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Mining Industry Consultants
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KUTCHO COPPER PROJECT
British Columbia, Canada

FEASIBILITY STUDY
NI 43-101 Technical Report

Prepared for: Kutcho Copper Corp.
Prepared by: CSA Global Consultants Canada Limited

Report Date: 22 December 2021
Effective Date: 8 November 2021

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Filename	R533.2021 Kutcho Feasibility NI 43-101 - FINAL (22 Dec 2021)
Last Edited	22/12/2021 2:47 PM
Report Status	Final

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Certificate of Qualified Person – Andrew Sharp

I, Andrew Willis Sharp, B.Eng. (Mining). P.Eng. FAUSIMM, do hereby certify that:

1) I am currently employed as Director Mining Engineering with CSA Global Consultants Canada Limited with an office at 1000-1100 Melville St, Vancouver, BC V6E 4A6, Canada.
2) This certificate applies to the Technical Report titled "NI 43-101 Feasibility Study Technical Report for the Kutcho Copper Project, British Columbia, Canada", with an Effective Date of 8 November 2021, (the "Technical Report") prepared for Kutcho Copper Corp. ("the Issuer").
3) I am registered as a professional engineer in good standing with Engineers and Geoscientists BC (#47907) and I am a Fellow in good standing of the Australian Institute of Mining and Metallurgy (#112949). I am a graduate from the University of Curtin, Kalgoorlie (1987). I have been involved or associated with the mining industry since 1987, in Australia, Malaysia, Ghana, Mexico, Papua New Guinea, Argentina, Bolivia, Colombia, Chile and Canada in production roles for 28 years and six years in consulting. In particular to the Kutcho Copper Project I have more than five years applied to underground mining in the same and similar techniques and more than 15 years in open cut mining in planning, geotechnical, and operational roles which includes VMS deposits and project work in British Columbia.
4) I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5) I completed a personal inspection of the Kutcho Project site on 23 June 2021 for a day.
6) I am responsible for Section numbers 1.1 to 1.4, 1.7, 1.9 to 1.18, 2 to 6, 15.1 to 15.4,15.7, 16.1 to 16.4, 18.1, 18.2, 18.4 to 18.6, 18.11.16, 19 to 27 of the Technical Report.
7) I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101.
8) I have had no prior involvement with the property that is the subject of the Technical Report.
9) I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.
10) As of the effective date of the Technical Report and the date of this certificate, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 22nd day of December 2021.

(Signed and Sealed)

Andrew Sharp

____

Andrew Willis Sharp, B.Eng. (Mining), P.Eng., FAusIMM

Certificate of Qualified Person – Robert Sim

I, Robert Sim, P.Geo, do hereby certify that:

1) I am an independent consultant of SIM Geological Inc. and have an address at 508–1950 Robson Street, Vancouver, British Columbia, Canada V6E 1E8.
2) I graduated from Lakehead University with an Honours Bachelor of Science (Geology) in 1984.
3) I am a member, in good standing, of Engineers & Geoscientists British Columbia, License Number 24076 and with Permit to Practice number 1003563.
4) I have practiced my profession continuously for 36 years and have been involved in mineral exploration, mine site geology and operations, mineral resource and reserve estimations and feasibility studies on numerous underground and open pit base metal and gold deposits in Canada, the United States, Central and South America, Europe, Asia, Africa, and Australia.
5) I have read the definition of "qualified person" set out in National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
6) I am responsible for the preparation of Sections 1.5, 1.6, 1.8, 7, 8, 9, 10, 11, 12 and 14 of the technical report titled, Kutcho Copper Project, Dease Lake, British Columbia, Canada, Feasibility Study NI 43-101 Technical Report, with an effective date of 8 November 2021 (the "Effective Date"), and a release date of December 22, 2021 (the "Technical Report").
7) I visited the Kutcho Project on 12 and 13 September 2018.
8) I am independent of Kutcho Copper Corporation and the Property, applying all of the tests in Section 1.5 of NI 43-101.
9) I have had prior involvement with the property that is the subject of the Technical Report. I was responsibly for the previous estimate of mineral resources with an effective date of 22 February 2019 that was described in a Kutcho Copper press release dated 4 March 2019.
10) I have read NI 43-101, Form 43-101F1 Technical Report ("Form 43-101F1") and the Technical Report and confirm the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.
11) As of the Effective Date, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 22nd day of December 2021.

"original signed and sealed"

____

Robert Sim, P.Geo

Certificate of Qualified Person – Shervin Teymouri

I, Shervin Teymouri, P.Eng., do hereby certify that:

1) I am currently employed as Principal Mining Engineer with Mineit Consulting Inc. with an office at 460-688 West Hastings St., Vancouver, BC V6B 1P1, CANADA.
2) This certificate applies to the Technical Report titled "NI 43-101 Feasibility Study Technical Report for the Kutcho Copper Project, British Columbia, Canada", with an Effective Date of 8 November 2021, (the "Technical Report") prepared for Kutcho Copper Corp. ("the Issuer").
3) I am registered as a professional engineer in good standing with Engineers and Geoscientists BC (#35469). I am a Geological Engineering graduate from the University of British Columbia (2005) and Master of Mining Engineering from the University of British Columbia (2007). I have been involved or associated with the mining industry since 2005, in China, Mexico, and Canada, in production roles for four years and 12 years in consulting. In particular to the Kutcho Copper Project, I have more than 10 years applied to underground mining in the same and similar techniques which includes project study work in British Columbia.
4) I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5) I did not visit the Kutcho Project site for inspection during or prior to the Study.
6) I am responsible for Section numbers, 15.5 to 15.6, 16.5, 18.11.14.1 and 18.11.15.1 of the Technical Report.
7) I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101.
8) I have had no prior involvement with the property that is the subject of the Technical Report.
9) I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.
10) As of the effective date of the Technical Report and the date of this certificate, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 22nd day of December 2021.

(Signed and Sealed)

Shervin Teymouri

____

Shervin Teymouri, BASc., M.Eng., P.Eng.

Certificate of Qualified Person – Marinus Andre de Ruijter

I, Marinus André De Ruijter, M.Sc. (Met Eng), P.Eng., do hereby certify that:

1) I am currently employed as Principal Metallurgical Engineer with Mineit Consulting Inc. with an office at 460-688 West Hastings St., Vancouver, BC V6B 1P1, CANADA.
2) This certificate applies to the Technical Report titled "NI 43-101 Feasibility Study Technical Report for the Kutcho Copper Project, British Columbia, Canada", with an Effective Date of 8 November 2021, (the "Technical Report") prepared for Kutcho Copper Corp. ("the Issuer").
3) I am registered as a professional engineer in good standing with Engineers and Geoscientists BC (#31031) and Northwest Territories and Nunavut (#L4672). I am a graduate of the University of the Witwatersrand, Johannesburg, South Africa, holding M.Sc. and B.Sc. (Metallurgical Engineering), and B.Sc. (Physics, Mathematics) degrees graduating respectively in 1979, 1973 and 1969. I have practiced my profession continuously since graduation.
4) My relevant experience with respect to the Kutcho Copper Project includes laboratory metallurgical test programs, interpretation of test results, and engineering design of copper and copper and zinc projects in Canada, United States of America, Mexico, Chile and Peru.
5) I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
6) I did not visit the Kutcho Project site for inspection during or prior to the Study.
7) I am responsible for preparation of Section numbers 13.1 to 13.3, 17.1 to 17.4, 17.6, 17.7, 17.8.3, 17.8.9, of the Technical Report.
8) I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101.
9) I have had no prior involvement with the property that is the subject of the Technical Report.
10) I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.
11) As of the effective date of the Technical Report and the date of this certificate, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 22nd day of December 2021.

(Signed and Sealed)

Marinus André De Ruijter

____

Marinus André De Ruijter, M.Sc. (Met Eng), P.Eng.

Certificate of Qualified Person – Brent Hilscher

I, Brent Hilscher, P.Eng, do hereby certify that:

1) I am currently employed as Vice President of Mineral Processing for ABH Engineering Inc. with an office at #315 2630 Croydon Dr Surrey, BC V3Z 6T3.
2) This certificate applies to the Technical Report titled “NI 43-101 Feasibility Study Technical Report for the Kutcho Copper Project, British Columbia, Canada”, with an Effective Date of 8 November 2021, (the “Technical Report”) prepared for Kutcho Copper Corp. (“the Issuer”).
3) I am a member in good standing of the Association of Professional Engineers and Geoscientists of the Province of British Columbia #37465. I am a graduate of the University of British Columbia in 1999 with a B.A.Sc. In Mining and Mineral Processing Engineering. I have practiced my profession continuously since 2000. I have had over 21 years of combined experience in process operations, engineering, economics, and design. I have worked on a variety of operations and engineering studies for gold, silver, copper, molybdenum, lead and zinc deposits throughout the world. I have personally led over 60 ore sorting studies or construction projects for the mining industry.
4) I have read the definition of “qualified person” set out in National Instrument 43-101 Standards of Disclosure for Mineral Projects (“NI 43-101”) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a “qualified person” for the purposes of NI 43-101.
5) I have not visited the Kutcho Project site.
6) I am responsible for Section numbers 13.4 and 17.5.1 of the Technical Report.
7) I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101.
8) I have had no prior involvement with the property that is the subject of the Technical Report.
9) I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.
10) As of the effective date of the Technical Report and the date of this certificate, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 22nd day of December 2021.

“original signed and sealed”

____

Brent Hilscher, P.Eng. B.A.Sc.

Certificate of Qualified Person – Kelly McLeod

I, Kelly McLeod, P. Eng., do hereby certify that:

1) This certificate applies to the Technical Report entitled "Feasibility Study Technical Report on the Kutcho Copper Project, British Columbia" with an effective date of 8 November 2021, (the "Technical Report") prepared for Kutcho Copper Corp.
2) I am currently employed as a Consultant, Metallurgy, with Allnorth. with an office at Suite 1200–1100 Melville Street, Vancouver, British Columbia, V6E 4A6.
3) I am a Professional Metallurgical Engineer registered with the APEGBC, P.Eng. #15868. I am a graduate of McMaster University with a Bachelors of Engineering, Metallurgy, 1984. I have practiced my profession intermittently since 1984 and have worked for the last 15 years consulting in the mining industry in metallurgy and process design engineering.
4) I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101. I am independent of the Issuer, Vendor, related companies or the project applying all of the tests in Section 1.5 of NI 43-101.
5) I have not personally visited the Kutcho Copper project site.
6) I am responsible for Sections 13.5, 17.5, 17.5.2, 17.8.1 and 17.8.2 of this Technical Report.
7) I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101.
8) I have had prior involvement with the property that is the subject of this Technical Report. I was responsible for Sections 13 and 17 of the JDS 2017 Prefeasibility Report.
9) As of the effective date of this Technical Report, to the best of my knowledge information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
10) I have read NI 43-101, and the Technical Report has been prepared in accordance with NI 43-101 and Form 43-101F1.

Effective Date: 8 November 2021

Signing Date: 22 December 2021

(signed and sealed)

Kelly McLeod, P. Eng.

Certificate of Qualified Person – James Garner

I, James Robert Garner, P. Eng., do hereby certify that:

1) I am currently employed as Engineering Group Lead, Civil with Allnorth Consultants Limited with an office at 100-275 Lansdowne Street, Kamloops, BC, Canada.
2) This certificate applies to the Technical Report titled "NI 43-101 Feasibility Study Technical Report for the Kutcho Copper Project, British Columbia, Canada", with an Effective Date of 8 November 2021, (the "Technical Report") prepared for Kutcho Copper Corp. ("the Issuer").
3) I am registered as a professional engineer in good standing with Engineers and Geoscientists BC (#49570) and I am a graduate from the University of British Columbia with a bachelor's degree in applied science specializing in civil engineering (2015). I have been involved in various aspects of civil engineering consulting since 2013.
4) I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5) I completed a personal inspection of the Kutcho Project site on 23 June 2021 for a day.
6) I am responsible for section numbers 18.3, 18.7 to 18.10, 18.11 (with the exception of 18.11.14.1, 18.11.15.1, and 18.11.16), and 18.12 of the Technical Report.
7) I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101.
8) I have had no prior involvement with the property that is the subject of the Technical Report.
9) I have read the NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.
10) As of the effective date of the Technical Report and the date of this certificate, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 22nd day of December 2021.

(Signed and Sealed)

James Garner

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James Garner, P.Eng.

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1 Summary

1.1 Introduction

This Technical Report was prepared by CSA Global Consultants Canada Ltd ("CSA Global") an ERM Group company in accordance with National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 43-101") and is suitable for filing with the Canadian securities commissions. This document was prepared for Kutcho Copper Corp. ("Kutcho") to disclose material information related to an updated Mineral Resource estimate and Feasibility Study ("FS") for the Kutcho Copper Project (the "Project"), located approximately 100 km east of Dease Lake in northern British Columbia, within the traditional territories of the Kaska Dena Nation and Tahltan Nation. The effective date of the Technical Report is 8 November 2021.The Technical Report summarizes the results of the Feasibility Study.

There are a number of Qualified Persons (QPs) responsible for various sections of the report and those are detailed in Section 2 and within the respective QP certificates.

A number of companies contributed to the Feasibility Study. Major contributors include:

- SIM Geological Inc. – Mineral Resource estimate
- Mineit Consulting Inc. – Mineral processing flotation and thickening, underground mine design, planning, underground Mineral Reserve
- Allnorth-Tahltan Inc. – Infrastructure design and capital, comminution design and plant processing costs
- ABH Engineering – Ore sorter test work, design and costing.

CSA Global was responsible for the open pit Mineral Reserve Estimate, open pit mining, financial evaluation, market studies, climate, hydrogeology, hydrology and overall project lead in evaluating the Project.

Other companies have provided important background information for the formulation of the study (where used but not relied upon) including:

- Onsite Engineering Ltd – Access road design and costing
- Terrane Geoscience Inc. – Geotechnical drilling support and geotechnical advice for open pit and underground mine design
- Piteau Associates Engineering Ltd – Tailings management facility (TMF) design.

The Mineral Resource Estimate was disclosed in accordance with NI 43-101 and Canadian Institute of Mining, Metallurgy and Petroleum ("CIM") standards and guidelines. All technical works have been undertaken using the May 2014 CIM Definition Standards for Mineral Resources and Mineral Reserves and CIM Best Practice guidelines.

All dollar values in this report are Canadian dollars ($ or C$) unless otherwise stated.

1.2 Project Description

The Project contains three main massive sulphide deposits: Main, Esso, and Sumac, that contain copper, zinc, silver and gold mineralization. Only the Measured and Indicated Mineral Resources of the Main and Esso deposits are used in the mine planning and economics of the Feasibility Study. The Sumac deposit is classed as an Inferred Mineral Resource and therefore was excluded from the study economics.

The Main deposit is designed to be mined primarily as a conventional loader and truck open pit operation, with a deeper and smaller portion mined by underground longitudinal longhole open stoping (LLHOS) with cemented rock fill (CRF). The Esso deposit is also designed to be mined using the same method. A total of 17.3 Mt of crusher feed is planned to be mined over an 11-year operating mine life preceded by 1.5 years of pre-production mining.

Approximately 14.5 Mt of the total crusher feed is supplied from the open pit and 2.8 Mt by the underground mines. A steady-state processing production rate of 4,500 tonnes per day (tpd) of crushed ore is expected be achieved by the end of the first year of operations. Low grade ore will be stockpiled and rehandled after the open pit is complete. The mine plan incorporates a potentially acid generating (PAG) rock management plan, and the site has a comprehensive water management program.

A conventional flotation process was designed for the separation of copper and zinc minerals from the high-pyrite content ore with two-stage crushing, ore sorting, SAG and ball milling, and two-stage concentration by rougher-cleaner flotation. A copper concentrate and a zinc concentrate will be produced. Thickened tailings are to be stored in a fully lined impoundment with a downstream embankment constructed from mine waste.

The approximately 119 km long access road between Highway 37 and the site will require upgrading before the major phase of construction can commence. Electrical power is to be supplied through an LNG powered 10 MW capacity power plant. The camp will have 250 rooms for the operational period and the site will employ a maximum of 380 people during construction.

1.3 Property and Location

The Property is located approximately 100 km due east of Dease Lake in the Liard mining division of northern British Columbia (BC), Canada.

The Property is within the Cassiar Mountains, just to the north of the continental divide between the Arctic and Pacific watersheds. The area is moderately rugged with elevations ranging from 1400 to 2200 masl. Most of the area is alpine with the tree line at approximately 1500 masl. Snow cover can persist for nine months of the year, particularly on shady north-facing slopes. Winters are long, cold and dry, while summers are short, cool and moist. The Property has an average annual temperature of -1.5 degrees Celsius (˚C) and experiences a mean annual precipitation of 816 mm, half of which is snow.

The site is accessible via a 900 m long gravel airstrip located 10 km west of the deposit and a 119km long seasonal road from Dease Lake that is presently only suitable for off-highway vehicles during the summer months.

The Project does not overlap with any private, provincial, or federal lands, nor does it encroach on any provincial or national parks, protected areas, or historic sites. There are no existing or past commercial developments, industrial facilities, or known permanent, seasonal, or temporary residences overlapped by the Property. No zoning bylaws apply to lands required for the Project. There are no agricultural land reserves near the Project.

1.4 History

In 1968 a joint venture exploration program operated by Imperial Oil Ltd identified mineralization on the Property during a follow up to a regional stream sediment geochemical program.

The Property has had multiple owners including Esso Minerals Ltd (EML) (formerly Imperial Oil, later Homestake Canada Ltd [Homestake]), Sumac Mines Ltd (SML) (a subsidiary of Sumitomo Metal Mining Co. Ltd), Homestake, American Reserve Mining Corporation (ARMC), Teck Cominco Ltd (TCL), Barrick, Western Keltic Mines (WKM), Sherwood Copper Corp., Capstone Mining Corp. and Kutcho Copper Corp. through the years. The property was acquired by Desert Star Holdings Corp. in 2017, which changed its name to Kutcho Copper Corp. in 2018.

Diamond drilling on the Property commenced in 1974 and diamond drilling, geological, geotechnical, geophysical, environmental, metallurgical, and engineering investigations were undertaken periodically through to the present day.

1.5 Geology and Mineralization

Located near the eastern end of an east-west striking narrow allochthonous belt of island arc volcanic rocks of Permo-triassic age, the Property contains three known Kuroko-type volcanogenic massive sulphide (VMS) deposits. They are aligned in a westerly plunging linear trend and from east to west they are called the Main, Sumac, and Esso deposits. The largest of the three, the Main deposit comes to surface near the eastern end of this trend, whereas the higher-grade Esso deposit occurs at depths from 400 m to almost 700 m below surface. Mineralization occurs as three deposits along a 3.5 km trend. Sulphide minerals occur in a series of massive sulphide lenses and include pyrite, sphalerite, chalcopyrite, bornite, minor chalcocite, and trace tennantite, galena, digenite, djurleite, and idaite. Gangue minerals include quartz, dolomite, ankerite, sericite, gypsum, and anhydrite.

1.6 Exploration/Drilling and Sampling

This report reflects exploration drilling and associated sampling information available for the Property as of 30 July 2021. The drilling activities and sampling practices used during all exploration on the property since 2000 are well documented and the resulting sample data was verified using quality control and quality assurance programs that follow accepted industry standards. The data generated during earlier drilling programs, conducted during the 1970s, 1980s and 1990s by reputable companies, was verified using a combination of methods including resurveying of the collar locations, resampling of historical drill core and conducting visual and statistical comparisons with the more recent data. This shows there are no significant differences between the older and more recent drilling. The author is of the opinion that the drilling database is adequate for use in the estimation of Mineral Resources.

1.7 Mineral Processing and Metallurgical Testing

The Kutcho deposit contains copper and zinc mineralization, which is intended for mining and, using flotation, the production of saleable grade copper and zinc concentrates with precious metal credits. Various metallurgical test programs have been conducted historically and have been reported in various studies, namely Wardop Engineering in 2007, SRK Consulting in 2008, JDS Energy and Mining in 2011, and an updated study by JDS in 2017. The present Feasibility Study has utilized the historical metallurgical information to develop a robust and reproducible flotation procedure to recover the copper and zinc using recent test programs at Bureau Veritas Commodities Canada (BVC), in 2018 and 2019, and Base Metal Laboratories (BML) in 2020 and 2021. In addition, in 2020 DRA Global Projects conducted a scoping level study for the hydrometallurgical recovery of gold, silver, and copper from the flotation tailings based on the results obtained from the BML 2020 study.

The metallurgical testing of mineralized sample material from the Kutcho deposit up to 2017 was reported in the preliminary Feasibility Study issued by JDS in 2017. Since the JDS PFS report was published in 2017, additional metallurgical test programs were conducted. These test programs included the following:

- BVC 2018, a mineralogical investigation of flotation products from an open cycle rougher test
- BVC 2019, a mineralogical investigation of flotation products from a locked cycle test
- DRA 2020, the evaluation of a hydrometallurgical process to recover gold, silver, and copper from the flotation tailings
- BML 2020, the development of a flotation flowsheet to recover copper and zinc
- BML 2021, additional test work to generate concentrate and tailings samples for settling tests.

The flotation results from the BML 2020 test work were ultimately developed to the stage where they could be reliably used for detailed process design for the recovery of the copper and zinc from the Kutcho mineralized material.

The most recent program completed with Base Metal Laboratories, Test Program BL714 in 2021, forms the major support to the Feasibility Study flotation estimates and a number of other process tests (such as settling, reagent addition rates, additions to comminution parameters, total concentrate assays, process water analysis, and others). Flotation testing was conducted as large-scale tests to generate the quantities of concentrate and tailings material required for the subsequent respective test programs. The Main Lens material was processed using the locked cycle test flowsheet which had been developed and optimized in the earlier test program (BL483). The Esso deposit material was treated using an open circuit flotation procedure but using the same reagents and following the procedure developed in BL483. In each test the standard reagents were added as developed previously, namely using zinc sulphate and the SO_2/Na_2S reagents for zinc and pyrite depression in the copper flotation stage which was followed by the zinc flotation stage and the activation of zinc with copper sulphate. In each case, collector and frother reagents were also added. The results obtained are detailed in Table 1-1.

Table 1-1: Flotation results for Main and Esso samples, BML report BL714

LCT Main Lens Sample, BL714, 2021									
Product	**Mass, %**	**Concentrate grade**				**Metal recovery, %**			
		Cu, %	**Zn, %**	**Au, g/t**	**Ag, g/t**	**Copper**	**Zinc**	**Gold**	**Silver**
Cu concentrate	4.2	27.9	6.06	1.94	293	70.8	13.6	24.8	47.1
Zn concentrate	1.1	3.17	56.2	2.28	195	2.1	32.7	7.5	8.1
Feed	100.0	1.66	1.87	0.33	26	-	-	-	-
Flotation conditions: Primary grind P_{80} 54 µm; Cu regrind P_{80} 12 µm; Zn regrind P_{80} 14 µm Sequential Cu-Zn flotation; Cu circuit, 1 cleaner stage; Zn circuit, 3 cleaner stages									
LCT Esso Lens Sample, BL714, 2021									
Product	**Mass, %**	**Concentrate grade**				**Metal recovery, %**			
		Cu, %	**Zn, %**	**Au, g/t**	**Ag, g/t**	**Copper**	**Zinc**	**Gold**	**Silver**
Cu concentrate	8.6	27.0	4.61	4.88	545	94.5	5.9	61.4	77.7
Zn concentrate	7.6	0.20	62.6	0.47	54	0.6	70.0	5.4	6.8
Feed	100.0	2.46	6.75	0.68	60	-	-	-	-
Flotation conditions: Primary grind P_{80} 55 µm; Cu regrind P_{80} 18 µm; no Zn regrind Sequential Cu-Zn flotation; Cu circuit, 1 cleaner stage; Zn circuit, 2 cleaner stages									

The results of the BL714 metallurgical test work were used in development of the mine plan processing recovery estimates.

Amenability and bulk test work was completed on ore sorter performance using x-ray transmission (XRT) and near infra-red (NIR) sensors. Ore sorting results based on the XRT technology showed that the XRT sorter rejected up to 18.7% of the sorter feed at metal grades significantly lower than the feed grades. The test also showed that it was not possible to discriminate sulphide types with the NIR technology. Data was also collected on the fine material that would by-pass the ore sorter.

1.8 Mineral Resource Estimate

The effective date for the Mineral Resource Estimate contained in this report is 30 July 2021. The estimate was prepared by Robert Sim, of SGI.

Mineral Resource estimates were generated using drillhole sample assay results and the interpretation of geological models which relates to the spatial distribution of copper, zinc, gold, and silver. Interpolation characteristics were defined based on the geology, drillhole spacing, and geostatistical analysis of the data. Grade estimates have been made using ordinary kriging into model blocks measuring 5 x 5 x 5 m (length x width x height) for the Main deposit and 5 x 2.5 x 5 m for the Sumac and Esso deposits. Potentially anomalous outlier grades have been identified and their influences on the grade models are controlled during interpolation using top cutting and outlier limitations. Specific gravities (SG), at the Main and Esso deposits, were estimated in model blocks using the inverse distance squared weighting (ID2) interpolation method. There is insufficient data to support estimation of SG at Sumac. The results of the modeling process were validated using a combination of visual and statistical methods to ensure the grade estimates are appropriate representations of the underlying sample data.

The Mineral Resources were classified according to their proximity to the sample data locations and are reported, as required by NI 43-101, according to the CIM Definition Standards for Mineral Resources and Mineral Reserves (May 2014). Mineral resources in the Measured category include areas of consistent mineralization where drillholes are spaced on a nominal 25 m pattern. Mineral Resources in the Indicated category have drilling on a regular 50 m pattern and Mineral Resources in the Inferred mineral category occur within a maximum distance of 100 m from a drillhole.

The estimates of Mineral Resources have been evaluated to ensure they exhibit reasonable prospects for eventual economic extraction. It is assumed that the Main deposit would be mined using a combination of open pit and underground extraction methods and the Esso and Sumac deposits would be mined using only underground mining methods based on the following technical and economic parameters:

- Mining (underground): US$43.00/t
- Mining (open pit): US$2.65/t
- Processing: US$20.50/t
- G&A: US$6.00/t
- Copper price: US$3.50/lb
- Zinc price: US$1.15/lb
- Gold price: US$1,600/oz
- Silver price: US$20.00/oz
- Copper process recovery: 87.6% (Main and Sumac), 94.5% (Esso)
- Zinc process recovery: 64.3% (Main and Sumac), 89.3% (Esso)
- Gold process recovery: 58.0% (Main and Sumac), 66.0% (Esso)
- Silver process recovery: 57.9% (Main and Sumac), 71.2% (Esso)
- Pit slope angle: 48.9°.

Based on the metal prices and recoveries listed here, recoverable copper equivalent (CuEqR) grades were calculated using the following formulae:

- Main and Sumac: CuEqR = (Cu% x 0.876) + (Zn% x 0.241) + (Au g/t x 0.441) + (Ag g/t x 0.006)
- Esso: CuEqR = (Cu% x 0.945) + (Zn% x 0.310) + (Au g/t x 0.466) + (Ag g/t x 0.006)

The projected cut-off grade for Mineral Resources considered amenable to open pit extraction methods at the Main deposit is determined to be 0.45% CuEqR. For Mineral Resources considered amenable to underground extraction methods, the cut-off grade at the Main and Sumac deposits is 1.05% CuEqR and at Esso it is 0.95% CuEqR. At these elevated cut-off thresholds, Mineral Resources at Main, Esso and Sumac form relatively continuous zones of mineralization that are considered amenable to underground mining methods.

The estimate of Mineral Resources, inclusive of mineral reserves is presented in Table 1-2. There are no known factors related to environmental, permitting, legal, title, taxation, socio-economic, marketing, political or other relevant factors which could materially affect the estimate of mineral resources. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

Table 1-2: Estimate of Mineral Resources (Effective Date 30 July 2021)

Resource category	Tonnes ('000)	CuEqR (g/t)	Cu (%)	Zn (%)	Ag (g/t)	Au (g/t)
Main Deposit (pit constrained, 0.45% CuEqR cut-off)						
Measured	7,213	2.31	1.64	2.35	24.7	0.36
Indicated	12,201	1.79	1.27	1.64	22.8	0.32
Measured + Indicated	**19,414**	**1.98**	**1.41**	**1.90**	**23.5**	**0.34**
Inferred	459	1.35	0.78	1.24	16.8	0.60
Main Deposit (below open pit, 1.05% CuEqR cut-off)						
Indicated	793	1.93	1.35	1.54	30.3	0.45
Inferred	1,717	1.87	1.19	1.90	26.1	0.49
Esso Deposit (0.95% CuEqR cut-off)						
Indicated	2,595	4.40	2.40	4.49	61.5	0.78
Inferred	1,624	2.15	1.32	1.59	35.8	0.42
Sumac Deposit (1.05% CuEqR cut-off)						
Inferred	9,086	1.49	1.06	1.53	16.2	0.16
ALL DEPOSITS COMBINED						
Measured	7,213	2.31	1.64	2.35	24.7	0.36
Indicated	15,590	2.23	1.46	2.11	29.6	0.41
Measured + Indicated	**22,802**	**2.26**	**1.52**	**2.18**	**28.1**	**0.39**
Inferred	12,886	1.62	1.10	1.58	20.0	0.25

Note: The Mineral Resource Estimate in Table 1-2 is considered to be amenable to a combination of open pit and underground extraction methods. Mineral Resources are inclusive of Mineral Reserves. At the Main and Sumac deposits, CuEqR = (Cu% x 0.876) + (Zn% x 0.241) + (Au g/t x 0.441) + (Ag g/t x 0.006). At the Esso deposit, CuEqR = (Cu% x 0.945) + (Zn% x 0.310) + (Au g/t x 0.446) + (Ag g/t x 0.006). Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. The effective date of the estimate of Mineral Resources is 30 July 2021. Table values may not sum correctly due to rounding.

1.9 Mineral Reserve Estimate

The effective date for the Mineral Reserve Estimate contained in this report is 8 November 2021 and was prepared by Andrew Sharp, P.Eng., of CSA Global. The Mineral Reserves are not in addition to the Mineral Resources but are a subset thereof.

The Project supports an economically viable open pit and underground mining operation. The Mineral Reserve Estimate is based on the Measured and Indicated categories of the Mineral Resource contained within the mine design. The Mineral Reserve Estimate has considered all modifying factors appropriate to the Project.

The reference point at which the Mineral Reserves are defined is where the ore is delivered to the process plant primary crusher.

There is no known mining, metallurgical, infrastructure, permitting or other relevant factors that could materially affect the Mineral Reserve estimate. However, it is important to note that permitting of the Project is not complete. Kutcho has initiated the environmental and social assessment process, including the permitting procedures to meet British Columbian provincial and Canadian federal regulations, and at the time of writing is gathering environmental data to meet statutory requirements.

The Mineral Reserves identified in Table 1-3 comply with the May 2014 CIM Definition Standards for Mineral Resources and Mineral Reserves and CIM Best Practice guidelines.

Table 1-3: Mineral Reserve for the Kutcho Copper Project (Effective Date 8 November 2021)

Mine section	Reserve category	Material type	Tonnage (Mt)	Diluted grade					Contained metal			
				Cu (%)	Zn (%)	Ag (g/t)	Au (g/t)	NSR (US$/t)	Cu (Mlb)	Zn (Mlb)	Ag (Moz)	Au (koz)
A. Esso Underground	Probable	Sulphide	2.23	2.14	4.13	54.9	0.71	199.48	105.3	202.6	3.9	50.7
B. Main Underground	Probable	Sulphide	0.60	1.40	1.54	32.2	0.57	120.17	18.5	20.4	0.6	11.0
C. Total Underground	Probable	Sulphide	2.83	1.99	3.58	50.1	0.68	182.68	123.8	223.0	4.6	61.7
D. Main Open Pit	Probable	Oxide	0.44	1.26	1.82	18.3	0.32	81.91	12.2	17.5	0.3	4.5
	Proven	Sulphide	6.80	1.64	2.38	24.5	0.37	135.15	245.7	356.5	5.4	80.8
	Probable	Sulphide	7.28	1.38	1.78	23.0	0.29	111.88	221.8	285.7	5.4	67.1
	Proven + Probable	**Sulphide + Oxide**	**14.51**	**1.50**	**2.06**	**23.5**	**0.33**	**121.85**	**479.5**	**659.3**	**11.0**	**153.0**
E. Total Open Pit and Underground	Probable	Oxide	0.44	1.26	1.82	18.3	0.32	81.91	12.2	17.5	0.3	4.5
	Proven	Sulphide	6.80	1.64	2.38	24.5	0.37	135.15	245.7	356.5	5.4	80.8
	Probable	Sulphide	10.11	1.55	2.28	30.6	0.40	131.68	345.5	508.7	9.9	128.8
	Proven + Probable	**Sulphide + Oxide**	**17.34**	**1.58**	**2.31**	**27.9**	**0.39**	**131.77**	**603.2**	**882.3**	**15.5**	**214.7**

Notes for the Mineral Reserve

1. *CIM definitions were followed for Mineral Reserves.*
2. *Mineral Resources are reported inclusive of Mineral Reserves.*
3. *The Inferred Mineral Resources do not contribute to the financial performance of the project and are treated in the same way as waste.*
4. *Sum of individual values may not equal due to rounding.*
5. *Metal prices – copper US$3.50/lb, zinc US$1.15/lb, silver US$20/oz, and gold US$1,600/oz.*
6. *No previous mining has occurred.*
7. *The reference point at which the Mineral Reserves are defined is where the ore is delivered to the crusher.*
8. *There is no known likely value of the following factors of mining, metallurgical, infrastructure, permitting or other relevant factor that could materially affect the estimate.*
9. *A complex NSR formula that varies on the basis of oxide and sulphide rock types and head grade was applied. It is fully documented within the NI 43-101 Feasibility Study.*

9a. *The oxide NSR formula may be approximated to ±1% accuracy for average head grades as: NSR (US$/t) = 50.26 x Cu% + 7.09 x Zn% + 0.14 x Ag g/t + 8.94 x Au g/t.*

9b. *The sulphide NSR formula may be approximated to ±1% accuracy for average head grades as: NSR (US$/t) = 57.82 x Cu% + 9.94 x Zn% + 0.34 x Ag g/t + 22.52 x Au g/t.*

Underground Specific Notes:

10. *Underground Mineral Reserve cut-off was C$129.45/t NSR.*
11. *The minimum pre-dilution mineable width applied is 2.5 m, average stope dimensions of 25 m height, 13.1 m wide and 42 m length. Minimum footwall dip of 47°.*
12. *A 0.5 m footwall and a 0.5 m hanging wall dilution is applied and wall dilution grades were taken from estimated block grades in these locations.*
13. *A net mining recovery and mining loss estimate after wall dilution was estimated as +2.2% tonnage and -6.2% grade.*
14. *Net mining dilution, mining recovery and mining loss of the Mineral Resource is estimated at 0.34 Mt (+12%) at 0.50% Cu, 1.13% Zn, 13 g/t Ag, and 0.12 g/t Au.*
15. *All stopes included in the Mineral Reserve were optimized to maximise net cashflow and must be cashflow positive after including access development capital costs.*

Open Pit Specific Notes:

16. *Open pit Mineral Reserve cut-off was C$38.40/t NSR for oxide ore and C$55.00/t NSR for sulphide ore. The sulphide grade is an operational cut-off and is above the breakeven cut-off of C$38.40/t NSR.*

17. *Mineral Resource between the breakeven and operational breakeven cut-off not included in the Mineral Reserve amounts to 1.24 Mt at 0.53% Cu, 0.63% Zn, 9.6 g/t Ag, and 0.13 g/t Au.*
18. *The mining SMU is 5 m x 5 m x 5 m. All ore is diluted to this block dimension and is considered the minimum recoverable dimension for the mining equipment and mining method selected.*
19. *Average dilution included in the Mineral Reserve is estimated as 1.17 Mt (+8.1%). Grades are derived from waste materials, estimated as 0.12% Cu, 0.13% Zn, 1.0 g/t Ag and 0.08 g/t Au.*
20. *Mining loss for material above the operational cut-off is estimated as 0.79 Mt (5.9%) at a grade of 0.91% Cu, 0.98% Zn, 15 g/t Ag, and 0.39 g/t Au.*
21. *The Open Pit Mineral Reserve lies within a pit design that is supported by geotechnical drilling and studies and optimized for net present value.*

1.10 Mining Method

Open pit mining will be by conventional loaders and trucks whereas underground mining will be done using longitudinal longhole open stoping (LLHOS) methods.

Open pit mining of the Main deposit is expected to commence during the construction period and ramp up over a two-year period before reaching an estimated average steady production rate of 1.8 Mtpa of ore at the end of Year 1. The open pit has an estimated eight-year life with a total of 14.5 Mt of ore mined at an average strip ratio of 5.6:1. The open pit is mined in two phases, Phase 1 and Phase 2. Direct processing of higher-grade ore and stockpiling of lower-grade ore (for processing at the end of the mine life) is a significant component of the mine plan. Phase 1 waste is used in construction of the TMF embankment and excess material is placed into a waste rock dump (WRD) located alongside the pit. The WRD will separately store PAG (potentially acid generating) and nPAG (not potentially acid generating) material. Once Phase 1 of the open pit mine is complete, Phase 2 PAG waste is backfilled directly into the void and excess is placed on the WRD (mostly nPAG). A significant portion of the WRD (all of the PAG rock component) will be rehandled into the Phase 2 void at the end of open pit mining operations. Including pre-production and ore/waste rehandling periods, the open pit life is 12.5 years.

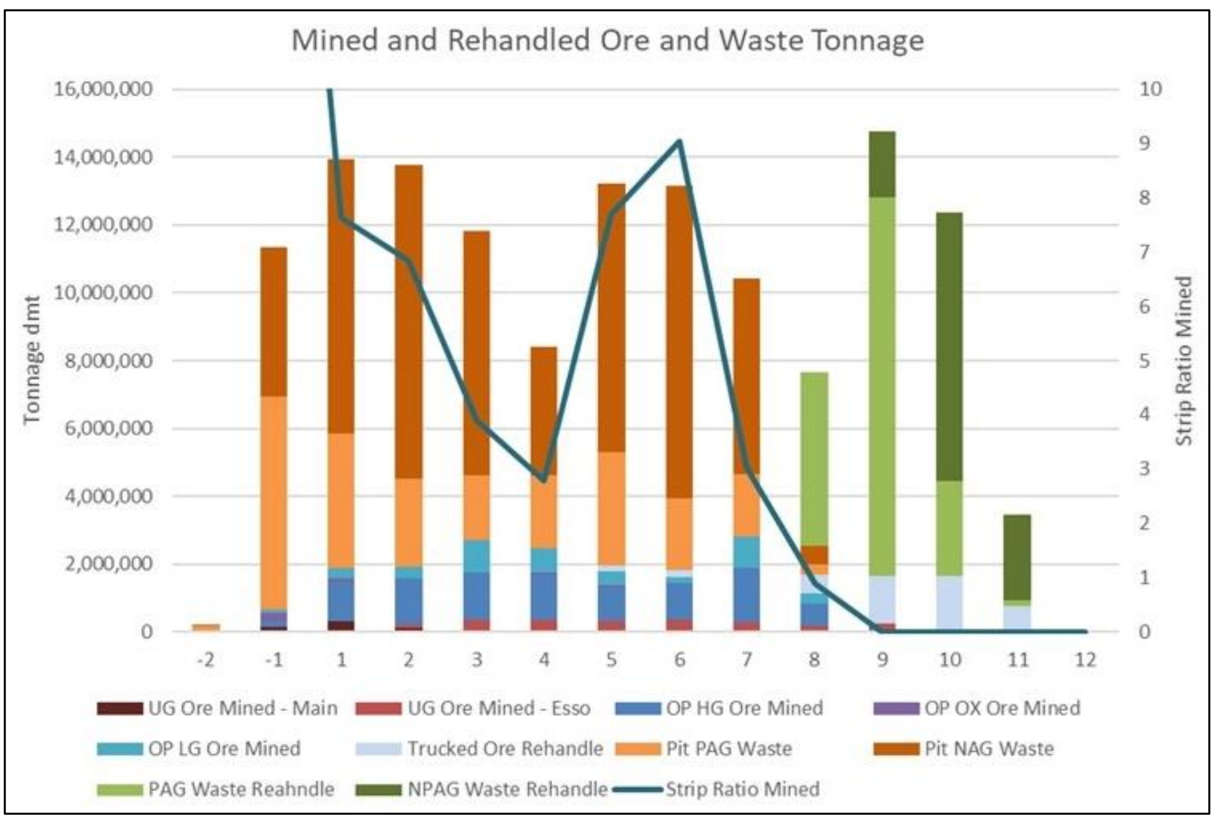


Figure 1-1: Life of Mine Material Movements

Figure 1-1 (above) shows total material movements and strip ratio by year for the open pit and underground mines (Abbreviations: HG – High Grade, LG – Low Grade, OX – Oxidized, nPAG – Not Potential Acid Generating, PAG – Potentially Acid Generating). The open pit mining equipment is planned to move up to 40,000 tpd. The truck fleet comprises up to eight 90-tonne class rigid body dump trucks. Ore and waste loading will be by two 10 m^3 wheel loaders and a 4 m^3 tracked excavator. A 7 m^3 wheel loader will be utilized for ROM stockpile rehandling. Blast hole drilling by two down-the-hole hammer drill rigs will be completed on 5 m benches for mixed ore and waste and 10 m benches for waste. Ancillary equipment will support the open pit mining operation, site surface activities and maintenance of the various access roads.

Production statistics for the underground and open pit mine plan are shown in Table 1-4.

Table 1-4: Summary of the underground and open pit mining operations

	Units	Pre-production	Early production	Steady state	Stockpile rehandle	Life of mine
		Years -2 to -1	Year 1	Years 2 to 8	Years 9 to 11	Years -2 to 11
Main and Esso underground mines						
Ore mined	Mt	0.15	0.32	2.09	0.27	2.83
Ore mined (1)	tpd	327	865	817	602	738
Underground mine life		10.5 years including pre-production				
Main open pit mines						
Ore mined	Mt	0.5	1.6	12.4	0.0	14.5
Waste mined	Mt	11.5	12.1	58.2	0.0	81.8
Strip ratio	W/O	22.8	7.7	4.7	0.0	5.6
Ore stockpile rehandle	Mt	0.0	0.0	0.9	3.8	4.7
Waste rehandle	Mt	0.0	0.0	5.1	26.5	31.6
Total material movement	Mt	12.0	13.7	76.6	30.3	132.5
	tpd	16,400	37,562	29,978	27,645	27,934
Open pit mine life		12.5 years including pre-production and rehandle				

Note: (1) Includes 1.25 years for pre-production and stockpile rehandle periods.

A single portal provides access to both the Main and Esso underground mines. The time between commencement on the underground portal and first ore mined is about nine months inclusive of portal construction. The first ore from Main is accessed about 340 m from the portal entrance. A spiral ramp then accesses additional mining levels before splitting to access Main ore further to the east. Access to Esso is from the Main spiral ramp. The Main underground mine is located beneath the Main open pit and is separated from it by a 25 m wide crown pillar. The top of the Esso deposit is approximately 400 m below surface and is accessed by an 1,800 m ramp extending from the Main ramp system. Further ramps are required to access the production stopes in the Main and Esso mines. The total ramp length for both mines is anticipated to be 5,437 m.

Underground mining of both the Main and Esso deposits is expected to be by longitudinal long-hole open stoping (LLHOS). The stopes will be backfilled using cemented rock fill (CRF) mostly comprised of ore sorter reject from the processing plant. Stope mining is planned to utilize temporary rib pillars and cable bolting as primary stope support measures. The underground portion of the Main deposit is expected to have a three-year mine life, extending from Year -1 to Year 2. Total ore production from Main underground is estimated to be 0.6 Mt, mined at an average rate of 600 tpd. On a quarterly basis it peaks at about 900 tpd. Underground mining of the Main deposit finishes early in the life of the open pit. Consequently, there is minimal interference between the two operations. The crown pillar is not planned for recovery.

Production from the Esso deposit is expected to commence in Year 2 and ramp-up to full production in the following year. Total projected ore production from Esso is 2.2 Mt at a steady state ore production rate of

0.3 Mtpa or 860 tpd. At these production rates, the mine life of Esso is estimated at nine years excluding pre-production which adds a further two years. Esso ore contains higher average grades of copper and zinc than the Main deposit. Consequently, even with the higher cost of underground mining, ore production from Esso is scheduled as early as possible.

The mine schedule allows for the stockpiling of low-grade ore from the open pit throughout operations and allowing higher grades to be processed earlier. The lower-grade stockpiles are processed in the final years of mine life, commencing in Year 8 but primarily from Year 9 through to Year 11. The ore sources for crusher feed and the crusher feed NSR C$/t are shown in Figure 1-2.

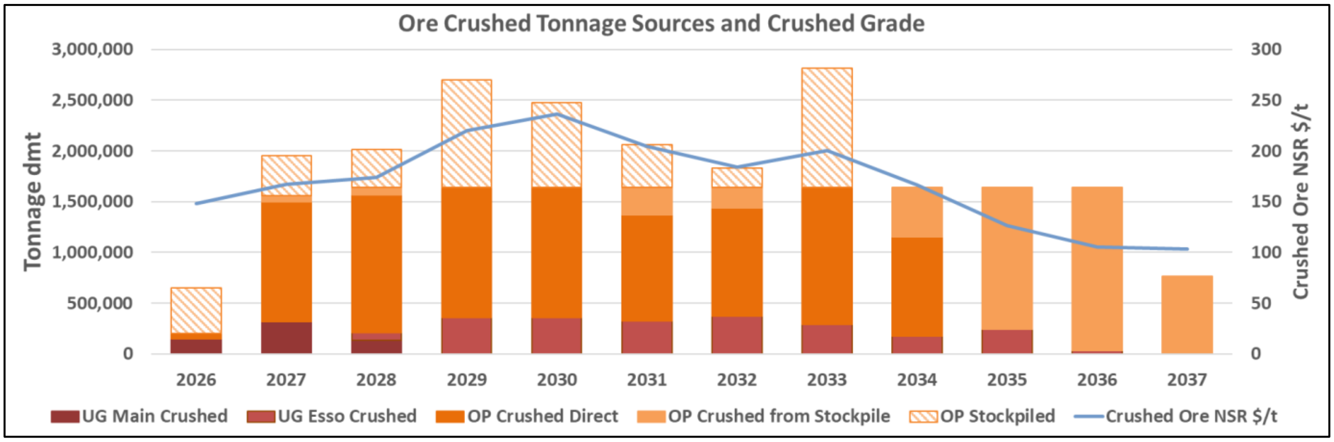


Figure 1-2: Crushed Ore Sources and Ore NSR C$/t

1.11 Recovery Methods

The crusher will process ore at a rate of 4,500 tpd with Life-Of-Mine (LOM) average head grades of 1.58% Cu, 2.31% Zn, 28 g/t Ag and 0.39 g/t Au. The ore is amenable to particle ore sorting. A pair of Tomra XRT particle ore sorters will reject an estimated 560 tpd of very low-grade rock (or about 13.5% of crusher feed). The rejected material is estimated to have an average grade of 0.12% Cu, 0.10% Zn, 3 g/t Ag and 0.08 g/t Au. The crushing and flotation circuits will process 3,940 tpd after the ore sorting circuit with an average LOM head grade of 1.79% Cu, 2.62% Zn, 31 g/t Ag and 0.43 g/t Au. The two-stage grinding circuit will target a product size of 80% passing (P_{80}) 55 µm, followed by a sequential flotation process to produce copper and zinc concentrates.

Based on test work, the overall LOM recoveries of produced metal for both concentrates combined from the flotation circuit are estimated to be approximately 88.6% for copper, 95.4% for zinc, 86.6% for silver and 60.5% for gold. The tailings will be pumped to a tailings management facility (TMF) and water recovered for re-use in the process plant. The crushing and ore sorter circuit will operate at an availability of 70%, while the grinding and flotation circuits will operate 24-hours per day, 365 days per year, at an availability of 92%. The concentrate filtration circuit will operate at an availability of 65%.

The plant will consist of the following unit operations:

- Primary crushing – A vibrating grizzly feeder and jaw crusher in open circuit, producing a final product P_{80} of approximately 95 mm.
- Secondary crushing and ore sorting – A cone crusher, triple deck screen and two XRT particle ore sorters producing an upgraded final product P_{80} of approximately 50 mm and a waste reject. Material smaller than 12.5 mm bypasses the ore sorter.
- Crushed material storage and reclaim – A 1,800-tonne bin with two reclaim belt feeders feeding the SAG mill feed conveyor.
- Ore sorter waste – 730-tonne bin with truck loadout.

- Primary grinding – A SAG mill in open circuit, producing a transfer size T_{80} of approximately 600 µm.
- Secondary grinding – A ball mill in closed circuit with a cyclone cluster, producing a final product P_{80} of 55 µm.
- Copper rougher and cleaner flotation – rougher flotation cells, rougher concentrate regrind (P_{80} of 15 µm), and cleaner flotation cells.
- Zinc rougher and cleaner flotation – rougher flotation cells, rougher concentrate regrind (P_{80} of 20 µm), and three stages of cleaner flotation cells.
- Concentrate dewatering, filtration and loadout – thickeners, stock tanks, and pressure filters for each of the zinc and copper concentrates.
- Final tailings thickener and disposal to the tailings management facility (TMF) – thickener and centrifugal pumps to discharge slurry to the TMF, a floating intake system and a shore-based pump reclaims TMF water for treatment before re-use in the process plant or as clean water release.

1.12 Project Infrastructure

In terms of infrastructure, the Project is a greenfield mine and so no infrastructure exists at the proposed mine site other than drill site access trails. An airfield and an exploration camp also currently exist at approximately 12 km to the west of the proposed mine site.

A single-lane access road is proposed to be built to connect the Project to Highway 37 immediately south of Dease Lake. The access road is designed to allow year-round access to the site for construction and operations. To enable the access road to be constructed in one construction season, a pre-construction winter road following the alignment of the existing access trail is proposed.

The major infrastructure required for the Project includes:

- Power generation utilizing liquified natural gas (LNG), LNG storage, and electrical distribution
- Crusher building housing a 4,500 tpd crushing, XRT particle ore sorters and bins for reject and fine ore storage
- Process plant building housing a SAG mill, ball mill, flotation circuit tailings thickeners, and concentrate filters
- Administration, open pit, plant, and maintenance offices
- Internal access roads
- Assay laboratory
- Warehouse
- Truck shop and wash bay
- Mine dry and underground mining offices
- Accommodation complex, associated buildings and services
- Diesel fuel storage
- Water treatment plant and discharge pipeline
- Various water storage and sedimentation ponds
- Water diversion channels, ditches, and sediment ponds
- Vent raises and underground mine portal
- Bulk explosives and detonator magazines
- Tailings management facility (TMF)
- Waste rock dumps (WRD)
- Temporary ore stockpiles.

The TMF is situated east of the open pit in the Playboy Creek catchment; tailings are thickened prior to disposal. This location was chosen amongst other sites investigated as it offers a combination of an embankment constructed from low-cost waste rock and improved storage efficiency. Additionally, the Playboy Creek site did not encroach on sensitive fish habitats, thus improving environmental and closure issues.

The TMF was designed to safely store 13 Mt of tailings along with site contact water (up to about one million cubic metres). The TMF is designed with a staged dam construction using downstream raises and competent nPAG waste rockfill. The embankment will be constructed as a starter dam in the pre-production period and then three downstream raises, principally using compacted nPAG waste rock from the open pit operation. The upstream face of the embankment and the entire impoundment will be lined with a composite system consisting of a geomembrane and locally sourced low permeability soil. The TMF will additionally be used to store contact water periodically and in response to seasonal demands.

Surface water management is designed to manage on site surface water during the life of the mine and to maintain environmental and operational performance for the Project. The general mine layout is designed to separate surface water into contact water and non-contact water catchments. The system captures, conveys, and treats all surface water within the active mine area (open pit, plant, waste rock dump and ore stockpile areas) which has come into contact with PAG rock. Treated water is designed to be conveyed by pipe to a designated discharge point on the Kutcho Creek, approximately 10 km to the west of the site. All other surface water within the site is designed to be collected and conveyed to a series of sediment ponds where sediment is to be removed from the water by sedimentation before being released to the receiving environment.

Waste rock from the open pit and underground mines will be stored in four separate locations; at the underground portal, the main waste rock dump (WRD), the in-pit dump, and the TMF embankment. The main WRD is designed to store both PAG and nPAG materials. The PAG WRD area is designed to be underlain by a layer of PAG rock. The area is also closest to the contact water pond for best management of potentially affected waters, which will be directed there from both the PAG and nPAG materials in the WRD.

As soon as practical in the mining schedule, Phase 1 of the open pit will be utilized as a backfill location for bothPAG and nPAG rock, although most will report to the WRD. At the completion of Phase 2 open pit mining, all remaining nPAG rock is planned to be rehandled and backfilled to the open pit void. The PAG in-pit backfill will be capped by a low permeability layer and nPAG rock.

Ore sorter reject rock is expected to be PAG and is planned to be used preferentially for underground stope backfill. Any excess reject rock is planned to be stored either in the PAG WRD or with the open pit backfill.

Two ore stockpiles are provided for the Project. A high-grade stockpile of around 250 kt is provided near to the crusher location, and a low-grade stockpile is designed to contain around 4.3 Mt of material (segregated oxidized and sulphide ore) to enhance metal recovery.

Some facilities are proposed to be located within the blast radius of the open pit and a Blast Management Plan will need to be developed to allow for blasting operations to be carried out safely.

During the Project construction and operational phases personnel from southern British Columbia and Yukon will be transported to site via air charter. Personnel from local communities and northern British Columbia will be bussed to site from Dease Lake.

Goods and materials required for the construction and operation of the Project are proposed to reach the site year-round over the main access road. Copper and zinc concentrates are proposed to be transported year-round to the port of Stewart for shipment to Asia in 5,000 dmt lots. The bulk terminal operator currently has empty and suitable concentrate storage facilities at the port to store the project’s concentrate based on the proposed production schedule.

1.13 Environmental Considerations

1.13.1 Land Use

The Project does not overlap with any private, provincial, or federal lands, nor does it encroach on any provincial or national parks, protected areas, or historic sites. No zoning bylaws apply to lands required for the Project. There are no forestry, agriculture nor agricultural land reserves near the Project. Placer and jade mining, guide outfitting, and trapping are the primary economic activities.

1.13.2 Regulatory Approval

Kutcho Copper does not anticipate that the Project will trigger a federal impact assessment under the Federal Impact Assessment Act as the Project's production rate of up to 4,500 tpd is below the threshold of 5,000 tpd. However, the Project does have several potential triggers relating to fish habitats, migratory birds, species at risk and explosives that could prompt more federal involvement with the EA process.

The Project Description for the Project was submitted under the British Columbia Environmental Assessment Act (2002) and the Project is reviewable under this Act. An Environmental Assessment (EA) is needed as the Project's proposed production capacity exceeds the threshold of 75,000 tpa of mineral ore for a new mineral mine. After submitting the Project Description, the Environmental Assessment Act was updated to the BC Environmental Assessment Act (2018) and the Project will now follow the 2018 Act assessment process. Since submitting the 2018 Project Description, Kutcho Copper has updated the mine plan for the Feasibility Study, so an amendment to the Project Description will be required.

Kutcho Copper initiated discussions to improve the road alignment, surface, and associated creek crossings in partnership with Tahltan and Kaska. Based on discussions with Tahltan and Kaska, the EAO has indicated that the access road could potentially be removed from the EA process. Kutcho Copper is not part of the Jade Boulder Road Association that has a special use permit for the road.

Kutcho Copper currently holds a MYAB permit for exploration This permit is valid for five years; it was issued in 2018 and will expire in March 2023. A suite of Federal and Provincial authorizations, licenses and permits that may be required for the Project have been identified.

1.13.3 First Nations and Community Considerations

The Project is located within the traditional territories of the Kaska Dena Nation and Tahltan Nation. Traditional use of lands and resources in the vicinity of the Project was documented for both the Kaska Dena Nation and Tahltan Nation. Kutcho Copper has engaged in regular communications with First Nations throughout the project. This contact continues to be maintained, and a formal consultation plan will be developed for the BCEAO with Input from both the Kaska Dena and Tahltan Nations. Kutcho Copper remains committed to working with First Nations through a jointly developed Consultation Plan. Kutcho currently has exploration agreements in place with both the Kaska Dena and Tahltan Nations.

1.13.4 Baseline Studies and Effects Management

Environmental and socio-economic investigations were conducted on the Project area in different periods and at differing levels of rigour. The historical data, pre-1980s data was largely excluded from consideration as the methods and data validation do not meet current standards. The Project’s most recent environmental and socio-economic baseline studies were initiated in 2005 and updated in 2012 and 2013 by Capstone Mining Corp. After acquisition from Capstone in 2017, Kutcho Copper started additional baseline studies in 2018, and these have continued through to 2021. These studies will complete the dataset required to support the preparation of an application to the BCEAO for an environmental assessment certificate and will represent a substantial baseline dataset collected over a long time period.

Kutcho Copper has endeavoured to eliminate potential environmental effects by refining the mine design, minimizing the project footprint, and not encroaching on fish-bearing watercourses. However, some effects are inevitable. The primary potential effects have been identified and mitigations developed.

1.13.5 Closure and Reclamation Plans

The Project would be developed, operated, and closed to leave the property in a condition that would mitigate potential environmental effects and restore the land as practical to meet end land use objectives agreed to with the community and First Nations. Kutcho Copper would start progressive closure during the operational life of the mine with a particular focus in the last three years of operation. During this time, mining would have ceased, processing of the stockpile would be ongoing, and backfilling of the open pit would continue. It is anticipated that the majority of the nPAG rock would be used in the closure of the TMF and the backfilling of the open pit. The PAG rock would be placed into the open pit and enclosed by nPAG material during the backfilling process. The open pit would be partially backfilled during operations and then completely backfilled with remaining PAG and nPAG material during the stockpile rehandling period at the end of mine life. The material would then be covered with subsoil, topsoil and revegetated.

Closure after processing finishes is anticipated to take 12 months. After closure, reclamation is expected to take a further 24 months. Under the BC Mines Act and the Health, Safety, and Reclamation Code for Mines in British Columbia, the primary objective of the reclamation plan would be to return, where practical, areas disturbed by mine operations to their pre-mining land use and capability.

Post-closure monitoring is expected to last between five and 10 years after closure with requirements confirmed during operations and at closure. Closure of the water treatment plant and associated facilities would occur when monitoring indicates that direct discharge of untreated flows meets Environmental Management Act permit limits.

A financial model for the closure and rehabilitation plan at the conceptual level was estimated. The estimate includes all costs for underground workings closure, land-forming, topsoil placement, revegetation, creation of post mine water rehabilitation of the facilities, refurbishment of the water treatment plant, and post mine water treatment costs. Even though the post-closure monitoring is expected to last between five and ten years, for the sake of conservatism, costs were included for water treatment and monitoring beyond this. The total estimated cost of the closure activities is approximately C$34 million and estimates for an environmental bond were included in the financial model.

1.14 Capital Cost Estimate

The Feasibility Study outlines an initial (pre-production) capital cost estimate of C$483 million (undiscounted), including a contingency of 10.6% (excluding mine closure bonding). Sustaining capital costs (undiscounted, including contingency) over the life of the mine are estimated at C$90 million.

Initial open pit mining capital costs cover site development for the pits, haul roads and stockpiles, and mining to provide construction material and expose ore for processing. Initial underground capital costs include the construction of the underground portal, development to access the Main and Esso deposits, and other infrastructure. The mobile mining fleets would be leased from the equipment suppliers. The initial deposits on leased equipment (typically 25%) are reflected as a capital cost, with the remainder paid off over several years and treated as a mining operating cost.

The TMF embankment material is supplied from the open pit nPAG waste material, with Phase 1 construction captured as a pre-production mining capital cost, and subsequent construction phases as a mining cost. The LNG power generating sets would be leased on the same basis as mobile mining equipment, with repayments reflected as part of the power cost applied to the various operations.

Initial, sustaining and closure capital costs were estimated based on Q2/3 2021, un-escalated Canadian dollars and are summarized in the table below. Vendor quotes were obtained for all major equipment. Some of the costs were developed from first principles, while some were estimated based on factored references and experience from similar projects.

A mine closure bond is allowed for, lodged at the commencement of construction activities, and thereafter linked to progressive site disturbance and post-closure management. Physical closure, rehabilitation, and post-closure management costs are estimated at C$34.5 million. A salvage value for equipment is applied at the end of mine life.

The weighted average of each input or cost element was used to calculate the contingency and these varied between 9.0% and 11.5% for each cost element. The contingency estimate totals C$54.9 million or 10.6% of the total direct and indirect capital costs. Contingency for the site rehabilitation is estimated separately at 15% and is included in the site rehabilitation estimate cost in order to determine the mine closure bond directly.

The capital cost estimate for the Kutcho Project is summarized in Table 1-5.

Table 1-5: Summary of capital costs

Component (1), (2)		Units	Initial	Sustaining	Closure	Total
Direct Costs	Mining costs	C$M	132.9	44.8	0.0	177.7
	Process plant	C$M	106.0	15.1	0.0	121.2
	On-site infrastructure	C$M	54.2	15.8	0.0	70.1
	Off-site infrastructure	C$M	31.7	0.0	0.0	31.7
EPCM and indirect costs		C$M	75.3	0.0	0.0	75.3
Owner's costs (including working capital)		C$M	35.9	5.6	0.0	41.5
Total capex without contingency		**C$M**	**436.1**	**81.3**	**0.0**	**517.5**
Contingency (3)		C$M	46.7	8.2	0.0	54.9
Salvage		C$M	0.0	0.0	-18.0	-18.0
Mine closure bond		C$M	10.0	9.3	-19.3	0.0
Closure and rehabilitation (4)		C$M	0.0	10.2	24.3	34.5
Total capex with rehabilitation		**C$M**	**492.8**	**109.0**	**-13.0**	**588.9**

Notes:

(1) All values stated are undiscounted.

(2) No inflation or depreciation of costs were applied; all costs are in 2021 money values. Major underground mobile equipment, all open pit mobile equipment and the power gensets are leased.

(3) Includes average contingency of 10.6%.

(4) Includes contingency of 15%.

1.15 Operating Cost Estimate

Operating costs were development on the basis of equipment productivities, vendor and contractor quotations or supplier costs for machinery, consumables, and services. Labour costs across all activities were estimated from file data and benchmarking. The location of the Project in northern British Columbia was also considered. Operating cost estimates are based on Q2/3-2021 un-escalated Canadian dollars.

The mining operating costs include the leasing of all mining equipment, and all activities operated and managed by the owner.

Process plant operating costs include all consumables (balls for the ball mill, reagents, and chemicals) power, external services, TMF operating costs, and plant maintenance to allow for the production of two concentrates. The battery limit for plant operating is at the point of production of the concentrate at site. Concentrate transport, port storage and onward costs are considered off-site costs.

General and administration costs were developed from file data of similar operations in Canada and include some zero-based cost estimation for items such as water treatment. Costs not developed by a zero-based method are covered by budget estimates provided by regionally based companies.

The average LOM mine operating cost is estimated to be C$65.89/t of ore crushed. Table 1-6 summarizes the LOM operating costs.

Table 1-6: LOM operating costs

Component	Units	Early production	Steady state	Stockpile rehandle	Life of mine [2]
		Year 1 [1]	Years 2 to 8	Years 9 to 11	Years 1 to 11
Open pit mining cost	C$/t OP ore mined	27.68	20.55	NA [3]	24.75
Underground mining cost	C$/t UG ore mined	62.17	57.88	25.61	55.38
Plant processing cost	C$/t ore crushed	28.06	28.48	27.53	28.21
General and administration	C$/t ore crushed	10.01	9.36	6.69	8.79
Total	**C$/t ore crushed**	**79.24**	**70.51**	**47.67**	**65.89**

Notes:
(1) Year 1 includes pro-rated adjustments for working capital.
(2) Pre-production tonnages and costs are not included in the LOM operating cost summary (these are Years 2 and -1 and are capitalized).
(3) No ore mined, rehandle period.

The cost of power delivered by LNG generators, calculated from first principals, is estimated at C$0.176/kWhr or C$0.195/kWhr including generator leasing costs. The cost of diesel delivered to site was determined to be C$1.17/L.

1.16 Economic Analysis

The economic analysis undertaken for this Feasibility Study assumes that a copper and a zinc concentrate would be produced. The copper concentrate would obtain value mainly from copper, but also from contained silver and gold. The zinc concentrate would primarily obtain value from zinc, but also gold and sometimes silver. Kutcho Copper intends to sell the concentrates to either traders or refineries and would not directly produce metals. Appropriate deductions for treatment, refining, transport and insurance costs are applied Penalties for impurities and deleterious elements have been applied to the copper and zinc concentrates. The analysis assumes that the Project is 100% equity financed.

Roskill Consulting Group Ltd (Roskill) was contracted in 2021 to complete a market overview of the relevant metals, price range recommendation, and concentrate sales terms. In addition, Kutcho provided consensus forecast metal prices from S&P Global IQ. Based upon analysis of the metal price historical and forecast data the exchange rate and base case metal prices used in the economic analysis are shown in Table 1-7.

Table 1-7: Exchange rate and metal prices used in the financial analysis

Parameter	Value
Exchange rate	0.76:1 (USD:CAD)
Copper price	US$3.50/lb (US$7,716/t)
Zinc price	US$1.15/lb (US$2,525/t)
Silver price	US$20.00/oz
Gold price	US$1,600/oz

There is no guarantee that any of the metal prices used in the base case or sensitivities cases are representative of future metals prices. The results of the pre- and after-tax economic analyses are provided in Table 1-8.

The financial evaluation presents the determination of the key economic performance indicators for the Project, including the Net Present Value (NPV), the payback period (time in years to redeem the initial capital investment), and the Internal Rate of Return (IRR) for the project. The discounted cash flow (DCF) model is reported at 100% attributable equity. Annual cash flow projections are estimated over the life of the mine based on the estimates of capital expenditures, production costs and sales revenues. The sales revenue is based on the production of a copper concentrate and a zinc concentrate that are planned to be sold to refineries based in the East Asia region.

The estimates of initial and sustaining capital expenditures and site production costs were developed specifically for this project. Total initial and sustaining capital is C$493 million and C$99 million respectively.

The Project has cash costs of US$1.11/lb of CuEq and an all-in sustaining cost of US$1.80/lb of CuEq. The NPV at assumed long term metal prices using a 7% discount rate is C$461 million and the internal rate of return (IRR) is 25% (Table 1-8). Payback of the initial capital occurs 3.4 years after commercial production commences (Table 1-8. The plant produces a payable total of 504 Mlb of copper, 531 Mlb of zinc, 8.3 Moz of silver, and 107 koz of gold over a 10.5-year processing life. The processing life is preceded by a two-year on-site construction period and commissioning period, which is itself preceded by a two-year off-site road construction period. The processing life is followed by a period for closure and rehabilitation.

Key financial outcomes are presented in Table 1-8.

Table 1-8: Summary of economic results

Parameter	Unit	Pre-tax results	After-tax results
NPV – no discount	C$M	1,264	841
NPV – 7% discount (1)	C$M	737	461
Annual average net operating income (2)	C$M	158	120
IRR	%	31.3%	24.8%
Payback period	production years	3.2	3.4

Notes:

(1) Method of discounting for NPV – all capital years are undiscounted; discounting commences in Year 1. The after-tax NPV for commencing discounting in Year -3 is C$360 million, the NPV for commencing discounting in Year -2 is C$388 million and the NPV for commencing discounting in Year -1 is C$431 million.

(2) Covers production Years 1-11 only, pre-production and Year 12 are excluded.

The sensitivity of the Project to a number of parameters was estimated (Table 1-9). Neither the cut-off grade, mine plan nor the processing plan were altered. The Project NPV is most sensitive to changes in metal price and factors such as exchange rate, head grade, and metallurgical recovery. Of the four metals, the project is most sensitive to variations associated to copper (the contribution to the Project economics by metal type is approximately 65% from copper, 23% from zinc, 6% from silver and 6% from gold). After metal price, IRR is equally sensitive to changes in operating and capital expenditure. NPV on the other hand is more sensitive to operating expenditure than capital costs. The project NPV is robust to changes in capital and operating expense. The IRR is estimated to be more sensitive to capital expense than operating cost. The NPV and IRR sensitivity of the project is estimated to be four times greater to metal price than capital or operating costs.

Table 1-9: Key sensitivity analyses for the Project

Parameter	Sensitivities	After-tax IRR (%)	After tax NPV (C$ M)
Revenue (metal price)	-25%	8.8	43
	Base Case	**24.8**	**461**
	25%	38.6	869
CAPEX	-25%	32.0	511
	Base Case	**24.8**	**461**
	25%	19.4	410
OPEX	-25%	31.8	615
	Base Case	**24.8**	**461**
	25%	18.3	306

Parameter	Sensitivities	After-tax IRR (%)	After tax NPV (C$ M)
Discount rate	-25% (5.3%)	24.8	540
	Base Case (7.0%)	**24.8**	**461**
	+25% (8.8%)	24.8	391

1.16.1 Royalties and Wheaton Precious Metals Purchase Agreement

Kutcho Copper Corp entered into a Precious Metal Purchase Agreement (PMPA) with Wheaton Precious Metals Corp. ("Wheaton") on 14 December 2017. The full terms of this agreement are not included in the Feasibility Study as it only comes into effect should Wheaton elect not to participate after the completion of the Feasibility Study. If not terminated the PMPA, as interpreted from the Agreement, changes certain revenue parameters which are disclosed in in further detail in Section 19.7. The injection of two deposit payments by Wheaton, one of C$76 million in the first year of construction (2025, Year -2), and a bonus deposit of C$26 million for achieving 4,500 tpd throughput in the first year of commercial production (2027, Year 1) reduces funding requirements and is estimated to increase the IRR to 25.9%. The net impact of investment and repayments to the Project is estimated to reduce the Project NPV from C$461 million to C$417 million.

1.16.2 Royal Gold Agreement

Royal Gold Inc. holds a "back-in" rights to certain claims as described in Section 4.5. Royal Gold is entitled to a royalty of 2% of net smelter returns on certain claims if they do not elect to exercise the "back-in" rights. Royal Gold acquired this royalty interest effective 1 October 2008, as part of the acquisition of a royalty portfolio from Barrick Gold Corporation. The economic assumption of this Technical Report is that Royal Gold will elect to not 'back-in' and therefore the only effect applied to the Project from this agreement is that the Royal Gold royalty is allocated to the economic analysis.

A 2% NSR royalty held by Royal Gold Inc. over certain portions of the Project was applied to the cash flow model for a total of C$26 million (undiscounted).

1.16.3 Taxes

Tax considerations included in the cash flow are:

- BC Mineral Tax: 13%
- BC Income tax: 12%
- Federal Tax: 15%.

Investment and new mine allowances have been applied against the BC Mineral Tax. A 2% provincial minimum tax payable on net current proceeds, which is credited against the Mineral Tax, is calculated based on operating profit less applicable capital cost deductions. The mining tax is deductible in computing provincial and federal income tax. Canadian Development Expenses and Canadian Exploration Expenses have been applied to the tax model. Total taxes payable over the life of the Project (undiscounted) are estimated at C$422 million after allowing for tax credits of C$30 million accumulated by Kutcho Copper to date. It is an assumption of the Project that tax deductions associated to the new mines provision will continue to apply.

1.17 Conclusions

The Project contains a substantial copper-zinc sulphide resource with gold and silver by-product credits. The Main deposit can be mined predominantly by open pit methods with some ore extraction by underground mining. The Esso deposit can be mined using underground methods. The Project, as described in this Technical Report, yields positive economic results based on industry-standard design methods of mining and mineral processing.

The QPs are not aware of any fatal flaws with the Project. As with most mining projects, there are several risks and opportunities that could impact the Project viability and those considered most pertinent to the Project are elaborated upon in the Report.

1.18 Recommendations

It is recommended that the Project progress towards detailed engineering. The environmental approval process requires a closure, which will be continued with the view to completion. Additional work around metallurgy, water quality prediction, some mining studies, and geotechnical data collection should be conducted to further de-risk the Project. These programs are estimated to cost an additional C$2.95 million prior to detailed engineering.

2 Introduction

This Technical Report was prepared by CSA Global Consultants Canada Ltd ("CSA Global") in accordance with National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 43-101") and is suitable for filing with Canadian securities commissions. This document was prepared for Kutcho Copper Corp. ("Kutcho Copper" or "Kutcho") to disclose material information related to an updated Mineral Resource estimate and Feasibility Study ("FS") for the Kutcho Copper Project (the "Project"), located approximately 100 km east of Dease Lake in northern British Columbia, within the traditional territories of the Kaska Dena Nation and Tahltan Nation. The effective date of the Technical Report is 8 November 2021. The Technical Report summarizes the results of the Feasibility Study.

2.1 Qualified Persons and Site Visits

The Qualified Persons ("QP" or "QPs") preparing this Technical Report are specialists in the fields of geology, exploration, mineral resource and mineral reserve estimation and classification, geotechnical, environmental, permitting, metallurgical testing, mineral processing, processing design, capital and operating cost estimation, and mineral economics.

None of the QPs or any associates employed in the preparation of this Technical Report has any beneficial interest in Kutcho Copper and neither are they insiders, associates, or affiliates. The results of this report are not dependent upon any prior agreements concerning the conclusions to be reached, nor are there any undisclosed understandings concerning any future business dealings between Kutcho Copper and the QPs. The QPs are paid a fee for their services in accordance with normal professional consulting practice.

The following individuals, by virtue of their education, experience, and professional association, are considered QPs as defined in the NI 43-101, and are members in good standing of appropriate professional associations.

Table 2-1: Qualified Person site visits

Qualified Person	Site visit date
Andrew Sharp, P.Eng.	23 June 2021
James Garner, P.Eng.	23 June 2021
Kelly McLeod, P. Eng.	No site visit
Marinus Andre de Ruijter, P.Eng.	No site visit
Brent Hilscher, P.Eng.	No site visit
Shervin Teymouri, P.Eng.	No site visit
Robert Sim, P.Geo.	12–13 September 2018

QP report author, Mr Andrew Sharp completed a personal inspection (site visit) of the Property on 23 June 2021. The site was viewed from helicopter and several landings occurred at sites of future potential installations including the Playboy Creek tailings management facility, waste rock dump, pit, diversion channel, Esso ventilation raise and the then proposed road crossing for the Andrea Creek. Mr Sharp observed no significant differences between electronic data and topological, hydrological and vegetation realities. Mr Sharp also observed several requested drillholes to obtain a sense of rock mass quality for open pit mining and found no observable difference between the core viewed and electronic reports for geotechnical rock mass parameters.

QP report author, Mr James Garner visited the site on 23 June 2021 together with Mr Andrew Sharp and viewed the same locations but was not actively engaged in core review. The QP observed no significant differences between electronic data used in the Project estimation against topological, hydrological and vegetation realities.

QP report author, Mr Robert Sim visited the Property on 12 and 13 September 2018 where he met with site personnel and observed ongoing drilling operations that were on the Main deposit at that time. Core handling and sampling procedures were observed and reviewed, as well as procedures used to collect SG measurements.

Mr Robert Sim completed a full tour of the deposit areas and the surrounding Property was conducted by helicopter, and stops were made to observe the current drilling activities as well as stops at several of the areas where previous drilling had occurred; this included a visit to an exposure of massive pyrite uncovered during drill site preparation in the Main deposit.

Drill core from a series of randomly selected holes was reviewed by Mr Robert Sim and compared to the information in the database. The geological descriptions appeared to be reasonable, and visual observations of the sulphide minerals present reflected the grades in the sample database.

The Project is at an advanced exploration stage and site visits by Kelly McLeod, Marinus Andre de Ruijter, Brent Hilscher, and Shervin Teymouri were not necessary to complete this Technical Report.

Table 2-2: QP responsibility matrix

NI 43-101 Form F1 Item	QP responsibility
Item 1: Summary	Andrew Sharp (1.1 to 1.4, 1.7, and 1.9 to 1.18)
	Robert Sim (1.5, 1.6, and 1.8)
Item 2: Introduction	Andrew Sharp
Item 3: Reliance on Other Experts	Andrew Sharp
Item 4: Property Description and Location	Andrew Sharp
Item 5: Accessibility, Climate, Local Resources, Infrastructure and Physiography	Andrew Sharp
Item 6: History	Andrew Sharp
Item 7: Geological Setting and Mineralization RSM	Robert Sim
Item 8: Deposit Types	Robert Sim
Item 9: Exploration	Robert Sim
Item 10: Drilling	Robert Sim
Item 11: Sample Preparation, Analyses, and Security	Robert Sim
Item 12: Data Verification	Robert Sim
Item 13: Mineral Processing and Metallurgical Testing	M. Andre de Ruijter (13.1 to 13.3)
	Brent Hilscher (13.4)
	Kelly McLeod (13.5)
Item 14: Mineral Resource Estimate	Robert Sim
Item 15: Mineral Reserve Estimate	Andrew Sharp (15.1 to 15.4, and 15.7)
	Shervin Teymouri (15.5 and 15.6)
Item 16: Mining Methods	Andrew Sharp (16.1 to 16.6) Shervin Teymouri (16.7 to 16.8)
Item 17: Recovery Methods	M. Andre de Ruijter (17 to 17.4, 17.6, 17.7, and 17.8.3 to 17.8.9)
	Brent Hilscher (17.5.1)
	Kelly McLeod (17.5, 17.5.2, 17.8.1, and 17.8.2)
Item 18: Project Infrastructure	Andrew Sharp (18.1, 18.2. 18.4 to 18.6, 18.11.16)
	James Garner (18.3, 18.7 to 18.10, 18.11 (with the exception of 18.11.14.1, 18.11.15.1 and 18.11.16), and 18.12)
	Shervin Teymouri (18.11.14.1 and 18.11.15.1)
Item 19: Market Studies and Contracts	Andrew Sharp

NI 43-101 Form F1 Item	QP responsibility
Item 20: Environmental Studies, Permitting, and Social or Community Impact	Andrew Sharp
Item 21: Capital and Operating Costs	Andrew Sharp
Item 22: Economic Analysis	Andrew Sharp
Item 23: Adjacent Properties	Andrew Sharp
Item 24: Other Relevant Data and Information	Andrew Sharp
Item 25: Interpretation and Conclusions	Andrew Sharp
Item 26: Recommendations	Andrew Sharp
Item 27: References	Andrew Sharp

2.2 Kutcho Copper Corp.

Kutcho Copper Corp. is a Canadian-based exploration and development company. Kutcho Copper's shares are listed on the TSX Venture Exchange ("TSX:V") under the symbol KC and on the OTCQX as KCCFF with its head office situated at:

1030 West Georgia St Suite 717
Vancouver BC V6E 2Y3
Tel: 604 628 5623

2.3 Sources of Information

The sources of information include data and reports supplied by Kutcho Copper as well as documents cited throughout the report and referenced in Section 27.

Background Project information was adapted from the most recent Technical Report titled Pre-Feasibility Study Technical report on the Kutcho project, British Columbia with an effective date of 15 June 2017 (JDS 2017).

Other major sources of information include:

- Wardrop Kutcho Project Pre-Feasibility Study dated 25 October 2007
- SRK Preliminary Economic Assessment Technical Report dated 31 May 2008
- Consolidated Management Consultants Electrical Power Options, 2011 Pre-Feasibility Study Report dated June 2010
- EBA Kutcho Project – Main Deposit Pre-feasibility Level Geotechnical Evaluation Report dated 29 November 2010
- EBA Kutcho Project – Esso Deposit Pre-feasibility Level Geotechnical Evaluation dated 29 November 2010
- Onsite Engineering Ltd. Onsite Kutcho Access Road, Geometric Road Design Report dated 17 December 2010
- Stantec-Mining (Stantec) Capstone Kutcho Project, Prefeasibility Level Ventilation Planning & Design Report dated 15 December 2010
- JDS Energy & Mining Inc. Kutcho Copper Project Pre-Feasibility Study, British Columbia with an effective date of 15 February 2011
- Terrane, 2021. Feasibility Geotechnical Underground Report: Kutcho Copper Project. Halifax: Terrane Geoscience Inc.
- Onsite Engineering Ltd. 2021. Kutcho Access Road, Geometric Road Design Report. Onsite Engineering Ltd. Dated 13 October 2021
- Piteau Associates Engineering Ltd., 2021. Kutcho Tailings Management Facility Geotechnical Assessment report prepared for CSA Global 2021

- CSA Global., 2021. Kutcho Copper Project – Water Management Study, CSA Global Consultants Report dated 17 December 2021.

2.4 Currency and Rounding

Unless otherwise specified or noted, the units used in this technical report are metric. Every effort was made to clearly display the appropriate units being used throughout this technical report. Currency is in Canadian dollars ($ or C$). United States dollars are shown as US$.

This report includes technical information that required subsequent calculations to derive subtotals, totals, and weighted averages. Such calculations inherently involve a degree of rounding and consequently introduce a margin of error. Where these occur, the QPs do not consider them to be material.

3 Reliance on Other Experts

The QP authors of this technical report are not qualified to comment on any legal and taxation considerations relating to the status of the Property and expresses no opinion as to the legal ownership status of the Property. The QP authors were reliant on Kutcho Copper with regards to the legal status of mineral claims and surface ownership information provided in the technical report. However, the QP authors did verify the mineral claims listed in this technical report against those listed as held by Kutcho Copper in the British Columbia Government Minerals Title Online (MTO) database on 9 December 2021.

3.1 Environmental

QP report author, Mr Andrew Sharp has relied upon Mr Mark Vendrig for the environmental information provided in Section 20, including all information in subsections 20.1, 20.2, 20.3, 20.4, 20.5, and 20.6. Mr Vendrig is an environmental expert and not a qualified person. Mr Sharp received this information from Mr Vendrig on 16 December 2021 as a Microsoft Word document, and it was subsequently reviewed by him and included in Section 20 of the technical report.

Mr Mark Vendrig is currently employed by Consult 5 Inc., an environmental consulting firm with an office at 1055 W Hastings Street, 3rd Floor, Vancouver, B.C., V6E 2E9, Canada, and has held the role of Company Director and Principal Consultant since 2010. Mr Vendrig is a graduate from the University of the Witwatersrand, South Africa (1991) and has been involved or associated with the mining industry since 1989, working in South Africa, USA, Canada, Ghana, Zimbabwe, Senegal, Zambia, Russia, and Canada in operation roles for 10 years and 20 years in environmental consulting. With reference to the Kutcho Copper Project he has 14 years of similar environmental experience in Canada and 16 years of similar environmental experience in planning and operational roles in other countries. Mr Vendrig visited the Kutcho Copper Project site on 12 October 2018 for a period of two days and continues to act as an environmental consultant to Kutcho Copper.

3.2 Taxation

The QP authors of this technical report have relied upon the information supplied by Kutcho Copper for the manner of calculation of taxation that was then used to generate the taxation outcomes presented in Section 22. Further details are outlined below:

- The Project was evaluated on an after-tax basis to provide an estimate of its potential economics. As of the effective date of this technical report, the Project is subject to the following tax regimes:
    - Canadian corporate federal income tax under the Income Tax Act (Canada) and provincial income tax under the Taxation Act (British Columbia) [collectively, “Income Tax”]; and
    - BC Mineral Taxes under the Mineral Tax Act (British Columbia) [“Mineral Tax”].
- The tax model used in the technical report was compiled by Kutcho Copper with assistance from Kenneth Chong, CPA-CA, Tax Partner from Dale Matheson Carr-Hilton Labonte LLP Chartered Professional Accountants (“DMCL”), third-party taxation professionals, to estimate the Income Tax and Mineral Tax applicable to the Project using mine costs, expenses, revenue and other financial information supplied by the Company.
- The calculations are based on the tax regime as of the date of the Feasibility Study and include estimates of Kutcho Copper’s existing unused tax deductions, expenditure pools and losses and expected expenditures, deductions and tax pool balances expected to be incurred from construction of the Project, the commencement of commercial production and throughout operation of the Project and were factored into calculating the Income Tax and Mineral Tax applicable to the Project.

- The tax model is utilized in the economic evaluation of the Project in Section 22 and where a summary of the Project taxation input values and outcomes are provided. The final taxation information advice utilized in the model was supplied by DMCL in a spreadsheet dated 26 October 2021.

4 Description and Location

4.1 Introduction

The Property is located approximately 100 km east of the community of Dease Lake in Northern British Columbia and is shown in Figure 4-1. The site is accessed from Dease Lake via an approximately 119 km long seasonal resource road (known as the "Jade Boulder Road"). The Property is located on national topographic system (NTS) map sheet 104I/1. The geodetic coordinates for the centre of the claim area are 58°12'N and 128°22'W. The Universal Transverse Mercator (UTM) coordinates in the North American Datum 1983 (NAD 83) Zone 9, for the centre of the Main deposit, are approximately 537500E and 6452000N.

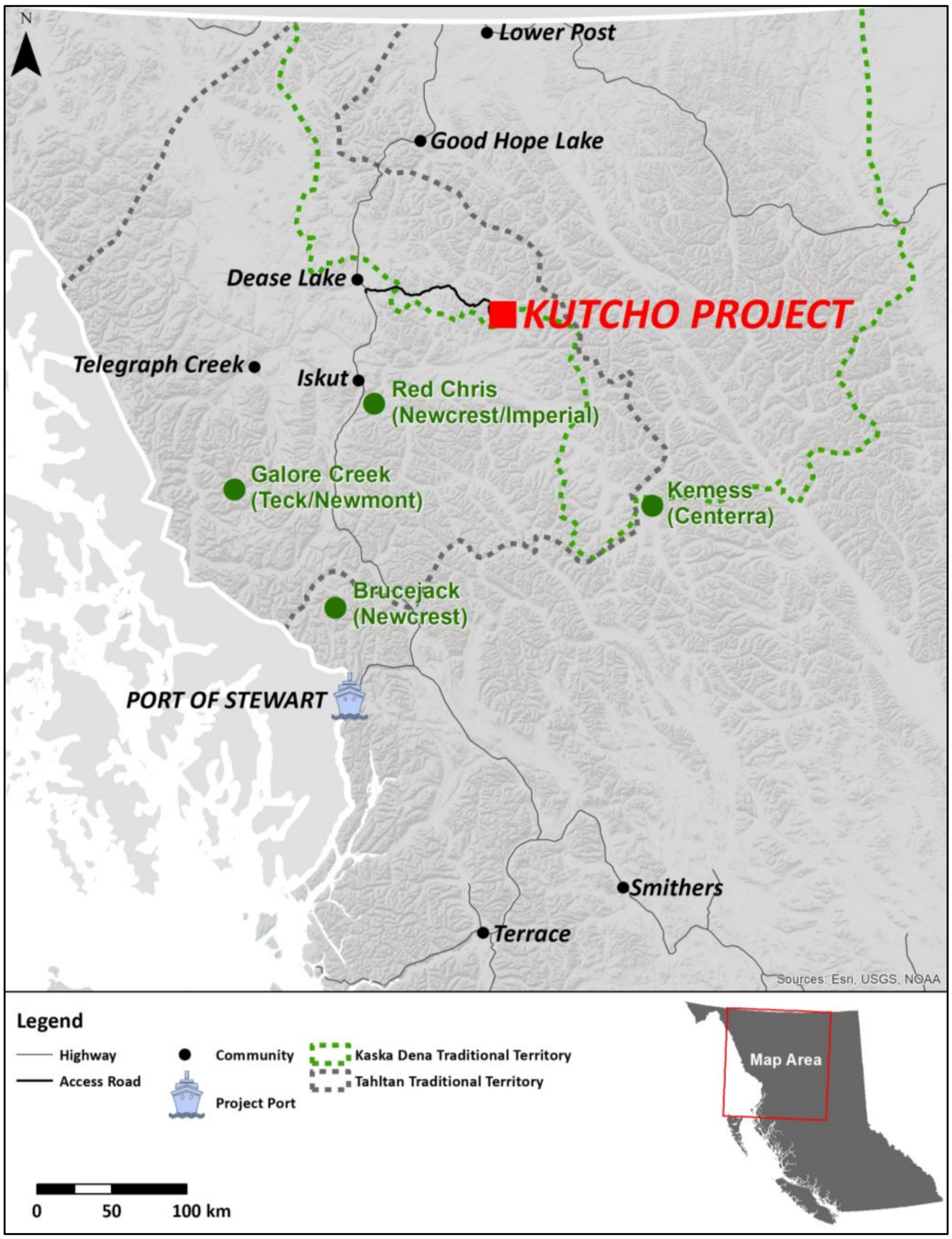


Figure 4-1: Kutcho Project Area

4.2 Land Tenure

The Project area contains 55 mineral claims and one mineral lease covering an area of 24,233.07 ha of Crown land. Mineral tenures are listed in Table 4-1 and shown in Figure 4-2. These claims are registered with the provincial government and entitle Kutcho to the subsurface mineral rights. Some of the land around the access road has placer mining claims which allows users access to surface mineral exploration and exploitation activities. The presence of placer mineral claims would not prevent Kutcho accessing the Project area but agreements with the placer claim holders would be required.

Kutcho Copper Corp. was previously named Desert Star Resources Ltd (Desert Star). Desert Star signed a definitive purchase agreement dated 15 June 2017 with Capstone Mining Corp. (Capstone) to acquire the Kutcho Project. Capstone owned 100% of the Project through its wholly owned subsidiary, Kutcho Copper Corp. Desert Star completed the acquisition in December 2017) and changed its name to Kutcho Copper Corp.

Table 4-1: Kutcho mineral tenure

Tenure no.	Claim name	Claim type	Issue date	Good to date	Area (ha)	Owner	% Ownership
552782		Claim	26/02/2007	11/04/2029	306.87	Kutcho Copper Holdings Inc.	100
552785		Claim	26/02/2007	11/04/2029	409.34	Kutcho Copper Holdings Inc.	100
552792		Claim	26/02/2007	11/04/2029	153.50	Kutcho Copper Holdings Inc.	100
552794		Claim	26/02/2007	11/04/2029	597.09	Kutcho Copper Holdings Inc.	100
552796		Claim	26/02/2007	11/04/2029	494.79	Kutcho Copper Holdings Inc.	100
552805		Claim	26/02/2007	11/04/2029	1074.74	Kutcho Copper Holdings Inc.	100
552809		Claim	26/02/2007	11/04/2029	136.42	Kutcho Copper Holdings Inc.	100
552812		Claim	26/02/2007	11/04/2029	136.37	Kutcho Copper Holdings Inc.	100
552814		Claim	26/02/2007	11/04/2029	357.90	Kutcho Copper Holdings Inc.	100
552816		Claim	26/02/2007	11/04/2029	306.77	Kutcho Copper Holdings Inc.	100
552820		Claim	26/02/2007	11/04/2029	340.92	Kutcho Copper Holdings Inc.	100
552823		Claim	26/02/2007	11/04/2029	921.83	Kutcho Copper Holdings Inc.	100
552911	Pass1	Claim	27/02/2007	11/04/2029	136.41	Kutcho Copper Holdings Inc.	100
552913	Add1	Claim	27/02/2007	11/04/2029	17.05	Kutcho Copper Holdings Inc.	100
552914	Add2	Claim	27/02/2007	11/04/2029	17.06	Kutcho Copper Holdings Inc.	100
556552	Add3	Claim	17/04/2007	11/04/2029	374.88	Kutcho Copper Holdings Inc.	100
556555	Add4	Claim	17/04/2007	11/04/2029	102.29	Kutcho Copper Holdings Inc.	100
569607		Lease	7/11/2007	7/11/2022	1090.00	Kutcho Copper Holdings Inc.	100
585957	Mother 1	Claim	7/06/2008	11/04/2029	426.64	Kutcho Copper Holdings Inc.	100
585958	Mother 2	Claim	7/06/2008	11/04/2029	409.55	Kutcho Copper Holdings Inc.	100
585959	Mother 3	Claim	7/06/2008	11/04/2029	375.29	Kutcho Copper Holdings Inc.	100
586844	Accent 1	Claim	25/06/2008	11/04/2029	426.47	Kutcho Copper Holdings Inc.	100
586846	Accent 2	Claim	25/06/2008	11/04/2029	273.02	Kutcho Copper Holdings Inc.	100
586848	South Fork 1	Claim	25/06/2008	11/04/2029	426.92	Kutcho Copper Holdings Inc.	100
586849	South Fork 2	Claim	25/06/2008	11/04/2029	426.88	Kutcho Copper Holdings Inc.	100
586850	South Fork 3	Claim	25/06/2008	11/04/2029	426.83	Kutcho Copper Holdings Inc.	100
586851	South Fork 4	Claim	25/06/2008	11/04/2029	426.86	Kutcho Copper Holdings Inc.	100
586852	Trondhjemite 1	Claim	25/06/2008	11/04/2029	426.65	Kutcho Copper Holdings Inc.	100
586854	Trondhjemite 2	Claim	25/06/2008	11/04/2029	426.69	Kutcho Copper Holdings Inc.	100
586855	Trondhjemite 3	Claim	25/06/2008	11/04/2029	426.55	Kutcho Copper Holdings Inc.	100
848105	Accent 3	Claim	4/03/2011	11/04/2029	238.92	Kutcho Copper Holdings Inc.	100

Tenure no.	Claim name	Claim type	Issue date	Good to date	Area (ha)	Owner	% Ownership
848106	Accent 4	Claim	4/03/2011	11/04/2029	153.51	Kutcho Copper Holdings Inc.	100
852142	Pyramid Peak	Claim	20/04/2011	11/04/2029	426.85	Kutcho Copper Holdings Inc.	100
852162	Tucho 1	Claim	20/04/2011	11/04/2029	426.68	Kutcho Copper Holdings Inc.	100
852163	Tucho 2	Claim	21/04/2011	11/04/2029	426.84	Kutcho Copper Holdings Inc.	100
852164	Tucho 3	Claim	21/04/2011	11/04/2029	426.91	Kutcho Copper Holdings Inc.	100
852165	The Sphinx	Claim	21/04/2011	11/04/2029	426.97	Kutcho Copper Holdings Inc.	100
852166	Nile River	Claim	21/04/2011	11/04/2029	222.07	Kutcho Copper Holdings Inc.	100
852167	South Road	Claim	21/04/2011	11/04/2029	187.56	Kutcho Copper Holdings Inc.	100
852344	Far East 1	Claim	23/04/2011	11/04/2029	426.82	Kutcho Copper Holdings Inc.	100
852345	Far East 3	Claim	23/04/2011	11/04/2029	85.31	Kutcho Copper Holdings Inc.	100
852346	Far East 2	Claim	23/04/2011	11/04/2029	426.89	Kutcho Copper Holdings Inc.	100
852347	Campview 1	Claim	23/04/2011	11/04/2029	306.97	Kutcho Copper Holdings Inc.	100
852348	Campview 2	Claim	23/04/2011	11/04/2029	340.97	Kutcho Copper Holdings Inc.	100
854561	Kutcho Fault	Claim	15/05/2011	11/04/2029	409.07	Kutcho Copper Holdings Inc.	100
858667	Accent 5	Claim	24/06/2011	11/04/2029	153.62	Kutcho Copper Holdings Inc.	100
861767	Accent 6	Claim	29/06/2011	11/04/2029	102.36	Kutcho Copper Holdings Inc.	100
1061247	K1	Claim	16/06/2018	16/06/2029	136.52	Kutcho Copper Holdings Inc.	100
1061249	K2	Claim	16/06/2018	16/06/2029	341.46	Kutcho Copper Holdings Inc.	100
1061251	Accent 6	Claim	16/06/2018	16/06/2029	289.96	Kutcho Copper Holdings Inc.	100
1061254	Far East 3	Claim	16/06/2018	16/06/2029	734.46	Kutcho Copper Holdings Inc.	100
1061255	Sphinx 2	Claim	16/06/2018	16/06/2029	1247.40	Kutcho Copper Holdings Inc.	100
1061257	South Fork 5	Claim	16/06/2018	16/06/2029	1127.28	Kutcho Copper Holdings Inc.	100
1061259	South Fork 6	Claim	16/06/2018	16/06/2029	1451.41	Kutcho Copper Holdings Inc.	100
1061263	Accent 10	Claim	16/06/2018	16/06/2029	1502.42	Kutcho Copper Holdings Inc.	100
1061264	Accent 11	Claim	16/06/2018	16/06/2029	341.29	Kutcho Copper Holdings Inc.	100

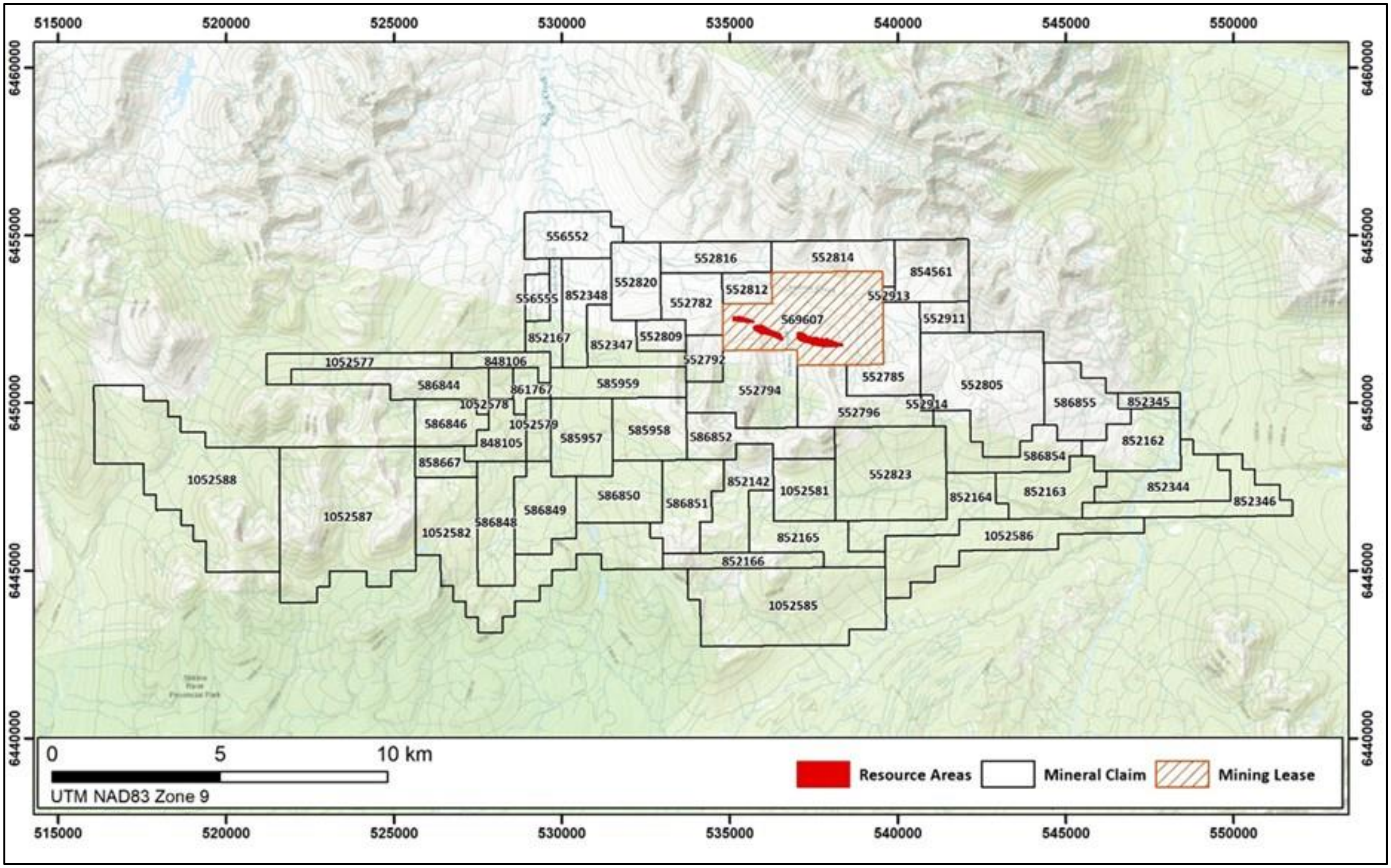


Figure 4-2: Kutcho Property Mineral Tenure Map

4.3 Tenure Obligations

Under Section 8(4) of the Mineral Tenure Act Regulation (MTAR), the value of exploration and development required to maintain a mineral claim for one year is C$5.00 per hectare during each of the first and second anniversary years, C$10.00 per hectare for each of the third and fourth anniversary years, C$15.00 per hectare for each of the fifth and sixth anniversary years and C$20.00 per hectare for subsequent anniversary years. Reporting of physical work is required on the claims to demonstrate activity.

Section 10 of the Mineral Tenure Act Regulation describes registering payment instead of exploration and development. The required payment to maintain a claim for one year is double the value of exploration and development that would be required to maintain the claim under section 8(4) or (5) for the anniversary year. This is C$10.00 per hectare for anniversary years 1 and 2, C$20.00 per hectare for anniversary years 3 and 4, C$30.00 per hectare for anniversary years 5 and 6; and C$40.00 per hectare for subsequent anniversary years.

A mineral claim is an exploration and development tenure, and a recorded holder may convert a claim to a lease to carry out production mining. Mineral claims provide access to subsurface rights. The access road runs through placer claims, but the Mining Right of Way Act provides for the right of a recorded holder to use access roads owned by a person or to use existing roads on Crown land or private land for the purpose of gaining access to a mineral title. These rights are not exclusive and may require the consent of other land users, tenure owners and claim holders.

At the time of writing all mineral claims and leases are in good standing.

4.4 Surface Rights, Access, and Land Use

Kutcho does not own the surface rights to the land. Surface rights are held by the Crown. The project site does not overlap with any mineral tenures not held by KCC.

The Project does not overlap with any private, provincial or federal titled or surveyed lands. Additionally, the Project does not overlap with any provincial or national parks, protected areas or historic sites. There are no Land Act tenures in the vicinity of the mine site and airstrip.

The Project is located within the Stikine Regional District (unincorporated). No zoning bylaws apply to lands required for the Project. The mine site, and a portion of the access road are located within the Dease-Liard Sustainable Resource Management Plan (SRMP). A section of the access road is located within the Tuya resource management zone of the Cassiar Iskut-Stikine Land Resource Management Plan. The access road overlaps with 28 mineral claims, 33 placer claims and one placer lease. Operating jade mines in the vicinity of the Project include Wolverine (Cassiar Jade Contracting Inc.), Letain (Cassiar Jade Contracting Inc.), Kutcho Creek Jade (Continental Jade Ltd), Provencher (Glenpark Enterprises Ltd), Dean Kutcho (Cassiar Jade Contracting Inc.), Jade Valley (United Oriental Mining Ltd) and Polar Jade (Jade West Group) (BC MEMPR 2018). The Mining Right of Way Act would apply in these cases to prevent access restriction to the Kutcho Project.

A section of the access road close to its junction with Highway 37 overlaps with range tenure RAN074881 held by the Iskut First Nation.

The mine site, and majority of the access road are located in guide outfitter area 601064. The mine site and access road are located in trapline TR0752T004. The access road passes through traplines TR0752T004, TR0619T005 and TR0622T011. Trapping occurs during the winter months when titleholders are permitted to lay trap boxes for fur-bearing species. A section of the access road overlaps a recreation reserve (map reserve designated under section 16 of the Land Act, Crown Land File #8000849) covering Wheaton Lakes along the Turnagain River. The mine site and majority of the access road is located within range tenure RAN 075525 held by BC Safaris Ltd.

There is no forestry or agriculture in the mining or the access road footprint. There are no agricultural land reserves in the vicinity of the Project. The Project does not overlap with any active or pending forestry tenures or forestry recreation sites. There are no water licences in the vicinity of the Project.

Kaska and Tahltan do make use of the land and resources for traditional purposes. Kaska sacred sites are within the project area but not near to or directly affected by the project.

4.5 Royalties

*4.5.1 **Royal Gold***

Royal Gold Inc. holds a back-In right to the area marked in blue in Figure 4-3. Royal Gold is entitled to a royalty of 2% of net smelter returns on this portion of the Kutcho Project if it elects to “not back-in” to the Project (SRK, 2008). The 2% Royal Gold royalty is applicable to 35% of the current Mineral Resource if Royal Gold elects not to “back-in” to the Project.

Following notice by Kutcho Copper that it has completed a Feasibility Study on the Project, Royal Gold will have 120 days to elect to “not back-in” for a 50% interest in the portion of the Kutcho Project. Royal Gold acquired this royalty interest effective 1 October 2008, as part of the acquisition of a royalty portfolio from Barrick Gold Corporation.

The back-in arrangement for the 50% interest stipulates that Royal Gold will pay to Kutcho Copper in aggregate an amount equal to 150% of Kutcho Copper’s eligible development expenditures. On the first anniversary of the delivery of the Back-In-Notice, a further amount equal to 150% of Kutcho Copper’s eligible development expenditures will be payable by Royal Gold. This applies only to that portion of the Property on which Barrick and AMI Resources previously held an interest (SRK 2008). Likewise, Royal Gold’s capital contribution and profit share is pro-rated on its portion of the tenure. A map showing the claims on which Royal Gold holds a royalty is given as Figure 4-3.

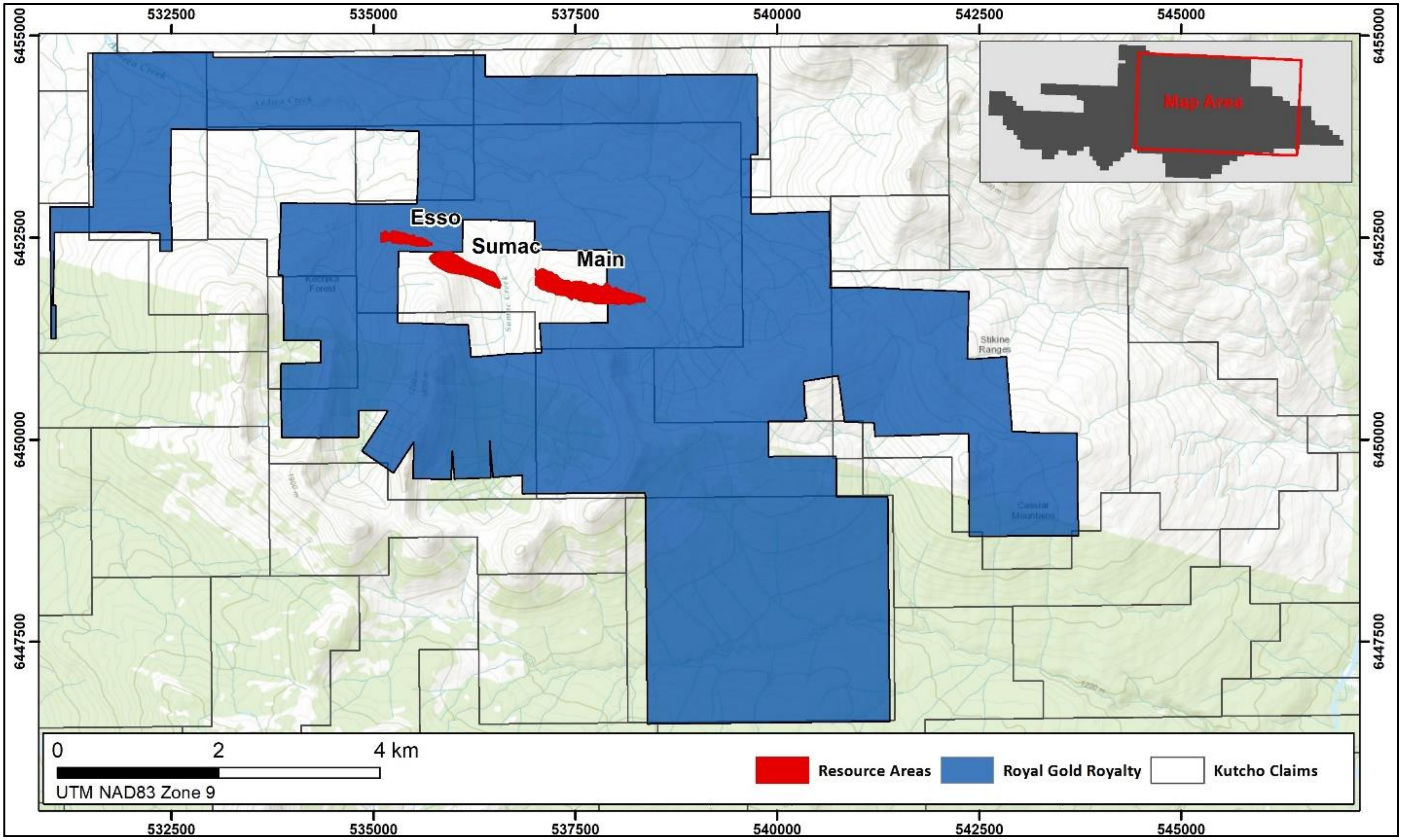


Figure 4-3: Kutcho Property Royal Gold Royalty Tenure

4.5.2 Sumac

The royalty rights of Sumac Mines Inc. were extinguished by an agreement signed between Sumac Mines Inc. and Kutcho Copper Corp. in 2021 and reported in a press release dated 6 July 2021. Certain payment terms within the agreement are contemplated to be completed after the publication of this report. However, for the purposes of this study, it was assumed that the Sumac royalty was extinguished.

4.5.3 Wheaton Precious Metals Purchase Agreement

In August 2017, Kutcho Copper Corp. and Wheaton Precious Metals Corp. (Wheaton) entered into an Early Deposit Precious Metals Purchase Agreement (PMPA) whereby Wheaton will be entitled to purchase 100% of the silver and gold production from the Project until 5.6 Moz of silver and 51,000 ounces of gold have been delivered, at which point the stream will decrease to 66.67% of the silver and gold production for the life of the mine. In consideration for the sale and delivery of refined gold and refined silver pursuant to the terms of the PMPA, Wheaton will pay to Kutcho Copper a deposit in cash against gold and silver purchases of US$65 million.

Wheaton will make ongoing production payments to Kutcho equal to 20% of the applicable spot prices of silver and gold delivered. An additional payment of up to US$20 million will be payable should the processing capacity increase to 4,500 tpd or more within five years of attaining commercial production.

Under the terms of the proposed PMPA, Kutcho Copper was advanced US$7 million as an Early Deposit to fund the Feasibility Study. The balance of the US$65 million is payable during construction of the Kutcho Project, subject to certain conditions being met. Wheaton will have the option to terminate the PMPA at any time after the Early Deposit payment and before delivery of the Feasibility Study documentation, or at any time on or after 14 December 2019 if the Feasibility Study documentation has not been delivered to Wheaton by such date, or at any time within 90 days after the delivery of the Feasibility Study documentation, or at any time there is a default.

The Project economics and financial models presented in the Feasibility Study do not include the application of the Wheaton PMPA terms although guidance is given for the potential impact.

4.6 Permits

The permits required for the development and operation of the Project, to the extent known, are set out in Section 20 – Environmental Studies, Permitting and Social or Community Impact.

Kutcho Copper currently holds a MYAB permit for exploration (Permit MX-1-612). This permit is valid for five years; it was issued in 2018 and will expire in March 2023. An annual MYAB permit notification of work is being prepared for review by First Nations and the provincial regulator to accommodate the 2022 exploration.

4.7 Risk and Liabilities

4.7.1 Environmental Liabilities

Kutcho does not own or operate the airstrip at the site and is not a current user of the Jade Boulder Road. Therefore, it has no liabilities associated with either. Kutcho does, however, operate a small camp at the airstrip, makes use of site roads, and has several exploration sites that would need to be remediated if the site were to be closed at this time. The MYAB permit issued for exploration over the site specifies what facilities are allowed and what activities Kutcho is permitted to undertake. It makes financial provision for closure and reclamation of the exploration facilities, sites, and roads. This bond is currently fully funded to the value of C$159,600. Additional funding of C$25,000 is required to remove structures left in 2021 exploration drilling and historical geochemical test work cribs.

Environmental liabilities that may be incurred by the Project construction and operation are discussed in Section 20.7.1.4.

4.7.2 Permits

The Company currently has the required permits to conduct exploration work on the Property but will need to proceed through additional permitting process to develop the Project as per the requirements of the Province of British Columbia the permitting process is outlined in Section 20.3.

4.7.3 Risks to the Property

No other factors or risks are known to exist that may affect access, title or the right or ability to perform the work on this Property as set out in this Feasibility Study. The continued development of the Project is, however, dependent on the approval of the various permits set out in Section 20.3 of this report.

5 Accessibility, Climate, Local Resources, Infrastructure and Physiography

5.1 Accessibility

The Kutcho property is located approximately 100 km east of Dease Lake, BC (Figure 4-1). Dease Lake is accessed by paved and well-maintained road from Smithers, about 600 km to the south, and Terrace, about 580 km to the south, via Highway 16 and Highway 37. Dease Lake is approximately 400 km from the Port of Stewart. From Dease Lake, an unsurfaced and unmaintained seasonal access road (Figure 18-3) runs to the property. Typically, this road is only suitable for use during the summer. In summer, 4x4 trucks can use the road but it is easiest with vehicles that additionally have low ground bearing pressure and high clearance.

Rapid and reliable access to the Project area is possible using fixed-wing aircraft and helicopter from Smithers, Dease Lake, and Whitehorse, landing at the 900 m long gravel airstrip located at the junction of Kutcho and Andrea creeks (Figure 18-2). The deposit area of the property is connected to the airstrip by a 10 km road. Currently this road has had culverts removed and is only passable by four-wheel drive trucks with good ground clearance.

5.2 Climate and Physiography

The property is located within the Cassiar Mountains, just to the north of the continental divide between the Arctic and Pacific watersheds. The area is moderately rugged with elevations ranging from 1250 to 2200 masl. Most of the area is alpine with tree line at approximately 1500 masl.

The proposed mine site and airstrip are situated within the spruce-willow-birch bio-geoclimatic zone, a subalpine zone characterized by white spruce and subalpine fir (Figure 5-1). Shrub-dominated ecosystems ranging from swamps and fens to dry colluvial scrub are well represented in the zone. At upper elevations, the landscape of this zone is characterized by scrub/parkland dominated by tall scrub birch and willows. Wetland vegetation in the zone include white spruce and tall willow swamps, sedge fens, and sedge marshes.

Figure 5-1: Area of the Proposed Mine Site with the Surface Expression of the Main Deposit in the Middle Ground

A total of 2,677 ha of wetlands, approximately 10% of the local study area, have been identified and mapped in the vicinity of the Project. A total of 22 wetland ecosystems have been identified. Most of the wetland classes identified are fens and alpine seepage sites.

The climate is characterized by prevailing westerly winds, rain-shadow effects, and precipitation That is highest in July and lowest in April. Winters are long and cold while the summer growing season is cool and short, with a mean annual temperature range between -0.7° and -3°C. The warmest and coolest months on average are July (10.0°C) and January (-12.6°C). Measured monthly average air temperatures were as low as 18.6°C in December 2008 and as high as 14.0°C in August 2008. The extreme minimum hourly temperature of 41.0°C was recorded in January 2008, while the extreme maximum temperature of 28.2°C was recorded in July 2009. With sub-zero temperatures from October to May, precipitation accumulates predominantly as snow, where it remains until spring. Temperature inversions are known to occur at the site.

Some of the valleys contain permafrost, especially above 1200 to 1400 masl. There are no glaciers or permanent ice fields within the proposed mine site area.

The average annual precipitation estimates for the proposed mine site are between 771 mm and 806 mm. Mean annual precipitation is estimated to be 800 mm, and both snowfall and rainfall increase locally with elevation at the Project site due to orographic cooling. The maximum snow depths recorded at the Deposit Station were 161 cm on 6 February 2009, and 160 cm on 11 April 2008. Snowmelt at the Project site typically begins in late April or early May and lasts for four to six weeks. Annual open water or lake evaporation was estimated to be in the range of 288–319 mm for the Andrea Creek catchment.

Between October 2008 and September 2009, the average wind speed observed at the Deposit Station was 2.1 m/s with maximum instantaneous wind gusts of approximately 18 m/s (65 kmph).

5.3 Local Resources

Dease Lake is a community of about 335 people and has basic services such as an airstrip, medical clinic, school, college extension campus, grocery store, and hotels. The Dease Lake community and surrounding area offers a pool of potential employees.

5.4 Infrastructure

The present site infrastructure includes the 900 m long gravel airstrip located at the junction of Kutcho and Andrea creeks, a 30-person exploration camp (located at the airport), core storage and logging shacks and a single lane four-wheel drive access gravel road connecting the site, airstrip and Dease Lake.

The proposed mining complex is planned to be located within an area enclosed by the Andrea, Sumac, and Playboy Creeks; refer to Section 18, Figure 18-1 for more information regarding site layout. The Project is located on Crown land and there are no known competing surface rights beyond securing appropriate permits to conduct exploration and development work. Power will be generated at site with liquefied natural gas (LNG) generators. Water sources for the Project have been defined and are laid out in Section 16.3 and will include contact-water run-off collection, wells for potable water and de-watering from mine operations. These sources will ensure that the Project has a positive water balance, and no water will be drawn from creeks or other surface water sources.

The Project site plan displays the planned location of the Project access road, tailings storage facility, waste rock stockpiles, mine locations, mineral processing facility, and other ancillary structures. The site plan is presented in Section 18 as Figure 18-1.

6 History

6.1 Exploration History

The history section is taken almost entirely from the JDS (2017) NI 43-101 Technical Report that supported the 2017 Pre-Feasibility Study.

The Property has had multiple owners through Its exploration history, with ownership by Exxon subsidiary, Imperial Oil Ltd, Esso Minerals Ltd (EML) which was later acquired by Homestake Canada Ltd (Homestake), Sumac Mines Ltd (SML) (a subsidiary of Sumitomo Metal Mining Co. Ltd), Homestake, American Reserve Mining Corporation (ARMC), Cominco Ltd (Cominco), Barrick, Western Keltic Mines (WKM), Sherwood Copper Corp., Capstone Mining Corp., and Kutcho Copper Corp. through the years.

6.1.1 1968 to 1988 – Imperial Oil, Esso Minerals and Sumac Mines

Mineralization on the Property was first discovered in 1968 by a joint venture exploration program operated by Imperial Oil Ltd.

Mineralization was identified by follow up prospecting anomalous to stream sediment samples collected during a regional drainage survey. Twenty claims were staked by W. Melnyk directly over the as-yet undiscovered Main Kutcho sulphide deposit. These claims were allowed to lapse when the other partners in the joint venture declined to fund further exploration.

After the statutes of the joint venture agreement expired, Imperial Oil returned to re-stake the area in 1972. However, SML had conducted stream sediment sampling earlier that year, and in response to anomalous samples R. Britten staked eight "two-post" claims along the anomalous stream and eight more claims along the geological strike direction, resulting in the cruciform claim outline overlying the western part of the main Kutcho sulphide deposit. Imperial Oil staked a much larger area surrounding the SML claims.

In 1973, the results of SML and EML exploration work prompted additional staking which resulted in claim boundaries which remained largely unchanged until today. Geological mapping, geochemical sampling, and airborne EM geophysical surveys defined drill targets for diamond drilling that commenced in 1974.

By 1982 approximately 60,000 m had been drilled by both companies, defining three sulphide lenses (Main, Sumac, and Esso). During this time EML also drilled additional exploration targets in other parts of the Property with moderate success. Environmental, metallurgical, and engineering studies were initiated by both groups in 1980.

A partnership agreement on engineering and development work was signed by EML and SML in 1983, made retroactive to 1981, the year SML began work driving an adit to collect a 100-tonne bulk sample. The agreement was essentially a 50/50 joint venture for development work and culminated in a Prefeasibility Study by Wright Engineers Limited in 1985. This study indicated an 11% internal rate of return (IRR), using a copper price of US$0.95/lb. Based on risk factors involved and long-term price projections for copper below the US$0.95/lb level, the companies put the Project on hold pending further exploration results. A limited amount of exploration work was done on EML's claims to the south of the main mineralized trend between 1985 and 1988; however, this work and the numerous geophysical surveys that had been undertaken indicated limited potential for additional open pit mineralization.

6.1.2 1989 to 2003 – Homestake, ARMC

In 1989, EML sold most of its mining assets to Homestake. In 1990, Homestake optioned the Kutcho property to ARMC, who funded a C$1.1 million exploration program (Homestake remained the operator) which included 7,031 m of drilling in 28 holes, mostly in outlying target areas, thereby earning a 20% interest. Exploration successfully confirmed the presence of extensive areas of favourable geology and alteration indicative of hydrothermal activity, but mineralization was not identified. One such exploration target was found 10 km southwest of the Kutcho deposit, a narrow zone of massive pyrite with a strike length in excess of 5 km which was intersected in four widely spaced drillholes but was barren of base or precious metals. ARMC carried out engineering studies but did no further exploration work, relinquishing the option in 1992 while retaining a 20% interest in Homestake's property.

Homestake optioned its Property to Cominco in 1992. Cominco carried out deep penetration EM geophysical surveys (UTEM) over the Esso zone targeting additional conductors along the Kutcho trend. Due to extensive cover of conductive argillaceous units in the hanging wall, the UTEM system was unable to detect the Esso deposit or other conductors at depth, leading Cominco to drop their option.

Homestake was purchased by Barrick in 2003.

*6.1.3 **Western Extension Exploration – Noranda and Atna***

Extensions of the Kutcho stratigraphy to the west were staked and worked by various companies in the past. Shortly after the discovery of the Kutcho deposits, Noranda staked the Kutcho Formation to the west of Kutcho Creek. Noranda conducted geophysical surveys and carried out a small drilling program.

The claims were allowed to lapse and were re-staked in 1995 by Gary Belik. Mr Belik carried out a detailed mapping program and optioned the claims to Atna Resources Ltd (Atna) in 1997. Atna conducted UTEM geophysical surveys and an extensive drilling program. Results of Atna's work were mixed, and although no additional deposits were discovered, significant weak to moderately mineralized alteration zones were intersected. Structural complexity and lack of clear geophysical targets prevented additional work and the option was terminated.

*6.1.4 **2004 to 2006 – Western Keltic Mines***

Negotiations by WKM to purchase the property from Barrick and Sumitomo were initiated in 2003 and concluded in early 2004, and the property was placed into Kutcho Copper Corp., WKM's wholly owned subsidiary. WKM carried out diamond drilling within the Main and Esso deposits during 2004 to confirm historical results and obtain material for metallurgical studies. A second round of drilling by WKM in 2005 tested the Main deposit's potential for up-dip and down-dip extensions, as well as western extensions to the Esso deposit. The Sumac deposit was also drilled in 2005 to test for higher grade zones. A third round of drilling in 2006 focused on infill drilling within the five-year pit area of the Main deposit. The Kutcho property was entered into the Mine Development Review Process in 2006 and environmental studies were initiated to provide baseline data for provincial and federal EA reviews (Wardrop, 2007).

*6.1.5 **2008 to 2011 – Sherwood Copper Corp., Capstone Mining, and Kutcho Copper Corp.***

In February 2008, Sherwood acquired 93% ownership in WKM and renamed it Kutcho Copper Corp. On 27 May 2008, Sherwood acquired 100% ownership of Kutcho Copper Corp. On 27 November 2008, Sherwood amalgamated with Capstone Mining under a plan of arrangement and Kutcho Copper Corp. became a wholly owned subsidiary of Capstone.

Kutcho Copper Corp. embarked upon a program of diamond drilling of 81 holes. Three holes were abandoned due to technical issues and 78 holes were completed for a total of 9,905 m of HQ drill core (63.5 mm). The program was designed to infill previous drilling, reducing drillhole spacing and increasing sample density, and to provide material for further metallurgical and environmental testing. The drill program refined the deposit geometry and improved confidence and Mineral Resource classification for the 2008 estimate.

Kutcho Copper completed a second diamond drill program in 2010. On 3 July 2010, a program totalling 17,970 m of infill and step-out drilling commenced on Esso deposit which led to significant changes in the Mineral Resource estimate. The 2010 holes were designed to infill previous drilling at Esso, reducing drillhole spacing and increasing sample density, and to provide material for further metallurgical and environmental testing. Previous drilling had defined the gross limits and overall geometry of the mineralized zone at Esso and as expected, the new drilling did not result in a material change to these limits or the geometry of the resource model. The 2010 program better defined higher-grade lenses within the deposit, eliminated an internal gap in the resource model at the west end of Esso deposit and provided more confidence to allow an upgrade in resource classification categories.

The 2011 program at the Property consisted of an airborne electromagnetic (EM) survey and subsequent exploration drilling. The EM survey used Geotech Ltd's proprietary time-domain electromagnetic VTEM system. The VTEM survey consisted of 1,649.4 line-km (plus tie-lines) covering 147.2 km^2. The survey flight lines were oriented at 004° azimuth, perpendicular to the strike of the host rock strata in the deposit area.

The geophysical program was followed by a drill program that tested nine high-priority VTEM targets. Drilling totalled 4,227 m of HQ size core in 20 holes (including two short, abandoned holes). The 2011 exploration program generated the following conclusions:

- The Sumac deposit extends further south, east, and up-dip than previously recognized.
- Anomalous VTEM responses were observed over the east end of the Sumac deposit and along the up-dip edge of the Esso stratigraphic horizon. This represents deeper levels of penetration and higher levels of sensitivity than previous airborne EM systems used at Kutcho.
- Graphitic mudstone horizons are commonly close to pyritic tuff horizons, and the mudstone itself may host exhalative sulphides.

6.1.6 Desert Star Resources

In 2017, Desert Star Resources Ltd acquired 100% of Capstone's wholly owned subsidiary, Kutcho Copper Corp. and Desert Star changed its name to Kutcho Copper Corp. A NI 43-101 Technical Report supporting a Pre-Feasibility Study prepared by JDS Associates was issued with an effective date of 17 June 2017.

6.2 Historical Mineral Resources and Reserves

Various historical resources and reserve estimates have been completed by previous operators of the Project within the Main (also known as the Kutcho zone), Sumac and Esso (also known as the Esso West zone) zones. These historical estimates have been superseded by the current mineral resource and mineral reserve estimates disclosed in Section 14 and 15 of this Technical Report.

The first known mineral resources or mineral reserves were reported in 1986 by Imperial Oil. These were listed as reserves for the three zones, Kutcho, Sumac, and Esso West. An "underground mineable reserve" was reported in 1991 for the Kutcho and Esso West zones (George Cross Newsletter No. 54, 1991). In 1996, a "mineable reserve" of the Kutcho zone was reported (BC MEMP, 1996). The 1986, 1991, and 1996 historical estimates were reported prior to the introduction of NI 43-101 and CIM definition standards.

Subsequent historical resource and reserve estimates were performed on behalf of WKM, Sherwood Copper, Capstone and Desert Star following the introduction of NI 43-101 and CIM and were compliant with these disclosure standards at the time. However, these historical estimates are superseded by the current mineral resource and mineral reserve estimates disclosed in Section 14 and 15 of this Technical Report.

7 Geological Setting and Mineralization

7.1 Regional Setting

The Kutcho property lies within the King Salmon Allochthon (KSA), of the Cache Creek terrane. The KSA is a narrow belt of Permo-Triassic island arc volcanic and sedimentary rocks, between two northerly-dipping thrust faults: the Nahlin fault to the north, and the King Salmon fault to the south (Figure 7-1). Penetrative foliation and axial planes of major folds are parallel to these east-west trending bounding faults.

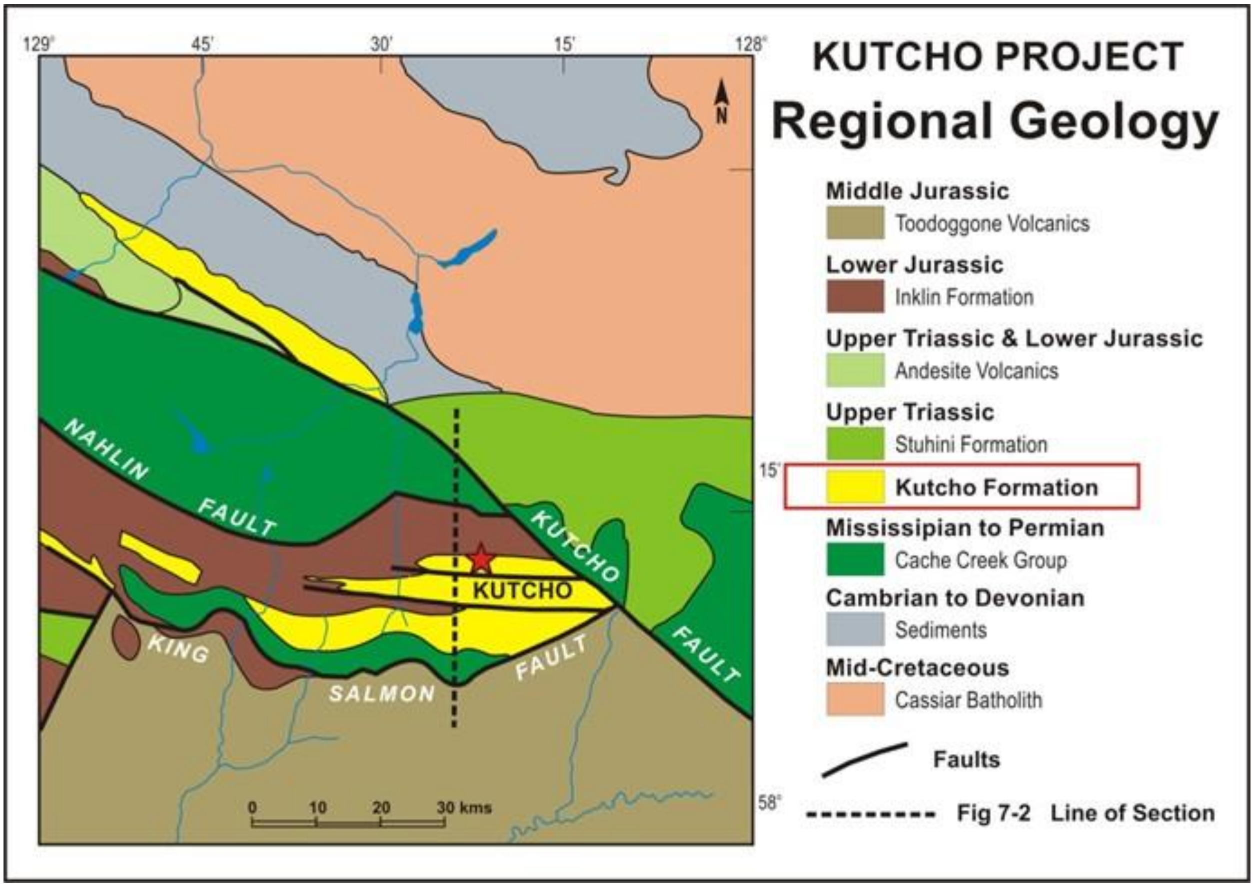


Figure 7-1: Regional Geology Setting
(Source: Wardrop, 2007)

The Lower Triassic bimodal volcanic and volcaniclastic rocks of the Kutcho assemblage are interpreted to have formed in a juvenile oceanic arc setting (Thorstad, 1983; Childe and Thompson, 1996).

The belt of volcanic rocks is thickest in the area where it hosts the VMS deposits, partly due to primary deposition, but also to stratigraphic repetition by folding and possibly thrusting. The KSA is terminated to the east (near the eastern edge of the property) by the Kutcho strike-slip fault (Mansy and Gabrielse, 1978; Gabrielse, 1978) but extends to the west for hundreds of kilometres. However, the host Kutcho Formation rocks thin to the west, and do not occur or are rarely exposed 10 km to the west of Kutcho Creek (Figure 7-1).

Stratigraphy of the KSA consists primarily of the Kutcho Formation, which is overlain by the limestone of the upper Triassic Sinwa Formation, which in turn is overlain by sediments (predominately argillite) of the Lower Jurassic Inklin Formation. Major folds are delineated by the Sinwa limestone and, where the Sinwa is absent, by the contact between the Kutcho and Inklin Formations (Figure 7-1 and Figure 7-2).

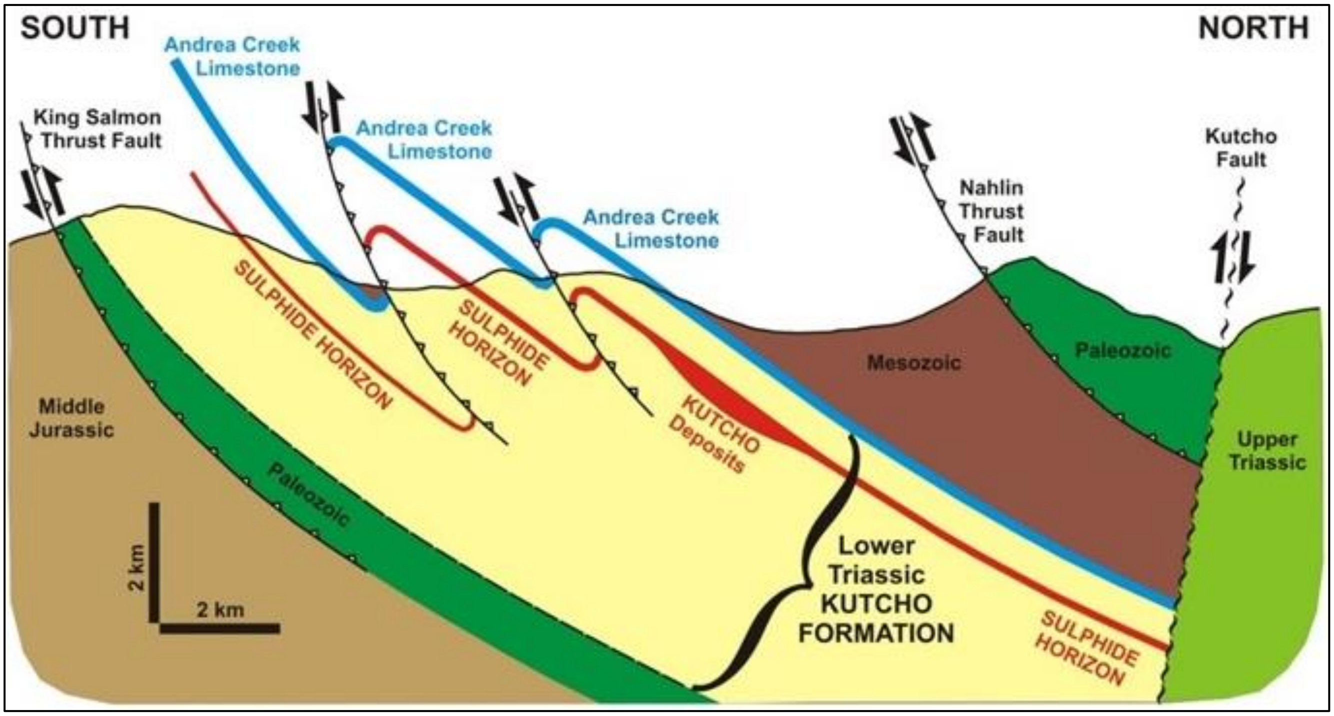


Figure 7-2: Kutcho Area Schematic Cross-Section
(Source: Wardrop, 2007)

7.2 Stratigraphy

The stratigraphy of the Kutcho property was described by CIM (1986), Thorstad (1983), and Holbek (1985), so it will only be briefly described herein. Stratigraphy is best understood in the upper part of the Kutcho Formation, where units are better exposed and drill information is available. The footwall stratigraphy (particularly away from the deposit area) is not well understood.

The lowest rocks in the section are exposed on the southern ends of Imperial and Sumac ridges and include interbedded basalts, basaltic tuffs and wackes, rhyolitic lapilli tuffs, and possible trondhjemite. The mafic rocks are fine to very fine-grained, chloritic, equigranular to weakly porphyritic, and are commonly referred to as greenstone in the field. The lapilli tuffs are pale grey and siliceous, and commonly contain very fine quartz phenocrysts and lenticular fragments from 0.5 cm to 3.0 cm in length. Textures can only be seen on weathered (but lichen-free) surfaces.

The trondhjemite unit is somewhat unclear. It is described by Pearson and Pantaleyev (1975) and CIM (1986) as a fine-grained, equigranular, plagioclase-rich unit, but it is very similar to some of the tuffaceous units as well. A weak but pervasive carbonate-chlorite-pyrite or propylitic alteration of this unit is subtle but discernible. Stratigraphy is shown in Figure 7-3.

Rocks overlying the greenstone-lapilli tuff package are termed the “ore sequence”, and consist of lapilli tuffs, crystal-lithic tuffs, and quartz and quartz-feldspar crystal tuffs. Away from the deposit area, these units tend to be thin, interbedded, and variably but weakly altered. Fine quartz-crystal ash tuff with silica rich laminations and rare thin zones of ferroan dolomite typically marks the distal exhalative zone. The sulphide zones occur at (or near to) the contact between footwall lapilli tuff and hanging wall quartz crystal tuff. In general, both lapilli fragments and phenocrysts are much coarser grained in the vicinity of the deposits, becoming progressively finer grained to the south and west. The quartz-feldspar crystal tuff is quartz-rich near the deposits, becoming more feldspar-rich to the south.

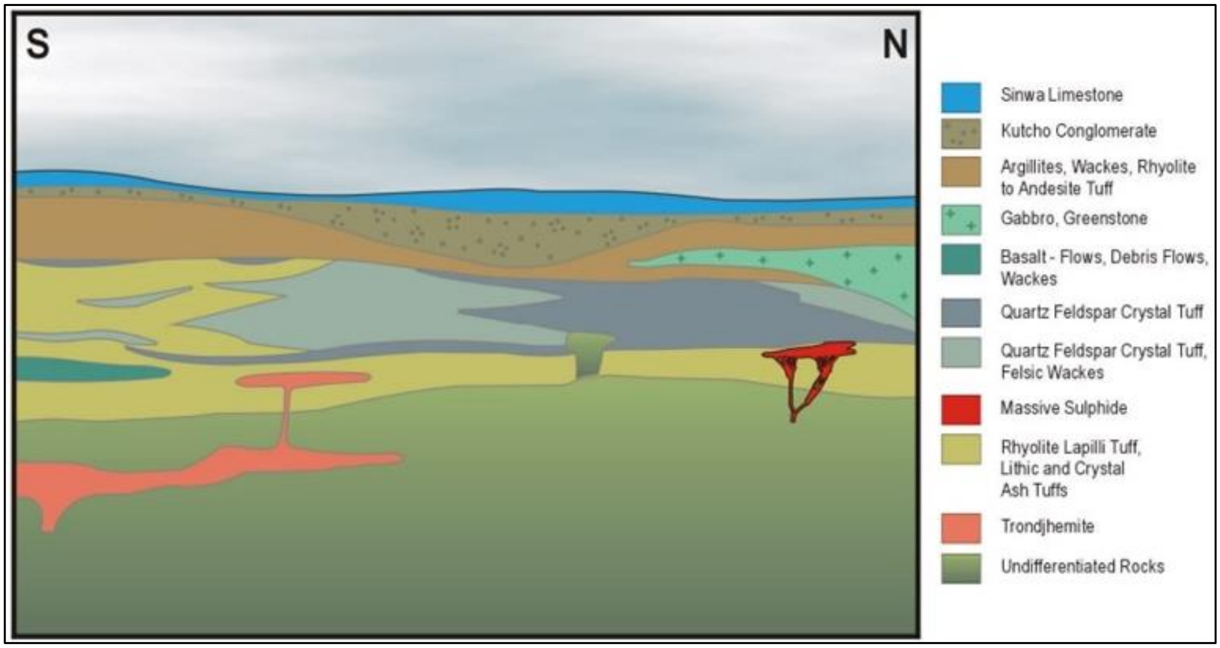


Figure 7-3: Property Stratigraphy Schematic (~ 10x vertical exaggeration)
(Source: Wardrop, 2007)

A large zone of feldspar crystal tuff with almost no free quartz occurs a few hundred metres south of the sulphide zones, and whether this unit is footwall, hanging wall, or a facies equivalent to the quartz-feldspar crystal tuff is as yet indeterminate. An interesting feature is the occurrence of a coarse breccia texture within the quartz-feldspar crystal tuff immediately over the sulphide zones. The breccia fragments are typically sub-rounded, from 2 cm to 30 cm in size, and are identical to crystal tuff matrix except for an increase in the amount of epidote from 1% to 2% to closer to 10%. This was interpreted to be a debris flow of semi-consolidated crystal tuff, shed from a flow dome complex and trapped in the graben or half-graben structure which hosts the sulphide lenses.

Rocks between the ore sequence and the overlying conglomerate unit are referred to as the Tuff-Argillite Unit (TAU) and consist of greywackes and argillite intruded by gabbroic to basaltic sills and dykes. In the area of the deposit, the gabbroic units are commonly coarse-grained and are commonly referred to as metagabbro. Higher in the section, to the east and west of the Kutcho deposits, this mafic unit becomes much finer grained, and an intrusive origin is not so clearly identified.

The amount of argillite increases in a westerly direction, supporting the concept that this direction is towards the deepening marine basin. The base of the TAU is interpreted to be a thrust fault, and there are numerous other fault zones within the unit as noted in drill cores and the adit. The basal thrust plane does not cause significant offset of the Sinwa limestone in the fold nose to the west, implying a scissor-type action with increasing movement to the east.

Overlying the TAU and truncating it to the west is the Kutcho conglomerate. This unit is a heterolithic, fragment-supported conglomerate composed of sub-rounded clasts, ranging in size from 1 cm to 38 cm (long axis), and derived from all the underlying lithologies. The conglomerate is conformably overlain and transitional into the Sinwa limestone, which in turn appears to be conformably overlain by Jurassic aged Inklin Formation argillite. However, it is quite possible that there could be a contact between Kutcho Formation argillite and Inklin Formation argillite higher in the section, which would be difficult to spot and could be unconformable.

Thorstad (1983) reported an Upper Triassic age for the the Kutcho Formation based on rubidium-strontium dating of volcanic rocks and regional stratigraphic constraints. Subsequent U-Pb dating provided Lower Triassic to lowermost Middle Triassic ages (Childe and Thompson, 1997).

7.3 Structure

Rocks of the Kutcho Formation are characterized by penetrative axial planar foliation that has a relatively constant strike direction of 270° to 290° with northerly dips from 45° to 65°. Minor but systematic changes in foliation from the east to west suggest low amplitude buckling of the fold axes. There appears to be a tendency for the dip of the foliation to decrease with structural depth, indicating that the axial planes are convex to the south.

Folds are open to tight, asymmetrical, inclined, and verge to the south. Fold plunges range from 0° to 30° in a westerly direction. Folds are most evident in well bedded competent units, and therefore spatial distribution of the fold data is heavily biased to the western property area, where these units predominate.

Two aspects of the structure that critically affect stratigraphic interpretations are the number and size of foliation-parallel thrust faults, and the degree to which the folds are propagated through the stratigraphic sequence. Neither of these aspects can be determined independently, and thus there remains considerable scope to reinterpret the stratigraphic position of various units locally. Foliation-parallel thrust faults are difficult to detect from surface outcrop, but can be inferred from missing stratigraphy, contact geometry, shearing, and topographic evidence. Faults of this type are consistent with the deformation style and are considered to be prevalent through the Project area.

Fold hinges outlined by the Sinwa limestone unit on Conglomerate Ridge (immediately east of Kutcho Creek) are difficult to trace in an easterly direction. Structural data (Holbek, 1985) indicate that the folds are cylindrical and therefore should be continuous within the depth of exposed stratigraphy. However, lithological competency contrasts are likely to result in disharmonic folding (Holbek and Heberlein, 1986), causing discontinuity of the axial plane towards the core of the fold. Stratigraphically thicker units will tend to produce a series of lower amplitude folds toward the core of the structure, which may explain why the fold axes so clearly outlined by the limestone unit on the western part of the Property are not at all evident to the east, in the vicinity of the Sumac and Main deposits. Therefore, a certain degree of flexibility needs to be maintained regarding structural and stratigraphic interpretations in the vicinity of the sulphide deposits (Wardrop, 2007).

7.4 Mineralization

Mineralization occurs as three deposits along a 3.5 km trend. Sulphide minerals occur in a series of massive sulphide lenses and include pyrite, sphalerite, chalcopyrite, bornite, minor chalcocite, and trace tennantite, galena, digenite, djurleite, and idaite. Gangue minerals include quartz, dolomite, ankerite, sericite, gypsum, and anhydrite.

The three deposits form a westerly plunging linear trend, termed the Main, Sumac, and Esso deposits from east to west. The Main deposit comes to surface at its eastern end, whereas the Esso deposit occurs at depths about 400 m below surface, as shown with drillholes and topography in Figure 7-4.

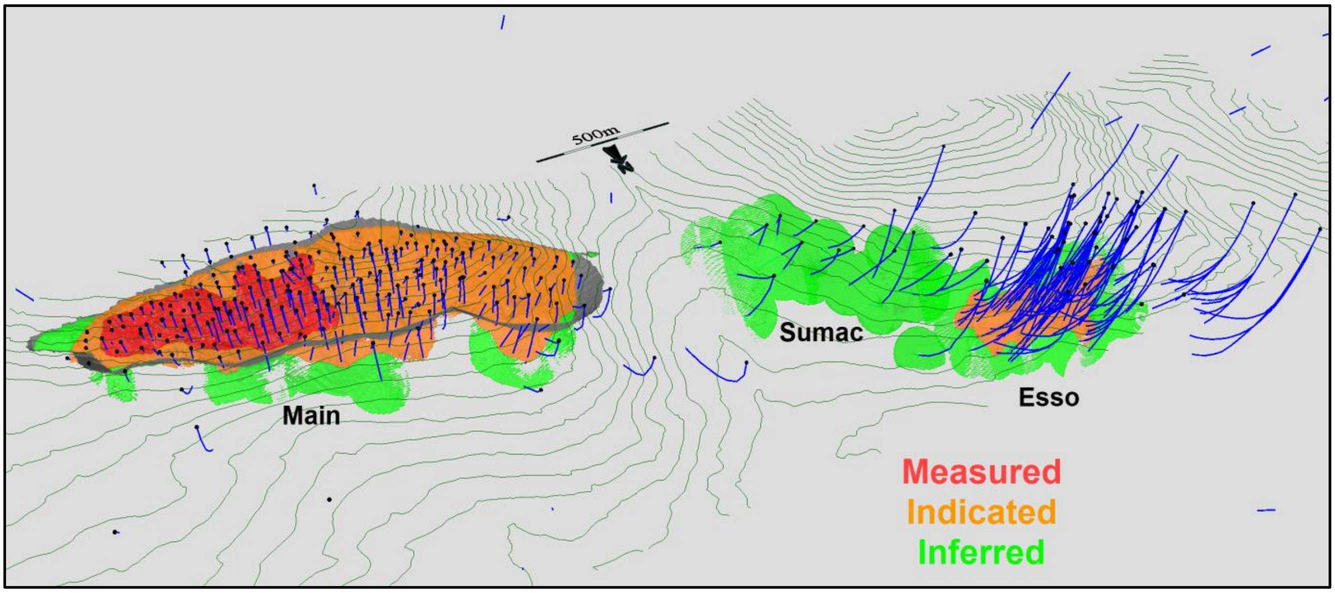


Figure 7-4: Main, Sumac and Esso Deposits Looking Southwest Showing Drillholes and Topography
(Source: Sim, 2021)

7.5 Main Deposit

The Main deposit has an elliptical, lenticular shape with approximate dimensions of 1,600 m long, 500 m wide (down-dip), and approaching 40 m in true thickness in some areas. The long axis of the deposit plunges to the west at about 12°, just slightly less than the regional fold axes. The deposit is conformable with stratigraphy, dipping moderately to the north. There is a gentle warping of the deposit, such that the dip of the deposit changes from east to west and north to south. The shallowest dip (about 40°) occurs at the south-eastern edge and becomes progressively steeper (to about 60°) at the north-western edge. In general, the up-dip edge of the sulphide lens is narrow and pinches out, whereas the down-dip edge is thicker and interlayered with tuffaceous rock.

Sulphide mineralogy of the deposit is relatively simple, consisting of pyrite, chalcopyrite, sphalerite, and bornite, with minor sulphide minerals chalcocite, tetrahedrite, digenite (and related minerals), galena, idaite, hessite, and electrum. Gangue minerals include quartz, dolomite, ankerite, sericite, gypsum, and anhydrite. Fluorite and barite have been observed, but do not occur in volumetrically significant amounts.

The internal stratigraphy of the Main deposit was determined by detailed drill core logging (Holbek and Heberlein, 1986) along a single longitudinal section of drillholes. The deposit appears to have formed from three hydrothermal-depositional cycles that began with barren pyrite at the base and grade into a copper-rich middle and zinc-rich top. Depositional cycles are commonly separated by layers of exhalative quartz and/or carbonate and minor volcanic ash. However, continued hydrothermal activity resulted in sulphide-replacement mineralization. Additional features also cause complexity to the internal sulphide stratigraphy, such as an irregular depositional surface, localized slumping of sulphide mineralization or chimney collapse, and late stage (post depositional) hydrothermal activity.

Areas of late overprinting by oxidized copper species and enrichment in precious metals are interpreted as indicators of vent areas and occur along a linear trend on the down-dip side of the deposit, with two hotspots near each end of the deposit. However, no well-defined areas of classical footwall stringer mineralization have been identified by drilling.

The upper contact of the sulphide mineralization is relatively sharp, with almost no sulphide minerals occurring in the hanging wall rocks except for scattered coarse grains of porphyroblastic pyrite. However, alteration of feldspar to sericite and carbonate in the hanging wall is intense and occurs for up to 50 m above the sulphide contact. It is common for a thin but persistent shear zone to occur at the sulphide-schist contact, which varies in thickness from 20 cm to 200 cm, and in many drillholes, carries some grade.

The base of the deposit consists of nearly barren massive pyrite with interstitial quartz. The contact between ore and the footwall pyrite zone can be either gradational or sharp. Below the footwall pyrite zone is quartz-sericite schist with bands of generally barren massive to semi-massive pyrite. The footwall pyrite content diminishes with depth away from the deposit but is noted to extend to a depth of 200 m below the central part of the deposit.

Interpretation of the shape of the sulphide zone, taken together with the observed volcanic and depositional textures of the enclosing rocks, suggests that the sulphide mineralization was deposited in a structural depression, likely a half-graben type structure. Fine mineralogical layering and sulphide-ash, sulphide-silica, or carbonate inter-layering, as well as framboidal and snowball textures in both the sulphide and carbonate minerals, suggests quasi-sedimentary deposition at the seawater-seafloor interface. Polished section analysis indicates that very little sulphide recrystallization has taken place.

7.6 Sumac Deposit

The Sumac deposit was initially identified by a chargeability anomaly in the mid-1980s and is located approximately 550 m west of the Main deposit, is nearly continuous with the Esso deposit and sits within a local depression relative to the Main and Esso deposits.

A total of 20 drillholes at approximately 100 m spacing define the Sumac deposit. Better intercepts include 1.45% Cu, 2.56% Zn, and 23.7 g/t Ag over 26.1 m, 1.37% Cu, 1.9% Zn, and 26.2 g/t Ag over 23.4 m, and 1.94% Cu, 2.66% Zn, and 43.2 g/t Ag over 10.1 m.

The Sumac deposit is finely banded but massive and competent, containing the highest sulphide content (>90%) of the three deposits. Alteration of the host stratigraphy around it is very similar to that of the Main and Esso deposits.

The Sumac deposit is not fully delineated with drilling at present but with current knowledge can be described as having an elongated lens shape that commences close to surface near the Sumac Creek, with a strike length of approximately 800 m x 300 m in the dip plane and is up to 25 m in true thickness but averages approximately 12 m thick.

7.7 Esso Deposit

The Esso deposit is the deepest and most westerly massive sulphide lens and lies between 350 m and 650 m below the surface. It was discovered by following the westward trend of mineralization down plunge beyond the Main and Sumac deposit areas. The Esso deposit has an elongated lens shape with a strike length of approximately 900 m x 400 m in the dip plane and is up to 25 m in true thickness but averages approximately 12 m thick.

Mineralization at the Esso deposit tends to be higher grade than at the Main or Sumac deposits and shows similar mineral zonation with copper or zinc in layers or zones, as well as zonation in thickness and grade from the central deposit area. Although the copper and zinc grades tend to be higher at Esso, the deposit contains significantly lower overall sulphide concentrations compared to the Main and Sumac deposits. Alteration at Esso is similar to the Main deposit, where sericite alteration of feldspars in the hanging wall is gradational from very weak at distances up to 50 m stratigraphically above to deposit, to very intense in proximity to the sulphide zone.

7.8 Other Mineralization

Other zones of mineralization include the footwall zone (FWZ), the hangingwall zone (HWZ) and the Jenn Area. The FWZ occurs approximately 120 m stratigraphically below the Main deposit and slightly up-dip and to the east of the centre of the Main deposit. The FWZ is relatively narrow, at 2–5 m thick, and relatively zinc-rich. The mineralization was only systematically drilled up to the historical Esso-Sumac property boundary, but several drillholes by SML (and more recently WKM) demonstrate that the FWZ does not extend for significant distances to the west and down-dip of its current position.

The HWZ is a narrow band of relatively low-grade mineralization that occurs within 1–2 m to up to 15 m stratigraphically above the Main deposit. The HWZ is interpreted over a strike length of 1,300 m x 400 m on the dip plane and is often only 1–3 m in thickness.

The Jenn area is on the eastern end of the property and was explored most actively by EML. Although significant alteration and some local mineralization were intersected in several drillholes, no resources have been defined in the Jenn area.

8 Deposit Types

Mineralization of the Project is part of the volcanogenic massive sulphide (VMS), or volcanic-hosted massive sulphide (VHMS) spectrum of deposits. These deposits are major sources of copper, zinc, lead, silver, and gold around the world. They form near the seafloor where hydrothermal fluids driven by magmatic heat are quenched through mixing with groundwater near seafloor lithologies. Massive sulphide lenses vary widely in shape and size and are often pod-like or sheet-like. These deposits are generally stratiform and may occur as multiple lenses (Shanks et al., 2012).

VMS deposits have been classified into various subtypes, depending upon the composition of the host rocks and the mineralization, and the tectonic setting of origin. The Kutcho deposits are VMS deposits of the Kuroko type or felsic volcaniclastic depending upon the classification scheme. Mineralization is related to felsic volcanism in island arc or back-arc tectonic settings. Perhaps the most significant feature of VMS deposits from an exploration perspective is their tendency to occur in clusters. Larger VMS camps can have up to 25 discrete deposits, and mineralized districts are common.

Features of the Kutcho deposits suggest that they formed at (or very near to) the water-seafloor interface in a structurally controlled depression, likely a half-graben type structure. The Kutcho deposits have some uncommon features: the absence of lead and barite is likely due to the low potassium content of the volcanic host rocks (and presumably the associated rhyolite dome), and the abundant carbonate is probably of exhalative origin.

Alteration associated with VMS deposits is well documented and provides a valuable exploration tool in that the area of alteration is much larger (by a factor of up to 10 to 100) than the actual sulphide deposit, thereby providing a large exploration target. Extensive studies of the alteration around the Kutcho deposits have been undertaken, and the chemical composition of the alteration is well zoned about the hydrothermal ventilation areas. This zonation allows the use of geochemical analysis of drill core within the alteration zone to provide vectors towards the hydrothermal ventilation area and, hopefully, the sulphide deposits.

9 Exploration

Most exploration of the Kutcho project area was carried out by previous operators and has been summarized in Section 6. This exploration was also described in the NI 43-101 Technical Report completed in 2017 by Desert Star Resources Ltd upon acquisition of the Project. Desert Star changed its name to Kutcho Copper Corp. upon completion of the acquisition.

The primary goal of the work conducted on the Kutcho Project since the previous (June 2017) Technical Report was to collect the required technical field data and information to support the Feasibility Study. This includes the drilling of holes for geotechnical and water monitoring purposes as well as to collect material for metallurgical testing.

In addition to the work related to the Feasibility Study, some minor surface exploration programs were conducted consisting of minor soil sampling, prospecting, and mapping.

9.1 Surface Exploration Programs

During 2018, a total of 83 soil samples, including field duplicates, were collected on the mine lease (claim number C569607) along north-south oriented grid lines spaced 100 m apart. Individual soil samples were collected at 50 m intervals using geotuls or mattocks to excavate past the organic cover, with a goal of sampling the "B-horizon" where possible. This work outlined a weak surface geochemical expression of the FWZ located stratigraphically below the Main deposit. Based on these results, additional soil sampling is recommended to the west to delineate drill targets and extend the known mineralization extents of the FWZ.

Prospecting and evaluation of historically identified mineralized occurrences was also undertaken in 2018. The primary objective of this work was to evaluate the merits of undertaking additional work, such as drilling, in order to provide a better understanding of these exploration targets.

10 Drilling

10.1 Drilling History

The first drillholes into the Main deposit were carried out nearly simultaneously by Esso Minerals Ltd (EML) and Sumac Mines Ltd (SML) in the summer of 1974, although within different areas of the deposit. The first two seasons of drilling were primarily exploratory, with relatively wide-spaced drillholes used to determine approximate extents of the mineralization. In 1976, SML carried out airborne EM geophysical surveys, which provided indications of the size of the conductive mineralization, as well as indications of the presence of the soon to be discovered Sumac deposit. Shortly thereafter both companies adopted a delineation drilling approach, with holes spaced on a regular grid pattern along north-south grid lines spaced at 200-foot (approximately 60 m) intervals, with individual drillholes spaced 200-feet apart. Additional infill drillholes were added on a 100 x 100-foot (approximately 30 x 30 m) pattern in some parts of the Main deposit.

Most of the drilling completed on the Main deposit are located along the north-south grid lines and similar approaches were used to drill at the Sumac and Esso deposits.

Additional drilling conducted through the 1980s and 1990s comprised 30,161 m of drilling. This included holes for geotechnical purposes at the Main and Esso deposits plus holes that tested regional exploration targets including the Treaty Creek and Jenn showings.

Beginning in 2004, HQ core drilling in the Main deposit was carried out by WKM to verify the previous drilling and to further delineate the deposit as well as to obtain material for metallurgical testing. The approach to drillhole locations for the 2004 and 2005 programs was to obtain a distribution of drillholes covering the entire deposit area, with specific drillhole locations placed where they would result in infilling areas of lower drillhole density. Because the HQ drill was a skid-mounted rig, drillhole locations were restricted to areas accessible to such a rig with minimal road building. The 2006 drilling program on the Main deposit was designed to infill the upper "starter pit" area of the deposit on approximate 30 m centres.

Sampling methods for drill core were similar for all the exploration drilling programs conducted on the property. Core sizes varied from BQ, NQ and HQ and sampling of the core was initially done using a mechanical splitter but was changed to a diamond saw after the first nine drillholes by SML and after approximately 30 drillholes by EML. Sample splitting by diamond saw was used ever since.

In 2008, HQ core drilling in the Main deposit was carried out by Capstone to infill gaps in previous drilling, verify historical drilling data, to obtain material for metallurgical testing and to collect detailed geotechnical data. The drilling was helicopter supported, facilitating access to collar locations not possible during previous programs, resulting in delineation of majority of the deposit with holes spaced on a regular 30 m grid pattern. A total of 9,897.7 m was drilled in 81 holes (78 holes for the Main deposit, including three holes totalling 69.2 m which were lost before intersecting the mineralized zone).

In 2010, Capstone conducted delineation drilled on the Esso deposit to increase drillhole density and to define the lateral extents of the mineralization. Material used for metallurgical testing was obtained from half (NQ) core samples and assay samples were derived from quarter core splits (retaining quarter core for future reference). Specific gravity (SG) measurements and geotechnical and geological core logging were also completed. Most holes completed during the 2010 program were started with HQ core to depths of about 200 m below surface before being downsized to NQ core Helicopter support was used for the drill program, again facilitating access to collar locations not possible in previous ground-support only programs. Overall, 34 holes (18,042.1 m) were drilled at the Esso deposit, including five holes (1,324.3 m) which were abandoned above the mineralized zone when it became apparent the hole could not hit the planned target location at depth.

In 2011, drilling conducted by Kutcho Copper testing nine geophysical VTEM exploration targets surrounding the Kutcho deposits as well as definition drilling at the Sumac deposit, totalling 4,227 m of HQ size core in 19 holes (including two short, abandoned holes). In addition to this exploration drilling, several vertical groundwater monitoring wells were drilled at Main and Esso for a total of 645 m of drilling.

During the summer of 2018, Kutcho Copper completed 69 drillholes (10,740 m) using two Boyles 37 diamond-drilling rigs provided by Cyr Drilling of Winnipeg, Manitoba. The drilling program was managed by Equity Exploration Consultants under the guidance of Kutcho Copper personnel. Majority of the program included close-spaced delineation drilling at the Main and Esso deposits but also included holes designed to provide geotechnical and hydrogeological information as well as holes to provide additional material for metallurgical testing purposes. The majority of drillholes were HQ diameter unless a reduction in diameter was required to reach target depths, in which case the holes were finished as NQ diameter

During the spring and early summer of 2021, a drilling program was conducted in order to obtain additional geotechnical data to support the design of an open pit at the Main deposit. Four drillholes were completed that targeted the footwall area of the proposed open pit.

Additional groundwater testing was also conducted on these drillholes. The 2021 program also included five holes drilled to obtain soil and rock samples for geotechnical test work in the area of the TMF embankment. There were no samples collected for laboratory (grade) analyses and, as a result, the drilling conducted during 2021 has no impact on the mineral resources contained in this report.

10.2 Drilling Completed

There is a total of 631 individual drillholes in the Kutcho drilling database. This includes all the delineation drilling conducted on the three mineral deposits, as well as holes that test proximal exploration targets, some of which are located more than 10 km away from the Kutcho deposits. The database also includes a number of holes that were drilled for geotechnical, hydrogeological, metallurgical or other purposes and, as a result, they do not have any associate sample assay (grade) data. The distribution of drilling by the various operators of the Project is summarized in Table 10-1. Note, this table does not include the limited geotechnical and hydrogeological drilling that was conducted during the summer of 2021.

Table 10-1: Summary of drilling by company (number of holes)

Company	Main	Sumac	Esso	Other	Total
EML/SML (1973–1983)	175	14	62	44	295
ARMC (1990)	3	-	-	25	28
Atna (1997)	-	-	-	9	9
WKM (2004–2006)	68	4	18	3	93
Capstone (2008)	81	-	-	2	83
Kutcho Copper (2010–2011)	1	6	36	11	54
Kutcho Copper (2018)	32	-	3	34	69
Total	**360**	**24**	**119**	**128**	**631**

Locations of drillholes that define the Esso, Sumac, and Main deposits are displayed on a plan map (Figure 10-1).

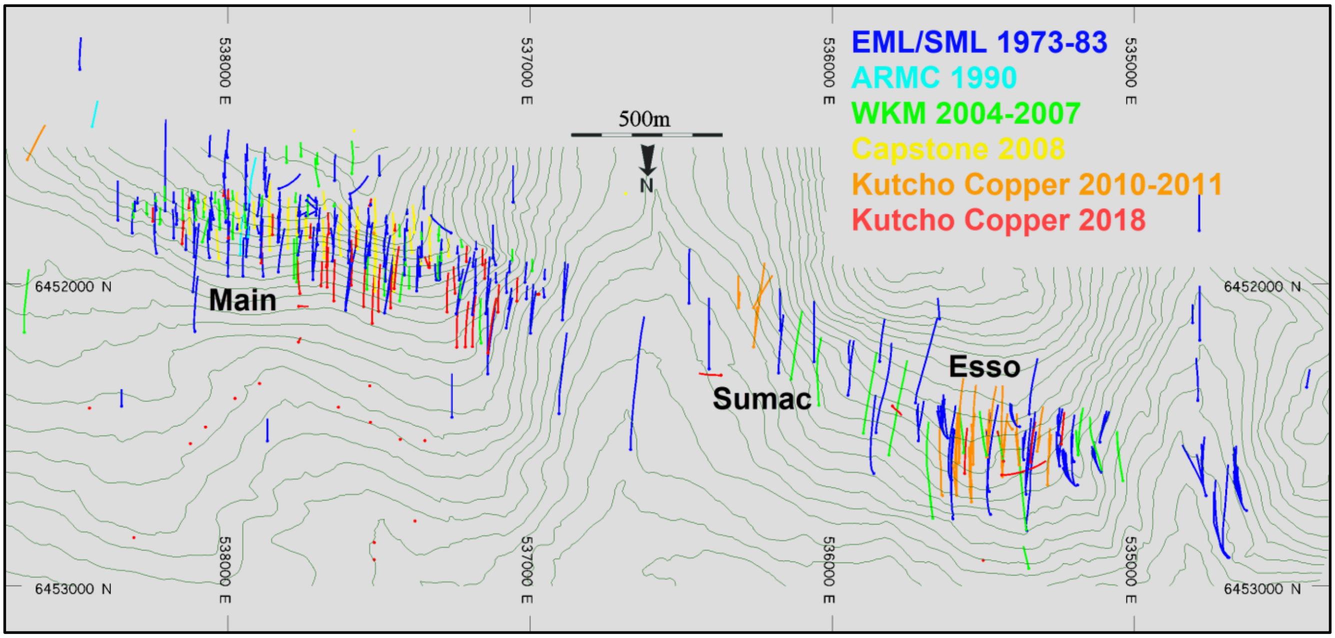


Figure 10-1: Plan Showing Drilling by Vintage in the Main, Sumac and Esso Deposit Areas

(Source Sim, 2021)

10.2.1 Collar Surveys

Up until 1983, all drillhole collars (and claim locations) were surveyed periodically during exploration programs by McElhanney Engineering Services Limited (MESL); all later WKM drillholes and many of the historical drillholes were surveyed or resurveyed by MESL in September 2006. Surveys conducted prior to 1983 were conducted using a local "mine grid" coordinate system. Since that time, all subsequent work on the Project was done using UTM coordinates. The older (pre-1983) survey locations were converted to UTM coordinates by MESL.

Most of the drill sites have been reclaimed, however, many of the hole collars can still be found in the field and most of the drill core remains stored on the property. A newer storage facility with core racks was constructed for the Esso deposit drill core in 1985, and all the previous stored core was relocated into the new facility during the period from 1985 to 1991. The Sumac deposit drill core had been stored on core racks located in the area between the Main and Sumac deposits. Due to decomposition of these racks, the core was removed and cross-piled nearby, and the core racks were dismantled. Approximately half of drill core from the Sumac and Esso deposits was relogged in 1984 and 1985 (Holbek and Heberlein, 1986) using the GEOLOG system.

The collar locations of holes drilled during the 2018 program were surveyed by Challenger Geomatics Ltd. Based out of Whitehorse, YT, using an RTK Differential GPS.

The locations of the holes drilled for geotechnical purposes in 2021 were surveyed using a handheld GPS instrument. These holes are clearly marked in the field and can be resurveyed with the RTK system if required.

10.2.2 Drill Core Diameter

Historical drilling mainly produced BQ size core (38 mm), and recoveries were generally very good with only rare core losses occurring in minor fault zones. Beginning with the 2004 program, changes in drilling equipment allowed for larger diameter (HQ and NQ) core to be produced. The 2018 drilling program used a combination of HQ and NQ core in the Main deposit and HQ, NQ (or BQTW in wedge branches) core within the Sumac and Esso deposits. All holes drilled in 2018 to collect material for metallurgical testing started with HQ size core with a reduction to NQ core in some of the deeper holes.

10.3 Pre-2018 Drilling Location, Spacing, and Objectives

Initial drilling on the Main deposit was carried out on 120 m spaced north-south oriented vertical cross sections with drillholes spaced approximately 60 m along section lines. As drilling progressed, this spacing between holes was subsequently reduced to approximately 30 m along cross sections spaced at 60 m intervals. WKM further reduced the drillhole spacing in selected areas to 30 m or less.

Most of the drillholes in the central part of the Esso deposit intersect the mineralized zone at approximately 50 m spaced intervals. Due to the strong north-dipping foliation present in the host rocks, holes drilled at the Esso deposit tend to flatten with depth, resulting in cored intervals that intersect the mineralized horizon roughly perpendicular to its dip.

The 2008 drill program conducted by Capstone was designed to infill an area of the Main deposit with close spaced holes, principally to increase the confidence in the resource classification and to better define higher grade trends as well as to provide sufficient sample material to conduct more extensive metallurgical test work. Additional data such as measured bulk densities, fracture density, and other geotechnical data were also recorded during this drilling program.

Following geological logging of the drill core generated during the 2008 program, the core was split to provide material for both assay samples as well as material for metallurgical testing. Metallurgical sample material was taken from half core samples which were sealed in nitrogen-filled bags and stored in nitrogen-filled pails for stable storage in an oxygen deprived environment. The remaining half core was further split, into quarter core samples, to provide material for analyses at the laboratory.

The main objective of the 2010 drill program was to increase the density of holes in the deposit in order to increase confidence in the resource classification, better define higher grade trends, and provide sufficient sample material to conduct more extensive metallurgical test work. Additional data such as measured bulk densities, fracture density, point-load tests and other geotechnical data were also recorded during this program. The result produced a more robust geological, geotechnical and resource model that could support a more robust economic assessment.

Several holes were drilled into the Sumac deposit during the 2011 program in order to gain a better understanding of the deposit. The current distribution of drilling at Sumac does not provide the level of confidence required to support resource estimates in the Indicated (or Measured) categories. Additional close-spaced drilling is required to upgrade the resources at Sumac. The Sumac holes were drilled at an azimuth of 180º, with inclinations ranging from -50º to -70º. This inclination ensured that most drillholes intersect at or near perpendicular to the mineralized target horizon, providing “true thickness” intercepts.

10.4 2018 and 2021 Drilling Program Details

Most of the holes drilled on the Main deposit during the 2018 program were oriented at an azimuth of 180º with inclinations ranging from -90º to -45º but averaging between -60º and -45º. These orientations were designed to provide drill intercepts that are roughly perpendicular to the mineralized target horizon.

10.4.1 ***Rig Alignment***

Drill rig alignment (azimuth and dip) was completed by Equity geologists with the help of a Reflex Azimuth Positioning System (APS) or SurveyTech DeviAligner. The APS is a GPS based compass that is not affected by local magnetic interference (natural or manmade) and produces true north azimuth measurements to within 0.5° with good GPS integrity. The DeviAligner gyrocompass is a gyroscope-based compass that like the APS is not affected by local magnetic interference. It has an advantage over the APS in that it does not rely on a GPS signal to determine azimuth. It also produces true north measurements to ±0.23°, depending on latitude.

10.4.2 Downhole Surveys

Downhole surveys were completed using both a Reflex EZ-TRAC system and Reflex EZ-GYRO. The EZ-TRAC is a "single-shot" high precision magnetic instrument that measures drillhole azimuth relative to magnetic north, drillhole dip and magnetic field strength. EZ-TRAC survey measurements were collected as the drillhole advanced to determine if the hole was progressing along the planned alignment or if deviation was occurring. The first survey was completed once the hole had penetrated several metres into bedrock below casing, followed by ~30 m intervals for the rest of the hole. Magnetic north azimuth readings were converted to UTM grid north by adding 18.57°, which is the grid declination in the Project area for 2018. Downhole surveys were not accepted and flagged as "false" if the corrected azimuth was strikingly different from azimuths on either side of it, which is usually a result of localized magnetic field interference (i.e. magnetic rock, drill string). In general, magnetic field strengths of 5600–6000 nT were accepted whereas those below 5600 nT and above 6000 nT indicated magnetic interference.

The Reflex EZ-GYRO is a true north seeking gyroscope that uses a micro-electromechanical system (MEMS) gyro that measures drillhole azimuth and dip deviation relative to true north. The gyroscope is not affected by magnetic interference and can be used inside drill rods or magnetically disturbed ground. At completion of drilling, survey measurements were made through the drill rods at 10–30 m intervals downhole, depending on the length of the hole, and a second survey was completed on the way out of the hole at 30–50 m intervals to provide a check on the quality of survey data. Azimuth readings were converted from true north to grid north by adding 0.53° convergence angle for the finalized data.

EZ-TRAC and EZ-GYRO survey measurements were not completed on vertical drillholes (i.e. hydrogeological and surficial geotechnical drillholes) as both tools are at their working limits with regards to hole dip and cannot produce reliable azimuth measurements. All metallurgical, resource infill and mine geotechnical drillholes were surveyed with both the EZ-TRAC and EZ-GYRO tools.

Overall, the EZ-GYRO data is preferred over the EZ-TRAC as there is more data for survey control, and there do not appear to be significant problems with the tool as evidenced by better matching and relatively consistent survey control on rig 37-5 over the course of the program.

10.4.3 Drillhole Collar Surveys

Drillhole collar surveys were completed for all 2018 drillholes and several holes drilled in 2011 that had previous survey control. In total, 68 of the 2018 drill collars plus an additional nine holes drilled in 2011 were surveyed following the 2018 drilling program. All surveying was completed by Challenger Geomatics Ltd ("Challenger") of Whitehorse, Yukon using an RTK differential GPS system (DGPS) with radio base stations set up in proximity to drilling sites to provide real time kinematic corrections. Leica GS16 and GS18 GNSS receivers were used to perform the surveys that were completed in NAD83 UTM Zone 9.

Challenger tied into an existing historical control point (5207) that was found to be within a few centimetres of the 2007 survey. Challenger also tied into two historical holes for check shots to confirm that the 2018 survey would match the survey completed in 2007. The co-ordinates and comparisons of the drillhole check is displayed in Table 10-2. Challenger also established three new control points around the Project site that could be used for future surveys.

Table 10-2: Co-ordinate comparison between the 2018 and 2007 survey on historical drillholes Wk06-76 and Wk0684

	2018 Challenger coordinates			2007 Historical coordinates			Change from 2018 to 2007		
	Northing	Easting	Elevation	Northing	Easting	Elevation	Northing	Easting	Elevation
WK06-76	6451782.64	538008.07	1616.61	6451782.609	538008.070	1616.64	0.045	-0.003	-0.031
WK06-84	6451819.74	538027.86	1608.81	6451819.74	538027.84	1608.82	0.000	0.011	-0.010

Core recoveries for the 2018 drill program were calculated by summing all the core recovered and then dividing by the total meters drilled not including overburden. Total core average recovery for the 2018 drill program was very good at 97.14%. This is similar to recoveries documented during previous drilling programs.

10.5 Geotechnical and Hydrogeological Drilling 2018 and 2021

A geotechnical field data collection program was designed to characterize the rock mass, hydrogeology, structural fabrics, and major structures (i.e. faults) associated with both the Main and Esso deposits. This data was collected with the intent of developing a geotechnical model suitable for open pit and underground mine design. The field data collection program consisted of geotechnical logging of oriented core, index strength testing (i.e. point load testing), packer testing, geomechanical sample collection, and optical/acoustic televiewer surveying. The initial geotechnical field program was completed between 12 June and 9 October 2018. An additional geotechnical field program was completed between 11 May and 13 June 2021.

Drill core was oriented for all mine geotechnical drilling and for one resource infill drillhole (KC18-278) that was used for the geotechnical program. Bottom of hole core orientation marks were also placed at the end of each run for several of the resource infill holes.

For the geotechnical holes, bottom of hole core orientation was completed from the base of overburden until EOH by the drill crews using a Reflex ACTIII RD core orientation system combined with a Gyro non-magnetic downhole survey tool. Measurements for dip and azimuth were collected with the Gyro at regular intervals in each hole. For the 2021 geotechnical program optical televiewer (OTV) and acoustic televiewer (ATV) downhole surveys were carried out on each geotechnical drillhole completed.

All geotechnical core logging was completed in accordance with accepted geotechnical logging standards and included the collection of the required parameters to calculate RMR76 (Bieniawski, 1976) and the Q-system (Barton et al, 1974). The logging consisted of interval logging or detailed logging of each core run. Each core run was 3 m long using a standard core barrel. For interval logging, data was collected on core recovery, Rock Quality Designation (RQD), discontinuity characteristics (e.g. alteration, weathering, and infill) and fracture counts.

Additional hydrogeological wells were installed during the 2018 program in and around the known deposits to better characterize the groundwater conditions and generate a more robust site water balance model. In order to develop an appropriate site water balance, hydrogeologists require a minimum of one year of data and that data collection is currently ongoing. One of the major data gaps in the previous studies, which this program compensates for, is the conditions at depth. Of particular consideration is the conditions in and around the Esso lens. Through this program and in addition to the historical data, a new and more detailed site water balance model was designed.

11 Sample Preparation, Analyses and Security

11.1 Drillhole Sampling and Analyses for Drill Programs Prior to 2008

11.1.1 ***Esso Minerals and Sumac Mines***

Neither EMC nor SML documented their sampling and analytical protocols used during the drilling conducted from 1973 to 1983, but both companies typically carried out their work following industry standards of the time.

Some of the EMC drill logs have duplicate sample data written adjacent to the original samples, suggesting that duplicate samples were a standard procedure. These duplicate samples display normal expected variations compared to the original sample grades, and it is possible that these duplicated grades were reanalysis of sample pulps by a different (umpire) laboratory. There is no documentation of any analytical problems related to samples collected by EMC and SML.

Based on the available information, both EMC and SML logged core at camps on site, marked samples, and split core for analytical samples. EMC used a manual splitter for the first three years and then switched to a diamond saw (preferable for massive sulphide mineralization), while SML used a manual splitter for the first year of drilling (nine holes) and then switched to a diamond saw thereafter.

EMC tended to take longer samples of about 2–3 m lengths (to a maximum of 5.4 m) during the initial two to three years of exploration, but took shorter, more geologically constrained samples in the later years. Additionally, in much of the early sampling by EMC, the geologists restricted their samples to areas exhibiting visual signs of significant mineralization, with no additional samples from above or below the mineralized zones. Based on current experience with the deposit, it is unlikely that EMC missed any significant grade; however, no samples of potential dilution are available.

SML tended to take shorter standard sample intervals (averaging 1 m in length) and samples were selected on the basis of geology and sulphide mineralogy, and they often included shoulder samples above and below zones of significant mineralization.

11.1.2 ***Western Keltic Mines***

EMC used Min-En Labs of North Vancouver (acquired by Bondar-Clegg in the 1990s) for almost all its analytical work, while SML used ALS Chemex Laboratories (Chemex, acquired by ALS in 1999), also of North Vancouver. Check samples for EMC were carried out by Terramin Labs in Calgary and SGS Lakefield Research Ltd in Lakefield, Ontario.

The core handling and sampling procedures used by WKM from 2004 to 2006 are described in Wardrop (2007). Drill core was transported to the core logging facility at site via helicopter or truck. Core was logged and marked for splitting. Mineralized drill core intersections were sawn in half, and one half was bagged and sent for analyses and the other half was retained in core boxes at site. Approximately 70% of the holes drilled by WKM were used to provide material for metallurgical testing. In these cases, one half of the core was packed in nitrogen-filled, sealed bags which were then packed within airtight, nitrogen-filled plastic pails. The remaining half core was further sawn and quarter core was sent for analyses and the remaining quarter core was returned to the core box and stored at site. New BQ boxes were used to hold the quartered HQ or NQ core. Sealed sample bags were then placed into rice bags for transport by charter aircraft to Dease Lake or Smithers, BC where it was then transported by trucks to ALS in North Vancouver for analyses.

Core samples (including blanks) were ground to 80% passing 100 mesh and were analyzed (including standards) using induced coupled plasma (ICP) methods for 33 elements following an aqua regia digestion. Copper, zinc, and silver values above the ICP detection limits (10000 ppm for copper and zinc; 100 ppm for silver) were assayed by atomic absorption methods following an aqua regia digestion. All samples were analyzed for gold by fire assay in 30 g subsamples, and sulphur was analyzed by Leco furnace.

Specific gravity measurements were done on split core using the process of weighing in air and weighing in water, following the same procedure as described in Section 11.2.

11.2 Drillhole Sampling and Analyses for 2008–2018 Drill Programs

This section of the report describes the drill core handling practices, sampling procedures, analyses of samples and security of the processes followed by Capstone and Kutcho Copper for all drilling programs that were conducted between 2008 and 2018.

Drill core was transported from the drill site to the camp at least once a day by helicopter or pickup truck, depending on the location of drilling and road accessibility. Boxes were opened, sorted and inspected for any irregularities. Geotechnical and geological logging was conducted. Sample intervals were marked with red crayon on the core and sample tags were assigned, and one part of the sample tag was stapled to the core box at the end of each sample interval. Sample tags for standards, blanks and duplicates were also stapled to the boxes in the same order that they were inserted.

Sampling of drillholes was primarily constrained to zones that showed visible signs of the mineralization. The standard sample intervals in mineralized zones ideally ranged from 1.0 m to 1.5 m long and could be up to 3.0 m where barren intervals occurred between mineralized zones. Typically, two additional shoulder samples were taken in (generally) unmineralized hanging wall and footwall rocks immediately adjacent to the mineralized zones. Assay sample intervals were selected so they do not cross geological contacts.

Some of the holes drilled by Capstone and Kutcho Copper were also used to provide material for metallurgical testing. In these cases, one half of the cut core was packed in nitrogen-filled, sealed bags which were then packed within airtight, nitrogen-filled plastic pails. The remaining half core was further sawn and quarter core was sent for laboratory analyses and the remaining quarter core was returned to the core box and stored at site. In the opinion of the QP, these smaller quarter core samples would not have a material impact on the quality of the sample data generated.

Following logging and the assignment of sample intervals, core boxes were photographed and then moved into a designated sampling building where core was cut in half using a diamond rock saw. Half of the core was retained in the core boxes and stored for future reference and the remaining half core was placed into a plastic sample bag together with the sample tag and sealed with zip ties. Typically, six to 10 bagged samples were then packaged into rice bags with security numbered zip seals.

Specific gravity (SG) measurements were conducted on split (cut) core pieces by site personnel in the core shack. Samples for SG measurements were typically selected at 10 m intervals down each drillhole and in the mineralized zones. SG measurements were conducted on every sample interval. The weight of individual samples was determined while suspended in air and then submerged in water using an Ohaus scale. The resulting SG values were calculated using the following formula:

$$SG = \frac{\text{Weight in Air}}{(\text{Weight in Air} - \text{Weight in Water})}$$

Sample weights were accurate to the nearest tenth of a gram. Samples selected for SG measurements were not sealed prior to weighing. The QP believes this does not impact the results because the rocks at Kutcho do not show any signs of porosity.

The sealed rice bags were transported from the on-site core processing facilities on fixed-wing charter planes, typically on return flights from crew and supply runs, generally on a weekly basis. When the plane arrived in Whitehorse, the expeditor received the samples from the hangar of the charter company and delivered the sealed rice sacks to the ALS geochemistry preparation laboratory at 78 Mt. Sima Rd, Whitehorse, Yukon, Canada. Although no irregularities were noted in the sample shipments, the charter company, expeditors and laboratory staff were given instructions to notify both Equity Exploration's team and Kutcho Copper's VP Exploration and Development of any problems or damage to any of the bags that would break the seals placed by the core logging team.

Kutcho samples were sent for analyses to ALS Chemex (ALS) in North Vancouver, BC. In 2008, samples were prepared at ALS in North Vancouver. During the 2010 and 2011 drilling programs, samples were prepared in the ALS sample processing facility in Terrace, BC and during the 2018 program, sample preparation was done at the ALS facility in Whitehorse.

ALS attained ISO 17025 accreditation by Standards Council of Canada for test procedures, including fire assay gold by AA, ICP, and gravimetric finish, multi-element ICP and AA assays for silver, copper, lead, and zinc. In addition, ALS is ISO 9001:2008 registered.

Drill core and blank samples were weighed, dried and crushed to 70% passing 2 mm, with a 250 g split pulverized to 85% passing 75 microns. Gold was analyzed by 30 g fire assay with an AAS finish, and silver was analyzed by ICP with AES finish on a 0.5 g aqua regia digest aliquot. Over-limits were triggered at Au >9 ppm and Ag >80 ppm, resulting in gold re-analysis using a 30 g fire assay with gravimetric finish and silver re-analysis using an ore grade method of aqua regia digest and ICP-AES analysis.

The base metal and 35 element analyses were completed by ICP with AES finish on a 0.5 g aqua regia digest aliquot with the copper, zinc and lead ore grade method triggered at >0.25%. Ore grade analyses for copper, zinc and lead were triggered below the mining cut-offs to improve the precision of the key economic elements.

A comprehensive quality assurance/quality control (QA/QC) program was used to monitor the precision and accuracy of all samples collected during drilling programs conducted from 2008 through to 2018. The various certified standards that were used during these drilling programs are shown in Table 11-1. The grade ranges of these standards reflect those typically present in the deposit.±

Table 11-1: Reference standards used in QA/QC programs

Standard	Mean Grade of Certified Elements ± 2 Standard Deviations				
	Cu (%)	**Zn (%)**	**Ag (g/t)**	**Au (g/t)**	**Pb (%)**
CDN-FCM-3 (1)	0.29 ± 0.02	0.54 ± 0.03	23.6 ± 3.3	0.40 ± 0.07	0.15 ± 0.01
CDN-ME-2 (2)	0.48 ± 0.018	1.35 ± 0.10	14.0 ± 1.3	2.10 ± 0.11	-
CDN-ME-6 (2)	0.61 ± 0.034	0.517 ± 0.04	101 ± 7.1	0.27 ± 0.028	1.02 ± 0.08
CDN-ME-11 (3)	2.44 ± 0.11	0.96 ± 0.06	79.3 ± 6.0	1.38 ± 0.10	0.86 ± 0.10
CDN-ME-18 (3)	1.93 ± 0.09	4.60 ± 0.22	58.2 ± 5.1	0.512 ± 0.070	0.098 ± 0.012
CDN-GS-P3A	-	-	-	0.338 ± 0.022	-
CDN-HZ-3(2)	0.61 ± 0.03	3.16 ± 0.16	27.3 ± 3.2	provisional 0.055 ± 0.010	0.707 ± 0.036
CDN-HZ-2	1.36 ± 0.06	7.2 ± 0.35	61.1 ± 4.1	0.124 ± 0.024	1.62 ± 0.11
CDN-HC-2	4.63 ± 0.26	0.259 ± 0.014	15.3 ± 1.4	1.67 ± 0.12	0.48 ± 0.04
CDN-HLHC	5.07 ± 0.27	2.35 ± 0.11	111.0 ± 8.6	1.97 ± 0.22	0.17 ± 0.01
CDN-HLLC	1.49 ± 0.06	3.01 ± 0.17	65.1 ± 6.7	0.83 ± 0.12	0.29 ± 0.03
OREAS-621 (4)	0.363 ± 0.016	5.22 ± 0.27	69.2 ± 5.3	1.25 ± 0.084	1.36 ± 0.078
OREAS-623 (4)	1.73 ± 0.0128	1.03 ± 0.06	20.4 ± 2.12	0.827 ± 0.78	0.250 ± 0.014
OREAS-624 (4)	3.10 ± 0.158	2.40 ± 0.156	45.3 ± 2.52	1.16 ± 0.106	0.624 ± 0.038

(1) 2008 use only. (2) Not used in 2008. (3) 2011 use only. (4) 2018 use.

Note: CDN Resource Labs, Delta BC. HL CRM are High Lake VMS deposit material. FCM are Campo Morado VMS material. HC and HZ CRM are derived from VMS deposit material. ME are Lookout, Niblack VMS material. GS-P3A is made from Bald Mountain, NV material (Carlin style mineralization). OREAS series sourced material from Gossan Hill Deposit.

The material used for blanks was derived from clean limestone or felsic rock purchased from a landscape supply company. This material was found to be material that was below detection for gold and silver and contained very low concentrations of copper, zinc, and lead.

Typically, two standards were blindly inserted for every 20 samples submitted to the laboratory, alternating between a blank and a certified standard reference material (CRM), with placement and type at the discretion of the logging geologist. A minimum of one blank and one CRM was included in each drillhole. An additional laboratory duplicate sample was included with every 20 samples taken. The laboratory duplicate samples alternated between a pulp duplicate sample and a coarse reject duplicate sample as selected by the core logger.

Typically, no field duplicates (taken from the other half of the core sample) were included, except in the 2018 drill program.

The results from the laboratory for the blanks and CRM were monitored on an ongoing basis during the drilling programs. CRM values outside of three standard deviations were considered failures, as were more than two consecutive values between two and three standard deviations above or below the mean.

If more than two control samples in a work order returned unacceptable values, the entire work order was rerun. If two or fewer control samples failed, the 20 samples in the analytical "batch" were rerun. In the case of a blank fail, all samples in the batch were reprocessed from coarse reject material.

Table 11-2 summarizes the control samples that were included with the drill core samples for the drilling conducted by Capstone and Kutcho Copper in 2008, 2010, 2011 and 2018.

Table 11-2: 2008 to 2018 Kutcho Project summary of quality control checks inserted with sample data

		Year							
		2008		2010		2011		2018	
Total samples collected		2,973		1,171		379		1,154	
CRM used		HLLC	34	HLLC	4	HLLC	8	621	13
		HLHC	22	HLHC	7	HLHC	2	623	17
		HC-2	26	HC-2	10	P3A	1	624	16
		HZ-2	28	HZ-2	18	ME-18	1		
		FMC-3	37	ME-2	19	ME-11	2		
				ME-6	13	ME-6	1		
				HZ-3	1	HZ-3	2		
Total CRM		147		72		17		63	
Blanks		77		60		15		15	
Paired data	Coarse reject duplicate	72		44		14		46	
	Pulp reject duplicate	72		47		18		38	
Total QC samples		368		223		64		210	
Frequency (%)		12		19		17		19.2	
Umpire checks (%)		2		3		0		3.5	

As an additional check on the sample assay data, representative suites of samples were selected and analyzed at a second external laboratory. At the end of the drill programs in 2008 and 2010, 2% to 3% of the sample pulp rejects produced at ALS were selected across a variety of grade ranges, but most were primarily within the ranges of potential economic interest. The pulp rejects from 2008 and 2010 were submitted to IPL Inspectorate in Vancouver, BC, together with one CRM added per every 10 samples submitted. All the samples were renumbered with new, sequential sample identifications and identical analytical methods were used.

Following the 2018 drilling program, approximately 3.5% of the pulps from that program were reanalyzed at Bureau Veritas located in Burnaby, BC. The results of all these re-assay programs show that the original grades can be reproduced at external umpire laboratories.

11.3 Quality Assurance and Quality Control of Pre-2008 Drill Programs

This section of the report describes the QA/QC programs prior to 2008 drill program.

WKM used a QA/QC program to monitor the results of its sampling that included the insertion of certified standards, blanks and duplicates analyzed at a second umpire laboratory (Eco Tech Laboratory Ltd in Kamloops, BC). There were no issues identified during any of its drill programs, with the exception of some minor contamination in some blank samples. The level of potential contamination was not considered significant, and the origin could be attributed to the source of the blank material that was used (produced from locally sourced limestone and felsic rocks that were believed to be barren of mineralization).

The October 2007 technical report, produced by Wardrop for WKM (Wardrop, 2007), states that *"data verification and quality control and quality assurance for the Kutcho Project was studied extensively by AMEC (Chong, 2007) who worked closely with company personnel in order to determine that all historical data was suitable for use in resource and reserve estimates and that all modern data met high standards for quality assurance and quality control"*.

11.4 Quality Assurance and Quality Control of 2008 Drill Program

Performance of the copper blanks was acceptable. Two failures were found to be caused by sample data entry errors by the core logger, and three failures were found to be caused by contamination from mineralized samples during processing at the laboratory. This was corrected and rectified by:

11) The core loggers were reminded to take care in recording sample types.

12) The laboratory personnel were advised of the contamination issue during the program.

13) Lab personnel were reminded of the standard operating procedures.

The overall performance of the blanks was acceptable for gold with very little suggestion of contamination due to inadequate cleaning between samples during the crushing and pulverization stages. One failure was re-assayed from coarse reject material, and it returned a marginal fail, suggesting the failure was due to inadequate cleaning between samples during the coarse crush. The laboratory was advised of the two failures, and it subsequently reminded staff of protocols for cleaning between samples. The blank had also failed for copper; the copper reanalysis from coarse reject was acceptable. A marginal failure after a high-grade interval was not re-assayed.

Blank performance for silver was acceptable, and very little between-sample contamination was evident.

Performance of the CRMs across all grade ranges was acceptable; the CRMs did show periods both above and below the mean but within acceptable standards.

Reproducibility of copper and silver results in duplicate samples prepared from coarse reject materials was excellent. Reproducibility of gold results in duplicate samples prepared from coarse reject materials was excellent, up to 1.5 g/t Au; higher gold results are more variable. This characteristic is likely due to inherent properties of the Kutcho deposits. The coarse nature of gold makes it difficult to reproduce results in the higher grade ranges.

Reproducibility of copper and silver results in duplicate samples prepared from pulp reject material was very good.

Reproducibility of gold results in pulp duplicates was acceptable.

At the end of the drill program in 2008, 2% of the sample pulp rejects at ALS were selected across a variety of grade ranges representative of the location and targets tested at the Project. The pulp rejects were submitted to IPL Inspectorate in Vancouver, BC, with one CRM added to every 10 samples. All the samples were renumbered with new, sequential sample identifications. Identical analytical methods were used. Results were shown to be acceptably reproducible.

For additional information regarding performance of quality control samples in 2008, please refer to "Preliminary Economic Assessment Revised Mining Option Kutcho Project British Columbia, JDS Energy & Mining Inc., July 2010".

11.5 Quality Assurance and Quality Control of 2010 Drill Program

This section of the report summarises the QA/QC program results during the 2008 drill program. Additional information regarding performance of quality control samples in 2008 is included in the Technical Report Preliminary Economic Assessment Revised Mining Option Kutcho Project British Columbia, JDS Energy & Mining Inc., July 2010.

Blank performance for copper was excellent (Figure 11-1). Five values that exceeded the upper threshold limit were not re-analyzed as the results were attributed to minor contamination after very high-grade samples exceeded 4% Cu. One failure was re-assayed and returned a similar value, indicating the contamination occurred at the coarse-crush stage of sample preparation after an 8% Cu sample. These instances of contamination are not considered systematic; however, there were many other very high samples submitted with well-performing blanks. The laboratory was notified of these instances and, consequently, reminded workers to take care during sample preparation and follow all standard operating procedures.

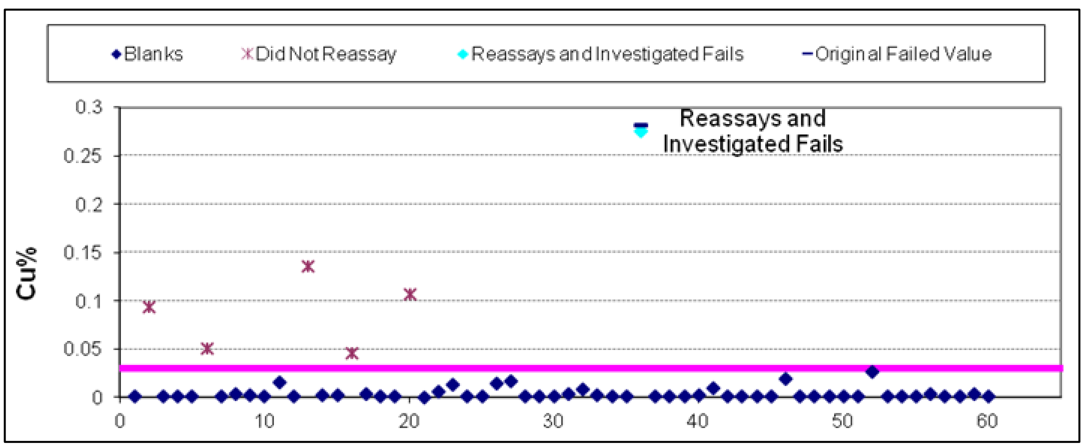


Figure 11-1: Copper Blank Performance from QA/QC in 2010 Drilling

(Source: JDS, 2011)

Blank performance for gold was excellent (Figure 11-2). Two values marginally exceeded the blank warning performance gate and were not re-assayed. Note that one of the two marginal fails is the same sample re-assayed for a copper blank failure.

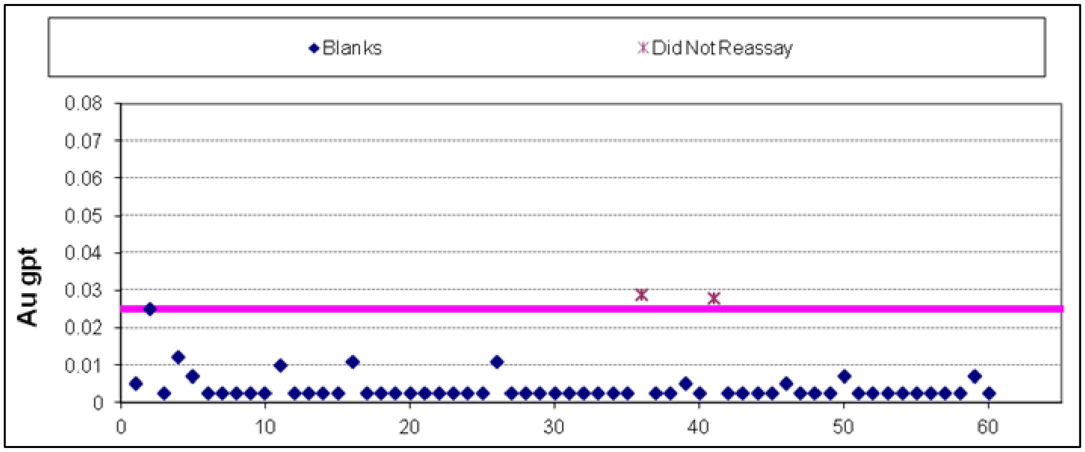


Figure 11-2: Gold Blank Performance from QA/QC in 2010 Drilling

(Source: JDS, 2011)

Performance of CRM HC-2 was very good for copper (Figure 11-3) and all copper values fell within acceptable limits.

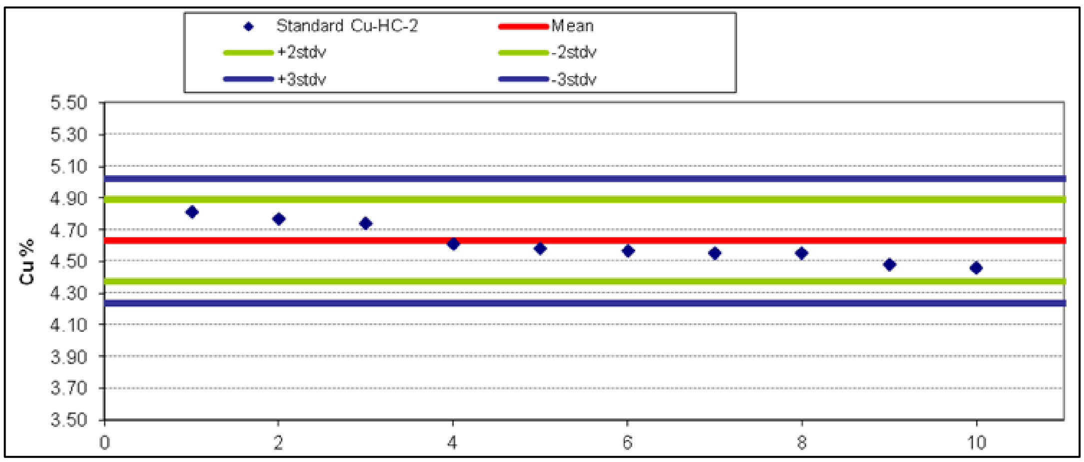


Figure 11-3: Copper CRM HC-2 Performance from QA/QC in 2010 Drilling

(Source: JDS, 2011)

Performance of CRM HZ-2 was very good for copper (Figure 11-4) with no values occurring outside of the acceptable limits.

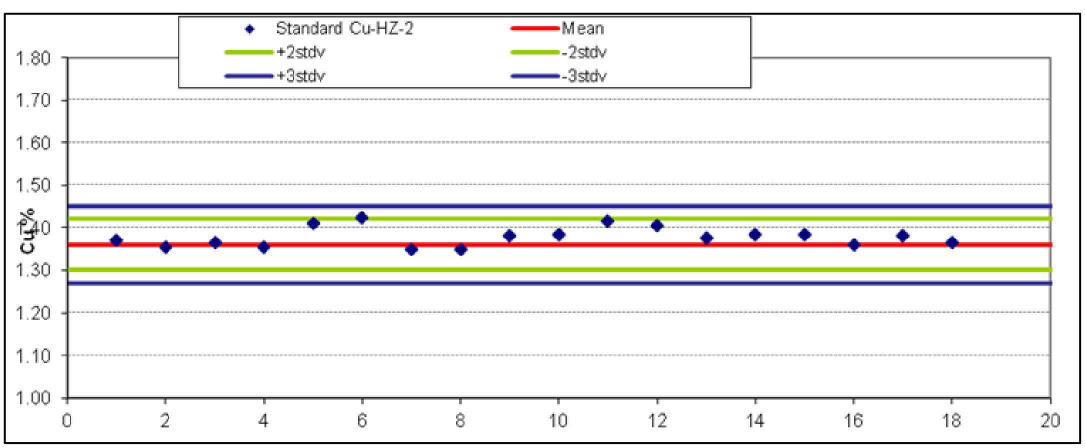


Figure 11-4: Copper CRM HZ-2 Performance from QA/QC in 2010 Drilling

(Source: JDS, 2011)

Performance of CRM HZ-2 for zinc (Figure 11-5) was within acceptable limits.

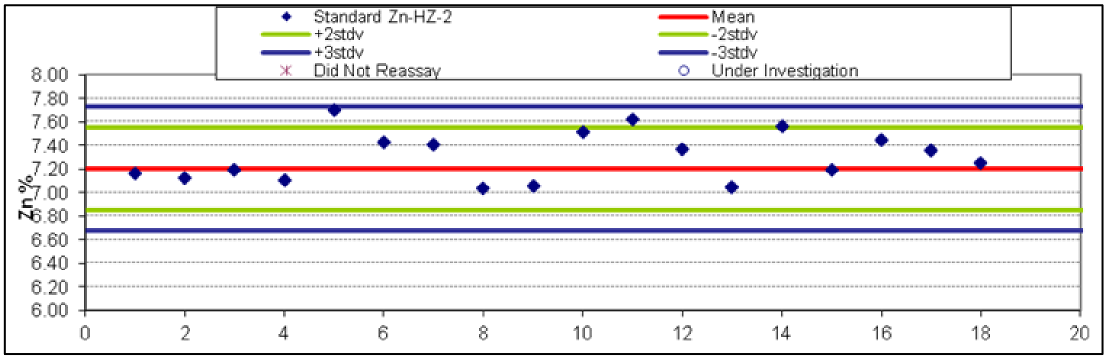


Figure 11-5: Zinc CRM HZ-2 Performance from QA/QC in 2010 Drilling

(Source: JDS, 2011)

Performance of the CRM ME-2 for zinc was acceptable (Figure 11-6). A single failed value was corrected upon reanalysis of a batch of 20 samples (the CRM also failed for copper and silver). Investigation into the failure revealed the analytical instrument had skipped four samples in the batch; ALS internal data review processes were reviewed with laboratory staff to ensure the error did not recur.

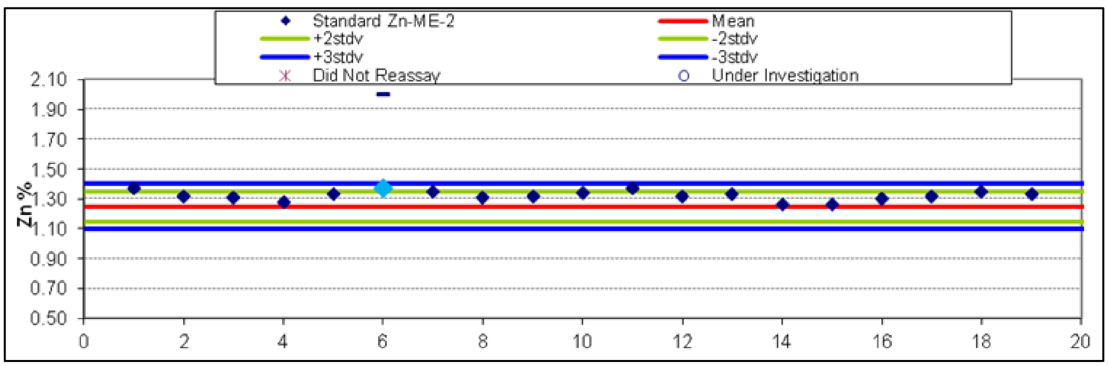


Figure 11-6: Zinc CRM ME-2 Performance from QA/QC in 2010 Drilling

(Source: JDS, 2011)

Performance of CRM HZ-2 for silver (Figure 11-7) was within acceptable limits throughout the drill program.

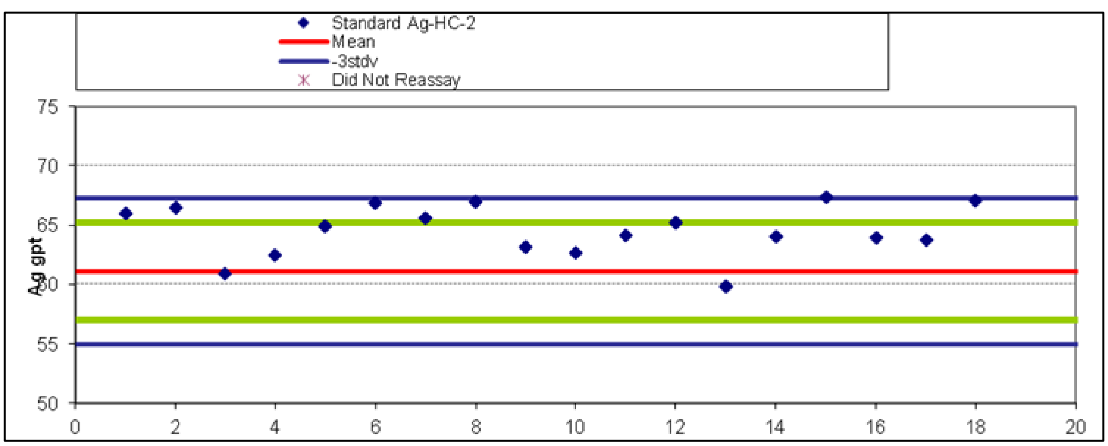


Figure 11-7: Silver CRM HZ-2 Performance from QA/QC in 2010 Drilling

(Source: JDS, 2011)

Performance of the CRM ME-2 for gold was acceptable (Figure 11-8). One marginally high failure occurred just outside three standard deviations above the mean performance gate, but it was not re-analyzed.

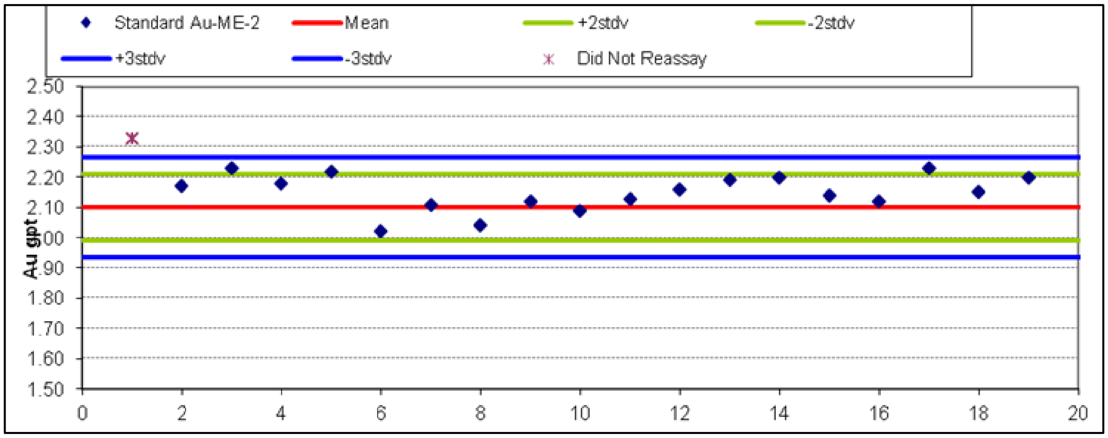


Figure 11-8: Gold CRM ME-2 Performance from QA/QC in 2010 Drilling

(Source: JDS, 2011)

The XY plots in Figure 11-9 and Figure 11-10 show comparisons of pulp duplicate samples that were both run in sequence at the same laboratory (ALS).

Reproducibility of copper in pulp reject duplicates was acceptable for the 2010 drill campaign Figure 11-9.

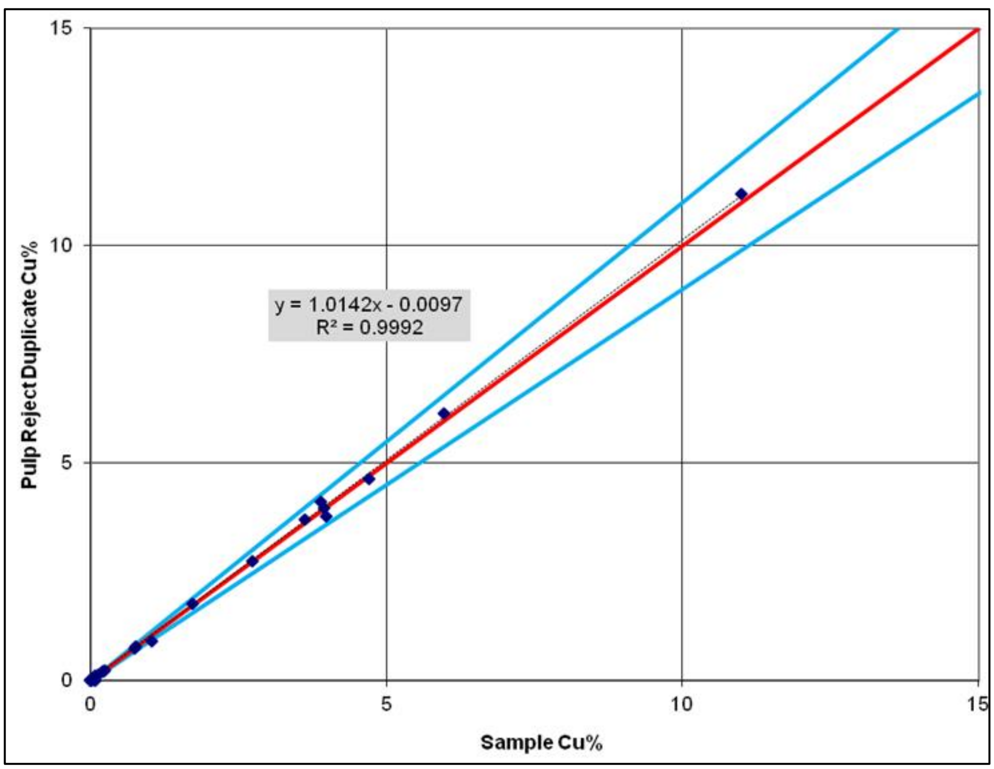


Figure 11-9: Sample Cu% vs Pulp Reject Duplicate Cu% from QA/QC in 2010 Drilling

(Source: JDS, 2011)

Reproducibility of zinc in pulp reject duplicates was acceptable for the 2010 drill campaign (Figure 11-10).

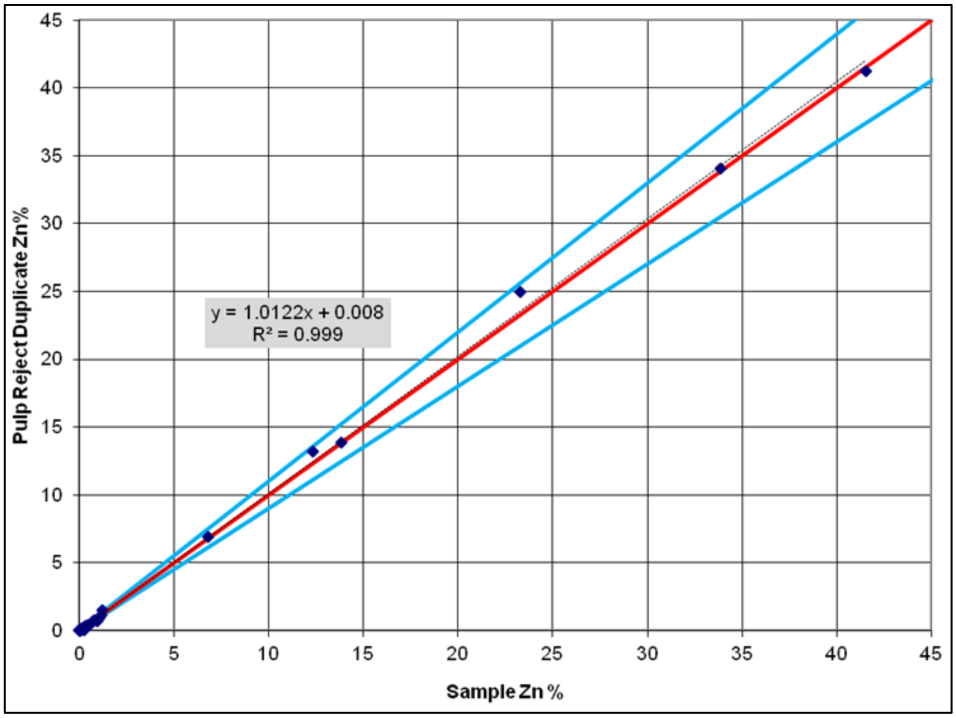


Figure 11-10: Sample Zn% vs Pulp Reject Duplicate Zn% from QA/QC in 2010 Drilling

(Source: JDS, 2011)

The XY plots in Figure 11-11 and Figure 11-12 show comparisons of coarse duplicate samples that were both run in sequence at the same laboratory (ALS). Reproducibility of copper in coarse reject duplicates was acceptable for the 2010 drill campaign (Figure 11-11). All duplicate pairs plot within the ±10% performance gate.

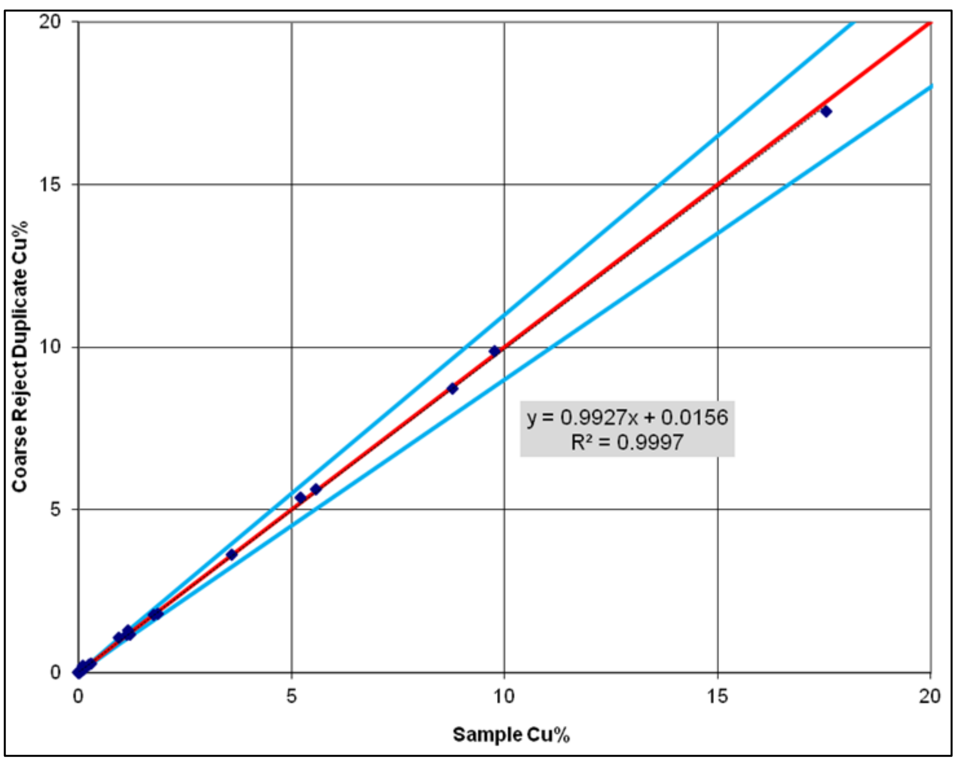


Figure 11-11: Sample Cu% vs Coarse Reject Duplicate Cu% from QA/QC in 2010 Drilling

(Source: JDS, 2011)

Reproducibility of zinc in coarse reject duplicates was acceptable for the 2010 drill campaign (Figure 11-12). Two of the duplicate pairs plot below the ±10% performance gate, indicating the original results were higher grade than the duplicate results.

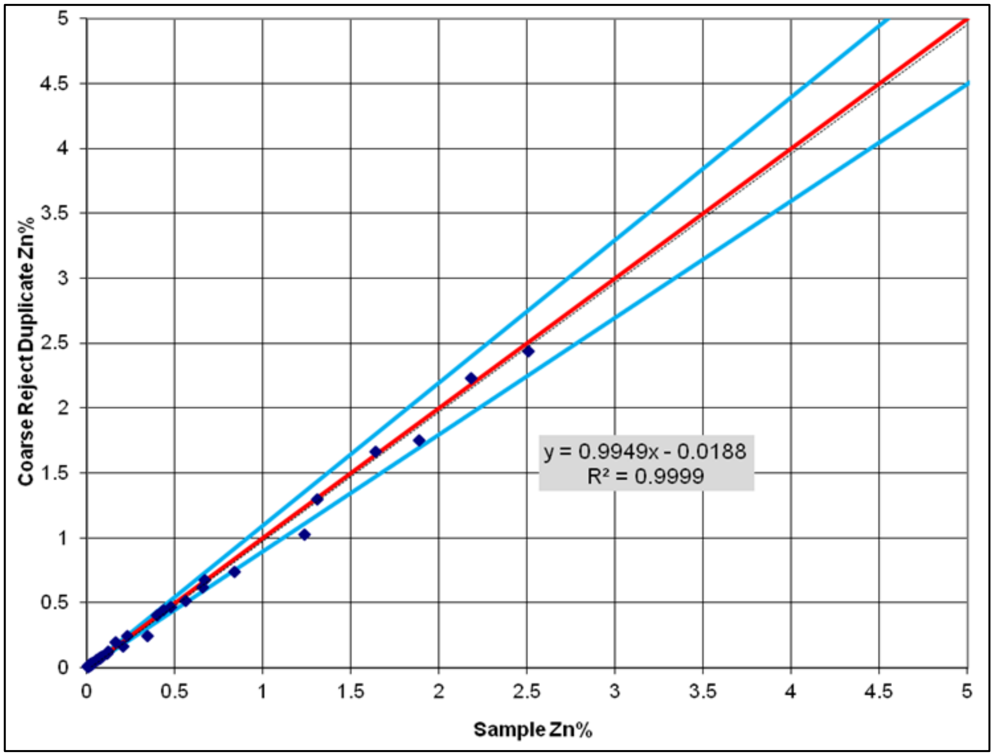


Figure 11-12: Sample Zn% vs Coarse Reject Duplicate Zn% from QA/QC in 2010 Drilling
(Source: JDS, 2011)

A comparison of analyses between laboratories was conducted at the end of the drill program, where 3% of the sample pulp rejects at ALS were selected across a variety of grade ranges and submitted to IPL Inspectorate in Vancouver, BC, with one CRM added every 15 samples. All the samples were renumbered with new, sequential sample identifications, and identical analytical methods were used at both laboratories. Results for the check analysis demonstrated a very strong correlation between the original sample and duplicate samples for copper, zinc, lead, and silver. Gold values in the between-lab check samples correlated well but were not as reproducible.

11.6 Quality Assurance and Quality Control of 2011 Drill Program

This section of the report shows some examples of the results of the QA/QC samples submitted during the 2011 drilling program. Additional control charts from the 2011 drilling program are included in the previous PFS technical report (JDS, 2017).

Performance of blanks was acceptable for copper and gold. Figure 11-13 and Figure 11-14 show QA/QC performance of blanks with the warning line in green (five times the detection limit) and the failures line in blue (10 times the detection limit). Failures that were not re-assayed are shown as red stars. Concerns were communicated to ALS, who reviewed the between-sample cleaning protocols with preparation personnel.

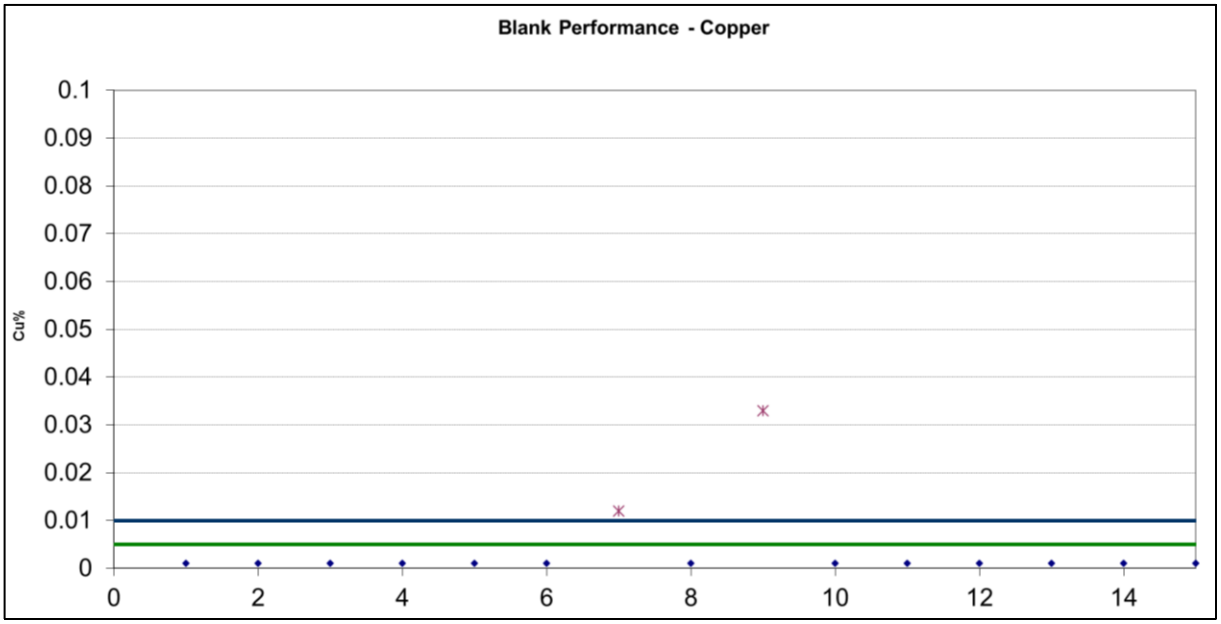


Figure 11-13: QA/QC Performance of Blanks (Copper) from QA/QC in 2011 Drilling
(Source: JDS, 2011)

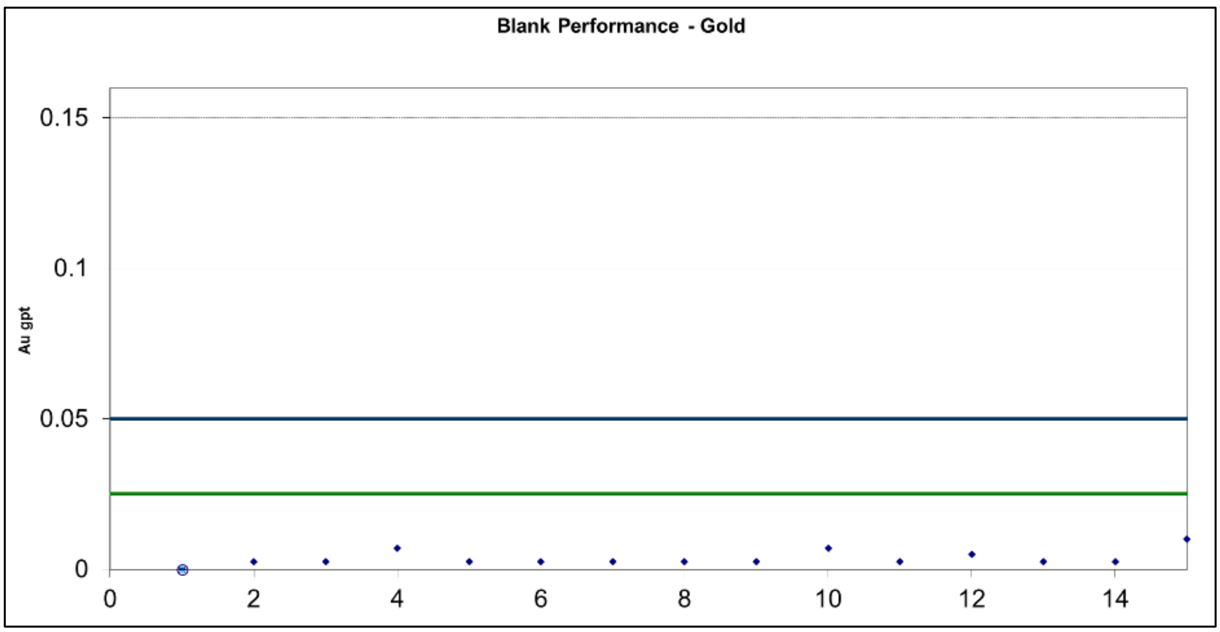


Figure 11-14: QA/QC Performance of Blanks (Gold) from QA/QC in 2011 Drilling
(Source: JDS, 2011)

Performance of CRM was acceptable for copper, zinc, silver, and gold in all cases. Examples from CRM HLLC are shown in Figure 11-15 to Figure 11-18. The mean grade of the CRM is shown as a red line, two standard deviations from the mean is shown as a green line, and three standard deviations from the mean is shown as a dashed blue line. One failure during the program for zinc in CRM HZ-3 was resolved upon re-assay of the batch of affected samples.

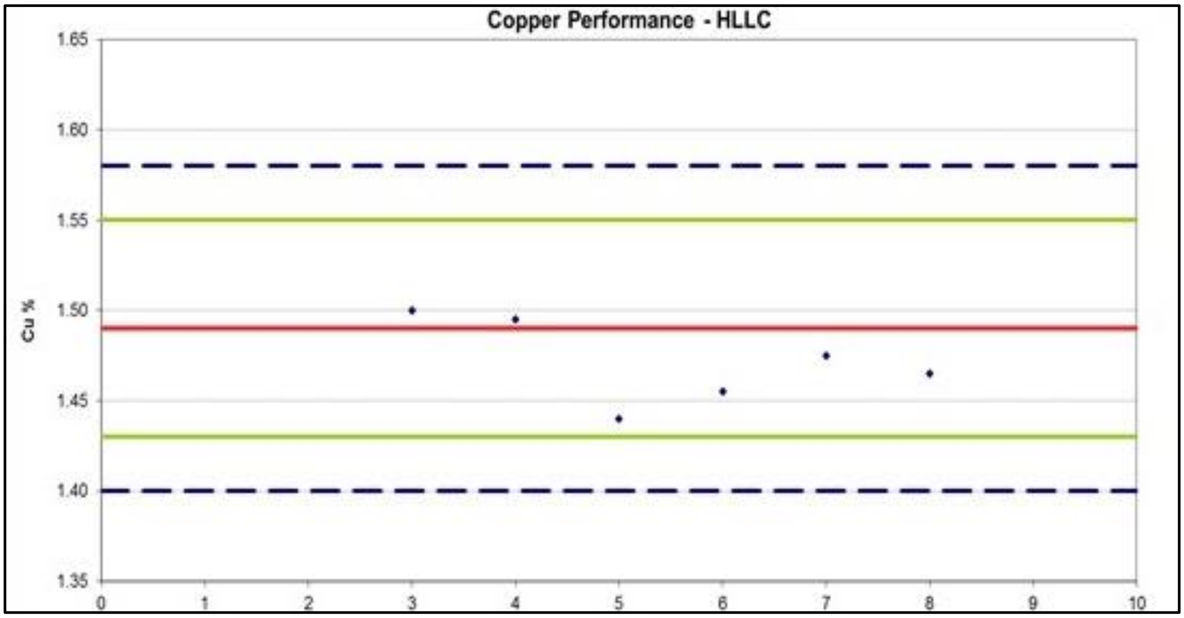


Figure 11-15: Performance of CRM HLLC from QA/QC in 2011 Drilling Cu%
(Source: JDS, 2011)

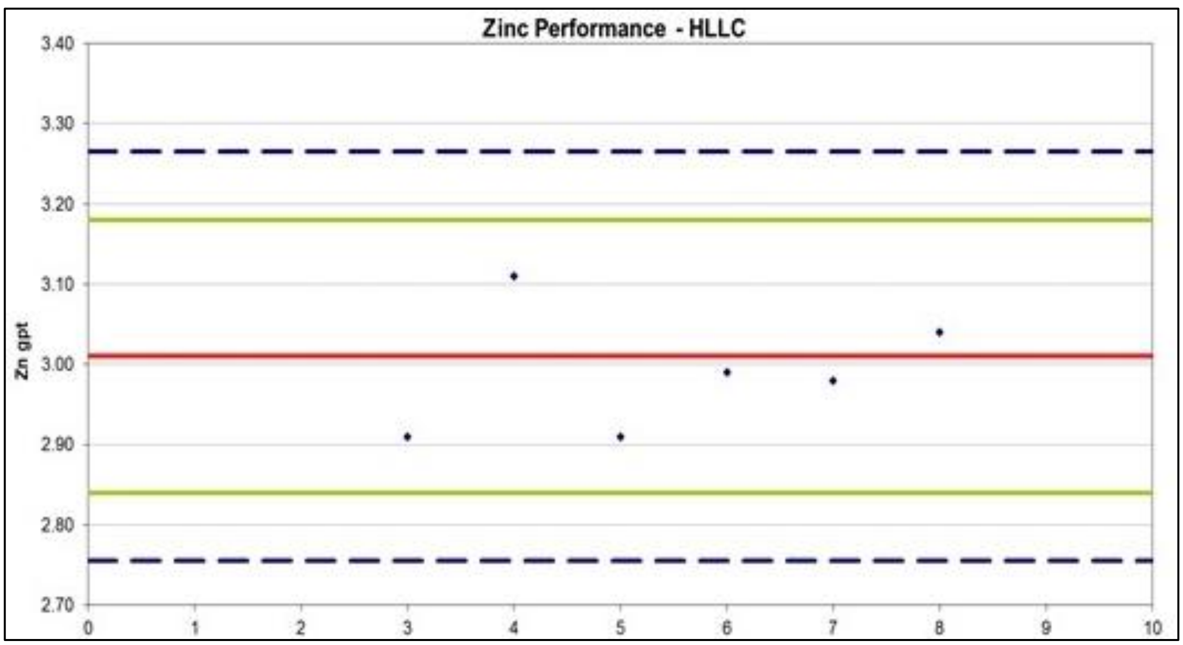


Figure 11-16: Performance of CRM HLLC from QA/QC in 2011 Drilling Zn%
(Source: JDS, 2011)

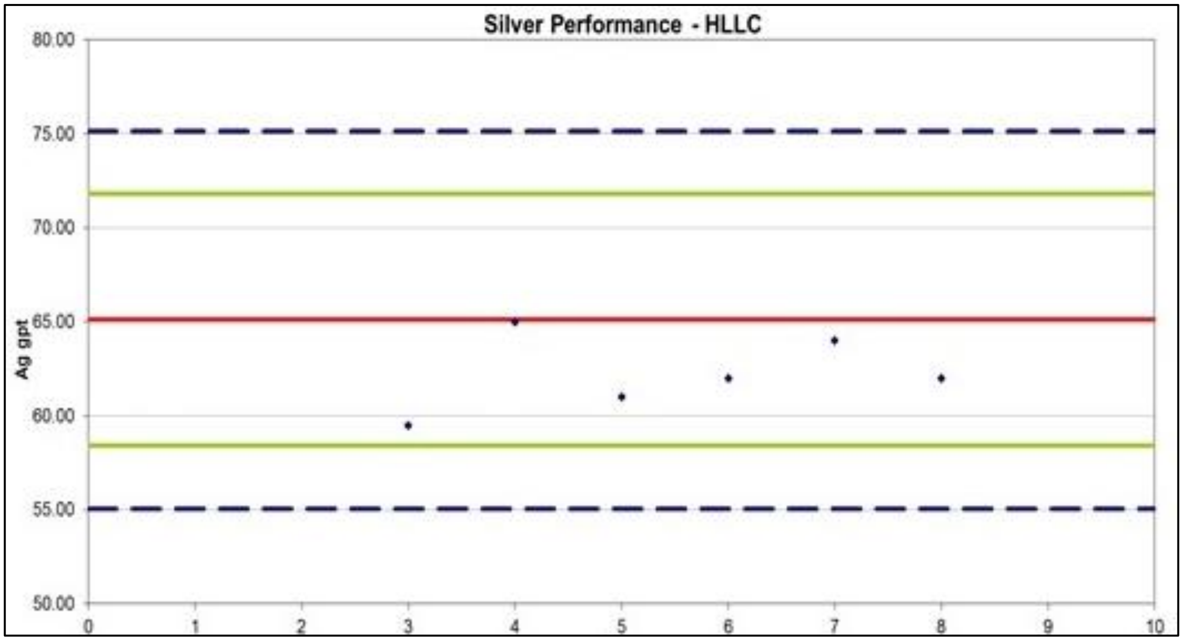


Figure 11-17: Performance of CRM HLLC from QA/QC in 2011 Drilling Ag ppm

(Source: JDS, 2011)

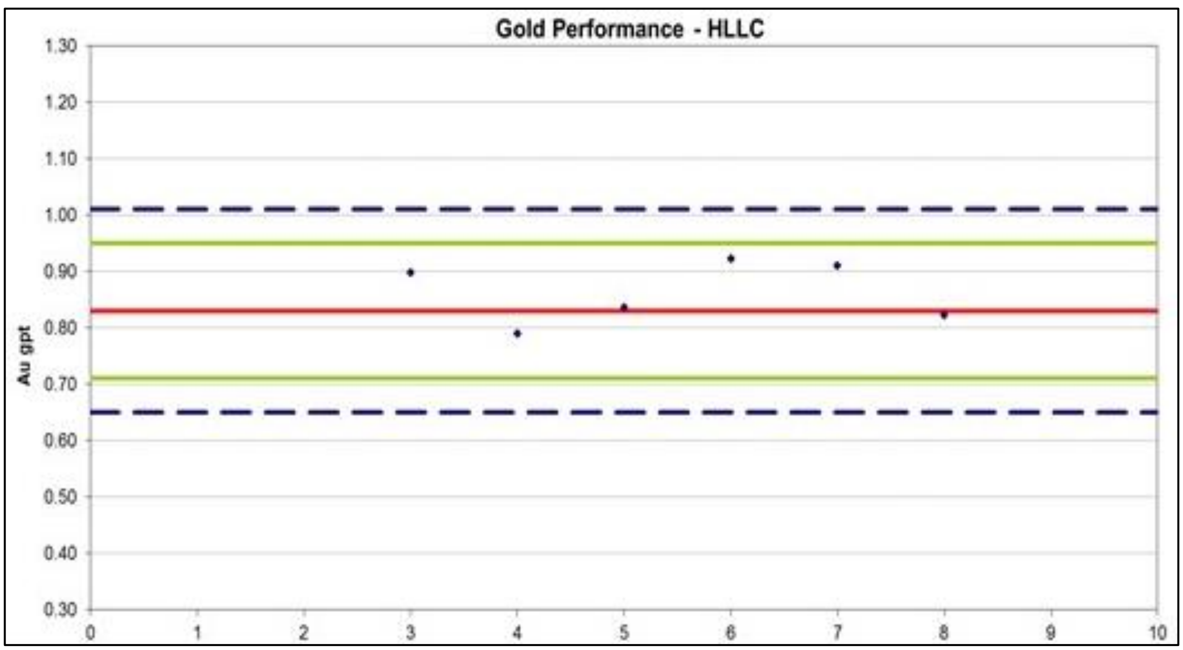


Figure 11-18: Performance of CRM HLLC from QA/QC in 2011 Drilling Au ppm

(Source: JDS, 2011)

11.7 Quality Assurance and Quality Control of 2018 Drill Program

This section of the report summarizes the results of the QA/QC samples submitted during the 2018 drilling program.

A total of 1,154 individual samples were collected for analyses during the 2018 drilling program, including 913 samples taken from the 37 new holes drilled during the program, as well as an additional 241 samples collected from previous drillholes where shoulder samples, those that bracket the mineralized zones, were not originally taken. These shoulder samples generally do not impact the estimate of mineral resources, but they do provide information on the nature of dilution material that may be encountered during mining. Included with the 1,154 samples submitted, there were an additional 207 samples derived from CRM or blanks, plus 40 preparation (pulp) duplicates, 38 coarse duplicates, and 30 field duplicates.

Blank material used during the 2018 program consisted of coarse crushed granite rock sourced directly from Cox Station Quarry in Abbotsford, BC. Fifteen samples of this material were submitted to ALS Minerals (ALS) in North Vancouver, BC to verify that the rock did not contain any base or precious metals of interest. Analytical results from the 15 blank samples show the material is below detection for gold and silver and contains very low concentrations of copper, zinc and lead relative to economic grades at the Kutcho deposits.

A total of 210 QA/QC samples (CRMs, blanks, field and prep duplicates) were submitted with the 2018 drill core samples, comprising 18.7% of the total samples submitted for this portion of the program. Just under half of the 2018 drilling QA/QC samples comprised blank samples because of multiple visually high-grade mineralized zones requiring the insertion of successive blank samples to monitor potential contamination during sample processing.

The performance of copper in blanks is shown in Figure 11-19. A small number of blanks exceeded the blank warning performance gate, but they were not re-analyzed; the results were attributed to minor contamination after, and between, very high-grade samples, some exceeding 4% Cu.

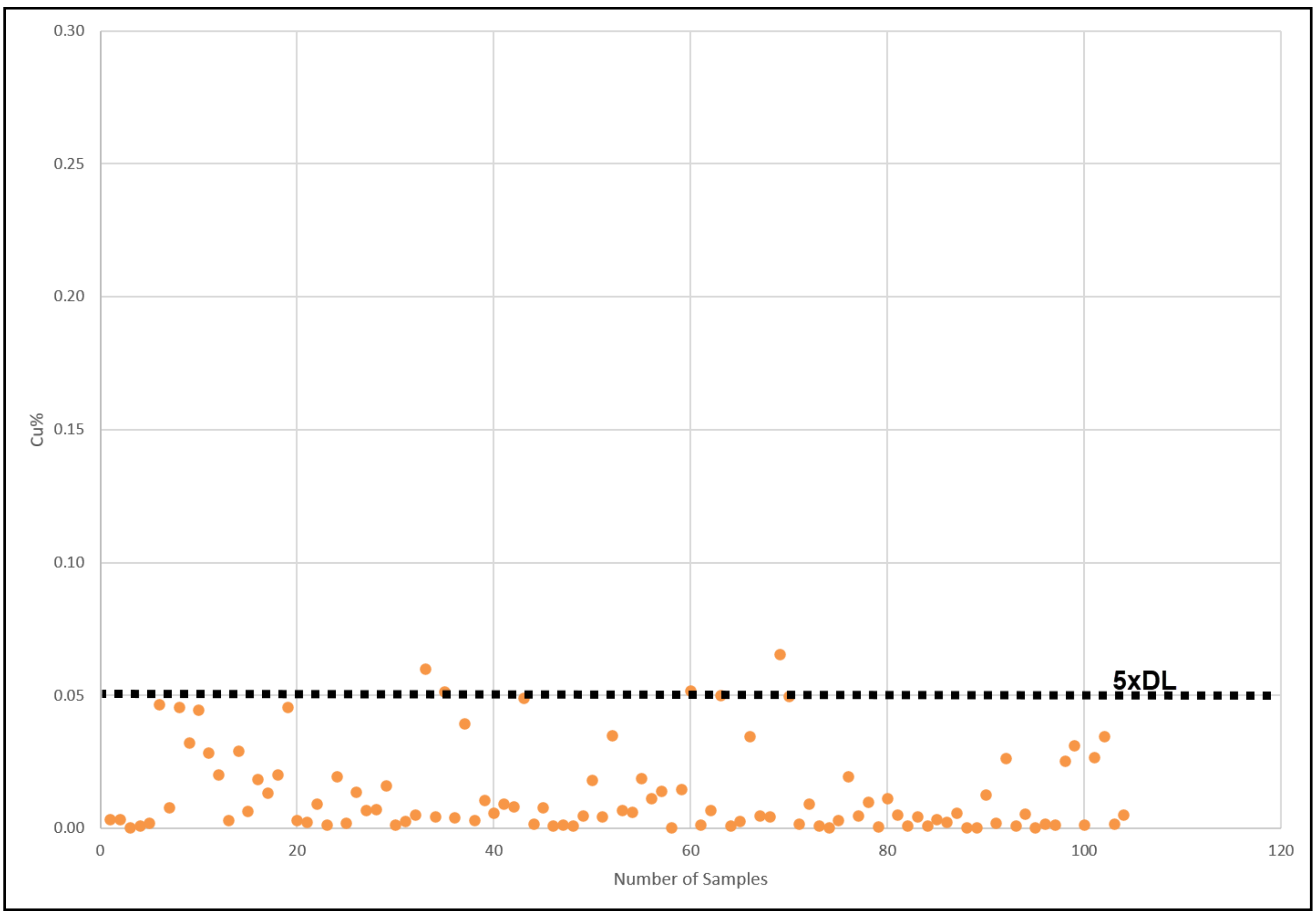


Figure 11-19: Copper Blank Performance from QA/QC in 2018 Drilling

(Source: SIM Geological, 2021)

In some instances where two blanks were inserted after a high-grade sample, the second blank did not exceed the blank warning performance gate; this indicated that the subsequent sample was not contaminated. The contamination most likely occurred at the coarse-crush stage of sample preparation. These instances of contamination are not systematic, as there were many instances where blanks inserted following high-grade samples did not show elevated results. The laboratory was notified of these instances and consequently addressed them in its sample preparation procedures. The magnitude of this potential contamination is relatively small in relation to copper grades typically present in the mineralized zones at Kutcho. Overall, the copper blank performance was deemed acceptable.

The performance of zinc in blanks (Figure 11-20) is similar to that shown for copper, with several samples exceeding the threshold limit attributed to possible contamination from previous high-grade samples, or the result of low zinc concentrations present in the locally sourced blank material. The magnitude of these zinc failures is not considered significant. There were no failures recorded for silver or gold in the blanks submitted during the program.

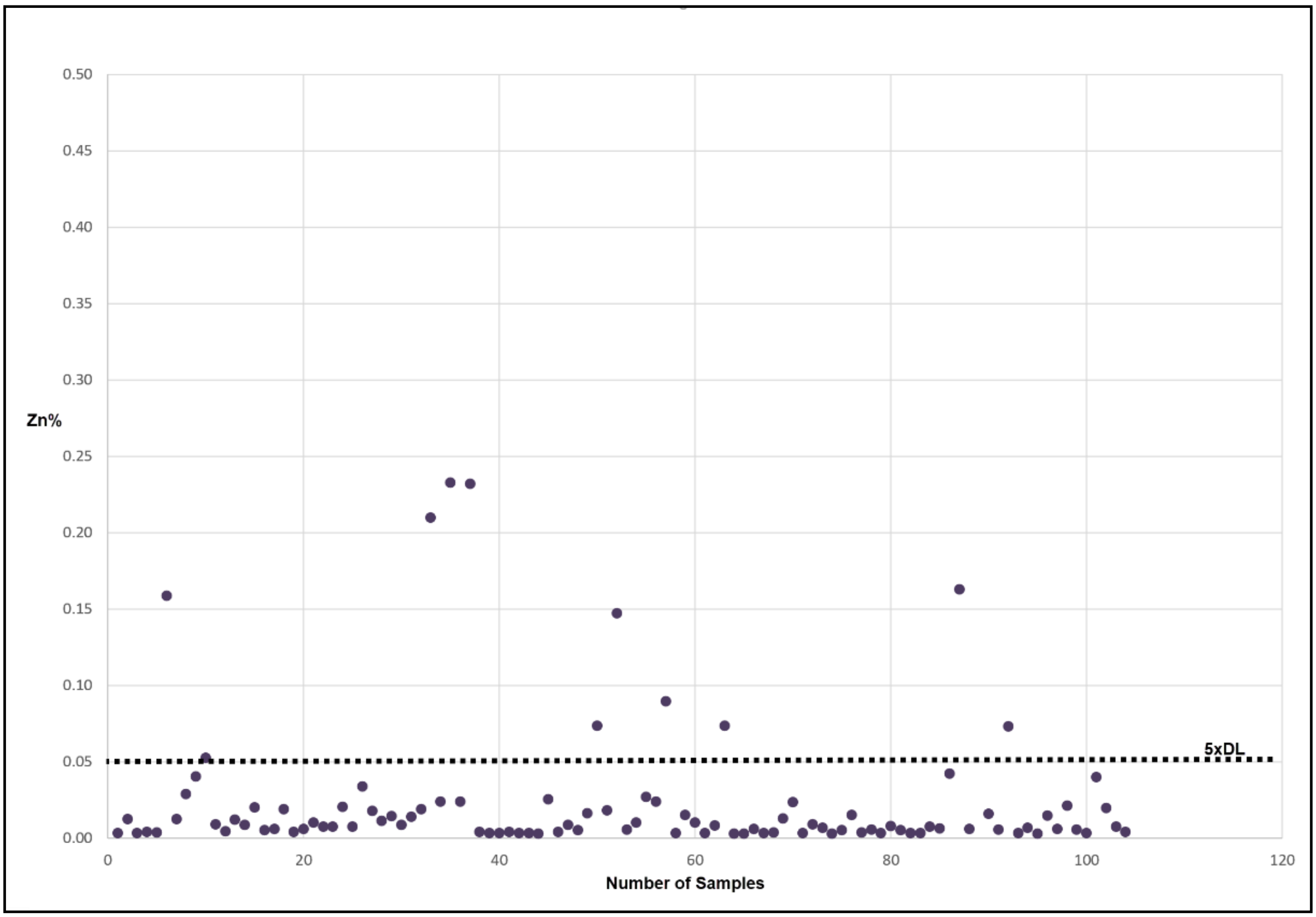


Figure 11-20: Zinc Blank Performance from QA/QC in 2018 Drilling
(Source: SIM Geological, 2021)

Performance of CRM throughout the 2018 drilling program was acceptable for copper, zinc, silver, and gold, and all results returned within ±10% of the expected values except one gold assay which returned lower than expected; this was not considered significant. Examples showing the results for copper and zinc in the three CRM standards used during the program are shown in Figure 11-21 and Figure 11-22.

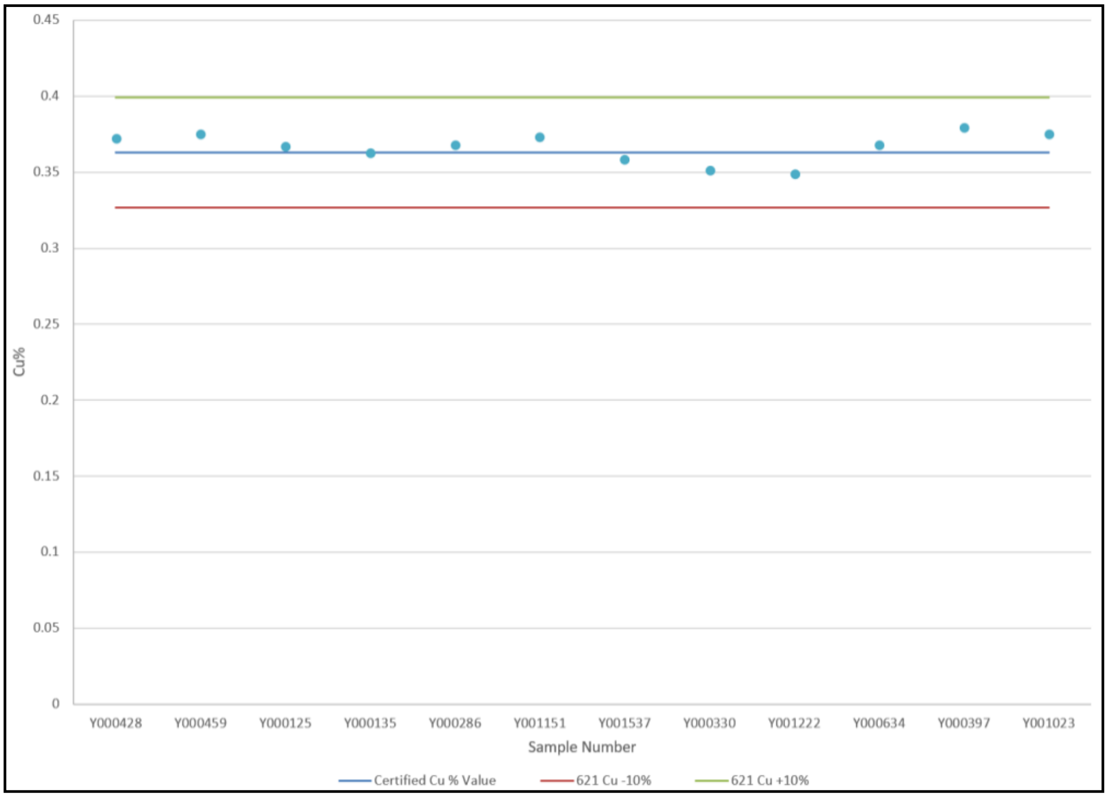


Figure 11-21: Performance of Copper in CRM OREAS 621 from QA/QC in 2018 Drilling
(Source: SIM Geological, 2021)

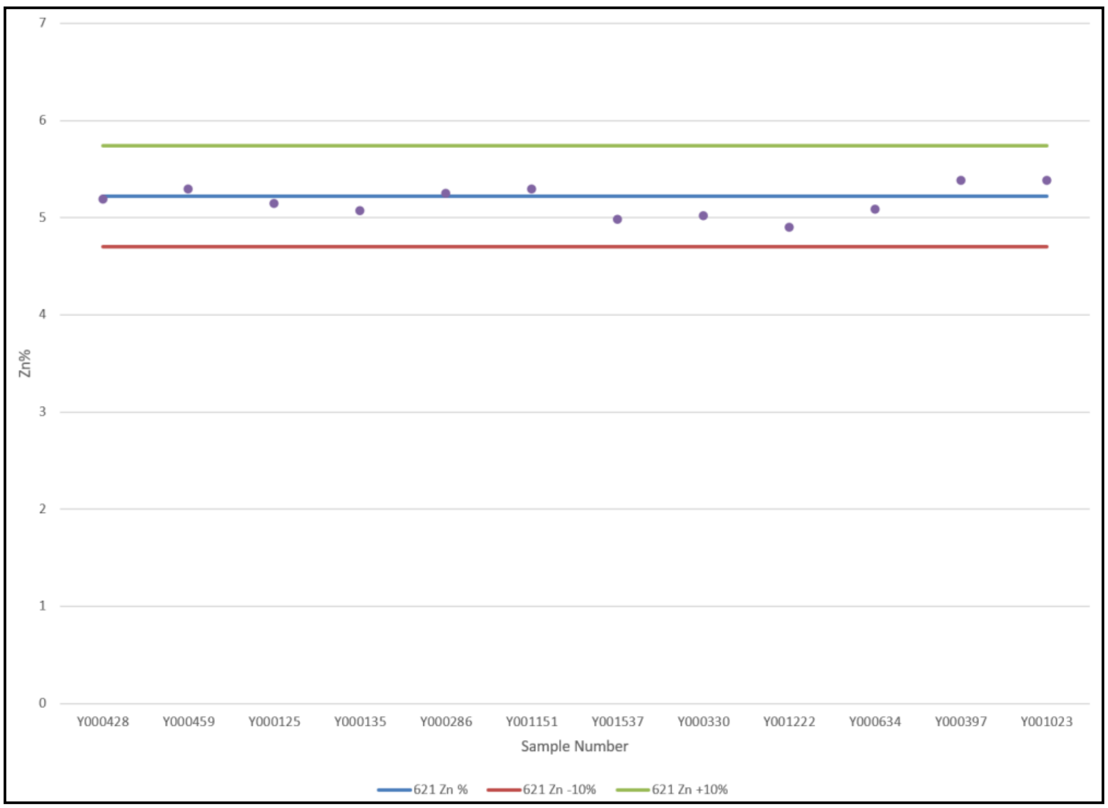


Figure 11-22: Performance of Zinc in CRM OREAS 621 from QA/QC in 2018 Drilling
(Source: SIM Geological, 2021)

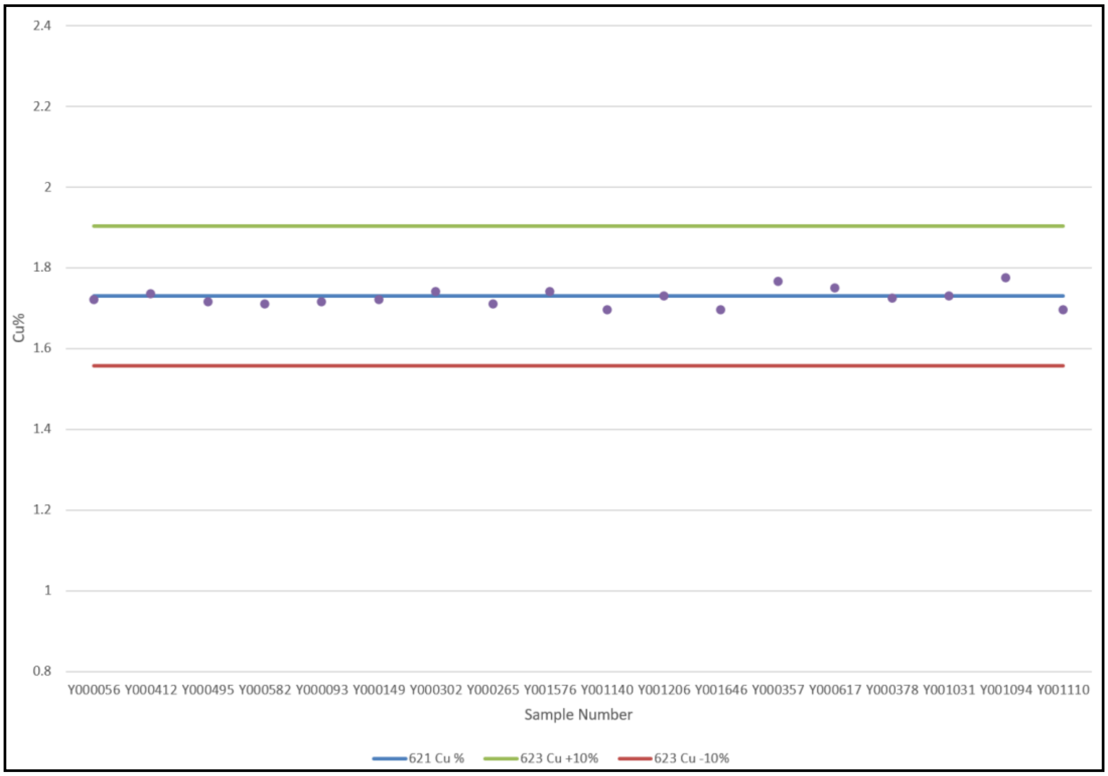


Figure 11-23: Performance of Copper in CRM OREAS 623 from QA/QC in 2018 Drilling
(Source: SIM Geological, 2021)

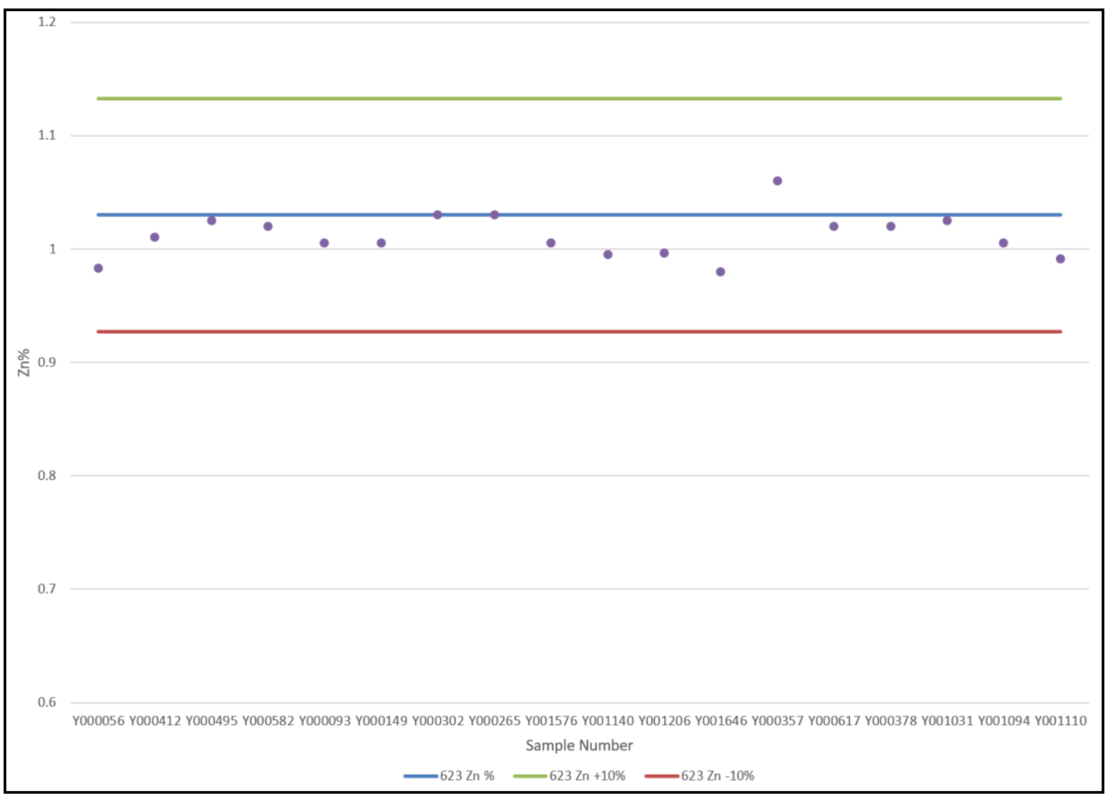


Figure 11-24: Performance of Zinc in CRM OREAS 623 from QA/QC in 2018 Drilling
(Source: SIM Geological, 2021)

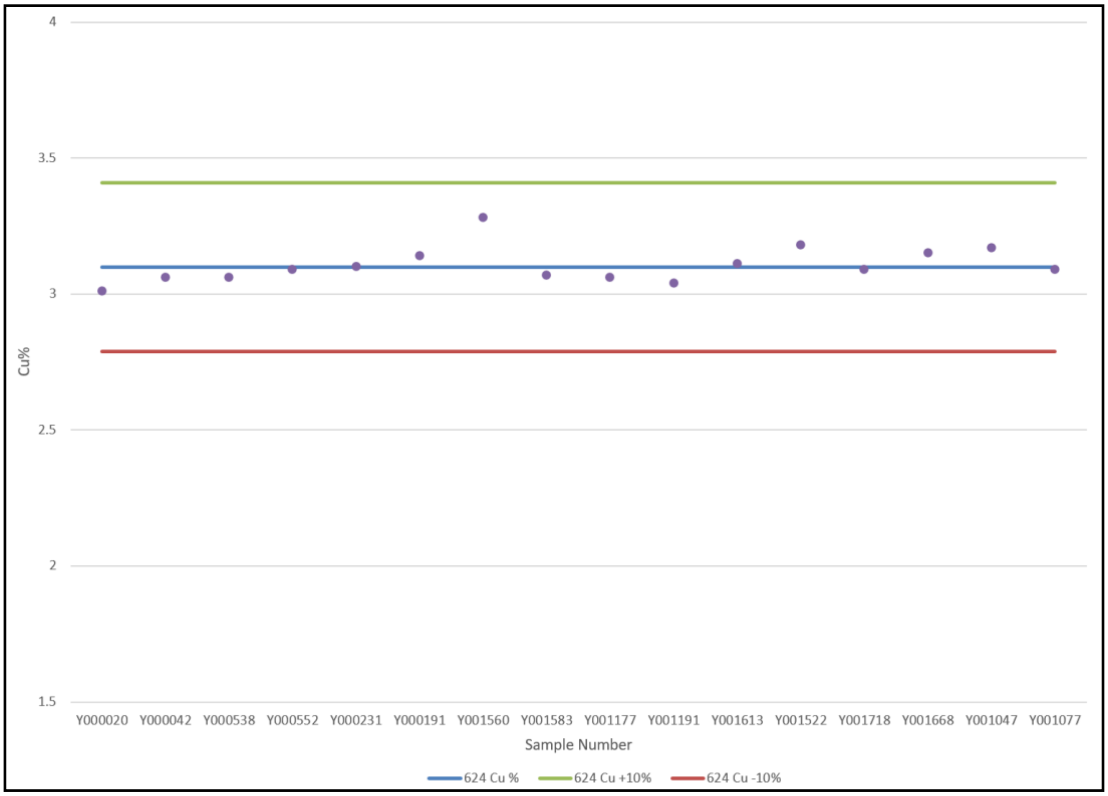


Figure 11-25: Performance of Copper in CRM OREAS 624 from QA/QC in 2018 Drilling
(Source: SIM Geological, 2021)

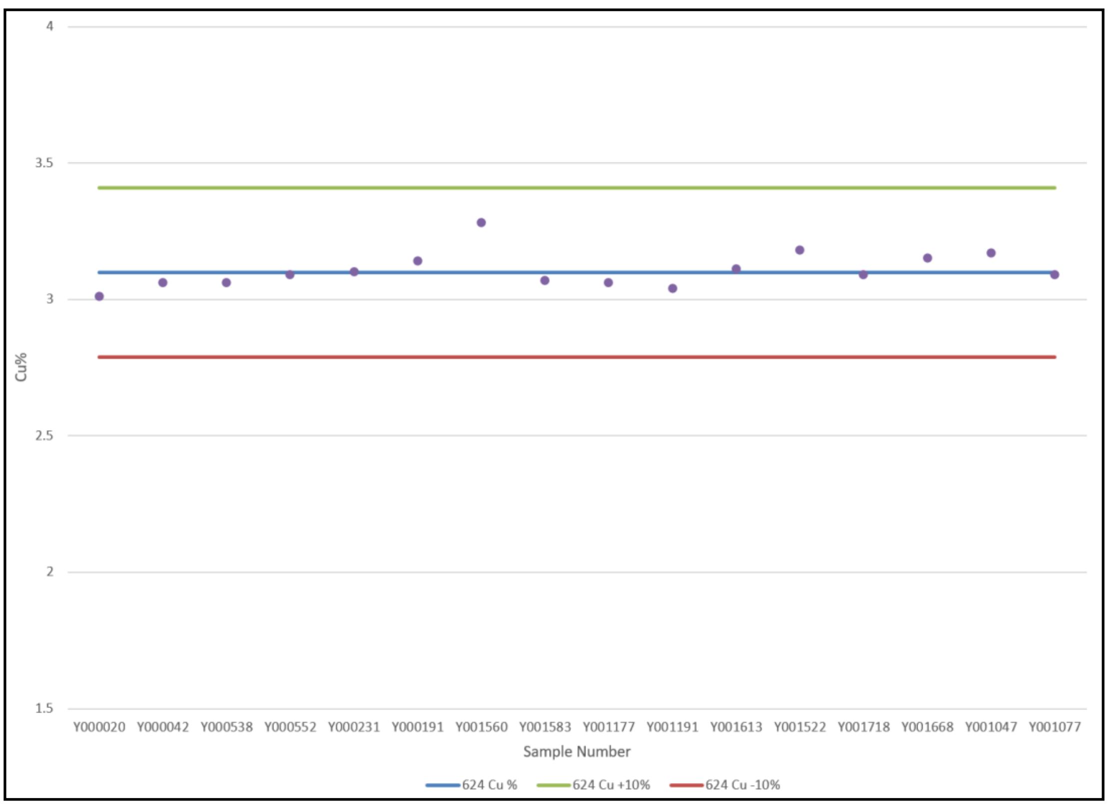


Figure 11-26: Performance of Zinc in CRM OREAS 624 from QA/QC in 2018 Drilling
(Source: SIM Geological, 2021)

For the first time in the life of the Project, field duplicate pairs were included as part of the 2018 QA/QC program. Field duplicates consisted of cutting the primary assay sample in half again along the length of the sample and labelling it with a unique sample ID number. A total of 22 field duplicates were taken during the program. The results for copper and zinc are shown in Figure 11-27 and Figure 11-28, respectively. The grades of these duplicate samples tend to be quite low, but they compare well throughout the grade ranges.

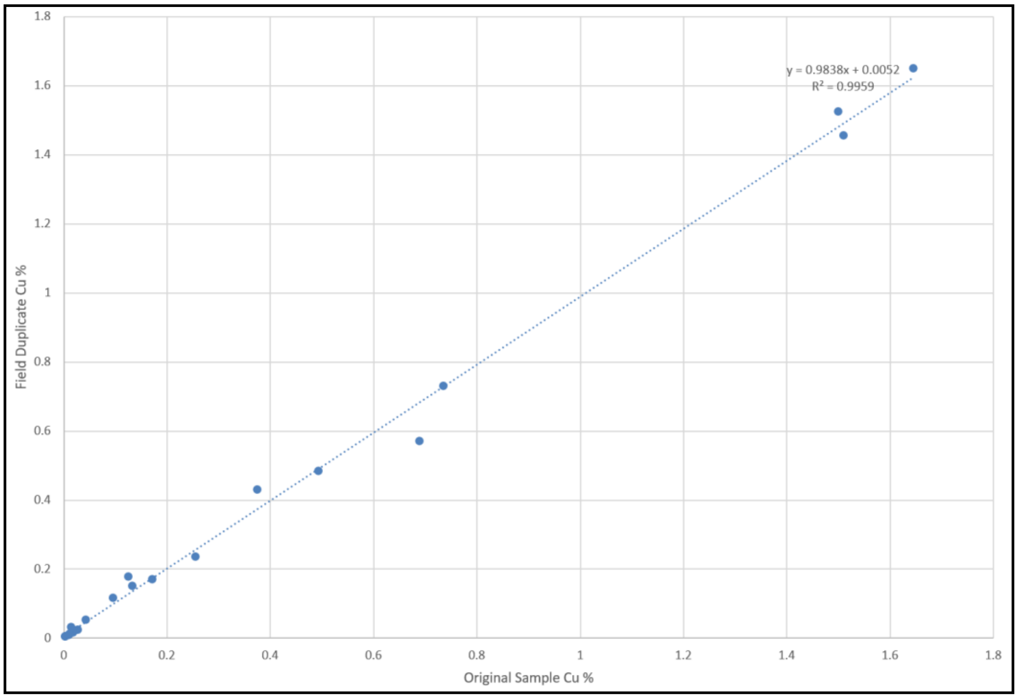


Figure 11-27: Field Duplicate Copper Performance

(Source: SIM Geological, 2021)

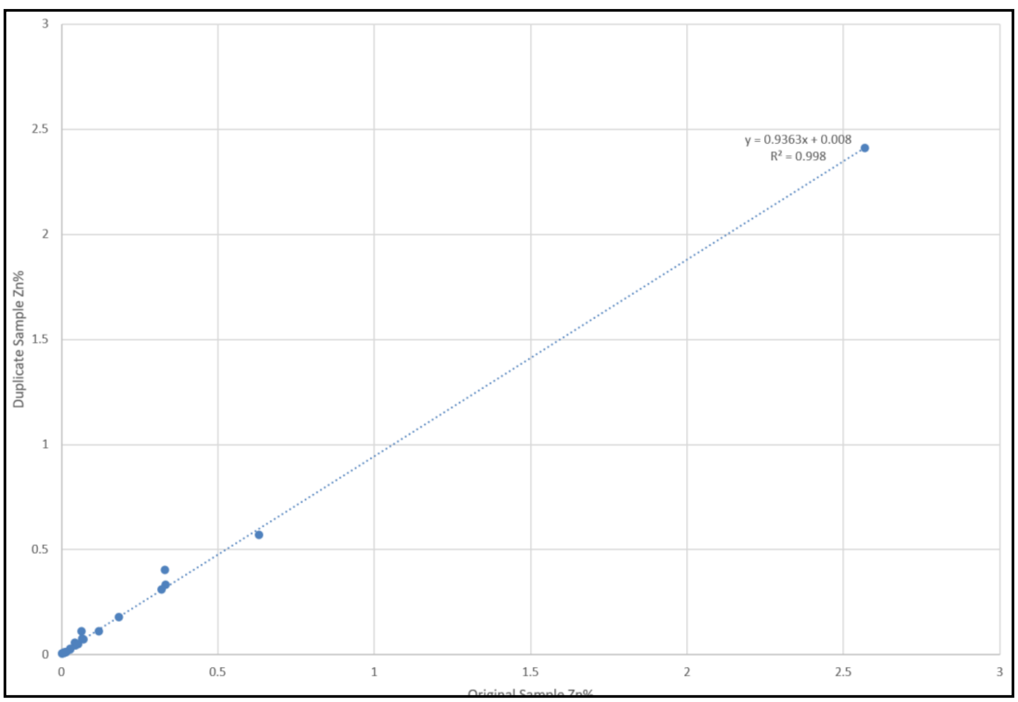


Figure 11-28: Field Duplicate Zinc Performance

(Source: SIM Geological, 2021)

Comparisons of coarse reject duplicates, where ALS prepared a split of the coarse reject and made a new pulp and analyzed it using the same methods as the original, are shown for copper and zinc in Figure 11-29 and Figure 11-30, respectively. Performance of the duplicates is acceptable with only isolated outlying results.

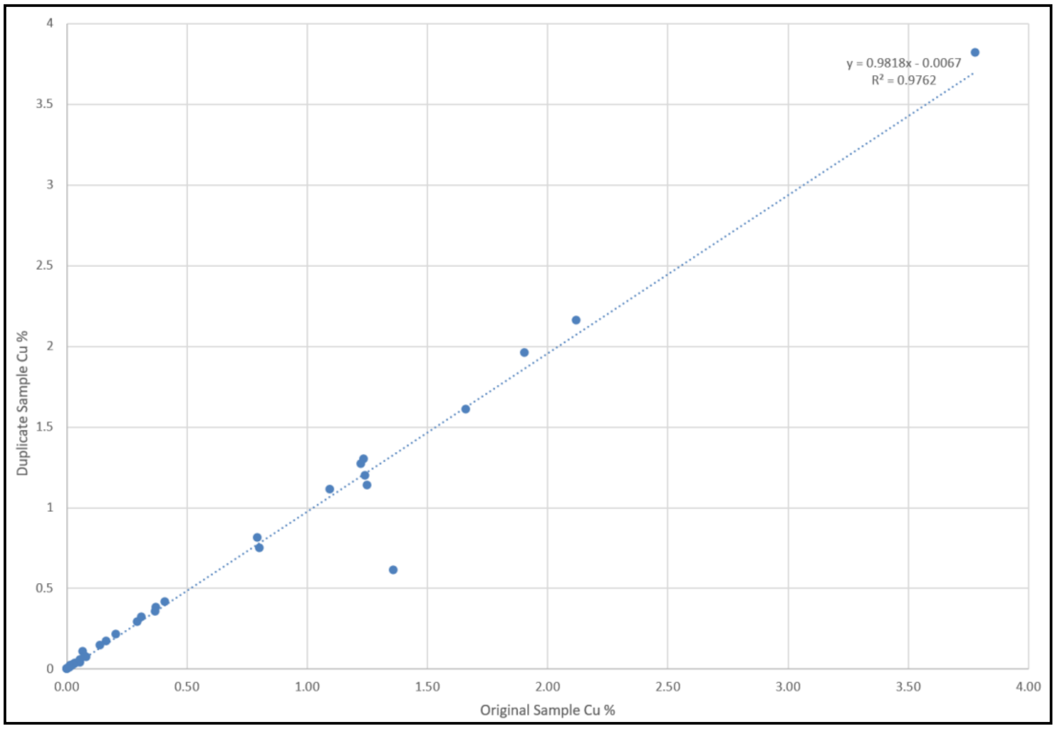


Figure 11-29: Coarse Duplicate Copper Performance

(Source: SIM Geological, 2021)

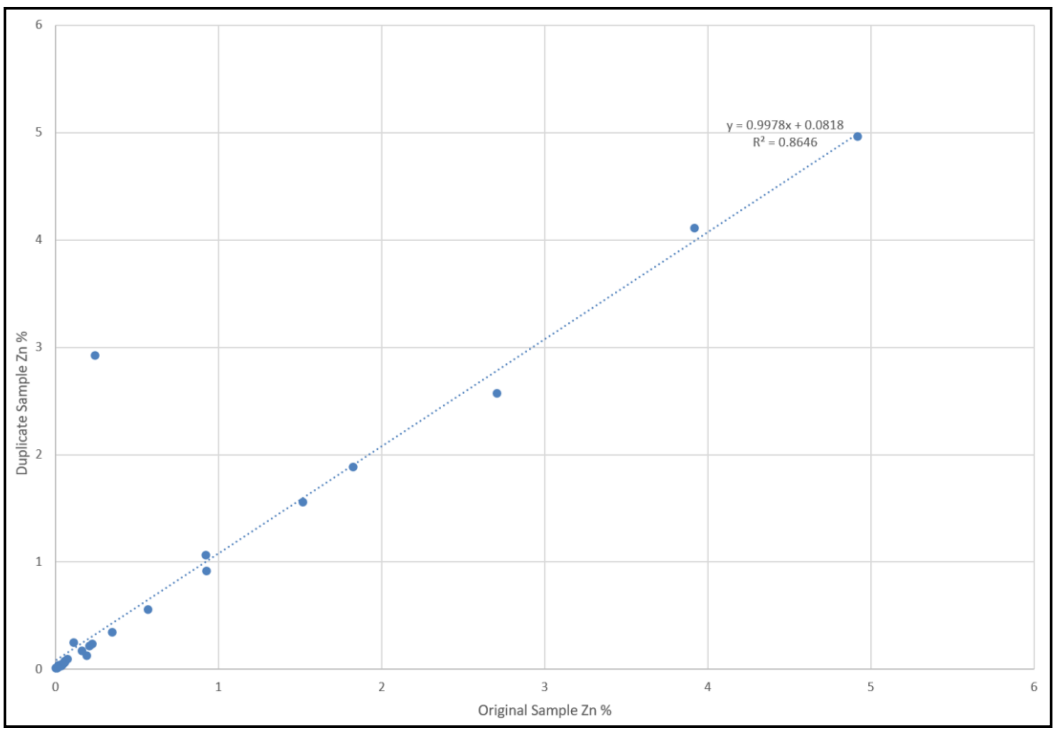


Figure 11-30: Coarse Duplicate Zinc Performance

(Source: SIM Geological, 2021)

At the end of the 2018 drilling program, the pulps from a total of 41 samples generated at ALS, selected across a range of grades, were sent to an outside umpire (Bureau Veritas) laboratory for analyses. Comparisons for copper and zinc (shown in Figure 11-31 and Figure 11-32, respectively) show acceptable reproducibility throughout the range of grades. More variability is seen in the gold and silver results but, overall, the results are also considered to be acceptable.

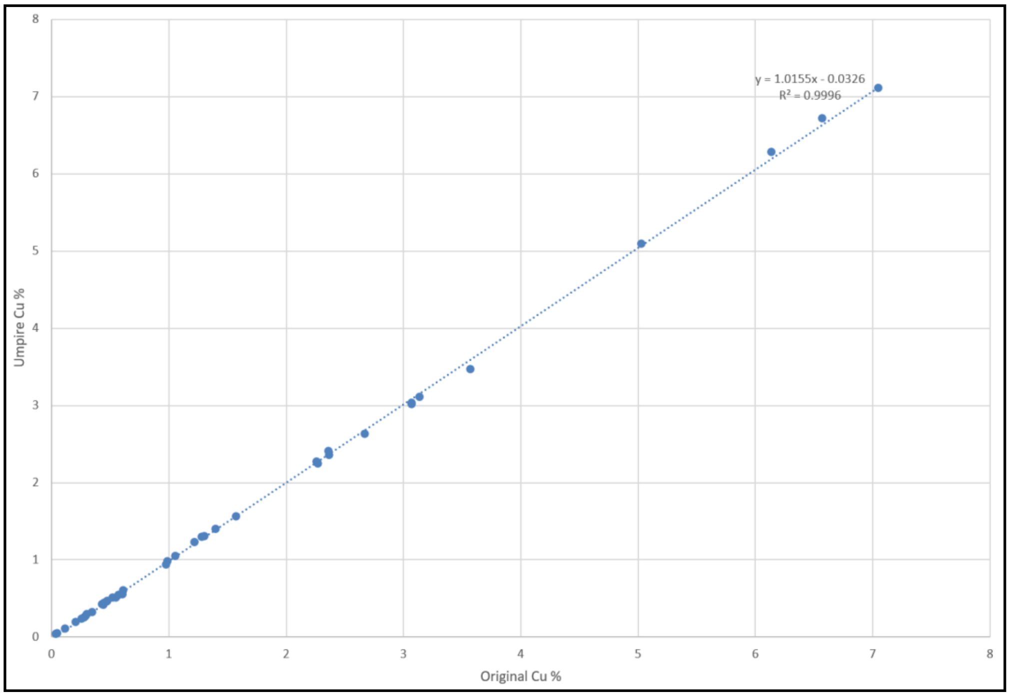


Figure 11-31: Umpire Copper Performance
(Source: SIM Geological, 2021)

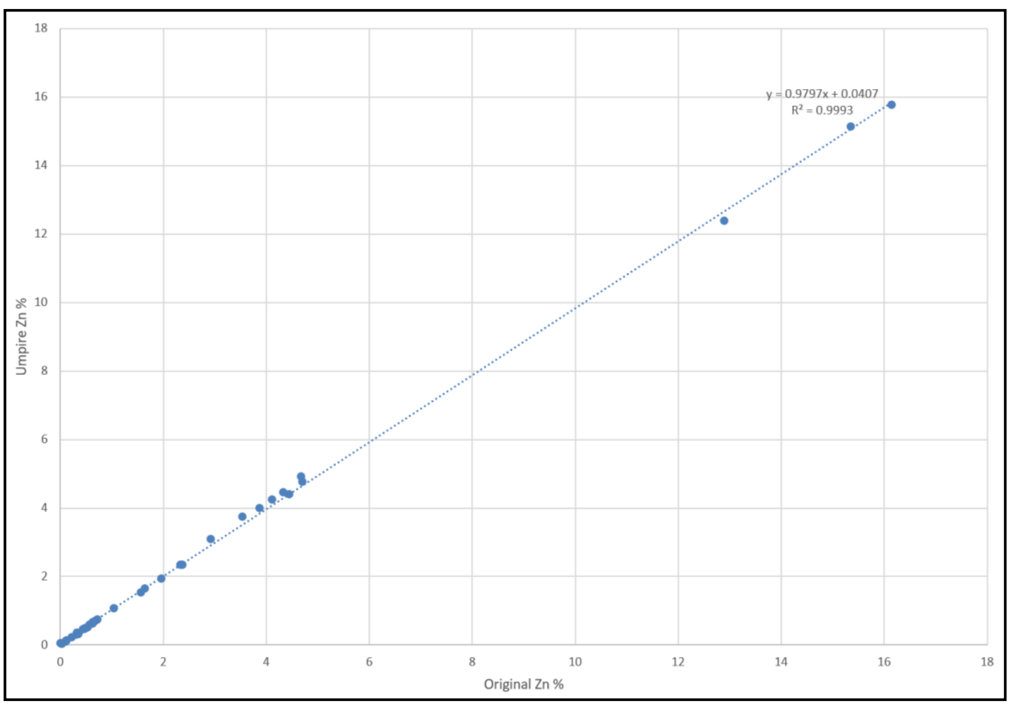


Figure 11-32: Umpire Zinc Performance
(Source: SIM Geological, 2021)

11.8 Qualified Person's Opinion on Sample Preparation, Analytical and Security Procedures

In the author's opinion, the core handling, sample selection and preparation, analysis, QA/QC and security protocols are consistent with common industry practices. The two laboratories that were used to analyze majority of the samples from the Project, ALS Global and Bureau Veritas, are independent, impartial, internationally accredited laboratories that meet ISO/IEC 17025-2015 and ISO-9001-2015 standards.

12 Data Verification

12.1 Drillhole Collar Validation

The locations of all drillhole collars have been surveyed using differential GPS. This includes the resurveying of older holes drilled in the 1970s, 1980s, and 1990s. Collar elevation data are validated by comparing surveyed elevations with the digital elevation model (DEM) generated by photogrammetry from orthophotos over the Project area. Most elevation differences at the drillhole collars were less than 1 m compared to the DEM surface.

12.2 Downhole Survey Validation

The downhole survey data were validated by searching for the presence of any large discrepancies between sequential dip and azimuth readings. No significant discrepancies were identified.

12.3 Drill Data Verification

All collars, surveys, geology and assay data were exported from the Project Geospark database into ASCII (.csv) format files and imported into MinePlan™ software. No identical sample identifications exist, all from/to data are either zero or positive values, no overlapping intervals are present, and no sample intervals exceed the total depth of its drillhole.

To validate the data, the following checks were performed:

- Maximum depth of a sample was checked against the depth of the hole.
- Sample grades below the detection limit were converted to a positive number equal to half the detection limit.
- Core recovery data were only recorded in holes drilled since 2004, and these results show average recoveries greater than 95%. There are no indications that grade is related to core recovery.

12.4 Comparisons of Vintages of Drilling Data

The distribution of drillholes was segregated into two vintages: "old" holes completed from 1974 to 1990 and "new" holes that were drilled from 2004 to present. The two sets cover similar spatial volumes of the deposit. Sample data were composited to similar lengths and de-clustered into model blocks using a nearest neighbour interpolator. The results compare model blocks that are within a maximum distance of 35 m from old and new drilling. The results indicate there is no significant difference in the drilling results encountered when using either the old or new drillholes.

12.5 Site Visit Observations

QP report author, Mr Andrew Sharp completed a personal inspection (site visit) of the Property on 23 June 2021. The Property was viewed from helicopter and several landings occurred at sites of future potential installations including the Playboy Creek tailings management facility, waste rock dump, pit, diversion channel, Esso ventilation raise and the then proposed road crossing for the Andrea Creek. During Mr Sharp's data verification process, he observed no significant differences between electronic data and topological, hydrological and vegetation realities. Mr Sharp also observed several requested drill holes to obtain a sense of rock mass quality for open pit mining and found no observable difference between the core viewed and electronic reports for geotechnical rock mass parameters.

QP author, Mr Robert Sim visited the property on 12–13 September 2018 and met with site personnel and observed drilling operations that were ongoing on the Main deposit at that time. Core handling and sampling procedures were observed and reviewed, as well as procedures used to collect SG measurements. A full tour of the deposit areas and the surrounding property was conducted by helicopter, and stops were made to observe the current drilling activities as well as stops at several of the areas where previous drilling had occurred; this included an exposure of massive pyrite uncovered during drill site preparation in the Main deposit.

Drill core from a series of randomly selected holes was reviewed and compared to the information in the database. The geological descriptions appeared to be reasonable, and visual observations of the sulphide minerals present reflected the grades in the sample database. No duplicate samples were taken by the Mr Sim to verify the results during the site visit. In the opinion of Mr Sim, the exploration activities used on the Project follow generally accepted industry standards.

12.6 Assay Database Verification

The sampling procedures adhere to accepted industry practices, and the results are monitored using a QA/QC program that also meets industry standards. The results of this work indicate that the sample database meets the criteria for precision and accuracy for use in the estimation of mineral resources.

Data validation, described in Wardrop (2007), includes analyses of the resampling by WKM of holes drilled between 1973 and 1983. The results showed very good correlation of both copper and zinc sample grades. There was more variability in the results for silver and gold, but these were still considered reasonable.

The 2007 Wardrop report includes the following statements: The QP for this (2007) report has reviewed the extensive data verification done by AMEC and is of the opinion that data relied upon is a true and accurate representation of the geology of this Project.

AMEC (2007) determined that individual table error rates fell within the ≤1% error rate considered sufficient for 0resource modelling at the prefeasibility level. The error rates were summarized as follows:

- 0.3% error rate for 100% of the collar table data reviewed
- 0.3% error rate for 23% of the downhole survey data reviewed
- 0.5% error rate for 6% of the geology table data reviewed (excluding pyrite field)
- 0.1% error rate for 56% of the assay table data reviewed (copper, zinc, gold, silver, sulphur, and mercury)
- 0.0% error rate for the top 170 high-grade samples within the assay table.

Equity Exploration was responsible for managing drilling activities on the site since 2018 and, as part of its due diligence, it regularly conducted manual checks on the information in the database.

Fifteen drillholes, five from each of the 2008, 2010 and 2018 drilling programs, were selected at random. The sample assay database values were manually checked against the certified assay certificates provided by the laboratory. There were no errors found.

12.7 Conclusion

In the QP report authors' opinion, the drilling and sampling practices, database management, validation, and assay QA/QC protocols are consistent with common industry practices. Therefore, the database is sufficiently accurate and precise for use in the estimate of Mineral Resources and Mineral Reserves.

13 Mineral Processing and Metallurgical Testing

13.1 Introduction

The Kutcho deposit contains copper and zinc mineralization, which is intended for mining and processing, using flotation, to produce both a saleable copper concentrate and a saleable zinc concentrate. Various metallurgical test programs have been conducted historically and have been reported in various studies, namely Wardop Engineering in 2007, SRK Consulting in 2008, JDS Energy and Mining in 2011, and an updated study by JDS in 2017. The present Feasibility Study has utilized the historical metallurgical information to develop a robust and reproducible flotation procedure to recover the copper and zinc using recent test programs at Bureau Veritas Commodities Canada (BVC), in 2018 and 2019, and Base Metal Laboratories (BML) in 2020 and 2021. In addition, in 2020 DRA Global Projects conducted a scoping level study for the hydrometallurgical recovery of gold, silver, and copper from the flotation tailings based on the results obtained from the BML 2020 study.

13.2 Historical Metallurgical Testing – a Review

The metallurgical testing of mineralized sample material from the Kutcho deposit up to 2017 was reported in the PFS issued by JDS in 2017. A summary of all the historical test programs conducted from 2000 to 2010 was described in this report by JDS. The salient points of the earlier test programs included the following results described in the following subsections.

13.2.1 Mineralogy

The Wardrop PFS 2007 reported the following sulphide minerals as being present, major amounts of pyrite with lesser amounts of valuable minerals, namely chalcopyrite, sphalerite, and bornite with minor chalcocite, tetrahedrite, digenite and related copper-sulphur species, galena, idiaite, hessite and electrum. The minerals digenite, idiaite, hessite and electrum have not been reported in any of the more recent studies.

SRK PEA 2008 reported that the grain sizes of the copper sulphide minerals were generally between 20 µm and 30 µm, while the sphalerite was coarser grained at between 5 µm and 230 µm. This observation led to the early studies adopting a primary grind P_{80} size of about 60–75 µm with a bulk copper-zinc flotation process followed with a bulk concentrate regrind P_{80} size to about 20 µm which in turn was followed by the copper-zinc separation stage. The copper rougher concentrate was reground to a P_{80} of 9 µm and upgraded in two to three cleaner stages. There was no regrind of the zinc rougher concentrate. The zinc rougher concentrate had three (or four) cleaner stages to upgrade the zinc concentrate. The results obtained are shown in Table 13-16 (titled Locked Cycle Test Results) together with the results of the other locked cycle tests. The reagents used were generally similar to those used by BML in the 2020 and 2021 test programs. However, the SRK PFS report presented a projected metallurgical balance based on the sequential flotation process and using recovery estimates based on open circuit test results. These projected values were used in the report and appear to indicate an opportunity for increased gold and silver recovery values when compared with the bulk circuit recovery values obtained in the locked cycle test. The projected values are also shown in Table 13-16 for comparison purposes.

The JDS PFS 2011 report also included a summary of the mineralogical features of the valuable minerals, namely the copper and zinc minerals, the relative abundances of the copper minerals, and the degree of liberation at a particle size P_{80} of 75 µm. This mineralogical data is given in the Table 13-1.

Table 13-1: Summary of mineralogical features

Mineral species	Copper – relative abundance (%)	Degree of liberation (%)	Average grain size (µm)	Main associations
Bornite	65.7	80.3	10	Pyrite
Chalcopyrite	30.2	58.5	20	Pyrite
Chalcocite	4.1	estimated as < 30	-	-
Sphalerite	-	69.8	20	Pyrite

The JDS 2011 test program also investigated the effect of primary grind size on the recovery of copper and zinc. The results indicated that both the copper and zinc recovery values in the rougher concentrate are relatively insensitive to grind size in the P_{80}size range of 42 µm to 80 µm. This led to the selection of the primary P_{80} grind size of 75 µm in the process design.

13.2.2 Sample Feed Grade

Based on the results of the locked cycle tests, and projected recoveries, the feed grades of the sample material tested in the various test programs, and the estimated projected feed grades, are given in the Table 13-2. The values were found to vary highlighting the difference between the grade of the Main and Esso deposits. However, with the more accurate representation of the sample material which was used and tested during the BML test programs, these BML values for the 2021 Mine Plans for Main and Esso are considered to be more accurate.

Table 13-2: Head grade values of various test programs

Report and sample	Copper (% Cu)	Zinc (% Zn)
BML – Main Deposit	1.69 to 1.87	1.76 to 2.43
BML – Esso Deposit	2.60	5.51
BML – 75% Main / 25% Esso Projected Values	1.94	3.10
BML – Mine Plan, Feasibility Study 2021	1.77	2.58
BML – KUTCHO COPPER Composite	1.81 to 1.92	1.80 to 1.96
BML – Avg S Composite	2.16	3.61
Wardrop PFS 2007	1.63	2.26
SRK PEA 2008	2.48	2.48
SRK PEA 2008 – Projected Values	1.73	2.34
JDS PFS 2011	1.85	2.88
JDS PFS 2017 – Projected Values	2.01	3.19
BVC 2019	1.87	2.45

Apart from the BML_KUTCHO COPPER and Avg S Composite samples which are known to be non-representative, the other samples which appear to be too high or too low is the copper value for SRK PEA 2008, and the JDS PFS feed grade values. However, as the understanding of the morphology of the deposit increased, sample materials for the test programs were selected to be more representative of the deposit.

13.2.3 Grind Size

Different primary grind and concentrate regrind sizes were used in the test programs. Table 13-3 summarizes the particle sizes tested.

Table 13-3: Summary of P_{80} primary grind and concentrate regrind particle sizes (µm)

Test program/ flotation procedure	Primary grind (µm)	Copper rougher concentrate (µm)	Zinc rougher concentrate (µm)
Wardrop 2007 – bulk	75	Bulk concentrate 15 µm, then copper rougher concentrate 10 µm	No regrind
SRK 2008 – bulk	74	Bulk concentrate 18 µm, then copper rougher concentrate 9 µm	No regrind
SRK 2008 – sequential	40	10	No regrind
JDS 2011 – sequential	75	40	No regrind
JDS 2017 – sequential	75	35	No regrind
DRA 2020 – sequential	75	15	20
BML – sequential	55	15	20

The initial test results giving the relationship between the recovery of copper and zinc versus the primary grind particle size, together with the available mineralogical data, and the use of the bulk flotation method, led to the initial indications that a relatively coarse primary grind P_{80} size of 75 µm was required for recovering the copper and zinc in a rougher concentrate. During the copper-zinc separation stage, the copper rougher concentrate was subsequently reground to finer sizes and refloated in a cleaner stage(s) in order to upgrade the concentrate to obtain the final marketable copper grade. In the case of zinc, no regrinding was deemed necessary, and a marketable product was obtained by refloating in several cleaner stages. However, with the adoption of the sequential flotation procedure, the primary grind P_{80} size was made finer to about 55 µm and the copper rougher concentrate was reground to a finer size for upgrading. Initially, the zinc rougher concentrate was not reground, but with further development of the flowsheet, it was demonstrated that the regrinding of the zinc rougher concentrate gave improved results. The test work subsequently showed that the optimal grind and regrind P_{80} sizes were 55 µm for the primary grind, and 15 µm and 20 µm respectively for the copper and zinc rougher concentrates.

*13.2.4 **Flotation Procedures***

Different flotation procedures have been used during the various test programs. The bulk sulphide mineral flotation method, followed by the separation of the copper and zinc minerals, was initially tested. However, it was replaced by the sequential flotation method which was further tested and developed into the present flotation procedure adopted for the process. The sequential procedure involved the depression of pyrite and sphalerite during the initial recovery of copper minerals into a copper rougher concentrate. The copper rougher concentrate was then reground and upgraded generally in one cleaner stage, although additional cleaner stages were also investigated. The copper rougher tailings, and the copper cleaner tailings were combined and conditioned to form the feed to the zinc flotation stage. The resulting rougher concentrate was subsequently reground and generally re-floated in three stages of cleaners to produce the marketable zinc concentrate product.

The Wardrop PFS 2007 test program utilized the bulk sulphide mineral flotation process, followed by the regrinding of the bulk concentrate and separation of the copper (with a further regrind stage) and the zinc minerals into separate concentrates. The SRK PEA 2008 flotation procedures adopted involved both the bulk flotation method, as well as the sequential copper-zinc flotation procedure. The details have been described in Section 13.2.1. The JDS PFS 2011 report described the flotation procedure used in the testing of the Kutcho material as follows: primary grind P_{80} of 75 µm, followed by a copper rougher flotation stage and a rougher-scavenger stage, then regrinding of the rougher concentrate, and three cleaner flotation stages. The zinc flotation stage included a rougher and a rougher-scavenger stage, and three cleaner stages with no regrinding of the rougher concentrate. The samples representing the projected annual compositions of the plant feed material were tested and the copper and zinc grades and recoveries determined. An overall Global Composite sample was also compiled and tested. The results were listed in the report, and the results of the Global Composite locked cycle test is given in Table 13-16 for comparison with other similar test results. The JDS PFS 2017 report updated the results of the earlier study conducted in 2011.

13.2.5 Flotation Reagents

The reagents used in the historical flotation of copper and zinc test programs varied and differences are briefly mentioned in this section.

The Wardrop PFS 2007 test program used SO_2-based reagents for the recovery of copper but used sodium cyanide in the zinc flotation stage for pyrite depression. Lime was used for pH control. The SRK test program essentially used the same reagents as the BML program, namely an SO_2-based pyrite and zinc depressant scheme, with some differences in the collector reagents. The JDS PFS 2011 test program used lime for pH control, various reagents for dispersion and depression, copper sulphate for the activation of sphalerite, sulphidation reagents, and two collector reagents, namely Aero 3477 (an alkyl dithiophosphate) and Promoter 7583 (a dialkyl dithiophosphinate). The frother reagent used in the test program was Frother F1064. The test program described the presence of zinc in the copper circuit as resulting from the activation of sphalerite by copper ions as a result of the reactive copper minerals present in the deposit. This led to the implementation of a high pH value to precipitate the copper ions in solution and the addition of sulphide and sulphite reagents to activate the copper particle surfaces in the copper flotation stage. The JDS PFS 2017 test program used the earlier JDS PFS 2011 reagent procedure as the basis for its test program. The DRA 2020 report followed the reagent regime which was being tested at the time and which was described in the BML report (BL483). This reagent scheme also used lime for pH control, copper sulphate and Aero 3477, but differed by excluding the dispersant and included the addition of sodium sulphide and zinc sulphate for the activation of copper minerals. The collector reagent used was Aero 3894 Promoter, a dialkyl thionocarbamate.

13.2.6 Concentrate Handling

The copper and zinc final product flotation concentrates were each dewatered and filtered in a conventional manner prior to being shipped off of the property to the nominated smelter

13.3 Review of the Metallurgical Test Programs Conducted After 2017

13.3.1 Introduction

Since the JDS PFS report was published in 2017, additional metallurgical test programs were conducted. These test programs included the following:

- BVC 2018, a mineralogical investigation of flotation products from an open cycle rougher test
- BVC 2019, a mineralogical investigation of flotation products from a locked cycle test
- DRA 2020, the evaluation of a hydrometallurgical process to recover gold, silver, and copper from the flotation tailings
- BML 2020, the development of a flotation flowsheet to recover copper and zinc
- BML 2021, additional test work to generate concentrate and tailings samples for settling tests.

The flotation results from the BML 2020 test work were ultimately developed to the stage where it could be reliably used for the detailed process design for the recovery of the copper and zinc from the Kutcho mineralized material.

13.3.2 Bureau Veritas Commodities, Mineralogical Studies, 2018 and 2019

A detailed mineralogical investigation was undertaken in 2018 and 2019. Specific metallurgical flotation test samples were investigated with respect to mineralogical composition and associations, liberation characteristics, and the effect of particle size on the recovery (losses) of copper and zinc during the respective flotation stages. The 2018 test program investigated rougher flotation products, while the 2019 test program investigated the products from a locked cycle test. In both studies, the term "copper sulphide minerals" was used generically, and no distinction was made between the various copper-bearing minerals chalcopyrite, bornite, chalcocite, covellite, enargite/tennantite and oxidized copper minerals.

The Kutcho Copper deposit is a massive sulphide mineral deposit containing about 94% by weight sulphide minerals, dominated by pyrite. The primary copper-bearing minerals are bornite and chalcopyrite, with the minor copper sulphide minerals chalcocite, covellite and enargite/tennantite. This high ratio of pyrite to copper sulphide minerals is anticipated to cause difficulties in the separation process. The zinc occurs as sphalerite. Two other sulphide minerals occurring in small amounts which have been identified are galena and pyrrhotite. Gold and silver-bearing minerals were not identified. Non-sulphide gangue minerals consist of different types of silicates including quartz, muscovite/illite, chlorite, kaolinite, and the iron-bearing minerals goethite and siderite.

13.3.2.1 Bureau Veritas Commodities, 2018

Table 13-4 and Table 13-5 give the mineralogical and chemical composition of the sample used to generate the rougher flotation test products which were studied to determine the copper-zinc behaviour characteristics during the flotation separation process. Gold assays were not conducted, and the mineralogical characteristics of gold were not investigated.

Table 13-4: Chemical analysis of the rougher flotation test feed sample

Element	Units	Assay
Copper, Cu	%	2.04
Lead, Pb	%	0.03
Zinc, Zn	%	4.46
Iron, Fe	%	40.7
Sulphur, S	%	48.5
Carbon, C	%	0.30
Silver, Ag	g/t	31.1
Arsenic, As	%	0.07

Table 13-5: Mineralogical analysis of the rougher flotation test feed sample

Sulphide minerals	Mass (%)
Chalcopyrite	1.43
Bornite	2.02
Chalcocite	0.09
Covellite	0.07
Enargite/Tennantite	0.31
Galena	0.03
Sphalerite	6.66
Pyrite	83.5
Pyrrhotite	0.05
Total	**94.2**

Non-sulphide minerals	Mass (%)
Iron Oxides	1.05
Quartz	1.62
Dolomite/Ankerite	1.76
Muscovite/Illite	0.65
Kaolinite	0.30
Others	0.46
Total	**5.83**

The test was conducted as a rougher flotation test only at the primary grind size P_{80} of 46 µm. The results of the rougher flotation test are given in Table 13-6, while the mineralogical observations from this test are discussed below.

Table 13-6: Rougher flotation test metallurgical results

Product	Mass (%)	Assays (%)			Distribution (%)		
		Cu	Zn	Fe	Cu	Zn	Fe
Cu Rougher Concentrate	7.8	8.92	13.7	28.8	34.0	24.0	6.0
Zn Rougher Concentrate	21.9	4.12	15.2	33.9	45.0	74.0	19.0
Tailings	70.3	0.60	0.13	43.4	21.0	2.0	76.0
Total/Feed	**100.0**	**2.02**	**4.48**	**40.2**	**100.0**	**100.0**	**100.0**

The mineralogical and particle size review of the flotation products from this test has resulted in the following observations. The copper sulphide minerals were liberated at about 61%, with the unliberated copper sulphide minerals occurring mainly in binary form mainly with pyrite and to a minor extent with sphalerite. At this liberation level it would theoretically be possible to recover 90% of the copper minerals into a rougher concentrate. Sphalerite was liberated at about 82% with the unliberated sphalerite occurring in binary form mainly with pyrite and to a lesser extent with copper sulphide minerals. At this extent of liberation, it would theoretically be possible to recover 90% of the sphalerite into a rougher concentrate.

Despite the high degree of liberation achieved for both the copper sulphide minerals and sphalerite, the actual results obtained for both flotation circuits were lower than the theoretically achievable recovery values. This is attributable to the poor functioning of the copper rougher circuit with misplaced sphalerite reporting to the copper rougher concentrate, and the incomplete recovery of recoverable/liberated copper minerals mainly as a result of particle size effects and partially as a result of binary particle losses to the sphalerite circuit. This resulted in the relatively high recovery of 45% of copper in the zinc flotation circuit.

An evaluation of the copper flotation concentrate indicated that the extent of misplacement of sphalerite in the copper rougher concentrate was significant and this has reduced the recovery of zinc in the zinc flotation circuit. A particle size analysis indicated that the recovery of copper sulphide minerals declined for particle sizes >25 µm, and that for sizes >38 µm all the copper sulphide minerals were lost in the copper rougher tailings, although some of these particles would likely report to the sphalerite rougher concentrate. Also, the sphalerite-copper sulphide binary particles were recovered in the zinc rougher stage.

An evaluation of the zinc flotation concentrate indicated that zinc rougher circuit is not as sensitive to particle size as the copper flotation circuit in that the zinc recovery only starts to decrease from 60 µm and coarser. However, the misplacement of sphalerite because of poor selectivity and its reporting to the copper rougher concentrate implies that this aspect of flotation will need to be closely monitored. A possibly more aggressive pyrite depression stage ahead of the zinc flotation stage will also improve the grade and recovery of the zinc in the zinc rougher concentrate.

A comparison of the particle size versus metal recovery values obtained for the copper and the zinc rougher flotation stages is given in Table 13-7.

Table 13-7: Particle size vs metal recovery, rougher flotation test

Particle size (µm)	Copper recovery (%)	Zinc recovery (%)
12	51	63
21	8	94
41	1	96
66	0	91

Also significant is the finding that the P_{80} of the copper rougher concentrate is 12 µm (without a concentrate regrind stage) while the P_{80} value for the zinc rougher concentrate is 45 µm also with no regrinding of the zinc rougher concentrate.

Although silver assays were conducted for this test, the silver was found to not be upgraded in either the copper flotation stage nor to the zinc flotation stage.

13.3.2.2 Bureau Veritas Commodities, 2019

Table 13-8 and Table 13-9 together give the mineralogical and chemical composition of the sample used to generate the flotation test products from a locked cycle test and which were studied to determine the copper-zinc behaviour characteristics during the flotation separation process. As with the earlier test program, no gold assays nor mineralogical characteristics were performed.

Table 13-8: Chemical analysis of the locked cycle flotation test feed sample

Element	Units	Assay
Copper, Cu	%	1.87
Lead, Pb	%	0.08
Zinc, Zn	%	2.45
Iron, Fe	%	28.0
Sulphur, S	%	32.6
Carbon, C	%	1.65
Silver, Ag	ppm	31.7
Arsenic, As	%	0.02

Table 13-9: Mineralogical analysis of the locked cycle flotation test feed sample

Sulphide minerals	Mass (%)
Chalcopyrite	1.40
Bornite	1.83
Chalcocite	0.27
Covellite	0.09
Enargite/Tennantite	0.13
Galena	0.09
Sphalerite	3.77
Pyrite	57.7
Pyrrhotite	0.03
Total	**65.3**

Non-sulphide minerals	Mass (%)
Iron Oxides	0.57
Quartz	9.13
Dolomite/Ankerite	9.95
Muscovite/Illite	7.30
Kaolinite	3.79
Others	3.92
Total	**34.7**

The test was conducted as a locked cycle flotation test at the primary grind size $P_{80}$of 60 µm. The results of the locked cycle flotation test are given in Table 13-10, while the mineralogical observations from this test are discussed below.

Table 13-10: Locked cycle flotation test metallurgical results

Product	Mass (%)	Assays %			Distribution %		
		Cu	Zn	Fe	Cu	Zn	Fe
Cu Concentrate	6.2	24.8	9.35	26.6	82.3	23.7	5.9
Zn Concentrate	3.1	0.62	53.00	8.16	1.0	67.3	0.9
Tailings	90.7	0.34	0.24	28.8	16.7	9.0	93.2
Total/Feed	**100.0**	**1.87**	**2.45**	**28.0**	**100.0**	**100.0**	**100.0**

The mineralogical and particle size review of the flotation products from this test has resulted in the following observations. The copper sulphide minerals were liberated at about 74%, with the unliberated copper sulphide minerals occurring mainly in binary form mainly with pyrite and to a minor extent with sphalerite. At this liberation level it would theoretically be possible to recover 90% of the copper minerals into a final copper concentrate with a grade of about 30% Cu. Sphalerite was liberated at about 79% with the unliberated sphalerite occurring in binary form mainly with pyrite and to a lesser extent with copper sulphide minerals. At this extent of liberation, it would theoretically be possible to recover 90% of the sphalerite into a rougher concentrate. Although the theoretical recovery values were not attained, the overall test results were a significant improvement over the earlier rougher test conducted.

An evaluation of the zinc flotation concentrate indicated that a final zinc concentrate grade product of about 50% Zn at a recovery of 85% was achievable. This result was not attained mainly as a result of misplacement of the zinc in the copper flotation circuit. The major source of zinc losses was the loss of fine-grained sphalerite <20 µm in size possibly due to the slower flotation kinetics of that size range. It was also shown that the zinc flotation process is not as sensitive to particle size as the copper flotation circuit. The particle size versus recovery values obtained for the zinc flotation were about 52% recovery at 5 µm, about 57% at 14 µm, about 80% at 19 µm and about 91% recovery at 58 µm. The recovery of pyrite in the zinc circuit was also significantly lower compared with the rougher test indicating that pyrite depression was successfully achieved in this test. The results are summarized in Table 13-11.

Table 13-11: Particle size vs metal recovery, locked cycle flotation test

Particle size (µm)	Copper recovery (%)	Zinc recovery (%)
5	82	52
14	92	57
19	82	80
58	13	91

Also significant is the finding that the P_{80} of the final product copper concentrate is 21 µm (which included a rougher concentrate regrind stage) while the P_{80} value for the final product zinc concentrate is 49 µm (no concentrate regrind stage).

No silver assays were conducted for this locked cycle test. However, lead and arsenic assays were conducted, and both were preferentially upgraded into the copper concentrate.

The main findings of the mineralogical investigation were the conclusions that the misplacement of pyrite will significantly affect the concentrate grades of copper and zinc, and that the recovery of copper occurs for a significantly finer particle size than that for zinc. The regrinding of the copper concentrate is considered essential, and the relatively less fine (coarser) regrind size of the zinc rougher concentrate is also demonstrated. However, the overgrinding has also been shown as leading directly to metal loss and should be avoided.

The locked cycle test results indicate that the maximum particle size for copper mineral flotation is coarser relative to the rougher flotation test results at about 14 µm, and that coarser copper particles are floated at about 60 µm compared with no coarse particles floated in the rougher test. Sphalerite is confirmed as having its maximum recovery at about 60 µm. The reason for the coarser particle size recovery in the locked cycle test is due to the lower pyrite content of the feed material tested, and the more efficient depression of pyrite in this test.

In conjunction with the observations from the test program discussed in the report BM483 (Section 13.3.4), it can be concluded that bornite constitutes the main fine-grained copper-bearing mineral lost as liberated particles in the size range <10 µm as referred to earlier in the BVC rougher flotation test observations. The higher proportion of bornite in the finer particle size range would also contribute to the copper losses observed.

13.3.3 DRA, Hydrometallurgical Plant Evaluation, 2020

In 2020, DRA conducted a study to evaluate the economic potential of including a hydrometallurgical process (cyanide leach) on the plant tailings material generated from the copper and zinc flotation processes in order to recover some of the contained gold and silver as well as some residual copper. The report used the results obtained from an ongoing test program which was being conducted at the Base Metal Laboratories and which had included some basic cyanide leaching tests.

The envisaged hydrometallurgical process flowsheet incorporated the bulk flotation of pyrite and other associated sulphide minerals, which included the associated gold and silver. It was anticipated that some residual copper sulphide minerals would also be recovered during the pyrite flotation stage. The study flowsheet proposed that the pyrite concentrate would be thickened and then leached with sodium cyanide. A counter-current decantation circuit would recover the pregnant solution which would be treated in a Merrill-Crowe circuit for the precipitation of gold and silver, while the copper in solution would be recovered in the SART process (sulphidization, acidification, recycling, and thickening) circuit. The study was based on the throughput of 3,500 tpd and estimated the following grade and recovery values (Table 13-12) over the LOM period of 10 years. The flotation grade and recovery values from the then ongoing BML test program were used in the study (subsequently issued as a Report No. BM483 in 2020).

Table 13-12: Concentrate products and grades

Copper concentrate products	
Mass recovery, %	6.7
Tonnage, t/year	85,362
Copper recovery, %	89.5
Gold recovery, %	37.9
Silver recovery, %	61.3
Copper grade, % Cu	27.0
Gold grade, g/t Au	2.9
Silver grade, g/t Ag	377

Zinc concentrate products	
Mass recovery, %	4.6
Tonnage, t/year	59,400
Zinc recovery, %	72.8
Gold recovery, %	18.3
Silver recovery, %	9.5
Zinc grade, % Zn	56.0
Gold grade, g/t Au	2.2
Silver grade, g/t Ag	89

SART process products	
Tonnage, t/year:	
Gold doré, oz	73,305
Gold, kg	2,280
Silver doré, oz	2,588,163
Silver, kg	80,501
Copper sulphide, t	37,257
Copper in copper s Sulphide, t	14,875

The power requirements and capital (capex) and operating (opex) cost estimates were calculated based on the available test work data and similar projects conducted by DRA. (2020). The comparative study returned a positive IRR of 38.8% with a payback period of 2.3 years on a pre-tax basis.

The risks identified in the study included the use of cyanide and its associated environmental permitting requirements and management, and the potential of significant increases in capex costs (especially with respect to construction costs) and opex costs (specifically with regards to reagent costs). In addition, no SART test work had been performed. The opportunities identified included potential improvements to the flowsheet and the equipment types and sizes, and as well as an increase in metal prices.

13.3.4 Base Metal Laboratories Test Program BL483, 2020

13.3.4.1 Introduction

The Kutcho polymetallic mineral deposit contains economic grades of copper and zinc with minor amounts of silver and gold. Recovery methods have focused on the flotation recovery of the metals as a copper concentrate containing by-product silver and gold, and a zinc concentrate. The mineralogically complex interlocking nature of the copper and zinc minerals, the occurrence of copper being hosted in different sulphide-bearing minerals, the fine-grained nature of the valuable minerals, and with pyrite forming the gangue host rock, have been the reasons that the separate recovery of copper and zinc was problematic. The pre-activation of sphalerite and its recovery into the first stage of separation, namely the copper flotation stage, has led to the conclusion that the conventional sphalerite depressant systems are not effective in this case. A more aggressive sulphur dioxide based (SO_2 containing reagents) depressant system was tested and demonstrated to be successful in separating the copper and zinc and producing saleable grade concentrates.

Flotation test programs have focused on two types of flowsheets, namely the bulk flotation process whereby a single copper-zinc concentrate is initially produced followed by the separation of the copper and zinc into separate concentrates, or the sequential flowsheet first producing a copper concentrate followed by the production of a zinc concentrate. Earlier test programs have used the bulk flotation method, while this present test program used the sequential method. The flowsheet incorporated a relatively fine primary grind to a P_{80} value of 55 µm followed by the copper rougher flotation step. The copper concentrate was reground to a P_{80} size of 15 µm and a cleaner concentrate was produced. Only one cleaner stage was required for the flotation recovery of copper. The copper flotation tailings were conditioned and floated to produce a zinc rougher concentrate. The rougher concentrate was reground to a P_{80} size of 20 µm. This was followed by three stages of cleaning with the zinc rougher tailings and the zinc first cleaner tailings being combined as the final tailings. This flowsheet was tested with several samples from the deposit using locked cycle flotation tests. The results were evaluated, and the process was deemed robust and reproducible. Consequently, it was selected as the flowsheet for the design of the Kutcho mine process plant.

In addition to defining the flotation procedure, dewatering and filtration tests were also performed on copper and zinc concentrate samples. The settling characteristics of the final tailings were also determined.

13.3.4.2 Sample Description

The sample material used in this test program consisted of various existing sample material which had been used in the test program at BV Laboratories, Richmond, BC, termed the KM Global Composite, the KUTCHO COPPER Composite and the Avg S Composite samples. The bulk of the sample material received for testing represented historical drill-core material, as split cores, obtained from the 2008 and 2010 drilling programs respectively, for the Main deposit and Esso deposit. New and more representative composite samples were prepared to represent the Main and Esso deposits. The sample preparation was based on QEMSCAN mineralogical results conducted to achieve the target pyrite content, as well as the copper and zinc ratios, as representing the respective Main and Esso lenses. These samples were used as the basis for the flotation test program which also included variability testing. The two samples which were prepared were labelled “Main 4 Composite” (MC4) and “Esso 2 Composite” (EC2), respectively. Table 13-13 compares the respective head grades of these two samples with the values from the Resource Estimate and the results indicate a very close correlation of these composite samples with the grades of the Resource Estimate.

Table 13-13: Composite samples head grade values

Sample	Head grade (% Cu)	Head grade (% Zn)
Main 4 Composite	1.67	2.30
Resource Estimate	1.74	2.38
Difference, %	**3.8**	**3.4**
Esso 2 Composite	2.53	4.56
Resource Estimate	2.52	4.76
Difference, %	**0.2**	**4.2**

These two samples, namely MC4 and EC2, resulted from a review of the resource model after test work had commenced using the KUTCHO COPPER Composite and Avg S composite samples. Initially, the KUTCHO COPPER Composite sample material was used as the representative sample material for the Main Lens in order to formulate the basic flotation flowsheet, and this was followed by further developmental testing. Similarly, the KM Global and Avg S Composite material was used for comminution testing with the unused material subsequently also used in the developmental testing. The significantly superior results obtained when testing the Avg S Composite sample material compared with the KUTCHO COPPER Composite material led to a review of the resource model which resulted in the use of mineralogical information to be used in the construction of a new representative sample of the Main Lens, termed the Main Composite Sample (MC1 and subsequently MC4) as detailed in Table 13-13 above. Apparently the KUTCHO COPPER Composite and KMG samples contained fault material which did not represent the material from the Main Lens. The MC1 sample was used during the flowsheet developmental test stage using the sequential flowsheet and the testing of the sulphur dioxide (SO_2)

reagent based depressant method. A significant finding was that a finer primary grind than a P_{80} size of 55 µm did not improve the recovery of copper and that selectivity actually decreased with finer grinding. The final optimization of the flotation flowsheet was conducted using the Main Composite 4 (MC4) sample which was prepared upon exhaustion of the MC1 sample. The final flowsheet iteration is represented by the flotation test LCT-73 and shown as Figure 13-1.

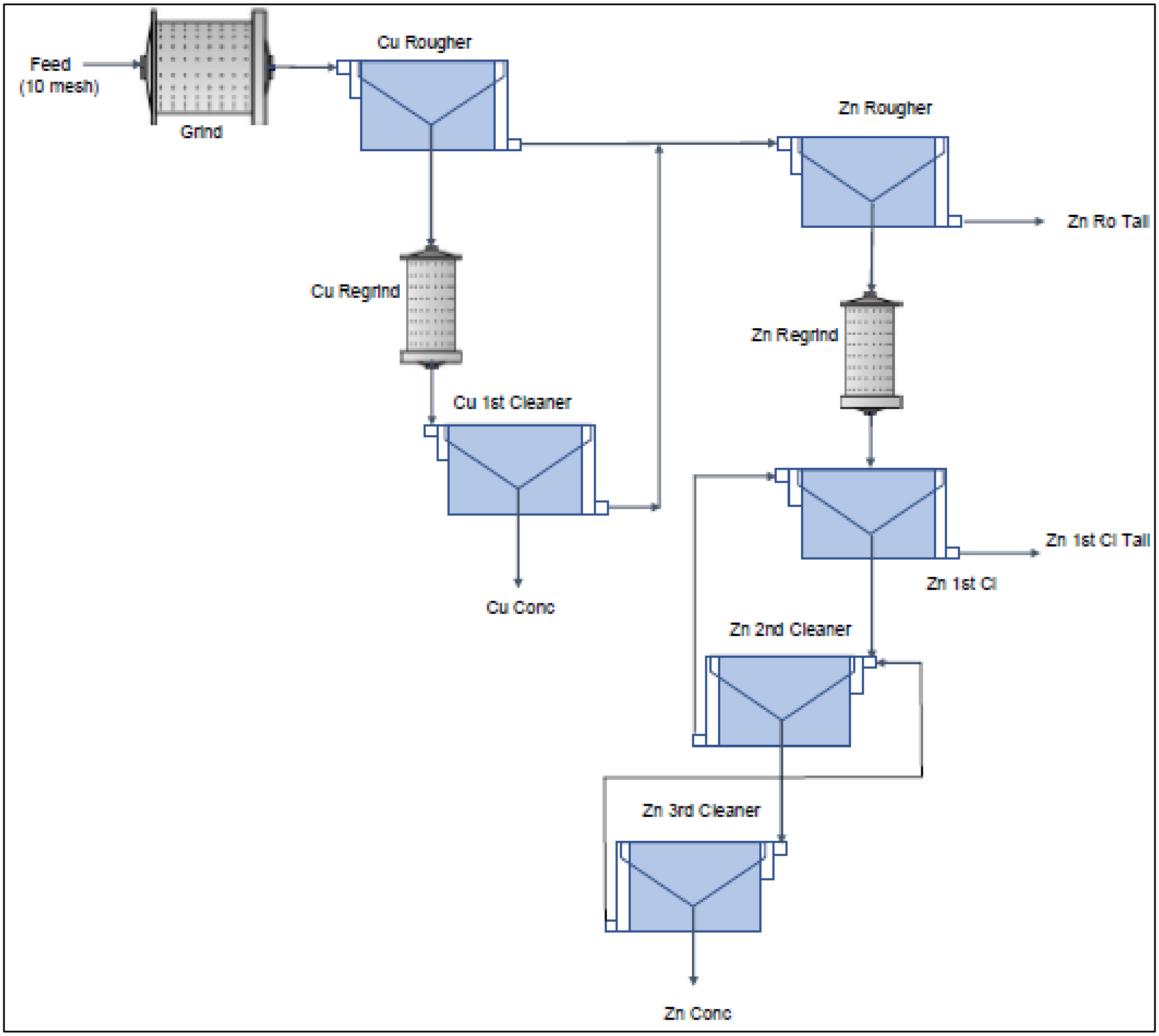


Figure 13-1: Flotation Flowsheet

Similarly, the Esso Lens sample EC1 was found to not be fully representative of the Esso deposit, and additionally consisted of aged drill core material. Sample EC1 was therefore used for developmental test purposes only. The sample EC2 was more representative and the test LCT-33 represents the results anticipated from the treatment of Esso material using the final flowsheet iteration. The test results from EC2 were better than the results from the Main Lens. A lower zinc content was obtained in the copper concentrate, and the zinc rougher concentrate did not require regrinding to achieve a marketable grade with reasonable recovery values.

13.3.4.3 Mineralogy

The Bureau Veritas Commodities study conducted an intensive particle size mineralogical evaluation, as described in Section 13.3.2. An additional mineralogical investigation was conducted by BML on sample material from the Main Lens deposit and the Esso Lens deposit in this study. The results are presented in Table 13-14 (and Table 13-15) and details the respective mineral composition, liberation, the chalcopyrite/bornite ratio, the pyrite content, and the copper mineral deportment of the samples together with the Main 4 Composite (MC4) sample. Note that the copper values are given as % Cu as a proportion of Total Cu.

Table 13-14: Copper mineral deportment and pyrite content values

Mineral/Item	Main Lens	Esso Lens	MC4 Sample
Chalcopyrite, % Cu	56.30	73.00	44.90
Bornite, % Cu	32.30	13.20	47.30
Chalcocite, % Cu	6.13	11.60	5.00
Covellite, % Cu	3.29	0.80	1.00
Tennantite/Enargite, % Cu	1.78	1.30	1.80
Malachite/Azurite, % Cu	0.20	0.10	0.10
Total, % Cu	100.00	100.00	100.00
Cp/Bn Ratio	1.74	5.53	0.95
Pyrite	43.0	10.0	47.1

The chalcopyrite/bornite ratio is seen to vary widely from about 1.7 in the Main Lens, to 5.5 in Esso Lens, with the ratio being even lower (namely about 0.9) for the MC4 sample composited for the metallurgical test program. The pyrite content of the is also seen to vary greatly for the two lenses (Main and Esso), while MC4 closely represents the Main Lens in relation to the pyrite content.

Bornite normally represents a more difficult copper mineral to recover by flotation, and generally a higher bornite content implies a reduced recovery of copper. Although not significantly different, this difference is shown in Table 13-15 where the results of the various tests are given indicating that material from the Main deposit (with a higher bornite content) gives lower copper recoveries than the material from the Esso deposit.

Table 13-15: Chalcopyrite and bornite liberation values, MC4 sample

Size fraction (µm)	Chalcopyrite	Bornite
-99 + 75	20.0	8.7
-75 + 53	27.6	16.9
-53 + 38	31.6	20.0
-38 + 20	39.8	29.2
-20 + 0	75.5	44.9

Liberation studies were also conducted on sample material from the MC4 sample. The sample was ground to a P_{80} size of 70 µm and then sized and evaluated for liberation and binary associations. The following table gives the liberation value for the different size fractions studied for both chalcopyrite and bornite.

Bornite liberation is poor even at the smallest P_{80} size range of 20 µm, while the liberation value for chalcopyrite is also relatively poor and only becomes reasonable for flotation recovery in the finest P_{80} size range smaller than 20 µm. By comparison, the findings of the BVC mineralogical evaluation for "copper sulphide minerals" was significantly higher although varied, namely 61% for a grind size P_{80} of 46 µm, and 74% for a grind size P_{80} of 60 µm. This result also highlights the varied nature of the copper-bearing minerals with different sample origins from different parts of the Kutcho deposit. The binary studies also indicated a relatively high value for bornite and pyrite and these bornite-pyrite composite particles would likely not be recovered during flotation.

13.3.4.4 Flotation Flowsheet Optimization

Flowsheet development originally used the KUTCHO COPPER, KM Global and Avg S Composite samples. This was subsequently followed by the Main and Esso Composite samples for further optimization studies. The initial test work also evaluated the bulk copper-zinc flotation procedure and the sequential copper-zinc procedure. Variability tests were also conducted using the original KUTCHO COPPER Composite sample, the Avg S Composite, and different Main and Esso Composite samples compiled over the duration of the test program. The effect of using inert versus steel regrind media was also determined, while a 50%/50% mixture of Main and Esso sample material was also conducted to establish whether the mixing of the ores resulted in any negative interactions.

Origin of Sample Material Tested

The origin of the sample material used for testing was described in an earlier section (see Section 13.3.4.2).

Study of the Depressant Systems

Conventionally, a zinc cyanide reagent system was employed in the depression of pyrite in the flotation of copper-zinc ores and the subsequent separation into two saleable copper and zinc concentrates. However, during the test program, it became apparent that using the zinc-cyanide depressant system was ineffective in the copper-zinc separation process in that copper recovery was reduced while the recovery of the faster floating and more reactive zinc was increased leading to a relatively high zinc content in the copper concentrate.

The SO_2-based depressant system was therefore tested and subsequently adopted and optimized and proved to be more successful in depressing the zinc in the copper flotation stage as well as depressing pyrite. Following extensive testing, this depressant system was used as the basis for the selective flotation of copper and zinc and the production of the two marketable concentrates. This system involved the conditioning of the slurry prior to copper flotation, initially with zinc sulphate, then followed by the addition of sodium sulphide, SO_2 as sulphurous acid while maintaining a pH value of 4.5 to 6.0. Lime was used to control the pH at 6.0 following which the collector reagents and frother was added and the flotation process commenced. Alternative sources of SO_2 were also tested, like sodium metabisulphite (SMBS), and varying the amounts of SO_2 addition were also evaluated. Various collector reagents were also tested, but the results obtained were inconclusive and further tests involving different collector reagents was not pursued further. The collector reagents used, namely Aero 3894 (a dialkyl thiocarbamate), and Aero 3477 (a dialkyl dithiophosphate), largely remained the same throughout the test program and were used as collector reagents for both the copper and the zinc flotation circuits. The frother reagents used were the conventional methyl isobutyl carbinol (MIBC) for the copper flotation circuit, and Frother W31, a stronger glycol-ether frother than MIBC, for the zinc circuit. Although the reagent SMBS is generally easier to control and easier to add to the circuit than SO_2 as sulphurous acid, the results obtained with SMBS were not as good as with using SO_2, and proportionally larger amounts of SMBS had to be used in the circuit. However, additional amounts of SO_2 were added to the conditioning stage as SMBS. Also, the stage addition of SO_2 was important for the effective use of SO_2 in the circuit and SO_2 provides the ability to control and lower the reactivity of zinc in the copper circuit by reducing the zinc content in the copper concentrate to < 5% Zn, although some copper recovery is also lost in the process. The amount of pyrite recovered was also better controlled with the SO_2-based system. During the testing, it was established that the regrinding step had to be conducted with an inert grinding media, such as ceramic material, rather than the more conventional steel or stainless steel grinding media.

This test program ultimately led to the definitive flowsheet as shown in Figure 13-1. The flowsheet involves the grinding of the material to a P_{80} grind size of 55 µm, followed by the sequential flotation of copper, then by the zinc flotation step. The copper rougher concentrate is reground to a P_{80} size of 15 µm followed by one cleaner flotation stage to produce a saleable copper concentrate product. The copper rougher tailings and the copper cleaner tailings are combined to feed the zinc flotation stage. The zinc flotation rougher concentrate is reground to a P_{80} size of 20 µm which is followed by three stages of cleaning for upgrading to a marketable zinc concentrate product. The cleaner tailings streams are returned to the preceding stage except the first cleaner tailings which are discarded together with the zinc rougher stage tailings.

Bulk Flotation versus Sequential Flotation Procedure.

Historically, some of the tests conducted on Kutcho material used the bulk flotation flowsheet, followed by the separation of the copper and zinc into the respective concentrates. The BML test program also conducted a limited open cycle test procedure comparing the results of using the bulk flotation flowsheet with the sequential flowsheet. The MC4 sample tested resulted in comparatively similar results although the sequential procedure resulted in slightly higher precious metal recovery values. The sequential method was retained as the preferred flotation procedure.

Variability Tests

A number of samples were specifically created for this series of open cycle tests by combining representative portions of potentially mineable core with barren material added for dilution of the sample. The results obtained varied as was expected, with a range of recovery values and grades obtained. However, it was also found that the overall effect of reducing the amount of pyrite in the sample being tested led to a significant improvement in flotation behaviour.

The results obtained from the variability test program can be summarized as follows.

- Copper Cleaner Concentrate: recovery varied between 52% and 89%, with concentrate grades varying from 17% to 27% Cu
- Copper Rougher Concentrate: copper recovery varied between 68% and 93% with between 19% and 39% Zn reporting to the copper rougher concentrate
- Zinc Cleaner Concentrate: recovery varied between 40% and 72%, with grades from 32% to 61% Zn
- Zinc Rougher Concentrate: zinc recovery varied between 47% and 78% with copper content varying between 2% and 10% Cu reporting to the zinc rougher concentrate
- Precious metal recovery values into the copper cleaner concentrate varied from 20% to 61% for silver, and 30% to 45% for gold
- Precious metal recovery values into the zinc cleaner concentrate were significantly lower compared with the copper concentrate and varied from 3% to 7% for silver and 3% to 8% for gold.

Locked Cycle Tests

A series of nine locked cycle tests were performed during this test program using various sample materials. All the tests were conducted using the flowsheet shown in Figure 13-1 with varying reagent additions, and possible changes to the number of cleaner stages in either of the circuits. The test results are all given in Table 13-16 and includes the Projected Flotation Data based on the proposed mine plan of early 2020 with the 75%/25% Main/Esso ore feed split, and the revised anticipated flotation results taking the effect of the ore sorter circuit into account. Various other locked cycle test results from historical test programs are also included for purposes of comparison.

Table 13-16: Locked cycle test results – actual and projected metallurgical results

LCT-12, KUTCHO COPPER Composite Sample, BL483, 2020									
Product	Mass, %	Concentrate grade				Metal recovery (%)			
		Cu, %	Zn, %	Au, g/t	Ag, g/t	Copper	Zinc	Gold	Silver
Cu Concentrate	4.8	31.3	6.32	2.20	314	80.0	16.3	29.7	47.0
Zn Concentrate	2.2	0.82	54.3	0.85	96	1.0	66.5	5.4	6.8
Feed	100.0	1.78	1.80	0.33	35.3	-	-	-	-
Flotation Conditions: Primary grind P_{80} 75 µm; Cu regrind P_{80} 17 µm; Zn regrind P_{80} 23 µm Sequential Cu-Zn flotation; Cu circuit, 2 cleaner stages; Zn circuit, 2 cleaner stages									
LCT-17, Avg S Composite Sample, BL483, 2020									
Product	Mass, %	Concentrate grade				Metal recovery, %			
		Cu, %	Zn, %	Au, g/t	Ag, g/t	Copper	Zinc	Gold	Silver
Cu Concentrate	7.5	26.6	4.65	2.76	347	92.3	9.7	36.1	70.6
Zn Concentrate	5.1	0.88	59.7	3.51	77	2.1	84.2	31.2	10.6
Feed	100.0	2.16	3.61	0.57	37	-	-	-	-
Flotation Conditions: Primary grind P_{80} 49 µm; Cu regrind P_{80} 16 µm; Zn regrind P_{80} 22µm Sequential Cu-Zn flotation; Cu circuit, 1 cleaner stage; Zn circuit, 2 cleaner stages									
LCT-18, KUTCHO COPPER Composite Sample, BL483, 2020									
Product	Mass, %	Concentrate grade				Metal recovery, %			
		Cu, %	Zn, %	Au, g/t	Ag, g/t	Copper	Zinc	Gold	Silver
Cu Concentrate	5.5	27.7	7.63	2.19	275	79.3	21.3	33.1	53.4
Zn Concentrate	2.4	1.89	55.2	1.99	97	2.4	68.3	13.3	8.3
Feed	100.0	1.92	1.96	0.36	28	-	-	-	-
Flotation Conditions: Primary grind P_{80} 75 µm; Cu regrind P_{80} 14 µm; Zn regrind P_{80} 23 µm Sequential Cu-Zn flotation; Cu circuit, 1 cleaner stage; Zn circuit, 2 cleaner stages									
LCT-19, KUTCHO COPPER Composite Sample, BL483, 2020									
Product	Mass, %	Concentrate grade				Metal recovery, %			
		Cu, %	Zn, %	Au, g/t	Ag, g/t	Copper	Zinc	Gold	Silver
Cu Concentrate	5.5	27.1	5.96	2.06	306	82.6	17.4	32.3	49.5
Zn Concentrate	2.0	0.86	58.6	1.33	90	1.0	63.2	7.7	5.4
Feed	100.0	1.81	1.89	0.35	34	-	-	-	-
Flotation Conditions: Primary grind P_{80} 75 µm; Cu regrind P_{80}14 µm; Zn regrind P_{80} 23 µm Sequential Cu-Zn flotation; Cu circuit, 1 cleaner stage; Zn circuit, 2 cleaner stages									
LCT-25, Main Composite 1 (MC1) Sample, BL483, 2020									
Product	Mass, %	Concentrate grade				Metal recovery, %			
		Cu, %	Zn, %	Au, g/t	Ag, g/t	Copper	Zinc	Gold	Silver
Cu Concentrate	3.9	33.1	5.10	1.99	307	75.6	11.5	27.9	39.9
Zn Concentrate	1.8	0.83	58.8	2.53	44	0.9	58.5	15.8	2.6
Feed	100.0	1.72	1.76	0.28	30.4	-	-	-	-
Flotation Conditions: Primary grind P_{80} 75 µm; Cu regrind P_{80} 15 µm; Zn regrind P_{80} 19 µm Sequential Cu-Zn flotation; Cu circuit, 3 cleaner stages; Zn circuit, 3 cleaner stages									
LCT-33, Esso Composite 2 (EC2) Sample, BL483, 2020									
Product	Mass, %	Concentrate grade				Metal recovery, %			
		Cu, %	Zn, %	Au, g/t	Ag, g/t	Copper	Zinc	Gold	Silver
Cu Concentrate	8.8	27.9	4.45	4.57	565	94.5	7.1	71.2	40.8
Zn Concentrate	8.4	0.62	58.2	2.95	96	2.0	89.3	11.6	25.2
Feed	100.0	2.60	5.51	0.99	70	-	-	-	-
Flotation Conditions: Primary grind P_{80} 49 µm; Cu regrind P_{80} 16 µm; no Zn regrind Sequential Cu-Zn flotation; Cu circuit, 1 cleaner stage; Zn circuit, 2 cleaner stages									

LCT-34, Main Composite 1 (MC1) Sample, BL483, 2020									
Product	**Mass, %**	**Concentrate grade**				**Metal recovery, %**			
		Cu, %	**Zn, %**	**Au, g/t**	**Ag, g/t**	**Copper**	**Zinc**	**Gold**	**Silver**
Cu Concentrate	5.9	27.2	9.12	2.45	292	85.0	22.0	57.4	47.0
Zn Concentrate	3.0	1.69	58.0	1.79	85	2.7	71.1	8.5	17.5
Feed	100.0	1.87	2.43	0.30	29	-	-	-	-
Flotation Conditions: Primary grind P_{80} 53 µm; Cu regrind P_{80} 17 µm; Zn regrind P_{80} 20 µm									
Sequential Cu-Zn flotation; Cu circuit, 1 cleaner stage; Zn circuit, 2 cleaner stages									
LCT-71, Main Composite 4 (MC4) Sample, BL483, 2020									
Product	**Mass, %**	**Concentrate grade**				**Metal recovery, %**			
		Cu, %	**Zn, %**	**Au, g/t**	**Ag, g/t**	**Copper**	**Zinc**	**Gold**	**Silver**
Cu Concentrate	4.9	29.0	10.40	2.24	285	84.3	21.3	34.3	49.8
Zn Concentrate	3.0	1.95	53.1	0.72	81	3.5	67.5	6.9	8.8
Feed	100.0	1.69	2.39	0.32	28	-	-	-	-
Flotation Conditions: Primary grind P_{80}54 µm; Cu regrind P_{80} 15 µm; Zn regrind P_{80} 17 µm									
Sequential Cu-Zn flotation; Cu circuit, 1 cleaner stage; Zn circuit, 2 cleaner stages									
LCT-73, Main Composite 4 (MC4) Sample, BL483, 2020									
Product	**Mass, %**	**Concentrate grade**				**Metal recovery, %**			
		Cu, %	**Zn, %**	**Au, g/t**	**Ag, g/t**	**Copper**	**Zinc**	**Gold**	**Silver**
Cu Concentrate	6.0	25.0	11.80	2.09	280	87.4	30.9	36.9	59.0
Zn Concentrate	2.7	1.80	54.6	3.03	98	2.8	63.8	23.0	9.2
Feed	100.0	1.72	2.30	0.34	29	-	-	-	-
Flotation Conditions: Primary grind P_{80} 49 µm; Cu regrind P_{80} 16 µm; Zn regrind P_{80} 22µm									
Sequential Cu-Zn flotation; Cu circuit, 1 cleaner stage; Zn circuit, 3 cleaner stages									
75% Main / Esso 25% Metallurgical Projection – LCT33 and LCT-73, BL483, 2020									
Product	**Mass, %**	**Concentrate grade**				**Metal recovery, %**			
		Cu, %	**Zn, %**	**Au, g/t**	**Ag, g/t**	**Copper**	**Zinc**	**Gold**	**Silver**
Cu Concentrate	6.7	26.0	9.41	2.91	375	89.8	20.3	38.8	64.5
Zn Concentrate	4.1	1.20	56.6	2.99	97	2.5	75.1	24.5	10.3
Feed	100.0	1.94	3.10	0.50	39	-	-	-	-
Flotation Conditions: Primary grind P_{80}55 µm; Cu regrind P_{80} 15 µm; Zn regrind P_{80} 20 µm									
Sequential Cu-Zn flotation; Cu circuit, 1 cleaner stage; Zn circuit, 3 cleaner stages									
2021 Mine Plan, Feasibility Study, Metallurgical Projections, September 2021									
Product	**Mass, %**	**Concentrate grade**				**Metal recovery, %**			
		Cu, %	**Zn, %**	**Au, g/t**	**Ag, g/t**	**Copper**	**Zinc**	**Gold**	**Silver**
Cu Concentrate	6.07	26.03	10.29	2.85	310	89.2	23.9	38.6	60.9
Zn Concentrate	3.46	1.43	55.01	3.15	80	2.8	72.8	24.5	9.0
Feed	100.0	1.77	2.58	0.42	31	-	-	-	-
Flotation Conditions: Primary grind P_{80} 55 µm; Cu regrind P_{80} 15 µm; Zn regrind P_{80} 20 µm									
Sequential Cu-Zn flotation; Cu circuit, 1 cleaner stage; Zn circuit, 3 cleaner stages									
Metallurgical Projection, Wardrop PFS, 2007									
Product	**Mass, %**	**Concentrate grade**				**Metal recovery, %**			
		Cu, %	**Zn, %**	**Au, g/t**	**Ag, g/t**	**Copper**	**Zinc**	**Gold**	**Silver**
Cu Concentrate	5.88	30.6	4.18	2.15	204	87.2	8.6	31.0	36.5
Zn Concentrate	4.12	0.51	54.45	0.19	22	1.0	78.8	2.0	2.7
Feed	100.0	1.63	2.26	0.32	26	-	-	-	-
Flotation Conditions: Primary grind P_{80} 75 µm; bulk conc regrind P_{80} 15 µm; Cu regrind P_{80} 10 µm									
Bulk Cu-Zn flotation; Cu circuit, 3 cleaner stages; Zn circuit, 4 cleaner stages, no Zn regrind									

LCT-F60, SRK PEA, 2008									
Product	Mass, %	Concentrate grade				Metal recovery, %			
		Cu, %	Zn, %	Au, g/t	Ag, g/t	Copper	Zinc	Gold	Silver
Cu Concentrate	5.51	34.8	3.43	4.34	385	85.9	7.8	44.3	50.0
Zn Concentrate	3.39	1.01	56.9	0.65	68	1.5	78.0	4.1	5.5
Feed	100.0	2.48	2.48	0.54	43	-	-	-	-
Flotation Conditions: Primary grind P_{80} 74 µm; bulk conc regrind P_{80} 18 µm; Cu regrind P_{80} 9 µm Bulk Cu-Zn flotation; Cu circuit, 3 cleaner stages; Zn circuit, 4 cleaner stages, no Zn regrind									
Projected Metallurgical Balance, SRK PEA, 2008									
Product	Mass, %	Concentrate grade				Metal recovery, %			
		Cu, %	Zn, %	Au, g/t	Ag, g/t	Copper	Zinc	Gold	Silver
Cu Concentrate	4.70	30.30	4.00	3.64	330	82.4	7.7	49.9	52.4
Zn Concentrate	3.06	0.50	55.0	0.43	37	0.9	72.0	4.0	4.0
Feed	100.0	1.73	2.34	0.27	26	-	-	-	-
Flotation Conditions: Primary grind P_{80} 40 µm; Cu regrind P_{80} 10 µm; no Zn regrind Sequential Cu-Zn flotation; Cu circuit, 2-3 cleaner stages; Zn circuit, 3-4 cleaner stages									
LCT-Global Composite Sample, JDS PFS, 2011									
Product	Mass, %	Concentrate grade				Metal recovery, %			
		Cu, %	Zn, %	Au, g/t	Ag, g/t	Copper	Zinc	Gold	Silver
Cu Concentrate	5.5	27.7	8.5	1.91	224	82.1	16.2	41.1	45.0
Zn Concentrate	3.7	1.60	54.0	0.52	37	3.3	69.5	7.0	5.0
Feed	100.0	1.85	2.88	0.25	27	-	-	-	-
Flotation Conditions: Primary grind P_{80} 75 µm; Cu regrind P_{80} 40 µm; no Zn regrind Sequential Cu-Zn flotation; Cu circuit, 3 cleaner stages; Zn circuit, 2 cleaner stages									
Metallurgical Projection, JDS PFS, 2017									
Product	Mass, %	Concentrate grade				Metal recovery, %			
		Cu, %	Zn, %	Au, g/t	Ag, g/t	Copper	Zinc	Gold	Silver
Cu Concentrate	5.66	27.6	7.3	2.5	269	84.7	7.3	41.2	48.0
Zn Concentrate	3.96	1.20	55.1	0.52	37	3.3	75.7	7.6	5.0
Feed	100.0	2.01	3.19	0.40	40	-	-	-	-
Flotation Conditions: Primary grind P_{80} 75 µm; Cu regrind P_{80} 35 µm; no Zn regrind Sequential Cu-Zn flotation; Cu circuit, 3 cleaner stages; Zn circuit, 3 cleaner stages									
LCT, Mineralogical Study, BVC 2019									
Product	Mass, %	Concentrate grade				Metal recovery, %			
		Cu, %	Zn, %	Au, g/t	Ag, g/t	Copper	Zinc	Gold	Silver
Cu Concentrate	6.2	24.8	9.35	n/a	n/a	82.3	23.7	-	-
Zn Concentrate	3.1	0.62	53.0	n/a	n/a	1.0	67.3	-	-
Feed	100.0	1.87	2.45	n/a	n/a	-	-	-	-
Flotation Conditions: Primary grind P_{80} 60 µm; Cu regrind P_{80} 19 µm; no Zn regrind Sequential Cu-Zn flotation; Cu circuit, 3 cleaner stages; Zn circuit, 3 cleaner stages									
Hydrometallurgical Study, DRA, 2020 (based on 75% Main Deposit / 25% Esso Deposit)									
Product	Mass, %	Concentrate grade				Metal recovery, %			
		Cu, %	Zn, %	Au, g/t	Ag, g/t	Copper	Zinc	Gold	Silver
Cu Concentrate	6.7	27.0	9.4	2.9	377	89.5	20.3	37.9	61.3
Zn Concentrate	4.6	1.2	56.0	2.2	89	2.5	72.8	18.3	9.5
Feed	100.0	1.85	2.72	0.49	34	-	-	-	-
Flotation Conditions: Primary grind P_{80} 55 µm; Cu regrind P_{80} 15 µm; Zn regrind P_{80} 20 µm Sequential Cu-Zn flotation; Cu circuit, 1 cleaner stage; Zn circuit, 3 cleaner stages									

The results from the locked cycle tests vary to a minor extent but show that the results are consistent for each sample type indicating that the flowsheet is robust and the results reproduceable. The results also demonstrate that the Esso material responds better to the flowsheet and that the recovery values are higher. Similarly, the KUTCHO COPPER Composite and Avg S Composite samples gave improved results compared with the Main Lens samples, specifically MC1 and MC4. However, the sample material from KUTCHO COPPER Composite and Avg S Composite was not representative and this was replaced by the MC samples. However, the KUTCHO COPPER Composite and Avg S Composite samples did allow basic test work to be conducted which ultimately developed into the final flotation process flowsheet.

The final optimization of the flowsheet was conducted using MC4 sample material with tests LCT-71 and LCT-73. The difference between these tests was in the number of cleaner stages for zinc, and that for test LCT-71, the copper cleaner tailings stream was recycled back to the feed stream of the rougher flotation stage. The results from test LCT-71 were similar to those for LCT-73 for copper and zinc grades and recoveries. However, test LCT-73 gave higher recoveries for gold and silver into the zinc concentrate, and it was decided to use the LCT-73 flowsheet to take advantage of the higher payable value of the zinc concentrate, in other words, discard the copper cleaner tailings rather than recycle to the head of the copper flotation stage. An additional concern was that the recycling of the reground (fine) cleaner tailings to the head of the flotation circuit would lead to the interference with the performance of the rougher circuit through the building up of a high circulating load.

The projected metallurgical recovery and grade values was determined for the original 75% Main and 25% Esso mine plan and was based on the results obtained from test LCT-33 (Esso sample EC2) and test LCT-73 (Main sample MC4). However, the ratio of Main and Esso ore has changed with the advent of the ore sorter installed in the processing circuit resulting in a different ore mix of underground material and open pit ore feeding the plant. However, blending tests conducted during the test program indicated that there were no negative synergies when mixing the two ore types from Main and Esso. It is therefore considered that the metallurgical behaviour will not be affected in any way.

A further review of the results of the various locked cycle tests conducted and reported in Table 13.16 indicates that:

- The samples tested early in the test program, namely the KUTCHO COPPER Composite, KM Global and Avg S Composite samples, helped develop the flotation flowsheet.
- The sequential flowsheet was demonstrated to be stable and robust and has given consistent results.
- The Esso material demonstrates improved metallurgical performance with respect to metal recoveries and grades, and the amount of zinc in the copper concentrate and copper in the zinc concentrate. Hypothetically, if possible, Esso should be mined and processed alone which could result in the shutting down of the zinc regrind circuit, and possibly the third cleaner stage, and obtaining a penalty-free concentrate which could also be used for subsequent blending purposes with the Main concentrates where necessary.
- The results from tests LCT-33 and LCT-73 were used to produce the anticipated projected recovery and grade details for the process.

13.3.4.5 Concentrate Analyses

Detailed concentrate analyses were conducted on the products from four of the locked cycle tests. The concentrates analyzed were from test LCT-17 (Avg S Composite sample), test LCT-33 (EC2 sample), test LCT-71 (MC4 sample), and test LCT-73 (MC4 sample). The results of the analyses are given in Table 13-17. The differences between the concentrates from Main and Esso can be clearly seen. Note that carbon and germanium analyses were not conducted (the germanium analysis would be required if the zinc concentrate was destined for a roaster and electrowinning plant).

Table 13-17: Copper and zinc concentrate analyses

Element	Units	LCT-17		LCT-33		LCT-71		LCT-73	
		Cu conc.	Zn conc.	Cu conc.	Zn conc.	Cu conc.	Zn conc.	Cu conc.	Zn conc.
Cu	%	26.6	0.88	27.9	0.62	29.0	1.95	26.0	1.20
Zn	%	4.65	59.7	4.45	58.2	10.4	53.1	9.41	56.6
Au	ppm	2.76	3.51	4.57	2.95	2.24	0.72	2.91	2.99
Ag	ppm	347	77	565	96	285	81	375	97
S	%	34.6	34.1	35.5	35.3	35.9	35.9	33.8	35.7
Al	%	0.04	0.01	0.24	0.07	0.18	0.07	0.16	0.05
As	ppm	403	274	315	181	413	3	365	6
B	ppm	< 10	< 10	< 10	< 10	< 10	< 10	< 10	< 10
Ba	ppm	< 10	< 10	< 10	< 10	< 10	< 10	< 10	< 10
Be	ppm	< 0.5	< 0.5	< 0.5	< 0.5	< 0.5	< 0.5	< 0.5	< 0.5
Bi	ppm	19	12	< 2	10	8	17	7	19
Ca	%	0.36	0.19	0.06	0.01	0.16	0.15	0.11	0.10
Cd	ppm	207	2750	193	2960	459	2520	447	2520
Co	ppm	33	12	< 1	< 1	38	21	24	20
Cr	ppm	81	24	36	16	40	493	34	423
Fe	%	19.4	2.2	28.2	4.48	22.3	6.66	22.5	7.0
Ga	ppm	< 10	< 10	< 10	10	< 10	< 10	< 10	< 10
Hg	ppm	9	96	< 1	23	15	12	15	16
K	%	< 0.01	< 0.01	< 0.01	< 0.01	< 0.01	< 0.01	< 0.01	< 0.01
La	ppm	< 10	< 10	< 10	< 10	< 10	< 10	< 10	< 10
Mn	ppm	185	80	41	78	83	152	73	112
Mg	%	0.22	0.08	0.16	0.07	0.09	0.07	0.08	0.05
Mo	ppm	23	49	27	12	17	4	10	5
Na	%	0.013	0.011	0.015	0.014	0.014	0.014	0.014	0.013
Ni	ppm	53	18	19	11	22	291	20	247
P	%	0.062	0.007	0.073	0.007	0.071	0.004	0.076	00.3
Pb	%	1.87	0.18	1.01	0.06	0.501	0.06	0.513	0.068
Sb	ppm	18	13	206	114	34	4	36	5
Sc	ppm	< 1	< 1	< 1	< 1	< 1	< 1	< 1	< 1
Se	%	0.001	< 0.001	0.002	< 0.001	0.010	0.004	0.009	0.004
Si	%	0.92	0.19	0.51	0.16	0.28	0.15	0.24	0.11
Sr	ppm	5	3	2	2	2	2	2	1
Te	ppm	74	39	68	21	48	< 1	64	< 1
Th	ppm	< 20	< 20	< 20	< 20	< 20	< 20	< 20	< 20
Ti	%	< 0.01	< 0.01	< 0.01	< 0.01	< 0.01	< 0.01	< 0.01	< 0.01
Tl	ppm	5	< 2	< 2	< 2	3	< 2	2	3
U	ppm	< 10	< 10	< 10	< 10	< 10	< 10	< 10	< 10
V	ppm	10	4	5	2	4	4	4	3
W	ppm	25	< 10	< 10	< 10	19	< 10	18	< 10
Y	ppm	< 1	< 1	< 1	< 1	< 1	< 1	< 1	< 1
Zr	ppm	129	30	89	8	110	5	129	6
Cl	%	< 0.01	0.01	< 0.01	0.01	< 0.01	< 0.01	0.02	< 0.01
F	%	< 0.01	< 0.01	< 0.01	< 0.01	< 0.01	< 0.01	< 0.01	< 0.01

The potential penalty elements could be the following:

- Copper concentrate: apart from the zinc contained in the copper concentrate, cadmium, lead, arsenic and antimony could potentially become penalty elements.
- Zinc concentrate: apart from the copper contained in the zinc concentrate, the following elements could incur penalties; cadmium, cobalt, arsenic and possibly lead.

13.3.4.6 Precious Metal Recovery

The presence of unrecovered gold and silver associated with the pyrite present in the tailings led to a set of scoping tests to determine whether this gold and silver could be recovered from a pyrite concentrate. The pyrite in the tailings from the copper and zinc flotation circuit can readily be recovered as a pyrite concentrate by flotation and this process would also collect the associated gold and silver into the pyrite concentrate. The pyrite was floated and recovered as a pyrite concentrate which was subsequently leached in cyanide solution. Dissolution values of up to 90% for gold, 80% for silver, and 90% for copper were obtained. The leaching time was 48 hours, using a 2,500 mg/L sodium cyanide strength solution after regrinding the pyrite concentrate to a P_{80} particle size of 24 µm. Sodium cyanide consumption values ranged between 3.1 kg/t for Esso material, and 4.7 kg/t for the Main sample. Lime consumption values were 0.3 and 0.4 kg/t respectively. A Merrill-Crowe circuit was then proposed for the recovery of the gold and silver, while the SART process would be used to recover the copper (the SART process involves sulphidization for the precipitation of the copper, acidification, recycling of the copper precipitate and the thickening of the precipitated copper). The additional recovery for the three metals was projected to be about 31% for gold, 18% for silver, and 7% for copper.

A scoping level evaluation was conducted by DRA in 2020 using the data from the ongoing BML test program. Although shown to be economically attractive, the concept has not been pursued any further.

*13.3.5 **Base Metal Laboratories Test Program BL714, 2021***

13.3.5.1 Introduction

An additional test program was conducted as a continuation of the BL483 program, particularly to generate additional quantities of concentrate and tailings for specific testing. The flotation tests would also serve to confirm the reproducibility of the flotation flowsheet and the procedure developed for the deposit process. Solid-liquid separation tests were subsequently conducted on the concentrates and tailings samples generated, while a part of the tailings sample was forwarded to another test facility for paste characterization tests. Concentrate density values up to 65% solids, and tailings density values up to 70%, were obtained indicating that the products responded normally to thickening and dewatering. The flocculant MF351 was found to be suitable for aiding the settling of tailings and concentrates. Pressure filter tests reduced the concentrate moisture content to 12%, or lower.

13.3.5.2 Sample Material

The samples used for this test program were selected from the historical sample material held in storage at Base Metal Laboratories in Kamloops, BC. The samples were prepared in accordance with the procedure used previously to prepare the Main Composite 4 (MC4) and Esso Composite 2 (EC2) samples. The samples were used for the comminution tests and the subsequent flotation and dewatering testing.

13.3.5.3 Flotation Tests

Flotation testing was conducted as large-scale tests in order to generate the quantities of concentrate and tailings material required for the subsequent respective test programs. The Main Lens material was processed using the locked cycle test flowsheet which had been developed and optimized in the earlier test program (BL483). The Esso Lens material was treated using an open circuit flotation procedure but using the same reagents and also following the procedure developed in BL483. In each test, the standard reagents were added as developed previously, namely using zinc sulphate and the SO_2/Na_2S reagents for zinc and pyrite depression in the copper flotation stage which was followed by the zinc flotation stage following the activation of zinc with copper sulphate. In each case the respective amounts of collector and frother reagents were also added. The results obtained are detailed in Table 13-18.

Table 13-18: Flotation results for Main and Esso samples, BL714

LCT Main Lens Sample, BL714, 2021									
Product	**Mass, %**	**Concentrate grade**				**Metal recovery, %**			
		Cu, %	**Zn, %**	**Au, g/t**	**Ag, g/t**	**Copper**	**Zinc**	**Gold**	**Silver**
Cu Concentrate	4.2	27.9	6.06	1.94	293	70.8	13.6	24.8	47.1
Zn Concentrate	1.1	3.17	56.2	2.28	195	2.1	32.7	7.5	8.1
Feed	100.0	1.66	1.87	0.33	26	-	-	-	-
Flotation Conditions: Primary grind P_{80}54 µm; Cu regrind P_{80} 12 µm; Zn regrind P_{80}14 µm Sequential Cu-Zn flotation; Cu circuit, 1 cleaner stage; Zn circuit, 3 cleaner stages									
LCT Esso Lens Sample, BL714, 2021									
Product	**Mass, %**	**Concentrate grade**				**Metal recovery, %**			
		Cu, %	**Zn, %**	**Au, g/t**	**Ag, g/t**	**Copper**	**Zinc**	**Gold**	**Silver**
Cu Concentrate	8.6	27.0	4.61	4.88	545	94.5	5.9	61.4	77.7
Zn Concentrate	7.6	0.20	62.6	0.47	54	0.6	70.0	5.4	6.8
Feed	100.0	2.46	6.75	0.68	60	-	-	-	-
Flotation Conditions: Primary grind P_{80} 55 µm; Cu regrind P_{80}18 µm; no Zn regrind Sequential Cu-Zn flotation; Cu circuit, 1 cleaner stage; Zn circuit, 2 cleaner stages									

The results obtained from the flotation tests were significantly poorer than the results previously obtained, respectively from test LCT-73 for Main Lens, and from test LCT-33 for Esso Lens. This was ascribed to a scale-up problem with the depressant reagents from typically testing a 2 kg sample to using 10 kg charges in this particular test. For the Main Lens, a 50 kg sample was used for the LCT (10 kg per cycle) followed by two 10 kg sample charges. For the Esso Lens, the open cycle tests used a 30 kg sample followed by two 10 kg sample charges. Also, since there was no excess sample material available, no trial tests could be undertaken since the complete sample had to be used to generate the required tailings material for paste testing. These poorer results have confirmed the need for close control of the plant feed, the reagent addition and the conditioning of flotation feed during plant operations and the need for some optical or "expert" system to assist the process plant operators.

13.3.5.4 Process Water Analysis

It was considered that a threat to the proper functioning of the flotation plant could be an accumulation of reagents in the recycled process water streams, particularly since relatively high dosages of sulphur-bearing reagents were being added to depress the flotation of pyrite and zinc in the copper flotation circuit. To determine whether an accumulation of any metal(s) could be occurring in the process water in the flotation circuit, the water from the Main Lens LCT tailings from each cycle was sampled and analyzed. Table 13-19 presents the results obtained from an analysis of the tailings water from each cycle. The results indicate that, possibly apart from sodium and copper, there does not appear to be a build-up of elements in the process water stream.

Table 13-19: Process water dissolved metals analysis

Metal	Units	Cycle 1	Cycle 2	Cycle 3	Cycle 4	Cycle 5
Aluminium	mg/L	< 0.0050	0.0044	0.0032	0.0025	0.0028
Antimony	mg/L	0.00458	0.00301	0.00213	0.00322	0.00302
Arsenic	mg/L	0.00094	0.00084	0.00092	0.00074	0.00092
Barium	mg/L	0.0669	0.0373	0.0366	0.0293	0.0360
Bismuth	mg/L	< 0.00025	< 0.00010	< 0.00010	< 0.00010	< 0.00010

Metal	Units	Cycle 1	Cycle 2	Cycle 3	Cycle 4	Cycle 5
Cadmium	mg/L	0.0095	0.0105	0.0118	0.0075	0.0103
Calcium	mg/L	628	421	388	339	396
Cobalt	mg/L	0.0347	0.0188	0.0250	0.0163	0.0186
Copper	mg/L	0.365	0.193	0.265	0.202	0.544
Iron	mg/L	< 0.050	< 0.020	< 0.020	< 0.020	< 0.020
Lead	mg/L	< 0.00025	0.00553	0.00253	0.00133	0.00068
Magnesium	mg/L	81.6	50.2	49.3	42.0	49.9
Manganese	mg/L	7.20	4.38	4.71	3.61	4.20
Molybdenum	mg/L	0.0257	0.0220	0.0201	0.0208	0.0271
Nickel	mg/L	0.1090	0.0974	0.1110	0.0810	0.1020
Potassium	mg/L	10.00	6.31	6.31	5.72	9.96
Selenium	mg/L	0.0777	0.0688	0.0659	0.0533	0.0642
Silicon	mg/L	3.39	3.15	3.34	3.01	2.97
Silver	mg/L	0.00704	0.00540	0.00542	0.00545	0.00899
Sodium	mg/L	126	149	163	151	187
Strontium	mg/L	1.050	0.734	0.708	0.624	0.716
Sulphur	mg/L	946	685	656	561	684
Uranium	mg/L	0.00845	0.01090	0.00714	0.00734	0.01050
Zinc	mg/L	11.20	9.36	11.20	6.96	8.00

Note that other elements were also tested, namely beryllium, boron, cesium, chromium, lithium, phosphorous, rubidium, tellurium, thorium, titanium, vanadium, tungsten, and zirconium, but these all reported below detection limit values in the water from all five cycles.

Although there was apparently no accumulation of elements in the flotation water, the design of the process water circuit has the respective copper and zinc concentrate and tailings thickener overflow solutions all reporting to the Water Treatment Plant (WTP) prior to its re-use in the flotation plant. The excess return water from the Tailings Management Facility will also be treated in the WTP. In this way, any reagents present in the process water will be degraded or oxidized or diluted prior to its return in the process water used in the flotation plant.

13.3.5.5 Dewatering Tests

Thickening tests were conducted for both concentrates and tailings samples. The concentrate samples also had flocculant scoping tests performed. The widely used flocculant Magnafloc MF351 was selected as giving the best results, and this flocculant was also used for the tailings thickening testing. Static thickening tests for the two concentrates at 20 to 30 g/t flocculant addition resulted in a 65% solids density value, while similar results were obtained for the tailings material. Dynamic thickening tests resulted loading rates between 0.3 t/m^2/h and 0.5 t/m^2/h while up to 0.7 t/m^2/h resulted in density values of 65–70% solids at flocculant dosage rates of 20–30 g/t.

Pressure filtration tests using sample material from both concentrates demonstrated that moisture values below 12% were attainable for both concentrates from both the Main and Esso Lens samples.

*13.3.6 **Discussion of Results***

Although the BML flowsheet was developed and demonstrated that it is robust and suitable in producing copper and zinc concentrates, the operation of the circuit and addition of SO_2-based depressants will need to be monitored closely to minimize the amount of zinc reporting to the copper concentrate and diluting the final grade of the concentrate and/or possibly incurring smelter penalties.

A comparison of the various results obtained from the different metallurgical test programs highlights that different results were reported in the various test reports. These differences can be ascribed to:

- The test programs and test procedures not being consistent, namely the testing of bulk flotation and sequential flotation, and the changes with the reagents used and the varying number of cleaning stages. This may have resulted in incomplete depression of copper and/or pyrite, and/or the different degrees of activation. The SO_2-based depressant reagents gave improved results compared with the zinc-cyanide system tested early in the program. However, different SO_2 reagents were tested until the present suite of SO_2 reagents was found to give the best results. Different collector and frother reagents were tested in various test programs but were generally found to give consistent results. Lime was used universally for pH control.
- Sample material selected for testing which changed over the years as the deposit became better understood and more accurately defined, and as the mine plan changed accordingly.
- The use of samples which had aged because of prolonged storage. Although not quantified, and also because many of the samples were stored under controlled conditions, it was regarded that this effect would not drastically change the results and conclusions drawn from the test programs.
- The differences in the metallurgical response for the copper minerals from both the Main and Esso deposits are as a result of the varying mineralogy, and also the differences in zinc response from the two deposits.
- The copper and zinc minerals in the Kutcho deposit were found to not follow the general particle size recovery relationship demonstrated by other copper-zinc deposits in the flotation process. The highest recovery for copper is in the range 10–20 µm, while for sphalerite the range is 40–60 µm. Copper mineral grains coarser than 38 microns are apparently not readily recovered, while for sphalerite the equivalent particle size appears to be >80 µm. The "normal" flotation particle size versus recovery for copper and zinc sulphide minerals indicates high flotation recoveries over the general size range from 10 µm to about 100 µm with the recovery only dropping off for coarser particles larger than about 125 µm. The apparent loss of very fine liberated particles into the tailings also indicates the importance of preventing overgrinding of the copper sulphide particles in particular.
- The findings of the mineralogical investigation conducted by BVC in 2019 and 2020 are surprising when compared with the earlier results such as those reported in the SRK 2008 report. SRK determined that a primary grind P_{80} size of 75 µm was optimal based on the test results available at the time. These results indicated that the primary grind values for both the copper minerals and sphalerite gave the highest recoveries but started decreasing for P_{80} particle sizes coarser than 80 µm. The recovery versus particle size was shown to be relatively insensitive to the primary grind size P_{80} in the range from 80 µm and finer up to about 42 µm which was the lowest P_{80} value tested. The regrind P_{80} size for the copper rougher concentrate was 40 µm, while there was no regrinding of the zinc rougher concentrate.
- The use of different primary grind and concentrate regrind sizes tested. Early test work indicated that a primary grind P_{80} size of 75 µm would give optimal results whereas a finer primary grind at 55 µm has given improved results. The copper rougher regrind value has remained at a P_{80} size of about 15 µm, but the zinc rougher regrind size has changed from a no regrind required to conducting a regrind at a particle size of about 20 µm.
- Different feed grades of the samples tested from the various areas of the deposit.
- The mineralogy of the deposit, including the significant presence of pyrite, the readily activated sphalerite as a result of the presence of copper, and the presence of several different copper mineral species each having its own characteristics such as particle liberation size, associations with other minerals, and response to flotation.
- The problem of upgrading of the copper concentrate is not considered to be as a result of a lack of grinding, but rather the incorporation of contaminants (pyrite and sphalerite) which are largely liberated and which

are subsequently misplaced into the copper concentrate owing to a lack of depression and/or inadvertent activation. In this regard, the chalcopyrite floats rapidly but the slower-floating bornite appears to be associated with the recovery of activated sphalerite.

13.3.7 Flotation Conclusions

The BML test program has led to the development of the flotation flowsheet shown in Figure 13-1. Test work has demonstrated that the flowsheet gives reproduceable and consistent results in producing marketable copper and zinc concentrate products and was adopted for the overall design of the flotation circuit. The flowsheet involves the grinding of the material to a P_{80} grind size of 55 µm, followed by the sequential flotation of copper, then by the zinc flotation step. The copper rougher concentrate is reground to a P_{80} size of 15 µm followed by one cleaner flotation stage to produce a marketable copper concentrate. The copper rougher tailings and the copper cleaner tailings are combined to feed the zinc flotation stage. The zinc flotation rougher concentrate is reground to a P_{80} size of 20 µm which is followed by three stages of cleaning for upgrading to produce a marketable zinc concentrate. The cleaner tailings streams are returned to the preceding stage except the first cleaner tailings which are discarded together with the zinc rougher stage tailings. The anticipated recoveries and grades are given in Table 13-16 but are reproduced in Table 13-20.

Table 13-20: Mine plan 2021, metallurgical projections

2021 Mine Plan, Feasibility Study, Metallurgical Projections, September 2021									
Product	Mass, %	Concentrate grade				Metal recovery, %			
		Cu, %	Zn, %	Au, g/t	Ag, g/t	Copper	Zinc	Gold	Silver
Cu Concentrate	6.07	26.03	10.29	2.85	310	89.2	23.9	38.6	60.9
Zn Concentrate	3.46	1.43	55.01	3.15	80	2.8	72.8	24.5	9.0
Feed	100.0	1.77	2.58	0.42	31	-	-	-	-

Part of the flotation test work was focused on the reduction of deleterious elements, particularly zinc in the copper concentrate using regrinding and reagent optimization to produce a concentrate that is considered saleable (refer to Section 19).

13.4 Mineral Sorting Test Work

Approximately 765 kg samples representing the Kutcho copper deposit was collected and sent to TOMRA, Germany, for the sorting study. The samples were mixed, crushed, and screened at 15 mm to remove excess fines (representing fines bypass).

A sensor-based sorting test was conducted on the +15 mm sample where particles are separately detected by a sensor and rejected by an amplified mechanical, hydraulic, or pneumatic process. The sensing technique used for the tested samples was x-ray transmission (XRT) due to the differences in the atomic density of the material containing mineralization and waste. For XRT sorting, a five-step cascade procedure was utilized where the product from each sorting step was reprocessed. The XRT conditions were highly selective to waste and producing rejects with the lowest metal concentration. With each additional scan, the selectivity to waste was reduced to produce rejects with increasing metal concentration. The final XRT product was reprocessed using Near Infrared (NIR) sensor to separate copper-zinc sulphides from the iron sulphides. The sorters used for the test work were TOMRA's COM Tertiary XRT and PRO Secondary COLOR-NIR. All the XRT and NIR sorted products, and the bypass fines (- 15 mm) were packed and sent to Base Metallurgical Laboratories Ltd (Kamloops, Canada) for metal assay.

13.4.1 Sorting Test Results

A single bulk composite sample was assembled from drill core intervals. The chosen material constituted a diverse selection of mineralization styles from the Main and Esso deposits, and it was inclusive of expected dilution haloes. The materials were blended at 20% Esso : 80% Main, which is approximately the average tonnage mix for most of the life of operation. The bulk composite grades were 1.37% Cu, 3.88% Zn, 27.8 g/t Ag and 0.44 g/t with fines and compare reasonably to the overall Mineral Reserve at 1.58% Cu, 2.31% Zn, 28 g/t Ag, and 0.39 g/t Au. The QP considers that for the ore sorter bulk sample the various types and styles of mineralization and the mineral deposit as a whole were represented appropriately to support the results obtained and used in the Feasibility Study. Recommendations for further work to support detailed design are included in Section 26.

Figure 13-2 summarizes the mass and metal grades obtained at each stage of the sorting test program. The XRT sorter sets represent the sensor's sensitivity to waste (1 being the least sensitive).

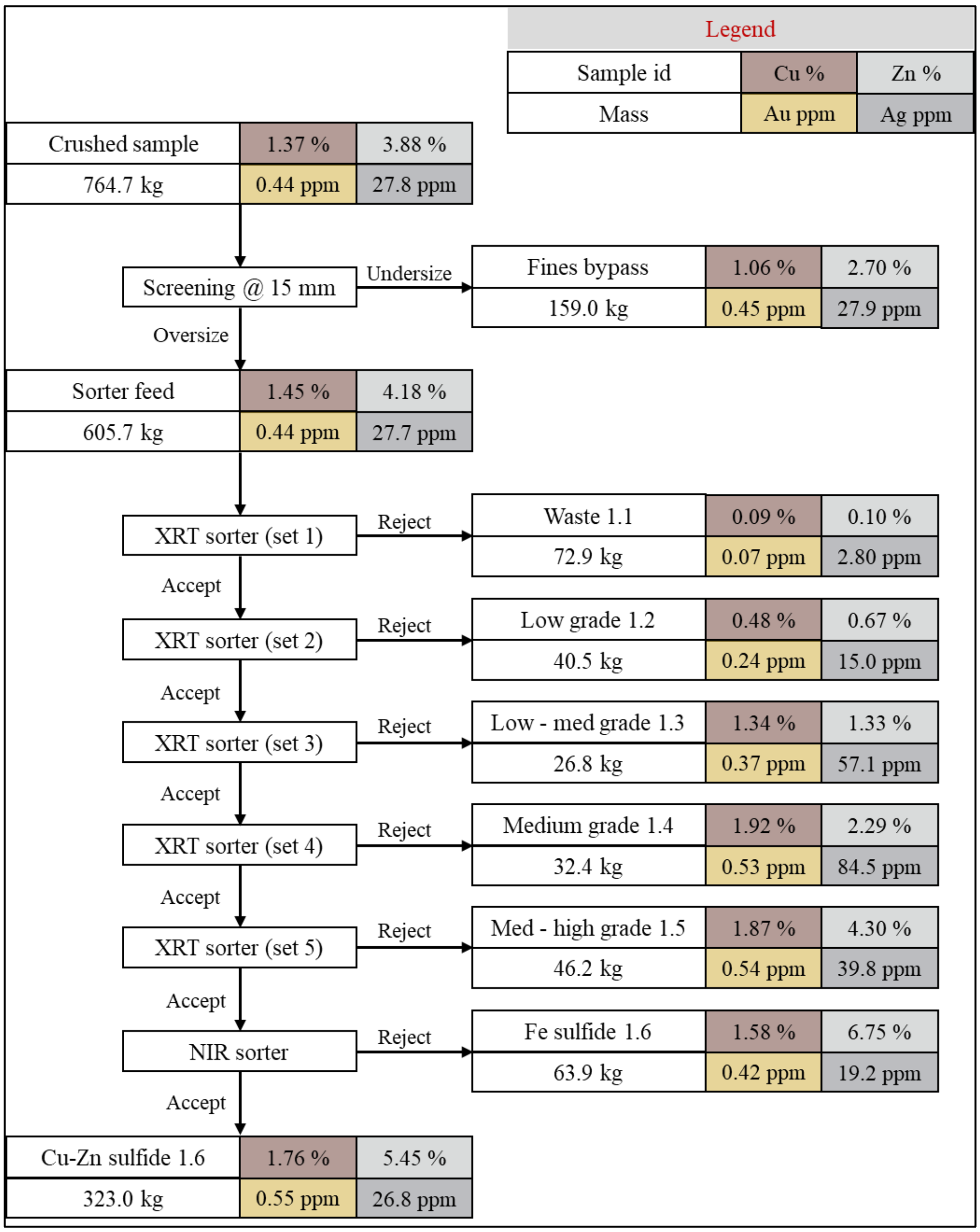


Figure 13-2: Result Summary from the Sorting Test Program

The results showed that the XRT sorter rejected up to 18.7% of the sorter feed (12% at set 1 and 6.7% at set 2) at metal grades significantly lower than the feed grades. The accept grades for the XRT sorter at set 3 to set 5 and NIR sorter was comparable to the feed grades. Overall, the results indicated that the XRT sorter at low sensitivity to waste could separate ore-bearing rocks from the waste rock. The NIR sorting results showed that separating different types of sulphides is not possible. The test results also showed a downgrading of the metal concentration in the fines bypass. The copper and zinc grades in the fines bypass were reduced to 1.06% and 2.70% compared to 1.37% and 3.88% in the feed. These downgrading factors were accounted for while determining the optimal mass pull and recovery from the ore sorter.

The results obtained from the sorting tests were used to construct the mass pull-recovery plots for the sorter for each metal of interest. Figure 13-3 shows the sorter grade-recovery analysis for each metal. Due to the very low effectiveness of the NIR sensor, only XRT sorting results were used for the analysis. The results indicated that the XRT sorter could reject 10–20% of the total feed to the sorter as waste while recovering 95–99% of the valuable metals, thus upgrading the metal concentration in the sorter product. The XRT mass pull-recovery data was further interpolated to estimate the optimal mass pull to maximize sorter performance.

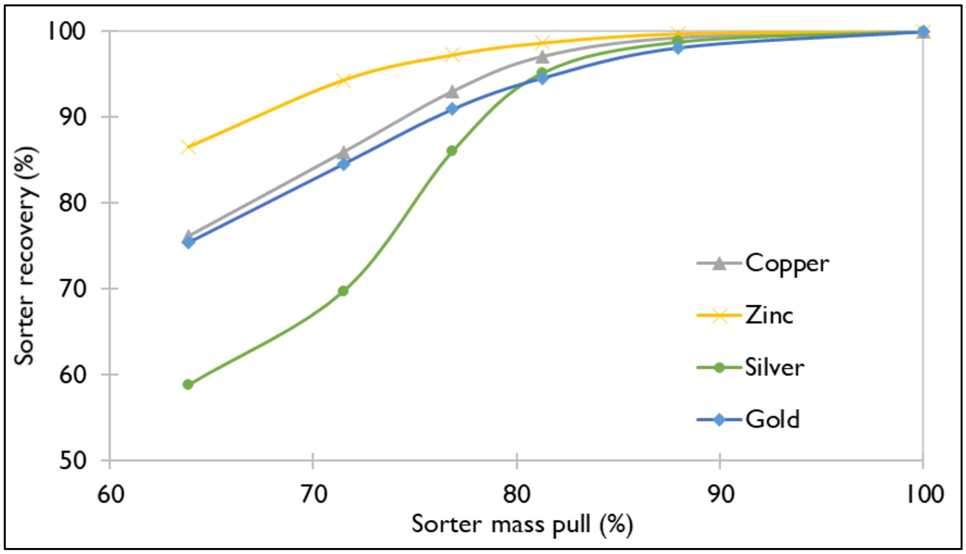


Figure 13-3: Sorter Mass Pull-Recovery for Different Metals

13.5 Comminution Test Work Summary

Comminution work complete prior to 2018 was reported in the JDS 2010 NI 43-101 PEA and JDS 2017 NI 43-101 PFS. The most recent grindability work summarized for this NI 43-101 FS includes the 2020 and 2021 metallurgical reports and the Bond Abrasion Index (Ai) and SMC data from 2018.

From 2018 to 2021 test work was carried out by Base Metallurgical Laboratories (BaseMet), Kamloops, BC, Bureau Veritas Laboratory (BV), Richmond, BC, and SGS Lakefield Research Limited (SGS), ON. Bond Abrasion (Ai) test results under BV Project 1801105 and SMC results completed under SGS Project 16768-01 provided by CSA Global were reviewed. The projects are listed below.

- BaseMet: Metallurgical Testing of Kutcho Copper, BL714, Report 1, 30 July 2021
- BaseMet: Metallurgical Testing of Kutcho Copper, BL483, Report 1, 10 November 2020 (Starkey SAGDesign S265 on the Main samples was conducted by Bureau Veritas Laboratory (BV) in Richmond, BC and reported in BL483)
- SGS: SMC Kutcho 16768-01, November 2018
- BV: Bond Abrasion Index 1801105, November 2018
- Cozamin: Metallurgical Investigation for Kutcho, December 2010 (JDS 2017 43-101 PFS)
- SGS: The Grindability Characteristics of Samples from the Kutcho Creek Deposit, Project 10933-001/002/004-Final Report, 18 August 2008 (Originally reported in the JDS 2010 43-101 PEA).

Table 13-21 summarizes the comminution test work from 2018 to 2021.

Table 13-21: Comminution test work completed

Year	Laboratory	Report no./data	Ore zone	Axb	SCSE	W_{SDT}	Dwi	Rwi	Bwi	Sd-Bwi*	Ai
2021	BaseMet	BL714	Esso	-	-	-	-	X	X	-	
			Main	-	-	-	-	X	X	-	
2020	BaseMet	BL483	Esso	-	-	-	-	-	X	-	
			Main	-	-	X	-	-	-	X	
2018	SGS	16768-01	Main	X	X	-	X	-	-	-	
	BV	1801105	Main	-	-	-	-	-	-	-	X

**Sd – Bwi Starkey SAG Design Ball Mill Work Index*
Source: BaseMet (2018/2021), CSA Global

The composites created for the latest test work were point samples and domain composites taken from individual drillholes using core from 2008–2010. Ore sorting was added to the flowsheet and is estimated to remove 12% of the crusher feed. No comminution test work was completed on the effect this will have on grindability. No test work was completed on samples that could be used for spatial geometallurgy. Sample origin and composite details as well as the full results can be found in the BaseMet reports listed above.

The Bond Low-energy Impact test to determine Bond Crusher Work Index (Cwi) was completed on 20 rock samples ranging in size between 50 mm and 75 mm in 2008. The Cwi was in the 83rd percentile for hardness with a Cwi of 15.5 kWh/t. In the most recent test work the average JK Drop Weight Axb for the Global Main sample was 78.1 and the median Bond Rod Mill Work Index (Rwi) of samples tested was 9.1 kWh/t. The Axb and Rwi results indicate the samples are in the moderately soft range of hardness. The material would be considered moderately abrasive based on previously reported test work with an average abrasion index (Ai) of 0.2 g.

The figure below summarizes the Bond Ball Mill (Bwi) Work Index (kWh/t) results for both Main and Esso deposits over a range of grind sizes (P_{80}) in microns. The results of the samples tested indicate the material is medium hardness at the final grind size P_{80} of 55 µm with a Bwi of approximately 14.1 kWh/t. The Bwi vs Grind Size results are shown in Figure 13-4.

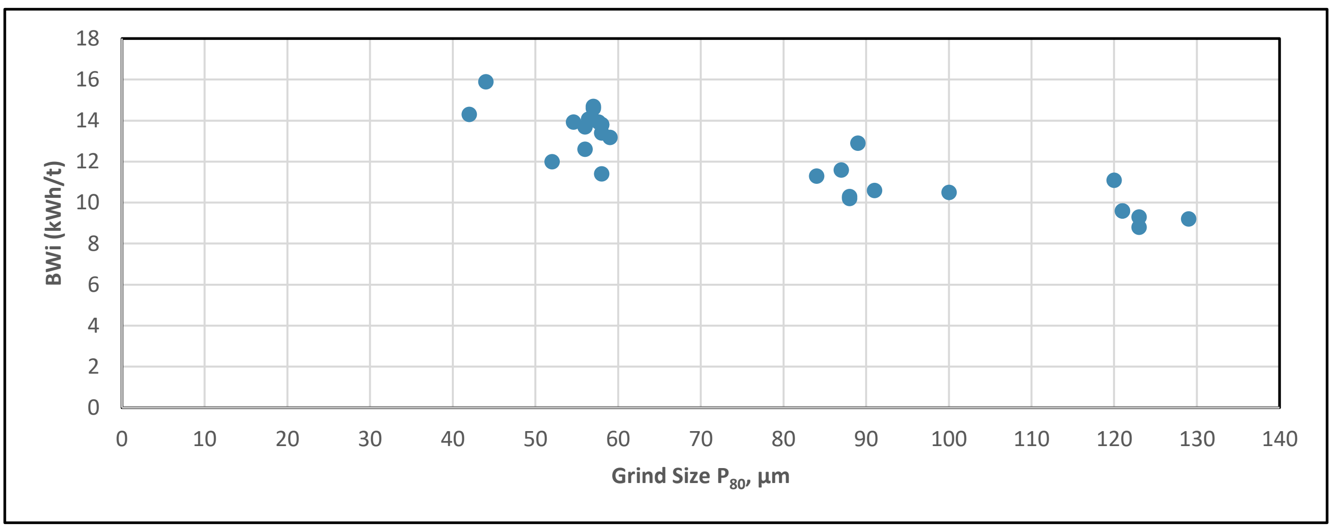


Figure 13-4: Bond Ball Mill Work Index vs Grind Size

13.5.1 BaseMet Test Program BL714 (2021)

Bond Rod Mill (Rwi) and Bond Ball Mill (Bwi) tests were complete on four composites from Esso Zone and four from Main Zone. The samples were prepared from intervals collected from eight drillholes. Each drillhole represents one composite. The results for the composites tested indicate the ore is relatively soft with Main samples being harder than Esso samples. The results are listed below in Table 13-22.

Table 13-22: Bond Rod Mill and Bond Ball Mill test results

Sample ID	Ore zone	Product size P_{80} (µm)	Rwi (kWh/t)	Product size P_{80} (µm)	Bwi (kWh/t)
KC10197	Esso	926	12.7	52	12.0
KC10175	Esso	835	10.0	58	11.4
KC10169	Esso	870	11.4	56	13.7
KC18-38W2	Esso	879	9.78	56	12.6
KC10197	Main	776	9.12	58	13.8
KC10197	Main	791	7.98	57	14.7
KC10197	Main	833	8.74	57	14.6
KC10197	Main	815	9.12	58	13.8

Source: BaseMet BL714 2021

13.5.2 BaseMet Test Program BL483(2020)

As part of BL483 three samples were sent out for SAG Design test work. Bond Ball Mill Work Index test work was completed on an Esso composite by BaseMet (2020).

13.5.2.1 Esso Comp #2

One Bwi was completed on the Esso composite indicating the sample was of medium hardness. The results are shown in Table 13-23 below.

Table 13-23: Esso Bond Ball Mill test results

Sample	Closing screen (µm)	Feed size F_{80} (µm)	Product size P_{80} (µm)	g/rev	Bwi (kWh/t)
Esso Comp. 2	75	1,757	58	1.32	13.4

Source: BaseMet BL483, 2020

13.5.2.2 SAG Design Tests

The SAG Design tests (SDT) were used to determine SAG mill specific pinion energy needed to grind ore from P_{80} 152 mm to P_{80} 1.7 mm denoted as WSDT in kWh/t and the ball mill work index Sd-Bwi in kWh/t. The overall SAG Design database indicates the WSTD is in the 13.6 percentile and the Sd-Bwi is in the 23rd percentile. Table 13-24 summarizes the results.

Table 13-24: SAG Design test results

Sample	Charge mass (g)	No. revs.	Solids SG	W_{SDT} (kWh/t)	Charge mass (g)	Feed size F_{80} (µm)	Product size P_{80} (µm)	Grams/ rev.	Bwi (kWh/t)
High Sulphur	11,979	737	4.26	3.85	2,251	1,132.5	59.3	1.476	13.19
KM Global A	9,557	702	3.73	4.20	1,795	1,107.1	54.6	1.539	12.12
KM Global B	9,604	774	3.41	4.61	1,828	1,165.4	56.4	1.549	12.23

Source: BaseMet BL483 (2020), Starkey SAG Design S265

13.5.3 SMC Kutcho 16768-01(2018)

The test work was completed by SGS on a high sulphur composite, and two global composites. The results put the samples in the soft range of hardness as shown in Table 13-25.

Table 13-25: SMC test results

Sample ID	A	b	A x b	SCSE (kWh/t)	DWI (kWh/m³)	Mia (kWh/t)	Mih (kWh/t)	Mic (kWh/t)	Relative density
High S Comp	74.2	1.25	92.8	NA	4.97	9.0	6.3	3.2	4.61
KM Global Comp – Dup	76.3	1.02	77.8	7.30	5.39	10.6	7.5	3.9	4.18
KM Global Comp	82.5	0.95	78.4	7.28	5.34	10.5	7.4	3.8	4.18

Source: SMC Kutcho 16768-01, November 2018 (spreadsheet only)

13.5.4 Bond Abrasion Index 1801105 (2018)

Bond Abrasion Index (Ai) was completed on six samples by Bureau Veritas Laboratory (BV). The results are shown in Table 13-26.

Table 13-26: Bond Abrasion Index test summary

Test ID	Composite	Ai (g)
AI-1	High S Composite	0.1394
AI-2	KM Global Composite	0.1203
AI-3	Average S Composite	0.0954
AI-4	Breccia Composite	0.1608
AI-5	Exhalite Composite	0.1109
AI-6	Low S Composite	0.0942

Source: Kutcho Copper Project 1801105 2018

14 Mineral Resource Estimate

14.1 Introduction

The Mineral Resource Estimate reported herein was prepared under the supervision of QP author, Mr Robert Sim, P.Geo., of SIM Geological Inc. (SGI) with the assistance of Mr Bruce Davis, PhD, FAusIMM of BD Resource Consulting Inc. Mr Robert Sim takes full responsibility for the Mineral Resource Estimate disclosed herein including all of Section 14. This section of the Technical Report describes the Mineral Resource estimation methodology and summarizes the key assumptions considered by the QP to prepare the Mineral Resource model for the copper, zinc, gold, and silver mineralization for the Main, Esso and Sumac deposits at the Kutcho Project. The effective date of Mineral Resource Estimate disclosed in this technical report is 30 July 2021.

The previous Technical Report for the Kutcho Project was dated 31 July 2017, with an effective date of 15 July 2017, and contained an estimate of Mineral Resources prepared by QP Garth Kirkham. During the 2018 summer field season, Kutcho Copper conducted additional delineation drilling on the Main and Esso deposits. SGI used the new drilling information to produce a new estimate of Mineral Resources with an effective date of 22 February 2019. The 2019 resource utilized an updated geological interpretation and a smaller block size that reflected the underground mining methods assumed at that time. The results, presented in a Kutcho Copper press release dated 4 March 2019, were similar to the previous estimate of Mineral Resources from July 2017 and are documented in an internal company report dated 22 February 2019 (there was no Technical Report produced at that time).

In the late spring of 2021, the Project Mineral Resources were updated to reflect new metal prices, operating costs and process recoveries. The assumed approach to mining of the Main deposit was also changed from (previously) underground to primarily open pit extraction methods and the block size in the Main resource model was increased to reflect this change in mining methods.

In the opinion of the QP author, the Mineral Resource estimate reported herein is a reasonable representation of the mineralization found at the Project at the current level of sampling. The Mineral Resource Estimate was disclosed in accordance with NI 43-101 and Canadian Institute of Mining, Metallurgy and Petroleum ("CIM") standards and guidelines. All technical works have been undertaken using the May 2014 CIM Definition Standards for Mineral Resources and Mineral Reserves and recent CIM Best Practice guidelines.

Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. There is no certainty that all or any part of the Mineral Resources will be converted into a Mineral Reserves upon the application of modifying factors.

Estimations were made from 3D block models based on geostatistical applications using commercial mine planning software (MinePlan® v15.80-2). The Project limits are based in the UTM coordinate system (UTM NAD83 Zone 9) using a nominal block size measuring 5 m x 5 m x 5 m for the Main deposit and 5 m x 2.5 m x 5 m (L x W x H) for the Sumac and Esso deposits. Drillholes, collared from surface, penetrate the sub-planar, north-dipping mineralized horizons from the hanging wall side.

The Main deposit strikes west-east for more than 1,600 mm and dips at -45° to -50° to the north. Drilling has intersected the mineralized Main zone horizon to depths approaching 400 m below surface. The spacing of drillhole pierce points range from 25 m to 50 m throughout most of the deposit, expanding to 100 m at the peripheries.

The Esso deposit has a west-east strike length of about 900 m and was drill tested from 400 m to 650 m below surface. Drillhole pierce points that intersect the mineralized horizon tend to be spaced between 25 m and 50 m intervals.

The Sumac deposit has a west-east strike length of 800 m and extends from surface to about 500 m below surface. Drillholes are variably spaced at 50–100 m or more.

Mineral Resource estimates were generated using drillhole sample assay results and the interpretation of geological models which relates to the spatial distribution of copper, zinc, gold, and silver. Lead was also evaluated, but the grades were too low to be considered in the estimate of Mineral Resources. Interpolation characteristics were defined based on the geology, drillhole spacing, and geostatistical analysis of the data. The Mineral Resources were classified according to their proximity to the sample data locations and are reported, as required by NI 43-101, according to the CIM Definition Standards for Mineral Resources and Mineral Reserves (May 2014).

14.2 Available Data

Kutcho Copper provided the drillhole sample data for the Kutcho Project on 9 November 2018. This comprised a series of Microsoft Excel® (spreadsheet) files containing collar locations, downhole survey results, geological information, and assay results for a total of 631 drillholes representing 146,425 m of drilling. Thirty-five of the holes were drilled for geotechnical or metallurgical purposes and, as a result, have no associated samples or assay results. These provide geological information but have not been sampled or analyzed for metal content. Assay results in the remaining 596 holes, totalling 140,389 m of drilling, were used to generate the estimates of Mineral Resources for the Kutcho Project.

There is a total of 360 drillholes at the Main deposit, 119 drillholes at the Esso deposit, and 24 drillholes at the Sumac deposit. The remaining drillholes are either exploratory in nature and test for satellite deposits and/or were drilled for geotechnical information. These provide geological information but have not been sampled or analyzed for metal content.

Most of the older drilling was conducted using BQ-size core holes. The more recent drilling is a combination of HQ and NQ (and rarely BTW) as holes are typically “downsized” when drilling problems are encountered at depth. Most of the mineralized intersections in the more recent drilling were sampled using NQ-size drill core.

Drilling on the Kutcho deposits began in 1974 and continued through to the most recent delineation drilling program conducted in 2018. Note that some additional drilling conducted for geotechnical and hydrogeological purposes in 2021 has no impact on the estimate of Mineral Resources contained in this report.

The sample results collected during drilling conducted since the year 2000 were monitored using a rigid QA/QC program that meets accepted industry standards. Sampling conducted prior to 2000 was likely monitored using certified standards and duplicates, but not a lot of supporting documentation was retained from that time. About one half of the holes intersecting both the Main and Esso deposits were drilled prior to 2000 (referred to as “old” holes) and the other half of the holes (“new” holes) were drilled after that date. Comparing the results from both vintages of drilling show that, although local grade differences exist, both new and old drillholes return similar grades and thicknesses of mineralization over larger volumes. These comparisons show that both new and old drilling provide similar results from a Mineral Resource perspective and, therefore, the assay data derived from both new and old drilling can be used to estimate the Mineral Resources, and there are no modifications required to any of the older data.

There are about 9,500 individual samples taken from drillholes that test the Main deposit, another 2,100 in the Esso deposit, and about 600 in the Sumac deposit; the majority were analyzed for a variety of elements (as part of a multi-element package). Individual sample intervals range from a minimum of 6 cm to a maximum of 5 m and average 1.05 m long at Main, 1.24 m at Esso and 1.38 m at Sumac. Sample data for copper, zinc, gold, and silver were extracted from the main spreadsheet database files and imported into MinePlan® to develop the Mineral Resource models. In addition to these, sample data for arsenic, mercury, antimony, and cadmium were also imported into MinePlan®; the available data for these potentially deleterious elements is limited because samples collected prior to 2000 were not analyzed for these elements.

The distribution of copper grades in drillholes in the Main deposit area is shown in Figure 14-1 and Figure 14-2. The distribution of copper grades in drillholes in the Esso deposit area is shown in Figure 14-3 and at the Sumac deposit in Figure 14-4.

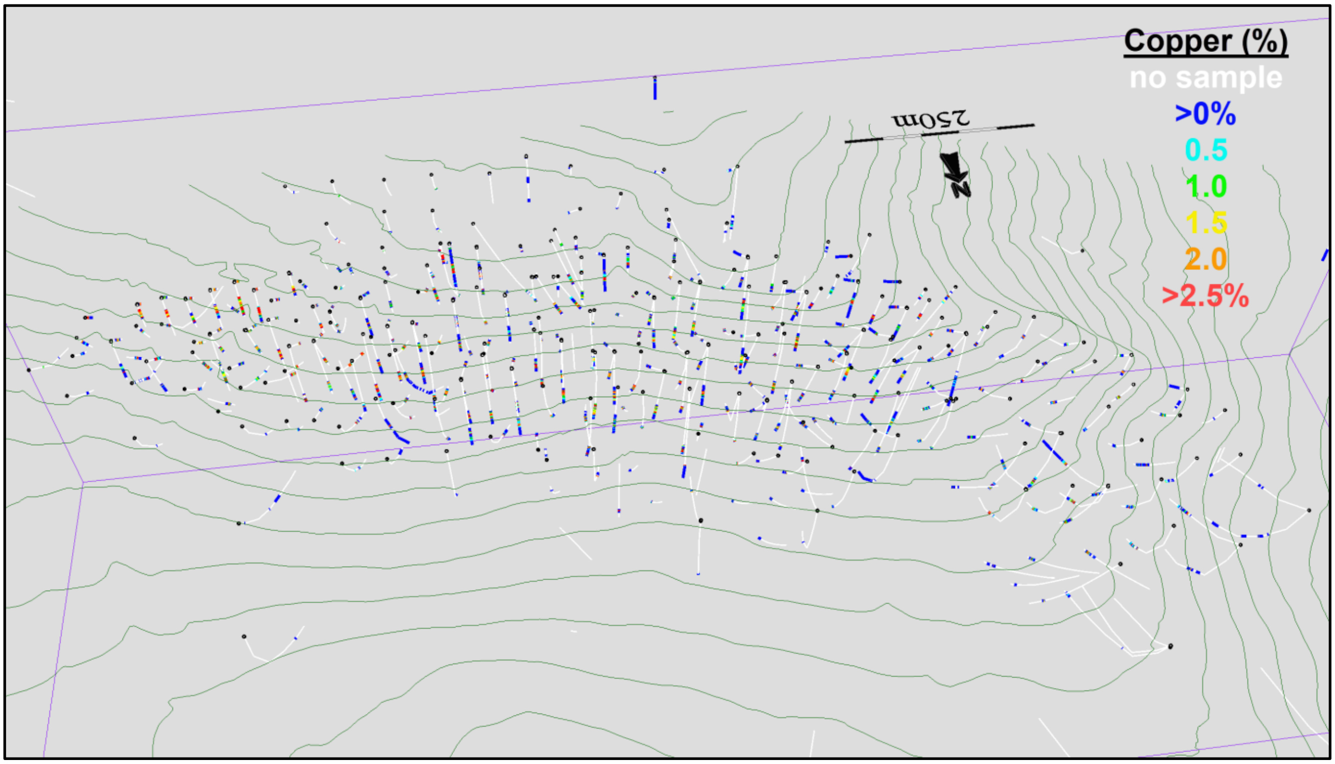


Figure 14-1: Isometric Longitudinal (approx.) Viewpoint of Copper Grades in Drilling on the Main Deposit

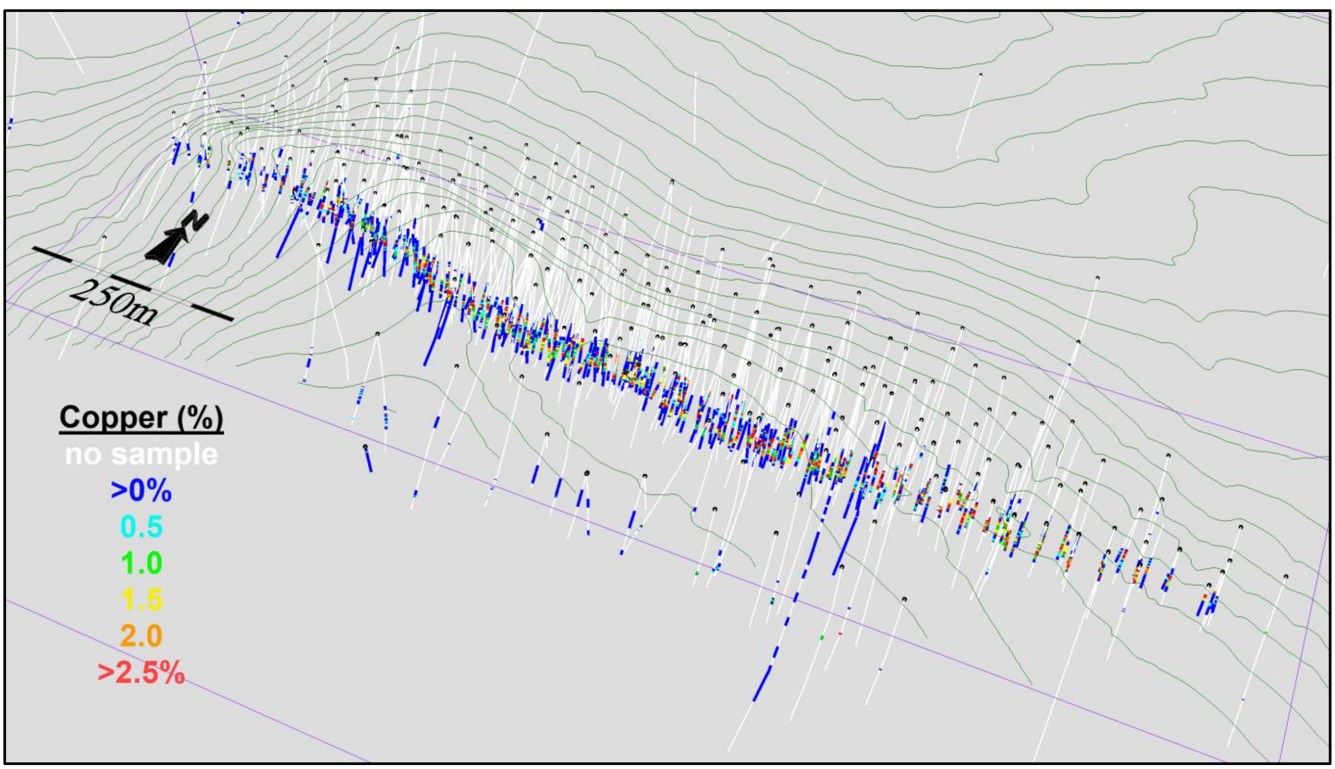


Figure 14-2: Isometric View Looking Down Dip of Copper Grades in Drilling on the Main Deposit

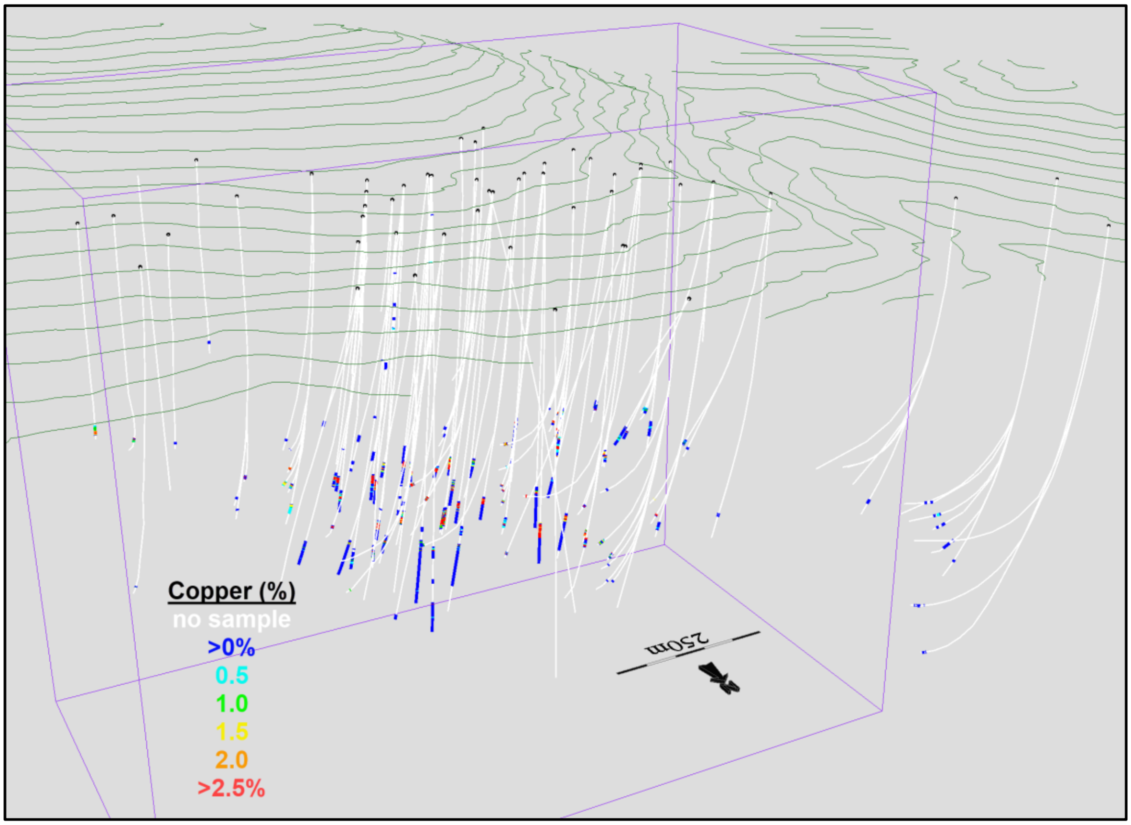


Figure 14-3: Isometric View of Copper Grades in Drilling on the Esso Deposit

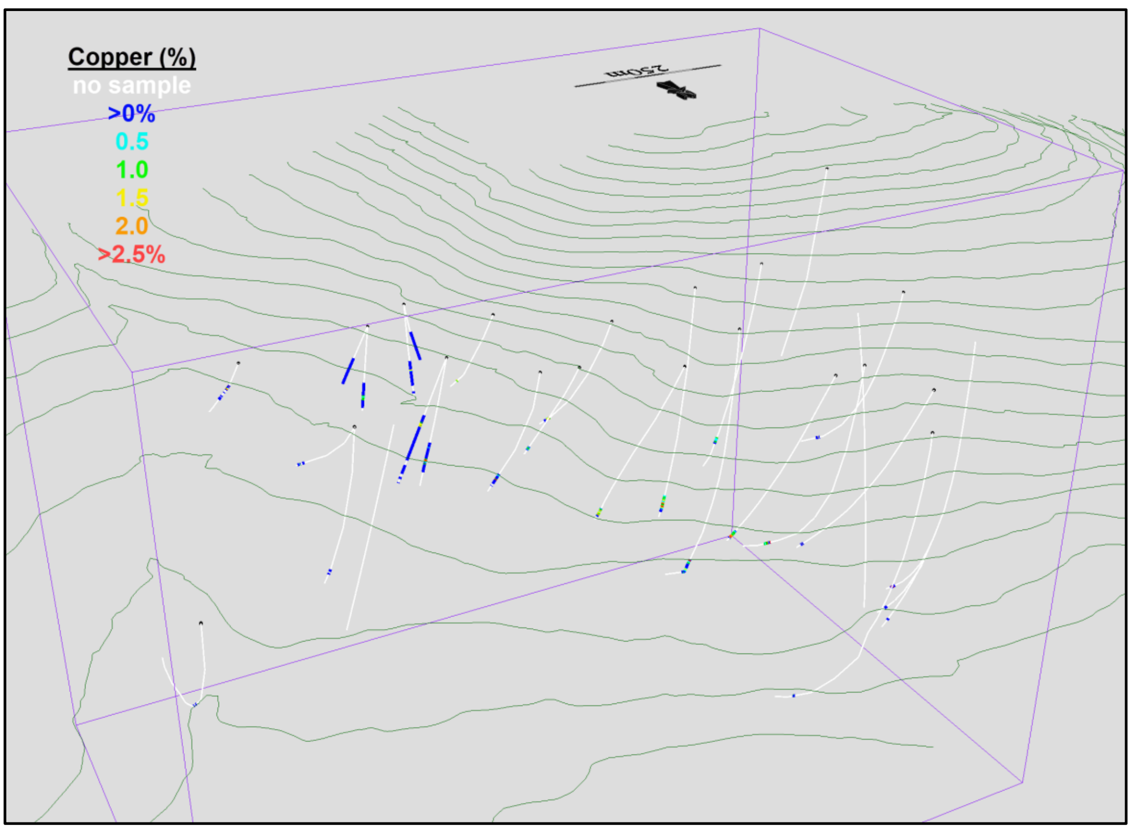


Figure 14-4: Isometric View of Copper Grades in Drilling on the Sumac Deposit

Specific gravity (SG) data are available in about two thirds of the drillholes that intersect the Main deposit and about one third of the holes at Esso, but only in six drillholes at Sumac. The distribution of SG data is considered sufficient to support the interpolation of SG values in the block models at Main and Esso, but not at Sumac.

Topographic data were provided in the form of a 3D triangulated digital terrain surface. The drillhole collars correlate very well with this topographic surface.

Geological information, derived from observations during core logging, provides lithology and alteration code designations. There are more than 85 different lithology types in the Kutcho database. The large number of lithology-type codes is primarily the result of multiple operators over many years using an inconsistent designation of the various rock types. There are eight different alteration types defined during core logging and these are available only in the newer drillholes completed since 2004. Attempts have been made by Kutcho Copper geologists to interpret lithological and alteration models using the available data, but these tend to be of poor quality due to the variability of the underlying geological information. Attempts to simplify or standardize the lithology and alteration information using the logging descriptions have not been very successful at the Project. Relogging of all available drill core is recommended to provide consistency of the geological information throughout the various vintages of drilling.

The degree of oxidation, based on visual observations, is recoded in about one half of the drillholes. This information suggests that (other than rare localized occurrences) the rocks at Kutcho do not show any signs of significant oxidation.

The statistical properties of the data in the vicinity of the Main deposit are shown in Table 14-1. The assay statistics for samples in the Esso and Sumac deposit areas are shown in Table 14-2 and Table 14-3, respectively.

Table 14-1: Summary of basic statistics of sample data for the Main deposit

Element	No. of samples	Total sample length (m)	Minimum	Maximum	Mean	Standard deviation
Copper (%)	9,546	10,055	0	35.5	0.796	1.453
Zinc (%)	9,590	10,141	0	41.9	1.063	2.469
Gold (g/t)	9,014	9,230	0.003	375	0.2193	3.4885
Silver (g/t)	9,495	9,992	0.1	859.4	13.64	28.67
Lead (%)	8,758	8,874	0	5.4	0.027	0.111
Arsenic (ppm)	6,230	6,364	1	2,640.00	97	180.5
Antimony (ppm)	6,174	6,091	1	264	5.5	10.2
Mercury (ppm)	6,174	6,091	1	101	2.9	4.3
Cadmium (ppm)	6,174	6,091	0.3	2,000.00	49.83	113.5
SG	7,732	6,660	1.07	7.69	3.314	0.685

Note: Original sample data are weighted by sample length. The data used in the corresponding table is restricted to 135 drillholes in the vicinity of the Main deposit.

Table 14-2: Summary of basic statistics of sample data for the Esso deposit

Element	No. of samples	Total sample length (m)	Minimum	Maximum	Mean	Standard deviation
Copper (%)	2,072	2,573	0	21.80	0.80	1.85
Zinc (%)	2,068	2,571	0	30.10	1.54	4.63
Gold (g/t)	2,065	2,556	0.002	160.000	0.332	3.959
Silver (g/t)	2,072	2,573	0.1	1500.1	21.0	76.08
Lead (%)	2,068	2,566	0	3.20	0.03	0.156
Arsenic (ppm)	543	578	1	2390	92	254.3
Antimony (ppm)	543	578	1	1360	17	87.2

Element	No. of samples	Total sample length (m)	Minimum	Maximum	Mean	Standard deviation
Mercury (ppm)	543	578	1	43	2.9	4.3
Cadmium (ppm)	543	578	0	2500	97.2	304.3
SG	1,783	1,887	1.00	7.00	2.98	0.347

Note: Original sample data are weighted by sample length. The data used in Table 14-2 is restricted to 135 drillholes in the vicinity of the Esso deposit.

Table 14-3: Summary of basic statistics of sample data for the Sumac deposit

Element	No. of samples	Total sample length (m)	Minimum	Maximum	Mean	Standard deviation
Copper (%)	634	873	0	5.18	0.311	0.635
Zinc (%)	664	942	0	11.55	0.45	1.005
Gold (g/t)	597	811	0.003	1.42	0.057	0.1109
Silver (g/t)	629	874	0.1	122	4.97	11.48
Lead (%)	593	797	0	0.33	0.012	0.028
Arsenic (ppm)	423	618	1	1,000	70.5	168
Antimony (ppm)	423	618	1	50	3.2	6.1
Mercury (ppm)	423	618	1	30	1.9	2.5
Cadmium (ppm)	423	618	0.3	502	18.4	49.28
SG	130	77	2.57	4.75	3.498	0.722

Note: Original sample data are weighted by sample length. The data used in Table 1.3 is restricted to 29 drillholes in the vicinity of the Sumac deposit.

As shown in Figure 14-1 through Figure 14-4, sampling of drill core at the Property is concentrated in the area of the north-dipping mineralized horizons and is primarily controlled by visual observations of sulphide mineralization. Therefore, all unsampled core intervals are assumed to be unmineralized, and, as a result, these were assigned default zero grades for Mineral Resource estimation purposes. Note: grades for copper, zinc, gold, and silver were assigned zero grade values where no sample data were available. This same approach was not used for missing data intervals for arsenic (As), mercury (Hg), antimony (Sb), and cadmium (Cd) because of the limited number of mineralized samples analyzed for these elements.

Several holes that were drilled for geotechnical or metallurgical purposes were never sampled or analyzed. Although these provide some information for geological interpretation, these holes were ignored for grade estimation purposes.

14.3 Geological Model, Domains and Coding

The Kutcho deposits are interpreted as volcanogenic massive sulphide (VMS) deposits, as described in Sections 7 and 8. Variable accumulations of massive to semi-massive sulphides occur as sheet-like zones striking east-west and dipping -50° to the north.

The Main deposit measures about 1,600 m along strike and 500 m along the dip plane and extends from surface to about 400 m below surface.

The Esso deposit is about 900 m along strike and 400 m along the dip plane and occurs from about 350 m to 750 m below surface.

The Sumac deposit is about 800 m along strike and extends to about 500 m below surface. The true thickness of zones with elevated copper and zinc is variable from 1 m to more than 40 m in some areas of the Main deposit and to about 25 m in the Esso and Sumac deposits.

Grade shell domains were interpreted based on a copper-equivalent cut-off threshold of 0.3% CuEq for the Main deposit and 0.7% CuEq for the Esso and Sumac deposits.

Metal prices were assumed to be US$3.00/lb copper, US$1.25/lb zinc, US$1,350/oz gold and US$17/lb silver. This resulted in the following formula:

$$CuEq\% = Cu\% + (Zn\%\ x\ 0.417) + (Au\ g/t\ x\ 0.656) + (Ag\ g/t\ x\ 0.008)$$

It should be noted that these metal prices differ from those used to define the Mineral Resources in Section 14.13 of this report because this work was initially conducted in 2019 and the values reflect the prices at that time. The mineralization tends to occur in well-defined zones which are surrounded by rocks that typically are completely unmineralized (these trends are shown in contact profiles presented in Section 14.5.2). The interpreted grade shell domains, using the metal prices listed above, essentially capture all the significant mineralization that is present at the Kutcho deposits. It should also be noted that the interpretation of the grade shell domains in all deposits does include some lower grade internal mineralization, below the defined threshold grades, to retain the shape and continuity of the interpreted domains.

Three shells were interpreted in the Main deposit area: a larger and more extensive Main zone that is straddled by a typically narrower and lower grade footwall zone and hanging wall zone. There is only one grade shell domain interpreted for each of the Esso and Sumac deposits. The shape and extent of the grade shell domains are shown in Figure 14-5 and Figure 14-6 for the Main deposit, Figure 14-7 and Figure 14-8 for the Esso deposit, and Figure 14-9 for the Sumac deposit.

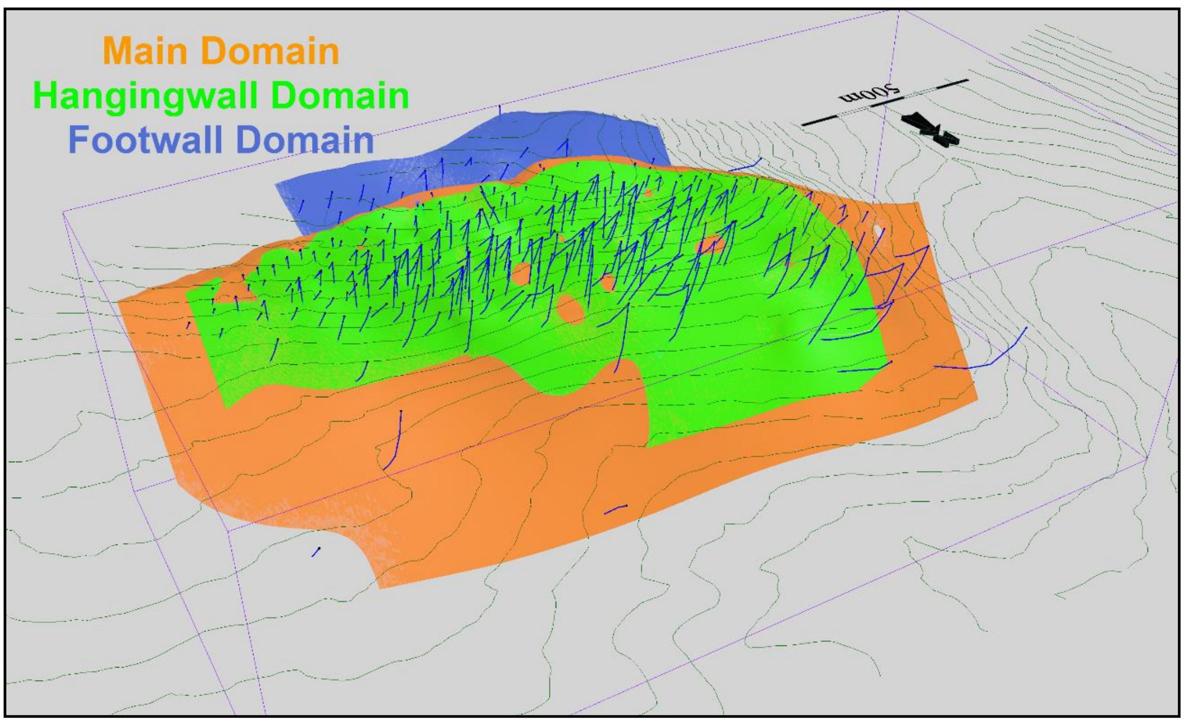


Figure 14-5: Isometric View of 0.3% CuEq Grade Shell Domains for the Main Deposit

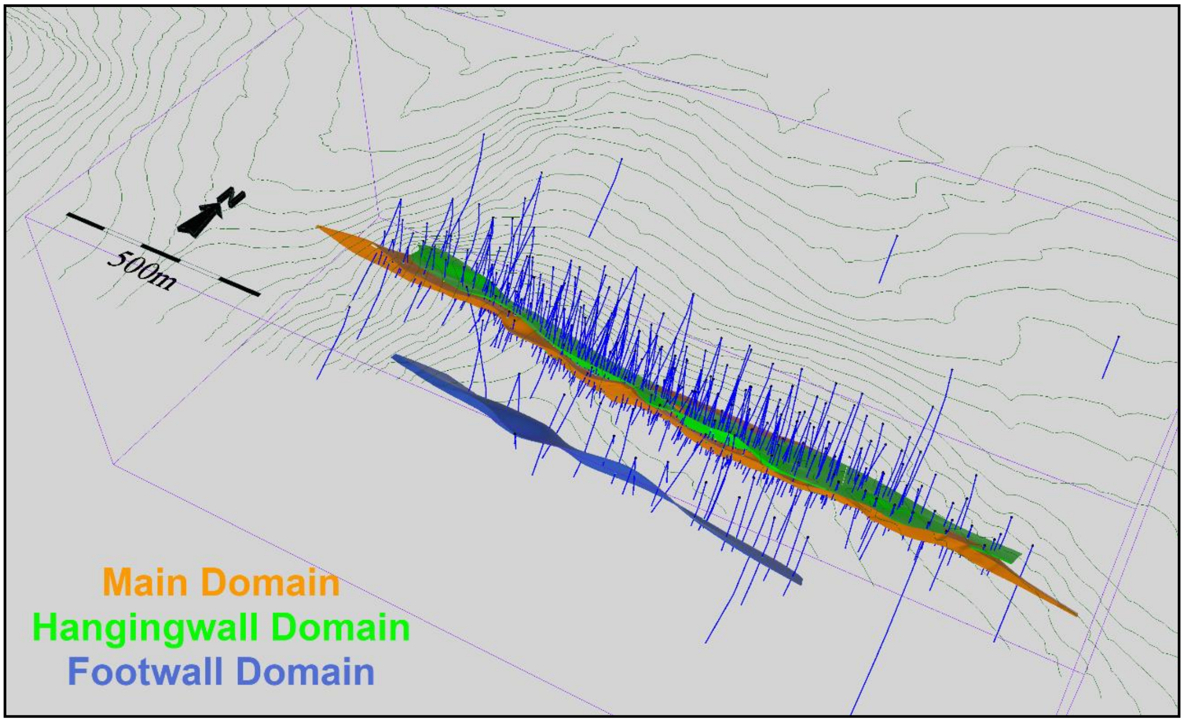


Figure 14-6: Isometric View Of 0.3% CuEq Grade Shell Domains for the Main Deposit

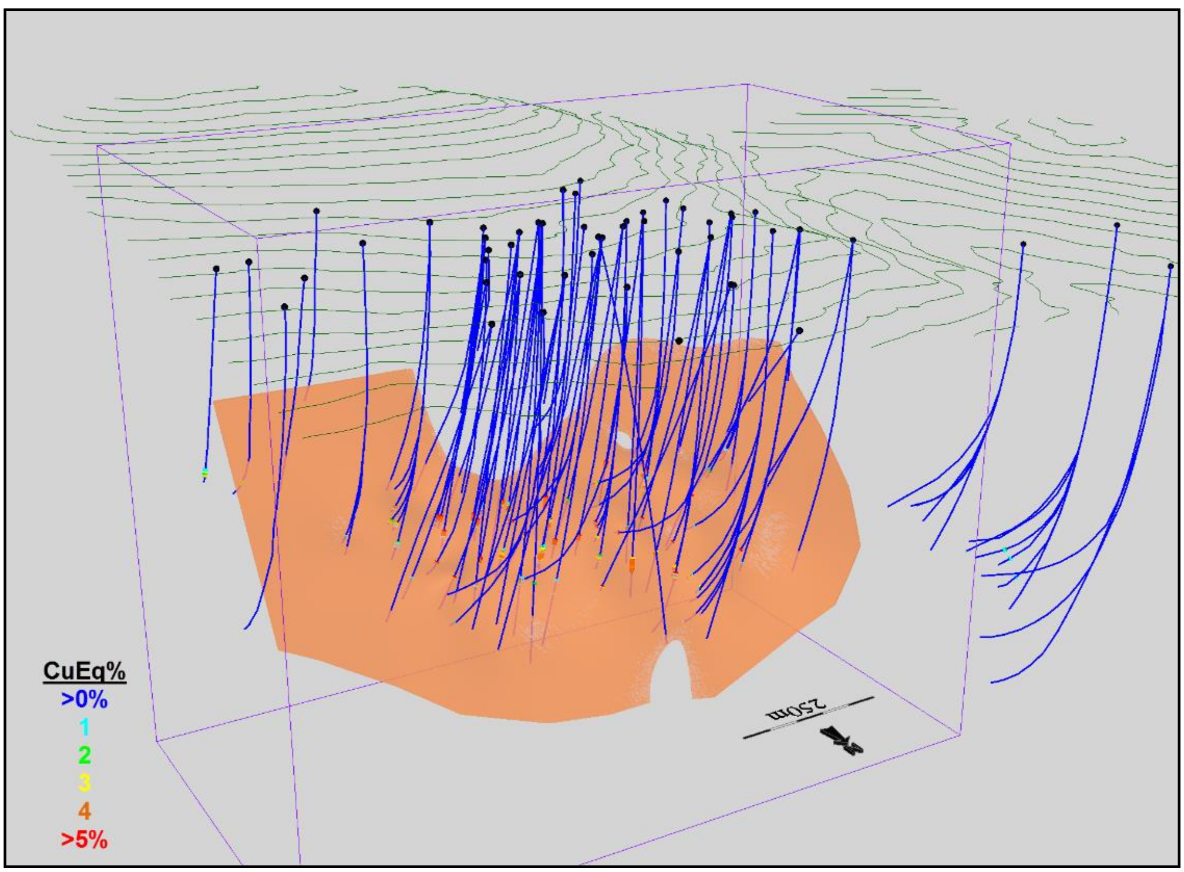


Figure 14-7: Isometric View of 0.7% CuEq Grade Shell Domain for the Esso Deposit

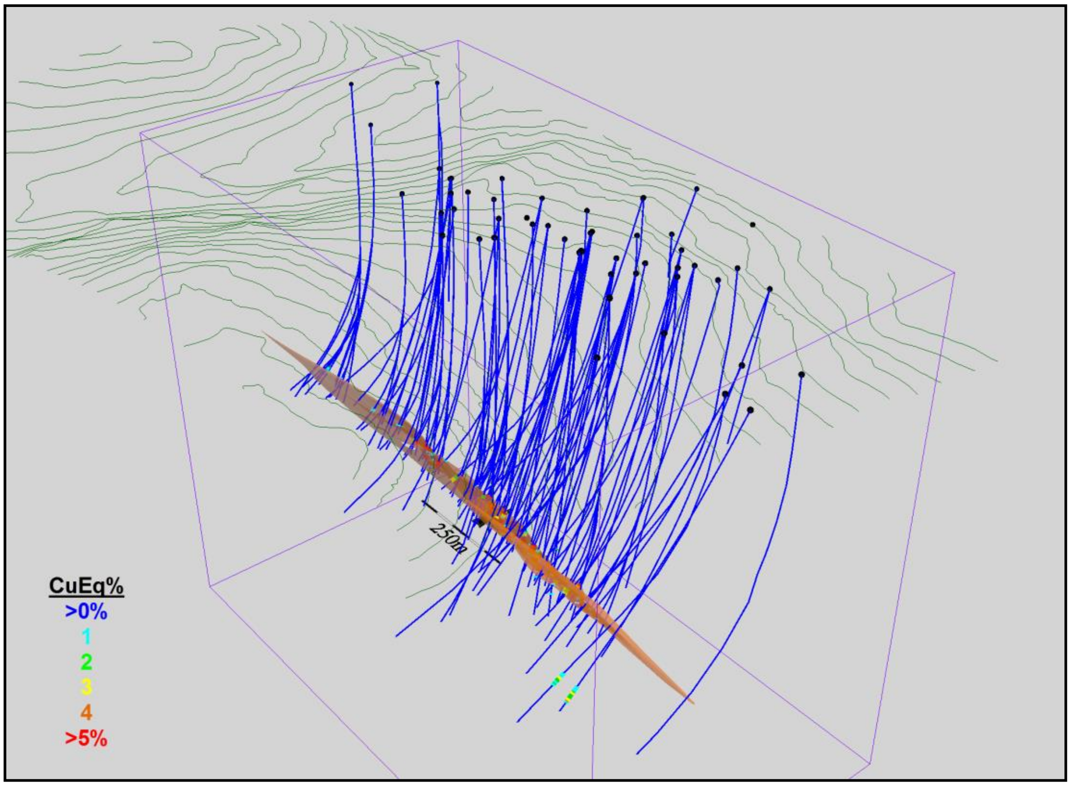


Figure 14-8: Isometric View of 0.7% CuEq Grade Shell Domain for the Esso Deposit

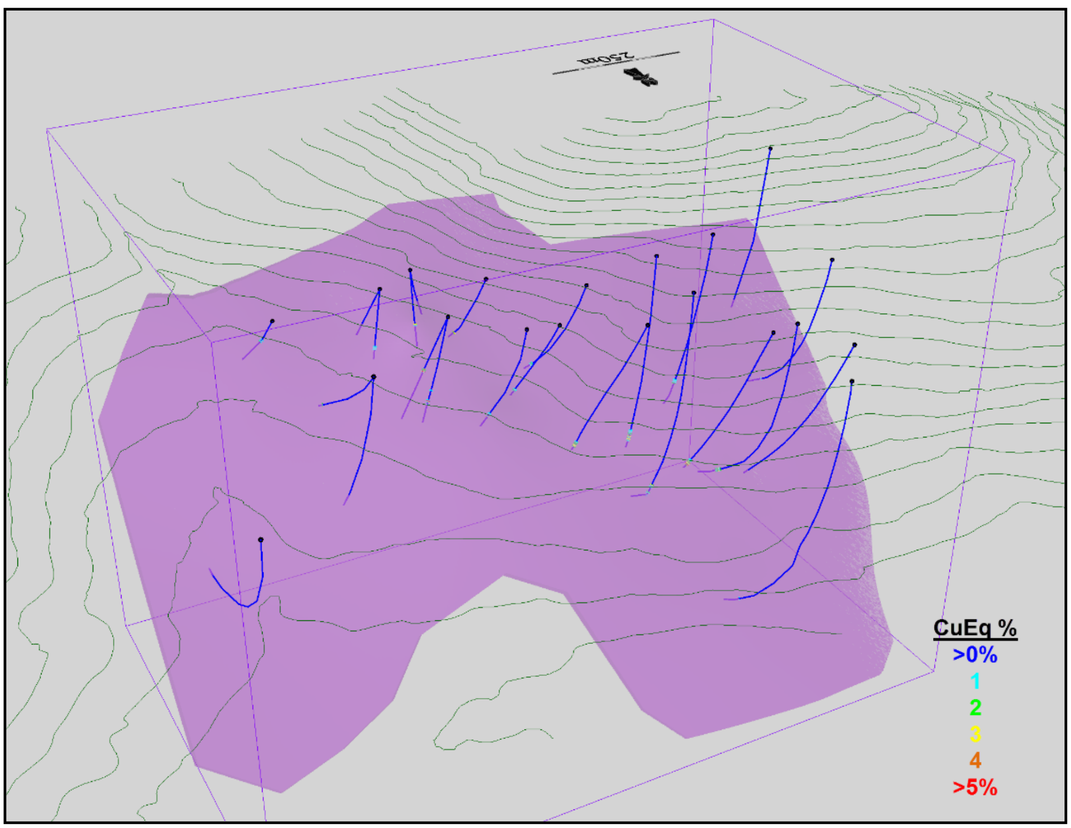


Figure 14-9: Isometric View of 0.7% CuEq Grade Shell Domain for the Sumac Deposit

As is typical of VMS deposits, the mineralized zones at Kutcho exhibit a somewhat banded nature in the metal distribution and, to retain this banding in the Mineral Resource block models, a dynamic search approach was used. During grade interpolations, search orientations are controlled using a "trend plane" which represents the centre of the grade shell domains. This approach retains the subtle banding shown in drilling in the block model.

14.4 Compositing

Compositing the drillhole samples helps standardize the database for further statistical evaluation. This step eliminates any effect that inconsistent sample lengths might have on the data. To retain the original characteristics of the underlying data, a composite length was selected that reflects the average original sample length. The generation of longer composites can result in some degree of smoothing which could mask certain features of the data. A composite length of 1 m was selected for the Kutcho deposits, reflecting the fact that the average length of samples located inside the Main zone grade shell domain is 0.97 m and 1.07 m at both the Esso and Sumac deposits.

Drillhole composites are length-weighted and were generated down-the-hole; this means that composites begin at the top of each hole and are generated at 1 m intervals down the length of the hole.

14.5 Exploratory Data Analysis

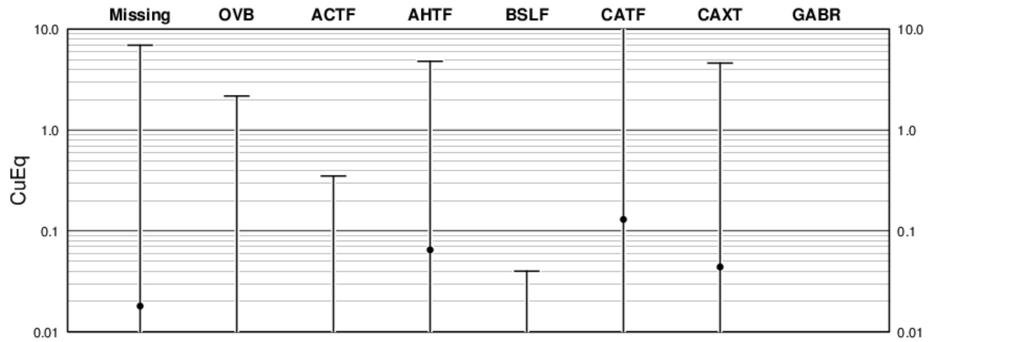
Exploratory data analysis (EDA) involves the statistical assessment of the database to better understand the characteristics of the data that may control grade. One of the main purposes of this exercise is to determine if there is evidence of spatial distinctions in grade which may require the separation and isolation of domains during interpolation. The application of separate domains prevents unwanted mixing of data during interpolation and, therefore, the resulting grade model will better reflect the unique properties of the deposit. However, applying domain boundaries in areas where the data are not statistically unique may impose a bias in the distribution of grades in the model.

A domain boundary, which segregates the data during interpolation, is typically applied if the average grade in one domain is significantly different from that of another domain. A boundary may also be applied if there is evidence that a significant change in the grade distribution has occurred across the contact.

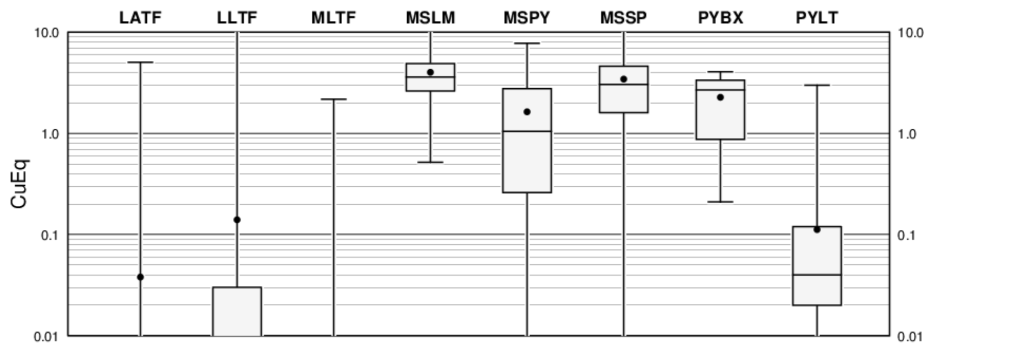
14.5.1 ***Basic Statistics of Sample Data***

The basic statistics of sample grades relative to some of the underlying lithology and alteration types recorded during core logging, as well as the distribution of sample data located inside versus outside of the grade shell domains, are evaluated using a series of boxplots.

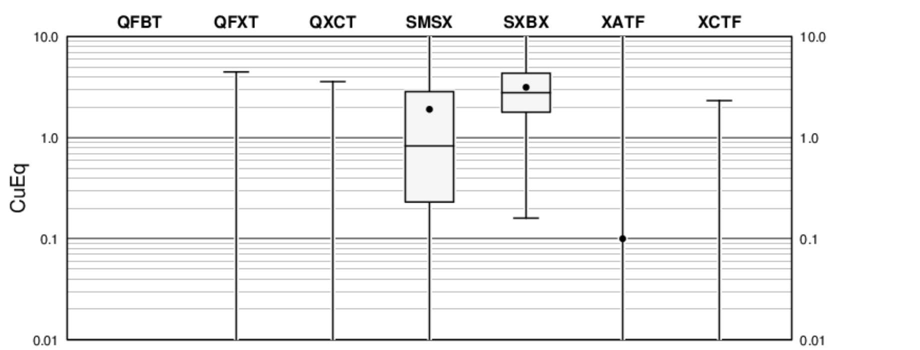
Figure 14-10 contains a series of boxplots showing the distribution of copper-equivalent grades by a series of logged lithology types. As stated previously, there are over 85 different lithology types in the geology database; these boxplots show some of the more common lithologies that are present. Only the massive sulphide (MS) and semi-massive sulphide (SMS) domains contain any appreciable mineralization, and the other rock types are essentially void of any significant grades. The magnitude of the grades in MS and SMS are similar, and there is quite a bit of overlap in the distribution between these rock types.



	Missing	OVB	ACTF	AHTF	BSLF	CATF	CAXT	GABR	
Number of data	2153	1580	500	1149	2335	773	800	4237	Number of data
Mean	0.018	0.001	0.008	0.065	0.0	0.13	0.044	0.0	Mean
Variance	0.051	0.002	0.001	0.156	0.0	0.59	0.052	0.0	Variance
Skewness	19.791	51.198	6.951	8.785	18.558	9.448	12.226	0.0	Skewness
Coef of Variation	12.199	48.016	3.138	6.078	17.619	5.888	5.212	0.0	Coef of Variation
Maximum	6.94	2.18	0.35	4.82	0.04	11.9	4.61	0.0	Maximum
Upper quartile	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	Upper quartile
Median	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	Median
Lower quartile	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	Lower quartile
Minimum	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	Minimum



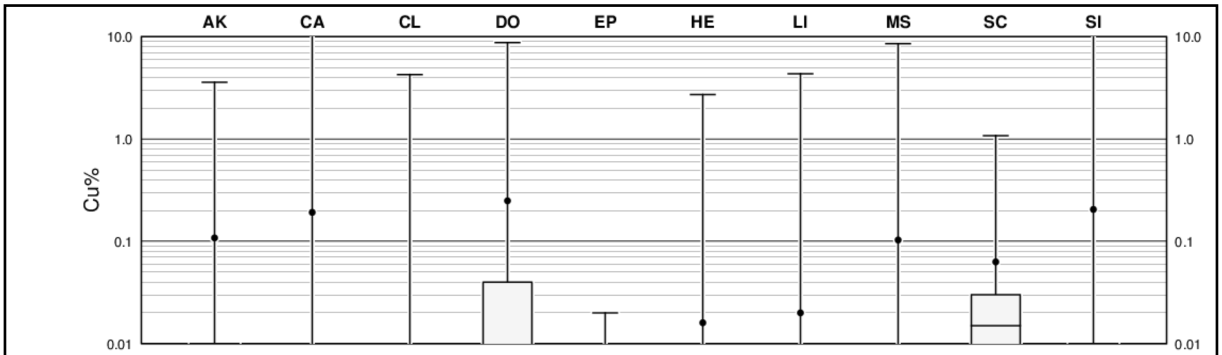
	LATF	LLTF	MLTF	MSLM	MSPY	MSSP	PYBX	PYLT	
Number of data	1760	7597	689	281	532	1607	24	599	Number of data
Mean	0.038	0.14	0.009	4.02	1.636	3.438	2.277	0.112	Mean
Variance	0.067	0.326	0.014	5.519	2.817	11.874	1.626	0.056	Variance
Skewness	13.384	8.494	17.533	2.621	1.122	13.759	-0.34	7.665	Skewness
Coef of Variation	6.722	4.065	12.84	0.584	1.026	1.002	0.56	2.113	Coef of Variation
Maximum	5.05	11.45	2.18	21.32	7.79	99.0	4.07	2.99	Maximum
Upper quartile	0.0	0.03	0.0	4.91	2.769	4.62	3.355	0.12	Upper quartile
Median	0.0	0.0	0.0	3.618	1.044	3.05	2.69	0.04	Median
Lower quartile	0.0	0.0	0.0	2.63	0.26	1.6	0.875	0.02	Lower quartile
Minimum	0.0	0.0	0.0	0.52	0.0	0.0	0.21	0.0	Minimum



	QFBT	QFXT	QXCT	SMSX	SXBX	XATF	XCTF	
Number of data	1697	11317	966	753	115	515	788	Number of data
Mean	0.0	0.001	0.008	1.91	3.153	0.1	0.006	Mean
Variance	0.0	0.002	0.019	7.204	3.415	0.7	0.008	Variance
Skewness	0.0	98.528	22.107	3.18	1.481	13.935	23.845	Skewness
Coef of Variation	0.0	53.768	17.195	1.405	0.586	8.38	15.258	Coef of Variation
Maximum	0.0	4.47	3.59	23.63	11.38	14.8	2.33	Maximum
Upper quartile	0.0	0.0	0.0	2.854	4.351	0.0	0.0	Upper quartile
Median	0.0	0.0	0.0	0.83	2.794	0.0	0.0	Median
Lower quartile	0.0	0.0	0.0	0.23	1.79	0.0	0.0	Lower quartile
Minimum	0.0	0.0	0.0	0.0	0.16	0.0	0.0	Minimum

Figure 14-10: Basic Statistics of Copper-Equivalent Grades in Samples Logged Lithology Type – Main Deposit

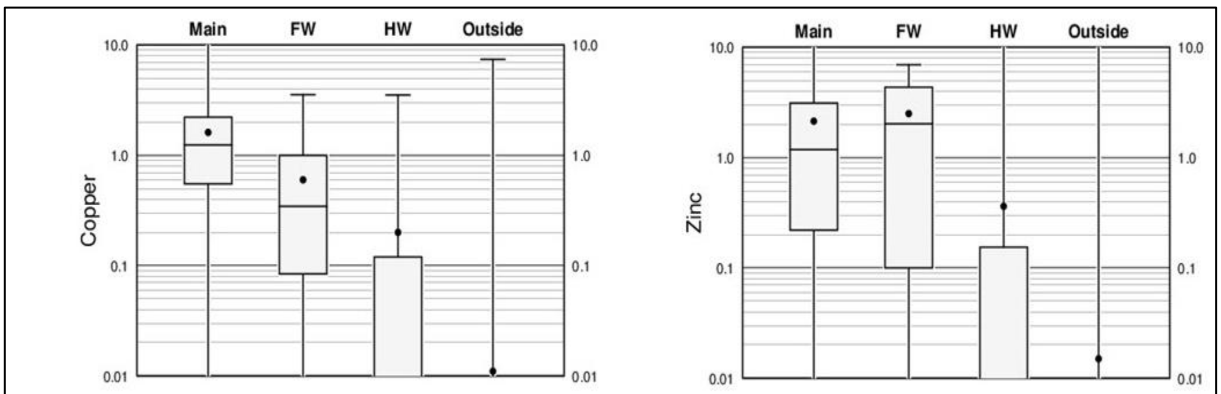
Figure 14-11 is a boxplot showing the distribution of copper by logged alteration type. There are no indications that the distributions of metal or penalty elements are controlled by the types of alteration that have been observed.



	AK	CA	CL	DO	EP	HE	LI	MS	SC	SI
Number of data	703	1564	689	362	2125	602	1006	3235	34	5608
Mean	0.108	0.192	0.008	0.249	0.0	0.016	0.02	0.103	0.063	0.205
Variance	0.158	0.687	0.028	0.806	0.0	0.037	0.044	0.231	0.036	0.654
Skewness	5.116	7.652	24.629	6.62	46.061	12.557	15.102	7.511	4.653	7.03
Coef of Variation	3.693	4.324	19.859	3.607	46.083	11.812	10.556	4.671	3.032	3.943
Maximum	3.58	13.7	4.28	8.76	0.02	2.72	4.35	8.55	1.08	14.17
Upper quartile	0.01	0.0	0.0	0.04	0.0	0.0	0.0	0.0	0.03	0.01
Median	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.015	0.0
Lower quartile	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Minimum	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0

Figure 14-11: Basic Statistics of Copper Samples by Logged Alteration Type – Main Deposit

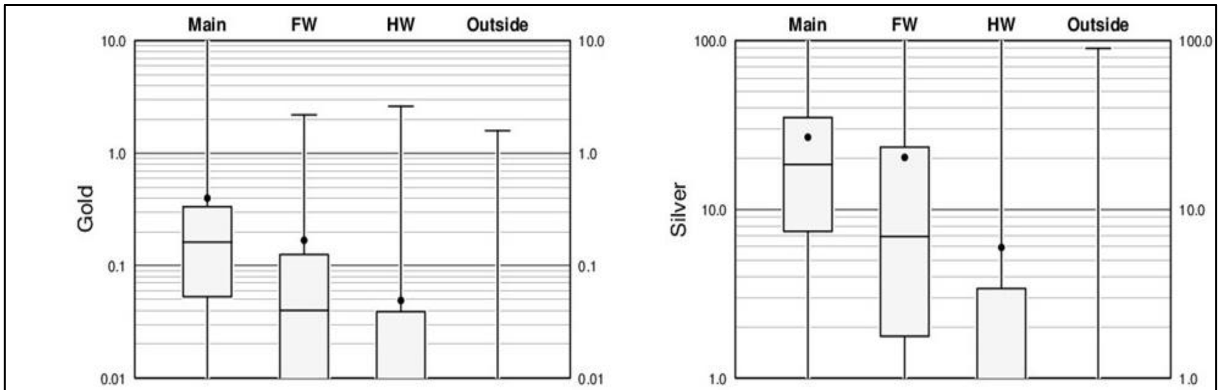
Figure 14-12 and Figure 14-13 are boxplots from the Main deposit that show the grade properties inside and outside of the interpreted grade shell domains. The results are very similar for the distributions of copper, zinc, gold and silver, with elevated grades inside, and essentially no metal present outside the interpreted grade shell domains. Boxplots of arsenic, mercury, antimony and cadmium (Figure 14-14) show similar marked differences inside versus outside of the grade shell domains at the Main deposit. It should be noted that, although the footwall rocks in many areas of these deposits comprise MS/SMS or stringer-type pyrite, there is very little to none of the elevated economic metals evident in these rocks



	Main	FW	HW	Outside
Number of data	4599	52	491	44308
Mean	1.613	0.6	0.2	0.011
Variance	2.494	0.506	0.208	0.01
Skewness	2.615	2.186	3.062	35.06
Coef of Variation	0.979	1.184	2.284	9.236
Maximum	18.13	3.54	3.53	7.42
Upper quartile	2.225	1.0	0.12	0.0
Median	1.24	0.345	0.0	0.0
Lower quartile	0.55	0.084	0.0	0.0
Minimum	0.0	0.0	0.0	0.0

	Main	FW	HW	Outside
Number of data	4599	52	491	44308
Mean	2.14	2.505	0.362	0.015
Variance	8.011	4.917	1.131	0.027
Skewness	3.344	0.439	5.308	68.853
Coef of Variation	1.323	0.885	2.937	10.911
Maximum	36.73	6.94	11.29	19.83
Upper quartile	3.12	4.34	0.154	0.0
Median	1.18	2.024	0.0	0.0
Lower quartile	0.22	0.1	0.0	0.0
Minimum	0.0	0.0	0.0	0.0

Figure 14-12: Boxplots of Copper and Zinc, Inside/Outside the Grade Shell Domains for the Main Deposit

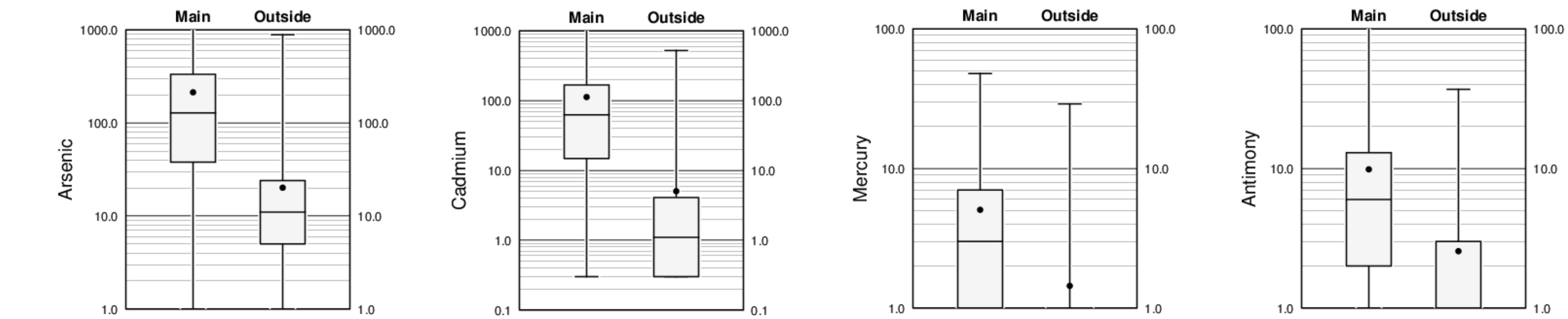


Gold	Main	FW	HW	Outside
Number of data	4599	52	491	44308
Mean	0.398	0.168	0.049	0.004
Variance	18.589	0.186	0.028	0.001
Skewness	62.9	3.839	10.372	34.274
Coef of Variation	10.84	2.575	3.463	6.366
Maximum	285.065	2.19	2.608	1.583
Upper quartile	0.335	0.126	0.039	0.0
Median	0.162	0.04	0.0	0.0
Lower quartile	0.053	0.0	0.0	0.0
Minimum	0.0	0.0	0.0	0.0

Silver	Main	FW	HW	Outside
Number of data	4599	52	491	44308
Mean	26.774	20.331	5.949	0.217
Variance	995.339	2408.35	483.716	2.197
Skewness	3.649	4.871	8.29	23.851
Coef of Variation	1.178	2.414	3.697	6.822
Maximum	481.1	287.5	273.7	90.1
Upper quartile	35.0	23.4	3.403	0.0
Median	18.5	6.9	0.0	0.0
Lower quartile	7.406	1.766	0.0	0.0
Minimum	0.0	0.0	0.0	0.0

Figure 14-13: Boxplots of Gold and Silver Inside/Outside the Grade Shell Domains for the Main Deposit

The Esso deposit examples shown in Figure 14-15 and Figure 14-16 show very similar results, with marked differences in grades inside versus outside of the grade shell domain. Similar grade properties are seen at the Sumac deposit.



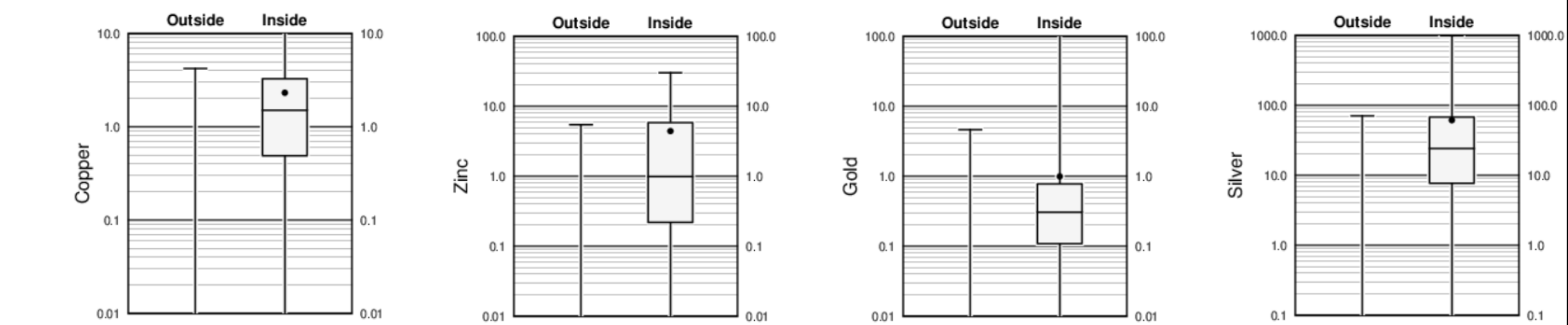
Arsenic	Main	Outside
Number of data	2511	3822
Mean	213.541	20.146
Variance	51137.7	1284.06
Skewness	1.75	11.314
Coef of Variation	1.059	1.779
Maximum	2640.0	886.0
Upper quartile	334.0	24.0
Median	128.415	11.0
Lower quartile	38.0	5.0
Minimum	1.0	1.0

Cadmium	Main	Outside
Number of data	2511	3535
Mean	112.938	5.046
Variance	18356.4	210.678
Skewness	2.366	15.908
Coef of Variation	1.2	2.876
Maximum	1642.8	526.0
Upper quartile	168.319	4.1
Median	62.852	1.1
Lower quartile	14.89	0.3
Minimum	0.3	0.3

Mercury	Main	Outside
Number of data	2511	3535
Mean	5.058	1.441
Variance	25.461	1.438
Skewness	2.552	5.787
Coef of Variation	0.998	0.832
Maximum	48.0	29.0
Upper quartile	7.0	1.0
Median	3.0	1.0
Lower quartile	1.0	1.0
Minimum	1.0	1.0

Antimony	Main	Outside
Number of data	2511	3535
Mean	9.868	2.555
Variance	168.942	8.19
Skewness	5.433	3.326
Coef of Variation	1.317	1.12
Maximum	189.0	37.0
Upper quartile	13.0	3.0
Median	6.0	1.0
Lower quartile	2.0	1.0
Minimum	1.0	1.0

Figure 14-14: Boxplots of Arsenic, Ccdmium, Mercury and Antimony Inside/Outside the Main Grade Shell Domains for the Main Deposit



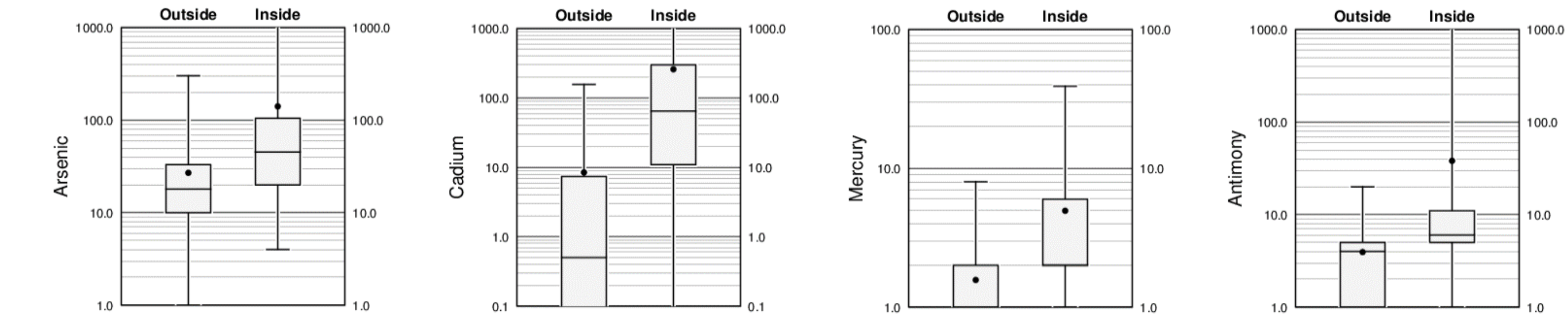
Copper	Outside	Inside
Number of data	60861	767
Mean	0.003	2.319
Variance	0.002	6.48
Skewness	38.888	2.102
Coef of Variation	14.215	1.098
Maximum	4.22	17.0
Upper quartile	0.0	3.28
Median	0.0	1.51
Lower quartile	0.0	0.49
Minimum	0.0	0.0

Zinc	Outside	Inside
Number of data	60861	767
Mean	0.006	4.424
Variance	0.007	48.597
Skewness	31.415	2.049
Coef of Variation	14.838	1.576
Maximum	5.48	30.1
Upper quartile	0.0	5.869
Median	0.0	0.987
Lower quartile	0.0	0.22
Minimum	0.0	0.0

Gold	Outside	Inside
Number of data	60861	767
Mean	0.001	0.995
Variance	0.0	42.74
Skewness	161.388	21.173
Coef of Variation	16.923	6.57
Maximum	4.65	160.0
Upper quartile	0.0	0.781
Median	0.0	0.308
Lower quartile	0.0	0.109
Minimum	0.0	0.0

Silver	Outside	Inside
Number of data	60861	767
Mean	0.085	61.343
Variance	1.159	11320.0
Skewness	33.255	4.278
Coef of Variation	12.678	1.734
Maximum	70.7	989.1
Upper quartile	0.0	67.916
Median	0.0	24.2
Lower quartile	0.0	7.7
Minimum	0.0	0.0

Figure 14-15: Boxplots of Copper, Zinc, Gold and Silver Inside/Outside the Grade Shell Domains for the Esso Deposit



	Arsenic Outside	Arsenic Inside	Cadium Outside	Cadium Inside	Mercury Outside	Mercury Inside	Antimony Outside	Antimony Inside
Number of data	375	187	375	187	375	187	375	187
Mean	26.994	140.969	8.461	259.82	1.569	4.946	3.932	38.096
Variance	920.384	119391.0	366.769	194873.0	0.685	34.079	12.23	15726.0
Skewness	3.552	4.813	3.748	2.616	3.326	2.743	1.49	7.305
Coef of Variation	1.124	2.451	2.263	1.699	0.527	1.18	0.889	3.292
Maximum	301.0	2327.0	156.0	2500.0	8.0	39.0	20.0	1360.0
Upper quartile	33.0	104.465	7.368	302.638	2.0	6.0	5.0	11.0
Median	18.0	45.21	0.5	64.754	1.0	2.0	4.0	6.0
Lower quartile	10.0	20.0	0.0	10.922	1.0	2.0	1.0	5.0
Minimum	1.0	4.0	0.0	0.0	1.0	1.0	1.0	1.0

Figure 14-16: Boxplots of Arsenic, Cadmium, Mercury and Antimony Inside/Outside the Grade Shell Domains for the Esso Deposit

14.5.2 Contact Profiles

Contact profiles evaluate the nature of grade trends between two domains. They graphically display the average grades at increasing distances from the contact boundary. Those contact profiles that show a marked difference in grade across a domain boundary indicate that the two datasets should be isolated during interpolation. Conversely, if a more gradual change in grade occurs across a contact, the introduction of a hard boundary (e.g. segregation during interpolation) may result in a much different trend in the grade model; in this case, the change in grade between domains in the model is often more abrupt than the trends shown in the raw data. Finally, a flat contact profile indicates no grade changes across the boundary; in this case, hard or soft domain boundaries will produce similar results in the model.

A series of contact profiles were produced to evaluate how the grades compare across the grade shell domain boundaries. The profiles in Figure 14-17 show distinct changes in copper-equivalent grades at the contact between all three grade shell domains. Similar results are also shown in the distributions of all other metals and penalty elements evaluated for the Main deposit.

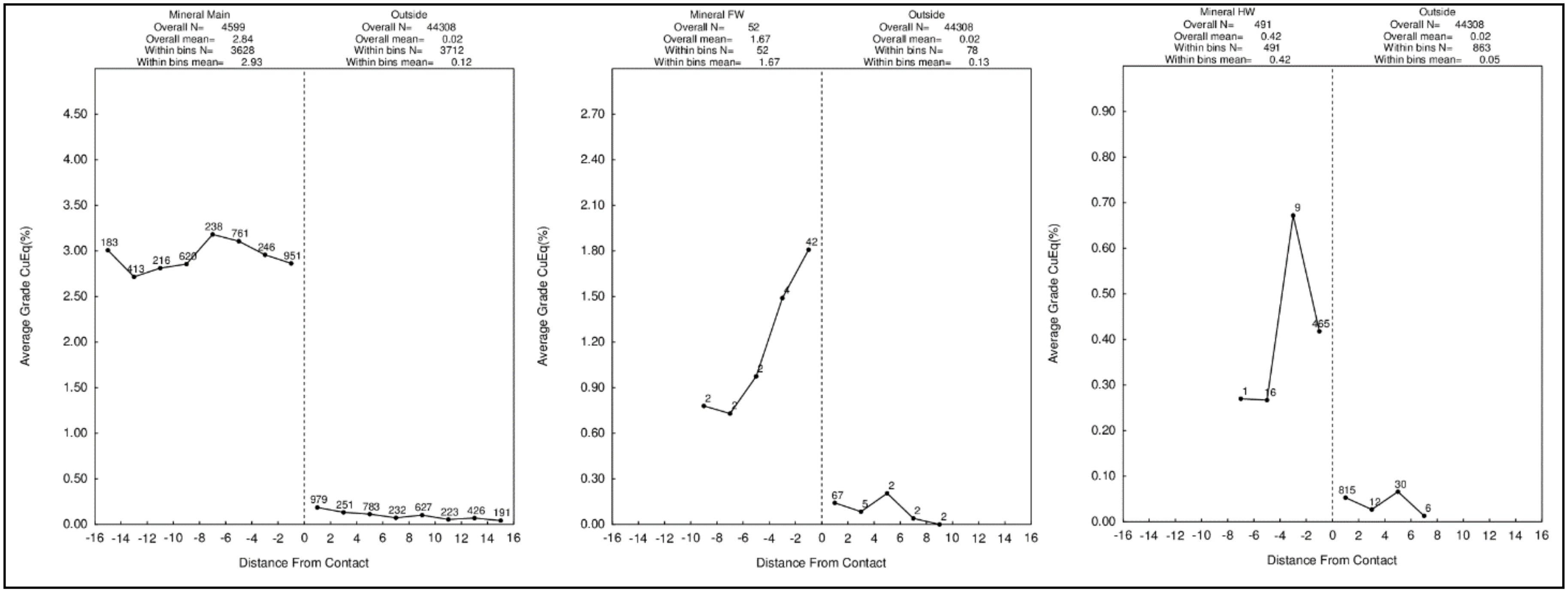


Figure 14-17: Contact Profiles of Copper Grades in the Main Deposit across the Grade Shell Domains Boundaries

Figure 14-18 shows abrupt changes in copper and zinc grades across the grade shell domain boundary at the Esso deposit. Similar results are also shown in the distributions of all other metals and penalty elements evaluated for the Esso deposit.

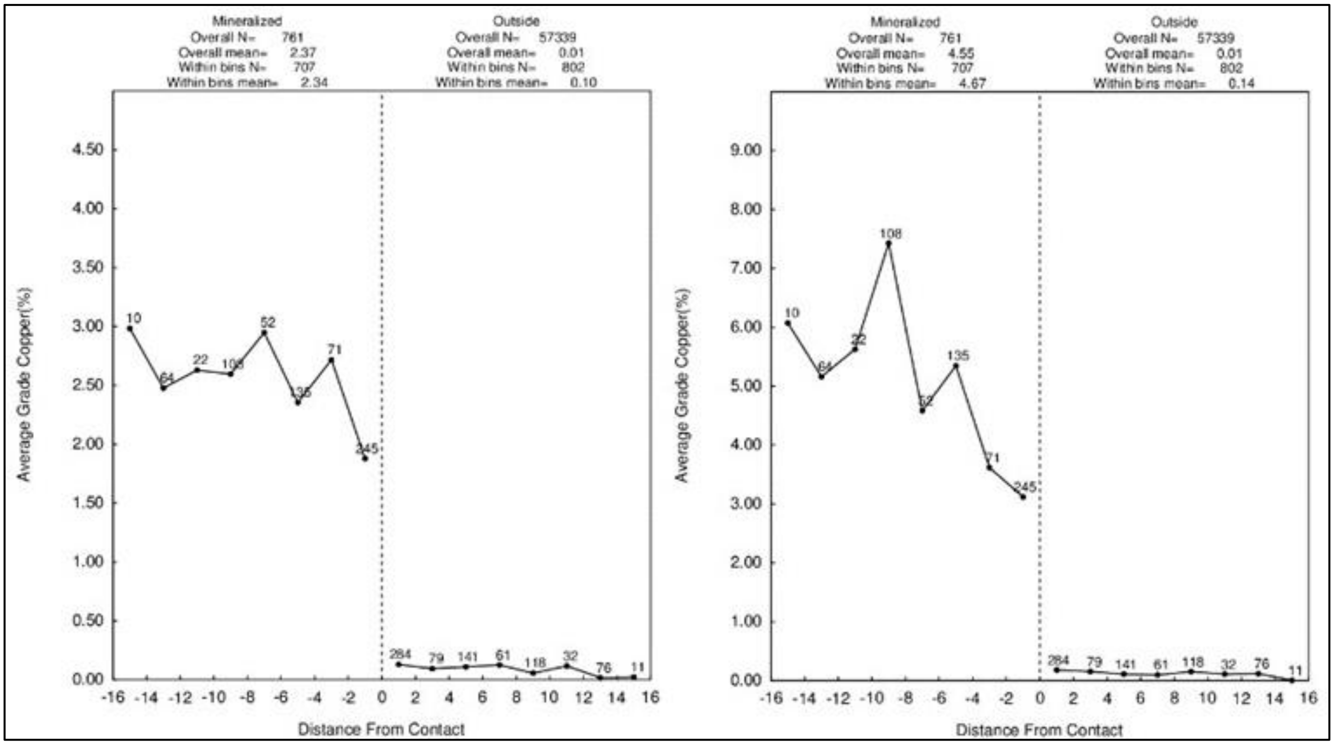


Figure 14-18: Contact Profiles of Copper and Zinc Grades in the Esso Deposit across the Grade Shell Domain Boundaries

14.5.3 Conclusions and Modelling Implications

The results of the EDA indicate that the distribution for all payable metal and additional elements of interest in the Mineral Resource block model are similar; the grade shell domains contain samples that are distinctly different from those located outside of the domains, and, as a result, these populations should remain segregated from one other during block grade interpolations.

The results of the EDA also show that almost all the potential economic mineralization is present inside the grade shell domains, and the surrounding rocks are essentially barren. However, grade estimates were made both inside and outside of the grade domain to gain a better understanding of the nature of the dilution material incorporated during mining.

Evaluation of grade versus lithology designations derived from core logging shows that essentially all significant grades are contained primarily in MS or SMS units, and the grade distributions are similar between these rock types. The interpreted grade shell domains mimic the shape and distribution of MS and SMS rocks at all the Kutcho deposit. There are no indications that the distributions of metals or penalty elements are related to the distributions of the various alteration assemblages.

14.6 Evaluation of Outlier Grades

Histograms and probability plots were evaluated to identify the presence of anomalous outlier grades or values in the composited (1 m) database. Following a review of the physical location of potentially erratic samples in relation to the surrounding sample data, it was decided that these would be controlled during block grade interpolations using a combination of traditional top-cutting and the application of outlier limitations. An outlier limitation controls the distance of influence of samples above a defined grade threshold. During grade interpolations, samples above the outlier thresholds are limited to a maximum distance-of-influence of 25 m for copper, zinc, gold and silver and 50 m for the other four penalty elements at the Main and Esso deposits. The outlier ranges are expanded to 50 m for all elements at the Sumac deposit which reflects the wider drillhole spacing.

The grade thresholds for all elements are shown in the Table 14-4 through Table 14-6. Overall, in the Main deposit, these measures have reduced the contained copper and zinc by about 1%, contained gold by 7%, and contained silver by 4%. At Esso, these measures have reduced the contained copper by 5%, zinc by 8%, gold by 27%, and silver by 9%. At Sumac, these measures have reduced the contained copper and zinc by 3%, gold by 2%, and silver by 4%.

Table 14-4: Treatment of outlier data at the Main deposit

Metal	Maximum	Top cut	Outlier
Main domain			
Copper	18.13	-	10
Zinc	36.73	-	25
Silver	481.1	300	150
Gold	285.065	40	15
Iron	50.10	-	-
Sulphur	56.31	-	-
Mercury	48	-	35
Arsenic	2640	1500	1000
Antimony	189	-	120
Cadmium	1642.8	1000	700
Footwall domain			
Copper	3.54	-	1
Zinc	6.94	-	3
Silver	287.5	40	20
Gold	2.190	-	0.3
Iron	15.35	-	-
Sulphur	31.60	-	-
Mercury	7	-	-
Arsenic	60	-	-
Antimony	20	-	-
Cadmium	177.2	-	-
Hanging wall domain			
Copper	3.53	-	1.5
Zinc	11.29	-	3.5
Silver	273.7	200	80
Gold	2.608	-	0.6
Iron	38.40	-	-
Sulphur	45.30	-	-
Mercury	11	-	-
Arsenic	1240	500	150
Antimony	24	-	-
Cadmium	343	-	150

Metal	Maximum	Top cut	Outlier
Outside domains			
Copper	7.42	4	2
Zinc	19.83	5	2.5
Silver	90.1	60	30
Gold	1.583	-	0.6
Iron	46.52	-	-
Sulphur	52.88	-	-
Mercury	29	-	-
Arsenic	886	-	200
Antimony	37	-	25
Cadmium	526	250	100

Table 14-5: Treatment of outlier sample data at Esso deposit

Metal	Maximum	Top cut	Outlier
Copper	17.00	-	13/25 m
Zinc	30.10	-	28/25 m
Silver	989.10	-	600/25 m
Gold	160.00	30	10/25 m
Mercury	39 – only 190 samples	-	20
Arsenic	2,327 – only 190 samples	-	1,500
Antimony	1,360 – only 190 samples	-	300
Cadmium	2,500 – only 190 samples	-	1,000

Table 14-6: Treatment of outlier sample data at Sumac deposit

Metal	Maximum	Top cut	Outlier
Copper	4.36	-	2.50
Zinc	11.55	8	5.00
Silver	99.3	-	60.0
Gold	1.083	-	0.70
Mercury	30	-	15
Arsenic	898	-	700
Antimony	50	-	25
Cadmium	502	-	200

The higher metal losses described here are due to a combination of skewed data distributions and the spacing of drillholes. These measures are considered appropriate for a deposit with this distribution of delineation drilling.

14.7 Variography

The degree of spatial variability in a mineral deposit depends on both the distance and direction between points of comparison. Typically, the variability between samples increases as the distance between those samples increases. If the degree of variability is related to the direction of comparison, then the deposit is said to exhibit anisotropic tendencies which can be summarized with the search ellipse. The semi-variogram is a common function used to measure the spatial variability within a deposit.

The components of the variogram include the nugget, the sill, and the range. Often samples compared over very short distances, even samples compared from the same location, show some degree of variability. As a result, the curve of the variogram often begins at some point on the y-axis above the origin: this point is called the nugget. The nugget is a measure of not only the natural variability of the data over very short distances, but also a measure of the variability which can be introduced due to errors during sample collection, preparation, and the assay process.

The amount of variability between samples typically increases as the distance between the samples increases. Eventually, the degree of variability between samples reaches a constant, maximum value: this is called the sill, and the distance between samples at which this occurs is called the range.

In this Technical Report, the spatial evaluation of the data was conducted using a correlogram rather than the traditional variogram. The correlogram is normalized to the variance of the data and is less sensitive to outlier values, generally giving better results.

Variograms (correlograms) were generated using the commercial software package Sage 2001© developed by Isaaks & Co. Multidirectional variograms for copper, zinc, gold, and silver were generated from the distributions of data located inside the grade shell domain, and the vertical distances are defined relative to the distance from the trend plane. The same variograms are used to estimate the grades both inside and outside of the domain. The results are summarized for the Main and Esso deposits in Table 14-7 and Table 14-8. Data are limited for the deleterious elements, and, as a result, reliable variograms could not be generated.

Similarly, reliable variograms could not be produced for the Sumac deposit due to limited data.

Table 14-7: Main deposit variogram parameters

Element	Nugget	Sill 1	Sill 2	1st Structure			2nd Structure		
				Range (m)	Azimuth (°)	Dip	Range (m)	Azimuth (°)	Dip
Copper	0.320	0.583	0.097	14	134	0	745	108	0
	Spherical			9	44	0	43	18	0
				8	90	90	8	90	90
Zinc	0.230	0.147	0.624	1029	106	0	26	40	0
	Spherical			71	16	0	16	310	0
				6	90	90	8	90	90
Gold	0.432	0.501	0.067	32	304	0	80	119	0
	Spherical			6	90	90	6	29	0
				5	34	0	6	90	90
Silver	0.350	0.528	0.122	25	342	0	701	108	0
	Spherical			8	72	0	35	18	0
				6	90	90	6	90	90

Note: Correlograms were conducted on 1 m composite sample data and Z-range is relative to trend plane.

Table 14-8: Esso deposit variogram parameters

Element	Nugget	Sill 1	Sill 2	1st Structure			2nd Structure		
				Range (m)	Azimuth (°)	Dip	Range (m)	Azimuth (°)	Dip
Copper	0.100	0.767	0.133	25	24	0	138	324	0
	Spherical			14	90	90	32	90	90
				10	294	0	28	54	0
Zinc	0.150	0.511	0.339	34	88	0	110	337	0
	Spherical			9	358	0	33	90	90
				7	90	90	25	67	0
Gold	0.350	0.551	0.099	28	98	0	66	9	0
	Spherical			9	8	0	17	99	0
				8	90	90	9	90	90

Element	Nugget	Sill 1	Sill 2	1st Structure			2nd Structure		
				Range (m)	Azimuth (°)	Dip	Range (m)	Azimuth (°)	Dip
Silver	0.350	0.481	0.169	28	117	0	91	351	0
	Spherical			12	90	90	22	81	0
				8	27	0	10	90	90

Note: Correlograms were conducted on 1 m composite sample data and Z-range is relative to trend plane.

14.8 Model Setup and Limits

Block models for each of the three mineral deposits were initialized in MinePlan®; the dimensions are shown in Table 14-9, Table 14-10 and Table 14-11. A nominal block size measuring 5 x 5 x 5 m (L x W x H) was selected for the Main Kutcho deposit, as this is projected to be mined primarily using open pit extraction methods. A smaller block size, measuring 5 x 2.5 x 5 m (L x W x H), was selected for the Esso and Sumac deposits which are projected to be mined using underground extraction methods. The block sizes are somewhat small relative to the spacing of drillholes, but they generally represent the selective mining unit (SMU) size typical of the extraction methods considered and, therefore, these are considered appropriate for the estimation of Mineral Resources. The extents of the three block models are represented by the purple rectangles shown in Figure 14-1 through Figure 14-9.

Table 14-9: Block model limits for Main deposit

Direction	Minimum	Maximum	Block size (m)	No. of blocks
X (east)	536500	538500	5	400
Y (north)	6451500	6452500	5	200
Z (elevation)	1000	1700	5	140

Table 14-10: Block model limits for Esso deposit

Direction	Minimum	Maximum	Block size (m)	No. of blocks
X (east)	534900	535950	5	210
Y (north)	6452200	6452800	2.5	240
Z (elevation)	700	1600	5	180

Table 14-11: Block model limits for Sumac deposit

Direction	Minimum	Maximum	Block size (m)	No. of blocks
X (east)	535550	536700	5	230
Y (north)	6451800	6452550	2.5	300
Z (elevation)	800	1700	5	180

Blocks in the model were coded using the grade shell domains; the percentage of the block inside the domain was determined and used as a weighting factor to calculate in-situ Mineral Resource s. Similarly, the percentage of the block outside of the grade shell domain was also determined and used as a weighting factor to calculate the grade of material outside of the domain for dilution purposes.

14.9 Interpolation Parameters

The block model grades for copper, zinc, gold, and silver were estimated using ordinary kriging (OK). The results of the OK estimation were compared with the Hermitian Polynomial Change of Support model (also referred to as the Discrete Gaussian Correction). This method is described in more detail in Section 14.10.2.

The OK models were generated with a relatively limited number of samples to match the change of support or Herco (Hermitian Correction) grade distribution. This approach reduces the amount of smoothing or averaging in the model, and, while there may be some uncertainty on a localized scale, this approach produces reliable estimates of the recoverable grade and tonnage for the overall deposit. Additional elements were estimated using the inverse distance squared weighting (ID^2) interpolation method.

All grade estimations use length-weighted composite drillhole sample data. The estimation parameters for the various elements in the Main domain of the Main deposit are shown in Table 14-12. Grades for copper, zinc, lead, gold, and silver in the footwall and hanging wall domains were all estimated using a maximum of 21 composites with a maximum of seven composites per drillhole. The estimation parameters for the various elements in the Esso deposit are shown in Table 14-13.

Table 14-12: Interpolation parameters for the main domain at the Main deposit

Element	Search ellipse range (m)			No. of composites			Other
	X	Y	Z (1)	Minimum/block	Maximum/block	Maximum/hole	
Copper	200	200	4	5	21	7	OK estimate
Zinc	200	200	4	5	21	7	OK estimate
Gold	200	200	4	5	15	5	OK estimate
Silver	200	200	5	5	27	9	OK estimate
Arsenic	300	300	10	5	21	7	ID2 estimate
Antimony	300	300	10	5	21	7	ID2 estimate
Mercury	300	300	10	5	21	7	ID2 estimate
Cadmium	300	300	10	5	21	7	ID2 estimate
SG	200	200	7	3	15	5	ID2 estimate

(1) Z-range is relative to the distance from the trend ellipse orientation with long axis north-south and west-east and vertical short axis.

Table 14-13: Interpolation parameters at the Esso deposit

Element	Search ellipse range (m)			No. of composites			Other
	X	Y	Z (1)	Minimum/block	Maximum/block	Maximum/hole	
Copper	200	200	3	4	12	4	OK estimate
Zinc	200	200	3	4	12	4	OK estimate
Gold	200	200	4	4	15	5	OK estimate
Silver	200	200	4	4	15	5	OK estimate
Arsenic	300	300	5	3	15	5	ID2 estimate
Antimony	300	300	5	3	15	5	ID2 estimate
Mercury	300	300	5	3	15	5	ID2 estimate
Cadmium	300	300	5	3	15	5	ID2 estimate
SG	200	200	7	2	9	3	ID2 estimate

(1) Z-range is relative to the distance from the trend ellipse orientation with long axis north-south and west-east and vertical short axis.

Grades at the Sumac deposit were estimated using the ID^2 interpolation method using a maximum of five composites from a single drillhole and a maximum of 15 composites in total to estimate a block grade.

SG values at Main and Esso were estimated using the ID^2 interpolation method. There is insufficient SG data at Sumac to support estimation into model blocks and, as a result, an average SG of 3.00 was assumed for blocks inside the grade shell domain at Sumac.

14.10 Validation

The results of the modelling process were validated using several methods. These include a thorough visual review of the model grades in relation to the underlying drillhole sample grades, comparisons with the change of support model, comparisons with other estimation methods and grade distribution comparisons using swath plots.

14.10.1 Visual Inspection

A detailed visual inspection of the block model was conducted in both section and plan to ensure the desired interpolation results. This includes confirmation of the proper coding of blocks within the grade shell domain. The estimated copper, zinc, gold, and silver grades in the model appear to be a valid representation of the underlying drillhole sample data. Examples of the distribution of copper and zinc grades in model blocks compared to the drillhole sample data are shown in Figure 14-19 and Figure 14-20 for the Main deposit and in Figure 14-21 and Figure 14-22 for the Esso deposit.

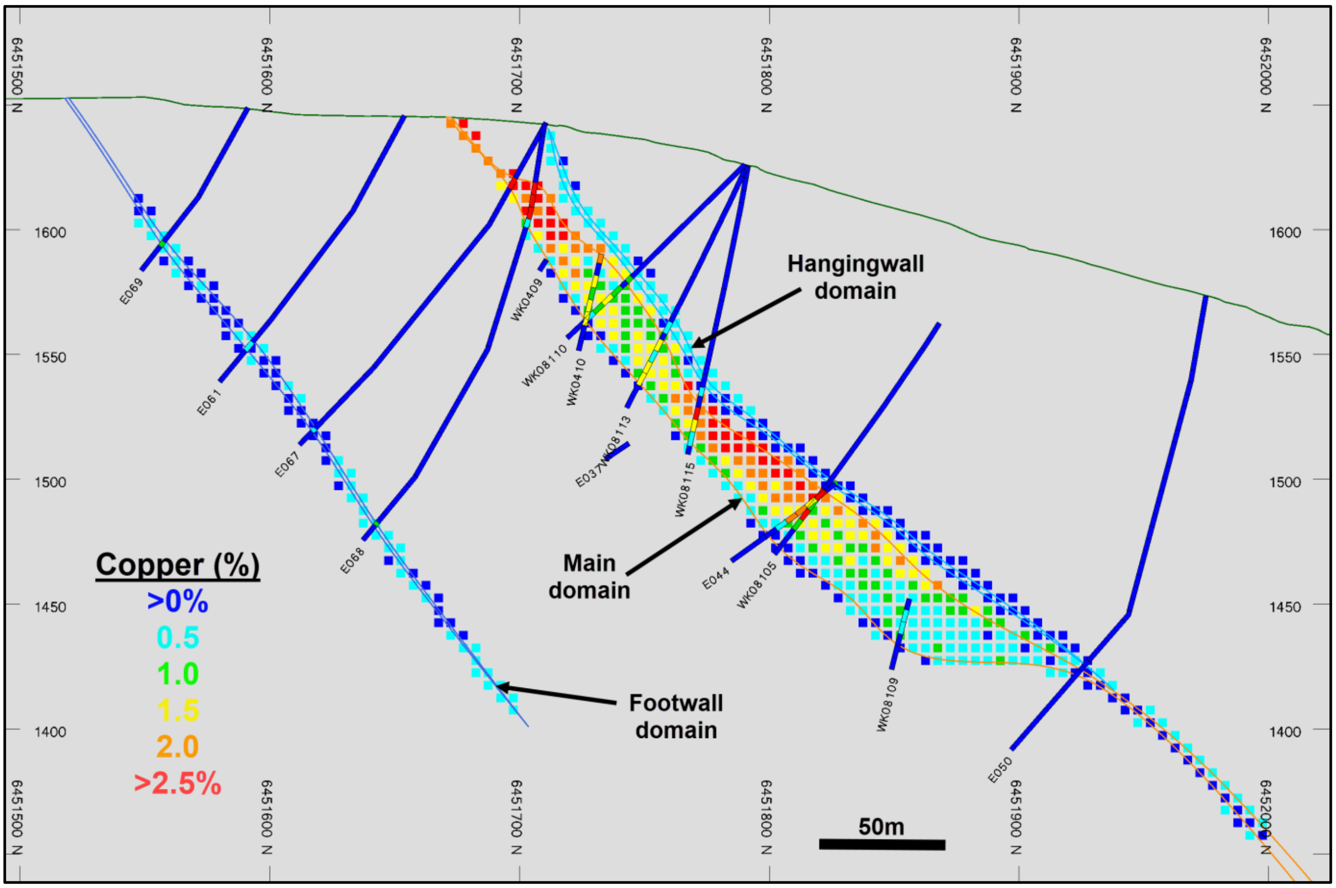


Figure 14-19: Vertical Cross-Section 537875E Showing Copper Grades in Drilling and Block Model in Main Deposit

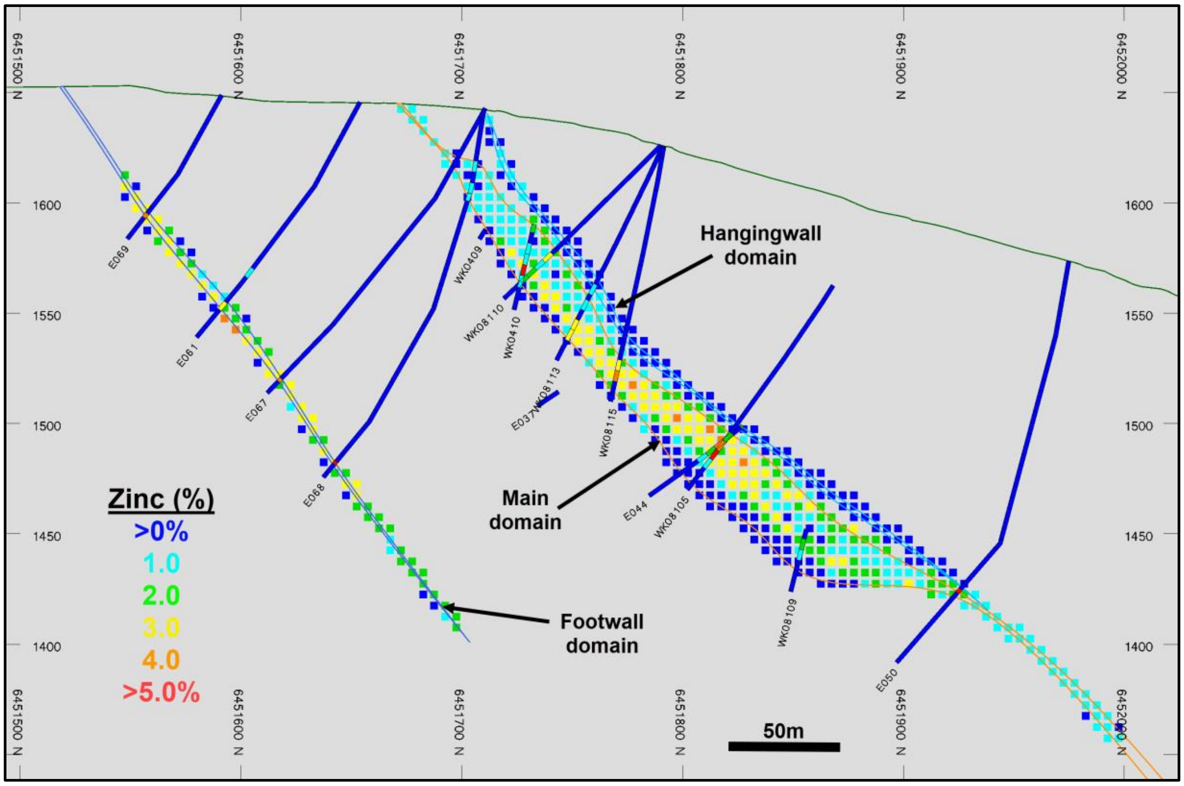


Figure 14-20: Vertical Cross-Section 537875E Showing Zinc Grades in Drilling and Block Model in Main Deposit

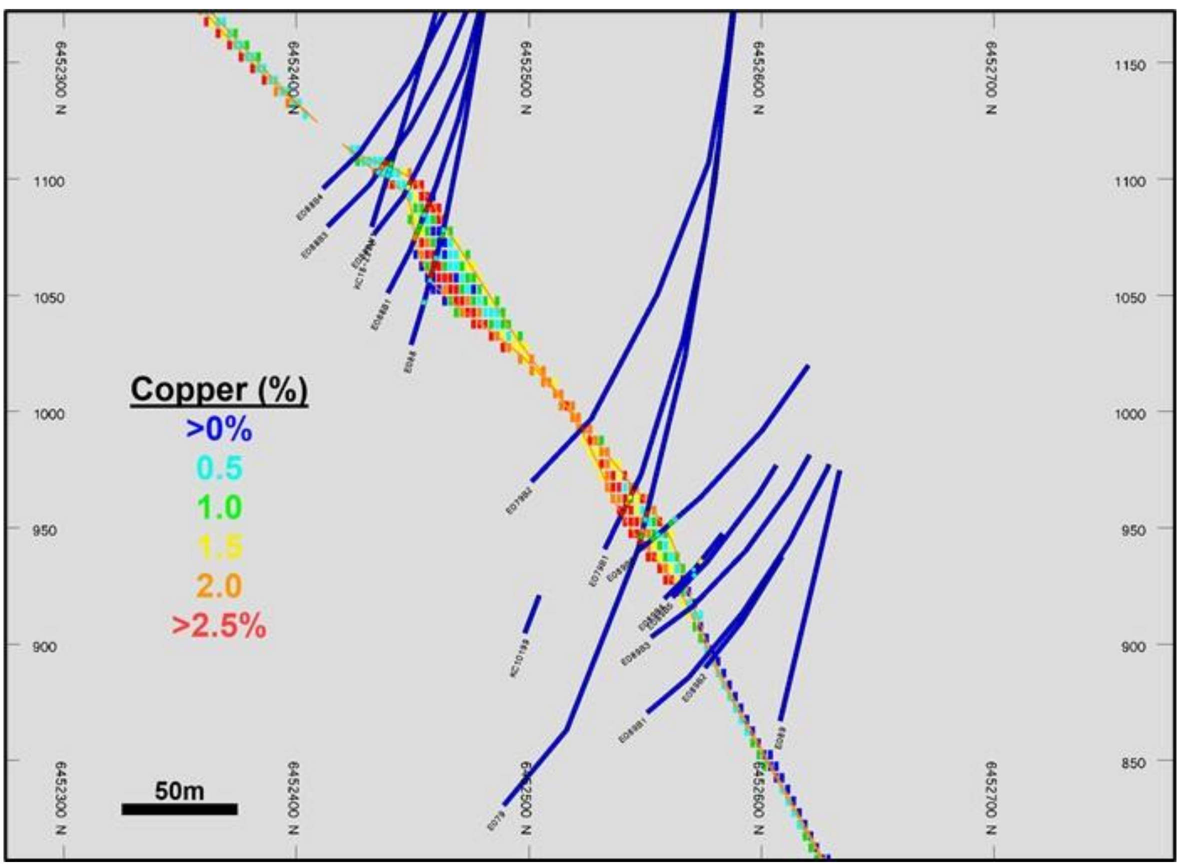


Figure 14-21: Vertical Cross-Section 535250E Showing Copper Grades in Drilling and Block Model in Esso Deposit

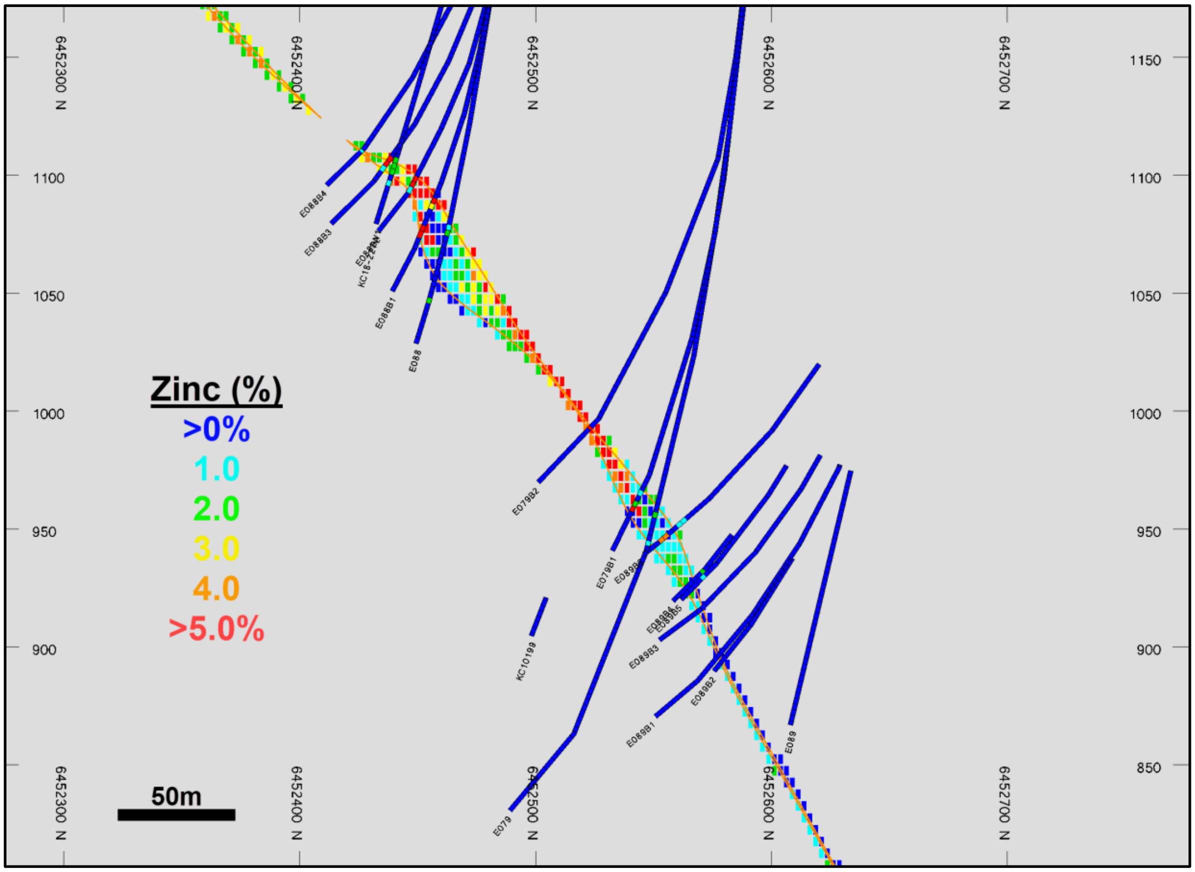


Figure 14-22: Vertical Cross-Section 535250E Showing Zinc Grades in Drilling and Block Model in Esso Deposit

14.10.2 Model Checks for Change of Support

The relative degree of smoothing in the block model estimates were evaluated using the Discrete Gaussian of Hermitian Polynomial Change of Support method (described by Rossi and Deutsch, Mineral Resource Estimation, 2014).

With this method, the distribution of the hypothetical block grades can be directly compared to the estimated (OK) model using pseudo-grade/tonnage curves. Adjustments are made to the block model interpolation parameters until an acceptable match is made with the Herco distribution. In general, the estimated model should be slightly higher in tonnage and slightly lower in grade when compared to the Herco distribution at the projected cut-off grade. These differences account for selectivity and other potential ore-handling issues which commonly occur during mining.

The Herco distribution is derived from the de-clustered composite grades which were adjusted to account for the change in support, going from smaller drillhole composite samples to the large blocks in the model. The transformation results in a less skewed distribution but with the same mean as the original de-clustered samples. The Herco analysis was conducted on the distribution of all metals in the Mineral Resource block model and a level of correspondence was achieved in all cases.

Examples showing the distributions of the copper and zinc models are shown in Figure 14-23 and Figure 14-24 for the Main deposit and in Figure 14-25 and Figure 14-26 for the Esso deposit.

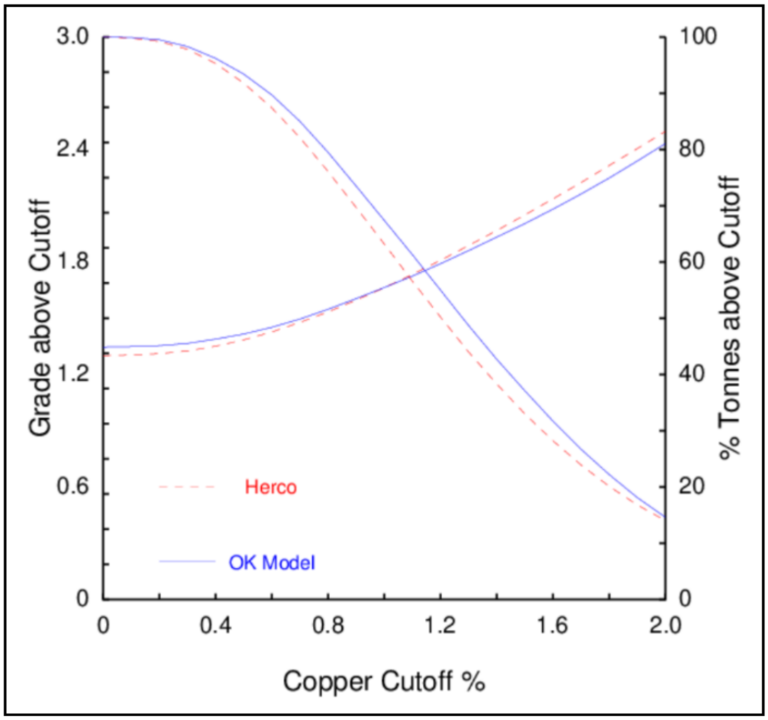


Figure 14-23: Herco Grade/Tonnage Plot for Copper Models in the Main Deposit

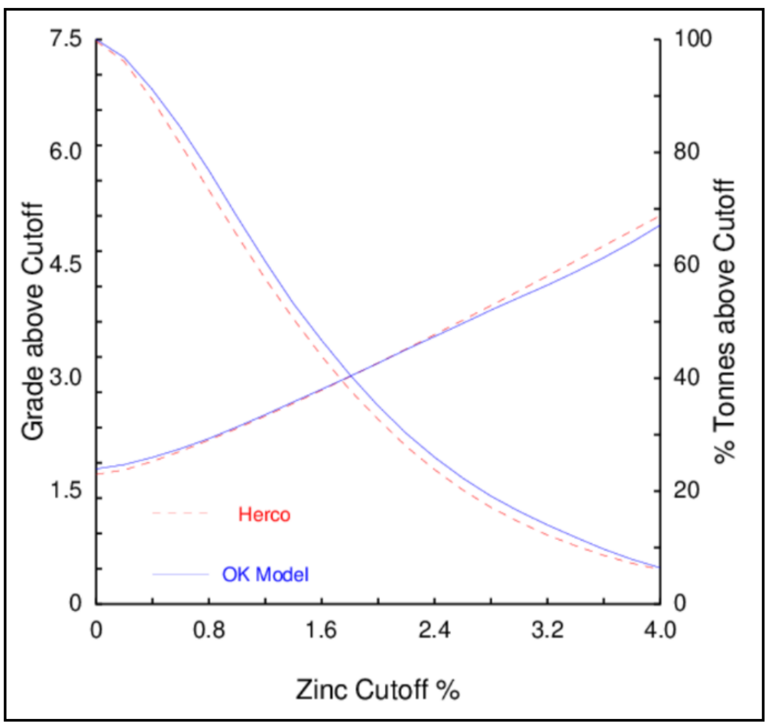


Figure 14-24: Herco Grade/Tonnage Plot for Zinc Models in the Main Deposit

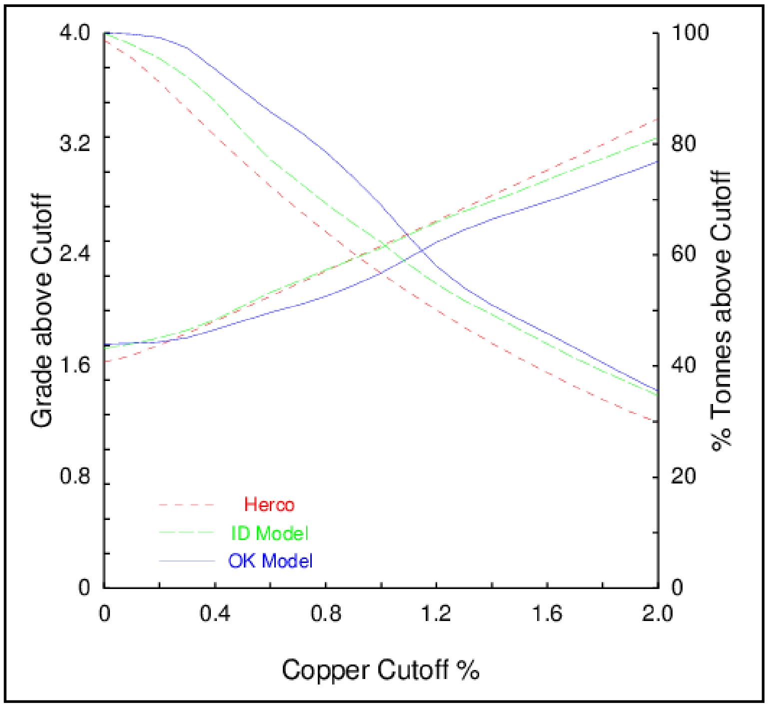


Figure 14-25: Herco Grade/Tonnage Plot for Copper Models in the Esso Deposit

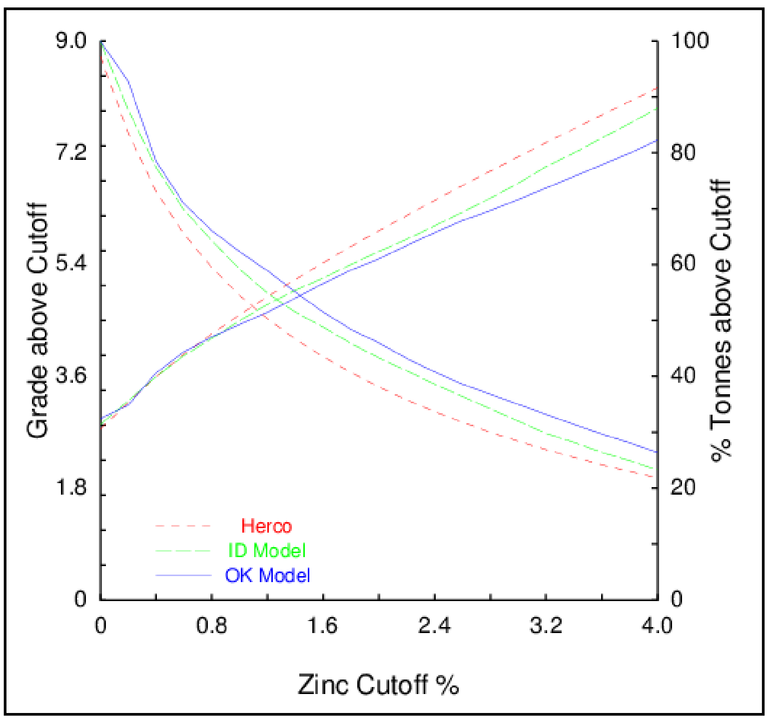


Figure 14-26: Herco Grade/Tonnage Plot for Zinc Models in the Esso Deposit

14.10.3 *Comparison of Interpolation Methods*

For comparison purposes, additional models for copper, zinc, gold, silver, and lead were generated using both the inverse distance weighted (IDW) and nearest neighbour (NN) interpolation methods. The NN model was generated using data composited to 2.5 m intervals.

Comparisons are made between these models on grade/tonnage curves. Examples of the grade/tonnage curves for the copper and zinc models are shown in Figure 14-27 and Figure 14-28 for the Main deposit and in Figure 14-29 and Figure 14-30 for the Esso deposit. There is good correlation between the OK and ID models throughout the range of cut-off grades. The NN distribution, generally showing less tonnage and higher grade, is the result of the absence of smoothing in this modelling approach. Similar results were achieved with the gold, silver, and lead models. Reproduction of the model using different methods tends to increase the confidence in the overall Mineral Resource.

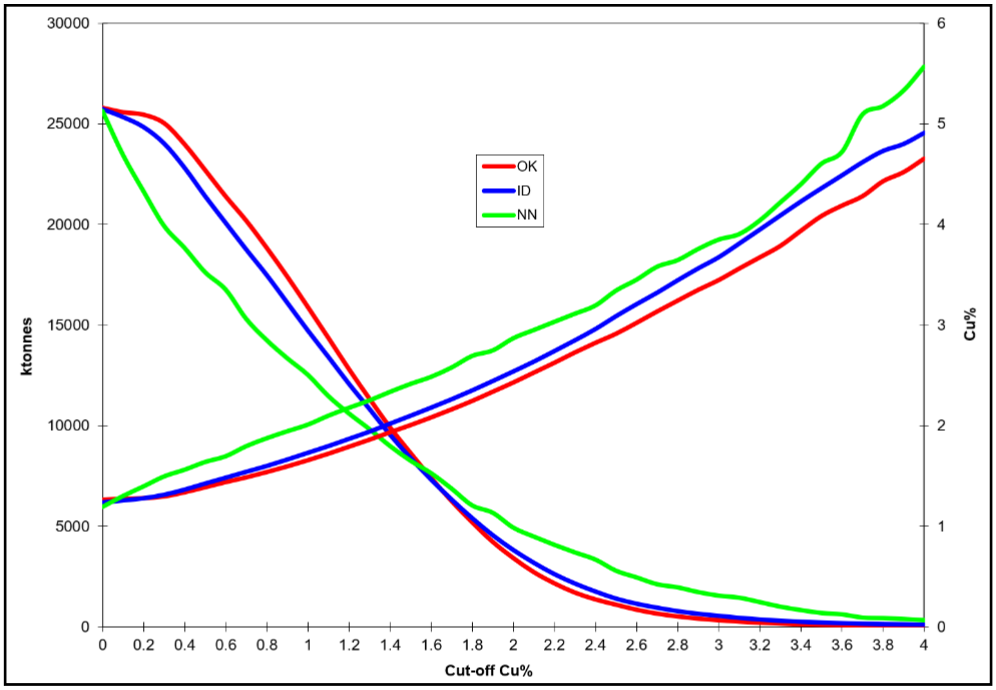


Figure 14-27: Grade/Tonnage Comparison of Copper Models in the Main Deposit

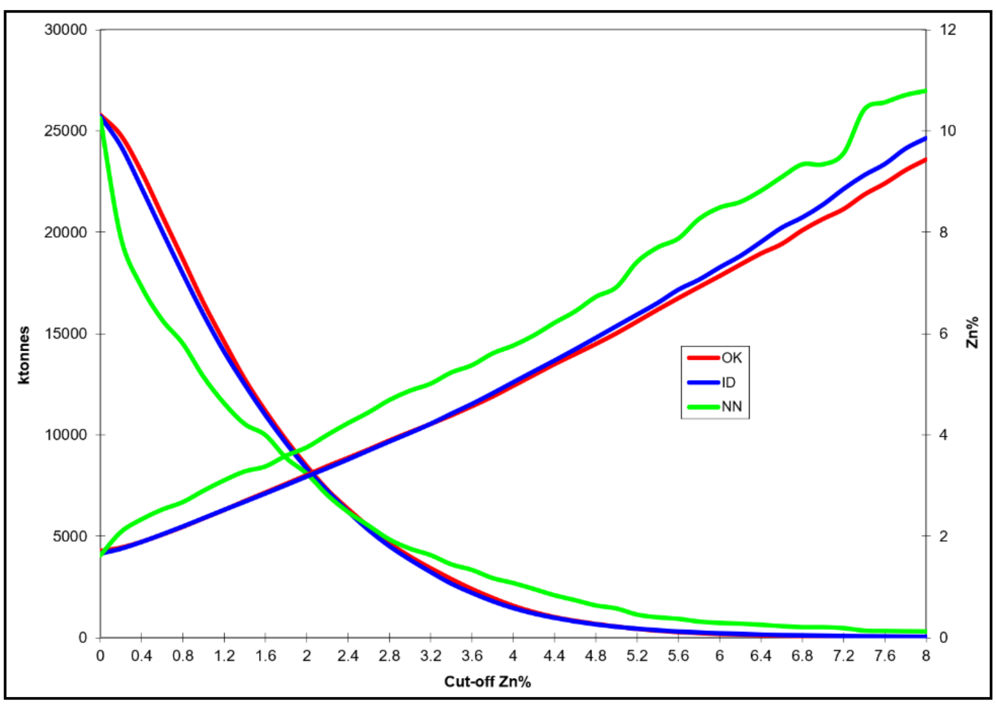


Figure 14-28: Grade/Tonnage Comparison of Zinc Models in the Main Deposit

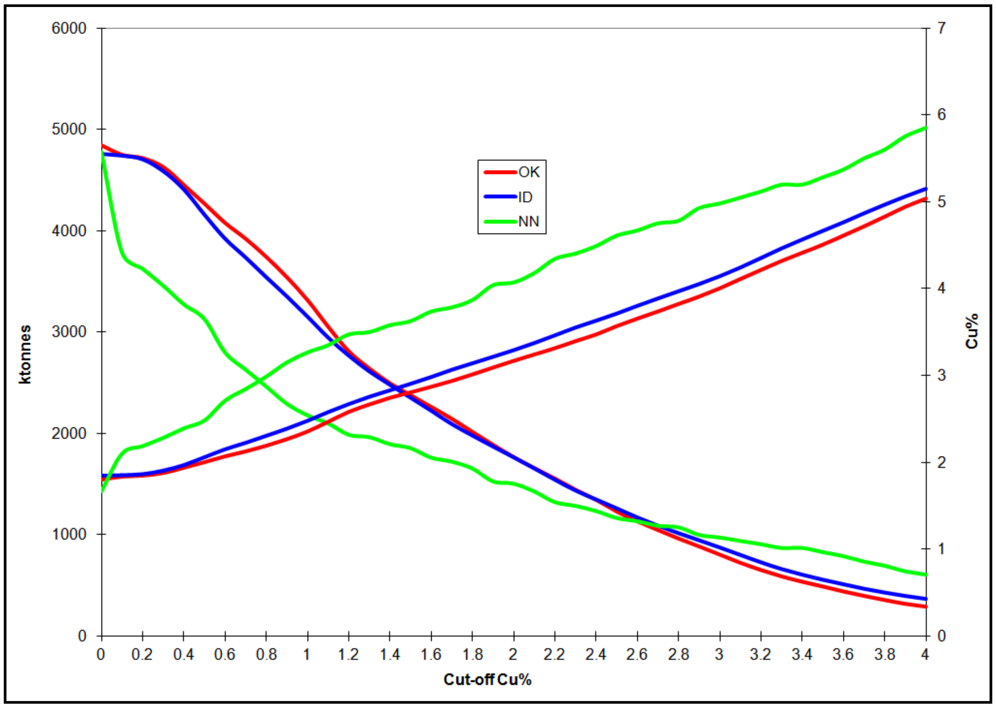


Figure 14-29: Grade/Tonnage Comparison of Copper Models in the Esso Deposit

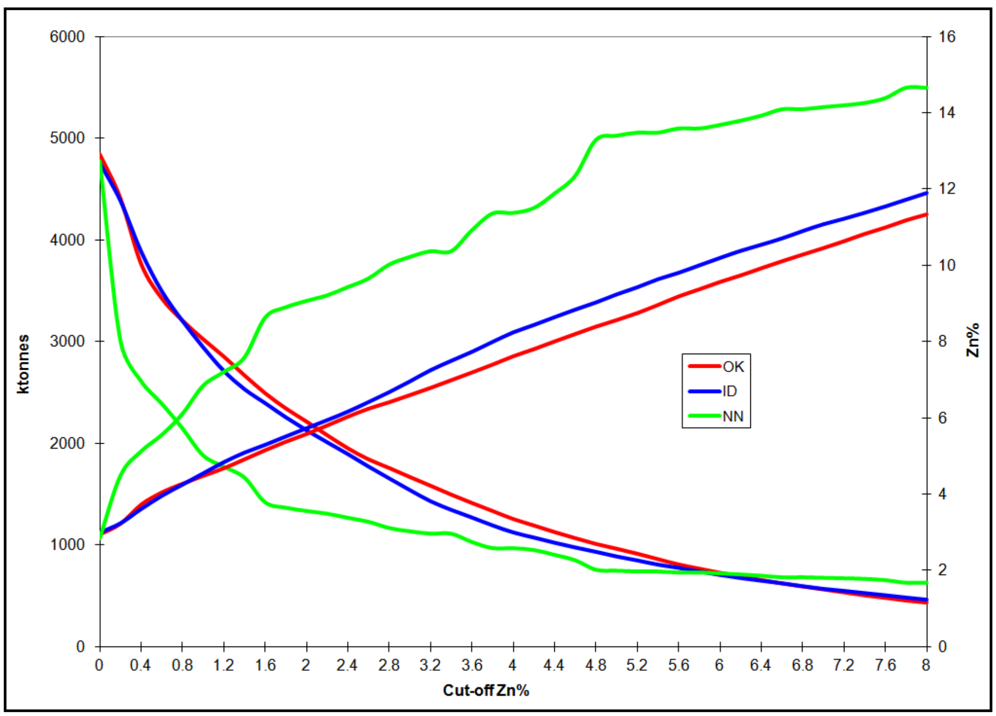


Figure 14-30: Grade/Tonnage Comparison of Zinc Models in the Esso Deposit

14.10.4 ***Swath Plots (Drift Analysis)***

A swath plot is a graphical display of the grade distribution derived from a series of bands, or swaths, generated in several directions through the deposit. Grade variations from the OK model are compared using the swath plot to the distribution derived from the de-clustered (NN) grade model.

On a local scale, the NN model does not provide reliable estimations of grade, but, on a much larger scale, it represents an unbiased estimation of the grade distribution based on the underlying data. Therefore, if the OK model is unbiased, the grade trends may show local fluctuations on a swath plot, but the overall trend should be like the NN distribution of grade.

Swath plots were generated in three orthogonal directions for all models. Examples of the copper and zinc distributions in north-south swaths are shown in Figure 14-31 for the Main deposit and in Figure 14-32 for the Esso deposit.

There is good correspondence between the models in most areas. The degree of smoothing in the OK model is evident in the peaks and valleys shown in the swath plots. Areas where there are large differences between the models tend to be the result of "edge" effects, where there is less available data to support a comparison. Note that most of the Mineral Resource occurs between 537000E and 538300E for the Main deposit and between 535150E and 535600E for the Esso deposit.

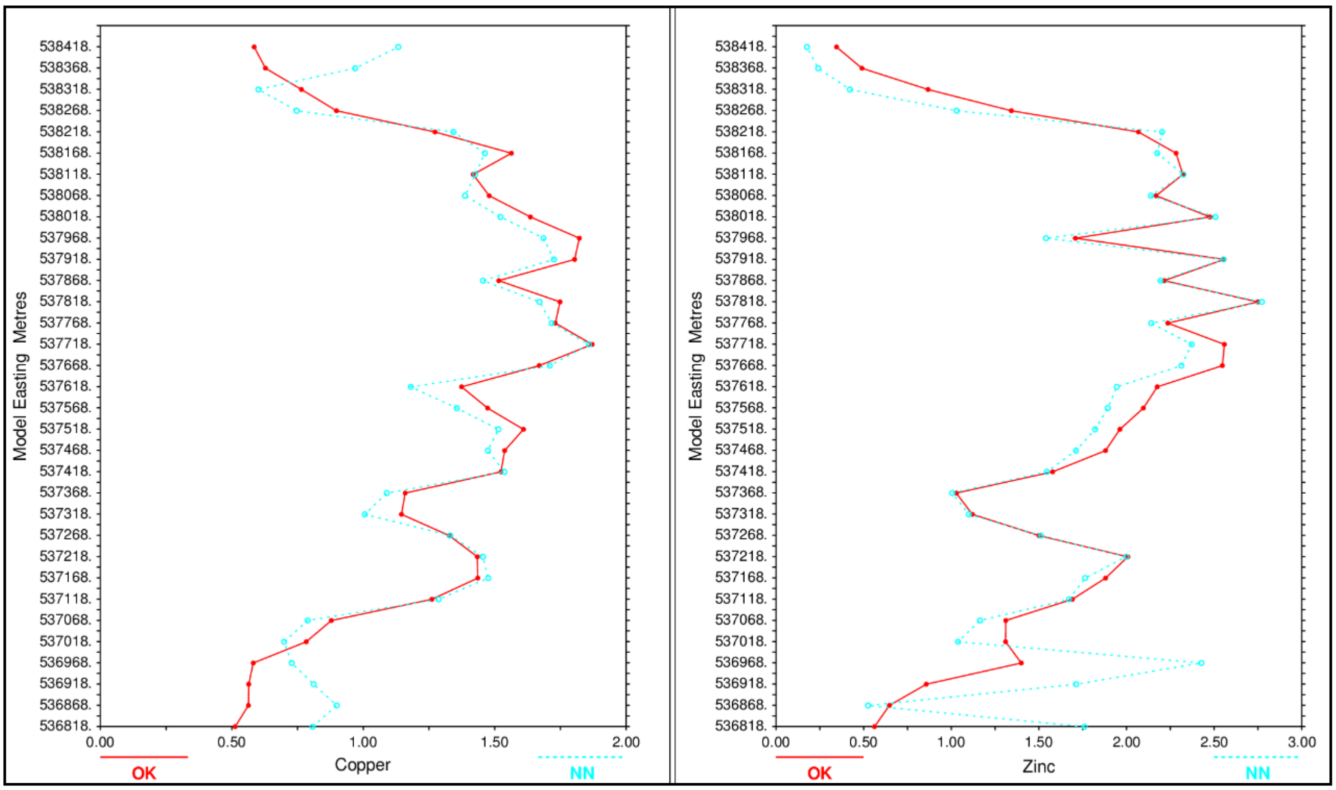


Figure 14-31: Swath Plot of Copper and Zinc OK and NN Models by Easting in the Main Deposit

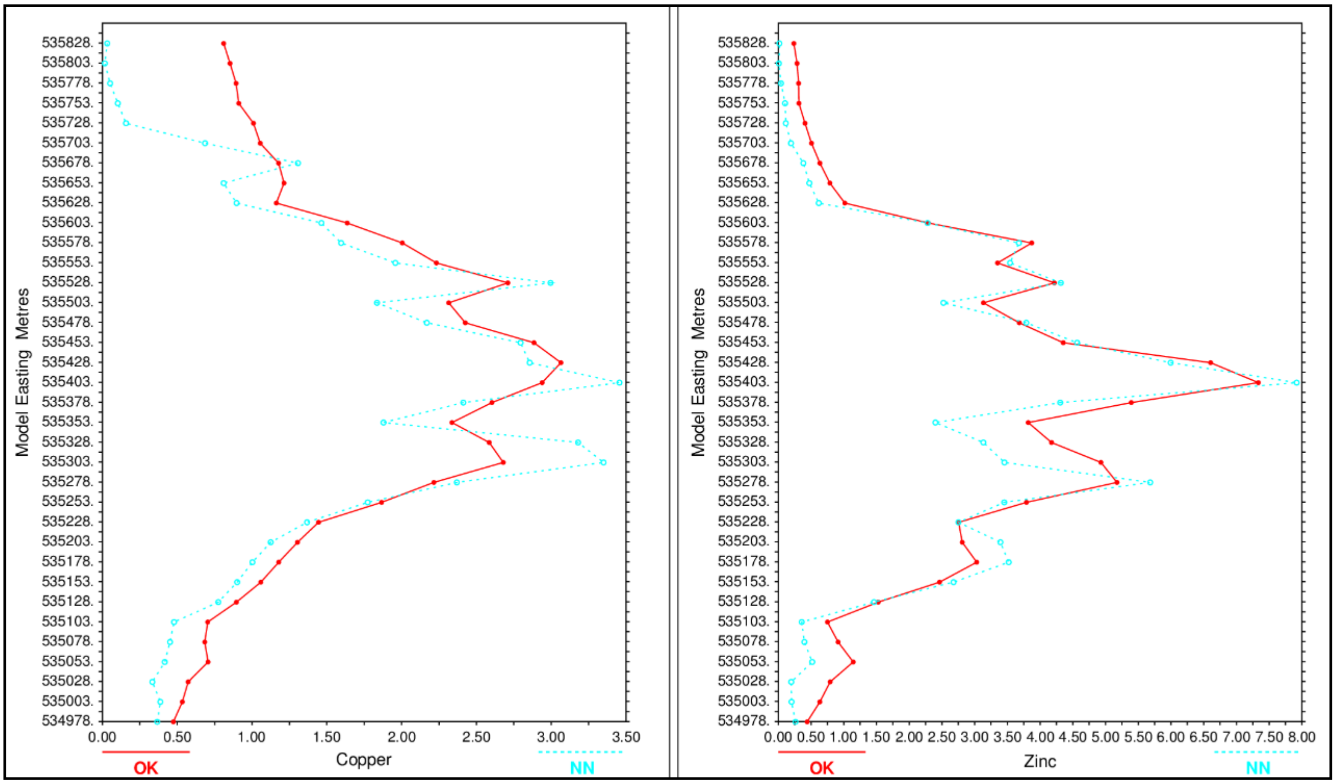


Figure 14-32: Swath Plot of Copper and Zinc OK and NN Models by Easting in the Esso Deposit

14.10.5 Conclusions

The validation results indicate that the OK models at the Main and Esso deposits and the ID models at Sumac are reasonable reflections of the underlying sample data.

14.10.6 Mineral Resource Classification

The Mineral Resources for the Kutcho deposit were classified in accordance with the CIM Definition Standards for Mineral Resources and Mineral Reserves (May 2014). The classification parameters are defined relative to the distance between sample data and are intended to encompass zones of reasonably continuous mineralization that exhibit the desired degree of confidence. These parameters are based on visual observations and statistical studies. Due to the polymetallic nature of the Kutcho deposits, classification parameters are based primarily on the nature of the distribution of copper-equivalent grades, and it was assumed that the deposit would be mined through a combination of open pit and underground extraction methods at a production rate of 4,500 tpd.

The following criteria were used to define the Mineral Resource categories:

- **Measured Mineral Resources**. Mineral resources in the Measured category require delineation with drillholes spaced on a nominal 25 m pattern.
- **Indicated Mineral Resources**. Mineral resources in the Indicated category include areas where grades are estimated using a minimum of three drillholes that are spaced at a maximum distance of 50 m.
- **Inferred Mineral Resources**. Mineral resources in the Inferred category include model blocks that do not meet the criteria for Measured or Indicated Mineral Resources but are within a maximum distance of 100 m from a drillhole.

At this stage of project evaluation, parts of the Main deposit are delineated by drilling on a consistent 25 m pattern, and, as a result, there are Mineral Resources in this deposit that are classified in the Measured category.

In the Esso deposit, there are no areas where there is a consistent distribution of drillholes on a 25 m pattern (or less), and, as a result, there are no Mineral Resources at Esso that can be included in the Measured category.

Most of the Main deposit was tested with drillholes spaced at 50 m or less, and, as a result, most of the Mineral Resources in the Main deposit are classified in the Indicated or Measured categories. Grades in the majority of the central part of the Esso deposit were estimated using a minimum of three drillholes spaced at a maximum distance of 50 m. This area meets the criteria required for Mineral Resources in the Indicated category.

Domains have been interpreted that encompass model blocks that are included in the Measured and Indicated categories. This step ensures consistency of classification across the deposits.

14.11 Mineral Resources

CIM Definition Standards for Mineral Resources and Mineral Reserves (May 2014) 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 eventual economic extraction" requirement generally implies that quantity and grade estimates meet certain economic thresholds and that Mineral Resources are reported at an appropriate cut-off grade that takes into account extraction scenarios and processing recovery.

It is assumed that the Main deposit would be mined using a combination of open pit and underground extraction methods and the Esso and Sumac deposits would be mined using only underground mining methods based on the following technical and economic parameters (US$):

- Mining (underground) $43.00/t
- Mining (open pit) $2.65/t
- Processing $20.50/t
- G&A $6.00/t
- Copper price $3.50/lb
- Zinc price $1.15/lb
- Gold price $1,600/oz
- Silver price $20.00/oz
- Copper process recovery 87.6% (Main and Sumac), 94.5% (Esso)
- Zinc process recovery 64.3% (Main and Sumac), 89.3% (Esso)
- Gold process recovery 58.0% (Main and Sumac), 66.0% (Esso)
- Silver process recovery 57.9% (Main and Sumac), 71.2% (Esso)
- Pit slope angle 48.9°.

Based on the metal prices and recoveries listed here, recoverable copper-equivalent (*CuEqR*) grades were calculated using the following formulae:

- Main and Sumac: CuEqR(%) = (Cu% x 0.876) + (Zn% x 0.241) + (Au g/t x 0.441) + (Ag g/t x 0.006)
- Esso: CuEqR(%) = (Cu% x 0.945) + (Zn% x 0.310) + (Au g/t x 0.466) + (Ag g/t x 0.006)

At the Main deposit, a resource constraining pit shell was generated using a floating cone algorithm based on recoverable copper equivalent grades that were calculated in model blocks on a whole block basis. This was restricted to model blocks in the Measured, Indicated, and Inferred categories.

Based on the parameters listed here, the projected cut-off grade for Mineral Resources considered amenable to open pit extraction methods at the Main deposit is determined to be 0.45% CuEqR. For Mineral Resources considered amenable to underground extraction methods, the cut-off grade at the Main and Sumac deposits is 1.05% CuEqR and at Esso it is 0.95% CuEqR. At these elevated cut-off thresholds, Mineral Resources at Main, Esso and Sumac form relatively continuous zones of mineralization that are considered amenable to underground mining methods. Based on these factors, the estimate of Mineral Resources is considered to exhibit reasonable prospects for eventual economic extraction.

The estimate of Mineral Resources for the Kutcho deposits are shown in Table 14-14. The distribution of Mineral Resources by class is shown in Figure 14-33 and Figure 14-34 for the Main deposit, Figure 14-35 and Figure 14-36 for the Esso deposit, and Figure 14-37 for the Sumac deposit. Figure 14-38 shows an isometric view of the distribution of Mineral Resources in all three deposits. All Mineral Resources are inclusive of Mineral Reserves.

Table 14-14: Estimate of Mineral Resources

Class	Tonnes (000)	CuEqR (%)	Cu (%)	Zn (%)	Ag (g/t)	Au (g/t)
Main Deposit (pit constrained, 0.45% CuEqR cut-off)						
Measured	7,213	2.31	1.64	2.35	24.7	0.36
Indicated	12,201	1.79	1.27	1.64	22.8	0.32
Measured + Indicated	**19,414**	**1.98**	**1.41**	**1.90**	**23.5**	**0.34**
Inferred	459	1.35	0.78	1.24	16.8	0.60
Main Deposit (below open pit, 1.05% CuEqR cut-off)						
Indicated	793	1.93	1.35	1.54	30.3	0.45
Inferred	1,717	1.87	1.19	1.90	26.1	0.49
Esso Deposit (0.95% CuEqR cut-off)						
Indicated	2,595	4.40	2.40	4.49	61.5	0.78
Inferred	1,624	2.15	1.32	1.59	35.8	0.42
Sumac Deposit (1.05% CuEqR cut-off)						
Inferred	9,086	1.49	1.06	1.53	16.2	0.16
ALL DEPOSITS COMBINED						
Measured	7,213	2.31	1.64	2.35	24.7	0.36
Indicated	15,590	2.23	1.46	2.11	29.6	0.41
Measured + Indicated	**22,802**	**2.26**	**1.52**	**2.18**	**28.1**	**0.39**
Inferred	12,886	1.62	1.10	1.58	20.0	0.25

Note: The Mineral Resource Estimate in Table 14-14 is considered to be amenable to a combination of open pit and underground extraction methods. Mineral resources are inclusive of mineral reserves. At the Main and Sumac deposits, CuEqR = (Cu% x 0.876) + (Zn% x 0.241) + (Au g/t x 0.441) + (Ag g/t x 0.006). At the Esso deposit, CuEqR = (Cu% x 0.945) + (Zn% x 0.310) + (Au g/t x 0.446) + (Ag g/t x 0.006). Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. The effective date of the estimate of Mineral Resources is 30 July 2021.

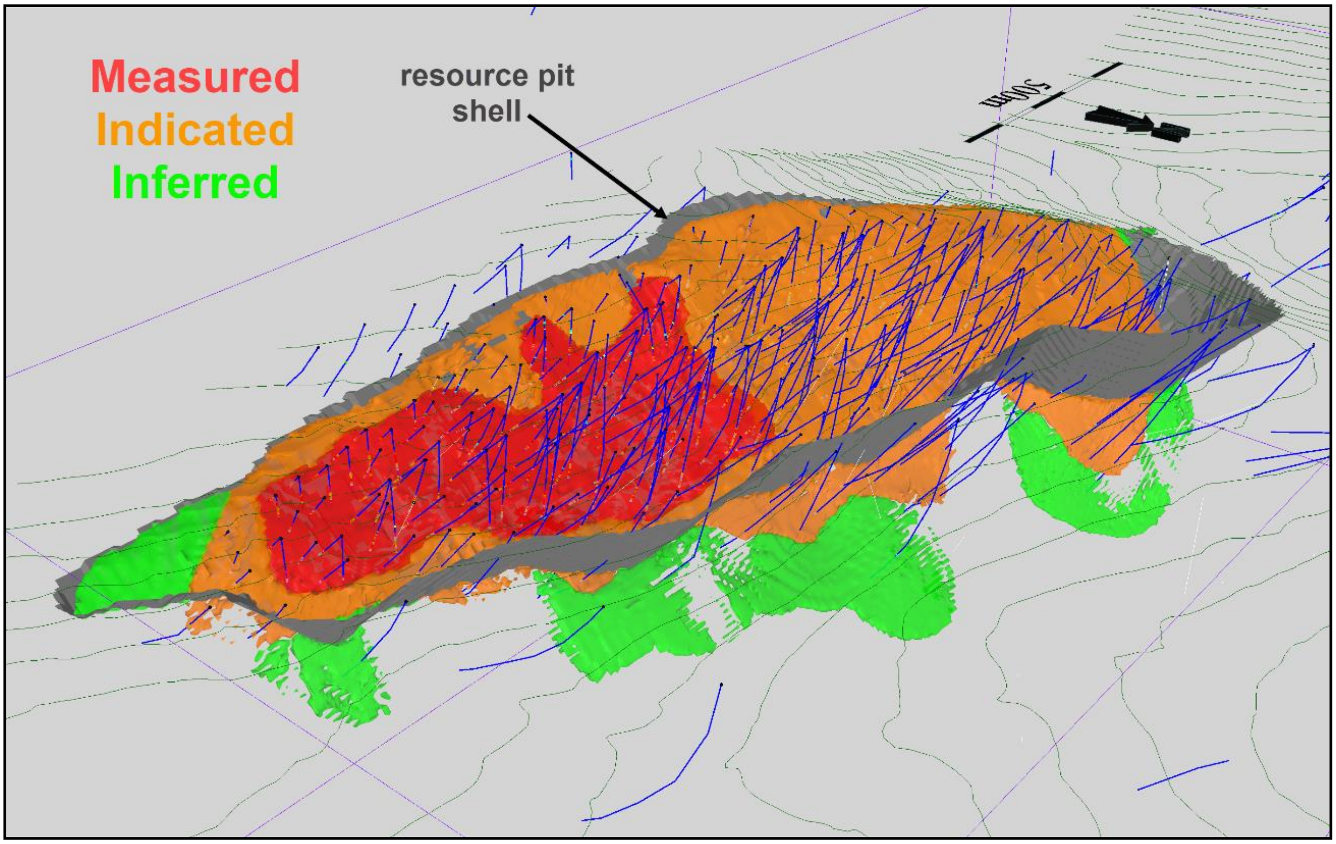


Figure 14-33: Isometric View of Base Case Mineral Resource at the Main Deposit

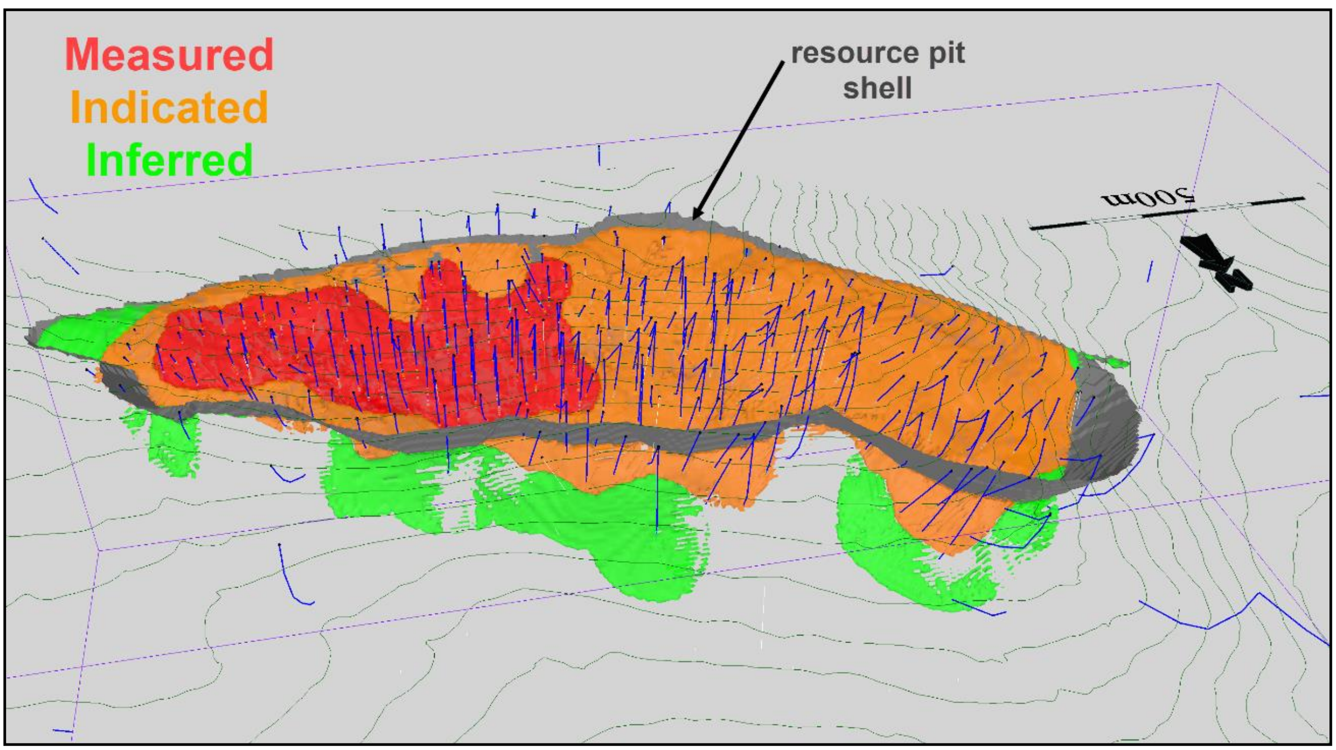


Figure 14-34: Isometric View of Base Case Mineral Resource at the Main Deposit

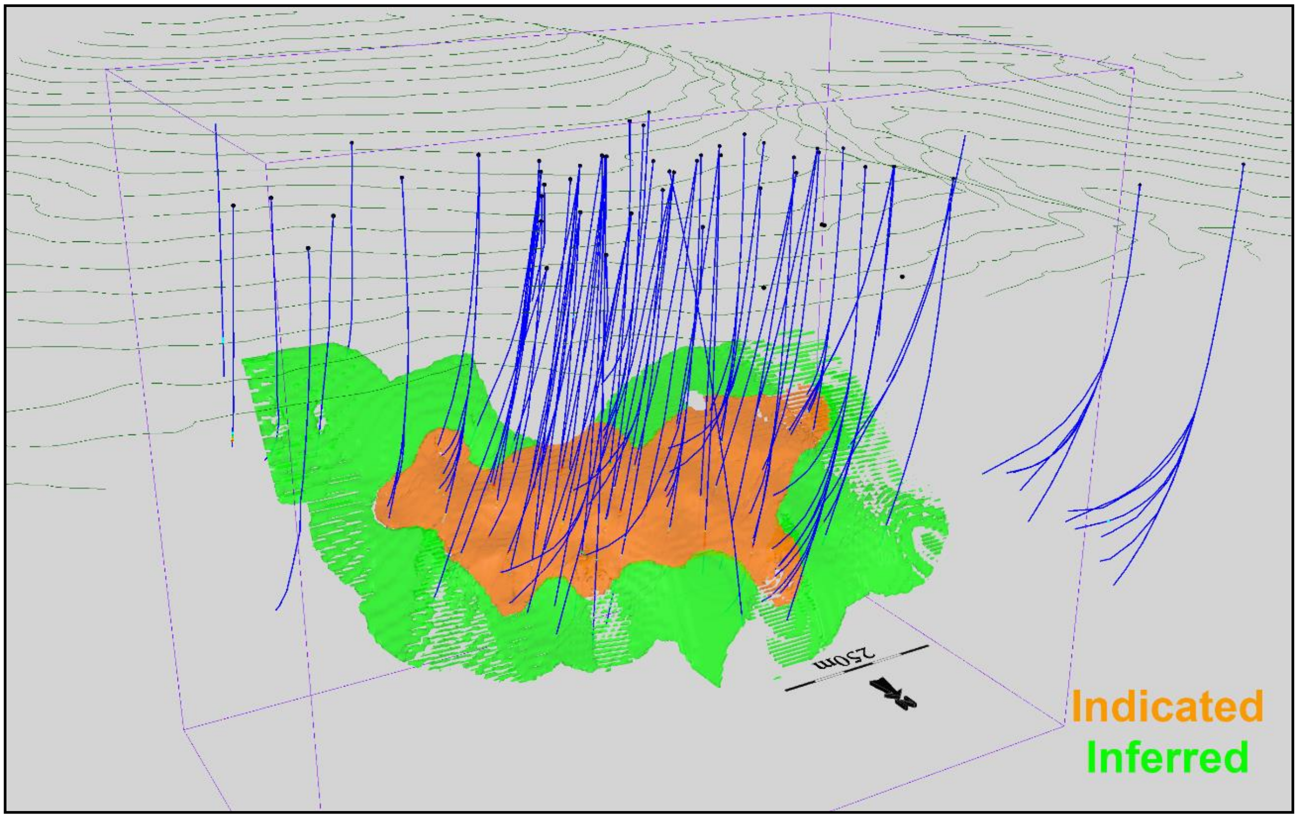


Figure 14-35: Isometric View of Base Case Mineral Resource at the Esso Deposit

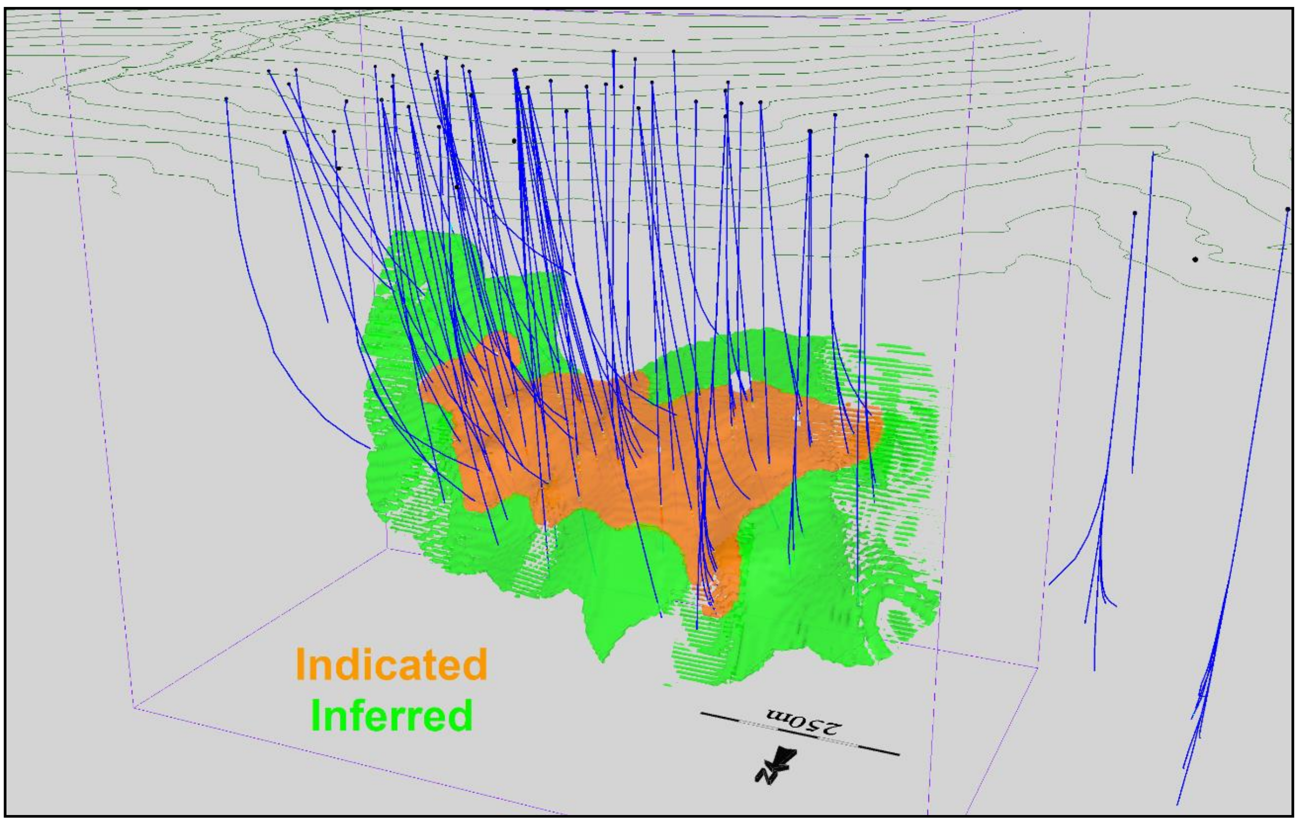


Figure 14-36: Isometric View of Base Case Mineral Resource at the Esso Deposit

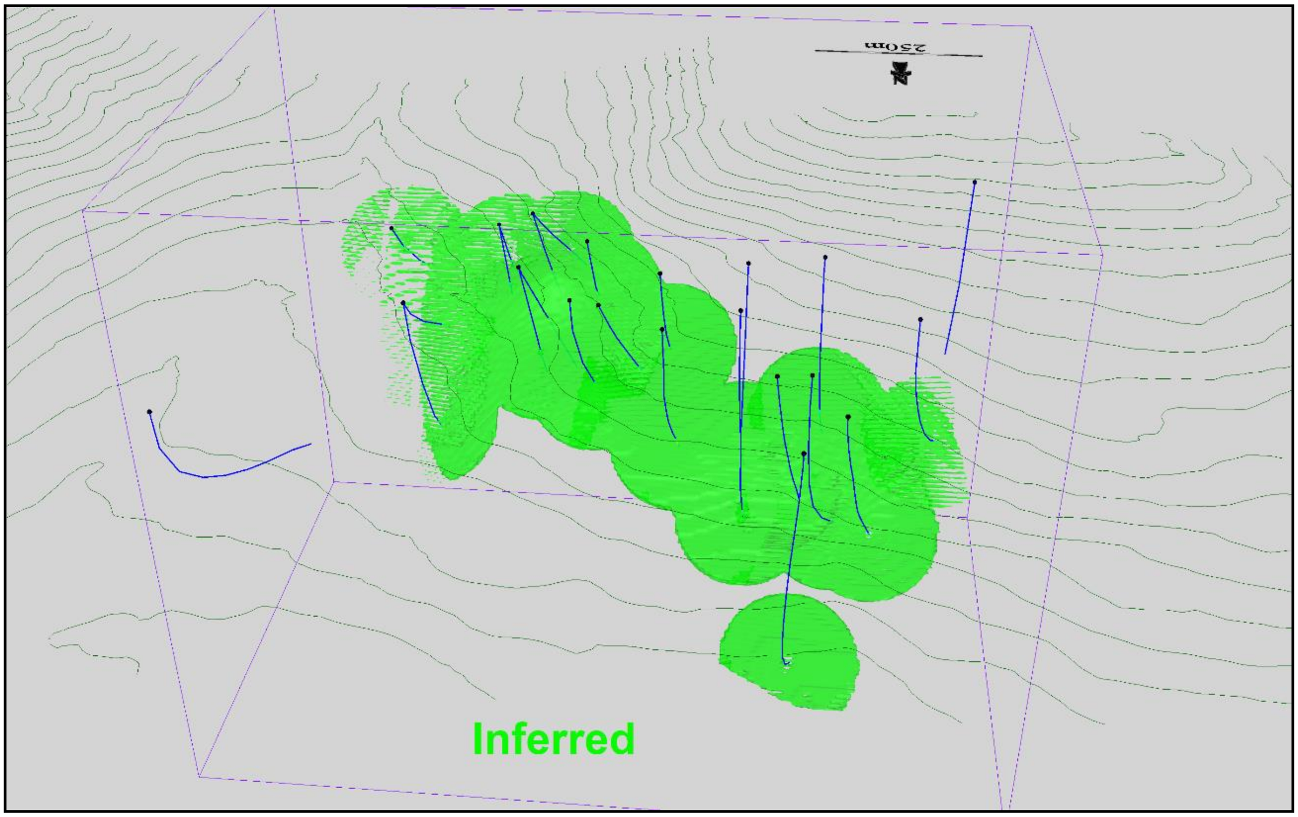


Figure 14-37: Isometric View of Base Case Mineral Resource at the Sumac Deposit

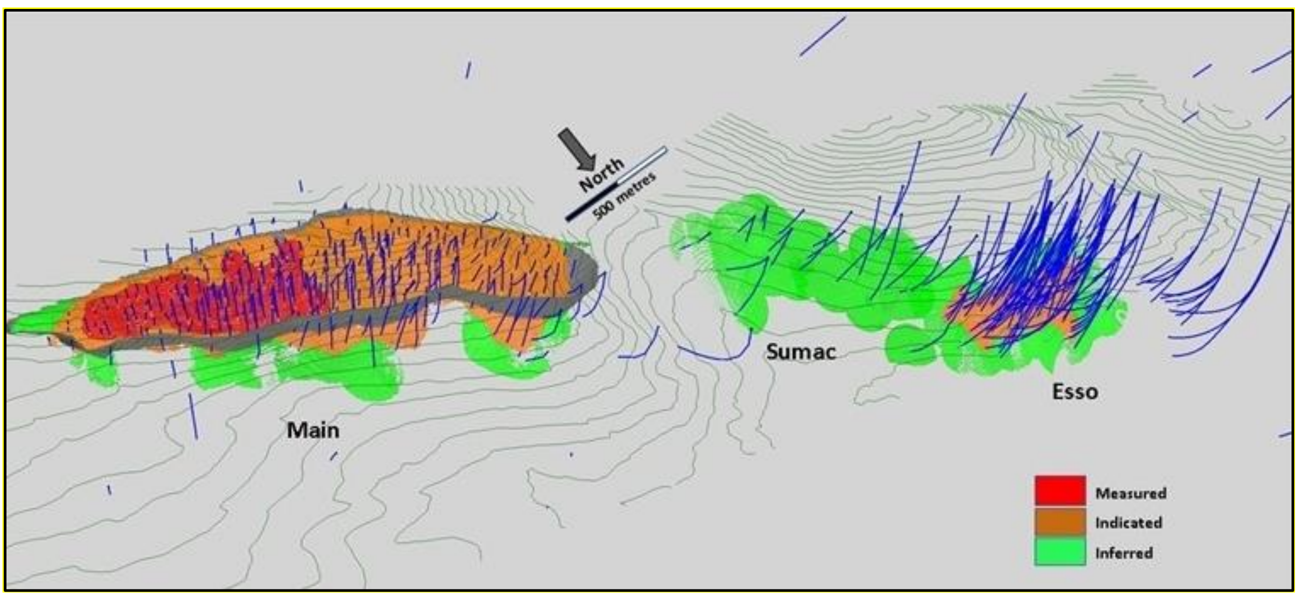


Figure 14-38: Isometric View of Base Case Mineral Resource for All Deposits

There are no known factors related to environmental, permitting, legal, title, taxation, socio-economic, marketing, political or other relevant factors which could materially affect the Mineral Resource.

Mineral Resources in the Inferred category have a lower level of confidence than that applied to Indicated Mineral Resources, and, although there is sufficient evidence to imply geological grade and continuity, these characteristics cannot be verified based on the current data. It is reasonably expected the majority of the Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.

Table 14-15 shows the estimate of Mineral Resources including the average densities (SG) and average grades for the four additional deleterious elements.

Table 14-15: Estimate of Mineral Resources with SG and penalty elements

Class	Tonnes (000)	CuEqR (%)	Cu (%)	Zn (%)	Ag (g/t)	Au (g/t)	SG	As (ppm)	Cd (ppm)	Hg (ppm)	Sb (ppm)
Main Deposit (pit constrained, 0.45CuEqR cut-off)											
Measured	7,213	2.31	1.64	2.35	24.7	0.36	3.85	239	126	5.2	10
Indicated	12,201	1.79	1.27	1.64	22.8	0.32	3.60	179	89	4.7	10
Measured + Indicated	**19,414**	**1.98**	**1.41**	**1.90**	**23.5**	**0.34**	**3.69**	**201**	**103**	**4.9**	**10**
Inferred	459	1.35	0.78	1.24	16.8	0.60	3.61	141	66	5.3	7
Main Deposit (below open pit, 1.05% CuEqR cut-off)											
Indicated	793	1.93	1.35	1.54	30.3	0.45	3.33	89	73	5.4	8
Inferred	1,717	1.87	1.19	1.90	26.1	0.49	3.25	80	64	4.2	7
Esso Deposit (0.95% CuEqR cut-off)											
Indicated	2,595	4.40	2.40	4.49	61.5	0.78	3.12	160	273	5.2	38
Inferred	1,624	2.15	1.32	1.59	35.8	0.42	3.07	162	197	3.9	35
Sumac Deposit (1.05% CuEqR cut-off)											
Inferred	9,086	1.49	1.06	1.53	16.2	0.16	3.00	405	88	5.6	12
ALL DEPOSITS COMBINED											
Measured	7,213	2.31	1.64	2.35	24.7	0.36	3.85	239	126	5.2	10
Indicated	15,590	2.23	1.46	2.11	29.6	0.41	3.51	171	119	4.8	14
Measured + Indicated	**22,802**	**2.26**	**1.52**	**2.18**	**28.1**	**0.39**	**3.61**	**193**	**121**	**4.9**	**13**
Inferred	12,886	1.62	1.10	1.58	20.0	0.25	3.06	322	98	5.2	14

Note: Mineral resources are inclusive of mineral reserves. The estimates in Table 14-15 are considered to be amenable to a combination of open pit and underground extraction methods. At the Main and Sumac deposits, CuEqR = (Cu% x 0.876) + (Zn% x 0.241) + (Au g/t x 0.441) + (Ag g/t x 0.006). At the Esso deposit, CuEqR = (Cu% x 0.945) + (Zn% x 0.310) + (Au g/t x 0.446) + (Ag g/t x 0.006). Mineral resources are not mineral reserves because the economic viability has not been demonstrated. The effective date of the estimate of Mineral Resources is 30 July 2021.

14.12 Sensitivity of Mineral Resources

The sensitivity of Mineral Resources to the cut-off grade is shown in Table 14-16 to Table 14-19. The base case cut-off grades are highlighted in the pertinent tables.

Table 14-16: Sensitivity of Mineral Resource to cut-off grade at the Main deposit (open pit)

Cut-off (CuEqR)	Tonnes ('000)	CuEqR (%)	Cu (%)	Zn (%)	Ag (g/t)	Au (g/t)
Measured and Indicated						
0.25	19,952	1.94	1.38	1.86	23.0	0.33
0.35	19,635	1.97	1.40	1.88	23.3	0.34
0.45 base case	**19,414**	**1.98**	**1.41**	**1.90**	**23.5**	**0.34**
0.55	19,197	2.00	1.42	1.92	23.7	0.34
0.65	18,895	2.02	1.44	1.94	23.9	0.34
0.75	18,508	2.05	1.46	1.97	24.2	0.35
0.85	17,988	2.09	1.48	2.01	24.5	0.36
0.95	17,371	2.13	1.51	2.05	25.0	0.36
1	17,012	2.15	1.53	2.08	25.2	0.37

Cut-off (CuEqR)	Tonnes ('000)	CuEqR (%)	Cu (%)	Zn (%)	Ag (g/t)	Au (g/t)
Inferred						
0.25	470	1.33	0.77	1.22	16.6	0.58
0.35	463	1.34	0.78	1.23	16.7	0.59
0.45 base case	**459**	**1.35**	**0.78**	**1.24**	**16.8**	**0.60**
0.55	448	1.37	0.80	1.25	16.9	0.61
0.65	438	1.39	0.81	1.27	17.1	0.62
0.75	424	1.41	0.82	1.28	17.3	0.64
0.85	398	1.45	0.84	1.33	17.7	0.65
0.95	353	1.52	0.88	1.43	18.5	0.67
1	329	1.56	0.90	1.49	19.0	0.68

Table 14-17: Sensitivity of Mineral Resource to cut-off grade at the Main deposit (underground)

Cut-off (CuEqR)	Tonnes ('000)	CuEqR (%)	Cu (%)	Zn (%)	Ag (g/t)	Au (g/t)
Measured and Indicated						
0.65	995	1.71	1.20	1.36	26.8	0.39
0.75	934	1.78	1.24	1.42	27.9	0.41
0.85	888	1.83	1.28	1.46	28.7	0.42
0.95	840	1.88	1.32	1.50	29.5	0.43
1.05 base case	**793**	**1.93**	**1.35**	**1.54**	**30.3**	**0.45**
1.15	752	1.98	1.38	1.58	30.9	0.46
1.25	705	2.03	1.42	1.63	31.7	0.47
1.35	661	2.08	1.45	1.67	32.5	0.48
1.45	608	2.14	1.49	1.73	33.2	0.50
Inferred						
0.65	2,122	1.68	1.07	1.69	23.9	0.43
0.75	2,031	1.72	1.10	1.73	24.5	0.45
0.85	1,932	1.77	1.13	1.78	25.0	0.46
0.95	1,835	1.82	1.16	1.83	25.5	0.47
1.05 base case	**1,717**	**1.87**	**1.19**	**1.90**	**26.1**	**0.49**
1.15	1,600	1.93	1.23	1.96	26.6	0.50
1.25	1,478	1.99	1.27	2.04	27.2	0.51
1.35	1,329	2.07	1.31	2.15	28.0	0.52
1.45	1,181	2.15	1.36	2.29	28.8	0.54

Table 14-18: Sensitivity of Mineral Resource to cut-off grade at the Esso deposit

Cut-off (CuEqR)	Tonnes ('000)	CuEqR (%)	Cu (%)	Zn (%)	Ag (g/t)	Au (g/t)
Indicated						
0.65	2,658	4.31	2.36	4.39	60.4	0.77
0.75	2,637	4.34	2.37	4.43	60.8	0.77
0.85	2,618	4.37	2.39	4.45	61.1	0.78
0.95 base case	**2,595**	**4.40**	**2.40**	**4.49**	**61.5**	**0.78**
1.05	2,576	4.42	2.42	4.52	61.8	0.79
1.15	2,542	4.47	2.44	4.57	62.3	0.79
1.25	2,501	4.52	2.47	4.64	63.0	0.80
1.35	2,460	4.57	2.50	4.70	63.6	0.81
1.45	2,418	4.63	2.52	4.77	64.2	0.82

Cut-off (CuEqR)	Tonnes ('000)	CuEqR (%)	Cu (%)	Zn (%)	Ag (g/t)	Au (g/t)
Inferred						
0.65	1,827	2.00	1.23	1.45	33.4	0.40
0.75	1,753	2.05	1.26	1.50	34.4	0.41
0.85	1,689	2.10	1.29	1.54	35.1	0.42
0.95 base case	**1,624**	**2.15**	**1.32**	**1.59**	**35.8**	**0.42**
1.05	1,539	2.21	1.35	1.66	36.6	0.43
1.15	1,415	2.31	1.40	1.77	37.7	0.45
1.25	1,290	2.42	1.46	1.89	38.8	0.47
1.35	1,147	2.56	1.53	2.05	40.3	0.50
1.45	1,012	2.71	1.60	2.25	41.8	0.53

Table 14-19: Sensitivity of Mineral Resource to cut-off grade at the Sumac deposit

Cut-off (CuEqR)	Tonnes (000)	CuEqR (%)	Cu (%)	Zn (%)	Ag (g/t)	Au (g/t)
Inferred						
0.65	10,026	1.43	1.01	1.46	15.3	0.15
0.75	9,892	1.44	1.02	1.48	15.5	0.15
0.85	9,718	1.45	1.03	1.49	15.7	0.15
0.95	9,460	1.47	1.04	1.51	15.9	0.16
1.05 base case	**9,086**	**1.49**	**1.06**	**1.53**	**16.2**	**0.16**
1.15	8,500	1.51	1.08	1.56	16.6	0.16
1.25	7,447	1.56	1.11	1.61	17.3	0.17
1.35	6,106	1.61	1.16	1.67	18.1	0.18
1.45	4,471	1.69	1.22	1.78	19.0	0.19

14.13 Comparison with the Previous Estimate of Mineral Resources

The previous estimate of Mineral Resources, generated by QP Robert Sim, had an effective date of 22 February 2019 and was presented in a Kutcho press release dated 4 March 2019. The previous estimate is presented in Table 14-20.

Table 14-20: Previous estimate of Mineral Resources for the Kutcho Project (effective 22 February 2019)

Class	Tonnes ('000)	CuEqR (%)	Cu (%)	Zn (%)	Ag (g/t)	Au (g/t)
Main Deposit						
Measured	5,831	2.66	1.92	2.78	28.7	0.48
Indicated	9,003	2.20	1.62	2.13	29.2	0.40
Measured + Indicated	**14,834**	**2.38**	**1.74**	**2.38**	**29.0**	**0.43**
Inferred	1,902	1.98	1.31	2.16	29.7	0.48
Esso Deposit						
Indicated	2,425	3.98	2.52	4.76	64.0	0.81
Inferred	1,025	2.30	1.60	2.23	41.4	0.52
Sumac Deposit						
Inferred	7,779	1.52	1.10	1.60	16.9	0.17
ALL DEPOSITS COMBINED						
Measured	5,831	2.66	1.92	2.78	28.7	0.48
Indicated	11,428	2.58	1.81	2.68	36.5	0.49
Measured + Indicated	**17,259**	**2.61**	**1.85**	**2.72**	**33.9**	**0.49**
Inferred	10,706	1.67	1.18	1.76	21.5	0.26

Note: The estimates in Table 14-20 are considered to be amenable to underground extraction methods and are inclusive of mineral reserves. The base case cut-off grade is 1.2% CuEqR based on the formula CuEqR = (Cu% x 0.825) + (Zn% x 0.302) + (Au g/t x 0.262) + (Ag g/t x 0.004).

Compared to the previous estimates of Mineral Resources, the current Mineral Resource shows increases in all three deposit areas. The significant increase in the Main deposit is primarily the result of a change from only underground extraction mining in 2019, to a combination of open pit and underground mining methods. Increases in resources at the Esso and Sumac deposits (and to some degree at the Main deposit) are attributed to changes in metal prices, operating costs and process recoveries, as listed below. It should be noted that there was no new drilling, other than for geotechnical or hydrogeological purposes, conducted at any of the Kutcho deposits since the previous estimate of Mineral Resource in February 2019.

Table 14-21: Comparison of operating parameters in the current and previous resource estimates

Operating parameter	Effective Date of Resources	
	30 July 2021	22 February 2019
Mining (underground)	$43.00/t	$34.00/t
Processing	$20.50/t	$18.00/t
G&A	$6.00/t	$10/t
Copper price	$3.50/lb	$3.00/lb
Zinc price	$1.15/lb	$1.25/lb
Gold price	$1,600/oz	$1,350/oz
Silver price	$20.00/oz	$17.00/oz
Copper process recovery	87.6% (M&S), 94.5% (Esso)	82.5%
Zinc process recovery	64.3% (M&S), 89.3% (Esso)	72.5%
Gold process recovery	58.0% (M&S), 66.0% (Esso)	40%
Silver process recovery	57.9% (M&S), 71.2% (Esso)	45%

15 Mineral Reserve Estimates

15.1 Introduction

The Mineral Reserve Estimate was disclosed in accordance with NI 43-101 and Canadian Institute of Mining, Metallurgy and Petroleum ("CIM") standards and guidelines. All technical works have been undertaken using the May 2014 CIM Definition Standards for Mineral Resources and Mineral Reserves and latest CIM Best Practice guidelines.

The Mineral Reserves are part of the Mineral Resources as stated in Section 14. The CIM Definition Standards 2014 define a Mineral Reserve as:

"A Mineral Reserve is the economically mineable part of a Measured and/or Indicated Mineral Resource. It includes diluting materials and allowances for losses, which may occur when the material is mined or extracted and is defined by studies at Pre-Feasibility or Feasibility level as appropriate that include application of Modifying Factors. Such studies demonstrate that, at the time of reporting, extraction could reasonably be justified."

The Project is constituted of the Main, Sumac and Esso deposits. The Main deposit crops out at surface and has in the past been considered as mineable by open pit only or open pit and underground mining methods. Recent work for this Feasibility Study examined the scale of operation (mining rate and plant throughput) as well as the balance between underground and open pit extraction of the Main deposit in order to maximize project value. As a result of this analysis the Mineral Reserve statement in this report considers the Main deposit to be mined mostly by open pit methods. The Sumac deposit is an inferred Mineral Resource and its economic viability is yet to be proven therefore it is not included in this Mineral Reserve Estimate. The top of mineralization of the Esso deposit is at a depth of about 400 m and can thus only be mined economically by underground mining methods. It was included in all previous Mineral Reserve estimates as being mined by underground methods. The Feasibility Study mining plan requires contemporaneous mining of the Main and Esso deposits.

The mine plan calls for the Main and Esso underground mines to have a mining extraction rate of just under 900 tpd. The combined open pit and underground ore production are planned to exceed the plant crushing capacity of 4,500 tpd and the excess low-grade and floatable oxidized ore (oxide) are planned to be stockpiled for processing at the end of mine life. This strategy is a key element of maximizing net present value through earliest processing of highest-grade ore and limiting the cost of initial capital. The processing plan includes an XRT particle ore sorter that removes waste material from the primary crushed feed and creates an upgraded SAG mill feed of 3,940 tpd. As such, it was considered most appropriate that the Mineral Reserve reflects the point at which ore is fed to the primary crusher as the reference.

15.2 Evaluation History

Mineral Reserves have been stated in previous Prefeasibility Studies for the Project and have considered various combinations of open pit and underground extraction. Proposed processing rates in those previous pre-feasibility studies varied between 2,700 tpd for the mostly underground cases and 5,700 tpd for the open pit dominated analysis.

For this Feasibility Study, a range of studies were undertaken to clarify the best path forward for mine extraction under current economic conditions. These 2021 studies that support this Feasibility Study include:

- Preliminary estimation that would extract the ore using only underground mining methods for both Main and Esso. This case was considered as a minimum surface disturbance option. For this preliminary study, underground stope optimization was completed, along with mine cost estimation with verification against peer operations. Capital estimation was completed by scaling previous estimates using the six-tenths rule. A simple scale-of-operation calculation using grade-tonnage curves of the Mineral Resource and scaled costs indicated that a mining rate of 3,500 tpd would maximize net present value. However, it was considered that the practicalities of the mining method would potentially limit the average mining rate to closer to 3,000 tpd. In addition, the indicated net present value for the 3,500 tpd case was not compellingly higher than the 3,000 tpd case and so the lower rate was adopted for further analysis. The results of this work were discussed informally with Kutcho.

- The geotechnical performance of accessing the Main and Esso stopes from both the hangingwall and footwall was examined. Footwall access was determined to be a geotechnically sounder and safer means of stope access. Development in the footwall of the orebody does generate more PAG waste that requires temporary surface storage before either retuning the material underground or into the open pit for closure.
- The economic cost of extracting the underground ore by transverse longhole open stoping (TLHOS), longitudinal longhole open stoping (LLHOS) and mechanized cut and fill (for thinner areas) was examined. LLHOS was concluded as being economically superior for the 3,000 tpd mining rate. This was reported in an internal CSA email dated 31 January 2021 and results disclosed to Kutcho). Cut and fill was locally better for from a marginal cost perspective in some thinner locations in the Esso deposit. However, the complexity of multiple mining methods outweighed the small economic benefit, and it was concluded as not being operationally justified. A simple economic and geotechnical analysis was conducted for heights between level of 20 m, 25 m and 30 m for LLHOS and TLHOS methods. It was concluded that 25 m levels delivered an extractable stope at the lowest cost (although there was not much difference to 20 m for LLHOS). Figure 15-1 shows the operating differential (revenue less operating costs) for LLHOS and TLHOS showing the clear performance benefit of LLHOS over TLHOS and the highest value for stopes of 25 m level heights.
- Surface crown pillar dimensions for an underground only mine at Main were provided by Terrane Geoscience Ltd (Terrane) assuming there would be no surface open pit (initially supplied by memorandum but more fully described in report Geotechnical Feasibility Investigation, 1 December 2021, Terrane Geoscience Inc.
- A mine plan was prepared for a 3,000 tpd Longitudinal Longhole open stope mine (for both Main and Esso) as the best case for underground-only extraction of the Main and Esso deposits. The plan included cemented paste stope fill. Consequently, the surface TMF would be of limited dimension and located near to that determined in the JDS 2017 PFS (for clarity, this is described as the "Best UG Only Case" for this report section).
- A simple open pit optimization and analysis using benchmark costs and scaling of previous capital estimates indicated that a predominantly open pit operation for Main would have a maximum net present value at a throughput of approximately 4,250 tpd (throughputs up to 5,500 tpd were evaluated). This was reported internally to CSA Global and briefly discussed in the report Kutcho Copper Project Gap Analysis Report, 14 December 2020 (CSA Ref R437.2020) for commencing open pit analysis at 4,000 tpd processing with 1500-1750 underground but this balance later moved to higher open pit contribution as optimization results were obtained and costs refined.
- For the predominantly underground mining case, a trade-off analysis was completed for two locations by Ausenco, either in Rusty Creek or placed on the nearby slope but outside of water courses (Kutcho Copper Project Feasibility Study Tailings Management Facility Trade-Off Study, July 2018). The preferred location was located between Playboy and Rusty Creeks. Piteau refreshed the trade-off analysis in November 2020, again for majority underground tailings storage, concentrating on the same final location selected by Ausenco but examining other methods of storage (Memorandum: Kutcho Copper, Update to Ausenco Tailings Disposal Trade-off Study, July 2018, 4 Nov 2020).
- Based on the outcomes of the preliminary optimizations a more complex pit optimization was undertaken using benchmark operating costs and scaled capital but examining the impact of the scale of underground extraction. The optimization net present value was not sensitive to the presence of an underground mine, although some extraction via underground methods of the Main deposit had marginal but accretive benefit. The analysis included Lerche-Grossman optimization, pit design, and preliminary WRD and TMF designs. The

value derived for the combined mainly open pit and underground mine had superior economic performance over the Best UG Only Case. This was first documented in a memorandum Kutcho FS – Open Pit Work Program Discussion dated 18 April 2021, (CSA Global ref R218.2021). The same report also examined conceptually the impact a particle ore sorter may have on project economics. This was followed by a more completed study by CSA Global "Kutcho Copper Project – Open Pit Mining Scoping Study", 8 May 2021 (CSA Global Ref R229.2021). The May 2021 report spurred further analysis on several project elements:

- Given a requirement for backfill for only 2.8 Mt of ore for the Esso and the Main underground mines, the method of backfill was re-examined given the capital burden that a paste backfill plant and distribution system would pose. Terrane Geoscience determined that cemented rockfill at 2–3% cement content would be a geotechnically acceptable stope fill for the Esso and Main stopes. This was first reported in memorandum form but replaced later by an encapsulating report Geotechnical Feasibility Investigation, 1 December 2021, Terrane Geoscience Inc. This was also more suitable at lower underground mining rates. The capital cost for a cemented rock backfill operation was also more aligned to the smaller underground mining operation. The offset would be a lower recovery to accommodate the temporary rib pillars.
- A Lerche Grossman analysis was completed to find the optimal position of the external-to-pit WRD with imposed constraints such as water drainage controls and minimizing the impact to Playboy Creek. It was considered that swapping the location of the TMF and WRD would be beneficial to the Project, and this initiated a review by Piteau Associates Engineering Ltd (Piteau). A TMF location analysis identified the non-fish bearing Playboy Creek as an improved location with respect to tailings storage efficiency and nPAG waste from the open pit could be utilized to build a low cost but robust embankment. The TMF would be a thickened tailings storage facility. The results of the WRD analysis utilizing the new conceptual location of the TMF in the upper reaches of Playboy Creek were provided as a CSA Global memorandum that dealt with WRD and TMF interactions: Kutcho FS – Preliminary Waste Rock Dump Designs Draft, 2 May 2021 (CSA Global Ref. R228.2021). It was determined that geotechnical drilling of the foundation location and soil and rock testing would be required and extended area of high-quality topography data (later organized by Kutcho). Designs and test results supporting the new feasibility TMF location are included in the report Kutcho Tailings Management Facility Geotechnical Assessment (Piteau, November 2021).
- It was determined that a plan to utilize the pit for waste rock backfill is economically favourable. The western end of the pit, which contains higher value ore, can be mined early and then backfilled with waste from a second phase located to the east (later termed Phase 1 and Phase 2 for the west and east divisions, respectively). An additional objective was to return all PAG waste stockpiled externally and cap that with nPAG waste. This waste re-handle plan has multiple benefits in that it will leave a post-closure pit surface with no surface water ponding and no nPAG material in free-draining external dumps after closure. The material can be re-handled whilst the plant processes low-grade thereby assuring an operational cashflow. Backfilling of the Phase 1 pit reduced mining costs marginally and also reduced the disturbance area of the operation. Optimization of the position of the WRD and backfill design in association with the Playboy Creek TMF were completed by CSA Global and reported in Kutcho FS – Preliminary Waste Rock Dump Designs Draft, 2 May 2021 (CSA Global Ref. R228.2021). The results of the new WRD position directly influenced waste haulage costs for the final optimization series.
- Preliminary analysis using typical industry values for particle ore sorter performance indicated a potentially positive impact to project net present value (NPV). Two amenability tests were arranged and based on the results of that work, and a bulk sample was processed. The program was managed by ABH Engineering. The bulk sample results proved a positive impact to Project NPV. ABH Engineering reported their bulk ore sorter test findings and cash flow analysis in the report Kutcho Copper – Particle Sorting Phase Two Study, 21 September 2021. The ore sorter results were combined into the final optimization process.

- It was identified that geotechnical drilling of the footwall rocks of the Main deposit to support the open pit optimization and design was required. Terrane Geoscience undertook geotechnical supervision of the drilling, rock parameter testing, analysis and reporting and results are included in Geotechnical Feasibility Investigation, 1 December 2021, Terrane Geoscience Inc. The report also covered underground geotechnical support advise for the Feasibility Study, including:
    - Crown pillar dimensions
    - Stope dimensions
    - Stope support recommendations
    - Development drive support recommendations
    - ELOS and non-ELOS dilution estimation.

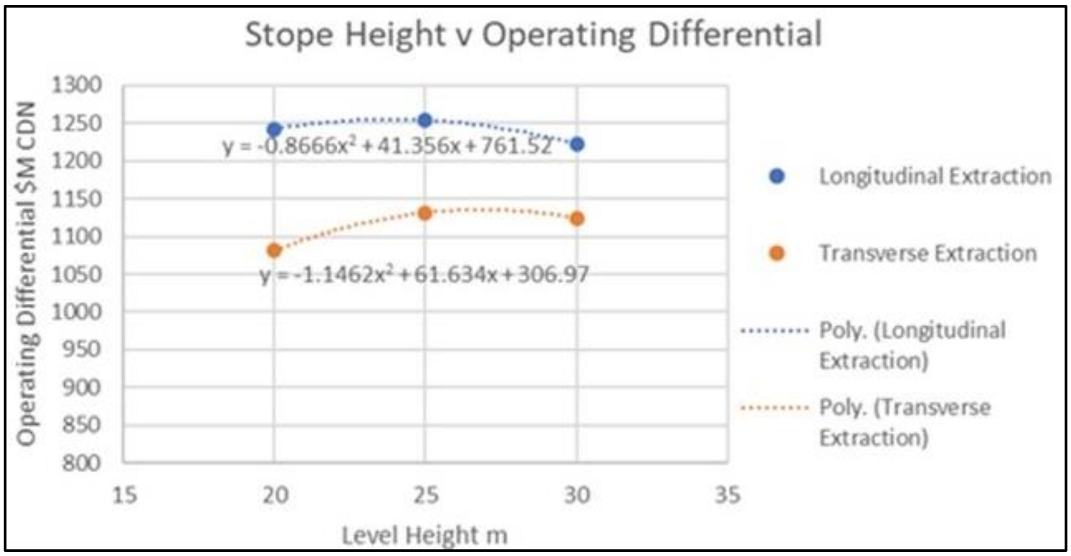


Figure 15-1: Operating Differential for Various Bench Heights for THLOS and LLHOS

The results of this formative work then became the base case mining strategy for the final round of analysis for the Feasibility Study. The mining strategy may be summarized as:

- Ore is mined mainly from the Main open pit and supplemented by high-grade underground ore from Main and Esso deposits. The open pit mining rate exceeds plant crushing capacity, allowing initially higher-grade ore to be sent to the plant and a low-grade ore stockpiled for later processing.
- The underground mine operating at a target rate of 1,000 tpd using longitudinal long-hole open stoping on 25 m levels, footwall ramp access and cemented rock fill.
- A process plant crushing 4,500 tpd, using an ore sorter to reject approximately 600 tpd of waste material.
- Tailings stored in the upper reaches of Playboy Creek catchment using a downstream-constructed embankment from open pit waste rock. Ore sorter rejects used for underground stope fill with the addition of a cement binder.
- Waste rock from the open pit placed in the TMF wall, an external waste rock dump, and in-pit backfilling. PAG waste that could not go to direct in-pit dumping to be rehandled into the pit at the end of operations. The low-grade stockpile ore processed at the end of mine life concurrent to pit backfilling with PAG and capping with nPAG material.
- Rehabilitation carried out during the operating life of the mine where possible, but mostly at the end of operations.

15.3 Open Pit Optimization

Sim Geological Inc. (SGI) provided a 5 m x 5 m x 5 m mineral resource block model for the Main deposit. The data was verified by tabulating the Mineral Resource in the original and newly developed mining block models to ensure the data was correctly received. Modifying factors, including oxidation, were applied to produce a mining block model that was then optimized to maximize cash flow using the Lerche-Grossman algorithm within the context of the base case mining strategy.

15.3.1 Block Model

Dimensions of the Main deposit model are summarized in Table 15-1.

Table 15-1: Main block model dimensions

Local coordinates	X	Y	Z
Minimum coordinates	536,500	6,451,500	1,000
Maximum coordinates	538,500	6,452,500	1,700
Block size (m)	5	5	5
Length (m)	2000	1000	700
No. of blocks in direction	400	200	140

The resource model included block volume below topography, density, copper, zinc, silver, and gold grades for mineralized and non-mineralized volumes, the percentage of mineralized volume within the block, resource category and density. Other values required for optimization and reporting would be later added as described in the following sections.

Rock-type codes were added from 3D solid models developed by Terrane Geoscience. The models guided pit optimization by defining PAG or nPAG material in association with geochemical data. Waste material defined as nPAG are assigned higher operating costs to allow for additional rehandle.

15.3.2 Oxidation

Whole blocks were considered as either partially or weakly oxidized (for simplicity termed “oxide”) or sulphide – no partial block assignment considered. All blocks were first coded as sulphide. A 3D solid model of the oxide was prepared and supplied by geologists from Equity Exploration Consultants Ltd. This solid model was used to assign oxide coding. If any block had a centroid within the oxide solid, it was coded as oxide. The QP reviewed drill core photographs to verify the model. The model correlated well with the photographs with an observable change in oxidation state in the core over most of the Project area. It is noted that thin zones of oxidation occur along fault zones that extend to a depth of 100–150 m below the oxidation boundary in places. These are not modelled, as the impact is localized and does not constitute a significant volume. Other than oxidation in fault zones, the oxidation is generally to a depth of 15 m.

Figure 15-2 displays a typical cross section of the Main Deposit with the pit shell, topography, oxidation layer and NSR of the mineralized blocks.

Oxide blocks were assigned reduced recovery and concentrate grade qualities. These are described in the NSR formula in Section 15.3.7.

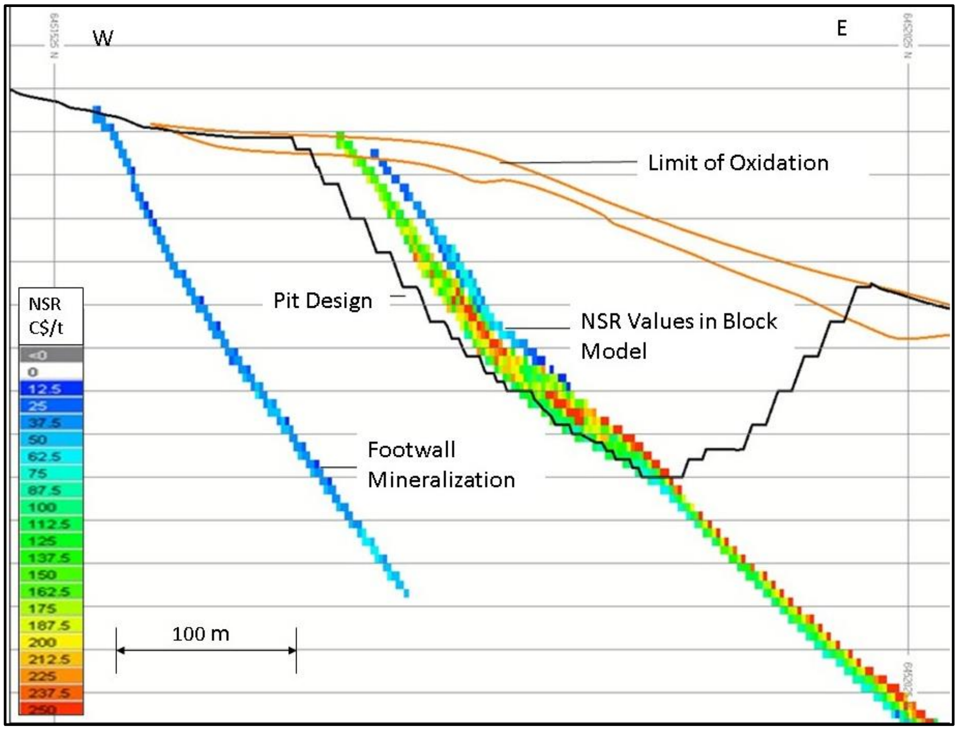


Figure 15-2: North-South Section 537750 Looking West Showing the Oxide Layer and Blocks Showing NSR C$/T

15.3.3 Dilution and Mining Loss

The Mineral Resource model blocks include a mineralization percentage field. For the Mineral Resource, each block describes a mineralized and non-mineralized volume, with associated grades. It was considered appropriate given the size of equipment considered in open pit mining and impacts of blasting that for the Mineral Reserve, the minimum recoverable mining unit would be the entire block size of 5 m x 5 m x 5 m. The mineable grade of each block is an average of mineralized and non-mineralized materials – the diluted grade. A net smelter return value (NSR) was estimated for each block based on the diluted grade. Average dilution is estimated as 1.17 Mt (8.1%) tonnes at 0.12% Cu, 0.13% Zn, 1.0 g/t Ag and 0.08 g/t Au. The dilution applied by this method was considered appropriate for the mining method and mineralization style.

The mining fleet would comprise a 4 m^3 bucket excavator and 10 m^3 loaders for ore mining in a 5 m bench configuration (both for blasting and mining of ore locations). These bucket sizes would allow both for selective mining where geology or grade variation is high or for high productivity where mineralization is more consistent. As well as give capacity to blast and mine waste on 10 m benches. No upside in ability to separate the mineralization at a finer scale than the 5 x 5 x 5 m SMU indicates using geological supervision was allowed although some mineralization boundaries have distinct visual attributes and potentially geological supervision may aid ore-waste definition. The ore sorter though may also be another means to gain additional efficiency at removing edge dilution. More test work would be required to determine if the ore sorter efficiency alters depending on the material local source. For the feasibility a single bulk test was available, so no further insight could be gained.

Similarly, through the application of a cut-off grade some blocks that contain a very small proportion of mineralization, are regarded as a mining loss. Mining loss to this effect is estimated as 0.79 Mt (5.9%) at a grade of 0.91% Cu, 0.98% Zn, 15 g/t Ag, and 0.39 g/t Au. The method of estimation of mining loss is considered appropriate for the mineralization and mining method being applied.

15.3.4 Classification

Inferred Mineral Resources do not contribute to the financial performance of the Project and are treated in the same way as waste. Sulphide Measured Mineral Resources are classed as Proven Mineral Reserves and Sulphide Indicated Mineral Resources as Probable Mineral Reserves. In the case of floatable oxidized ore (oxide), the lack of direct test work means there is some doubt of the confidence on results and so for this category Measured Mineral Resources and Indicated Mineral Resources are classed as Probable Mineral Reserves.

15.3.5 Royalty

The optimization process assumes a 2% NSR royalty for all mineralization.

During the preparation of the Feasibility Study Kutcho Copper renegotiated the Sumac Royalty and this agreement is no longer in effect. For the economic model, lease boundaries associated to the Royal Gold Royalty were coded into the model on a block majority basis (discussed further in Section 22). The application of a global 2% NSR royalty for the optimization process is therefore conservative for the Main underground and Phase 2 of the Main open pit. Figures for understanding the areas of the Mineral Reserve that have applicable Royal Gold Royalty can be found in Section 19.

The optimization process also assumed that the Wheaton precious metals price agreement (PMPA) streaming deal would be in effect. This provides assurance that if Wheaton enforced the PMPA the pit optimization would be appropriate. If not, the optimization would be slightly conservative.

15.3.6 PAG and nPAG Material

The Project has a long history of geochemical assessment going back to the deposit's discovery in 1982. Recent consolidation of the various studies has generated a database with acid base accounting (ABA) information used to classify the different waste rock materials from an ARD perspective. The ABA data was rationalized to align with 11 lithologies defined by Terrane Geoscience (Table 15-2). The rock types are tabulated based on their stratigraphic position from the footwall moving upwards towards the hangingwall within the pit area.

Classification of geochemical categories is based on the widely accepted method of the neutralization potential to acid potential (NP:AP) ratio. Materials which display an NP:AP ratio greater than three have a very low potential for AMD generation, whilst materials with ratios below this are more likely to develop AMD.

The average NP and AP values for each lithological unit are provided. The units which are lowest stratigraphically (LLTF through to MSSL) generally have average AP values which are higher than their average NP values, indicating they are on average acid producing. Conversely, the remaining units which are higher stratigraphically (QXTF upwards) all have NP values which are significantly higher than their AP values, indicating these materials are unlikely to generate net positive acidity and collectively represent a considerable acid-buffering resource.

Classification of the waste rock material into PAG and nPAG was based on this ratio with the classifications as follows:

- NP:AP <1 – sample is PAG
- NP:AP =1-3 – sample is uncertain PAG
- NP:AP >3 – sample is nPAG.

Table 15-2 shows the number of samples from each of the 11 lithology types which received these different classifications based on their NP:AP ratio.

Table 15-2: Classification of lithologies on the basis of AP and NP

Rock code	No. of samples	% of samples	Average acid potential (kg $CaCO_3$/t)	Average neutralization potential (kg $CaCO_3$/t)
LLTF	257	43	150	90
AHTF	21	4	50	87
SMSX	27	5	376	27
CBEX	13	2	204	151
MSSL	18	3	466	58
QXTF	175	30	9	108
MDST	6	1	9	90
ARGL	11	2	14	101
GABR	59	10	6	116
VSLT	2	0	4	178
CNGL	3	1	16	129
Total	**592**	**100**	**107**	**96**

Table 15-3: Classification of lithologies on the basis of PAG and nPAG

Rock code	PAG (NP:AP<1)		Uncertain PAG (NP:AP=1-3)		nPAG (NP:AP>3)	
	No. of samples	%	No. of samples	%	No. of samples	%
LLTF	130	51%	72	28%	55	21%
AHTF	6	29%	6	29%	9	43%
SMSX	26	96%	1	4%	0	0%
CBEX	9	69%	2	15%	2	15%
MSSL	16	89%	1	6%	1	6%
QXTF	4	2%	6	3%	165	94%
MDST	0	0%	0	0%	6	100%
ARGL	0	0%	1	9%	10	91%
GABR	2	3%	4	7%	53	90%
VSLT	0	0%	0	0%	2	100%
CNGL	0	0%	0	0%	3	100%
TOTAL	**193**	**33%**	**91**	**15%**	**306**	**52%**

It was decided that the Uncertain PAG classifications would be added to the PAG classifications to create a combined classification category (PAG + Uncertain PAG). This reduces the complexity of material handling and storage management. This also delivers a conservative classification scheme which provides assurance that lithologies with the potential to generate acid are classified as PAG. The final classification scheme is shown in Table 15-4.

Table 15-4: Classification scheme used in material handling and storage management

Rock code	PAG (NP:AP<1)		Uncertain PAG (NP:AP=1-3)		Zone
	No. of samples	%	No. of samples	%	
LLTF	202	79%	55	21%	Footwall
AHTF	12	57%	9	43%	
SMSX	27	100%	2	0%	Ore
CBEX	11	85%	1	15%	
MSSL	17	94%	6	6%	

Rock code	PAG (NP:AP<1)		Uncertain PAG (NP:AP=1-3)		Zone
	No. of samples	%	No. of samples	%	
QXTF	10	6%	10	94%	Hanging wall
MDST	0	0%	6	100%	
ARGL	1	9%	10	91%	
GABR	6	10%	53	90%	
VSLT	0	0%	2	100%	
CNGL	0	0%	3	100%	

This final classification exemplifies the sharp divide (highlighted in Table 15-4 across the ore to hanging wall rock codes) between those rock types which may be considered mostly PAG (LLTF, AHTF, SMSX, CBEX, and MSSL) and those which may be considered dominantly nPAG (QXFT, MDST, ARGL, GABR, VSLT, and CNGL). This differentiation occurs along the geological stratigraphic boundary between the ore zone and the hangingwall and therefore lends itself to classification of each of these structural units as either PAG or nPAG. The LLTF and AHTF rock types are contained within the footwall of the deposit; this entire rock unit is classified as PAG. The SMSX, CBEX and MSSL units are located within the ore horizon and are also classified as PAG. The remainder of the units (QXTF, MDST, ARGL, GABR, VSLT, and CNGL) are located within the hanging wall of the deposit and are classified as nPAG.

The mine model was loaded with rock codes from geological models supplied by Terrane Geoscience and then coded as PAG or nPAG based on the majority rock code: ore and footwall zones were classified as PAG and the hangingwall rocks classified as nPAG.

*15.3.7 **Net Smelter Revenue Formula***

The calculation for the NSR formula used in the pit optimization required the development of a NSR coefficient formula that considered:

- Metal prices
- Head grade to the crusher
- Concentrate payability formula (in some cases variable depending on the grade of concentrate)
- Penalties (in the case of zinc in copper concentrate a variable formula depending on grade of the zinc in the concentrate, all others were considered invariable and based on average concentrate grades)
- Metallurgical/Flotation recovery per metal to each concentrate (these were developed from regression analyses from the BML 2021 metallurgical data)
- Concentrate grade (again based on 2021 BML test work data for which regression analyses were applied)
- Ore sorter mass pull and metal recovery efficiencies
- Concentrate shipping and transport charges
- Concentrate refining and treatment costs
- Royalties.

The metal recovery formulae and concentrate grade formulae were derived using regression analysis of results from metallurgical test work completed in 2021 by BML. The formulae were applied equally to the Main and Esso deposits. The recovery and concentrate formulae applied in derivation of the NSR formula are shown in Table 15-5.

Table 15-5: Recovery and concentrate formulae applied in derivation of the NSR formula

Item	**Formula**
Ore Sorter	
Mass pull	87.0%
Copper recovery	98.7%
Zinc recovery	99.3%
Silver recovery	96.9%
Gold recovery	98.1%
Copper Concentrate (grades are the stage head (ore sorter accept))	
Copper concentrate grade	=0.19923+0.03682*Cu%
Stage copper recovery	=IF(0.67275+0.15027*Cu%-0.017623*Cu%^{2}>0.97,0.97,0.67275+0.15027*Cu%-0.017623*Cu%^{2})
Stage zinc recovery	=IF((IF(Zn%>5.51,0.0711,0.77318-0.25586*Zn%+0.023307*Zn%^{2}))>0.2,0.2,(IF(Zn%>5.51,0.0711,0.77318-0.25586*Zn%+0.023307*Zn%^{2})))
Stage silver recovery	=IF(Aggpt>70,0.7,-0.000109*Aggpt+0.2657+0.013971*Aggpt)
Stage gold recovery	=IF(Augpt>70,0.7,-0.12718*Augpt^2+0.30645+0.22668*Augpt)
Zinc Concentrate (grades are the stage head (copper concentrate circuit tail))	
Zinc concentrate grade	=IF(Zn%>5,0.5875,0.009187*Zn%+0.53044)
Stage copper recovery	=IF(Cu%>0.32,0.1,-1.3669*Cu%+0.5494)
Stage zinc recovery	=0.95246+0.00479*Zn%
Stage silver recovery	=IF(Aggpt>20,-0.002473*20*20+0.10663*20-0.75037,-0.002473*Aggpt^2+0.10663*Aggpt-0.75037)
Stage gold recovery	=IF(Augpt>0.55,-0.3088*0.55*0.55+0.36641*0.55+0.31367,-0.3088*Augpt^2+0.36641*Augpt+0.31367)
Oxide	
Ore sorter	No impact
Zinc and copper concentrate recoveries	Multiply result by 0.90
Zinc and copper concentrate grades	Multiply result by 0.75

Note: recovery formula results are 0 to 1, 1 being 100% - in head grade 5% = 5, 5 g/t = 5.

Based on the input prices, costs and recovery formula, a spreadsheet estimator of NSR values were developed for combinations of head grades. An example calculation for a set of input head grades is shown in Table 15-6. Although precise, the spreadsheet however was too complex for conversion to direct block model NSR estimation, so a set of formulas derived from regression analysis for comparisons of head grade and NSR grade coefficients were developed.

Two sets of regression based NSR formulae were developed, one for sulphide and one for oxidized material ("oxide").

The NSR regression-based formulae are still moderately complex and are summarized for the base case open pit optimization for revenue factor of 1.0 in Table 15-6.

Table 15-6: Regression based NSR formula applied in the pit optimization revenue factor 1.0

Item	Units	Regression based NSR Formula
Assign capped coefficient grades		
Copper	% (0-100)	cu%c = cu%d, capped to 2.7%, cu%d is the diluted grade
Zinc	% (0-100)	zn%c = zn%d capped to 4.5% for sulphide and 5.0 for oxide, zn%d is the diluted grade
Silver	g/t	agpc = agpd, capped to 70 gpt, agpd is the diluted grde
Gold	g/t	aupc = aupd, capped to 0.8 gpt, aupd is the diluted grde
Estimate NSR coefficient Sulphide (Lith 20+)		
Copper NSR coefficient	US$/%	nsccu=-0.5246*cu%c*cu%c+8.2045*cu%c+44.698
Low grade Zinc NSR coefficient cross metal modification	US$/%	nscz2=(0.5231-cu%d)*zn%d+(9.6532+0.1967*cu%d)
Mid grade Zinc NSR coefficient	US$/%	nscz1 = 0.0985*zn%d*zn%d+0.8542*zn%d+4.3435
Zinc NSR coefficient	US$/%	nsczn = nscz1 when nscz1 > nsczn2, otherwise nsczn = nsczn2
Silver NSR coefficient, low zinc <=3%	US$ / g/t	nscag=(agpc*agpc*agpc*7/10000/10000-311/1000/10000*agpc*agpc+0.003363*agpc+0.0583)*(1.0+(3.0-zn%d)/3.0*0.12)
Silver NSR coefficient, high zinc >3%	US$ / g/t	nscag=(agpc*agpc*agpc*7/10000/10000-311/1000/10000*agpc*agpc+0.003363*agpc+0.0583)
Gold NSR coefficient	US$ / g/t	nscau= -2.3056*aupc*aupc+4.6005*aupc+6.9847
Estimate NSR coefficient Oxide (Lith 1-20)		
Copper NSR coefficient	US$/%	nsccu=-0.2318*cu%c*cu%c+6.3417*cu%c+38.874
Zinc NSR coefficient cross metal modelification	US$/%	nscz2=(0.5231-cu%d)*zn%d+(9.6532+0.1967*cu%d)
Zinc NSR coefficient	US$/%	nsczn=-0.6134*zn%d*zn%d+6.1556*zn%d-1.9099
Silver NSR coefficient, low zinc <=3%	US$ / g/t	nscag=agpc*agpc*agpc*14/10000/10000-4274/10000/10000*agpc*agpc+0.003953*agpc+0.0637
Silver NSR coefficient, high zinc >3%	US$ / g/t	nscag=(agpc*agpc*agpc*7/10000/10000-311/1000/10000*agpc*agpc+0.003363*agpc+0.0583)*14/32
Gold NSR coefficient	US$/gpt	nscau=(-3.014*aupc*aupc+5.864*aupc+7.3364)/3*7.2
Estimate NSR per metal, multiply coefficients by diluted grades		
NSR contribution from copper	US$/t	nsrcu=nsccu*cu%
NSR contribution from zinc	US$/t	nsrzn=nsczn*zn%
NSR contribution from silver	US$/t	nsrag=nscag*aggpt
NSR contribution from gold	US$/t	nsrau=nscau*augpt
Estimate NSR	**C$/t**	**NSR = (NSRCu+NSRZn+NSRAg+NSRAu)/0.76**

Note that the NSR formulae applied in the optimization are slightly different for some parameters than the final economic model. The main difference being that the optimization process assumed that the Wheaton PMPA would be applied. This is considered not material to the outcome. The optimization NSR estimate is approximately 5% lower than the NSR calculations for the Project base case economic analysis.

The NSR formulae are complex in their derivation and application. Verification of the formulae and results have been undertaken for a selection of blocks as well as in the reserve estimation. The formula may, however, be reliably represented by simpler linear functions when grades are near to the Mineral Reserve average. The simpler NSR formulae that match the base case economic analysis (within 1% of total) presented in Section 22 are:

$$\text{Oxide NSR (US\$/t)} = (50.26 \text{ x } Cu\%) + (7.09 \text{ x } Zn\%) + (0.14 \text{ x } Ag\_g/t) + (8.94 \text{ x } Au\_g/t)$$

$$\text{Sulphide NSR (US\$/t)} = (57.82 \text{ x } Cu\%) + (9.94 \text{ x } Zn\%) + (0.34 \text{ x } Ag\_g/t) + (22.52 \text{ x } Au\_g/t)$$

Note: For copper and zinc grades 1% = 1, silver and gold 1 g/t = 1.

Table 15-7: Detailed results for concentrate costs and NSR coefficients for an example feed grade of 1.75% Cu, 2.6% Zn, 31.3 g/t Ag, and 0.46 g/t Au

	Units	Ore Sorter Feed					Copper Concentrator					Zinc Concentrator					TOTAL				
		Copper	Zinc	Silver	Gold	Total	Copper	Zinc	Silver	Gold	Total	Copper	Zinc	Silver	Gold	Total	Copper	Zinc	Silver	Gold	All
Stage head grade	% or g/t	1.75	2.60	31.30	0.46		1.99	2.98	35.46	0.51		0.21	2.51	14.31	0.34		1.75	2.60	31.30	0.46	
Stage recovery	%	1.3%	0.7%	1.9%	3.1%		90.2%	21.7%	62.4%	38.9%		26.4%	96.4%	26.9%	40.2%						
Recovery to head	%	1.3%	0.7%	1.9%	3.1%		89.1%	21.6%	61.2%	37.7%		2.5%	75.0%	9.9%	23.8%		91.6%	96.6%	71.1%	61.5%	
Concentrate	t/t					0.134					0.059056					0.035230					0.094285
Concentrate grade (reject of ore sorter)	% or g/t	0.16	0.13	4.51	0.11		26.36	9.50	324.35	2.93		1.26	55.35	88.11	3.10						
Tail grade (accept of ore sorter)	% or g/t	1.99	2.98	35.46	0.51		0.21	2.51	14.31	0.34		0.16	0.09	10.94	0.21		0.16	0.10	9.98	0.20	
Concentrate after losses in handling/transit	t/t										0.059					0.035					0.094
Payable	%						96.60%	0.00%	90.00%	90.00%		0.00%	85.00%	0.00%	46.00%						
Refining charge as a % of payable	% of payable						2.43%	0.00%	2.50%	0.31%		0.00%	0.00%	0.00%	0.00%						
Payable	US$/t ore						116.23	0.00	11.06	8.00	135.29	0.00	41.91	0.00	2.58	44.49	116.23	41.91	11.06	10.58	179.78
Refining charge	US$/t ore						2.83	0.00	0.28	0.03	3.13	0.00	0.00	0.00	0.00	0.00	2.83	0.00	0.28	0.03	3.13
Treatment charge	US$/t ore						3.46	1.25	0.00	0.00	4.71	0.16	6.87	0.00	0.00	7.03	3.62	8.12	0.01	0.00	11.74
Shipping and marketing	US$/t ore						5.93	2.14	0.01	0.00	8.08	0.11	4.71	0.00	0.00	4.82	6.04	6.85	0.01	0.00	12.89
Penalty charges	US$/t ore						0.09	0.57	0.00	0.00	0.66	0.00	0.00	0.00	0.00	0.00	0.09	0.57	0.00	0.00	0.66
Gross royalty	US$/t ore						0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
TC + RC + Shipping + Gross royalty	US$/t ore						12.31	3.96	0.29	0.03	16.58	0.26	11.58	0.00	0.00	11.85	12.57	15.54	0.29	0.03	28.43
NSR	US$/t ore						103.93	-3.96	10.77	7.98	118.71	-0.26	30.33	0.00	2.58	32.64	103.66	26.37	10.77	10.56	151.35
NSR metal taxes and royalty	US$/t ore						2.08	-0.08	0.22	0.16	2.37	-0.01	0.61	0.00	0.05	0.65	2.07	0.53	0.22	0.21	3.03
NSR inclusive of Royalties	**US$/t ore**						**101.85**	**-3.88**	**10.55**	**7.82**	**116.34**	**-0.26**	**29.72**	**0.00**	**2.53**	**31.99**	**101.59**	**25.84**	**10.55**	**10.34**	**148.33**
Payable	US$/t concentrate or US$/oz						1,973.10	0.00	187.71	135.84	2,296.64	0.00	1,192.59	0.00	73.38	1,265.97					
Refining charge	US$/t concentrate or US$/oz						47.98	0.00	4.69	0.42	53.10	0.00	0.00	0.00	0.00	0.00					
Treatment charge	US$/t concentrate or US$/oz						58.75	21.18	0.07	0.00	80.00	4.46	195.51	0.03	0.00	200.00					
Treatment charge	US$/t concentrate or US$/oz						100.68	36.30	0.12	0.00	137.10	3.06	134.02	0.02	0.00	137.10					
Penalty charge	US$/t concentrate or US$/oz						1.50	9.75	0.00	0.00	11.25	0.00	0.00	0.00	0.00	0.00					
Gross royalty	US$/t concentrate or US$/oz						0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00					
TC + RC + Shipping + Gross royalty	US$/t concentrate or US$/oz						208.91	67.23	4.89	0.43	281.45	7.51	329.53	0.05	0.00	337.10					
NSR	US$/t concentrate or US$/oz						1,764.20	-67.23	182.82	135.41	2,015.20	-7.51	863.05	-0.05	73.38	928.87					
NSR metal taxes and royalty	US$/t concentrate or US$/oz						35.28	-1.34	3.66	2.71	40.30	-0.15	17.26	0.00	1.47	18.58					
NSR inclusive of Royalties	**US$/t concentrate or US$/oz**						**1,728.91**	**-65.88**	**179.16**	**132.70**	**1,974.89**	**-7.36**	**845.79**	**-0.05**	**71.91**	**910.29**					

15.3.8 Optimization Costs

The input costs for the optimization were adapted from a preliminary optimization and economic model completed in May 2021 by CSA Global. Where available, updates to the costs were applied.

The base mining cost model used in the optimization was inclusive of drilling, blasting, loading, hauling (to the WRD), ancillary (including pumping), equipment ownership costs (leasing) and management. Additional to the base mining cost a block a waste rehandle cost was ascribed if the block were determined to be PAG waste (if PAG ore or nPAG waste this would not apply). The base mining cost also included modification based on elevation. Blocks beneath the 1565 masl were also increased in cost due to reduced productivities, increased pumping costs and increased pit perimeter costs (pre-splits and dewatering drilling) per tonne mined.

The final optimization mining cost model is shown in Figure 15-3 (the green series). Also shown on Figure 15-3 are two series that reflect the mining cost of Phase 1 and Phase 2 from the Project economic analysis. The later derived costs of the economic analysis have the benefit of additional work particularly on application of shortened lease terms, which were beneficial to Phase 2.

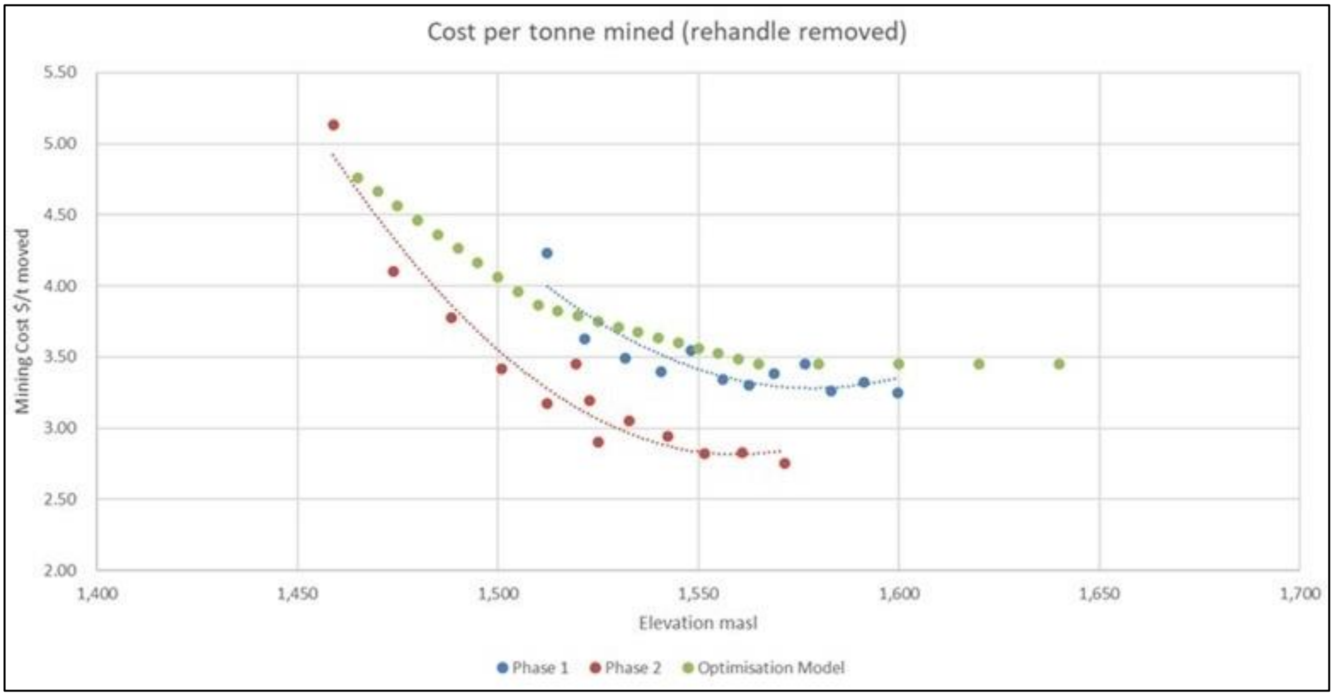


Figure 15-3: Optimization Mining Cost Model and Economic Model Results (Phase 1 and Phase 2)

It is considered though that a lower economic analysis cost result for Phase 2 costs does not reflect an opportunity to expand Phase 2. Phase 2 expansion is limited by the short strike length of quality ore (as discussed previously). It is in this area that an underground method of extraction performs favourable to open pit extraction. Lastly, the lower equipment lease cost is a product of Phase 1 mining and the certain size of the fleet and the pit. Expanding Phase 2 leads to the requirement of additional equipment and so an added ownership cost. For these reasons, it is considered that the position and size of the Phase 2 pit is appropriate.

PAG waste blocks were assigned a waste rehandle cost of C$3.66/t to allow for 100% rehandle of all PAG material with an additional 100% rehandle of nPAG material. After the optimization process was complete, mine scheduling and design of the pit backfill determined that the amount of nPAG rehandle could be reduced to 70% of the PAG quantity. Consequently, the rehandle factor applied to the optimization mining costs is moderately conservative.

Low-grade sulphide and oxide materials were assigned a cost of C$2.00/t for rehandle. An NSR value of C$85/t was used to discriminate between low-grade and high-grade sulphide ore. All oxide ore was scheduled for initial stockpiling and processing at the end of mine life. All high-grade sulphide ore is trucked directly to the crusher or the short-term, high-grade stockpile located adjacent to the crusher.

Ore processing costs were estimated on the basis of data available in May 2021 and are thus slightly different to the processing costs applied in the final economic analysis. The optimization ore processing cost is shown in Table 15-8.

Table 15-8: Optimization ore processing costs

Item	Units	Processing
Rehandle	C$/t	0.18
Processing	C$/t	27.00
Administration	C$/t	7.86
TMF sustaining	C$/t	3.32
Sustaining capital	C$/t	1.43
Total	**C$/t**	**39.79**

The processing cost applied in the final economic analysis, whilst different in each cost element, the total is very similar at C$39.86/t.

15.3.9 Geotechnical Considerations

Terrane Geoscience Inc (Terrane) was engaged by Kutcho Copper to undertake a Feasibility Study level geotechnical assessment for the Project:

- Data review of available geological and geotechnical reports
- Design of an oriented geotechnical drill program
- A geotechnical and hydrogeological site investigation
- Sample collection and laboratory testing
- Development of a 3D structural/fault geological model
- Rock mass classification and establishment of geotechnical domains
- A 3D geotechnical model
- Underground mine design
- Reporting.

Terrane (2021) designed the geotechnical field-data collection program with the aim of characterizing the rock mass and identifying structural fabrics and major structures associated with both the Main and Esso deposits. The field-data collection program consisted of detailed geotechnical logging, index strength testing, packer testing, geomechanical sample collection and optical/acoustic televiewer surveying. Two field programs were completed, one in 2018 and the other in 2021. Terrane's work adds to prior geotechnical programs conducted by other companies.

Terrane (2021) summarise the geology of the Main deposit as consisting of interbedded tuffaceous rocks in the footwall, and quartz-feldspar tuff with subordinate volcanic siltstone and argillite in the hangingwall. Massive to semi-massive sulphide lenses (which contains the copper/zinc/silver/gold mineralization) straddle the contact between the two zones. Primary stratigraphic relationships have been overprinted by the strong influence of north dipping reverse faults. Terrane considered the rock mass conditions at the Main deposit as ranging from poor (generally in the footwall) to good rock (generally in the hangingwall).

Terrane (2021) provided design recommendations for the proposed Main open pit which are summarized in Table 15-9. The recommendations are a product of the information collected and studies performed and included kinematic analysis and limit equilibrium modelling on benches and the overall pit wall slope.

Table 15-9: Summary of open pit mine recommendations – Main deposit

Design sector	Domain	Bench face angle (°) (1)	Inter-ramp angle (°) (1)	Bench height (m) (2)	Overall slope angle (°) (1)	Catch bench width (m)
1	FW	59	41	15	40.5	8.2
2	FW	59	41	15	40.5	8.2
3	FW	59	41	15	40.5	8.2
4	FW	59	41	15	43.0	8.2
5	HW	75	55	20	54.0	8.5
6	HW	75	55	20	54.0	8.5
7	HW	75	55	20	54.0	8.5
8	HW	75	55	20	53.0	8.5
9	HW	75	55	20	53.0	8.5

Notes: (1) A geotechnical berm or a ramp after a vertical height of 120 m. Overall slope angle assumed to be equal to or less than the value presented. (2) Maximum recommended bench height.

The design recommendations made by Terrane (2021) were:

- Trim blasting of the footwall
- Pre-split blasting of the hangingwall
- Slope dewatering and depressurization of the footwall using horizontal drain holes
- Routine bench face maintenance, geotechnical monitoring and ongoing data collection throughput the life of the mine.

It is considered by the QP that the mining methods and costs applied in the Feasibility Study meet these recommendations.

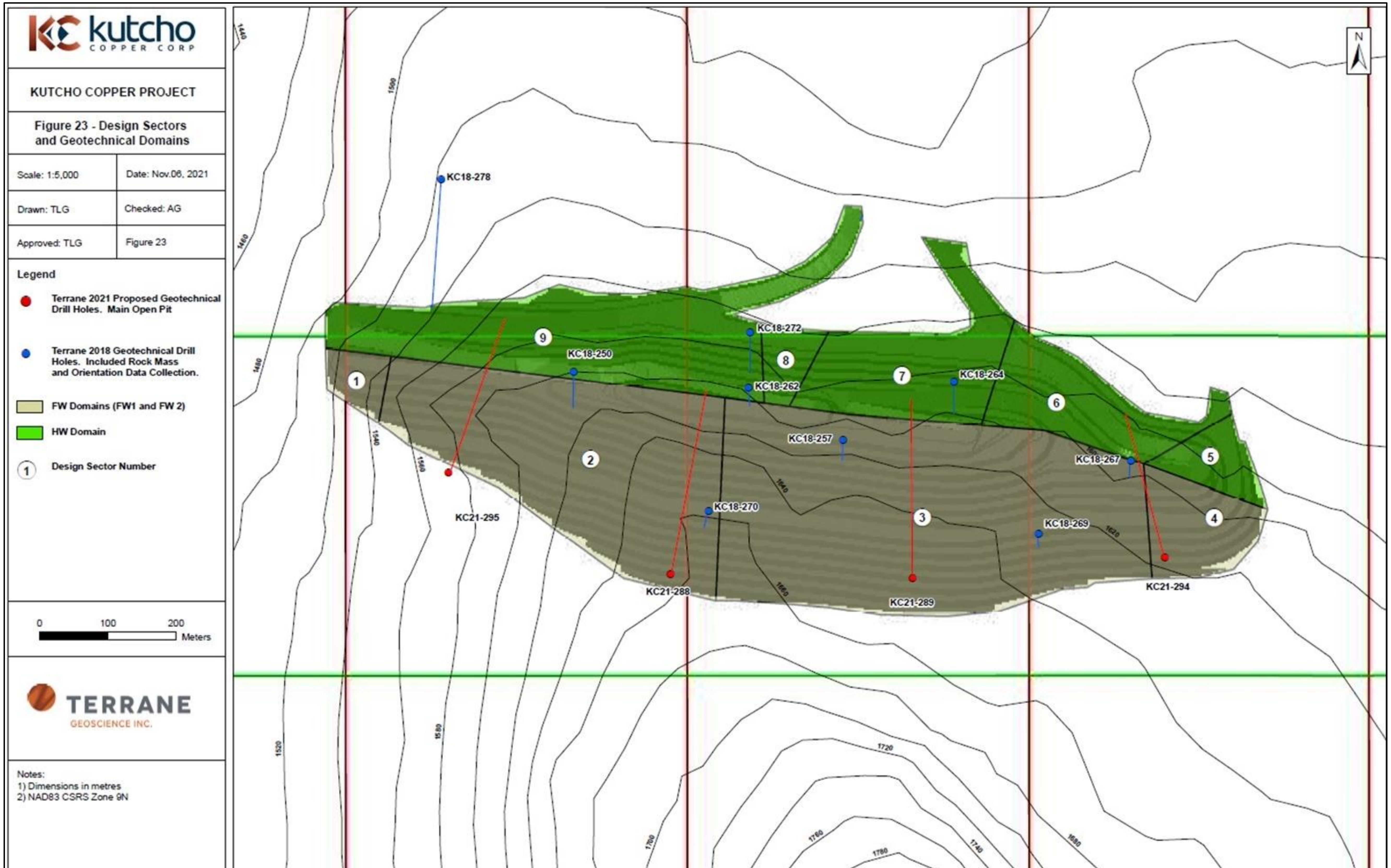


Figure 15-4: Design Sectors and Geotechnical Drillholes in the Main Deposit

The geotechnical guidance from Terrane was adapted to the optimization process to merge inter-ramp angle (IRA), catch-berm and ramp access considerations. A trial process of optimization and design was completed to refine the final optimization overall slope angles (OSAs). Ramp access design allows for two phases of mining (essentially a requirement for two sets of ramps, one for each phase). No catch berm is required on the hangingwall or north side of the pit due to the reduced height of the wall as compared to the footwall. A single catch berm is required on the footwall. The adapted optimization and design angles for optimization design are included in Table 15-10 and Figure 15-5 (note that the sectors provided for optimization are not the same numeral coding as those in the Terrane (2021) report table).

Table 15-10: Optimization design: pit wall sectors and parameters

Parameters	Unit	Sector 1	2	3	4	5	6	7	8
Easting	minimum	West Edge	537,050	537,150	537,250	537,550	537,650	537,960	Footwall
Easting	maximum	536,000	537,150	537,250	537,550	537,650	537,950	East Edge	Footwall
Bench	m	20	20	20	20	20	20	20	15
Bench face angle	°	75	75	75	75	75	75	75	59
Catch berm	m	8.5	8.5	8.5	8.5	8.5	8.5	8.5	8.2
OSA	°	28.7	39.0	45.2	47.5	54	47.7	41.9	39.6

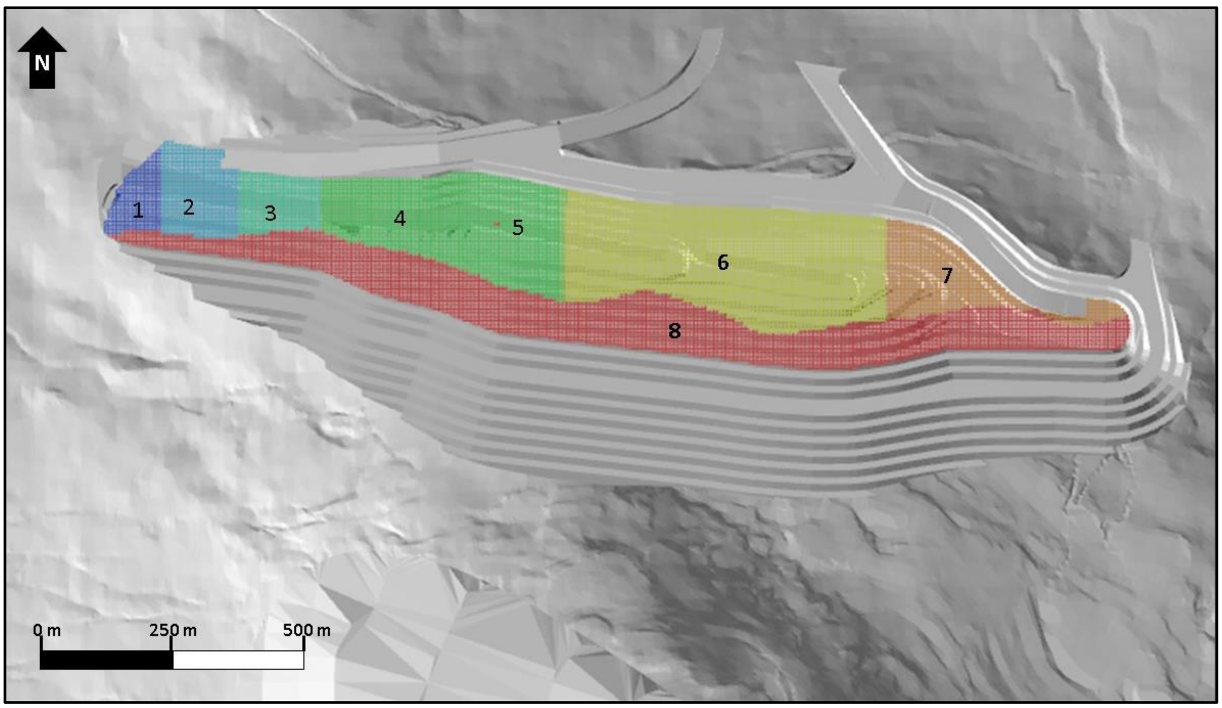


Figure 15-5: Sectors Used in Application of the Optimization OSA and Pit Design Wall Parameters

Note. Numbers refer to sectors and the associated colours are showing blocks coded for those sectors for a particular bench.

15.3.10 Optimization Process

Pit shells of the optimum NPV for various revenue factors (RF) of the base case were created using the Pseudo-flow algorithm in the Deswik software (Psuedo-flow is a means of optimization of the discounted cashflow and equivalent to the Lerches-Grossman algorithm). The NPV revenues were created at increments of 5% or 10%. A discount rate of 8% and a mining extraction limit of 11.7 Mtpa were applied to optimize the discounted cash flow. The results of the optimization process are shown in Table 15-11 and Figure 15-6. Shells at RF 0.4 and 0.8 as these re the best representations of the final pit designs for phases 1 and 2. RF 0.4 is at approximately 50% of the total design tonnage.

Table 15-11: Results of optimization

Stage	RF	Waste (kt)	Mill feed (kt)	Total movement (kt)	S.R.	Cu (%)	Zn (%)	Ag (g/t)	Au (g/t)	NSR(2) C$/t=	Cashflow(1) (C$M)
1	0.30	13,775	2,298	16,072	6.0	1.95	3.11	30.7	0.48	65	436
2	**0.40**	**36,793**	**6,300**	**43,092**	**5.8**	**1.80**	**2.67**	**27.4**	**0.41**	**77**	**885**
3	0.45	57,144	9,945	67,090	5.7	1.70	2.42	26.0	0.36	80	1,197
4	0.50	61,937	11,150	73,088	5.6	1.64	2.33	25.5	0.36	86	1,253
5	0.55	67,795	12,302	80,097	5.5	1.60	2.23	24.9	0.35	91	1,301
6	0.60	71,208	13,140	84,348	5.4	1.57	2.17	24.5	0.34	97	1,325
7	0.65	73,311	13,791	87,102	5.3	1.53	2.12	24.1	0.34	103	1,338
8	0.70	76,940	14,529	91,468	5.3	1.51	2.07	23.7	0.33	108	1,354
9	0.75	78,065	14,956	93,020	5.2	1.49	2.04	23.4	0.33	114	1,359
10	**0.80**	**81,680**	**15,612**	**97,292**	**5.2**	**1.46**	**2.00**	**23.1**	**0.32**	**120**	**1,369**
11	0.90	95,939	17,449	113,388	5.5	1.40	1.91	22.6	0.32	129	1,389
12	1.00	99,499	18,280	117,779	5.4	1.37	1.86	22.2	0.32	140	1,390
13	1.10	104,348	19,159	123,507	5.4	1.34	1.81	21.9	0.31	150	1,389
14	1.20	107,450	19,781	127,231	5.4	1.32	1.78	21.6	0.31	161	1,384
15	1.30	110,423	20,353	130,776	5.4	1.30	1.75	21.3	0.31	171	1,378
16	1.40	113,536	20,858	134,394	5.4	1.28	1.73	21.1	0.30	182	1,371
17	1.50	117,219	21,364	138,584	5.5	1.26	1.70	20.9	0.30	192	1,363
18	1.60	120,898	21,803	142,701	5.5	1.25	1.68	20.7	0.30	203	1,358

Notes: (1) Undiscounted cashflow using an RF of 1.0 to assess value within the shell. (2) NSR determined at a RF of 1.0.

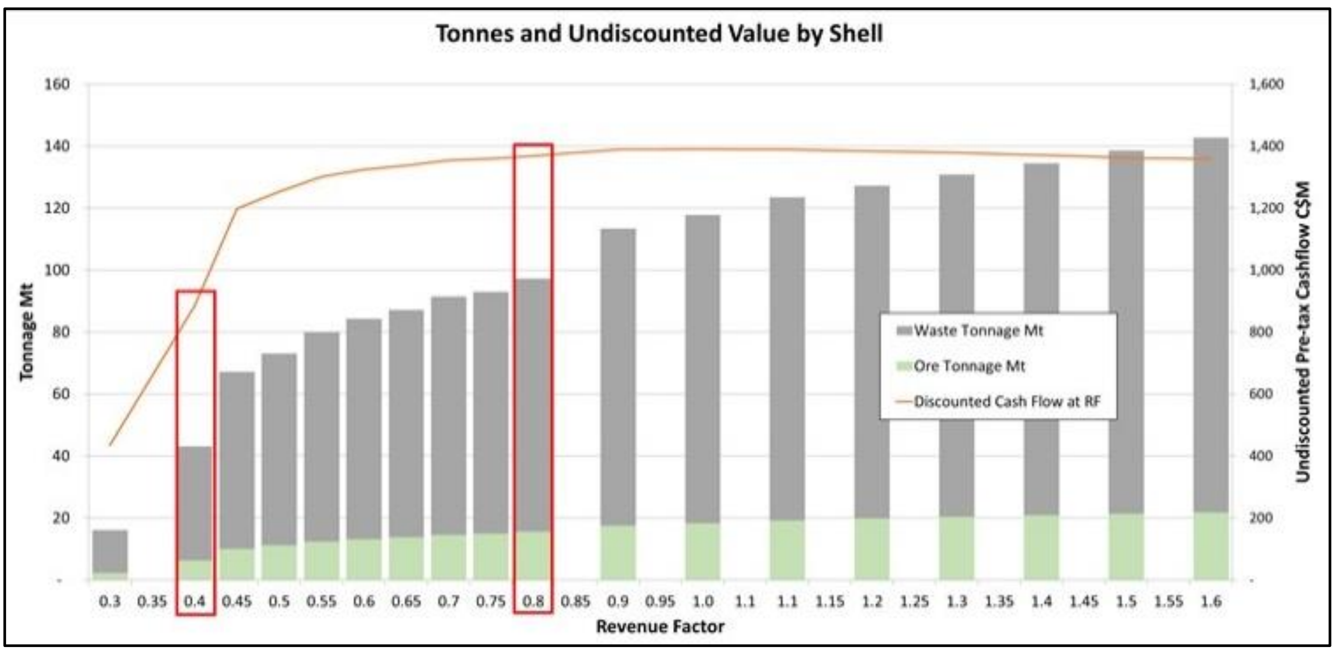


Figure 15-6: Mill Feed Tonnage (left-hand axis) and Discounted Cash Flow C$M (right-hand axis) by Shell

Shell No. 9 (revenue factor 0.75) was used as an initial guide the pit design process. Shell No. 9 is shown on Figure 15-8. The final design ended being very close to Shell 10 (revenue factor 0.80). For higher revenue factors than 0.8, there is a large increase in strip ratio as the optimization process chases a short section of mineralization at depth in the western part of the pit. Furthermore, the short strike length of this mineralization requires lower wall angles to accommodate more access ramping. Once lower angles are added back to the optimization, the optimization shrinks back to shell 10. Proving shell 10 to be a robust selection. Figure 15-7 shows the incremental strip ratio and the incremental margin for each shell (compared to its next smaller shell). The margin is calculated as undiscounted cashflow at a RF of 1.0 over costs per tonne of ore of the increment.

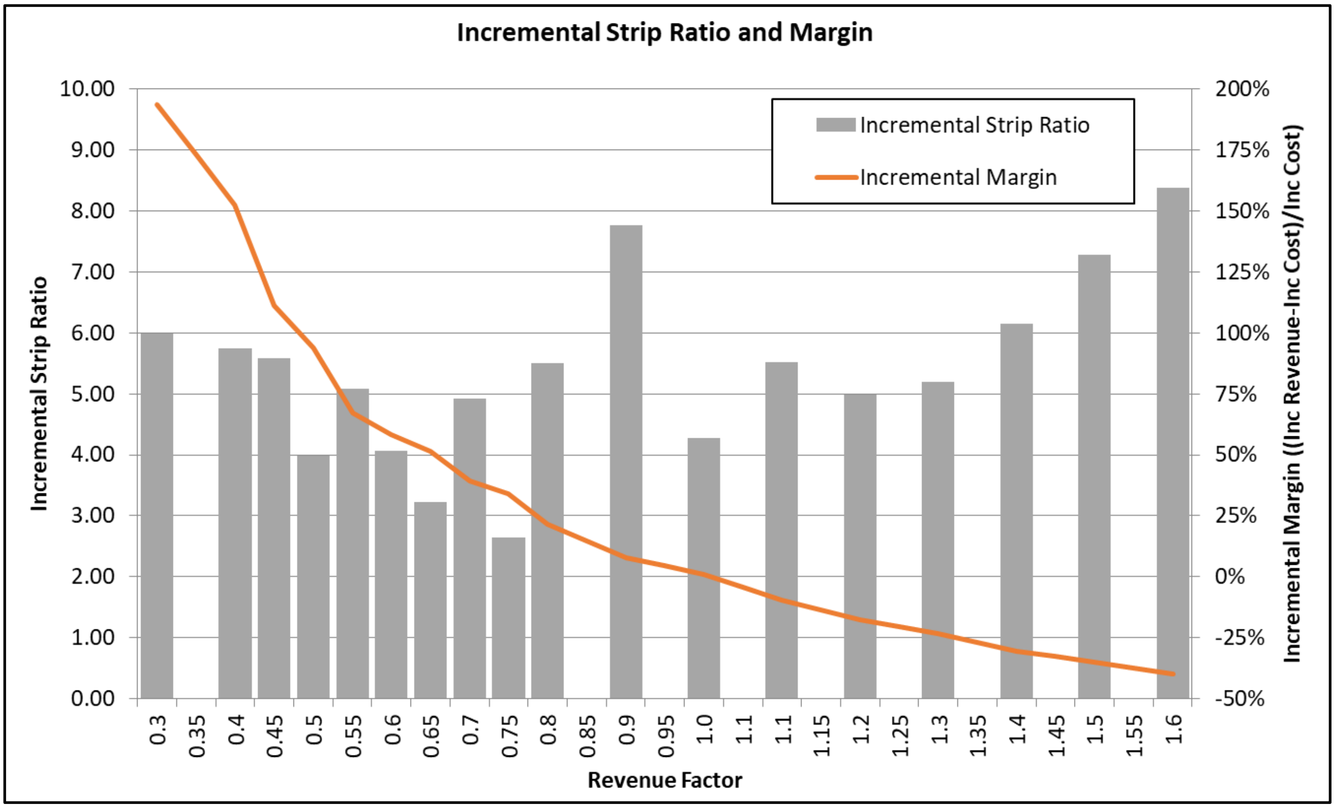


Figure 15-7: Incremental Strip Ratio and Margin

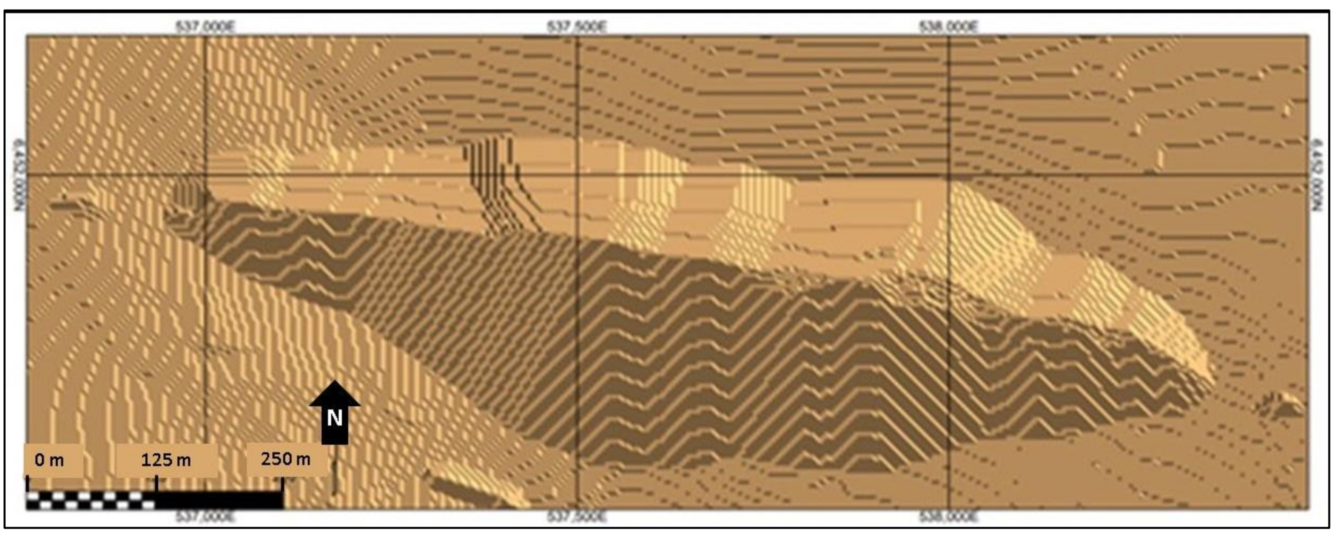


Figure 15-8: Plan View of Shell No. 9 Initially Selected for Initial Design of Phase 2

The eastern end of revenue factor 0.40 (shell no. 2 – boxed area in Figure 15-9) was used to guide the construction of the Phase 1 pit. The final design completed between shell no. 2 and shell no. 3 delivering around half of the tonnage within shell no. 8. The Phase 1 pit is the key element of the Project payback period.

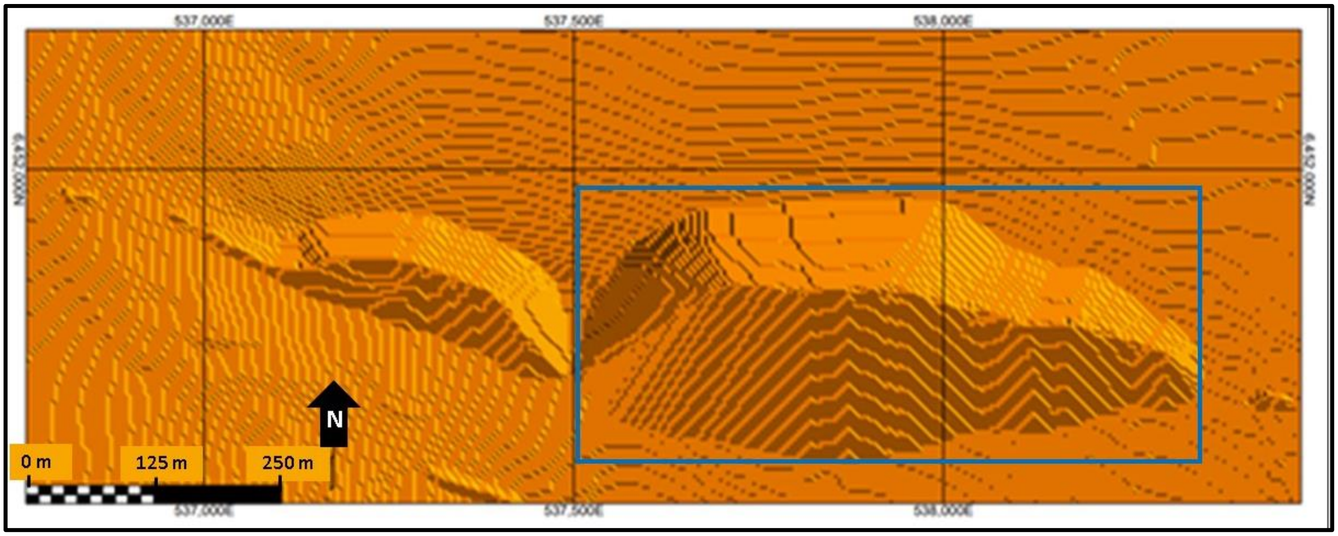


Figure 15-9: Plan View of Shell No. 2 Used for Design of Phase 1 (blue boxed portion)

15.3.11 Underground interaction

Optimizations incorporating a maximum NSR value mimicking underground extraction were performed in the May 2021 analysis undertaken by CSA Global. This analysis determined that the Main pit optimization at an RF of 0.75 (shell no. 9) or 0.80 (shell no. 10) is not sensitive to the presence of an underground mine (i.e. the optimization did not perceptibly change with or without adding the limiting cost of underground extraction to the optimization). Therefore, any underground mine beneath the Main open pit would be completed after the Main open pit mining was completed.

Figure 15-10 shows a long section of the Main deposit with NSR values shown for blocks of 20 m strike x 25 m depth based on underground extraction. The figure overlays the approximate Phase 1 and Phase 2 pit depths (blue line). Beneath the pit, boxes in red highlight permanent crown pillars. Terrane supplied advice that the Figure 15-10 crown pillar should be no thinner than 20 m for the anticipated stope spans. The designs completed for the Feasibility Study comply with these recommendations.

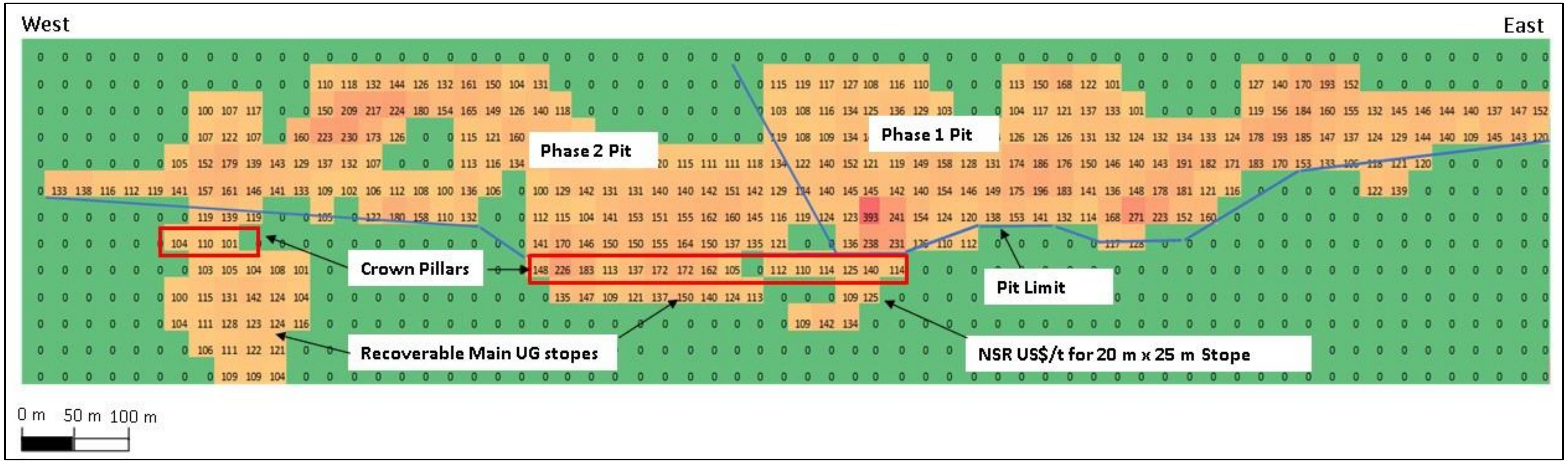


Figure 15-10: Long Section of Main Looking North Showing NSR C$/t for Underground Extraction (20 m x 25 m stope blocks)

Note The possible crown in red box and the base of the pit design (blue line).

15.3.12 Constraints

The optimization did not require the application of area limited areas. For instance, the sensitive environmental areas of the creeks and archeological site are well outside the maximum expansion of any optimization footprint. Similarly there are no ownership restrictions within the optimization area nor are there any facilities that such as the TMF or roads that require limitation of the optimization.

15.3.13 Operational Cut-off

Table 15-11 (above) outlines the Measured and Indicated Mineral Resources within the pit shells using a break-even cut-off. However, for the economic analysis a slightly higher "operational" cut-off was utilized. Consequently, there is some material between the breakeven cut-off and the operational cut-off that is marginally economic. The margin on this material is slight (less than 10% of total costs) and greater tailings capacity will be required to store this material. Without surety of storage design and the small margin, this material was not considered as a Mineral Reserve.

The breakeven cut-off accounts for the material to be processed at the end of mine life (requiring trucked rehandle). Based on the final economic analysis the break-even cut-off is C$39.84/t and is shown in Table 15-12. The operational cut-off was estimated as C$55.00/t which gives a 7.5% margin as a means to give plan robustness (revenue over costs), and this was applied in the Mineral Reserve estimation.

Table 15-12: Breakeven cut-off (applies to rehandle period Year 9 to 11)

Breakeven cut-off	C$/t
Rehandle	1.65
Processing	27.53
Administration	6.69
Sustaining capital (including TMF)	2.53
Total	**38.40**

Mineral resources between the breakeven and operational cut-offs not included in the Mineral Reserve amounts to 1.24 Mt at 0.53% Cu, 0.63% Zn, 9.6 g/t Ag, and 0.13 g/t Au. Utilizing the economic analysis base case the margin before tax and additional storage costs for this material is around C$4/t. The impact of not including this material in the mine plan is considered not material to the Project. Also, should the Wheaton PMPA be affirmed, the NSR of this material would further erode. The QP considers it appropriate not to include this marginal material in the Mineral Reserve.

15.3.14 Consideration of Inferred Mineral Resources

Inferred Mineral Resources was not used in the development of the Mineral Reserve. However, an optimization was completed at an RF of 0.75 to determine if there would be a change to the pit design this material should be converted. This evaluation was undertaken to understand the potential impact on infrastructure design (roads, TMF, WRD). The optimization determined that a few small extensions were possible, but most were of inconsequential dimension and would probably not be pursued further. One small extension to the pit design to the east of Phase 1 (Figure 15-11) might be possible if successfully converted. Only temporary roads exist in this area, and it is recommended that Kutcho Copper review their future development drilling requirements (see Section 26). There is no guarantee that Inferred Mineral Resources may be converted to Mineral Reserves.

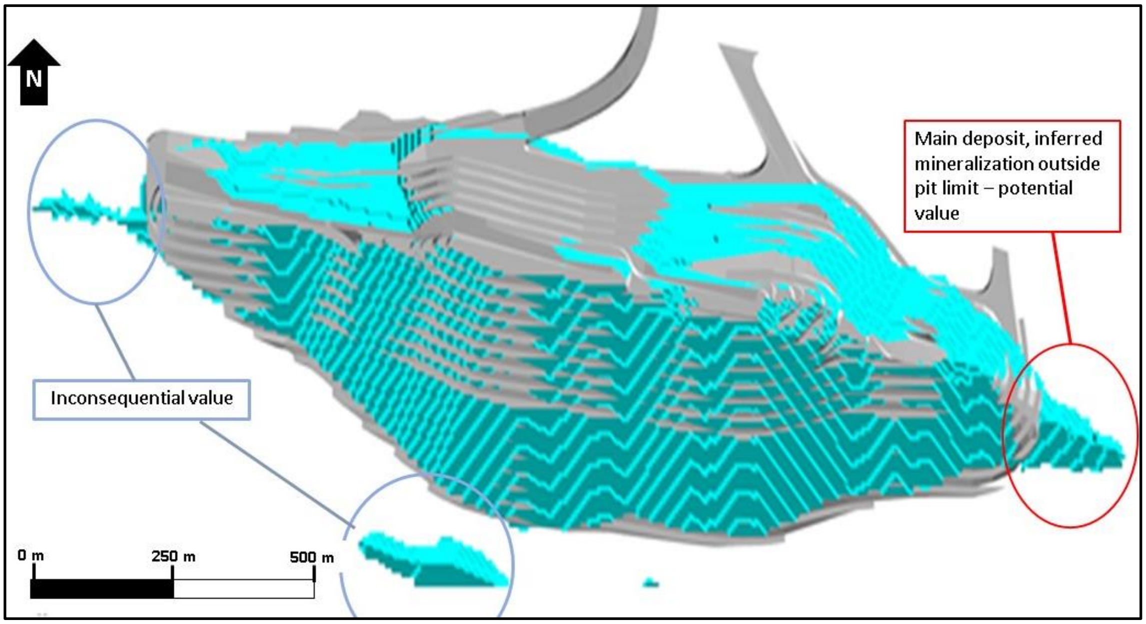


Figure 15-11: Plan View of Pit Design Overlain with Optimization Shell

The amount of Inferred Mineral Resource which is largely contained in the eastern extension is estimated as 0.17 Mt at 0.94% Cu, 1.49% Zn, 18 g/t Ag, and 0.54 g/t Au (1% of the current Mineral Reserve tonnage) at a strip ratio of approximately 2:1 (W:O) and includes both oxide and sulphide material types.

Also, there is almost no Inferred Mineral Resource within the pit design. Inferred Mineral Resource within the current pit design is estimated at 0.06 Mt with grades of 0.94% Cu, 1.49% Zn, 18 g/t Ag and 0.54 g/t Au (0.3% of the current Mineral Reserve).

The impact of Inferred Mineral Resource either within or in extension to the current pit design should they prove viable (and there is no guarantee they would be), would not be material to the Project.

15.4 Open Pit Mine Design

Open pit design guidance was developed by Terrane and pit-slope design criteria are described in Table 15-13. Ramps were designed for use with 90-tonne class, fixed-body dump trucks. Ramp gradients were designed at 10%. The last four 5 m benches were designed with single lane access ramps as high productivities are not required. High-productivity ramp switchbacks utilized a 7.8 m internal radius. The footwall catch berm was designed to allow equipment access to either end for Phase 1 and only the western end for Phase 2.

Table 15-13: Ramp width

Item	Units	Dual lane	Single lane
Truck running width	m	4.9	4.9
Angle of repose	°	35	35
Ground clearance	m	0.9	0.9
Safety berm height	m	1.8	1.8
Crest damage allowance	m	1.3	1.3
Safety berm width	m	5.1	5.1
Running width	m	17.1	7.3
Drainage channel	m	1.5	1.5
Total calculated width	m	25.0	15.3
Total allowed width	**m**	**25.0**	**15.0**

All benches were designed horizontal; a minor slope inclination would be beneficial in detailed designs to promote water drainage.

The designs followed optimization pit shells as described in the optimization section of this report with the final design closely matching pit shell 10 (0.8 RF).

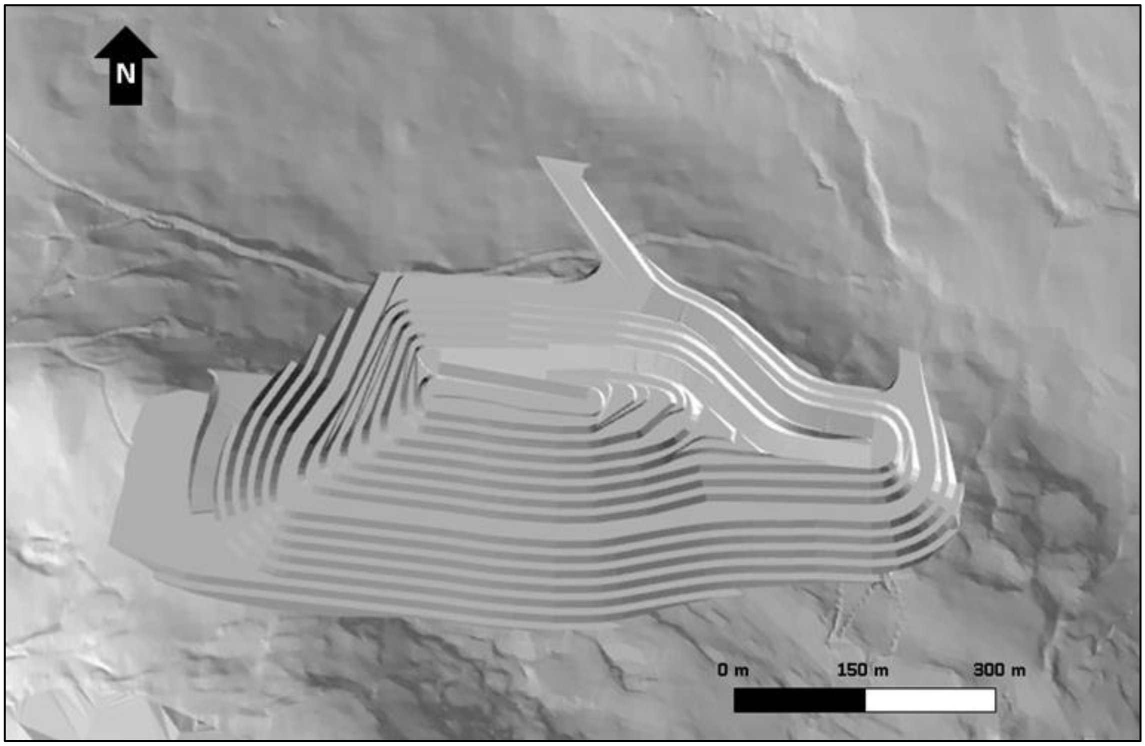


Figure 15-12: Open Pit Showing Phase 1 Only

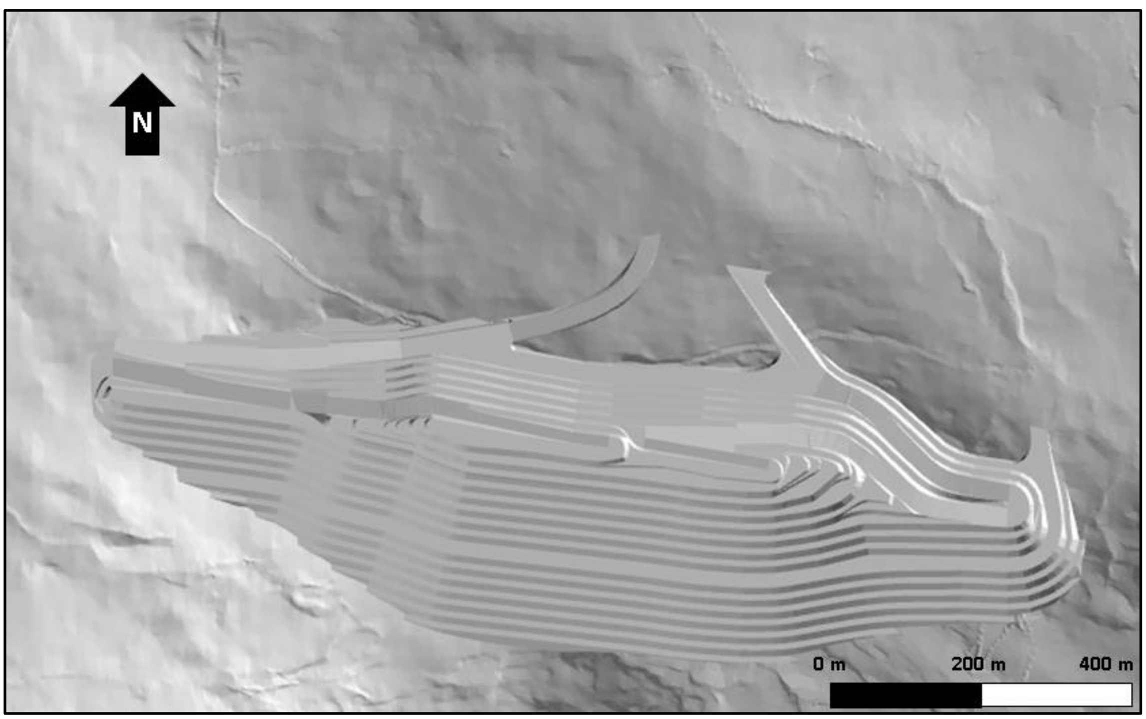


Figure 15-13: Open Pit Showing Phase 1 and Phase 2

15.5 Underground Mine Optimization

The Main and Esso underground mine optimization was undertaken by Mineit. No optimization was conducted on the Sumac deposit. However, the decline accessing Esso was located such that it would enable efficient access to Sumac should mining be contemplated.

Mining would be completed using the Longitudinal Longhole open stoping method with cemented rock fill for both Main and Esso deposits. The mining rate target was 1,000 tpd. The mining rate and method were the main controls on setting the mining cost estimate and cut-offs applied. Mining access was designed by footwall ramp. Stopes measure on average 40 m long, 25 m high, and 12 m wide. Articulated trucks (45-tonne) are allocated to ore and waste haulage with a variety of LHD loader bucket classes depending on task. Drilling will be performed by jumbo and production long-hole rigs.

15.5.1 Block Model

The probable Mineral Reserve Estimate is based on the Indicated Mineral Resource block models prepared by SGI for the Main and Esso deposits. These models were supplied as a 5 m (X) x 2.5 m (Y) x 5 m (Z) with mineralization defined as a percentage and grades defined for mineralization and non-mineralization portions of each block.

Sub-blocked models were prepared for the Main and Esso deposits, by adding a sub-block model over the original partial block model. The sub-blocked model has a parent cell of 5 m (X) x 2.5 m (Y) x 5 m (Z), with sub-blocking to 0.5 m x 0.25 m x 0.5 m. In the sub-block model, each block contains a single set of grades, the sub-block is either mineralization or not mineralization.

To validate the sub-blocked models, the reported tonnages and grades were compared to the reported results from the original partial block models. Both models were compared, and the grades and tonnages are within 0.3%.

15.5.2 Dilution and Mining Loss

Terrane (2021) estimated that, based on the area of the stope hangingwall, an empirical dilution estimate gave a stope hangingwall dilution under 0.5 m for Main and Esso. Terrane (2021) also modelled hangingwall relaxation numerically which gave larger expected dilutions. On this basis, the stope wall rock dilution was estimated to be 0.95 m and 1.15 m for Main and Esso, respectively. Stope-back dilution recommendations from Terrane were incorporated into an additional dilution estimate.

Underground stope optimizations were commenced prior to receipt of the final Terrane 2021 report and were thus based on preliminary information. As such, the optimization utilized dilution of 0.75 m in the footwall and 0.75 m in the hangingwall for stopes <100 m depth (from surface to 1510 masl toe). For stopes below this depth, 0.5 m was applied to the footwall and the hangingwall. Due to the presence of the open pit, all Main underground stopes 1.0 m total equivalent linear overbreak/slough (ELOS) was applied. These values were similar to the recommendations received from Terrane for Main underground but were slightly less (more optimistic) than those for Esso by about 1%. This variation is not considered material.

Table 15-14 summarises the various sources of dilution and mining losses considered appropriate for the Main and Esso deposits. The average grade reduction of 6.2% was incorporated into the optimization process. The tonnage variation was not possible to add as a parameter to the optimization process but was included as a modification to the Mining Reserve.

Table 15-14: Non-ELOS mining dilution and mining loss for Main and Esso underground mines

Non ELOS mining dilution or mining loss	Units	Value
Average stope dimensions		
Stope length	m	42.5
Stope height	m	25
Stope width	m	12.5
Non-ELOS unplanned dilution and mining losses		
End wall and back dilution (Terrane)	%	0.5%
Effective end-wall dilution (based on back dilution)	%	0.1%
Mucking dilution	%	1.2%
Ore pass dilution	%	0.1%
Drill deviation	%	0.5%
Footwall hang-up ore loss	%	1.5%
Ore loss due to truck and loader misdirection	%	0.0%
Pillar recovery		
Temporary rib pillar (TRP) recovery	%	90%
TRP fill dilution	%	15%
TRP width	m	7.5
Tonnage loss from TRPs	%	2.6%
Dilution from fill (total)	%	3.9%
Total ore loss	%	4.1%
Total ore dilution	%	6.3%
Net dilution impact of non-ELOS impacts		
Net Tonnage Impact	**%**	**+2.2%**
Net Grade Impact	**%**	**-6.2%**

Most non-ELOS dilutions and mine losses were based on experience and simple volumetric calculations. The success of recovery of the temporary rib pillar (TRP) will depend strongly on local rock conditions, water inflow and the quality of the cemented rock backfill.

Overall, the net mining dilution, mining recovery and mining loss of the Mineral Resource is estimated at:

- 0.34 Mt (+12%) at grades of 0.50% Cu, 1.13% Zn, 13 g/t Ag, and 0.12 g/t Au.

*15.5.3 **Classification***

The Inferred Mineral Resource does not contribute to the financial performance of the Project and is treated in the same way as waste. In practical terms for the optimization process, any block with an inferred classification was given a zero grade. In this manner any dilution involving inferred blocks would be conservatively estimated due to their zero grade. It was considered appropriate for the Mineral Reserve that Indicated Mineral Resources are translated into Probable Mineral Reserves. There are no Measured Mineral Resources in the underground mine areas.

*15.5.4 **Royalty***

This followed the same methodology as that applied to the open pit optimization process.

*15.5.5 **PAG and nPAG Rocks***

Underground mine development will be almost entirely in footwall rocks. As such, all on average waste would be designated as PAG. However, to date, some 20% of the ABA sample data collected in the footwall is classified nPAG. Given the more selective blasting and loading capabilities of an underground mining operation compared to the Main open pit, it was considered appropriate that a greater degree of waste segregation could be achieved and thus 20% of the volume was considered nPAG. This nPAG material would be stockpiled at the mouth of the portal to use as the base for the crushing unit and other underground mining activities.

15.5.6 Optimization Cost Inputs and the Cut-Off

Operating costs were based on data available to CSA Global as of May 2021. These costs differ in cost elements to the final economic analysis, but in total are within 1%. The operating costs applied to the optimization do not include waste development costs.

Table 15-15: Underground optimization operating cost

Operating cost	C$/t
Mine	51.41
Plant (incl sustaining)	33.22
General and Administration	11.45
Stockpile Rehandle	0.05
Surface TMF	3.54
Total	**99.67**

A series of stope optimizations and mining schedules were completed in May 2021 that examined the benefits to the Project NPV by increasing the stope cut-off above the operating cost in increments of 10% (0, 10%, 20%, 30% and 40%). The mining plan had the highest NPV with underground cut-offs at 30% above the breakeven stoping cost. Based on that analysis the same limit was applied so the final cut-off was 1.3 x C$99.67/t = C$129.45/t.

15.5.7 NSR Calculations

NSR cut-off calculations for underground mining followed the same process as the open pit mine optimization process.

15.5.8 Geotechnical Considerations

Geotechnical design parameters are applied in estimates for stope dilution, stope dimensions, crown pillar dimensions, support requirements in temporary drives, permanent drives and stopes, and backfill specifications. Recommendations for the next stages of project development are included in Section 26. Geotechnical considerations relating to operational decisions are included in Section 16 and for operating cost impacts in Section 21.

15.5.9 Hydrogeological Considerations

CSA Global modelled water ingress to the Esso and Main underground mines, details of which are included in Section 20. Capital and operating costs for pumping, storage and processing of the water for eventual release are included either in the specific mining costs or overall project costs. Operational considerations are detailed in Section 16 and costs in Section 21.

15.5.10 Optimization Process

After the initial loading, preparation and verification of the sub-celled block models, the step-wise process for optimization of the sub-celled block model was:

- All Inferred Mineral Resource blocks set to zero grade
- All block grades reduced by -6.2% for non-ELOS dilution, mining recovery and mining loss
- The NSR was calculated.

The stope optimizer was run using:

- Minimum mining width (before ELOS) of 2.5 m
- Minimum stope dip angle 47°
- No stope partitions were allowed
- Stope vertical height of 25 m
- Stope length of 20 m (this allows accumulated stopes to be either 20 m, 40 m or 60 m in length matching the Terrane recommendations)
- ELOS of 0.75 m on footwall and 0.75 m on hangingwall for stopes < 100 m depth (from surface to 1510 masl toe)
- ELOS of 0.5 m on footwall and 0.5 m on hangingwall for stopes > 100 m depth
- Total operating cost of 1.3 x C$99.67/t = C$129.45/t.

The positive value stope blocks as a product of the stope optimiser at this point are shown in Figure 15-14 and Figure 15-5. Dark green blocks in the Main diagram were given 0.75 m dilution. All stopes are shown prior to open pit removal, crown pillar removal, and isolated stope block removal.

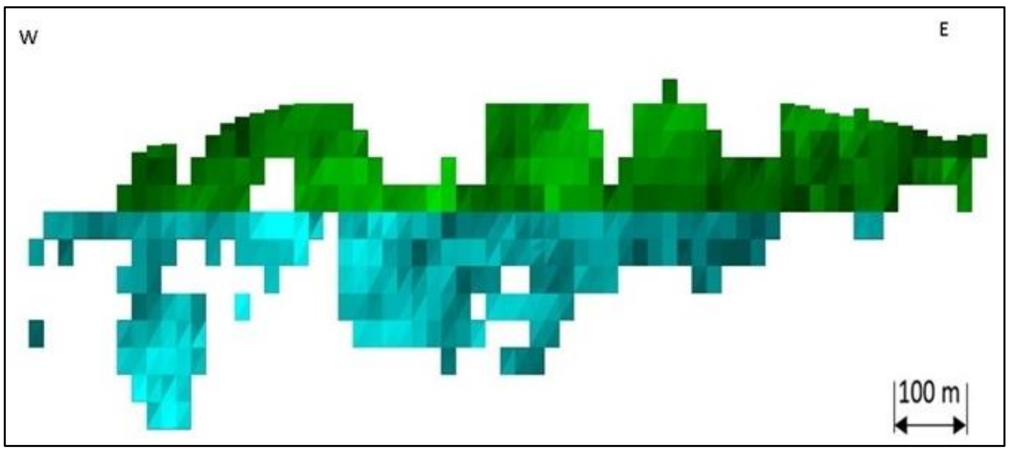


Figure 15-14: Longitudinal Section Looking North Showing Main Deposit Stope Optimization Results
Note: Green blocks have 1.5 m ELOS, blue 1.0 m ELOS.

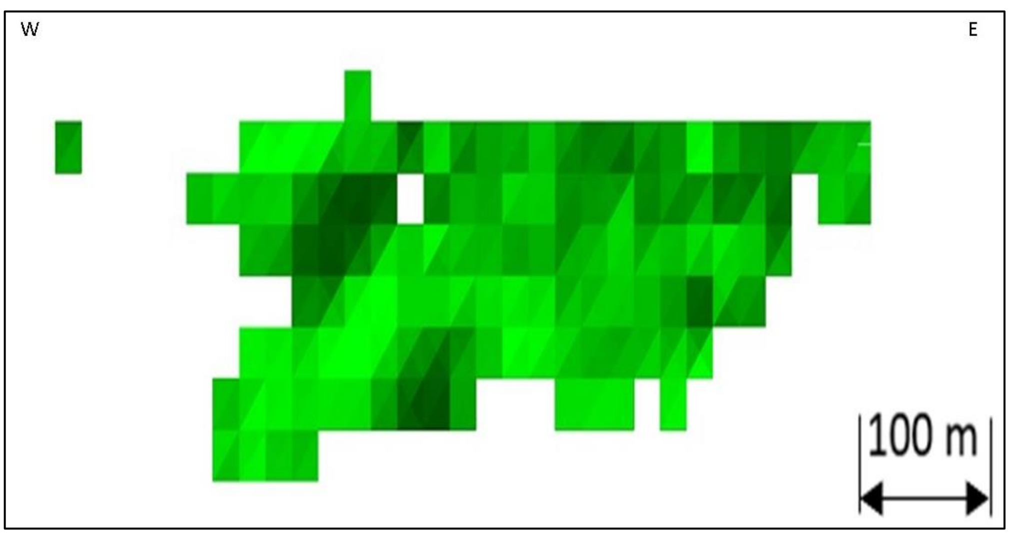


Figure 15-15: Longitudinal Section Looking North Showing Esso Deposit Stope Optimization
Note: Results – all blocks received 1.0 m ELOS.

After the stope optimization process was complete, in order to assemble the Mining Reserve the next steps were conducted on the stope blocks that met the stope optimiser constraints:

- Crown blocks and blocks within the Main open pit were excluded
- Isolated low-value blocks excluded
- Ensure each level NSR exceeds cost of access development and feasible safe access, ventilation can be achieved
- Design stope access drives and dilute as appropriate and sum ore tonnage in stopes and drives
- Add +2.2% tonnage to account for net mining loss and dilution.

15.6 Underground Mine Design

The design process for the ramp to the Main and Esso mines considered:

- Separate Esso and Main declines
- Various portal locations on either side of Sumac Creek
- Consideration of a shaft access for Esso.

The final selection of the portal location near to the Main open pit provided the lowest haulage costs to the plant.

The Mineral Reserve considers the extraction costs of stopes as well as the cost of access development. The underground mining of both Main and Esso deposits is accretive to the Project NPV after consideration of capital, another condition of consideration for Mineral Reserves. Development designs are discussed in detail in Section 16.

Also, in the process for consideration of Mining Reserves was the practical implication of the interaction of the open pit and underground operations. It is planned that the Main underground mine will extract the ore and back fill the stopes well in advance of open pit mining. The vertical separation between underground mining and open pit mining is greater than 100 m. Coordination of open pit and underground mining will be required for blasting. Traffic flow between the open pit and underground equipment was minimized within geographical constraints, but training and safe work practices will have to be completed and maintained for the operational life. The long-term emergency egress from the mine will be through the Esso ventilation raises, which will be far from open pit operations. There are no known open pit interactions with the underground that could materially affect the estimate.

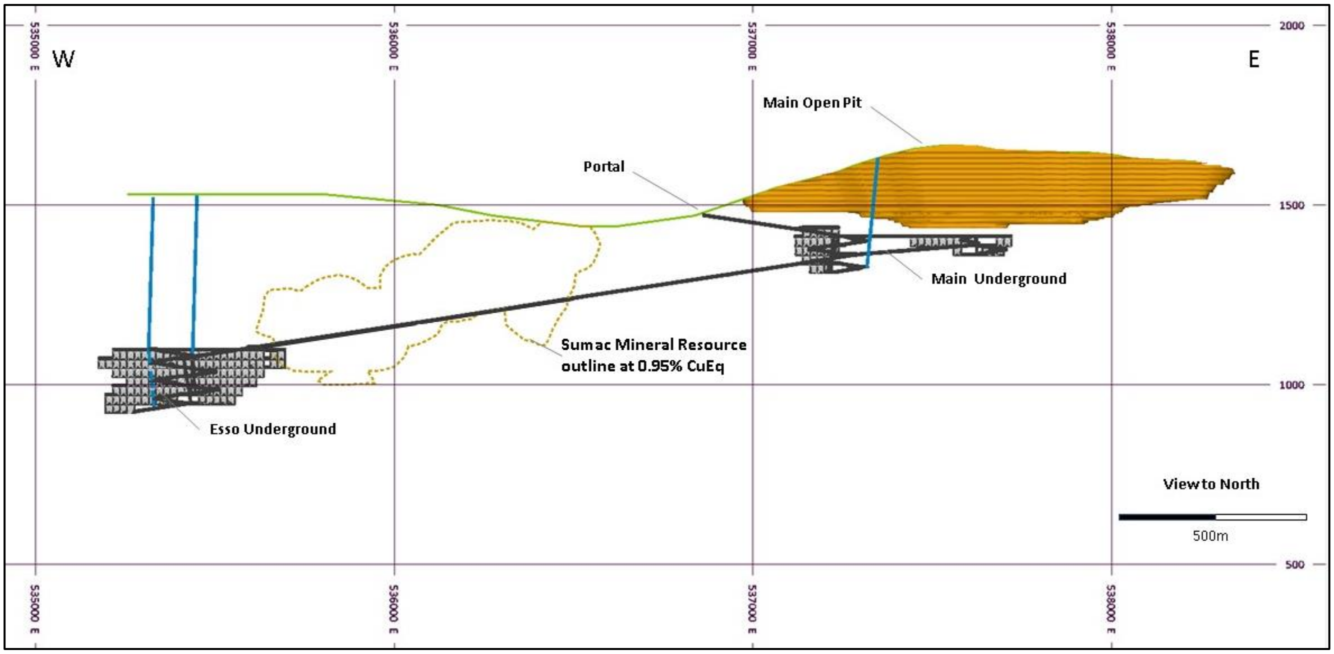


Figure 15-16: Long Section Showing Main Open Pit, Underground and Esso Underground

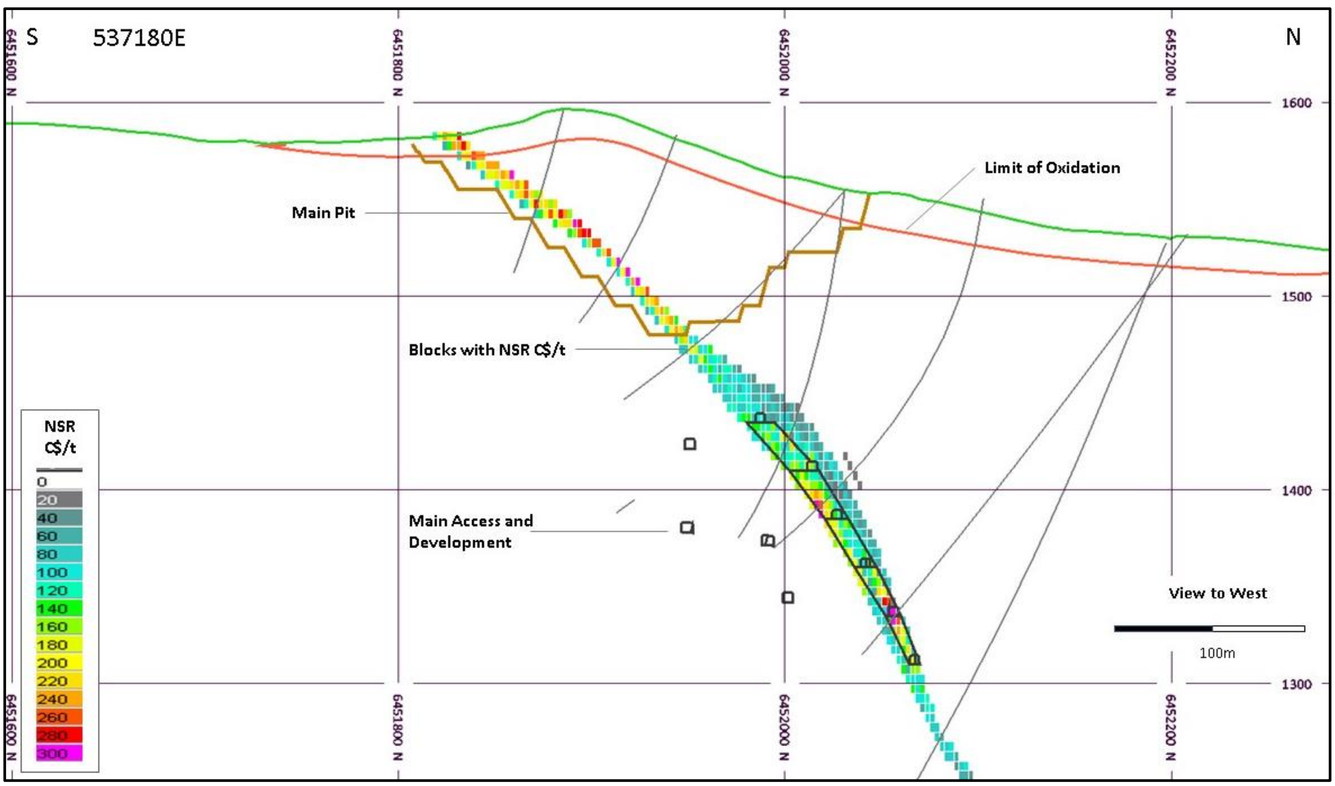


Figure 15-17: Relationship of the Main Underground and the Phase 2 Open Pit Design on Section 537180E

15.7 Mineral Reserves

The Project supports an economically viable open pit and underground mining operation. The Mineral Reserve Estimate is based on the Measured and Indicated categories of the Mineral Resource contained within the mine design. The Mineral Reserve Estimate has considered all modifying factors appropriate to the Project.

The reference point at which the Mineral Reserves are defined is where the ore is delivered to the process plant primary crusher.

There is no known mining, metallurgical, infrastructure, permitting or other relevant factors that could materially affect the estimate. It is important to note that permitting of the Project is not complete. Kutcho Copper has initiated the environmental and social assessment process, including the permitting procedures to meet British Columbian provincial and Canadian federal regulations, and at the time of writing is gathering environmental data to meet statutory requirements.

The Mineral Reserves identified in Table 15-16 comply with CIM definition standards for mineral resources and mineral reserves.

Table 15-16: Mineral Reserve for the Kutcho Copper Project (Effective Date 8 November 2021)

Mine section	Reserve category	Material type	Tonnage (Mt)	Diluted grade					Contained metal			
				Cu %	Zn %	Ag g/t	Au g/t	NSR US$/t	Cu Mlb	Zn Mlb	Ag Moz	Au koz
A. Esso Underground	Probable	Sulphide	2.23	2.14	4.13	54.9	0.71	199.48	105.3	202.6	3.9	50.7
B. Main Underground	Probable	Sulphide	0.60	1.40	1.54	32.2	0.57	120.17	18.5	20.4	0.6	11.0
C. Total Underground	Probable	Sulphide	2.83	1.99	3.58	50.1	0.68	182.68	123.8	223.0	4.6	61.7
D. Main Open Pit	Probable	Oxide	0.44	1.26	1.82	18.3	0.32	81.91	12.2	17.5	0.3	4.5
	Proven	Sulphide	6.80	1.64	2.38	24.5	0.37	135.15	245.7	356.5	5.4	80.8
	Probable	Sulphide	7.28	1.38	1.78	23.0	0.29	111.88	221.8	285.7	5.4	67.1
	Proven + Probable	**Sulphide + Oxide**	**14.51**	**1.50**	**2.06**	**23.5**	**0.33**	**121.85**	**479.5**	**659.3**	**11.0**	**153.0**
E. Total Open Pit and Underground	Probable	Oxide	0.44	1.26	1.82	18.3	0.32	81.91	12.2	17.5	0.3	4.5
	Proven	Sulphide	6.80	1.64	2.38	24.5	0.37	135.15	245.7	356.5	5.4	80.8
	Probable	Sulphide	10.11	1.55	2.28	30.6	0.40	131.68	345.5	508.7	9.9	128.8
	Proven + Probable	**Sulphide + Oxide**	**17.34**	**1.58**	**2.31**	**27.9**	**0.39**	**131.77**	**603.2**	**882.3**	**15.5**	**214.7**

Notes for the Mineral Reserve:

1. *CIM definitions were followed for Mineral Reserves.*
2. *Mineral Resources are reported inclusive of Mineral Reserves.*
3. *The Inferred Mineral Resources do not contribute to the financial performance of the project and are treated in the same way as waste.*
4. *Sum of individual values may not equal due to rounding.*
5. *Metal prices – copper US$3.50/lb, zinc US$1.15/lb, silver US$20/oz, and gold US$1,600/oz.*
6. *No previous mining has occurred.*
7. *The reference point at which the Mineral Reserves are defined is where the ore is delivered to the crusher.*
8. *There is no known likely value of the following factors of mining, metallurgical, infrastructure, permitting or other relevant factor that could materially affect the estimate.*
9. *A complex NSR formula that varies on the basis of oxide and sulphide rock types and head grade was applied. It is fully documented within the NI 43-101 Feasibility Study.*

9a. *The oxide NSR formula may be approximated to ±1% accuracy for average head grades as: NSR (US$/t) = 50.26 x Cu% + 7.09 x Zn% + 0.14 x Ag g/t + 8.94 x Au g/t.*

9b. *The sulphide NSR formula may be approximated to ±1% accuracy for average head grades as: NSR (US$/t) = 57.82 x Cu% + 9.94 x Zn% + 0.34 x Ag g/t + 22.52 x Au g/t.*

Underground Specific Notes:

10. *Underground Mineral Reserve cut-off was C$129.45/t NSR.*
11. *The minimum pre-dilution mineable width applied is 2.5 m, average stope dimensions of 25 m height, 13.1 m wide and 42 m length. Minimum footwall dip of 47°.*

12. *A 0.5 m footwall and a 0.5 m hanging wall dilution is applied and wall dilution grades were taken from estimated block grades in these locations.*
13. *A net mining recovery and mining loss estimate after wall dilution was estimated as +2.2% tonnage and -6.2% grade.*
14. *Net mining dilution, mining recovery and mining loss of the Mineral Resource is estimated at 0.34 Mt (+12%) at 0.50% Cu, 1.13% Zn, 13 g/t Ag, and 0.12 g/t Au.*
15. *All stopes included in the Mineral Reserve were optimized to maximise net cashflow and must be cashflow positive after including access development capital costs.*

Open Pit Specific Notes:

16. *Open pit Mineral Reserve cut-off was C$38.40/t NSR for oxide ore and C$55.00/t NSR for sulphide ore. The sulphide grade is an operational cut-off and is above the breakeven cut-off of C$38.40/t NSR.*
17. *Mineral Resource between the breakeven and operational breakeven cut-off not included in the Mineral Reserve amounts to 1.24 Mt at 0.53% Cu, 0.63% Zn, 9.6 g/t Ag, and 0.13 g/t Au.*
18. *The mining SMU is 5 m x 5 m x 5 m. All ore is diluted to this block dimension and is considered the minimum recoverable dimension for the mining equipment and mining method selected.*
19. *Average dilution included in the Mineral Reserve is estimated as 1.17 Mt (+8.1%). Grades are derived from waste materials, estimated as 0.12% Cu, 0.13% Zn, 1.0 g/t Ag and 0.08 g/t Au.*
20. *Mining loss for material above the operational cut-off is estimated as 0.79 Mt (5.9%) at a grade of 0.91% Cu, 0.98% Zn, 15 g/t Ag, and 0.39 g/t Au.*
21. *The Open Pit Mineral Reserve lies within a pit design that is supported by geotechnical drilling and studies and optimized for net present value.*

16 Mining Methods

16.1 Introduction

Open pit mining is expected to be by conventional front-end loader and truck whereas underground mining will be done using longitudinal long-hole open stoping (LLHOS) methods.

Open pit mining of the Main deposit is expected to commence during the construction period and ramp up over a two-year period before reaching an estimated average steady production rate of 1.8 Mtpa of ore at the end of Year 1. The open pit has an estimated eight-year life with a total of 14.5 Mt of ore mined at an average strip ratio of 5.6:1. The open pit is mined in two phases, Phase 1 and Phase 2. Direct processing of higher-grade ore and stockpiling of lower-grade ore (for processing at the end of the mine life) is a significant component of the mine plan. Phase 1 waste is used in construction of the TMF embankment and excess is placed into a waste rock dump (WRD) alongside the pit. The WRD will separately store PAG (Potentially acid generating) and nPAG (not potentially acid generating) material. Once Phase 1 of the open pit mine is complete, Phase 2 PAG waste is backfilled directly into the void and excess is placed on WRD (mostly nPAG). A significant portion of the WRD (all the PAG rock component) rehandled into the Phase 2 void at the end of open pit mining operations. Including pre-production and ore/waste rehandling periods, the open pit life is 12.5 years.

Figure 16-1 shows total material movements and strip ratio by year for the open pit and underground mines (Abbreviations: HG – High Grade, LG = Low Grade, OX = Oxidized, nPAG = Not Potential Acid Generating, PAG – Potentially Acid Generating). The open pit mining equipment is planned to move up to 40,000tpd. The truck fleet comprises up to eight 90-tonne class rigid body dump trucks. Ore and waste loading will be by two 10 m^3 wheel loaders and a 4 m^3 tracked excavator. A 7 m^3 wheel loader will be oxidized for ROM stockpile rehandling. Blast hole drilling by two down the hole hammer drill rigs will be completed on 5 m bench intervals for mixed ore and waste and 10 m benches for waste. Ancillary equipment will support the open pit mining operation, site surface activities and maintenance of the various access roads.

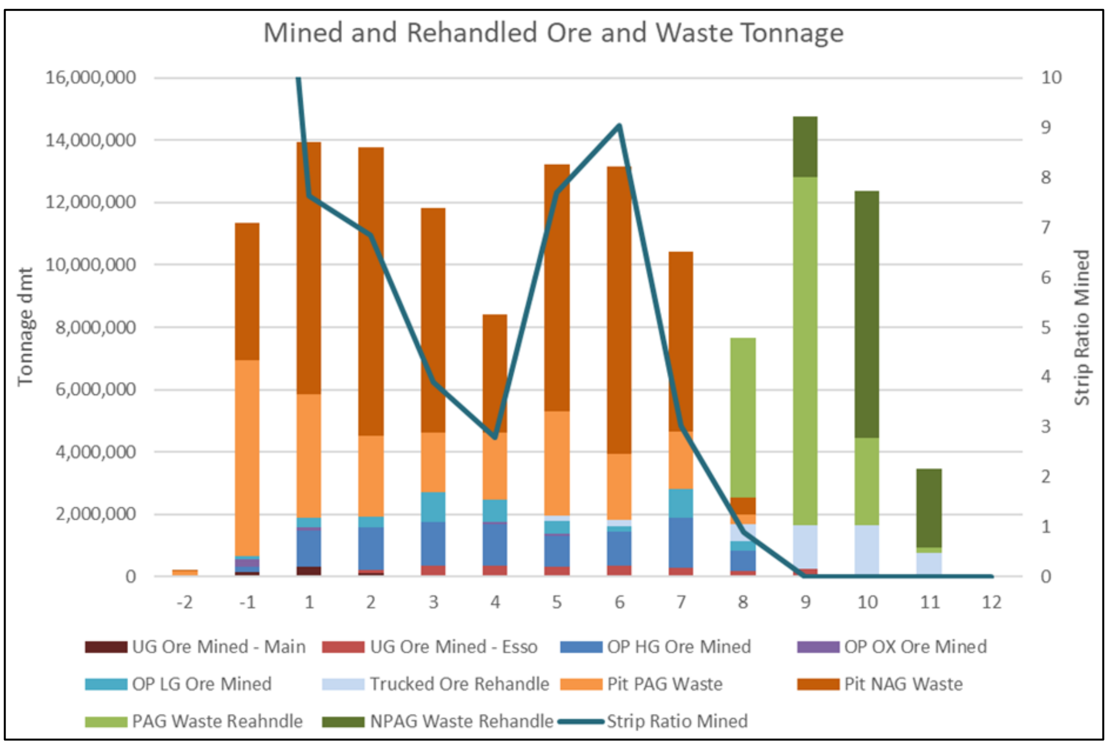


Figure 16-1: LOM Material Movements

Summary production statistics of the underground and open pit mine plan are shown in Table 16-1.

Table 16-1: Summary of the underground and open pit mining operations

		Pre-production	Early production	Steady state	Stockpile rehandle	Life-of-mine
		Years -2 to -1	Year 1	Years 2 to 8	Years 9 to 11	Years -2 to 11
Main and Esso Underground Mines						
Ore mined	Mt	0.15	0.32	2.09	0.27	2.83
Ore mined (1)	tpd	327	865	817	602	738
Underground mine life		10.5 years including pre-production				
Main Open Pit Mine						
Ore mined	Mt	0.5	1.6	12.4	0.0	14.5
Waste mined	Mt	11.5	12.1	58.2	0.0	81.8
Strip ratio	W/O	22.8	7.7	4.7	0.0	5.6
Ore stockpile rehandle	Mt	0.0	0.0	0.9	3.8	4.7
Waste rehandle	Mt	0.0	0.0	5.1	26.5	31.6
Total material movement	Mt	12.0	13.7	76.6	30.3	132.5
	tpd	16,400	37,562	29,978	27,645	27,934
Open pit mine life		12.5 years including pre-production and rehandle				

(1) Includes 1.25 years for pre-production and stockpile rehandle periods.

A single portal provides access to both the Main and Esso deposits. The time between first commencement on the underground portal and first ore mining is short at around nine months. The first ore is accessed from the Main deposit at 340 m from the portal entrance. The access ramp then spirals to access additional mining levels before splitting to access Main ore further to the east and also driving west to the Esso deposit. The ramps would be utilized to haul ore and waste to the surface, backfill from the surface to the stopes and to provide access for personnel, equipment, materials, and services. Additionally, the ramps form part of the mine dewatering and air ventilation circuit. The Main underground mine is located beneath the Main open pit and is separated from it by a 25 m wide crown pillar. The top of the Esso deposit is approximately 400 m below surface and is accessed by an 1,800 m connecting ramp extending from the Main ramp system. Further ramps are required to access the production stopes in the Main and Esso mines. The total ramp length for both mines are anticipated to be 5,437 m.

Underground mining of both the Main and Esso deposits is expected to be by longitudinal long-hole open stoping (LLHOS). The stopes will be backfilled using cemented rock fill (mostly ore sorter reject from the processing plant) and are planned to utilize temporary rib pillars and cable bolting as primary stope support measures. The underground portion of the Main deposit is expected to have a three-year mine life, extending from Year -1 to Year 2. Total ore production from Main underground is estimated to be 0.6 Mt and is planned to be mined at an average rate of 600 tpd although it on a quarterly basis it peaks at about 900 tpd. Underground mining of the Main deposit finishes early in the life of the open pit. Consequently, there is minimal interference between the two operations. The 25 m crown pillar is not planned for recovery.

Production from the Esso deposit is expected to commence in Year 2 and ramp up to full production in the following year. A generalized development schedule of Esso and Main development is shown in Figure 16-2. Total projected ore production from Esso is 2.2 Mt at a steady state ore production rate of 0.3 Mtpa or 860 tpd. At these production rates, the mine life for Esso is estimated at nine years excluding pre-production which adds a further two years. As reflected in Table 16-1, Esso ore contains higher average grades of copper and zinc than the Main deposit. Consequently, even with the higher cost of underground mining ore production from Esso is scheduled as early as possible.

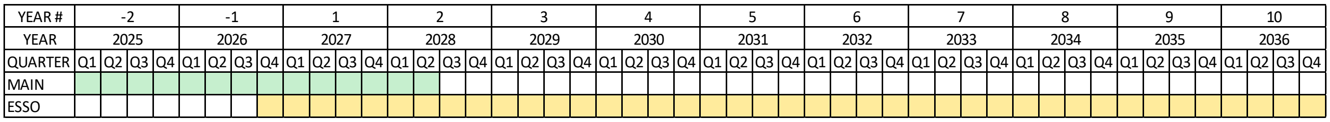


Figure 16-2: Generalized LOM Development Schedule

The mine schedule allows for the stockpiling of low-grade ore from the open pit throughout operations, allowing higher grades to be processed earlier. The lower-grade stockpiles are processed in the final years of mine life, commencing in Year 8 but primarily from Year 9 through to Year 11. The ore sources for crusher feed and the crusher feed NSR C$/t are shown in Figure 16-3.

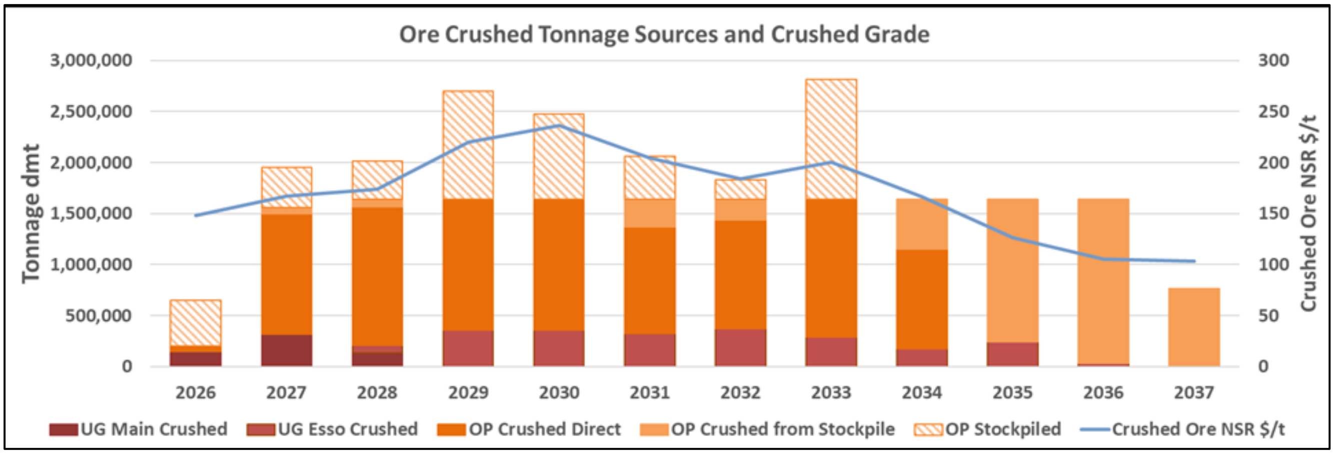


Figure 16-3: Crushed Ore Sources and Ore NSR C$/t

16.2 Geotechnical Evaluation

16.2.1 Geotechnical Database and Analysis

Prior to 2018, a total of 100 geotechnical boreholes were drilled at the Main and Esso deposits (81 at the Main deposit and 19 at the Esso deposit). The 81 boreholes at the Main deposit were evaluated but not used because of logging inconsistencies. The 19 boreholes at the Esso deposit did not contain sufficient geotechnical information to perform a geotechnical evaluation. The drilling was under the direct supervision of Kutcho Copper personnel for the duration of the drilling programs. Kutcho Copper personnel also logged (geologically and geotechnically) and photographed the rock cores.

As part of the 2010 drilling program, field and laboratory testing were conducted to evaluate the compressive strength of intact rock by unconfined compressive strength tests, and the tensile strength by Brazilian tensile tests.

In 2018, Terrane Geoscience Inc. (TGI) initiated a geotechnical investigation on the Kutcho Project site at both the Main and Esso deposits. The core size used was HQ3 and all holes were oriented. Optical and acoustic televiewer surveys were completed on each hole. A total of 36 samples were taken and were sent to the Norman B. Keevil Institute of Mining Engineering & Rock Mechanics at the University of British Columbia for analysis. The samples were analyzed for:

- Unconfined compressive strength
- Unconfined compressive strength with elastic modulus
- Indirect tensile (Brazilian)
- Direct shear
- Triaxial testing.

The geotechnical field data collection program was designed with the aim of characterizing the rock mass, hydrogeology, structural fabrics, and major structures (i.e. faults) associated with both the Main and Esso deposits. This data was collected with the intent of developing a geotechnical model suitable for open pit and underground mining. The field data collection program consisted of geotechnical logging of oriented core, index strength testing (i.e. point load testing), packer testing, geomechanical sample collection, and optical/acoustic televiewer surveying. The initial geotechnical field program was completed by Terrane between 12 June and 9 October 2018. An additional geotechnical field program was completed by Terrane between 11 May and 13 June 2021 (Terrane Geoscience Inc., 2021).

Drilling was completed by CYR Diamond Drilling (2018) and Konaleen Drilling (2021) in 12-hour shifts under contract to Equity Exploration Consultants Ltd. A total of 22 HQ3 size (61.1 mm) triple tube holes, and three wedges (two HQ, one NQ) were completed totalling 5,608.5 m of drilling between the Main and Esso deposits. A total of 2,294.4 m was drilled at Main and 3,314.2 m at Esso. Drill core was oriented using a Reflex Instruments ACT III orientation instrument combined with a Gyro non-magnetic downhole survey tool. Measurements for dip and azimuth were collected with the Gyro at regular intervals in each hole. All core was logged by Terrane staff at the drill (2018) or at the core logging facility at the Kutcho camp (2021). All geotechnical core logging was completed by an Intermediate Geological Engineer from Terrane with assistance from a Junior Geological Engineer (Terrane Geoscience Inc., 2021).

All core logging was completed in accordance with accepted geotechnical logging standards and included the collection of the required parameters to calculate RMR76 (Bieniawski, 1976) and the Q-system (Barton et al., 1974). The logging consisted of interval logging or detailed logging of each core run. Each core run was 3 m long using a standard core barrel. For interval logging, data was collected on core recovery, Rock Quality Designation (RQD), discontinuity characteristics (e.g. alteration, weathering, and infill) and fracture counts (Terrane Geoscience Inc., 2021).

In addition to interval logging, Terrane completed discrete logging of all orientated core discontinuities. This included measurement of the alpha and beta angles to evaluate structural fabric orientations. Additionally, brittle discontinuities and faults encountered within the geotechnical drill core were logged and characterized (Terrane Geoscience Inc., 2021).

16.2.1.1 Index Testing (Point Load Testing)

Point load testing was conducted approximately every 9 m (every three core runs) of drill core to develop a database of indicated strengths along the entire length of the drillhole. Point load tests were completed as per "ASTM D5731 – Standard Test Method for Determination of the Point Load Strength Index of Rock and Applications to Rock Strength Classifications." These were used in conjunction with the geomechanical laboratory testing results to determine the rock mass strength characteristics. (Terrane Geoscience Inc., 2021). A total of 727-point load tests were completed on geotechnical drillholes at Main and Esso.

16.2.1.2 Hydrogeological Investigation

Hydrogeological investigations were completed during the 2018 and 2021 field program which consisted of packer testing to determine in-situ hydraulic conductivity in selected geotechnical drillholes at the Main and Esso deposits. (Terrane Geoscience Inc., 2021)

16.2.1.3 Geomechanical Laboratory Testing

The 2018 geomechanical laboratory testing was completed by PHC Inc at the University of British Columbia, Rock Mechanics Laboratory (UBC), Vancouver, BC. The 2021 testing was completed by Natural Resources Canada at their Canmet MINING Rock Mechanics Laboratory (Canmet) in Ottawa, Ontario. The testing conducted consisted of Unconfined Compressive Strength (UCS) testing, Brazilian Tensile Strength testing, direct shear strength testing, and Triaxial (confined) Compressive Strength testing (TCS). In total, 186 geomechanical tests were completed on the 21 drillholes, 127 samples from the Main deposit and 59 samples from the Esso deposit.

The results of the geomechanical testing were analyzed by Terrane Geoscience Inc. (2021). Terrane provided recommendations for mine design that were followed in design of the proposed open pit and underground mine workings.

Triaxial Compressive Strength

Triaxial Compressive Strength (TCS) testing is conducted by encasing the specimen in a heat shrink tubing and applying a confining pressure (σ_3) to the sample, while simultaneously subjecting the specimen to an axial load (σ_1) until failure occurs. The load (kN) at failure when divided by the cross-sectional area of the core specimen results in the triaxial compressive strength (σ_1) of the rock at the applied confining pressure (σ_3). The TCS tests were completed at confining pressures of 1.5 to 10.0 mPa. The TCS testing was conducted in accordance with ASTM D7012 (Standard Test Methods for Compressive Strength and Elastic Moduli of Intact Rock Core Specimens under Varying States of Stress and Temperatures, 2019).

Direct Shear Testing

Direct shear testing was completed on natural discontinuities from eighteen core specimens. A direct shear sample is composed of two separate core specimens that fit together along a natural break or discontinuity. Direct shear testing is completed by applying a normal load (i.e. perpendicular) to the discontinuity being tested while also monitoring the shear stress (σ_t) that results in the displacement of one block relative to the other. To determine the shear strength a multi-stage testing procedure was used that applies three different normal loads (1, 2, and 3 mPa) to the core specimen and monitoring the shear stress induced when movement occurs. After each normal load is applied and movement occurs the discontinuity is repositioned back to its original position. The relationship between the applied normal load and the shear strength can then be plotted to determine the shear strength envelope. From these data points a peak and residual shear strength of the discontinuity can be determined using statistical regression analysis. All testing was completed in accordance with ASTM D5607-16 (Standard Test Method for Performing Laboratory Direct Shear Strength Tests of Rock Specimens Under Constant Normal Force, 2019) (Terrane Geoscience Inc., 2021).

The results of the direct shear testing were used to estimate the shear strength of natural discontinuities within the rock mass (e.g., foliation, joints, faults, and veins). The shear strength values were plotted against the applied average normal stress (σ_1) to determine the friction angle for Mohr-Coulomb strength criteria (Terrane Geoscience Inc., 2021).

Brazilian Tensile Testing

Splitting tensile strength tests, also known as Brazilian tensile testing, were conducted on 79 core specimens. Brazilian tensile testing is completed by subjecting a cylindrical core specimen (disc) to a diametrical load. The load causes tensile deformation normal to the loading direction, yielding a tensile failure. Using the resulting ultimate load and knowing the dimensions of the cylindrical core specimen allows for calculation of the indirect tensile strength of the rock. All testing was completed in accordance with ASTM D3967-16 (Standard Test Method for Splitting Tensile Strength of Intact Rock Core Specimens, 2019). (Terrane Geoscience Inc., 2021).

The results of the Brazilian tensile testing were used in conjunction with the results of both TCS and UCS testing to determine the material constant parameter mi for the generalized Hoek-Brown criteria (Hoek et al., 2002).

16.2.2 *Geological Discontinuity Features–- Rock Fabric and Major Geological Structures*

16.2.2.1 *Rock Fabrics*

Main Deposit

Stereonet analysis was completed for each drillhole and for each geotechnical domain for the Main deposit discontinuity orientation data. Additionally, plots of dip vs. depth for each drillhole, and change in average dip of the primary foliation (schistosity) S1 across strike was also completed. This data is summarized below (Terrane Geoscience Inc., 2021).

The Main deposit is characterized by one primary discontinuity set, S1, which is observed as a penetrative foliation (schistosity) that can be broken into two subsets, S1a and S1b. These subsets represent steep and shallow limbs of second order folds on the limbs of the larger scale, south vergent thrust structures. S1a is the dominant set of the two observed subsets. (Terrane Geoscience Inc., 2021).

Stereonet analysis of the fabric orientation data for the Main deposit indicates that the deposit is composed of one structural domain with changes in strike of the S1 discontinuity set from west to east. Drillhole analysis of the average dip of the S1 discontinuity set with depth does not indicate a systematic change with depth but rather a series of shallower and steeper dips that are interpreted as the change from S1a to S1b (Terraneoscience Ge Inc., 2021).

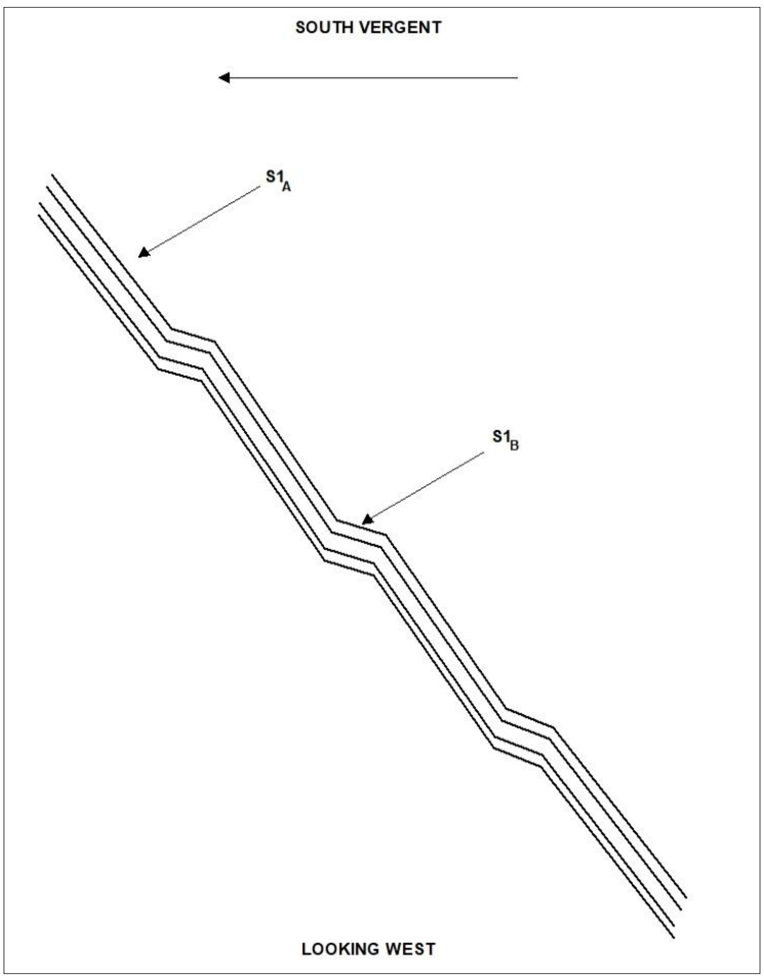


Figure 16-4: Section View Showing Discontinuity Sets in Relationship to Regional Deformation (not to scale)
(Source: Terrane Geoscience Inc., 2021)

Table 16-2: Summary of Main deposit discontinuity sets

Discontinuity set	Dip (°)	Dip direction (°)	Dip direction range	Dip range
S_{1a}	62	012	019–004	57–67
S_{1b}	41	010	023–357	35–48

(Source: Terrane Geoscience Inc., 2021)

Esso Deposit

Stereonet analysis was completed for each drillhole and for each geotechnical domain for the Esso deposit orientation data. Additionally, plots of dip vs. depth for each drillhole and change in average dip of the primary foliation (schistosity) S1 across strike was also completed. This data is summarized below and in Appendix C of the Terrane Geoscience report (2021).

The Esso deposit is characterized by one primary discontinuity set, S1, which is observed as a penetrative foliation (schistosity). A secondary discontinuity set JS1 which is a conjugate to S1a was observed locally throughout the rock mass (Terrane Geoscience Inc., 2021).

Table 16-3: Summary of Esso deposit discontinuity sets

Discontinuity set	Dip (°)	Dip direction (°)	Dip direction range	Dip range
S_1	63	009	0–- 019	54–- 72
JS_1	57	126	114–- 138	47–- 66

(Source: Terrane Geoscience Inc., 2021)

16.2.2.2 Major Geological Structures

Main Deposit

The fault surfaces modelled at Main reflect the structural history of the area, consisting of three north dipping thrust faults forming a sub-parallel set to the Nahlin and King Salmon Faults to the north and south, respectively. A summary of each modelled fault is provided in Table 16-4 (Terrane Geoscience Inc., 2021).

Table 16-4: Major fault summary – Main deposit

Fault ID	Type	Dip (°)	Dip direction (°)	Average thickness (m)	Maximum thickness (m)	Confidence
Upper Main Fault	Thrust	55	10	0.4	0.7	High
Main Fault	Thrust	48	7	1.3	2.3	High
Lower Main Fault	Thrust	50	7	0.6	1	High
Fault 6	Dextral strike-slip	72	56	0.1	0.1	Low
Fault 7	Dextral strike-slip	61	59	0.1	0.1	Low

Note: Dip and dip direction is the mean dip from stereonet analysis of each faults modelled vertices.
(Source: Terrane Geoscience Inc., 2021)

Steeply dipping crosscutting faults were again modelled at Main and are considered later than the primary thrust faults. Evidence for these structures occurs in both faulted intervals in drillholes as well as minor offsets to the sulphide lenses. Main Fault 6 and Main Fault 7 are both considered low confidence and insignificant from a geotechnical perspective (Terrane Geoscience Inc., 2021).

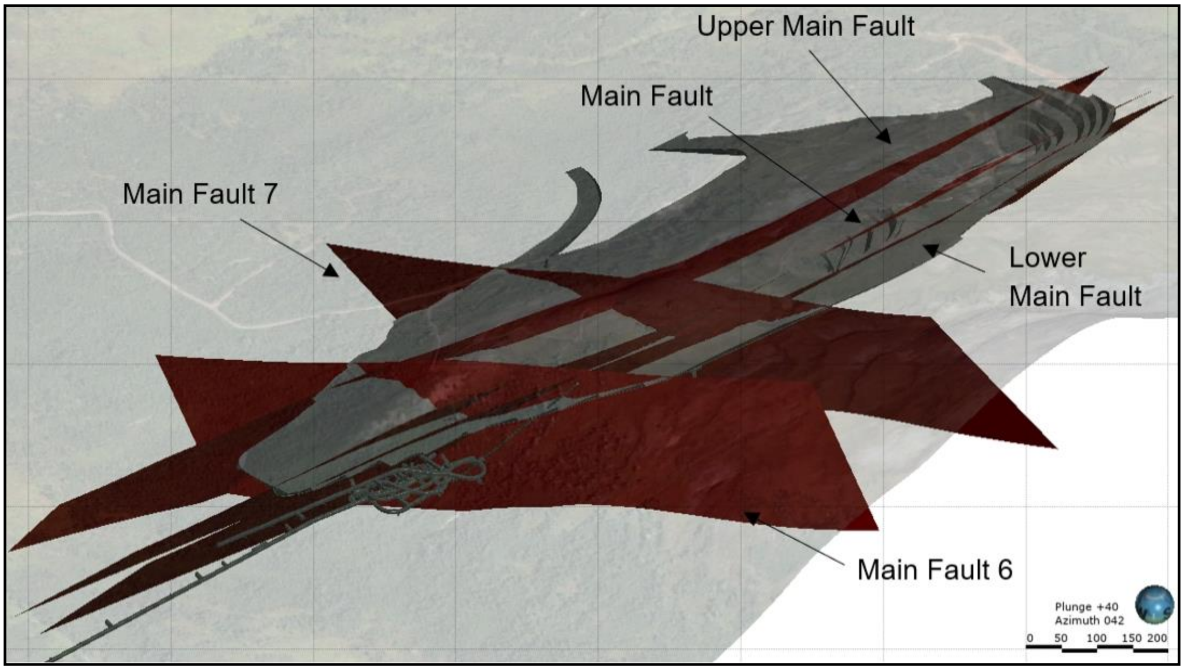


Figure 16-5: 3D View of Major Faults at the Main Deposit
(Source: Terrane Geoscience Inc., 2021)

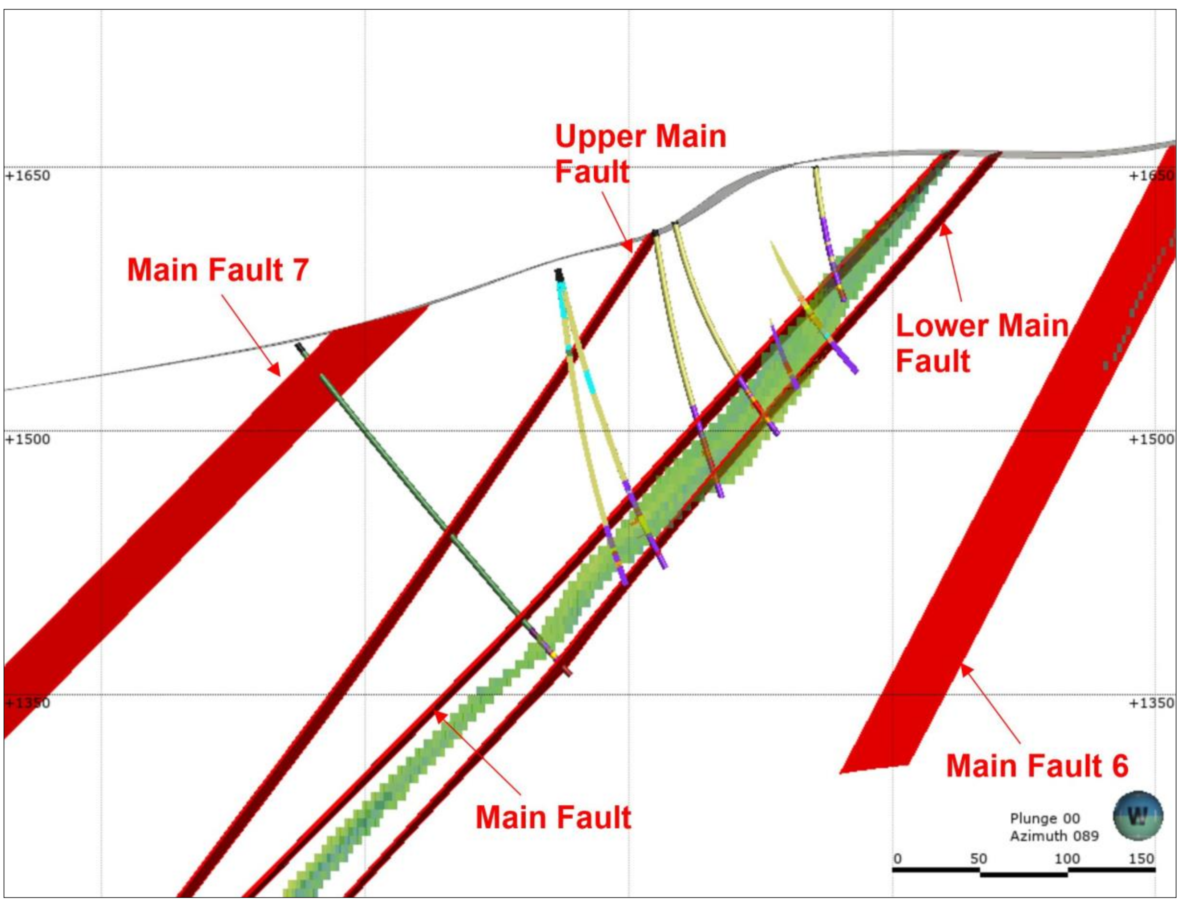


Figure 16-6: Section View of Main Deposit Showing Modelled Thrust Faults (looking west)
Notes: Section width is 25 m; fault widths do not represent true thickness.
(Source: Terrane Geoscience Inc., 2021)

Esso Deposit

Like Main, the fault surfaces established during modelling reflect the structural history of the area, with at least three north dipping thrust faults forming a sub-parallel set to the Nahlin and King Salmon Faults to the north and south, respectively. A summary of each modelled fault is provided in. The three faults broadly correlate across the Kutcho Property (Terrane Geoscience Inc., 2021).

Table 16-5: Major fault summary – Esso deposit

Fault ID	Type	Dip (°)	Dip direction (°)	Average thickness (m)	Maximum thickness (m)	Confidence
Upper Esso Fault	Thrust	61	2	0.1	1.7	High
Esso Fault	Thrust	58	4	3.1	11.7	High
Lower Esso Fault	Thrust	56	5	6.8	10.3	High
Fault 5	Dextral strike-slip	88	292	0.1	0.1	Low
Upper Esso Fault	Thrust	61	2	0.1	1.7	High

Note: Dip and dip direction is the mean dip from stereonet analysis of each faults modelled vertices.
(Source: Terrane Geoscience Inc., 2021)

Initial assessment identified a series of offsets along the mineralized horizons at the Sumac-Esso boundary where mineralization appears to be displaced by up to 70 m along a north-south trend. A preliminary interpretation was made of a set of steeply dipping approximately north-northwest trending strike slip faults, possibly as either transfer faults associated with the thrust system or a late dextral strike-slip fault associated with the Kutcho fault to the east. Drill testing in 2018 did not conclusively identify the presence of this steeply dipping structure, although the drillhole orientations were not favorable to accomplish this. This structure is considered to be of low confidence in the model (Terrane Geoscience Inc., 2021).

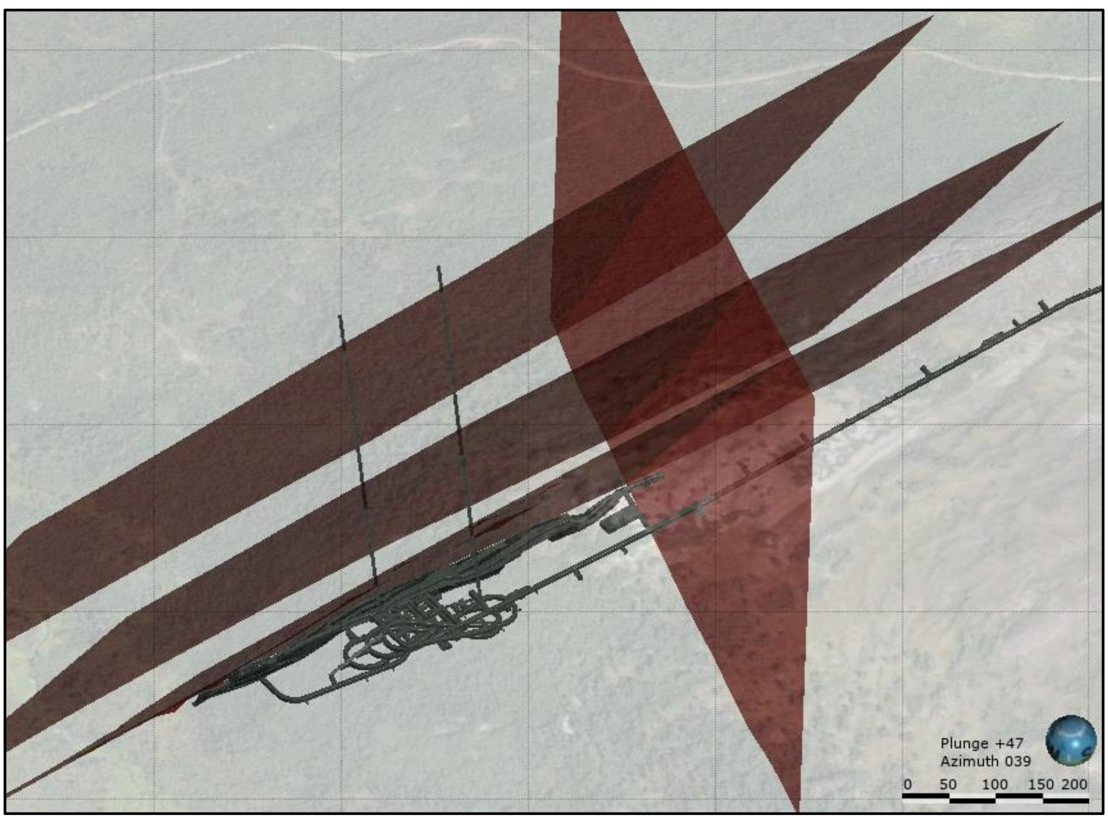


Figure 16-7: 3D View of Major Faults at the Esso Deposit
(Source: Terrane Geoscience Inc., 2021)

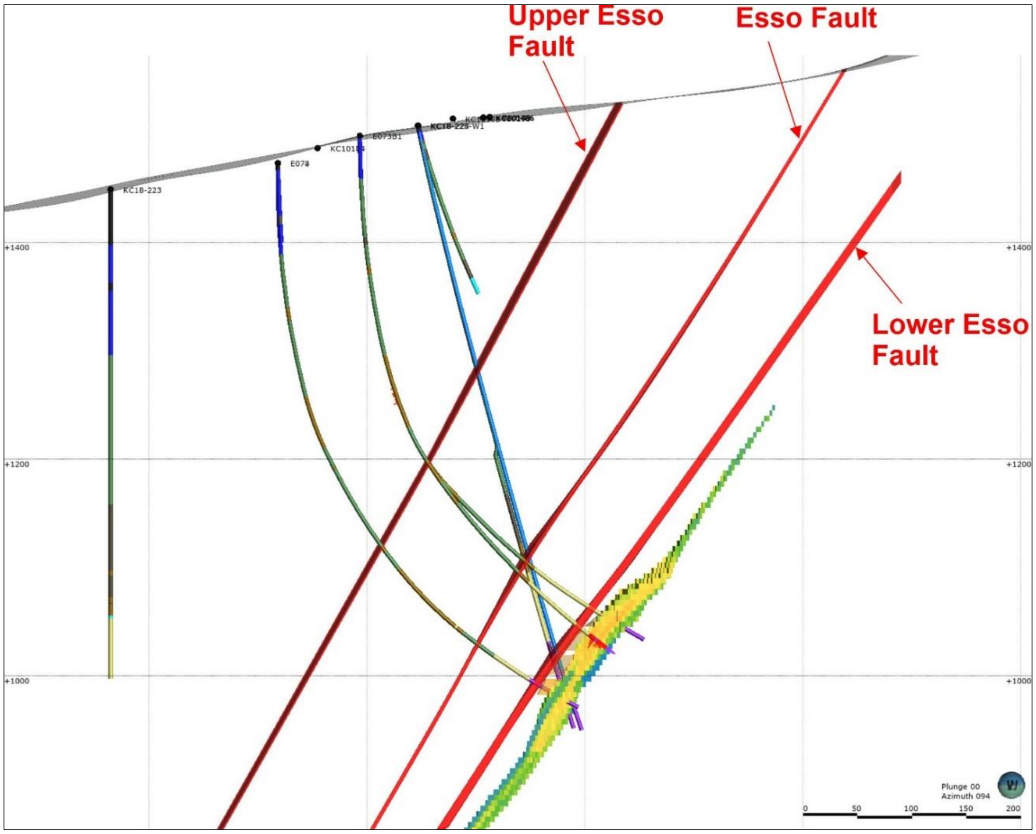


Figure 16-8: Section View of Esso Deposit Showing Modelled Thrust Faults (looking west)
(Source: Terrane Geoscience Inc., 2021)

16.2.3 Rock Mass Assessment and Geotechnical Model

16.2.3.1 Main Deposit

The geology of the Main deposit consists of interbedded tuffaceous rocks in the footwall and the hanging wall is dominated by quartz-feldspar tuff with lesser volcanogenic siltstone and argillite (Table 16-6 and Figure 16-9). Massive to semi-massive sulphide lenses straddle the contact between the two zones. Primary stratigraphic relationships have been overprinted by the strong influence of north-dipping thrust faults and repetition of strata is again assumed. Sub-volcanic gabbroic intrusions are volumetrically significant in the hanging wall (Terrane Geoscience Inc., 2021).

Table 16-6: Modelled geological units

Rock code	Colour[1.]	Lithology description
ARGL		Argillite
AHTF		Ash tuff
CBEX		Carbonate Exhalite
CNGL		Conglomerate
GBBR		Gabbro
LLTF		Lapilli Tuff
MSSL		Massive Sulphide
QFXT		Quartz Feldspar Crystal Tuff
SMSX		Semi-Massive Sulphide
VSLT		Volcanic Siltstone

(Source: Terrane Geoscience Inc., 2021)

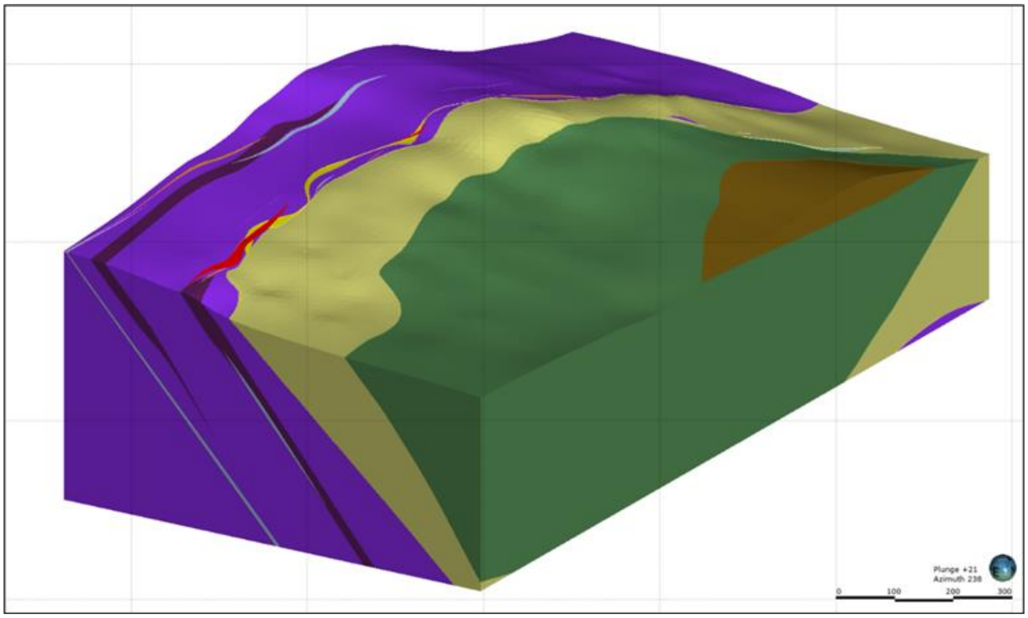


Figure 16-9: 3D Oblique View Showing Main Deposit Geology Solids (view to southwest)
(Source: Terrane Geoscience Inc., 2021)

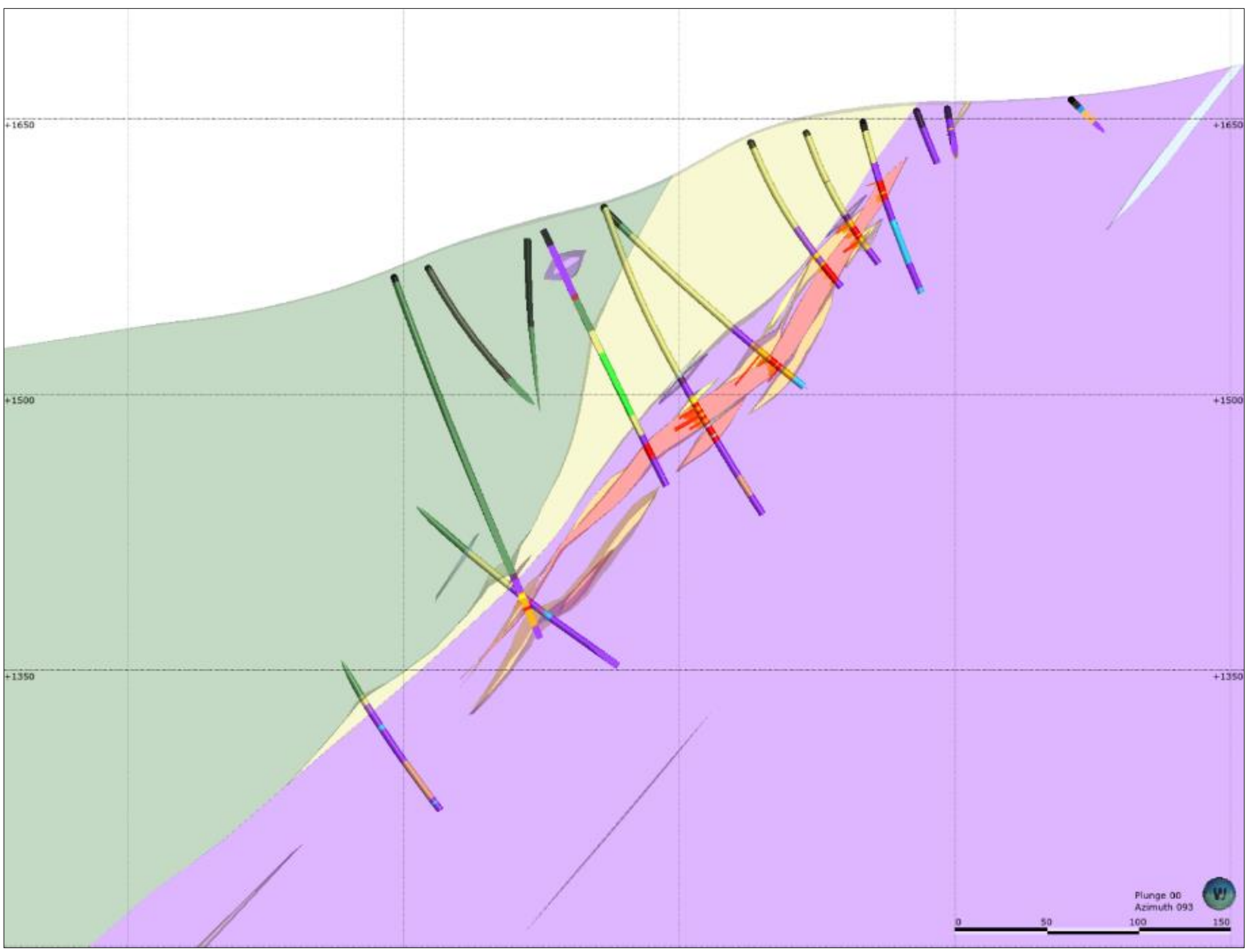


Figure 16-10 Cross-Section Through the Main Deposit Showing Massive Sulphide Lenses (looking west)
(Source: Terrane Geoscience Inc., 2021)

The rock masses that compose the Main deposit were found to be Fair to Good quality. The surrounding local rocks tend to be in the poor to very poor range as they are associated with fault zones or near-surface structures. The Main rocks were preliminarily classified into one of eight geotechnical domains. There were initially four primary domains, each one further separated into shallow (0 m to 20 m or 100 m) and deep zones (20 m or 100 m to 250 m) for a total of eight domains. Upon further review and analysis, the Main rocks were re-classified into one of five geotechnical domains. These final geotechnical domains are illustrated in Table 16-7.

Table 16-7: Final Main deposit geotechnical domains

Geotechnical domain	Description
HW	Gabbro, quartz feldspar oxidized tuff
OZ	Massive to semi-massive sulphide
FW_1	Felsic tuff, 0 – 125 m below surface
FW_2	Felsic tuff, greater than 125 m below surface
FLT	Brittle Faults / Major Structures

(Source: Terrane Geoscience Inc., 2021)

Table 16-8 presents the five domains with the average rating value for two rock mass classification systems: the Norwegian Geotechnical Institute's NGI Q-System and Bieniawski's Rock Mass Rating (RMR_{76}).

Table 16-8: Final Main deposit rock mass classification and intact rock strength parameters

Geotechnical domain	RMR_{76} (1)	Q' (2)	UCS (mPa) (3) (4)
HW	61	16.6	70 / 58.5
OZ	61	23.4	93
FW1	39	4.8	22
FW2	54	8.9	63
FLT4.	N/A	0.7	N/A

Notes:
(1) Average RMR76 within the geotechnical domain.
(2) Geometric mean of Q' values within the domain. Geometric mean is the average of the logarithmic values of a dataset. Discrete structures, Q' only reported.
(3) Average values from valid laboratory UCS tests and point load testing.
(4) Average UCS for immediate hanging-wall to ore zone, value used for stope design.
(Source: Terrane Geoscience Inc., 2021)

16.2.4 *In-Situ Stress*

To date, there have been no in-situ stress measurements for the Kutcho Project area. As a result, Terrane has estimated in-situ stress based on engineering judgement and commonly accepted assumptions.
The following estimated conditions have been assumed:

- Sigma V = σV = ɣ * Z = vertical in-situ stress
- Sigma H = σH = σ1 = 2 * σV = maximum horizontal in-situ stress (perpendicular to mineralization strike)
- Gamma = ɣ = unit weight of the rock = 0.028 MN/m^3
- Depth = Z (metres). (Terrane Geoscience Inc., 2021).

16.3 Site Water Balance and Mine Water Modelling

Site water management is central to maintaining an appropriate environmental and operational performance for the Project. The principle adopted for site water management is to intercept and control contact water flowing within the operational areas and direct it to the contact water dam (CWD) for treatment to allow reuse in the process plant or clean release to the Kutcho Creek. Some untreated water will be permanently stored in the TMF as moisture in tailings and a small amount will leave site in the concentrate product. The site water balance indicates that the Project will have a positive water balance requiring seasonal treated water release.

16.3.1 Monthly Water Balance Model

A monthly water balance model was developed for the Project to quantify water volumes from site runoff and dewatering under average, dry and wet year conditions on a monthly basis. The water balance was evaluated for three scenarios: average and wet and dry (1-in-100-year return period) conditions. Water balance results for Mine Year 3 under average year climate conditions are summarized in Figure 16-11.

A monthly water balance was developed for mine dewatering and contact water to determine the required water treatment plant (WTP) capacity during mining operations. Use of the tailings management facility (during winter and peak freshet) to store contact water requiring active treatment will facilitate constant rates of water treatment and discharge to Kutcho Creek during the open water season and reduce the required maximum treatment rate of the WTP. A maximum open water season WTP discharge of 2.9 Mm^3 (at a constant rate of 182.4 L/s) is predicted in Year 3 for a 1 in 100-year return period wet year.

A preliminary mass loading model was developed from the water balance model for the Project site, during the mine construction and operations periods to provide WTP influent water quality predictions to support WTP design. Water treatment requirements were developed using predictions of WTP influent water quality during the construction and operations period that were based on conservative assumptions for loading sources (e.g. potential acid generating and non-potential acid generating rock in the waste rock facility and open pit backfill, and exposed rock surfaces in underground works pit wall).

Lorax Environmental Services Ltd (Lorax) prepared preliminary estimates of assimilative capacity in the Kutcho Creek and developed associated preliminary WTP discharge criteria to support WTP design. Preliminary WTP discharge criteria estimates were compared to WTP influent predictions to inform which chemical constituents require treatment to meet receiving environment criteria in the Kutcho Creek.

Further refinements to the model, including development of the water balance and water quality predictions in the receiving environment and extending the model into closure and post-closure, are required to support environmental assessment and permitting.

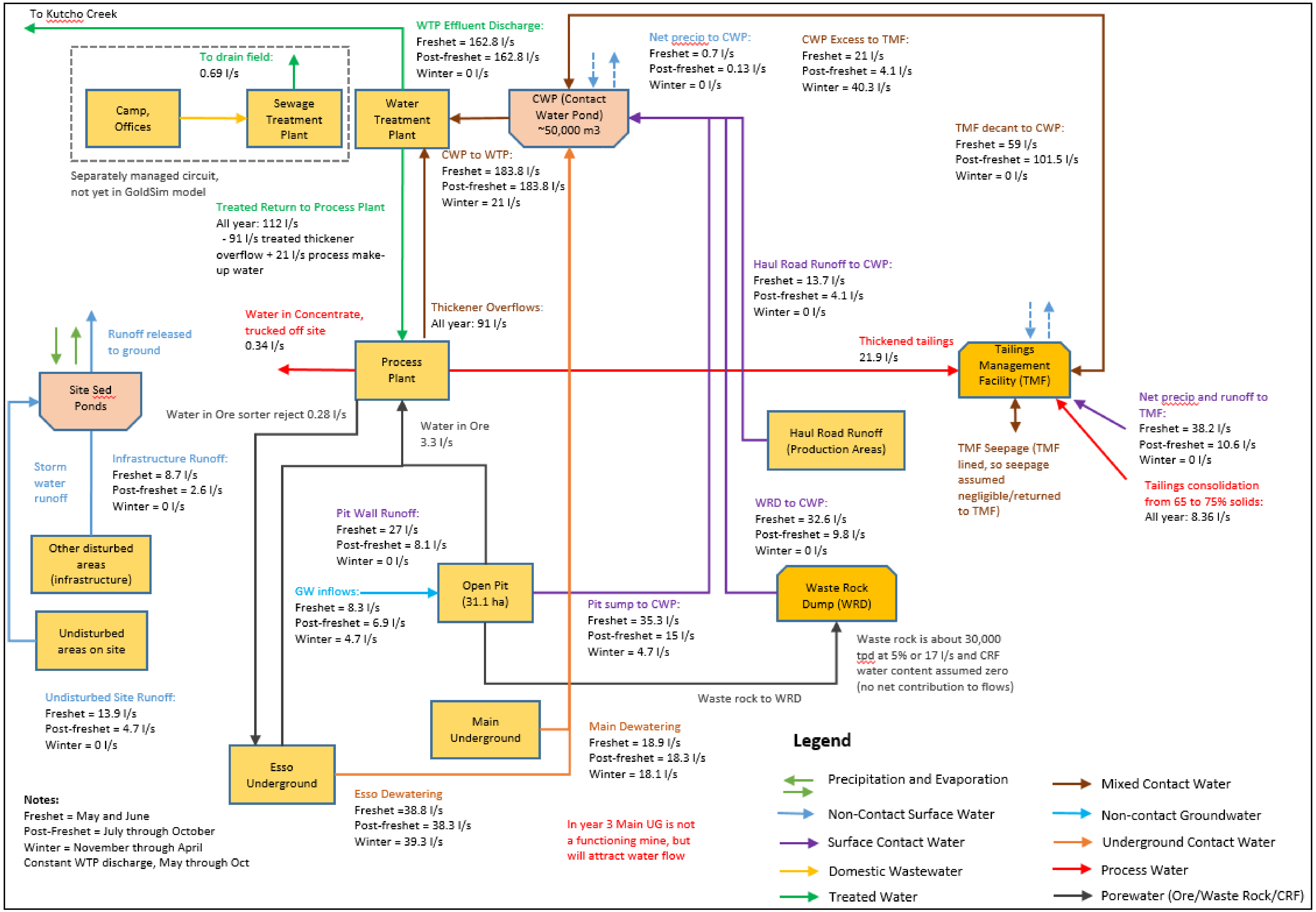


Figure 16-11: Kutcho Project Year 3 Site Water Balance, Average Climate

16.3.2 Numerical Groundwater Modelling

A hydrogeological data review was completed to assess the current level of hydrogeological understanding of the Kutcho project area. A conceptual hydrogeological model of the Project area was developed based on the review completed and this formed the basis for the numerical groundwater modelling completed.

A numerical groundwater flow model was developed to predict groundwater inflows to the Main open pit, Main underground and Esso underground mines throughout the life of mine and to inform dewatering requirements. The model was also used to predict groundwater level drawdown within the area immediately surrounding the proposed mine as a result of mine dewatering. The extent of the groundwater model, the model grid, topography and key groundwater monitoring boreholes are illustrated in Figure 16-12.

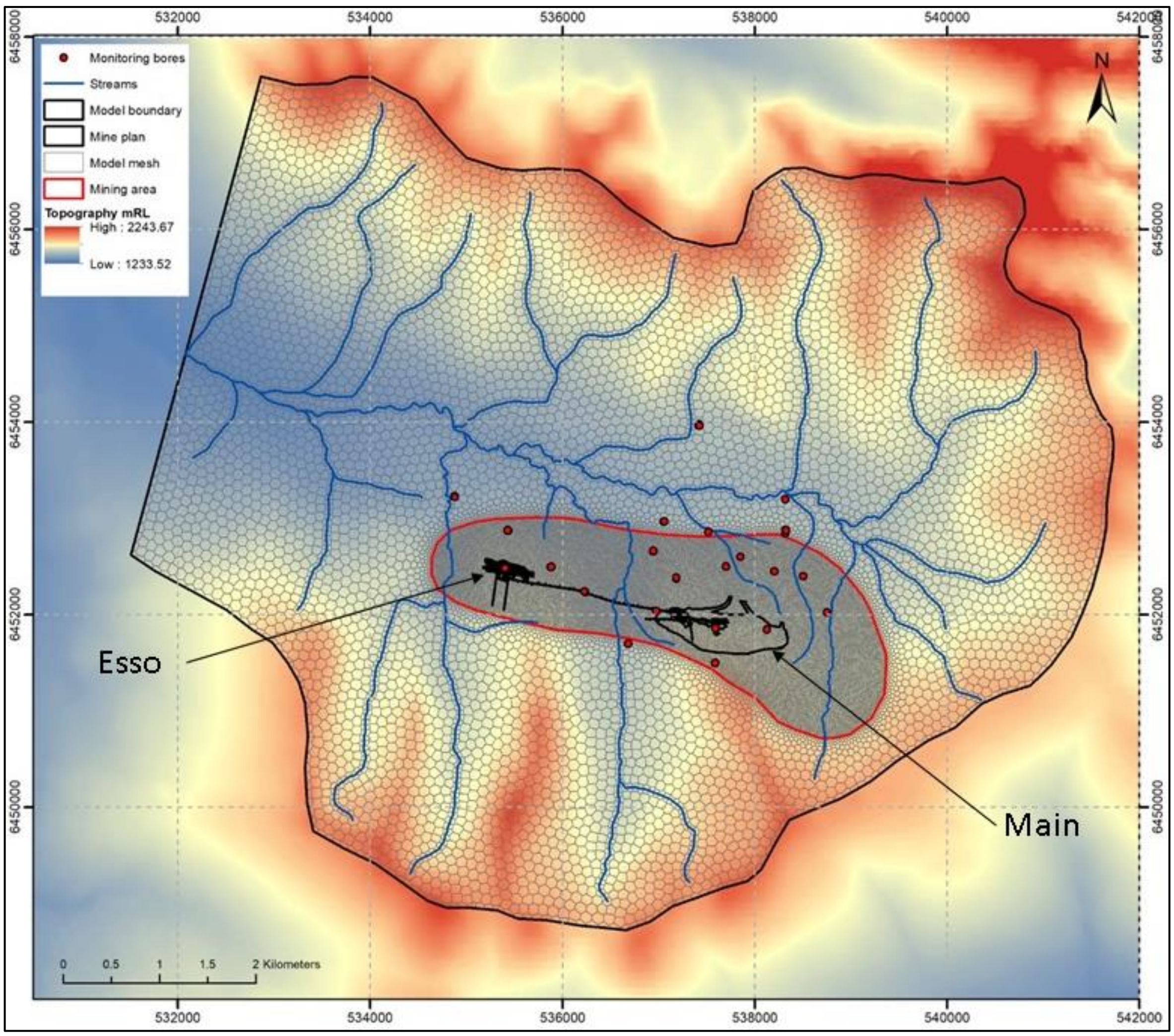


Figure 16-12: Numerical Groundwater Model Domain, Grid, Topography, and Monitoring Boreholes

The numerical groundwater flow model previously developed for the Kutcho Project in 2013 was upgraded based on newly available data, including additional data and insights on geology, hydrogeology (hydraulic properties, groundwater levels, etc.), hydrology (surface water flows) and climate. The modelling software was also updated from Visual Modflow to the more recent MODFLOW-USG with an AlgoMesh user interface. The model extends approximately 9 km west to east and approximately 8 km north to south, covering an area of roughly 72 km^2. The new mine plans were incorporated into the new groundwater model to assess the interaction between the mine development and the groundwater environment.

16.3.3 Mine Groundwater Inflows

The numerical groundwater model was used to predict groundwater inflows to the mine throughout the mine life. The model predicted inflows to the various components of the mine are illustrated in Figure 16-13.

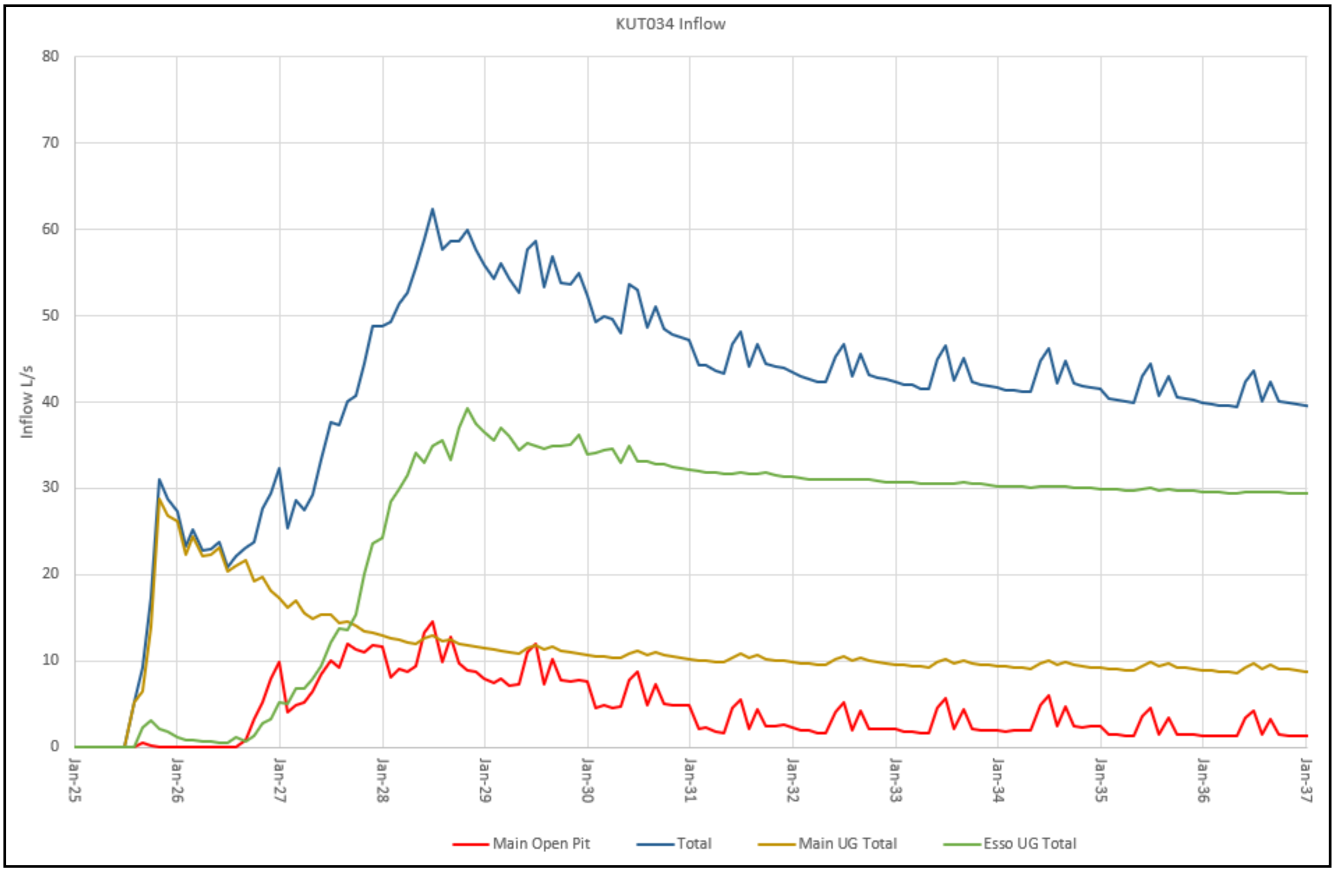


Figure 16-13: Predicted groundwater mine inflows

The groundwater model results suggest that mine groundwater inflows will increase as the mine extends towards its maximum depth. Mine groundwater inflows of up to approximately 30 L/s are predicted as the Main underground decline and ramp are developed (Year 1 of mining), increasing up to a peak total mine inflow rate of approximately 62 L/s over the next three years as mining proceeds at the Main open pit and with development of the Esso unground. Groundwater inflows gradually decrease after the mine attains its maximum aerial extent and over the remaining life of mine, decreasing from the peak of 62 L/s down to approximately 40 L/s.

Mine inflow rates are notably affected by the freshet (May/June snowmelt period) and fluctuate seasonally as a result. In the spring months (May/June) there is a sharp increase in mine groundwater inflow rates of approximately 3–5 L/s due to recharge from the snowmelt, followed by a similar decline as the freshet abates. This seasonal fluctuation is most obvious in the Main open pit inflow predictions. Inflow rates remain relatively steady over the winter months.

Structural lineaments (e.g. regional faults) represent zones of potential enhanced hydraulic connection, linking the surface water and the groundwater environments. In addition, the many drillholes within the deposit areas potentially link the surface water and groundwater environments. This combination of geological structures and open resource drillholes leads to discrete enhanced hydraulic connections between surface waters (snowmelt and rainfall runoff) and the underlying rocks resulting in the potential for oxidized elevated seasonal groundwater inflows to the mine workings, which should be incorporated into the mine dewatering designs.

16.3.4 Groundwater Level Drawdown

Natural groundwater levels will be drawn down in response to mine dewatering. Groundwater level drawdown will increase throughout the mine life, with the maximum drawdown roughly corresponding with the final stage of mining. The predicted water table drawdown will extend radially from the mine, except where influenced by geological structures, pinching out of lithologies, fault induced lithological boundaries, surface topography and/or recharge occurring along drainage channels/water bodies.

Modelling indicates that the extent of groundwater level drawdown within the shallower units will be generally restricted by the influence of surface watercourses which will act as recharge zones, reducing the amount of groundwater drawdown in these areas. The groundwater level drawdown is predicted to extend approximately 0.5–1.5 km from the Main open pit and directly above the Esso underground in the shallower/upper hydrogeological units (effectively the impact predicted to be observed in the near surface groundwater table). However, in the deeper lithologies (impacted primarily by mine dewatering of the deep Esso underground mine) the impact on groundwater is predicted to extend approximately 2–3 km from the mine areas. The predicted groundwater drawdown impact in these deeper rocks is more conical in distribution and extends further than that predicted in the upper units, as a result of less complexities and less influence of surface topography, creek leakage and recharge at this depth.

16.4 Open Pit Mining

Section 15 details the Mineral Reserve estimation, pit optimization, pit design, mining dilution, ore loss and geotechnical information specific to the open pit design. Where relevant to the mining methods, additional details are included with each described mining activity. There is only one open pit which is located on the Main deposit.

16.4.1 Material Physical Properties

The material physical properties and assumptions used in the estimation of mining equipment requirements and the mine plan are summarized in Table 16-9. Density was estimated directly into the Mineral Resource block model but average values of the ore or waste were used where stockpile movements were estimated. Moisture content was not measured; the applied value is based on other similar deposits. Similarly, the swell factor is based other similar deposits; it was applied to ore and waste equally. As a result of the material properties truck and loader capacities are tonnage limited and not volume limited.

Table 16-9: Material physical properties

Parameters	Units	Value
Dry bulk density – ore (average within pit)	t/m^3	3.70
Dry Bulk density – waste (average within pit)	t/m^3	2.77
Moisture content	%	5%
Swell factor	%	35%
Ore mining loss estimated	%	7%
Ore dilution estimated	%	9%

Ore mining loss and dilution are detailed in Section 15.

16.4.2 Mining Methods

Mining will be by conventional open pit drill and blast followed by load and haul. The mining cycle commences with mine planning based on the Mineral Reserve model and long-term material movement requirements. Areas ore and waste blasting conducted on 5 m benches will be designated. Bulk waste blasting will be on 10 m benches. Where blasthole drilling comes close to or intersects the mineralization drill cutting will be sampled and assayed for grade control. Grade control sampling will be used to refine ore block geometries leading to ore/waste definition and blast pattern modification to minimise dilution.

Two types of loading equipment were selected for the mine. Wheel loaders with 10 m^3 capacity buckets will mine the bulk waste and some of the ore/waste benches where ore differentiation is simple. A tracked excavator with a 4 m^3 bucket was selected for pit wall cleaning and mining in ore when the ore/waste geometry is complex. The excavator was selected for its ability to mine on a 5 m bench in combination with the 90-tonne class rigid body haul truck. For this reason, the 5 m x 5 m x 5 m block size was considered appropriate for definition of the SMU and no further dilution is anticipated. Blasting on 10 m benches will need to consider the front-end loader in order to maintain a safe recovery spoil height and blasts will be designed to limit heave.

The sequencing of drilling, grade control, blasting, water control and excavation will require expert planning and operations co-ordination. It will be imperative that the mining team are oxidized on grade delivery to the plant whilst maintaining all other mining metrics.

The compact nature of the operation and small number of mining equipment means that an automated truck dispatch system will not be required. Management of hauling from source to destination will be via a centralized control room equipped with computer equipment, telephone network and a CB Radio TX/RX system.

Pit wall stability monitoring will be completed using laser distance measuring to fixed prisms. Real time monitoring equipment like radars were not considered to be required.

The mine will operate on a 355-day per year basis (10 scheduled days of outages) with two 12-hour shifts. Shift idle time is provided in Table 16-10.

Table 16-10: Shift idle time

Open pit equipment idle time	Hours per shift
Blasting	0.25
Breaks/Meals	1.00
Shift change	0.50
Miscellaneous (moves, fuel, power, cleanup, idle)	0.50
Weather	0.33
Total Operating Delays	**2.58**

16.4.2.1 Drill and Blast

Rock fragmentation will be accomplished through blasting. There may be small amounts of free-dig or rippable material in the upper 5–15 m of the pit, but this was not incorporated into planning considerations. The only exception is that initial road cuts less than 5 m in depth are assumed to be rippable. This assumption applies to capitalized road construction costs.

The industry-accepted Kuz-Ram blast fragmentation prediction model, informed by geotechnical data (Terrane, 2021), was used to design blast patterns. The fragmentation model predicts that 8% of the shot material is estimated to be larger than 400 mm and less than 0.1% will be above 750 mm for the 5 m benches. The small proportion of oversize material will be reduced in size by the rock breaker installed at the primary crusher. For the 10 m waste benches, oversize material was less important in shot design, with the size limitation being the loader bucket size. Table 16-11 lists the parameters for 5 m benches (50%: 50% ore: waste mix), 10 m benches (bulk waste) and pre-splits. The 10 m benches are entirely within hangingwall rocks. The 5 m benches are by majority in the footwall and ore rocks. The blasting parameters reflect the majority rock units for each bench height. The blast pattern designs assume that a staggered pattern is utilized.

Drilling for blasting will be completed by two diesel-powered, 287kW, down-the-hole (DTH) hammer drill rigs. The rigs are capable of pre-split and regular production blasthole drilling for 90–165 mm (3½-6") diameter holes Table 16-11 shows drilling penetration rate based on experience and geological and geotechnical information.

Table 16-11: Blast pattern parameters and drilling penetration rate estimation

Drilling	Units	5 m benches	10 m benches	Pre-split hangingwall
Bench height	m	5.0	10.0	10.0
Hole diameter	mm	127	154	75
Sub-grade drilling	m	0.8	0.9	
Hole length	m	5.8	10.9	10.35
Burden	m	3.0	3.7	
Spacing	m	3.3	4.5	0.75
SG	T/BCM	3.70	2.77	
Yield per blast hole	tonnes	183	461	
Charge Length	m	3.5	6.0	
Stemming Length	m	2.3	4.9	
Charge density	kg/m^3	0.77	0.56	
Charge weight per hole	kg/hole	37.7	93.0	
Powder factor	**kg/t**	**0.22**	**0.20**	
Yield per metre drilled	t/m	31.6	42.3	
Yield per metre drilled	m/m			13.80
UCS	mPa	35.0	70.0	70.0
Job rating		75%	75%	75%
Nominal penetration rate	m/hr	67.0	59.5	59.5
Drilling time per blast hole	min	5.2	11.0	10.4
Moving, align time	min	2.0	2.0	3.0
Collaring	min	1.0	1.0	1.0
Rod change	min	0.0	1.0	2.0
Grade control sampling time	min	1.5	0.0	0.0
Total cycle time per hole	min	9.7	15.0	16.4
Penetration rate	**m/hr**	**36**	**44**	**38**

ANFO was planned to be used for all production blasts which provides environmental savings through reduction of nitrate contamination of contact water. Maintaining all shots with ANFO will require dedicated effort in water management before shot loading. For contingency planning two bulk storage silos were provided in the magazine area to give flexibility in product storage.

Equipment for removing water from the blast holes prior to charging was provided for.

Blasting equipment (all leased) includes:

- 2 x 80-tonne AN prill/emulsion silos
- Blast truck
- Front end loader/forklift
- Loader-side dump for stemming
- Explosives truck
- Pick-ups
- Dewatering truck.

The blasting crew was estimated to require four blasters, two helpers with two crews. An engineer and foreman will be responsible for supervision and design.

Pre-split blast design was developed on the basis hangingwall crest length. Scheduling of the pre-split drilling campaigns required consideration of timing of crest exposure and the limitation of drilling operational hours available per period.

Pre-split holes will be detonated with a 50% decoupling ratio using emulsion or emulsion-packaged explosives. It is recommended that the pre-splits be fired in conjunction with a 10–15 m buffer blast at the final wall position. Footwall trim shots will be required to reduce wall vibration and damage, but no pre-splitting will occur.

Table 16-12 sets out the site ANFO consumption. The open pit averages an ANFO consumption of 0.21 kg/t of rock mined of while the underground mine will average 0.69 kg/t.

Table 16-12 ANFO consumption per annum

ANFO		Total	-2	-1	1	2	3	4	5	6	7	8
Open pit	tonnes	19,838	114	2,348	2,824	2,805	2,372	1,676	2,623	2,600	2,094	384
Underground	tonnes	2,432	70	323	295	492	269	235	234	232	232	49
Total	**tonnes**	**22,271**	**184**	**2,671**	**3,119**	**3,297**	**2,641**	**1,911**	**2,857**	**2,832**	**2,326**	**433**

16.4.2.2 Grade Control

Review of the block model indicates that ore and waste are intermingled but that sampling of blast holes for grade control can be limited to a 1:1 waste:ore ratio. One sample will be taken per hole. Mining in ore areas constitutes only 20% of the activity time in the pit and as such the grade control process will require a crew of two technicians, a grade control geologist, and overseen by a senior geologist. It will also be possible to manage ore mining on day shift to allow for maximum verification and control. The cost of grade control is included in mine management operating costs. Grade control samples are to be assayed at an off-site laboratory.

16.4.2.3 Ore and Waste Loading

Loading of ore and waste into 90-tonne class rigid body, off-highway trucks will be performed by two 10 m^3 capacity bucket wheel loaders and a 4 m^3 capacity tracked excavator. The two 10 m^3 wheel loaders will also be used for stockpile and waste rehandle. Table 16-13 includes the loader productivity factors for a range of material types and loading conditions utilized in the mine production schedule.

Table 16-13: Loader productivity parameters

Parameter	Units	Wheel loader (10 m^3)					Wheel loader (7 m^3)		Excavator (4 m^3)	
		Ore	Waste	Long term stock	Topsoil	ROM (no truck)	ROM (no truck)	Conc.	Ore	Waste
Bucket capacity (heaped)	m^3	10.7	10.7	10.7	10.7	10.7	6.0	6.0	4.0	4.0
Material weight	dmt/bcm	3.23	2.77	3.70	2.00	3.70	3.70	4.50	3.70	2.77
Bulk (swell) factor		1.35	1.35	1.35	1.50	1.35	1.35	1.35	1.35	1.35
Dry bulk density	dmt/lcm	2.40	2.05	2.74	1.33	2.74	2.74	3.33	2.74	2.05
Moisture	%	5%	5%	5%	20%	5%	5%	9%	5%	5%
Wet bulk density	wmt/lcm	2.52	2.16	2.88	1.67	2.88	2.88	3.66	2.88	2.16
Fill factor	%	90%	90%	75%	90%	75%	85%	65%	90%	90%
Tonnes/pass	wmt	24.3	20.8	23.1	16.1	23.1	14.7	14.3	10.4	7.8
Maximum weight	wmt	21.7	21.7	21.7	21.7	21.7	14.5	14.5	8.0	8.0
Truck or bucket capacity	wmt	90.0	90.0	90.0	70.0	21.7	14.5	14.5	90.0	90.0

Parameter	Units	Wheel loader (10 m³)					Wheel loader (7 m³)		Excavator (4 m³)	
		Ore	Waste	Long term stock	Topsoil	ROM (no truck)	ROM (no truck)	Conc.	Ore	Waste
Average # passes for full cycle	passes	4.0	4.0	4.0	4.0	1.0	1.0	3.0	9.0	12.0
Truck spot time	sec	30	30	30	30	0	0	300	30	30
First bucket cycle time	sec	15	15	15	15	90	90	90	15	15
Subsequent bucket cycle time	sec	42	42	42	60	42	42	42	42	42
Loader productivity in loading cycle	%	90%	90%	90%	90%	90%	90%	90%	90%	90%
Load time per truck or bucket	min	3.17	3.17	3.17	4.17	2.44	2.44	8.00	7.06	9.39
Maximum productivity	trucks/hr	18.9	18.9	18.9	14.4	24.5	24.5	7.5	8.5	6.4
Maximum productivity	wmt/hr	1,705	1,705	1,705	1,008	533	356	109	765	575
Truck availability to loading unit	%	80%	80%	80%	80%	na	na	82%	80%	80%
Productivity	**dmt/NOH**	**1,296**	**1,296**	**1,296**	**645**	**267**	**267**	**81**	**582**	**437**
Truck payload weight limited	wmt	90.00	90.00	90.00	90.00	na	na	42.00	90.00	90.00
Truck payload weight limited	dmt	85.5	85.5	85.5	72.0	na	na	38.2	85.5	85.5

Two 7 m^3 wheel loaders will be used for primary crusher feed and concentrate loading. Included in the ancillary fleet is a wheel loader with a multi-fitting coupling that can allow it to be used either as a 7 m^3 wheel loader or wheel dozer (primary use) adding further flexibility.

16.4.2.4 Ore and Waste Hauling

Ore and waste haulage will be performed by 90-tonne class, rigid body, off-highway haul trucks. Haul truck productivity was estimated using truck productivity software. Haul road profiles were generated at 10 m or 20 m increments for each pit phase. The software was then used to estimate truck productivity given the loading units and physical characteristics for the ore and waste. Stockpile rehandle productivities were similarly estimated. Haul route profiles were digitized in Deswik 3D CAD software and laded to the haulage software to automatically generate the haulage elevation profiles. Speed limits for curves, intersections and differing rolling resistance factors for permanent (3%) or temporary (5%) road surfaces were added to the productivity software. The truck speed was limited to 40 kph.

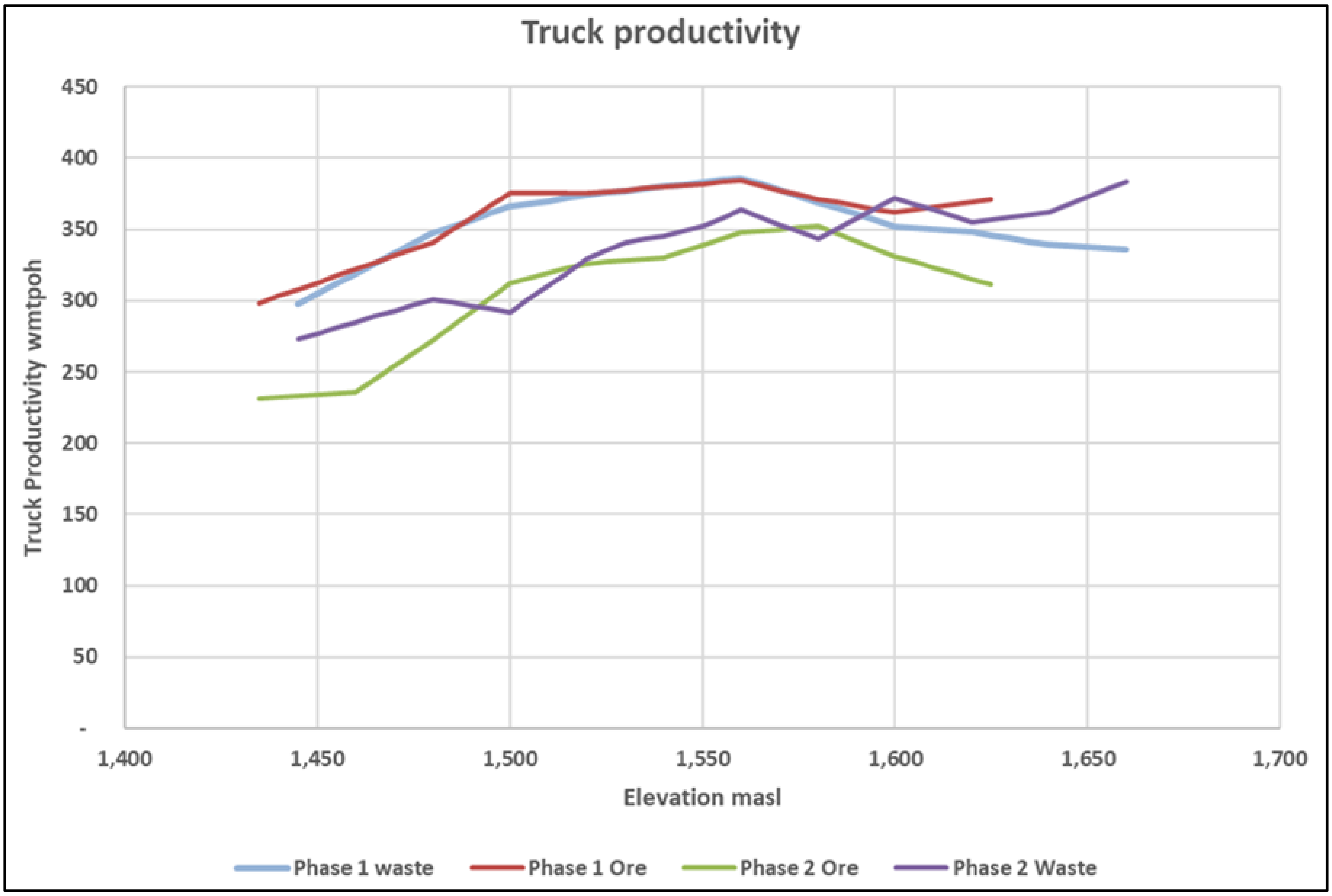


Figure 16-14: Truck Productivity as a Function of Bench Elevation (wet metric tonnes per operating hour)

A fleet of up to eight trucks will be required although the number of trucks in service will vary depending on the ore and waste mining dictates. Rotation of equipment during periods of lower production requirements will ensure the fleet ages evenly.

16.4.2.5 Ancillary Equipment

Dumping and bench clean-up will be undertaken by two 350 kW dozers and a 370 kW multi-purpose wheel loader with a dozer blade attachment.

Mine haul roads, service roads and the main access road will be undertaken by:

- One 35-tonne articulated dump truck
- One articulated 35,000-L water truck
- One vibratory roller (road and TMF embankment compaction)
- Two 220-kW graders
- One snow plough
- One 175-kW tracked dozer
- One tool carrier
- One utility backhoe (2 m^3 loader bucket and 0.3 m^3 excavator bucket).

Maintenance equipment for the site includes:

- One tyre handler
- One tractor and lowboy trailer (for tracked equipment – 50-tonne capacity)

- One fuel and lube truck (11,300-L diesel capacity)
- One mechanical service truck (20-tonne)
- One welder truck (4-tonne)
- Two field forklifts
- One shop forklift
- One 40-tonne mobile crane.

Emergency response equipment includes an ambulance and fire truck.

Other equipment:

- Eight lighting towers
- 12 crew cab pick-up trucks
- Two crew buses.

Blasting equipment is described above.

16.4.3 Production Schedule

The open pit production schedule recognises the contribution from the underground mines and the plant crusher throughput capacity. The development of the underground ore supply schedule is discussed in Section 0. Underground mining, given the higher grade of the ore, is given priority over open pit ore to plant feed. Consequently, the open pit schedule was developed after allowing for underground ore supply. Open pit crusher feed gives priority to high-grade sulphide ore, then the oxidized ore and finally low-grade ore. The distinction between low-grade and high-grade sulphide ore varies throughout the open pit mine life in order to maximise project NPV and to ensure the high-grade stockpile does not exceed 250 kt.

Process plant tonnage throughput was modified for start-up as shown in Table 16-14.

Table 16-14: Crushing circuit throughput modification during start-up

Period from start-up	Q1	Q2	Q3	Q4	Onwards
Throughput factor	50%	85%	95%	99%	100%

Figure 16-15 shows the plant crusher feed tonnage and sources of ore for a feed rate of 4,500 tpd.

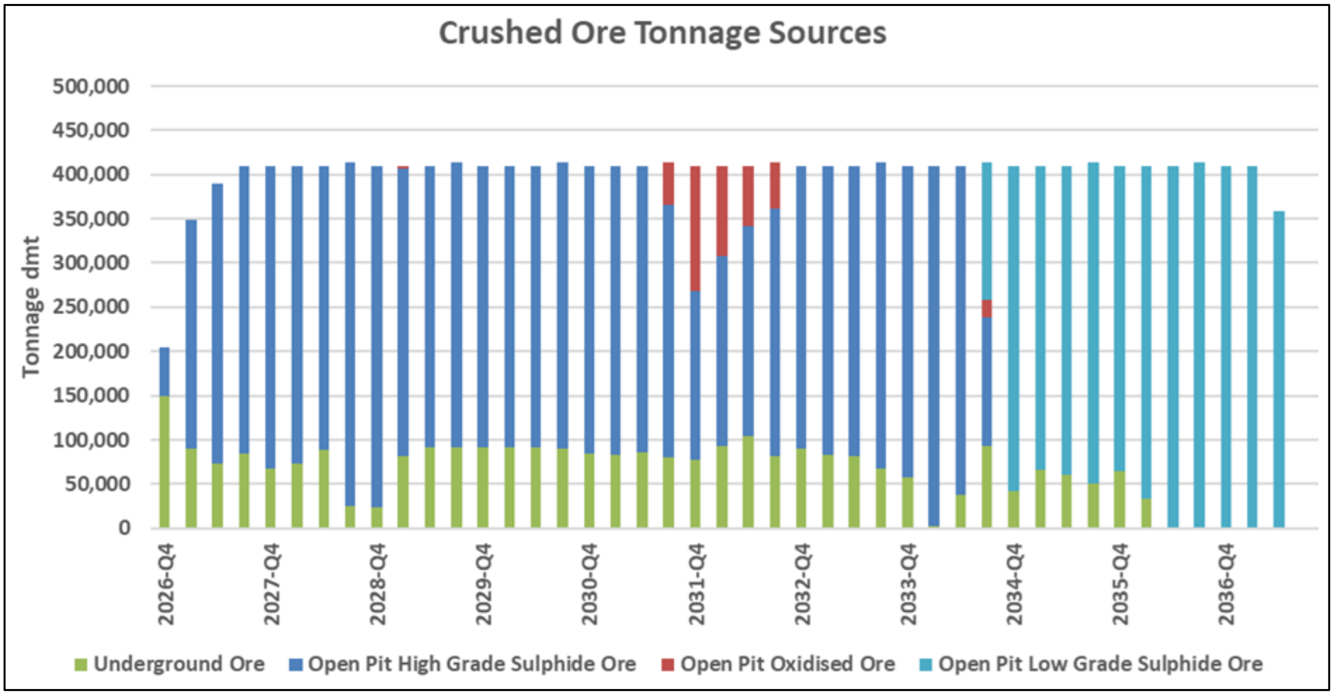


Figure 16-15: Quarterly Crusher Feed Tonnages and Sources of Ore

The variation in NSR value differentiating high-grade and low-grade sulphide ore and waste is shown in Figure 16-16. The NSR cut-off increases when high-grade ore is plentiful at the end of Phases 1 and 2 whereas the cut-off grade is lower at the start of operating life and at the commencement of Phase 2 of the availability of open pit ore is limited.

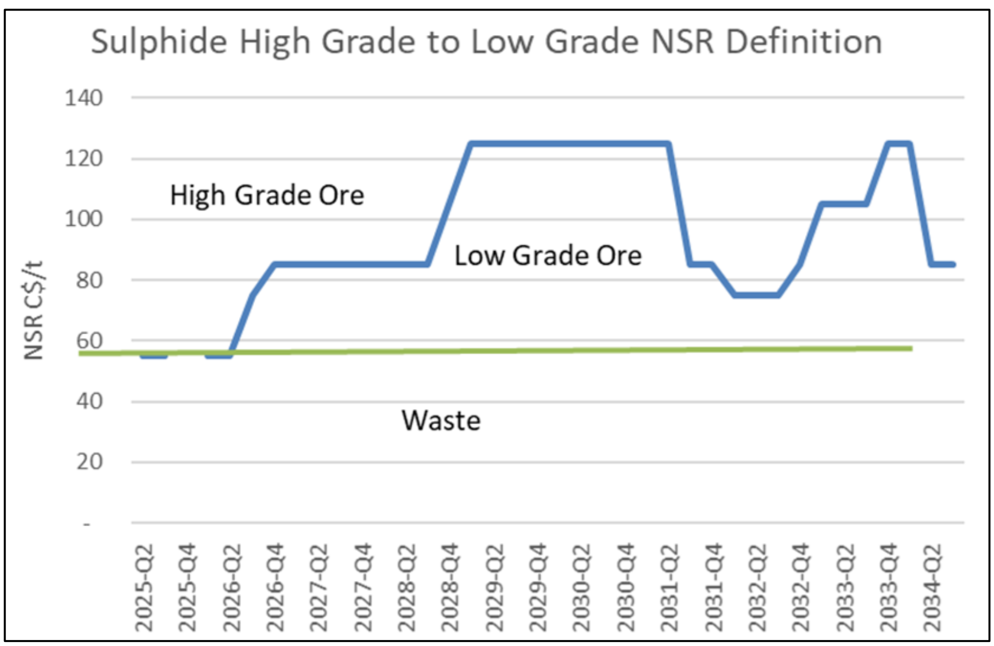


Figure 16-16: NSR Differentiation Between High Grade, and Low Grade Ore, and Waste

The high-grade stockpile is maintained at less than 275 kt. When the high-grade stockpile is empty, top-up feed to the plant is maintained from the low-grade stockpile, which prioritises the higher-grade oxidized ore before low-grade sulphide ore.Figure 16-17 shows the end-of-period (EOP) high-grade stockpile levels over the life of the operation.

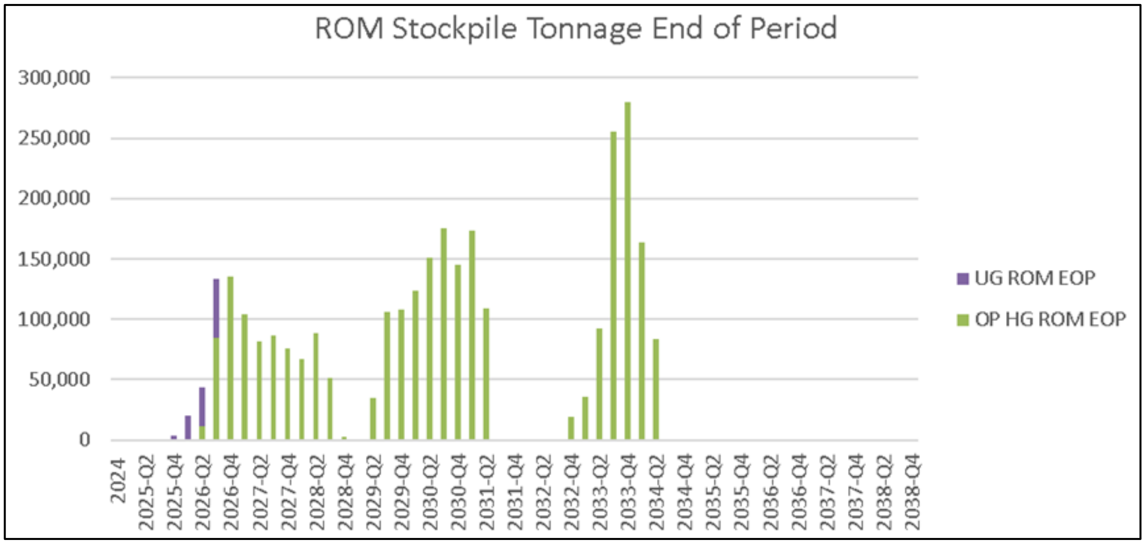


Figure 16-17: End-of-Period High-Grade Stockpile Levels

The low-grade stockpile contains two ore types – oxidized ore and low-grade sulphide ore. The higher value oxidized ore is consumed by the time the Phase 2 pit commences. The low-grade stockpile has a maximum capacity of 4.4 Mt. Figure 16-18 shows the quarterly end-of-period levels for all ore stockpiles.

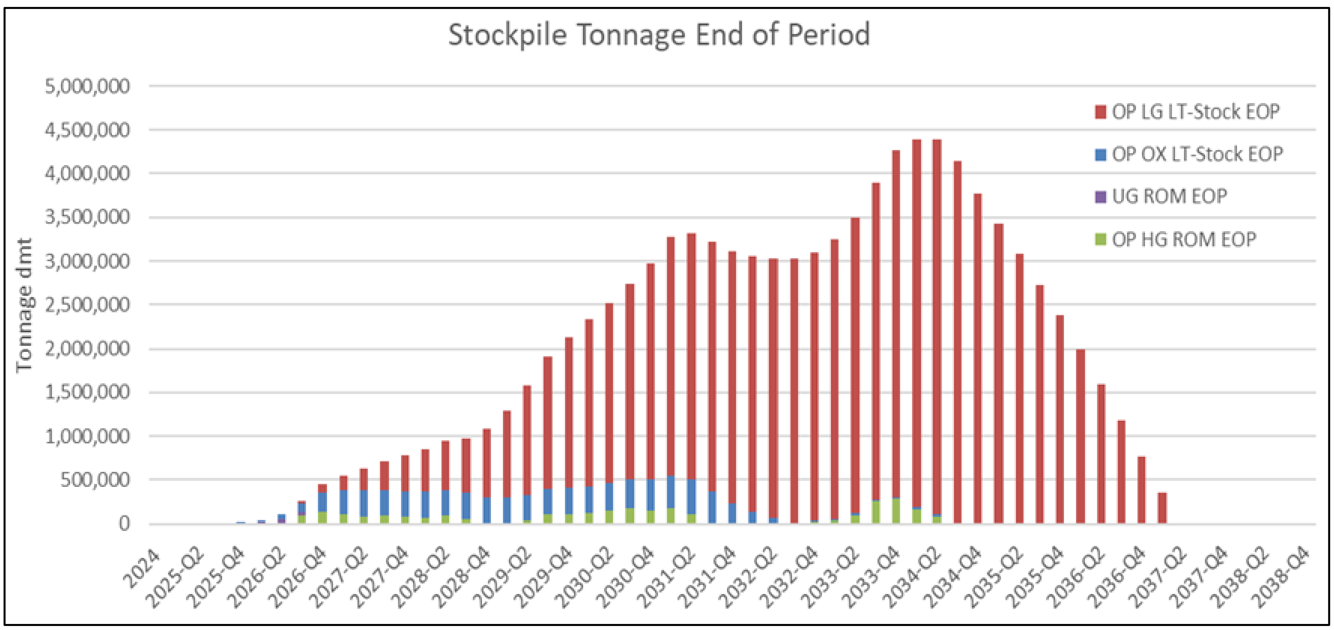


Figure 16-18: End-of-Period Ore Stockpile Tonnages

Truck productivities for hauling ore to the crusher and stockpiles, and rehandling stockpiled ore to the crusher, as well as waste rehandling, and topsoil haulage determined truck haulage hours for scheduling. An adequate number of trucks were then allocated per period and by iteration, a material handling solution found to maximize the NPV. The resulting material handling schedule is shown in Figure 16-19.

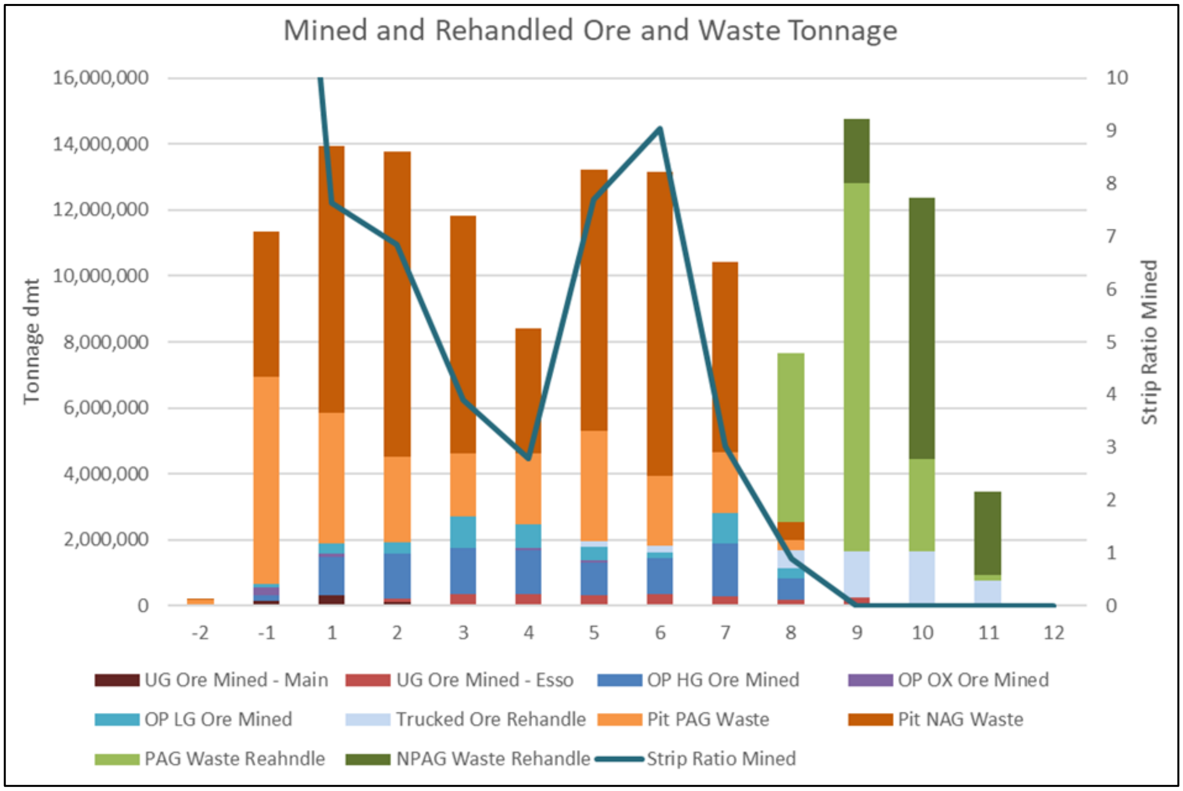


Figure 16-19: Annual Ore and Waste Production Schedule

The material handling schedule is also limited by the bench sink rate and associated non-productive activities when commencing new benches, such as building sumps, ramps, and issues of working in confined working areas. The maximum sink rate allowed in the schedule was 60 m per annum or one bench per month. This limitation affects the commencement and termination of each pit phase. A total of six quarters of production (17% of the open pit life) are affected which is considered to be reasonable.

Equipment operating hours are discounted by the product of mechanical availability and utilization. Scheduling allowed for reduced mechanical availability as a function of equipment age. Mechanical availability typically commenced at 90% and reduced to 85% at the end of equipment life.

The number of trucks in service fluctuates as a function of strip ratio of waste to high grade ore. The high-grade ore definition value changes through time also depending on strip ratio, but also as an aid in maximizing NPV. Table 16-15 shows the major equipment counts in service by year.

Table 16-15: Open pit mobile equipment by year

Equipment	-2	-1	1	2	3	4	5	6	7	8	9	10	11
Drills	1	1	2	2	2	2	2	2	2	1	-	-	-
Wheel loader 10 m³	1	2	2	2	2	2	2	2	2	2	2	2	1
Excavator 4 m³	1	1	1	1	1	1	1	1	1	1	-	-	-
Wheel loader 7 m³	-	1	2	2	2	2	2	2	2	2	2	2	1
90-t haul truck	2	7	8	8	8	6	8	8	8	5	8	7	3
Tracked dozer	1	2	2	2	2	2	2	2	2	1	1	1	1
Wheel dozer	1	1	1	1	1	1	1	1	1	1	1	1	1
Grader	1	2	2	2	2	2	2	2	2	2	2	2	1
Other ancillary	9	19	37	37	37	38	39	39	39	35	32	32	17
Total	**14**	**44**	**58**	**58**	**58**	**57**	**60**	**60**	**60**	**57**	**54**	**48**	**25**

Diesel consumption per operating hour for open pit equipment was based on OEM specifications with due modifications for equipment duty. Open pit mining equipment is the largest consumer of diesel. Project diesel consumption is provided in Table 16-16.

Table 16-16: Project diesel consumption per annum (million litres)

Diesel (ML)	Total	-2	-1	1	2	3	4	5	6	7	8	9	10	11
Open pit	59.6	0.3	4.5	5.9	6.0	5.8	4.8	6.2	6.2	5.7	3.7	5.1	4.3	1.1
Underground	10.4	0.1	1.1	1.5	1.4	1.1	1.1	1.1	1.1	0.6	0.9	0.5	-	-
Plant	1.3	-	0.0	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1
Power plant	4.0	1.2	1.7	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1
Other	7.5	-	4.0	2.5	0.3	-	-	0.3	-	0.3	-	-	-	0.3
Total	**82.8**	**1.6**	**11.4**	**10.1**	**7.8**	**7.1**	**6.1**	**7.8**	**7.5**	**6.8**	**4.8**	**5.8**	**4.5**	**1.5**

Note: Other includes TMF contractors' equipment and other plant and site infrastructure construction equipment.

16.4.3.1 Pre-Stripping

The pre-production period is scheduled to allow both the underground and open pit mines to attain steady state throughput within Year 1. Approximately 11.5 Mt of waste will be mined during this period. It is estimated that this waste will contain 4.4 Mt of nPAG waste of which 2.5 Mt is planned to be directed to the TMF Phase 1 embankment. The rest of the open pit nPAG waste will be used to as an under-liner pad at the PAG WRD. Any remaining waste will be placed on the WRD.

During the pre-production period, 503 kt of ore is to be mined from the open pit: 190 kt of high-grade sulphide ore, 213 kt of oxidized ore (stockpiled for later processing), and 100 kt of low-grade sulphide ore that is placed on the long-term low-grade stockpile. A further 150 kt of high-grade sulphide ore is planned to be mined from the Main underground mine during this period. It is planned that 204 kt of ore will be processed in the plant during the pre-production commissioning period.

16.4.4 ***Production Plans***

The phased mining development plans are presented in Figure 16-20 through to Figure 16-27. These plans start at the end of Year -2 (2025) and progress through to the rehabilitated landform in Year 12 (2038).

16.4.4.1 Year -2 Activities

The site access road will require up to two years of construction (estimated to be 2023 and 2024 or otherwise described as Years -4 and -3) to allow access for site construction equipment and materials. Some minor access road construction activities will occur in 2025 (Year -2).

The first major site construction year (Year -2) is scheduled to include:

- Development of all site roads and initial mine roads
- Clearing, grubbing and topsoil removal for 42% of the project area and completion of earthworks for the plant and commencement of foundation works
- Installation of the diesel power generator and power distribution for construction (later to be the operational back-up unit)
- Construction of the mine workshops, camp accommodation facilities, establishment of the mobile crushing plant (for TMF, initial underground stope rock fill and plant construction materials); water diversion channels, contact water pond, sedimentation ponds and the water treatment plant.

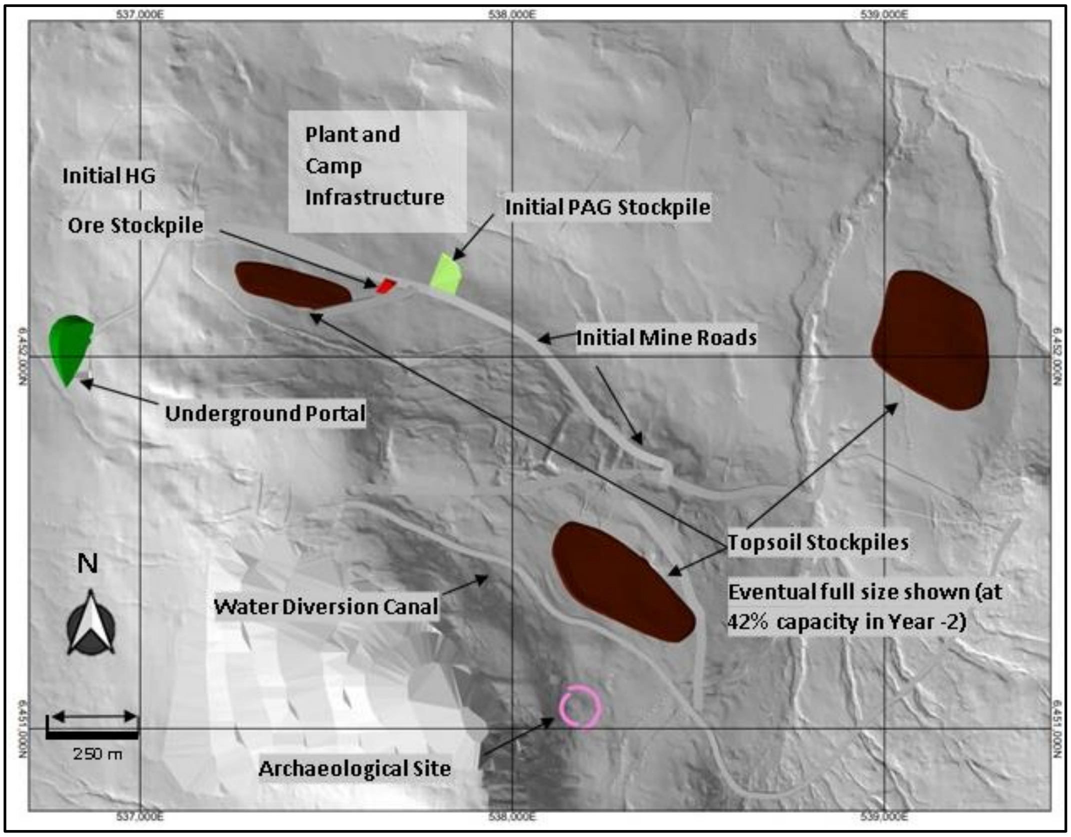


Figure 16-20: End of Year -2 Project Site Development

16.4.4.2 Year -1 Activities

The second year of site construction activities is scheduled to include:

- Clearing, grubbing and topsoil removal for 24% of the project area (for a progressive total of 66%)
- Completion and commissioning of the process plant
- Office complex construction
- Completion of and commissioning of LNG power plant
- Construction of the TMF Phase 1 to allow commencement of ore treatment
- Mining in the Phase 1 pit, targeting primarily waste rock for the TMF and sufficient ore to commission the process plant
- Underground ore mining at Main at full capacity and access development to Esso underway
- Decommissioning of construction camp extension.

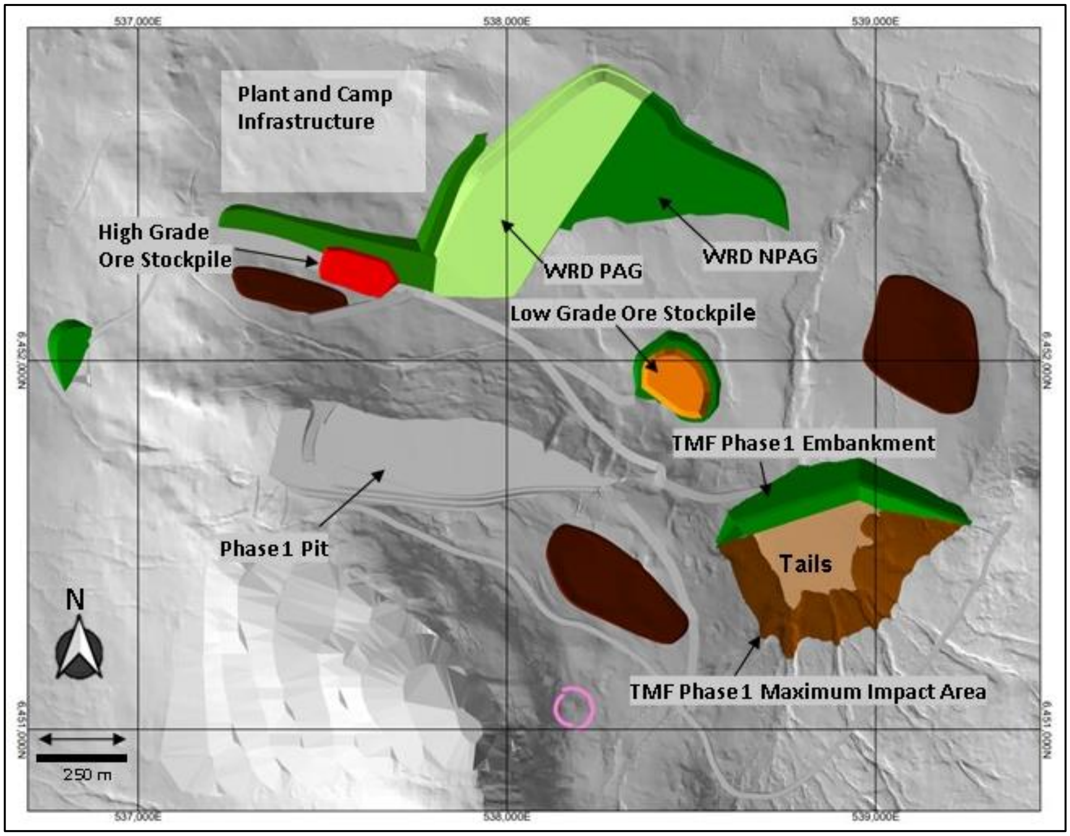


Figure 16-21: Development End of Year -1 Project Site

16.4.4.3 Years 1 and 2 Activities

The first two years of operational activities is scheduled to include:

- Clearing, grubbing and topsoil removal for 15% of the project area (progressive total of 81%)
- Process plant at full capacity and at steady state operations
- Construction of the TMF embankment Phase 2
- Mining of the Phase 1 pit, which to the end of Year 2 remains free draining (pumping is required from Year 3 onwards)
- Development of access ramps to Phase 2 to allow operational flexibility if required
- Access to Esso is completed, first Esso ore is produced, and Main underground terminates.

16.4.4.4 Years 3 to 5 Activities

The third to fifth years of operation is scheduled to include:

- Clearing, grubbing and topsoil removal for the remaining 14% of the project area (progressive total of 95%)
- Esso underground access development ceases and stope mining is at steady state mid-life capacity
- Phase 3 TMF embankment is completed
- Mining the Phase 1 pit completes and Phase 2 mining commences
- Commencement of Phase 1 in-pit backfill with waste rock.

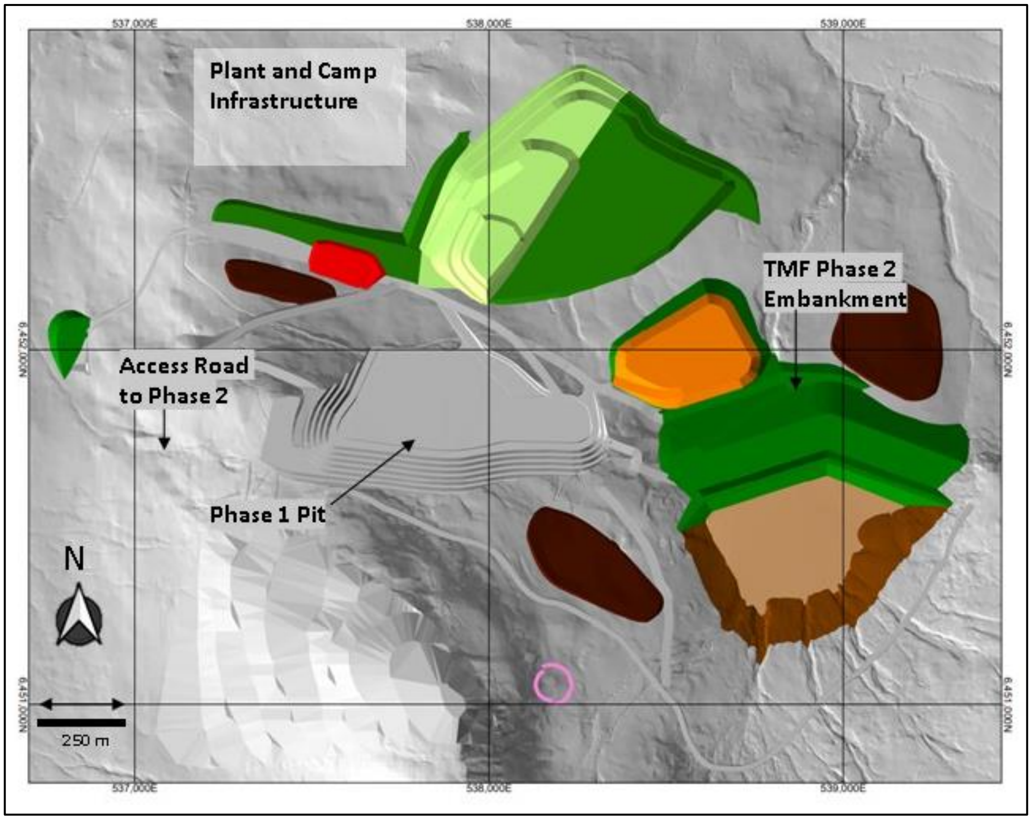


Figure 16-22: End of Years 3, 4 and 5 Activities

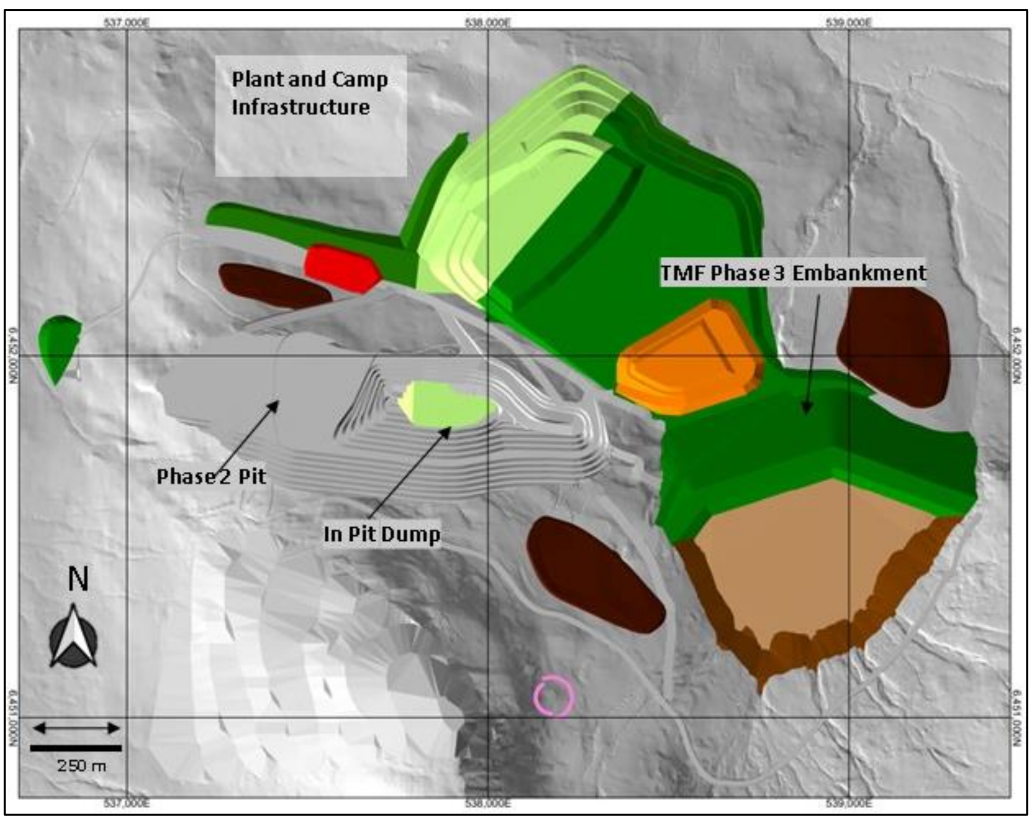


Figure 16-23: End of Year 5 Project Site Development

16.4.4.5 Years 6 and 7 Activities

The sixth and seventh year of operation is scheduled to include:

- Clearing, grubbing and topsoil removal for the remaining 5% of the project area (progressive total of 100%)
- Phase 2 mining continues to near termination
- The low-grade stockpile is at maximum capacity
- Backfilling pit dumping in the Phase 1 pit approaches full capacity with capping of the PAG with nPAG material commencing
- Development of the WRD is at its maximum elevation
- The final Phase 4 TMF embankment is completed.

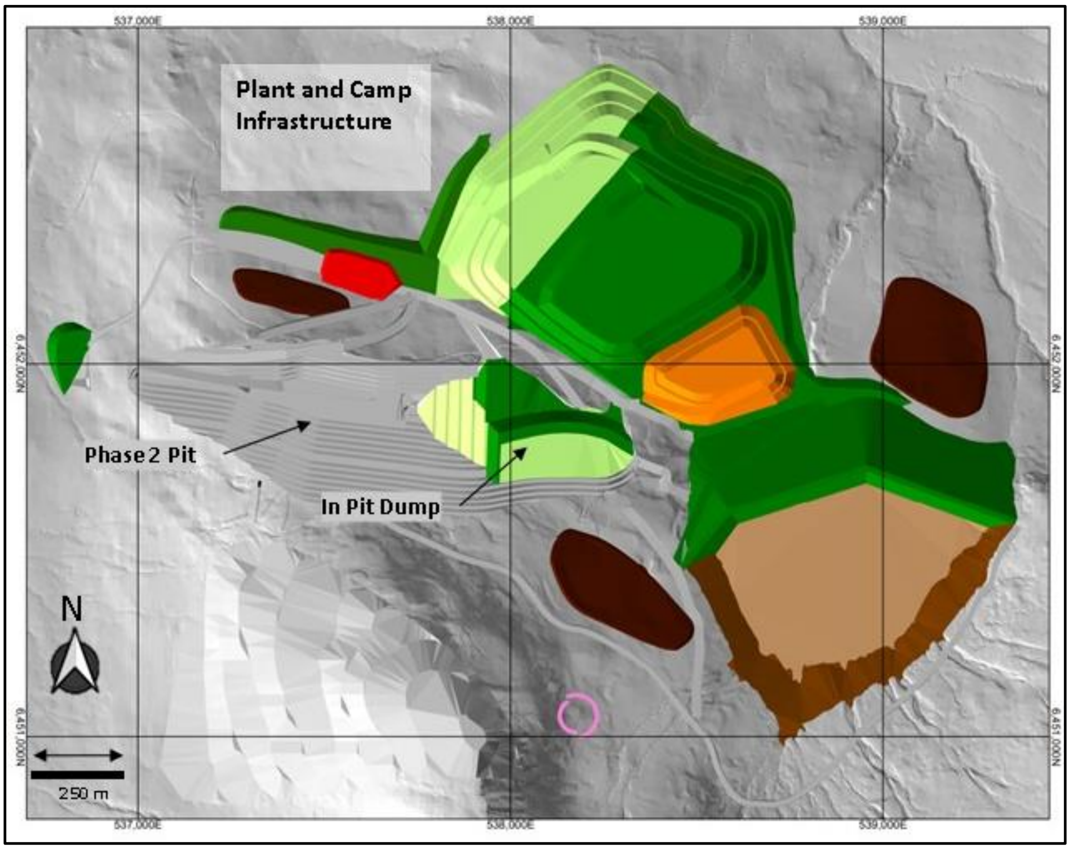


Figure 16-24: End of Year 7 Project Site Development

16.4.4.6 Years 8 and 9 Activities

Operational Years 8 and 9 are scheduled to include:

- Termination underground mining activities and closure
- Termination of open pit pumping
- Termination of Phase 1 pit backfill
- Continued progress with Phase 2 backfill with progressive nPAG cover
- Reclamation of the low-grade stockpile as the only source of process plant feed.

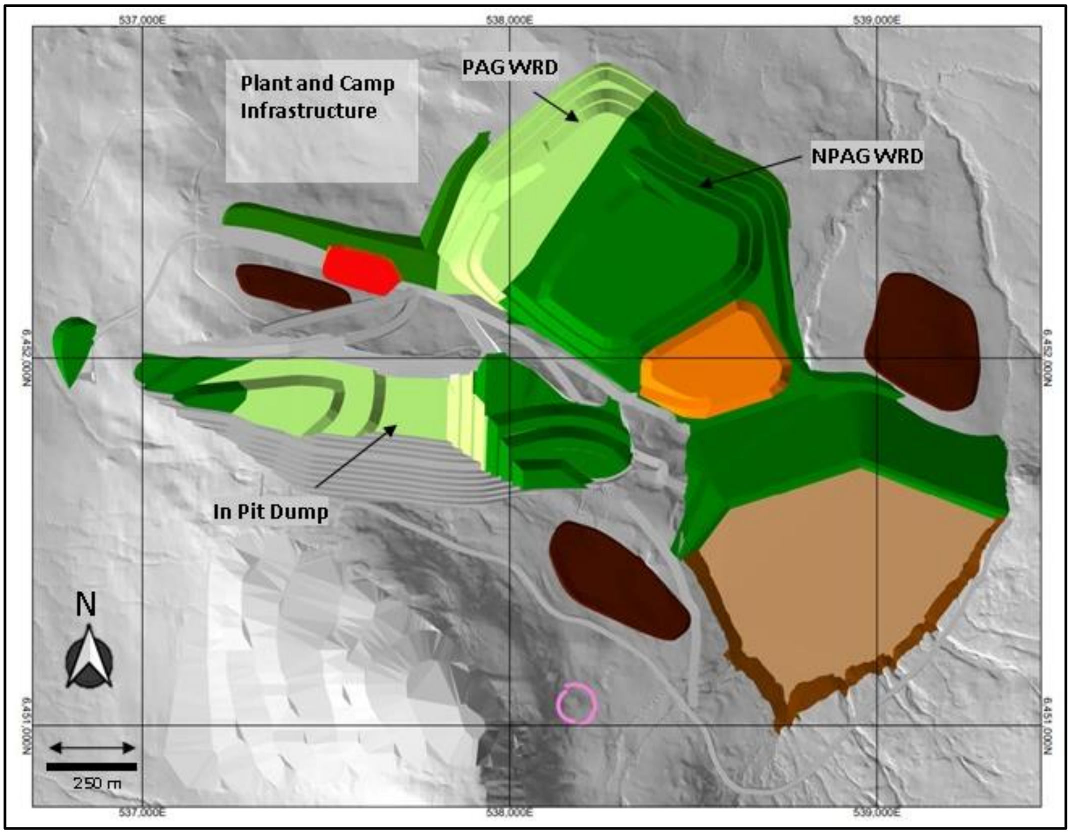


Figure 16-25: End of Year 9 Project Site Development

16.4.4.7 Years 10 and 11 Activities

Years 10 and 11 of operation is scheduled to include:

- Processing of the low-grade stockpile finishes
- Continuation and termination of in-pit backfill
- Commence covering TMF with impermeable layer as part of rehabilitation.

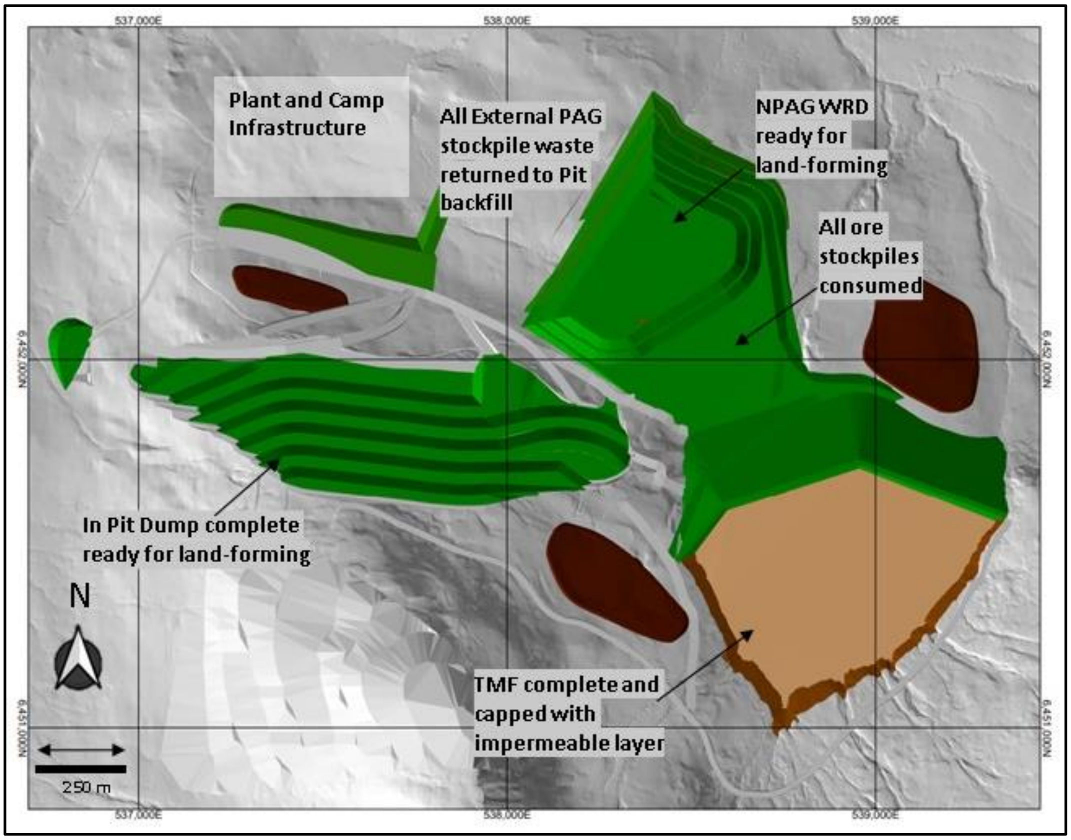


Figure 16-26: End of Year 11 Project Site Development.

1.1.1.1 Post-Closure Activities (Year 12+)

The site rehabilitation is discussed in Section 20:

- Land forming of the WRD and in-pit dumps
- TMF covered in an impermeable layer
- Reclamation of the process plant infrastructure, camp, roads and canals not required for long term sediment control
- All areas topsoiled and revegetated, water sediment control structures established
- Long-term water monitoring and treatment to final closure (see Section 20).

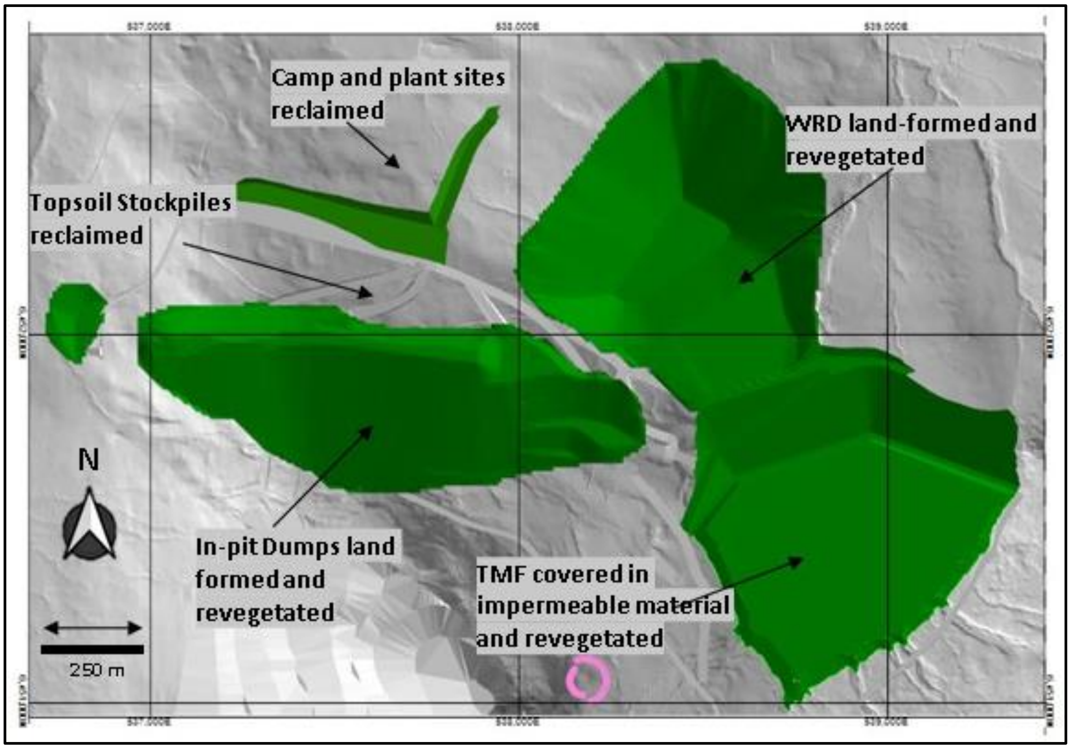


Figure 16-27: Post Operations Project Site Development

16.4.5 Expected Life of Mine

The life of the Project is 12.5 years, including two years of pre-development activity. The final two years of Project life consist of reclaiming and processing the low-grade ore stockpile and rehandle of waste rock into the pit void. The operating life of the open pit exclusive of pre-production and rehandle periods is eight years.

16.4.6 Waste Rock Dump

Waste rock will either be PAG (footwall rock) or nPAG (hangingwall rock). nPAG waste will be used for the construction of the TMF embankment. Waste (both PAG and nPAG) will also be placed directly as in-pit backfill but will mostly be directed to an external waste rock dump (WRD). PAG waste will be segregated in one area of the WRD closest to the CWP and underlain by a layer of nPAG waste. The WRD will be contained within a ditch drain that will direct all run-off to the CWP.

At the end of the life of the Phase 2 pit, all PAG waste will be rehandled into the pit and capped with a layer of impermeable soils and then nPAG rock.

The WRD, TMF embankment and the in-pit (backfilled) dump are an integrated waste storage package that is the product of forming the best outcomes from environmental requirements and economic performance. The size and placement of the WRD is a product of cost optimization (minimization) in combination with consideration of maintaining run-off drainage flow around the toe to the CWP, providing storage area for low-grade ore, maximizing the use of Phase 1 in-pit backfill and encroaching as little as possible into the Playboy Creek drainage as possible. A series of annual site development plans are provided at the end of this Section.

The WRD cost optimization (minimization) was undertaken using the Lerche-Grossman algorithm (CSA Global as reported in memorandum format "Kutcho FS – Preliminary Waste Rock Dump Designs" dated 2 May 2021). The optimization was carried out in two parts – the first until Phase 1 is completed, and the second after Phase 1 is backfilled. The optimization included constraints such as limiting disturbance of Playboy Creek and maintaining drainage to the CWP. The final design then added road alignments, crest and toe designs to surveyed topography and adjustments of the total volume to the mine plan total volumes.

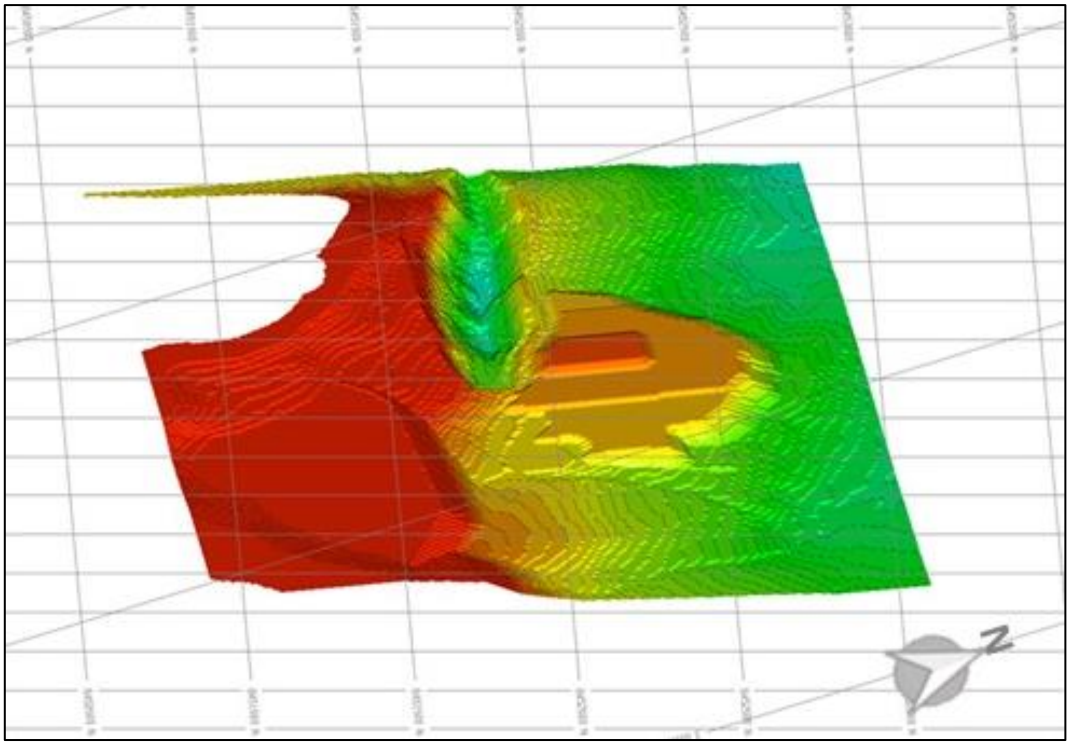


Figure 16-28: Isometric View Looking West Showing the Topography with the Total Pit and WRD Surface as a Result of Optimization at the End of Phase 1 (not to scale)

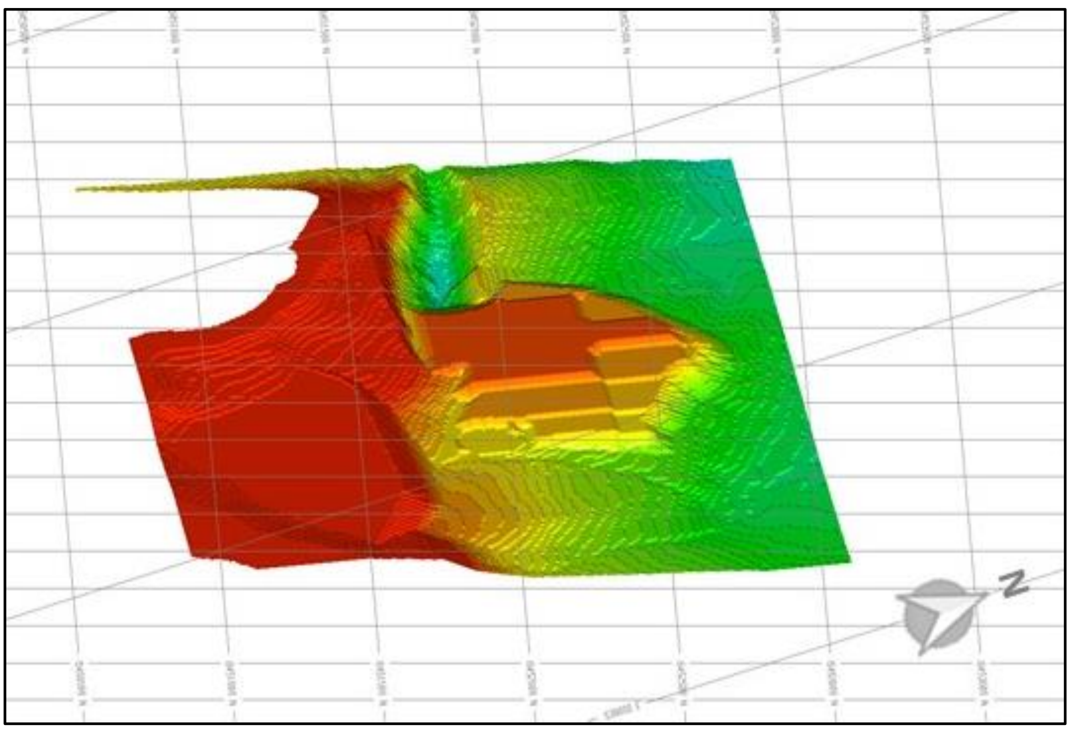


Figure 16-29: Isometric View Looking West Showing the Topography with the Total Pit and WRD Surface as a Result of Optimization at the End of Mine Operation (not to scale)

Design parameters for the WRD are:

- 20 m lift heights
- Maximum of five lifts
- Angle of repose on benches 35°
- Overall slope angle 28.6°
- Ramp width 25 m.

Figure 16-30 shows the WRD close to the end of mine life and before waste rock backfilling of Phase 2 commences.

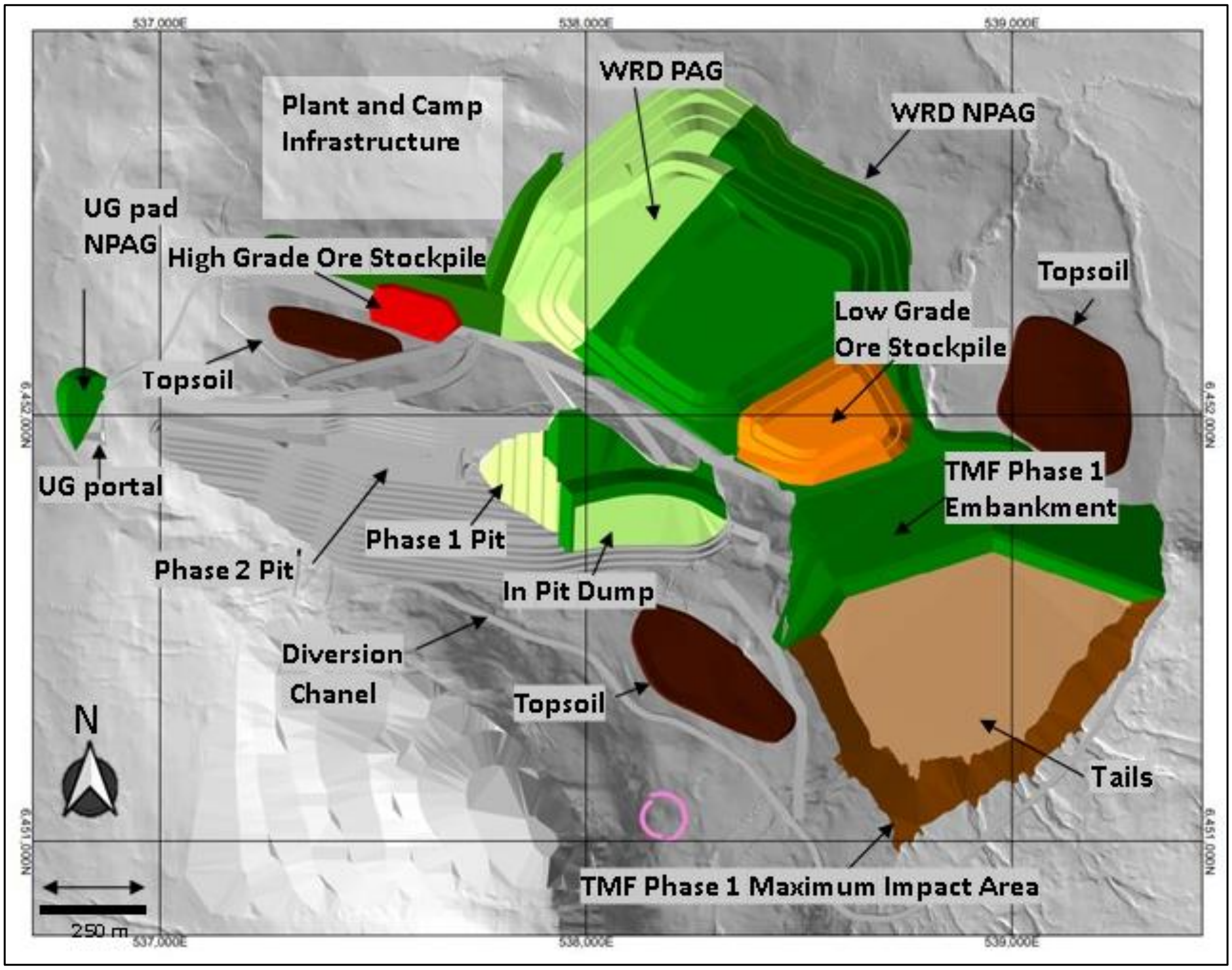


Figure 16-30: Open Pit Development in 2033 (Year 7)

After waste rehandle is completed the remnant WRD and in-pit backfilled dump will be shaped to a maximum overall slope gradient of 18.6° (3H: 1V) as described in Section 20. The TMF embankment is planned to be constructed at the at a slope that will require no further shaping for closure and rehabilitation and will be progressively reclaimed with no further land-forming requirements. Total stockpiling is depicted graphically in Figure 16-31. The WRD will store a maximum of 38.6 Mt of nPAG waste rock and 18.7 Mt of PAG waste rock. After pit backfilling is complete the WRD will contain 26.2 Mt of nPAG waste rock only as some of the nPAG and all the PAG waste rock will have been rehandled into the pit.

The TMF embankment will contain 12.8 Mt of nPAG waste rock. The pit will ultimately contain 18.3 Mt of nPAG waste and 25.8 Mt of PAG waste, for a total of 44.1 Mt.

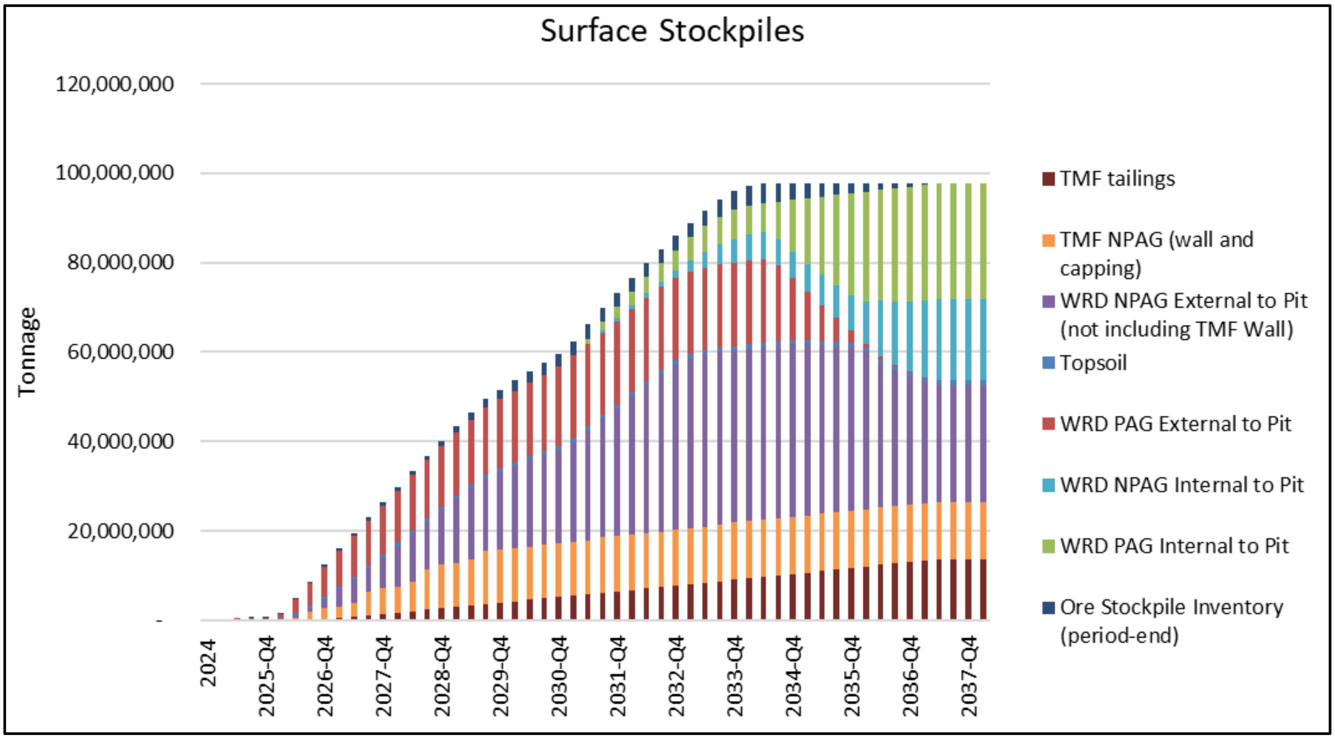


Figure 16-31: Surface Stockpile Development and Movement

16.4.7 *Open Pit Site Water Management Overview*

Within context of the overall site water management system, the open pit water management has the following features:

- Non-contact water from slopes above the pit are directed away from the pit area by a water diversion channel.
- Working benches in the pit will be inclined at 2% to the west to allow free drainage and direct water to collection drains that will then flow to the CWP. Details are shown in Section 18. This system will function until the 1560 masl bench after which free drainage away from the pit will no longer be possible.
- Below 1560 masl, the benches will still be inclined to the west, but water will be collected in one or more temporary sumps constructed in the pit floor. Up to three mobile diesel-powered pumps will be used to lift mine contact water to the water drains around the pit edge, from where the water will drain to the CWP via the external pit channel system. Once Phase 1 is complete, pontoon-mounted pumps will be used.
- Near-horizontal footwall groundwater depressurization wells will drain onto the footwall benches and direct the water as set out above.
- The pit pumping system is designed with a maximum capacity that exceeds anticipated freshet water volumes. However, the pumping capacity is not designed to cope with storm surges; this will be managed by sump capacity, and for extreme events, the pit will flood.
- All roads will be constructed with a substrate and compacted. This will minimise water erosion and particle entrainment. Haul roads will have drainage channels and culverts to collect water and direct it either to the external drainage system or to the pit sumps.
- The mine will aim to use as much ANFO as possible (rather than emulsion or other high density bulk explosives). ANFO has a lower nitrate load than emulsions and will reduce the contamination of water with nitrates and thus reduce water treatment costs.

- Waste rock mining will differentiate between nPAG and PAG materials (see Section 15 for more details). The footwall rocks are predominantly PAG. These will be stored either in a designated location in the WRD or within designated areas in the open pit Phase 1 and later Phase backfill areas. The WRD PAG area will be founded on a layer of acid-buffering nPAG material. All surface drainage from the WRD will be directed to the CWP.
- The ore is designated as PAG material and ore stockpiles will be founded on a nPAG substrate.
- Modelling suggests that the main underground operation will serve as a sump for the open pit. The underground pumping system is designed to cope with the predicted inflows; the open pit water management plan assumes no drainage into the underground workings and has sufficient capacity for independent and unaided dewatering.

16.4.7.1 Pit Dewatering and Depressurization

Pit wall depressurization requirements were based on geotechnical objectives to achieve target pore water pressures within the pit walls. Depressurization requirements are usually informed based on output from groundwater models of geotechnically critical sections of the pit wall, often associated with identified geotechnical weaknesses.

Terrane determined that the hangingwall rocks are classed as good or strong rock that should not require active pore water control measures. Limit-equilibrium modelling with high groundwater conditions suggests that even without groundwater drawdown the hangingwall slopes meet the design criteria factor of safety (FOS) of 1.3.

Terrane recommend a passive slope depressurization program for the footwall rocks means of horizontal drains. The Feasibility Study design was based on preliminary guidance that specified 30 m deep holes drilled with a vertical spacing of 15 m and in the bench toe. Terrane's final recommendation had a program of the same total hole length but using fewer, longer holes. This difference in design recommendation is not material to the Feasibility Study cost estimate nor geotechnical stability. The effect of the Main underground as a gallery drainage system requires to be examined for detailed design of the final selected passive drainage system.

16.4.7.2 Pit Inflows

The two sources of water inflows into the proposed pit are:

- Groundwater inflows – once the pits extend below the local water table groundwater inflows will occur through the bulk rock-mass and any permeable structures intercepted by the pits. For the footwall rocks this flow is facilitated with the passive drain hole system.
- Surface water (direct precipitation, snowmelt, and runoff) inflows – incidental rainfall which falls within the pit footprints (and does not infiltrate into the ground) and any surrounding surface catchments which are not prevented from draining to the pits.

Both groundwater and surface water pit inflows are dependent on the dimensions of the pit as it is developed; the site water balance was completed as a function of annual pit development. Above the 1560 masl elevation, occasional pit pumping may be required. Below the 1560 masl elevation, pit pumping is required. This applies to Phase 1 and Phase 2.

The site water balance includes estimates for groundwater and surface water flows developed from numerical modelling and site measurements of hydrological and hydrogeological parameters.

Total water inflows from surface and ground water to the pit will vary on a seasonal (monthly) basis but on average range between 5 L/s in winter to 35 L/s in freshet to 15 L/s after freshet. Until the pit is lower than the 1560 masl elevation, water will be directed to flow to the external drainage system (which eventually flows to the CWP) with occasional requirements for pumping. Pump capacity is designed for freshet flow rates. Storm events will lead to active bench flooding that will be recovered within about three days based on a 1:25 year event.

Mine sumps will be serviced by up to three diesel-motor powered centrifugal pumps capable of handling the maximum expected freshet flow of 35 L/s) in column heights less than 60 m. The pumps will sometimes be required to be staged in 20 m lifts. Water will be transported using 150 mm HDPE pipe. When the hydraulic head exceeds 60 m a single electrically powered pump capable of 90 m lift at 35 L/s will be added. The pump network has a maximum static head design of 120 m but will be less than this for much of the pit life.

The 1560 masl major catchment berm on the footwall will be used for redirection of water away from the pumping system. The catchment berm will be accessible to equipment and so can be kept clear of snow and debris.

*16.4.8 **Open Pit Mining Personnel***

Four operators are allocated per major piece of mining equipment. Operator hours are matched to equipment operating hours available. This implies that operators are flexible to styles of equipment or there are at least a reasonable proportion of fully trained operators that can give flexibility of operator assignment. Operator numbers are therefore shown as a total pool. The management team incorporates a full-time trainer and two rotating non-productive trainee roles to allow for a constant supply of fresh personnel and cross training opportunities.

During start-up lower staff and operator productivities are applied. After the cessation of open pit mining activities and during the stockpile processing and waste backfilling period, operator and management positions are reduced appropriately (no trainee positions are provided for).

Annual mine equipment operator complements are summarized in Table 16-17.

Table 16-17: Mine equipment operators per annum

	-2	-1	1	2	3	4	5	6	7	8	9	10	11
Equipment operators	9	59	67	67	65	53	73	74	67	40	52	43	23
Blasting	2	13	14	14	14	14	14	14	14	4	0	0	0
Maintenance	11	40	47	47	47	40	50	50	47	31	35	29	18
Management	10	20	22	22	22	22	22	22	21	16	10	9	8
Total	**32**	**132**	**149**	**151**	**147**	**129**	**159**	**160**	**148**	**91**	**96**	**82**	**49**

16.4.8.1 Open Pit Mine Management

Open pit mine management will cover all duties required for operational and technical leadership of the open pit mining operation. Some roles will be multi-skilled to ensure that necessary skills are always present on all shift rotations. They will utilize a range of computer programs to conduct required technical evaluations and will have a range of specialized equipment.

Annual requirements for mine management personnel are summarized in Table 16-18.

Table 16-18: Mine management personnel per year

Open pit mine management	Shift	-2	-1	1	2	3	4	5	6	7	8	9	10	11
Mine Superintendent	Day	1	1	1	1	1	1	1	1	1	1	1	1	1
General Mine Foreman	Day	1	1	1	1	1	1	1	1	1	0	0	0	0
Mine Foremen	Shift	0	4	4	4	4	4	4	4	4	4	4	4	3
Mine Clerk	Day	0	1	1	1	1	1	1	1	1	1	1	0	0
Mine Labourer	Shift	1	2	2	2	2	2	2	2	2	2	0	0	0
Training Coordinator	Day	1	1	1	1	1	1	1	1	1	1	1	1	1
Equipment Trainee	Shift	0	2	2	2	2	2	2	2	2	0	0	0	0
Chief Mine Engineer	Day	0	1	1	1	1	1	1	1	1	0	0	0	0
Senior Mine Engineers	Day	0	1	1	1	1	1	1	1	1	1	0	0	0
Senior Surveyor	Day	1	1	1	1	1	1	1	1	1	1	1	1	1
Survey Technician	Day	2	2	2	2	2	2	2	2	2	1	1	1	1
Senior Geologist	Day	1	1	1	1	1	1	1	1	1	1	0	0	0
Senior Geotechnical Engineer	Day	1	1	1	1	1	1	1	1	1	1	0	0	0
Geologist – Grade Control	Day	0	1	1	1	1	1	1	1	1	1	1	1	1
Geology Technician	Day	0	2	2	2	2	2	2	2	2	1	0	0	0
Total		**10**	**20**	**22**	**22**	**22**	**22**	**22**	**22**	**21**	**16**	**10**	**9**	**8**

16.4.8.2 Open Pit Mine Maintenance Personnel

Open pit mine maintenance personnel requirements were based on the equipment operating hours and are shown in Table 16-19.

Table 16-19: Mine maintenance personnel per year

Open pit mine maintenance	Shift	-2	-1	1	2	3	4	5	6	7	8	9	10	11
Maintenance Superintendent	Day	1	1	1	1	1	1	1	1	1	1	1	1	1
Maintenance General Foreman	Day	1	1	1	1	1	1	1	1	1	1	1	1	1
Maintenance Foreman	Day	1	4	4	4	4	3	4	4	4	3	4	3	2
Mechanical Foreman	Day	1	1	1	1	1	1	1	1	1	1	1	1	1
Maintenance Planner	Day	0	2	2	2	2	2	2	2	2	2	2	1	1
Maintenance Clerk	Day	0	1	1	1	1	1	1	1	1	1	1	1	1
Mechanics HD	Shift	3	13	15	16	16	13	17	17	16	9	10	8	5
Mechanics LD	Shift	1	6	8	8	8	7	8	9	8	5	5	4	2
Auto Electricians	Shift	0	1	2	2	2	1	2	2	2	1	1	1	1
Welders	Shift	1	6	8	8	8	7	8	9	8	5	5	4	2
Tyreman	Shift	2	2	2	2	2	2	2	2	2	2	2	2	2
Trainee	Shift	0	2	2	2	2	2	2	2	2	1	1	1	1
Total		**11**	**40**	**47**	**47**	**47**	**40**	**50**	**50**	**47**	**31**	**35**	**29**	**18**

16.5 Underground Mining

The underground mining component of the Project will see a total of 2.83 Mt of ore extracted from the Main and Esso deposits. Access to the Main deposit will be via a ramp from surface starting at 1470 masl. The Main deposit will feature two spiral ramps to facilitate crosscuts and access to the mineralization, while the Esso deposit will feature one spiral ramp. Access to the Esso deposit will be a via an 1,800 m ramp connecting the 1335 masl elevation of Main to the 1090 masl elevation of Esso.

The selected mining method for the Project is longitudinal longhole open stoping (LLHOS). Stopes will be drilled from the upper ore drives. Stopes will be mucked using load/haul/dump machines (LHDs) and ore will be transported to ore passes, then loaded onto 45t capacity articulated dump trucks and hauled out of the mine via the ramp. The haul trucks will take ore sorter rejects from the plant and deliver it to the cement mixing plant prior to delivery into the depleted stopes as cemented rock fill (CRF). Throughout the life of the underground operation, the average of 880 tpd at an average 1.99% copper, 3.58% zinc, 50 g/t silver and 0.68 g/t gold.

16.5.1 Underground Geotechnical Design

Terrane Geoscience Inc. (TGI) produced recommendations for the underground mine design based on their data collection and analysis. Summaries of those recommendations and their use in the mine plan are described throughout the following report sections.

16.5.1.1 Stope Geomechanical Design

The geomechanical design assumes that the desired mining method will be supported LLHOS with cemented rock (CRF) backfill. The Stability Graph Method for open stope design, initially proposed by Mathews et al. (1981) and later modified by Potvin (1988) as the Modified Stability Graph was used to assess the stability of open stopes for the Main deposit. ((Terrane Geoscience Inc., 2021).

The hydraulic radius is summarized on the right-hand side of Figure 16-32 and is equal to the surface area of the design face (L x h) divided by the perimeter of the design face (2L+2h), where, for the hanging-wall and footwall, L is the stope strike length and h is the stope down-dip height. (Terrane Geoscience Inc., 2021).

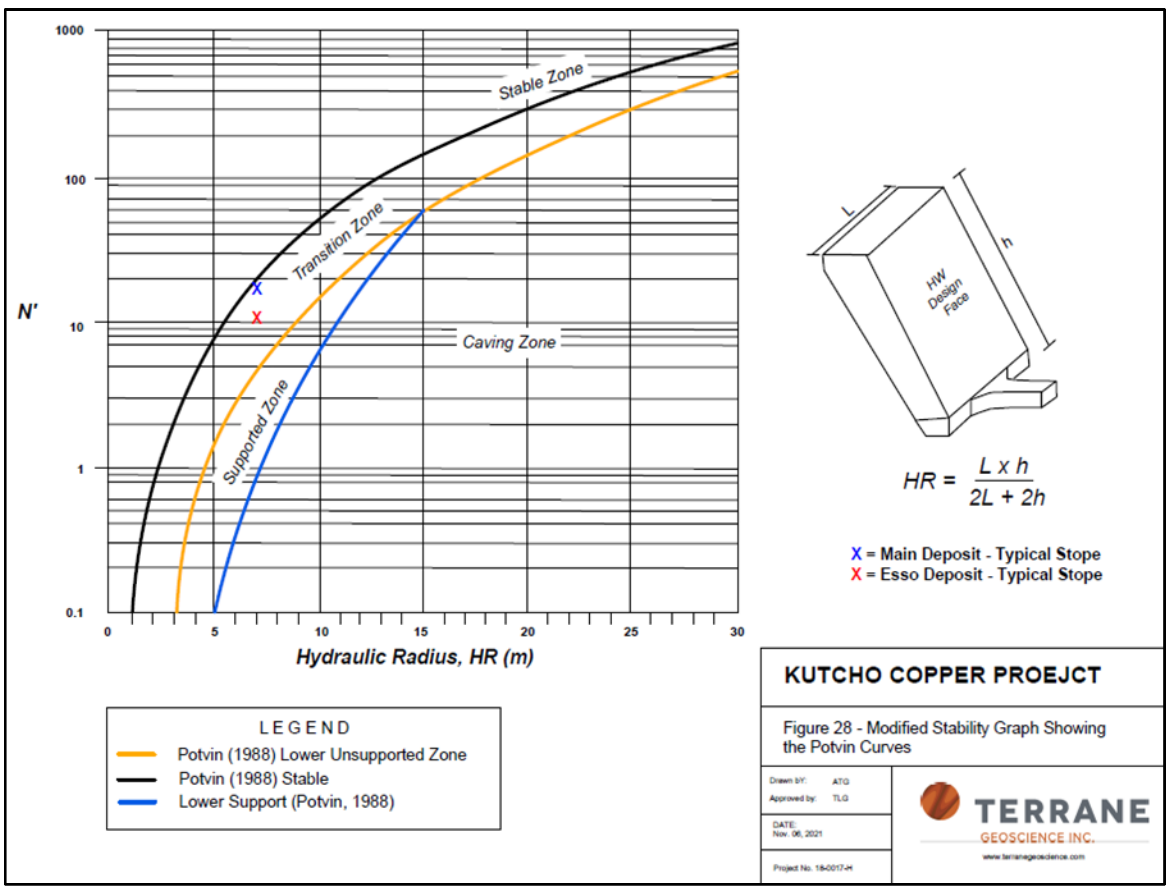


Figure 16-32: Modified Stability Graph Showing Potvin Curves
(Source:Terrane Geoscience Inc., 2020)

The modified stability number, N' is calculated by the equation:

$$N' = Q' \, x \, A \, x \, B \, x \, C$$

Where:

- **Q'** is the modified Q-system rock mass classification in which the final term Jw/SRF is equal to 1 (i.e. dry conditions and medium in-situ stress)
- **A** is the rock stress factor (varies from 0.1 to 1) and is the ratio of the rock UCS strength to the induced stress in the stope face
- **B** is the joint orientation adjustment factor (varies from 0.2 to 1) and is a measure of the influence of the orientation of discontinuities with respect to the surface being analyzed
- C is the gravity adjustment factor (varies from 2 to 8) and accounts for the mode of failure in open stopes (i.e. gravity fall, slabbing or sliding). (Terrane Geoscience Inc., 2021).

16.5.1.2 Main Deposit

Design criteria for supported stopes is based on the halfway point within the supported zone of the Modified Stability Graph (between the lower unsupported curve and the lower support curve) as acceptable design criteria.

The following assumptions and data inputs are summarized from the Terrane 2021 report:

- Non-man entry to production levels.
- Stope orientation: Average dip of 45° and average strike of 277°
- S1–- the primary foliation is considered the critical discontinuity set on all design faces due to its frequency of occurrence.
- Maximum mining induced stress assessment in Examine 2D (Rocscience, 2018) of a 25 m high x 15 m wide open stope used for "A" adjustment factor. The analysis was performed for stope backs, hanging wall and footwall at depths below 250 m.
- Sublevel intervals of 25 m (floor to floor) were considered in this analysis. Figure 16-33 shows a schematic of a 25 m inter-level vertical distance with a dip of 45° yielding a 37.3 m inclined height.

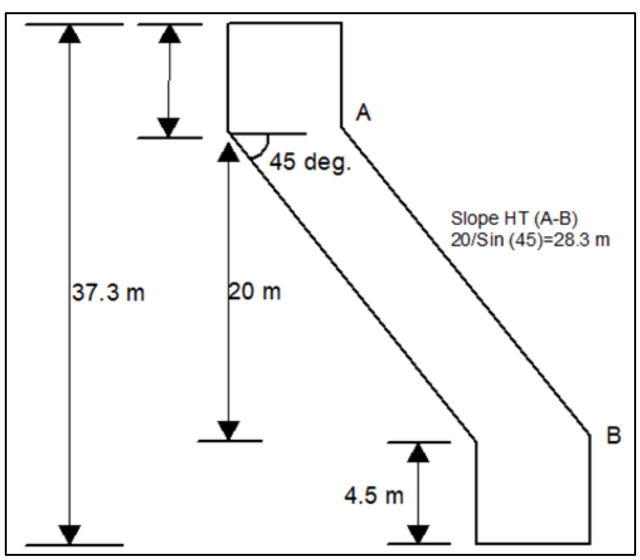


Figure 16-33: Typical Geometry for a 25 m (floor to floor) Sublevel Interval. 37.3 m (floor to back)
(Source:Terrane Geoscience Inc., 2021)

16.5.1.3 Esso Deposit

The design criteria for unsupported stopes uses the average transition zone (i.e. halfway between the stable zone curve and the lower unsupported curve) as the limit for stope design. This approach is considered a compromise between using the more aggressive lower unsupported curve and the more conservative stable curve for design. Similarly, for designing supported stopes, the halfway point within the supported zone (between the lower unsupported curve and the lower support curve) was selected as the acceptable design criteria (Terrane Geoscience Inc., 2021).

Our design criteria for supported stopes uses the average transition zone between the lower unsupported curve and the lower support curve.

The following assumptions and data inputs are summarized from the Terrane 2021 report:

- Non-man entry to production levels.
- Stope orientation: Average dip of 50° and average strike of 276°.
- S_1 - the primary foliation is considered the critical discontinuity set on all design faces.
- Maximum mining induced stress assessment in Examine 2D (Rocscience, 2018) of a 20 m high x 20 m long open stope used for “A” adjustment factor. The analysis was performed for stope backs, hanging wall and footwall at depths between 350 m and 650 m.
- Sublevel intervals of 25 m (floor to floor) were considered in the analysis. shows a schematic of a 25 m inter-level vertical distance with a dip of 45° yielding a 37.3 m inclined height.

The design from Terrane recommends unsupported longitudinal stopes for areas where the ore zone is up to 15 m wide (i.e. across strike footwall to hangingwall) and supported longitudinal stopes up to a maximum of 20 m wide.

16.5.1.4 Stope Design – Longitudinal Longhole

The following tables summarize the results of the final longitudinal longhole stope design for both the Main and Esso deposits. An assessment of stope backs, footwall, hangingwall, and end walls was evaluated. It was determined that the hangingwall dictates the final longitudinal stope design. The results are summarized below for 5 m to 20 m widths and for 25 m sublevels (floor to floor), and for Q' mean values.

Table 16-20: Final Main deposit LLHOS stope design summary – supported – 100 m to 250 m depth

Depth (m)	W (m)	L (m)	Sublevel H (m)	Slope HT (m)
100 – 250	5	60	25	37.3
	10	60	25	37.3
	15	60	25	37.3
	20	60	25	37.3

(Source:Terrane Geoscience Inc., 2021)

Table 16-21: Final Esso deposit LLHOS stope design summary – supported – to 650 m depth

Depth (m)	W (m)	L (m)	Sublevel H (m)	Slope HT (m)
≤650	5	48	25	35
	10	48	25	35
	15	48	25	35
	20	48	25	35

(Source:Terrane Geoscience Inc., 2021)

The Empirical Estimation of Wall Slough or Equivalent Linear Overbreak / Slough (ELOS) (Clark and Pakalnis, 1997) for the unsupported cases of a 45° dip (Main) and a 50° dip (Esso) are estimated to be limited to 0.5 m to 1.0 m. Additional information on ELOS is presented in Section 15.5.2

16.5.1.5 Cable Bolt Design

Cable bolt designs were based on empirical designs and were completed by TGI. Guidelines for design utilized relationships between RQD, Jn, and HR and densities of cable bolts. Also considered were densities based on N'/HR ratio. The proposed design uses the point anchor approach, in which cable bolts on the back and hanging wall are installed from sublevel drifts. Stand-off distances for Main are recommended to be 30 m from access development to the ore zone. This recommendation is applicable to both hangingwall and footwall designs. (Terrane Geoscience Inc., 2021).

Table 16-22: Cable bolt design summary

Spacing (m)	Type and diameter	No. of bolts in each ring	Spacing along strike (m)	Length (m)	Face plate size (mm)	Drillhole diameter (mm)	No. of cable bolts at each access
2.0 x 2.0	Double strand (16 mm)	6	2	10	200 x 200 x 12 (thick)	51-54	10

Note: Design is applicable to both Main and Esso Deposits. (Terrane Geoscience Inc., 2021)
(Source:Terrane Geoscience Inc., 2021)

Table 16-23: Cable bolting parameters

Cable bolting parameters	Units	Value
Cable bolt length	metre	10.0
Drill metres for cable bolting	metre/bolt	11.0
Drilling cycle time	min/bolt	27.8
Cable Bolt installation cycle time	min/bolt	15.0
Total cable bolting cycle time (in 50 min hours)	hours/bolt	0.9
Cable bolter time	hours/bolt	0.9
Cable bolting labour time	hours/bolt	1.1

16.5.1.6 Ground Support Design

Based on the estimated rock mass classification, ground support classes have been developed for standard lateral development and production drifts for the Main and Esso deposits. The following tables illustrate the preliminary ground support designs for both Main and Esso. For the recommended support designs to work as a system it is good practice to install rock bolts through surface support such as mesh and shotcrete (where recommended by the ground support design).

General ground support recommendations were developed for permanent and temporary workings as part of this Feasibility Study. Permanent excavations will include the ramp, muck bays, safety bays, passing bays and associated three-way intersections. Temporary openings are limited to ore drives and cross cuts. These ground support recommendations are based on the available geotechnical data presented within this report. It should be noted that geotechnical data along the alignment of the ramp from the surface portal to the Main deposit and down to the Esso deposit is limited. As more geotechnical data becomes available, these design recommendations should be reviewed (Terrane Geoscience Inc., 2021).

Permanent Workings

Permanent underground developments, such as the ramp, are given an ESR of 1.6 and have a span of 5.0 m which results in an equivalent dimension of 3.1. The following tables summarize the empirical support in relation to the Rock Mass Quality and Rock Support chart for a typical 5.0 m excavation span associated with the permanent development (Terrane Geoscience Inc., 2021).

Table 16-24: Support class based on rock mass quality and rock support – permanent workings

Domain	Deposit	Q-System	Support class (1)	Support class description (1)
FW_1 (0-125 m)	Main	3.2	3	Systematic pattern bolting (2)
FW_2 (>125 m)	Main	3.5	3	Systematic pattern bolting (2)
FW (between Esso and Main)	N/A	9.6	1	Systematic pattern bolting
FW	Esso	6.0	1-3	Systematic pattern bolting
Fault	N/A	0.1	5	Systematic pattern bolting and 9–12 cm of fibre reinforced shotcrete.

Notes: (1) After Grimstad and Barton, 1993. (2) Assumed to includes high tensile mesh rather than shotcrete. (Source:Terrane Geoscience Inc., 2021)

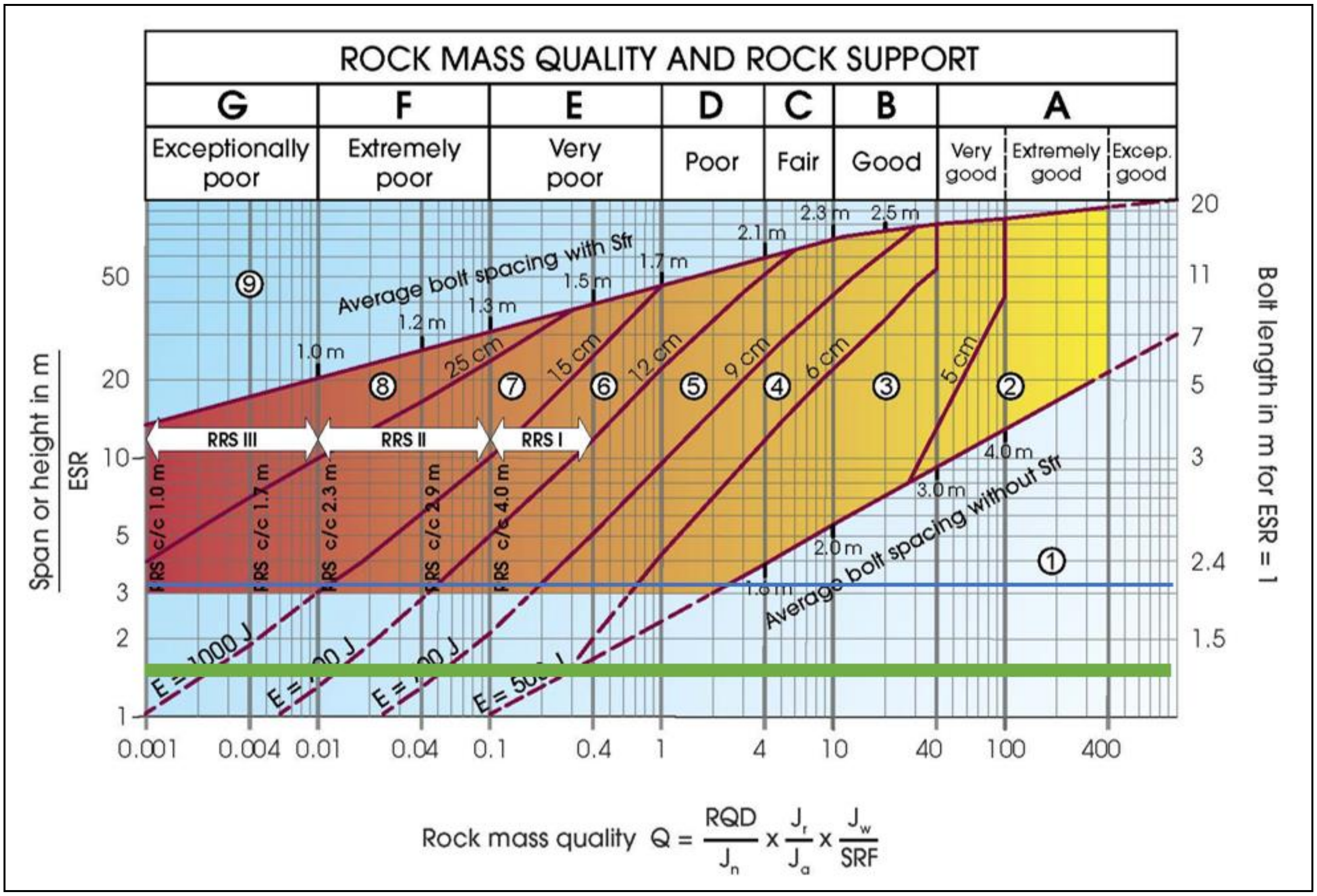


Figure 16-34: Rock Mass Quality and Rock Support Design Chart

Notes: Blue line represents an ESR of 1.6 for a span of 5.0 m (permanent development). Green line represents an ESR of 3.0 for a span of 4.5–5.0 m (temporary development). Figure adapted from Grimstad and Barton, 1993. (Source:Terrane Geoscience Inc., 2021)

Temporary Workings

Temporary underground developments such as the ore drives and crosscuts are given an ESR of 3.0 and have a span of 4.5 m and 5.0 m, respectively. This results in an equivalent dimension of 1.5 to 1.6. The temporary development is limited to the Main FW_2, Main ore zone, Esso FW and Esso ore zone geotechnical domains. Table 16-25 summarizes the empirical support recommendations a typical 4.5–5.0 m temporary excavation span.

Table 16-25: Support class based on rock mass quality and rock support - temporary workings

Domain	Deposit	Q-System	Support class	Support class description
FW_2 (>125 m)	Main	3.5	1	Systematic pattern bolting
Ore Zone	Main	9.3	1	Systematic pattern bolting
FW	Esso	6.0	1	Systematic pattern bolting
Ore Zone	Esso	5.2	1	Systematic pattern bolting
Fault	N/A	0.1	5	Systematic pattern bolting and 12 – 15 cm of fibre reinforced shotcrete.

16.5.1.7 Temporary Rib Pillar Design

Diaphragm Pillar (Rib Pillar) design was complete by TGI. At the Main Deposit, stope widths of up to 20 m and stope lengths of up to 40 m are expected to be achievable with the use of a 7 m wide rib pillar design. For stope widths greater than 20 m and stope lengths of up to 20 m, a 9.5 m wide rib pillar design is recommended.

Table 16-26: Rib pillar design – Main deposit (0–250 m)

Stope width (hangingwall to footwall)	L (along strike)	Sublevel	Pillar width (m)	Sigma 1 (Mpa)	Wp/Hp	Pillar load/ UCS	Class	Colour code for Figure 16-35
10	40	25	7.0	16.0	0.7	0.20	Stable	Red
15	40	25	7.0	15.8	0.5	0.18	Stable	Blue
20	40	25	7.0	15.1	0.4	0.19	Stable	Yellow
25	20	25	7.0	13.4	0.3	0.17	Stable	Lt. Green
30	20	25	9.5	12.6	0.3	0.15	Stable	Dk. Green
35	20	25	9.5	12.3	0.3	0.15	Stable	Dk. Green

Notes:

- *Average Pillar Load from Examine 3D stress modelling with one side of the pillar backfilled with paste. In-situ stress is assumed to be perpendicular to the ore zone.*
- *Wp (Pillar Width) is defined as the dimension of the pillar normal to the maximum induced stress (sigma 1), which in the case of a rib pillar with the in-situ stress normal to the ore zone is the length along strike.*
- *Hp (Pillar Height) is defined as the dimension of the pillar parallel to the maximum induced stress (sigma 1), which is the case of a rib pillar with the ins-situ stress normal to the ore zone is the width (hangingwall to footwall) of the ore zone.*
- *Unconfined Compressive Strength (UCS) of ore zone has an average value of 81 Mpa.*

(Source: Terrane Geoscience Inc., 2021)

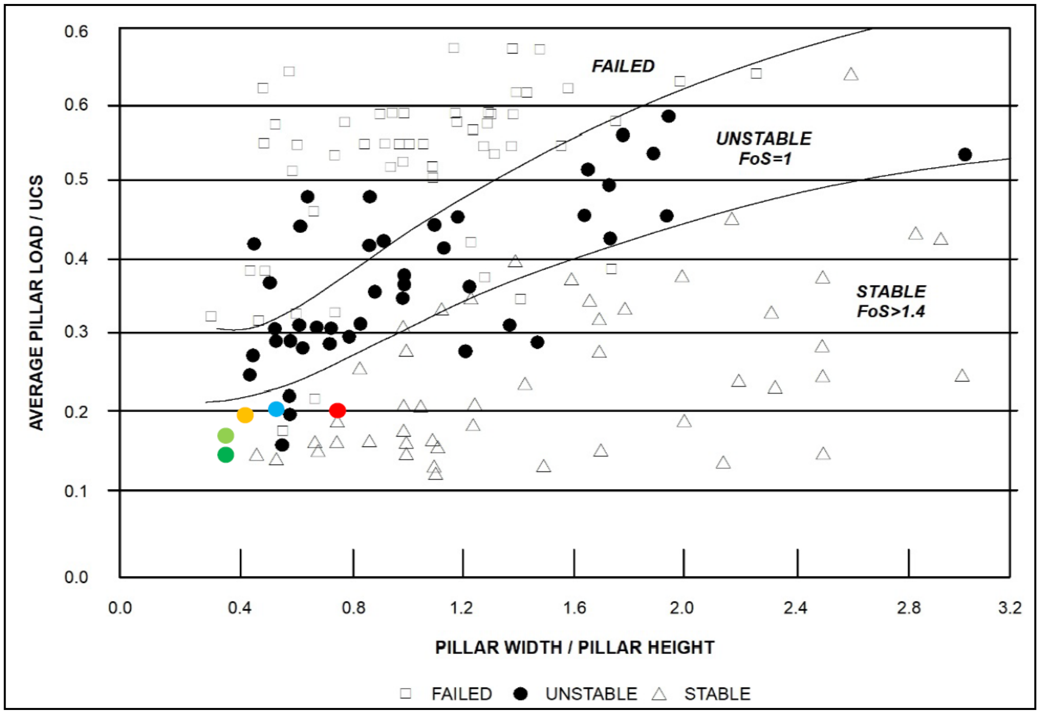


Figure 16-35: Pillar Load / UCS vs. Pillar W / H

(Source: Terrane Geoscience Inc., 2021)

16.5.1.8 Summary of Ground Support Recommendations

The empirical support methods and support classes summarized above along with the Unwedge analysis provide an estimate of the support that is required to maintain the stability of the underground developments. Practical operational considerations such as standard bolt lengths and bolting patterns matching typical mesh panels are also considered as part of ground support recommendations (Terrane Geoscience Inc., 2021).

The following tables summarize the recommended ground support designs for permanent and temporary developments based on the analyses presented in the previous sections.

Table 16-27: Ground support for permanent underground development for 5.0 m wide x 5.0 m high

Dev. class	Domain	Location	Support class (1)	Anchor type	Anchor spacing (m)	Anchor length (m)	Strength (kN)	Mesh	Shotcrete
Permanent	FW_1 (0–125 m) – Main	Back	3	#6 Rebar	1.2 x 1.2	2.4	120	High-tensile steel (3)	N
		Walls	3	Split set (SS-33)	1.2 x 1.2 (2)	1.8	85	High-tensile steel (3)	N
	FW_2 (>125 m) – Main	Back	3	#6 Rebar	1.2 x 1.2	2.4	120	High-tensile steel (3)	N
		Walls	3	Split set (SS-33)	1.2 x 1.2 (2)	1.8	85	High-tensile steel (3)	N
	FW (Main-Esso) (4)	Back	1	#6 Rebar	1.2 x 1.2	2.4	120	# 9 Welded wire mesh (5)	N
		Walls	1	Split set (SS-33)	1.2 x 1.2 (2)	1.8	85	# 9 Welded wire mesh (5)	N
	FW – Esso	Back	3	#6 Rebar	1.2 x 1.2	2.4	120	High-tensile steel (3)	N
		Walls	3	Split set (SS-33)	1.2 x 1.2 (2)	1.8	85	High-tensile steel (3)	N

Dev. class	Domain	Location	Support class (1)	Anchor type	Anchor spacing (m)	Anchor length (m)	Strength (kN)	Mesh	Shotcrete
	Fault (6)	Back	5	#6 Rebar	1.2 x 1.2	2.4	120	# 9 Welded wire mesh (5)	9–12 cm Fibre-reinforced
		Walls	5	#6 Rebar	1.2 x 1.2	1.8	120	# 9 Welded wire mesh (5)	9–12 cm Fibre-reinforced

Notes:
(1) From Empirical Rock Mass Quality and Rock Support Design Chart (Grimstad and Barton, 1993)
(2) First row of anchors assumed to start approximately 1.4 m above the floor.
(3) Minimum tensile strength of 1,700 N/mm² (example MINAX 80/4 GeoBrugg).
(4) Decline from Main to Esso.
(5) Galvanized welded wire mesh, 100 mm by 100 mm squares, 3.7 mm diameter.
(6) For costing purposes it is recommended that approximately 2.5% of the development workings be assumed to fall within the Fault geotechnical domain.

Table 16-28: Ground support for temporary underground development for 4.5 m to 5.0 m

Dev. class	Domain	Location	Support class (1)	Anchor type	Anchor spacing (m)	Anchor length (m)	Strength (kN)	Mesh	Shotcrete
Temporary	FW_2 (>125 m) – Main	Back	1	#6 Rebar	1.5 x 1.5	2.4	120	# 9 Welded wire mesh (3)	N
		Walls	1	Split set (SS-33)	1.5 x 1.5 (2)	1.8	85	# 9 Welded wire mesh (3)	N
	Ore Zone Main	Back	1	#6 Rebar	1.5 x 1.5	2.4	120	# 9 Welded wire mesh (3)	N
		Walls	1	Split set (SS-33)	1.5 x 1.5 (2)	1.8	85	# 9 Welded wire mesh (3)	N
	FW Esso	Back	3	#6 Rebar	1.5 x 1.5	2.4	120	# 9 Welded wire mesh (3)	N
		Walls	3	Split set (SS-33)	1.5 x 1.5 (2)	1.8	85	# 9 Welded wire mesh (3)	N
	Ore Zone Esso	Back	5	#6 Rebar	1.5 x 1.5	2.4	120	# 9 Welded wire mesh (3)	N
		Walls	5	#6 Rebar	1.5 x 1.5	1.8	85	# 9 Welded wire mesh (3)	N

Notes:
(1) From Empirical Rock Mass Quality and Rock Support Design Chart (Grimstad and Barton, 1993).
(2) First row of anchors assumed to start approximately 1.4 m above the floor.
(3) Galvanized welded wire mesh, 100 mm by 100 mm squares, 3.7 mm diameter.

16.5.2 Underground Mining Method

16.5.2.1 Longhole Open Stoping

Access to the Main deposit will be via a ramp from surface starting at 1470 masl. The Main deposit will feature two spiral ramps to facilitate access to the mineralization, while the Esso deposit will feature one spiral ramp. Access to the Esso deposit will be a via an 1,800 m ramp connecting the 1335-level of Main to the 1090-level of Esso. The spiral ramp access into the orebody is an important feature of the mining method in defining and planning how the equipment and materials will move in and out of the stopes. The spirals are organized to be centrally located in a block of mineralization so that there is equal haulage distance from each end to the central spiral. The spiral and all access development lies in the footwall of the mineralization. The core of the spiral will also contain manway access, ore passes, and ventilation raise(s) enabling level connectivity for airflow, ore transport, and emergency egress. The spiral will also host drill chambers that will be used to provide stope definition drilling which is the first step of the detailed stope design and ore drive location planning. The stope definition drillholes will be completed on a 12.5 m x 12.5 m pattern.

Open stope dimensions were derived through geotechnical investigation as reported by Terrane. After application of the supplied dimension controls, the average stope dimensions are 12.5 m wide, 40 m long, and 25 m high. However, the stopes may be as narrow as 2.5 m and as wide as 25 m. The stop length decreases for wider stopes to maintain stable opening. The Main deposit dips at an average of 47° from horizontal, and the Esso deposit is slightly steeper at 50° on average. The stope design followed Terrane recommendation for stope dimensions, temporary rib pillar sizes and ground support.

After drift development is complete, rings of blast holes are drilled in a parallel or fan pattern from the top level to the bottom level within the orebody. Blast holes will be double primed and only charged with packaged emulsion and ANFO immediately prior to blasting. Several rows will typically be pre-charged to minimize the loading crew's exposure to the open stope brow. Temporary rib pillars (TRP) will have more specialized detonation control as described elsewhere in the section.

An initial slot will be developed by drilling a drop raise made up of closely spaced blast holes. Once the initial slot was blasted and to retain a minimum pillar below the top drift, the entire stope will be blasted and then mucked using a remote-operated LHD. All remote mucking will be carried out using a 3.0 m^3 LHD equipped while non-remote (manned) mucking will be done with either a 3.0 m^3 or a 4.0 m^3 LHD.

Mined out stopes will be backfilled with cemented rock fill (CRF) containing 2% to 3% cement as a binder. Curing for 28 days will be allowed before extracting the adjacent stope. Figure 16-36 through Figure 16-44 illustrate the downhole longhole open stoping method.

The dominant mode of drilling used will be downhole stoping with occasional use of up-hole stoping when required. In up-hole stoping, a series of parallel rings will be drilled from the bottom drift into the back to the limit of the ore body. An inverse raise is drilled on the extremity of the stope and then blasted in one shot. Longhole rings will be blasted into the void where they will be mucked by a remote-operated load-haul-dump loader (LHD).

Once the stope designs are complete, ore drifts will be driven on 25 m levels. The first drifts will commence at the bottom of the defined mineralization block and progress upwards. Once two drifts are completed, ventilation between levels will be established and the stope mining sequence will commence and is described in the following nine stages below. The stages repeat for addition stopes and levels. Note that in the figures, the stope lengths are not to scales, and with depth, their length will vary based on local rock mass conditions. The mine has a team of geotechnical experts to monitor and give production guidance.

Stage 1 – Ore drive development and stope drilling (Figure 16-36):

- Ore access drifts will be used to access the mineralization and begin excavating the ore drives.
- Cable bolts will be installed as necessary.
- The slot raise will be drilled, along with the first stope and the temporary rib pillar (TRP).

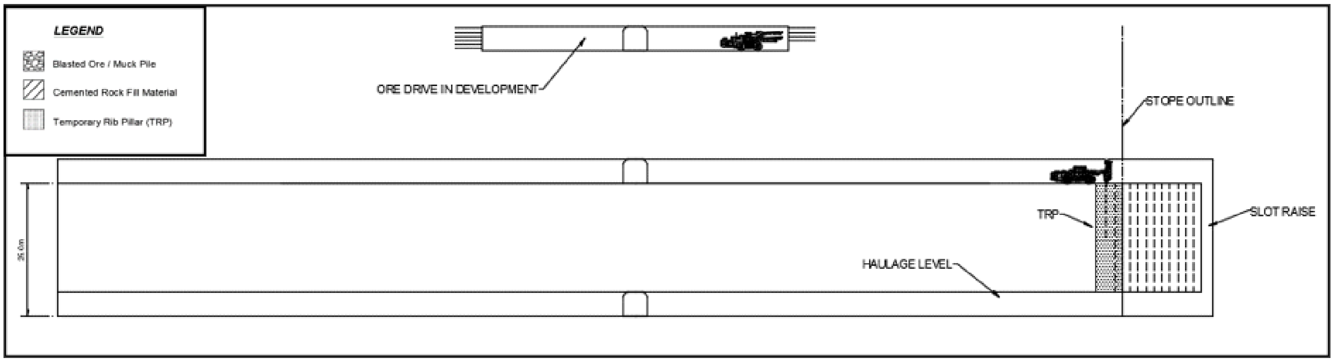


Figure 16-36: Stage 1 – Ore Drive Development and Stope Drilling (not to scale)

Stage 2 – Stope blasting and mucking (Figure 16-37):

- The stope is drilled and the blast holes charged, after which the stope is blasted and mucked.
- The TRP is drilled and charged blast holes adjacent to the TRP will be drilled at the same time as the TRP blast holes so that no drilling will take place in the vicinity of the charged TRP. Remote detonators will allow detonation at a later stage in the mining sequence remote wireless blasting technology.
- While one stope is being mucked, another is being drilled on the opposite end of the ore drive.

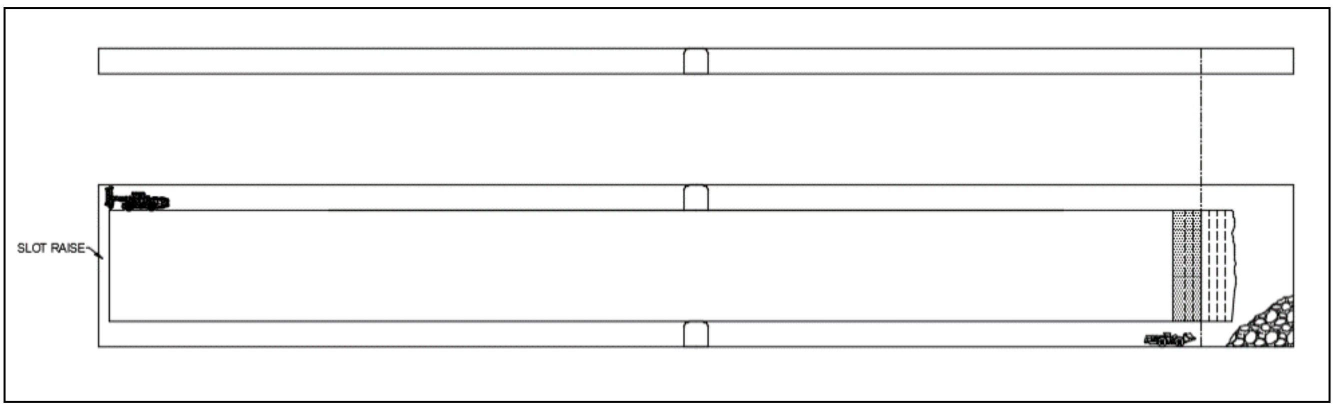


Figure 16-37: Stage 2 – Stope Blasting and Mucking (not to scale)

Stage 3 – Stope drilling and backfilling (Figure 16-38):

- Once the ore is mucked, the first stope can be backfilled with cemented rock fill (CRF).
- During backfilling, drilling continues at the opposite end of the ore drive.

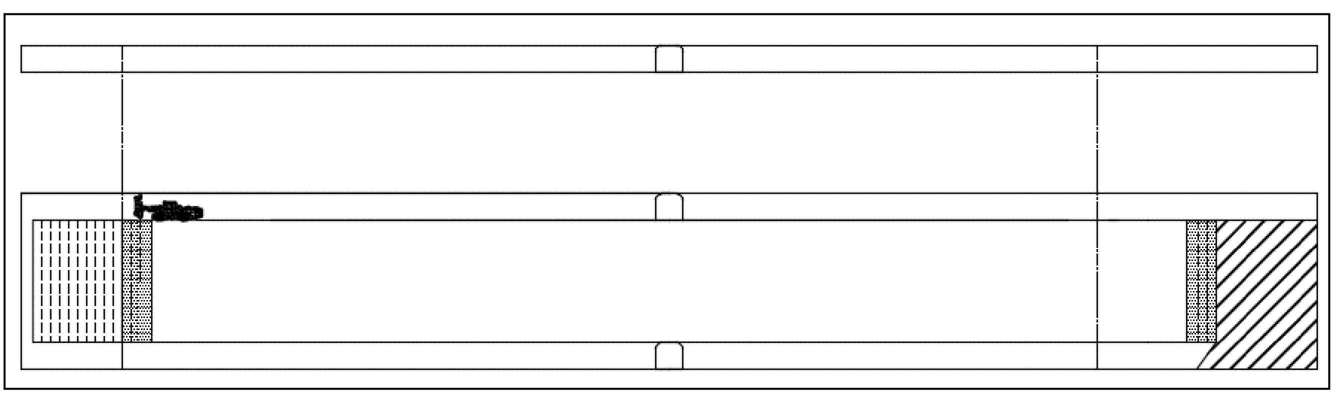


Figure 16-38: Stage 3 – Stope Drilling and Backfilling (not to scale)

Stage 4 – Stope blasting, mucking, and drilling (Figure 16-39):

- Once the second stope was drilled, it can be blasted and mucked.
- While the first backfilled stope is curing, the TRP allows the third stope to be drilled.

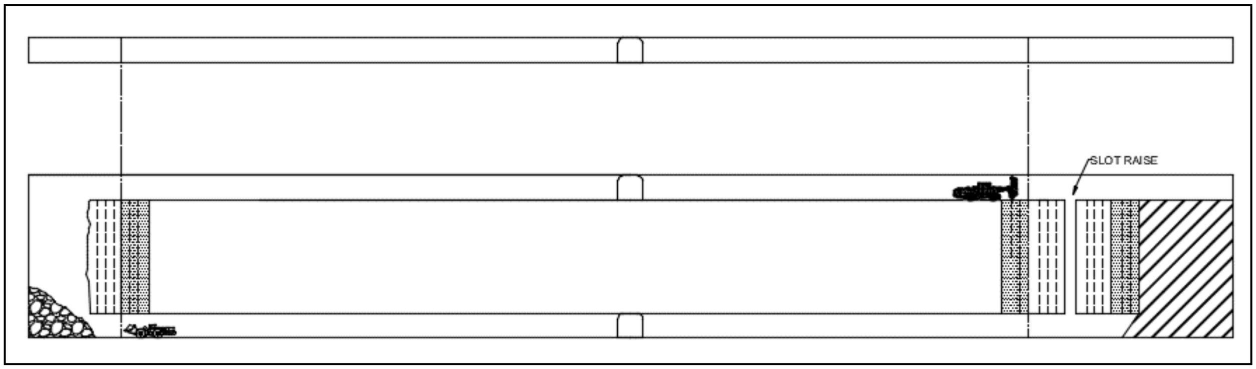


Figure 16-39: Stage 4 – Stope Blasting, Mucking, and Drilling (not to scale)

Stage 5 – Stope blasting, mucking, and backfilling (Figure 16-40):

- The third stope will be blasted and mucking will begin.
- After the second stope is mucked, it will be backfilled with CRF.

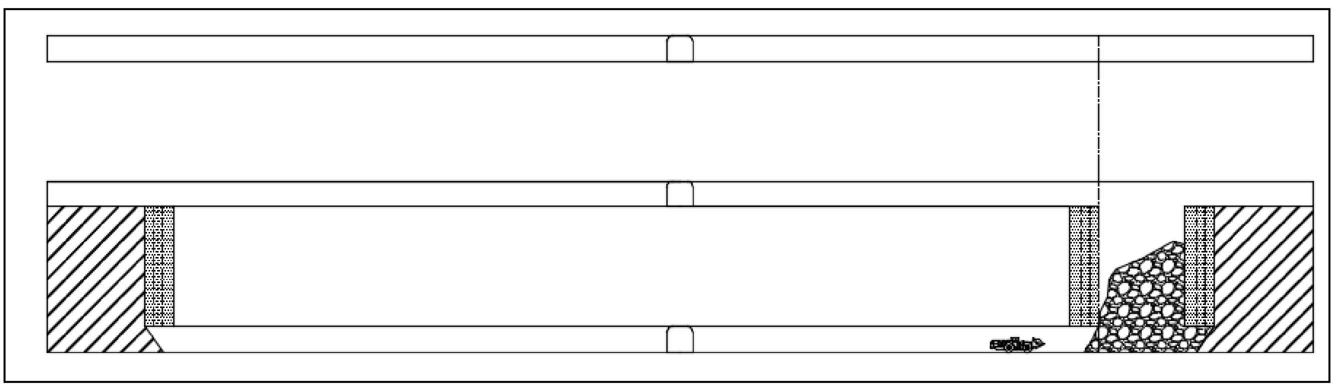


Figure 16-40: Stage 5 – Primary Stope Drilling, Mucking and Backfilling (not to scale)

Stage 6 – Stope drilling and TRP recovery (Figure 16-41):

- Once the first backfilled stope has cured, the first TRP can be blasted and mucked.
- Drilling will continue in the fourth stope.

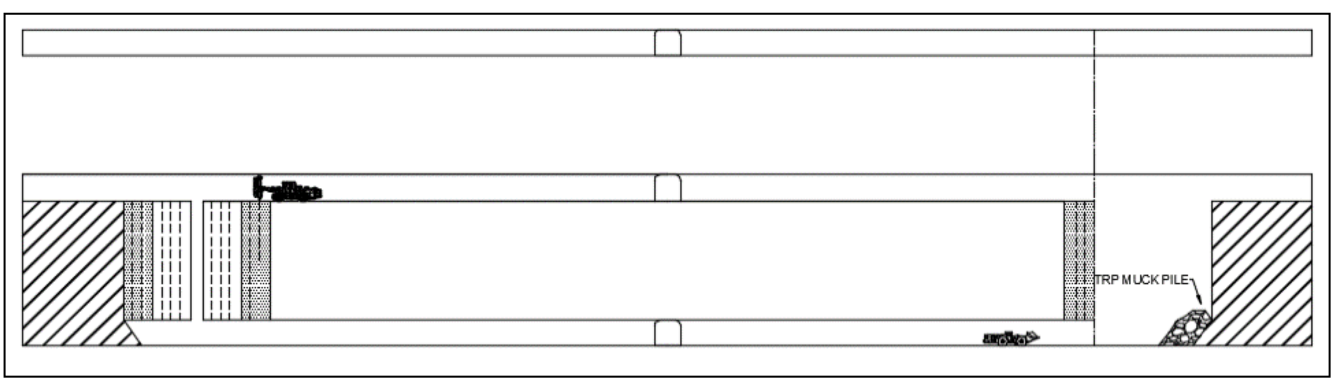


Figure 16-41: Stage 6 – Stope Drilling and TRP Recovery (not to scale)

Stage 7 – Stope drilling, blasting, mucking, and backfilling (Figure 16-42):

- While the second backfilled stope is curing, the fourth stope can be blasted and mucked.
- At this point, drilling begins on the fifth stope, one level above the current working level.

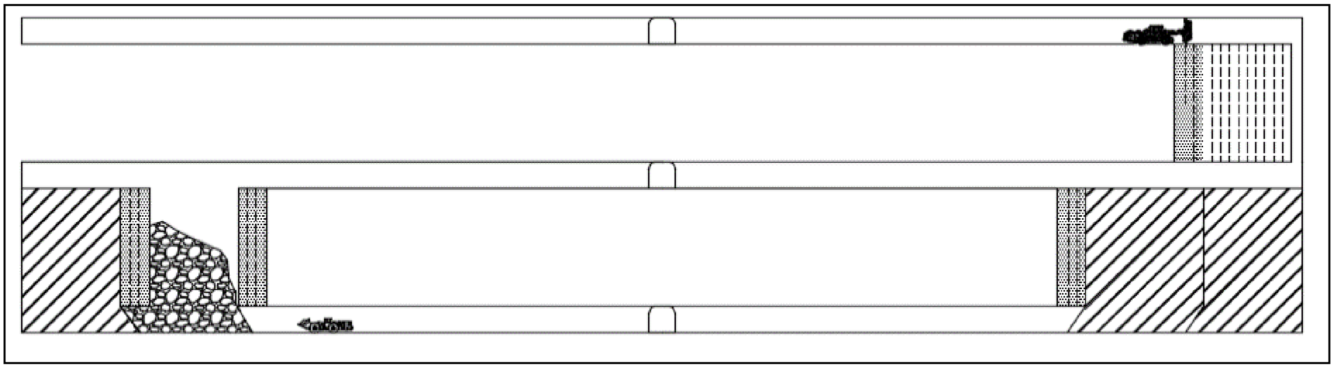


Figure 16-42: Stage 7 – Stope Drilling, Blasting, Mucking, and Backfilling (not to scale)

Stage 8 – Blasting, mucking, and TRP recovery (Figure 16-43):

- Once the second backfilled stope has cured, the second TRP can be blasted and mucked.
- Once the fifth stope is drilled, it can be blasted and mucked.

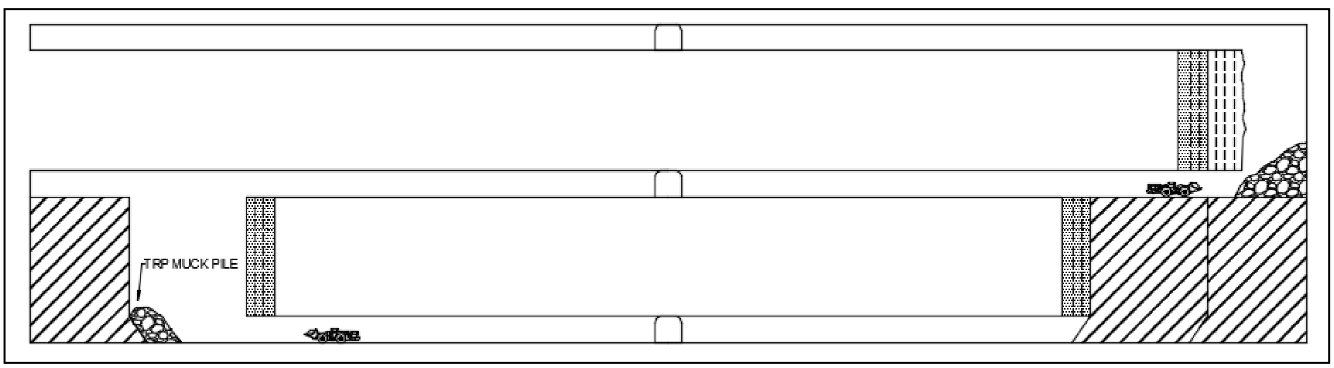


Figure 16-43: Stage 8 – Blasting, Mucking, and TRP Recovery (not to scale)

Stage 9 – Stope drilling and backfilling (Figure 16-44):

- Once the ore is mucked, the stope can be backfilled with CRF.
- During backfilling, stope drilling continues at the opposite end of the ore drive.
- The sequence can now be extended to a fourth drive level.

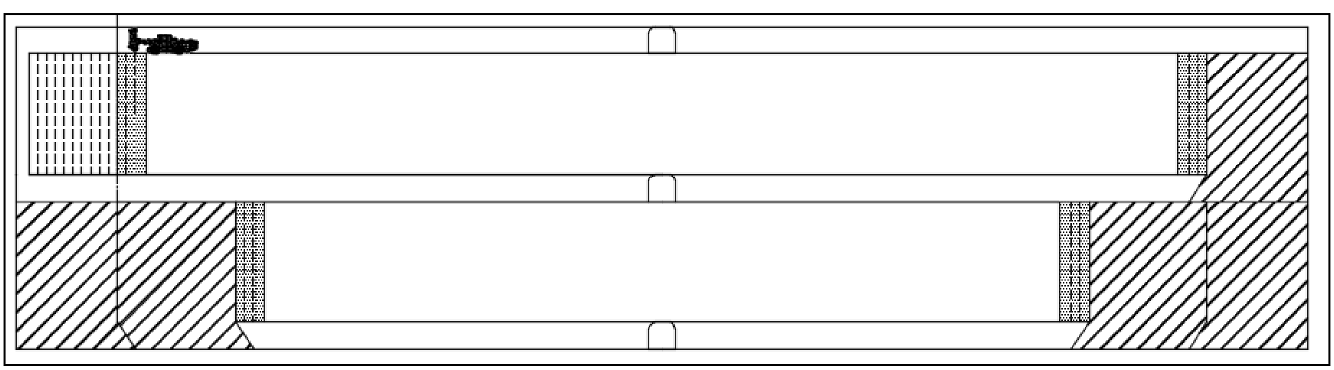


Figure 16-44: Stage 9 – Stope Drilling and Backfilling (not to scale)

The idealized sequence shown above is modified in the final mine schedule. Sometimes an up-hole drilling method will be utilized in areas where the top sill is not easily accessible. The diameter of the up-holes will be smaller to facilitate emulsion charging. The following table summarizes the average stope parameters.

Table 16-29: Average stope parameters for Main and Esso deposits

Average stope parameters	Units	Value
Height	m	25
Width	m	12.5
Length[1]	m	40
Stope dip	°	47-54
Drive height	m	5
Hole depth	m	26.9
Density in-situ	t/m³	3.56
Density loose	t/m³	2.64
Swell	%	35%
Stope volume	m³	6,250
Ore tonnes before losses and dilution	t	22,250
CRF required	t	12,963

Note: Typical stope length is 40 m but can vary between 20 m and 60 m depending on rock mass condition, stope thickness, and stope depth.

Figure 16-45 provides an example section view of the standard 25 m inter-level height for downhole blasting.

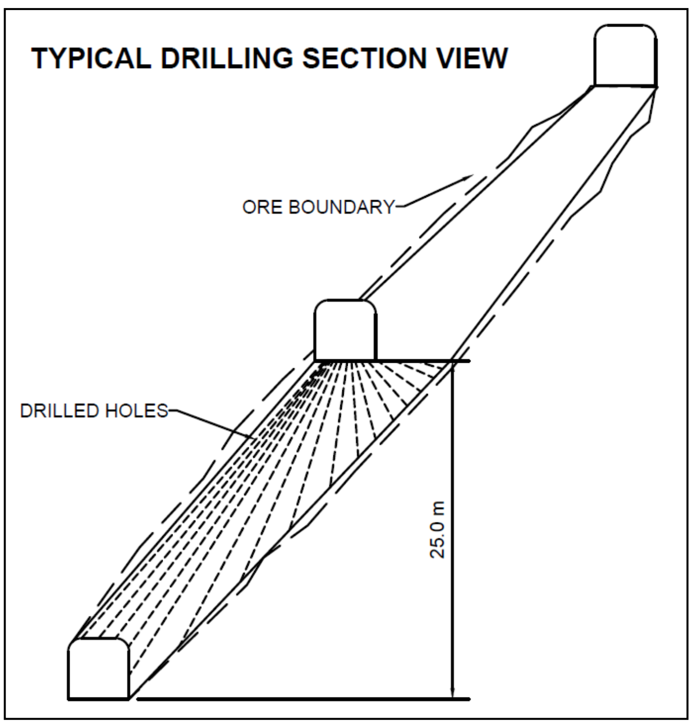


Figure 16-45: Typical Drilling Section View for a Typical Stope (not to scale)

16.5.2.2 Dilution and Recovery

The mining plan allows for mining losses, mining recoveries and mining dilution. These are detailed in Section 15. The net impact of these modifying factors is that all extracted stope ore grades will be reduced by 6.2% and a net gain of 2.2% will be applied to tonnage.

16.5.3 Underground Mine Production

Pre-production development is scheduled to take nine months and will include the Main access portal, Main central spiral ramp, and partial development of the Main-Esso connecting ramp. Access drifts, ore drives, safety bays and passing bays will be excavated during this period.

All levels for both Main and Esso deposits have similar access features as they are using the same mining method. Figure 16-46 shows overall access features for both Esso and Main deposits.

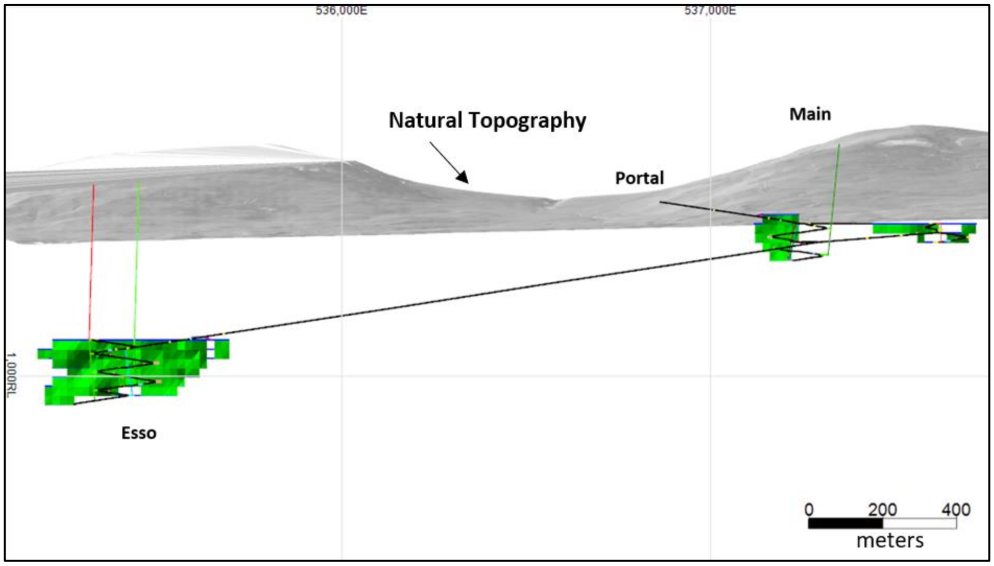


Figure 16-46: Sectional View Looking North of the Main and Esso Underground Mines

16.5.3.1 Underground Development

The primary access for the Main mine will be a single, straight decline from a starting floor elevation of 1470 masl. The cross-sectional dimensions will be 5 m high x 5 m wide to provide adequate clearance for equipment, ventilation, and services. Two spiral ramps will be driven off the primary access ramp, providing access to the Main deposit ore zones. The ramps will be driven at a maximum grade of ±15%. Figure 16-47 provides an overview for Main.

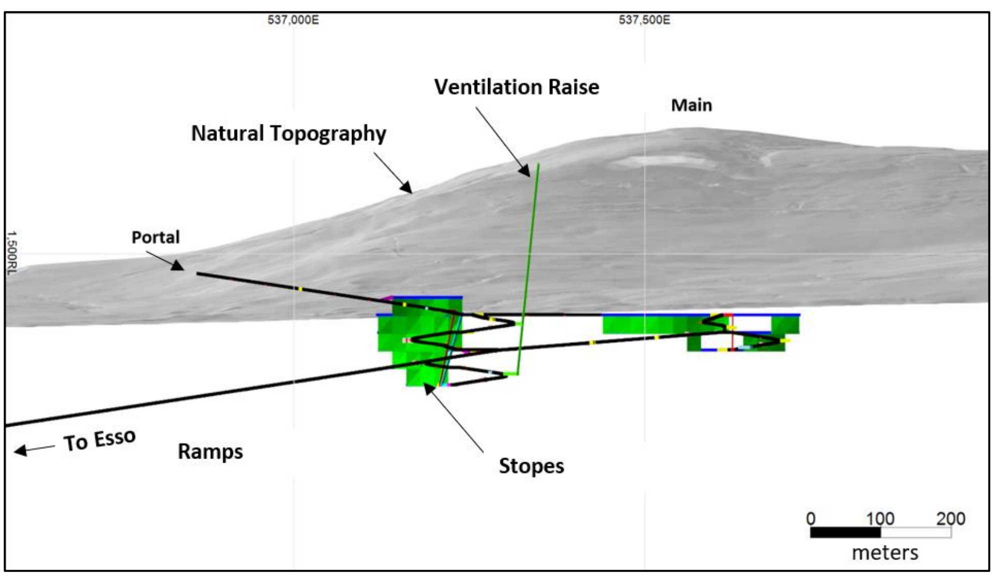


Figure 16-47: Sectional View Main Underground Mine (looking north)

Access to the Esso deposit will be via an 1,800 m connecting ramp extending from the Main access spiral. Cross sectional dimensions of the ramp will be 5 m high x 5 m wide and it will have an average grade of -15%. This ramp will connect the Main at the 1335 masl at the top of the Esso deposit at an elevation of 1090 masl. The access ramp will connect to a spiral ramp which will extend to the base of the Esso deposit. Lateral accesses will be driven off the spiral ramp and driven east and west to access the mining stopes. Although not designed for exploration purposes, the Esso access ramp could be drifted to the east for future exploration drilling of the Sumac deposit. Additionally, the Main–Esso connecting ramp may be used for accessing the Sumac deposit should its economic extraction be warranted.Figure 16-48 provides overview of access to Esso and Figure 16-49 provides the overall view of the fully developed mine.

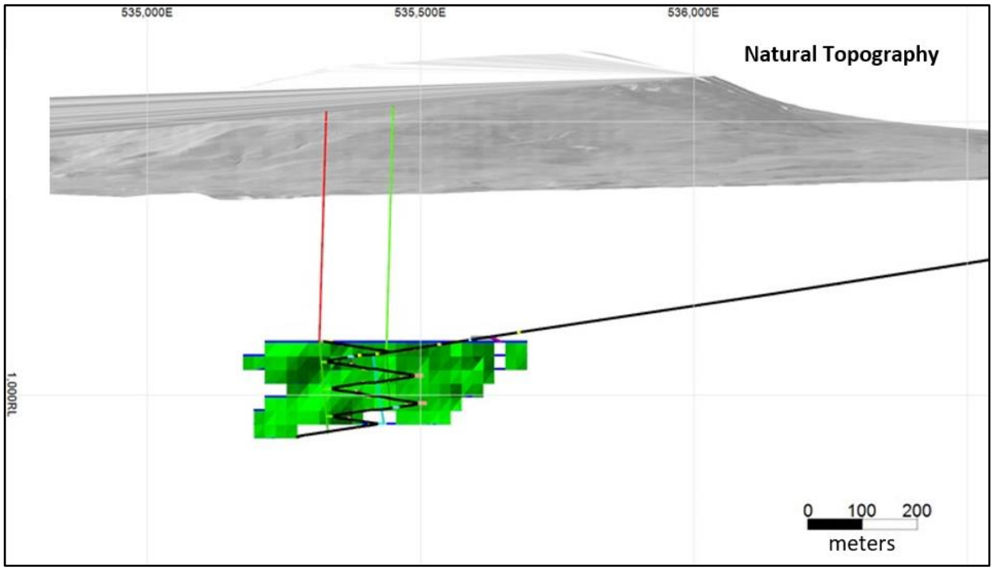


Figure 16-48: Esso Access Development and Stopes (view to the north)

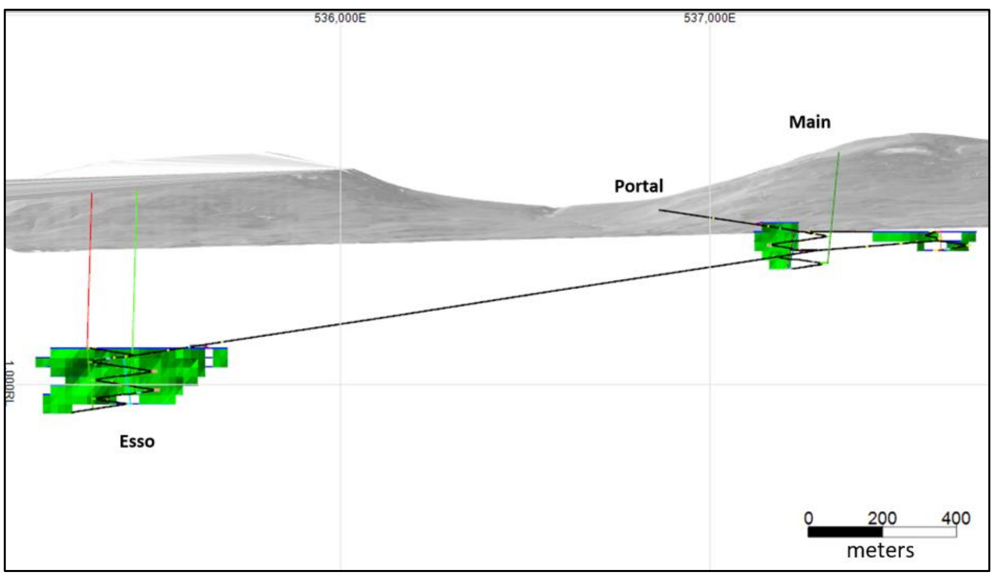


Figure 16-49: Fully Developed Underground Mine (looking to the north)

The pre-production development period is estimated to be approximately nine months. All lateral capital development is assumed to be carried out by Kutcho.

The underground mine will require an extensive system of raises for ventilation and ore handling. Each spiral ramp will have a central ore pass system with dump points at various levels to minimize internal haulage. A truck loading chute will be installed at the bottom of each ore pass.

There will be two exhaust raises to surface, one at the Main deposit and one at the Esso deposit. Fresh air will be supplied to the Main deposit via the portal and to Esso via a ventilation raise. An internal system of fresh air raises will also be required to complete Main deposit's primary and secondary ventilation circuits.

The ramp to Esso mine will have one intake ventilation raise to surface, and a series of internal raises which will be used to ventilate the ramp during development and will form part of the primary ventilation circuit. One additional raise to surface and additional internal ventilation raises will be required for exhaust air. Figure 16-50 will provide an overall perspective on the ventilation raises, access and spirals.

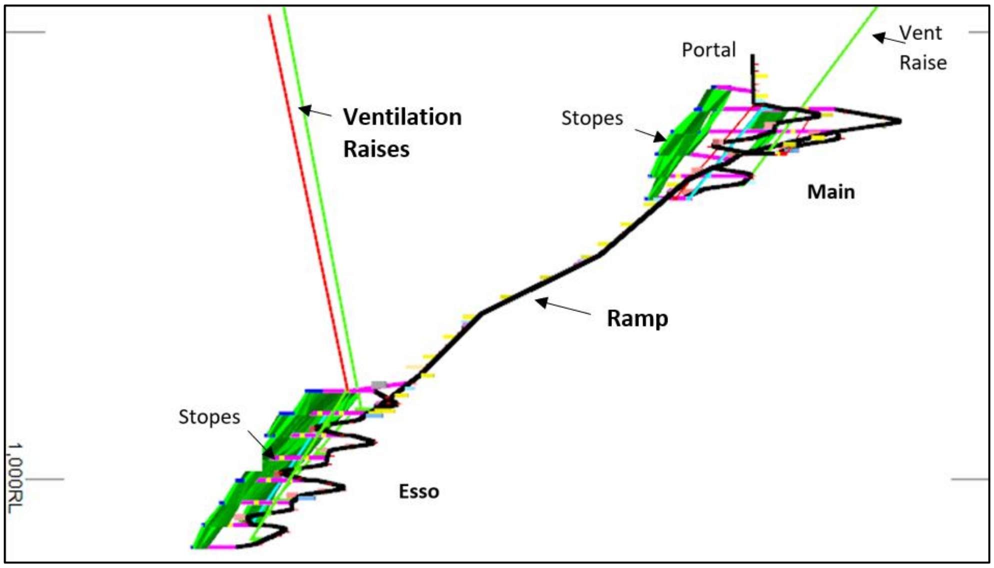


Figure 16-50: Isometric View Looking West of All Underground Development and Stopes (not to scale)

Ore drives will be accessed from the spiral ramps at 25 m vertical intervals as defined by the planned stoping heights. Lateral ramp development in the footwall is set back from the ore body by a minimum of 25 m from the ore contact and faults (data provided by Terrane – Faults_REV2.zip March 11, 2021). This setback of 25 m provides long-term geotechnical stability and provides adequate space for the placement of a fresh air raise and other ancillary services between the ramp and level development.

Major underground infrastructure includes ventilation raises, ore passes, pumping stations, settling ponds, electrical substations, explosive and detonator magazines, and other ancillary installations.

Table 16-30: Ancillary development and excavation dimensions for Main and Esso

Description	Section	Length (m)	Quantity
Electrical bays	4.0 m wide x 4.0 m high	10.5	11
Escape way access	5.0 m wide x 5.0 m high	Variable	7
Escape way raise	2.0 m wide x 2.0 m high	Variable	2
Exploration access	6.0 m wide x 7.0 m high	12.5	14
Explosive magazine	5.0 m wide x 4.0 m high	22.5	1
Level access	5.0 m wide x 5.0 m high	Varies	20
Preventive-maintenance	16.0 m wide x 8.0 m high	42.5	1
Mucking bays	5.0 m wide x 5.0 m high	14.5	34
Ore drives	4.5 m wide x 4.5 m high	Varies	18

Description	Section	Length (m)	Quantity
Ore pass access	5.0 m wide x 5.0 m high	Varies	11
Ore pass raise	3.0 m wide x 3.0 m high	Varies	2
Passing bay	4.0 m wide x 5.0 m high	20	5
Ramp	5.0 m wide x 5.0 m high	Varies	_
Refuge chamber	3.5 m wide x 5.0 m high	22.5	2
Safety bays	2.0 m wide x 2.0 m high	4.5	61
Settling pond	11.0 m wide x 5.0 m high	4.5	3
Sump	5.0 m wide x 5.0 m high	14.5	10
Ventilation access	5.0 m wide x 5.0 m high	Varies	10
Ventilation raise	3.0 m wide x 3.0 m high	Varies	3

Table 16-31: Minimum distance between bays

Description	Minimum distance between bays
Safety bays	50 m
Mucking bays	150 m
Passing bays	450 m
Electrical bays	450 m

16.5.4 Ramp and Access Development

All ramp and access development will be in the footwall of the Main and Esso orebodies. Access to the Main orebody will be via a ramp starting from a surface elevation of 1470 masl. Two spiral ramps will branch off the main ramp and provide access to the Main underground orebody. An 1,800 m long ramp will connect the access level at 1335 masl of the Main deposit to the access level at the 1090 masl for the Esso deposit. Esso has one spiral ramp to provide access to the Esso orebody.

The design of the Main–Esso connector ramp considers the shortest distance between the two orebodies whilst giving safe offset to the intervening Sumac deposit.

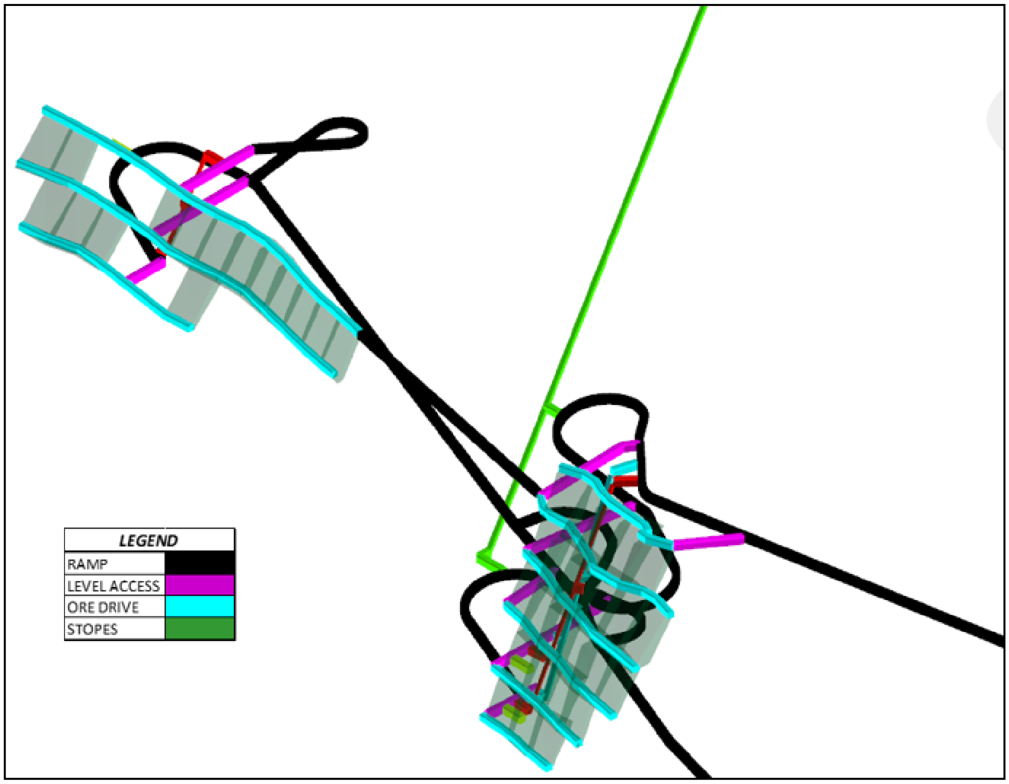


Figure 16-51: Isometric View Looking to the Southeast of Main Deposit Showing Development and Stopes (not to scale)

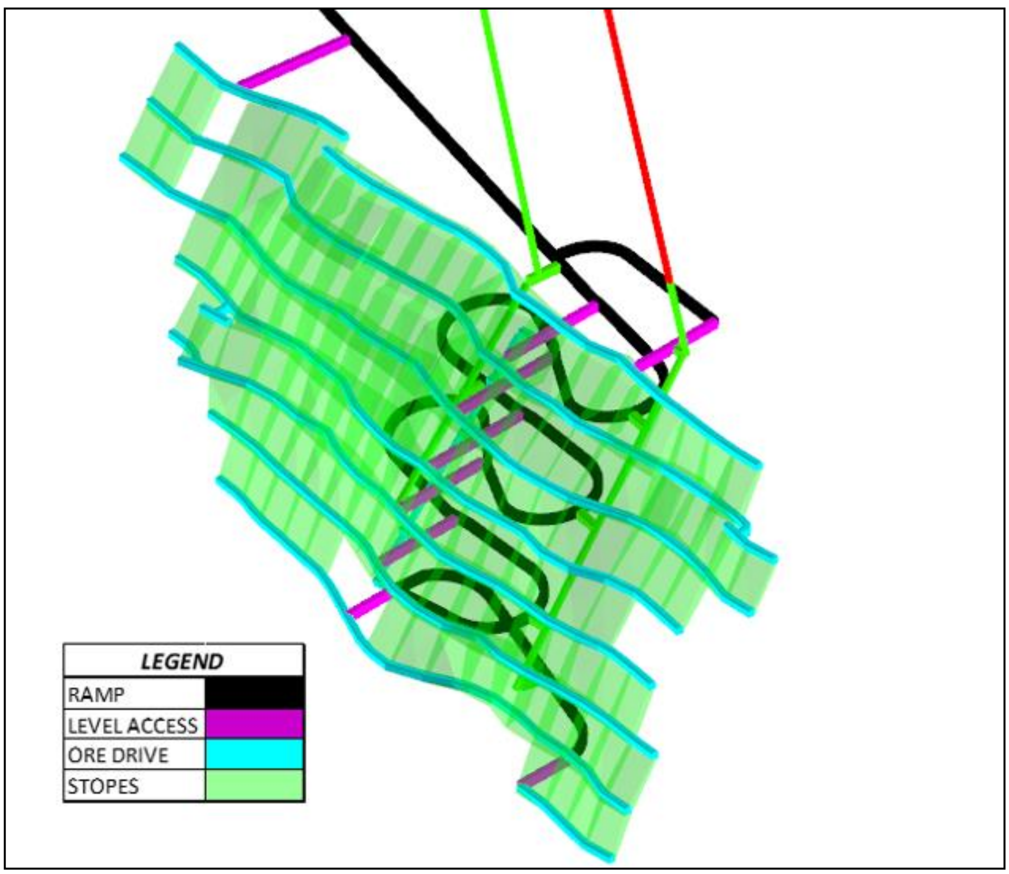


Figure 16-52: Isometric View Looking to the Southeast of Esso Deposit Design (not to scale)

Table 16-32: General ramp parameter summary

General ramp parameter summary	Units	Value
Height	m	5.0
Width	m	5.0
Advance per blast	m	4.39
Arch radius	m	1.0
Perimeter (backs and walls)	m	14.1
Face area	m^2	24.6
Over break	%	5
Density insitu	t/m^3	2.8
Density loose	t/m^3	2.07
Swell	%	35
Tonnes per round	t	433.4
Tonnes per metre	t	72.2

16.5.4.1 Lateral Development Standards

The lateral development standards consider mobile equipment size and clearance requirements, provision of ventilation ducting and other services, ground support requirements, and the ultimate intended use of the excavation. All development was sized to accommodate the largest anticipated equipment plus an allowance of 1 m minimum clearance on each side. The ore drives (4.5 m wide) are designed to provide adequate room for efficient production drilling. Table 16-33 shows the standard lateral development dimensions. A fleet of LHDs and trucks will be used for waste material transport from the various underground working areas through the internal ramp system and to surface. Some waste material will remain underground to be used as backfill material. An ore pass system will be utilized for both the Main and Esso deposits; ore passes facilitate the transportation of ore to the lower levels where it will be loaded into trucks and hauled to surface.

Table 16-33: Standard lateral development dimensions

Lateral development	Lateral development	
	Width (m)	Height (m)
Ramp (Primary and Secondary Declines)	5.0	5.0
Passing bays	4.0	5.0
Electrical bays	4.0	4.0
Ore drives	4.5	4.5
Escape way access	5.0	5.0
Exploration access	6.0	7.0
Refuge chamber	3.5	5.0
Explosive magazine	5.0	4.0
Level access	5.0	5.0
Mucking bays	5.0	5.0
Safety bays	1.8	2.0
Settling ponds	11.0	5.0
Sump	5.0	5.0
Ventilation raise access	5.0	5.0

Table 16-34: Standard vertical development dimensions

Name	Vertical development
	Diameter (m)
Escapeway	2.0 x 2.0
Ore pass	3.0 x 3.0
Ventilation	3.0 x 3.0

Table 16-35: Development distances

Development area	Units	Main	Esso	Total
Drifting	m	4,765	7,637	**12,402**
Permanent waste drives	m	2,651	4,040	**6,692**
Development ore drives	m	1,915	3,307	**5,222**
Ventilation raise	m	378	1,230	**1,608**
Escape ways	m	198	-	**198**
Ore passes	m	134	153	**286**

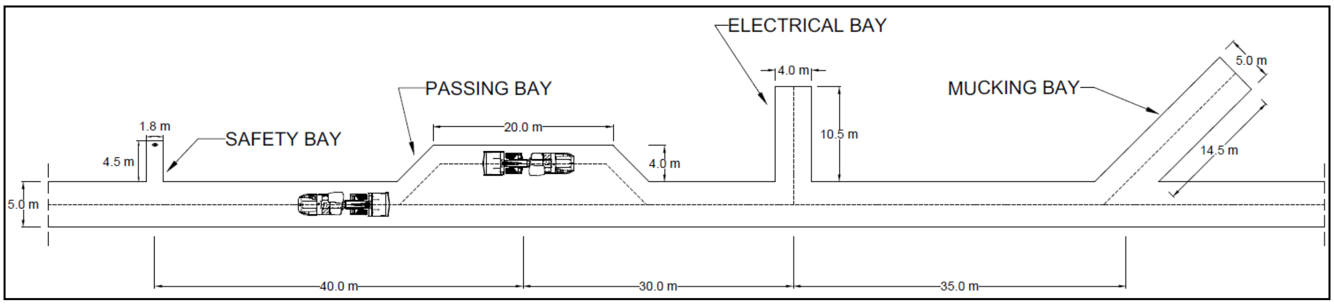


Figure 16-53: Ramp Bay Concepts – Plan View

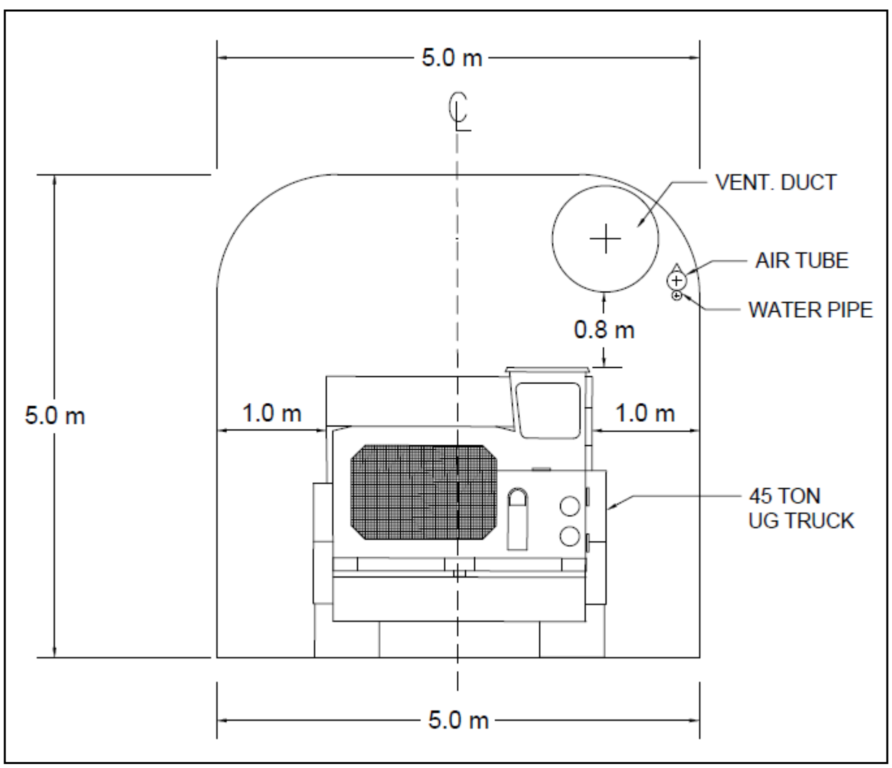


Figure 16-54: Typical Ramp Cross-Section View

16.5.4.2 Crown Pillar

The minimum recommended crown pillar height between the Main deposit open pit and the underlying stopes was determined to be 25 m based on recommendations from Terrane in the preliminary study. Crown pillars were excluded from Mineral Reserves. Underground mining in on the Main deposit will be well in advance of the open pit progress. The minimum coeval operational separation is greater than 100 m.

16.5.4.3 Mine Schedule Stope Extraction Order

LLHOS method requires backfilling of the mined-out stope prior to extracting the stope directly above it since restricting the mine to one working face will delay the mining cycle and production rate. Mining will start at the lowest level and work up in both the Main and Esso deposits. Mining will start at the easternmost and westernmost limits of the deposit and retreat towards the middle. This will allow multiple working faces and stopes to be accessed.

Stope mining will begin on the 1310 masl elevation of the Main deposit, and the first stope at Main is scheduled to be blasted during Q4 2025 (Year -2). Mining of the Main deposit will continue retreating towards the east with primary access e being the central spiral ramp. The first stope at the easternmost end of the Main underground mine (1360 masl elevation is scheduled to be blasted) during Q2 2026 (Year -1) and mining will continue retreating towards the west as development of the east ramp continues.

The first stope in the Esso deposit is scheduled to be mined during Q4 2027 (Year 1). Mining will commence on the 945 masl elevation and retreat towards the west. Sequentially, the next stope will be on the western side of the deposit on the 920 masl elevation. Mining on this level will retreat towards the east.

The extraction sequence for extraction of the TRP's utilizes remote wireless detonation.

Remote wireless detonation makes use of the latest electronic blasting systems. This system allows for groups of in-hole primers to be wirelessly initiated by firing command. The system eliminates the need for down-lines and surface connecting wires, communicates through rock, air and water to initiate blasts reliably and safely, removing people from harm's way. This wireless initiating system can:

- Eliminate the need to work near rill or the brow and reduce the impact of brow loss
- Significantly reduce loss and underperforming holes caused by lead damage/dislocation while eliminating rework
- Produce more uniform fragmentation and improved draw extraction through reliable initiation.

16.5.4.4 Slot Raise Drill Pattern

The slot raise will be blasted in one round, with the following parameters, as illustrated in Table 16-36.

Table 16-36: Slot raise drill pattern parameters

Parameter	Value
Blasthole diameter (mm)	106–116
Burden (m)	2.6
Spacing (m)	3.1

The slot raise burden and spacing distances are calculated based on generally accepted empirical rules and adjusted to fit the particular slot geometry. It is common to have different design standards for slot rings and production rings. Improved reliability of slot development is crucial for overall production of the entire stope, therefore reduced burden and spacing values are common. One of the general empirical slot design rules states that the burden and spacing distances are approximately 75% of the "normal" burden and spacings of the main production blast holes. This reduced burden and spacing increases the drill density, and significantly improves the reliability of slot firings.

16.5.4.5 Production Rings Drill Pattern

The production ring pattern for the downhole rounds uses the following parameters:

Table 16-37: Production rings drill pattern parameters

Parameter	Value
Blast Hole Diameter (mm)	106–116
Burden (m) (for 25 mH stopes)	1.8
Spacing (m)	3.1

The ring burden dimension is a function of the stope width and drillhole diameter whereas toe spacing is a function of both burden and stope geometry. Proposed ring burden and spacing values are calculated based on widely accepted empirical rules for underground stopes of similar size and geometry.

For example, Orica guidelines suggest the following practical relationships for underground blasting, as illustrated in Table 16-38.

Table 16-38: Practical burden and spacing for underground blasting recommended by Orica

Parameter	Value
Burden (m)	23 to 27 x blasthole diameter
Spacing (m)	1 to 1.5 x burden

Note that reported values for toe spacing are calculated design values whereas hole positions are adjusted to ensure even drillhole distribution during the design process depending on stope size and geometry. As such, actual toe spacing values may be insignificantly different from the calculated design values (less than 10% deviation from design spacing).

16.5.4.6 Underground Production Rate

Underground ore production will begin at the Main deposit at the end of 2025 (Year -2) and will continue until 2028 (Year 2) when Esso production begins. Production at Esso is scheduled to continue until 2036 (Year 10). average daily underground ore production rates were calculated by dividing the scheduled annual ore production by 360 working days per year. Annual average underground ore production rates (not including Year -2) range from a maximum of 1,028 tpd to a minimum of 92 tpd as production winds down. The average mine life ore production rate is 654 tpd or 775 tpd excluding Years -2 and 10. Including waste, the average LOM production rate is 820 tpd or 964 tpd excluding Years -2 and 10.

Table 16-39 summarizes the major underground operating targets. Table 16-40 summarizes the LOM grades and production rates for underground mining.

Table 16-39: Major operating targets

Major operating targets	Unit	Value
Target production rate	tpd	1,000
Operating days	days per year	360
Approximate total tonnage UG (Main+Esso)	kT	2,827

Table 16-40: LOM underground head grades and production rate for Main and Esso

Year #	LOM UG	-2	-1	1	2	3	4	5	6	7	8	9	10
Year	Average	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036
Production (tpd)	654	9	406	877	585	988	995	905	1028	808	488	671	92
Waste (tpd)	497	100	862	689	338	-	-	-	-	-	-	-	-
Copper grade (%)	1.98	1.14	1.39	1.40	1.59	1.67	2.17	2.34	2.36	2.21	2.40	2.09	1.42
Zinc grade (%)	3.58	1.19	1.13	1.62	2.47	2.97	3.82	4.41	4.68	4.67	4.23	4.43	4.41
Silver grade (g/t)	50.1	29.5	34.4	32.0	33.5	51.8	56.2	67.6	56.6	46.8	59.4	50.7	44.2
Gold grade (g/t)	0.68	0.5	0.42	0.69	0.48	0.58	0.72	0.92	0.67	0.66	0.67	0.75	1.08

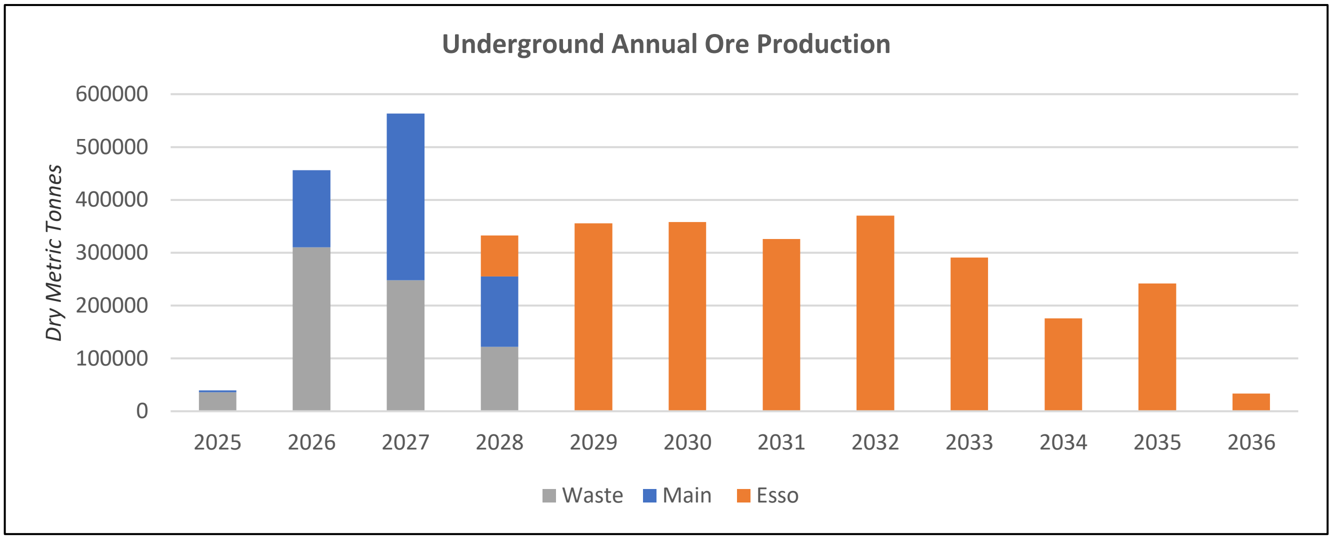


Figure 16-55: Underground Annual Production Rate

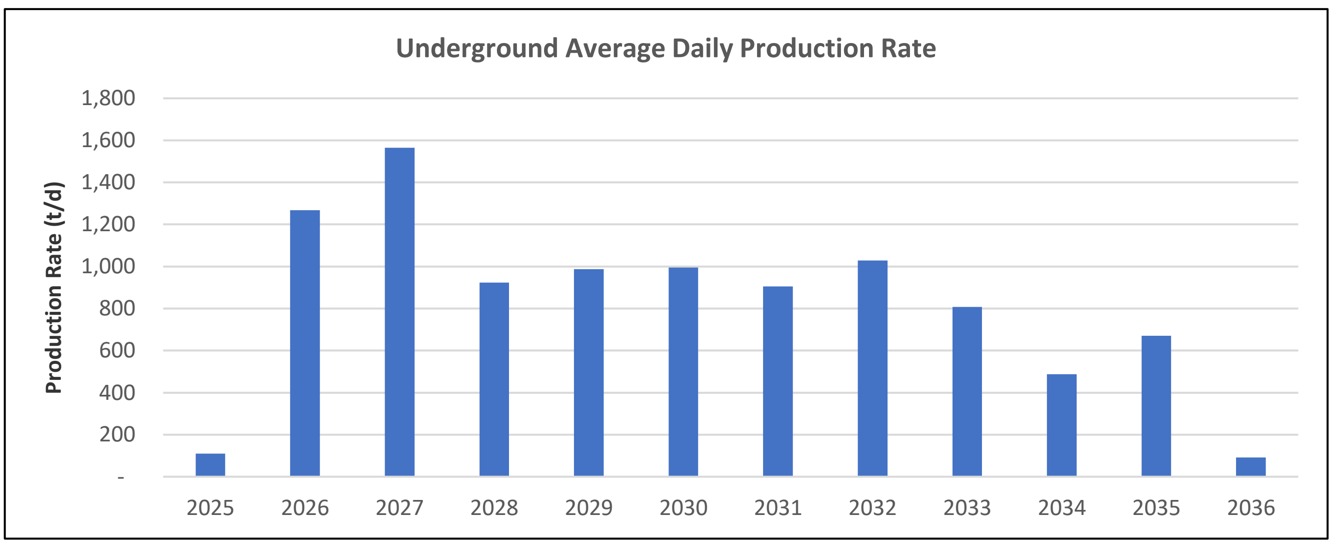


Figure 16-56: Underground Average Daily Production Rate

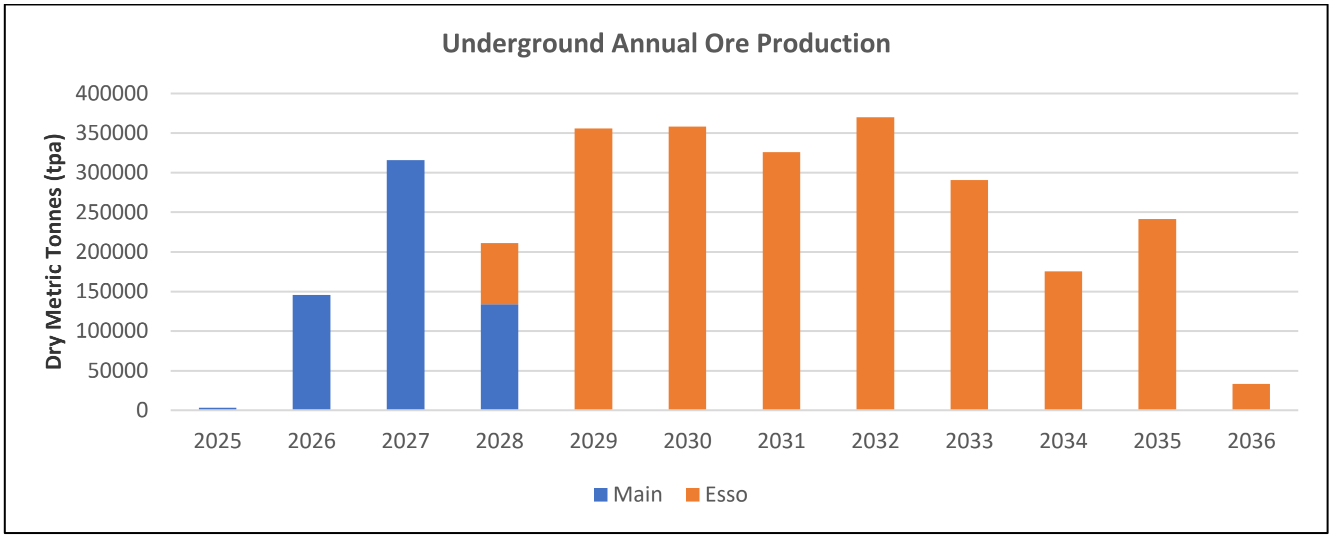


Figure 16-57: Underground Annual Ore Production

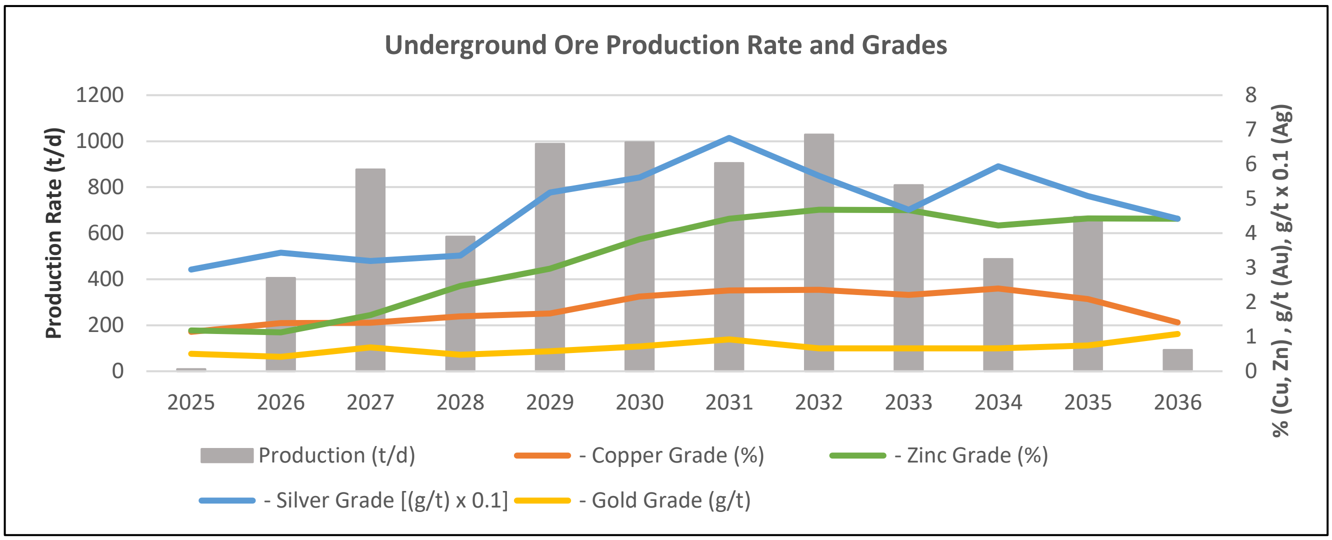


Figure 16-58: LOM Underground Ore Grades and Production Rate

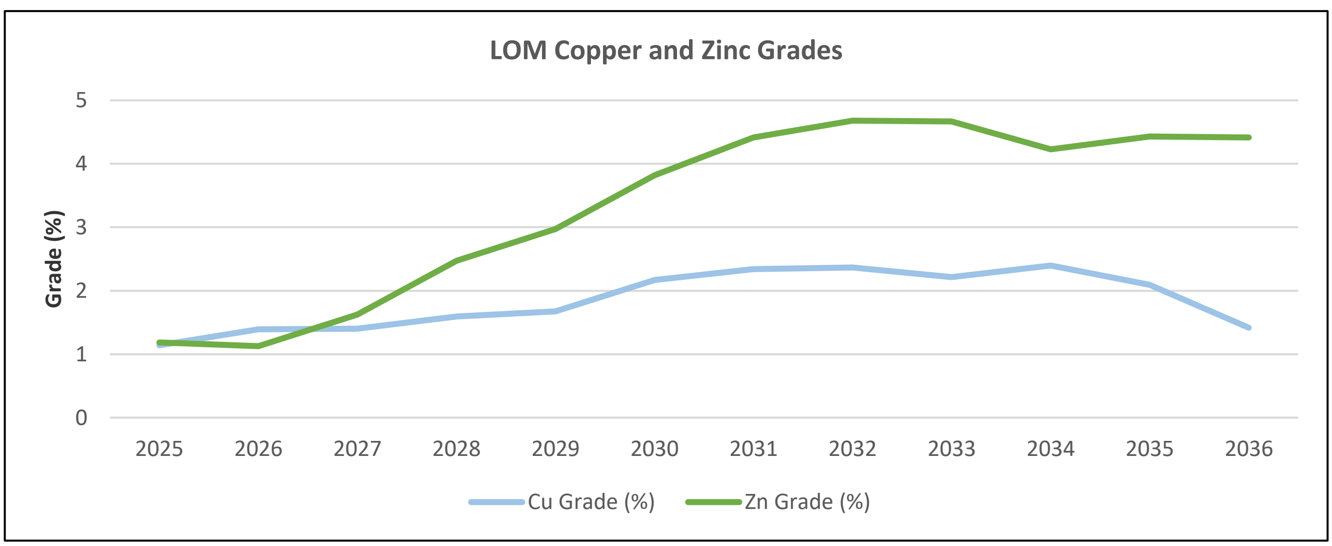


Figure 16-59: LOM Underground Copper and Zinc Grades

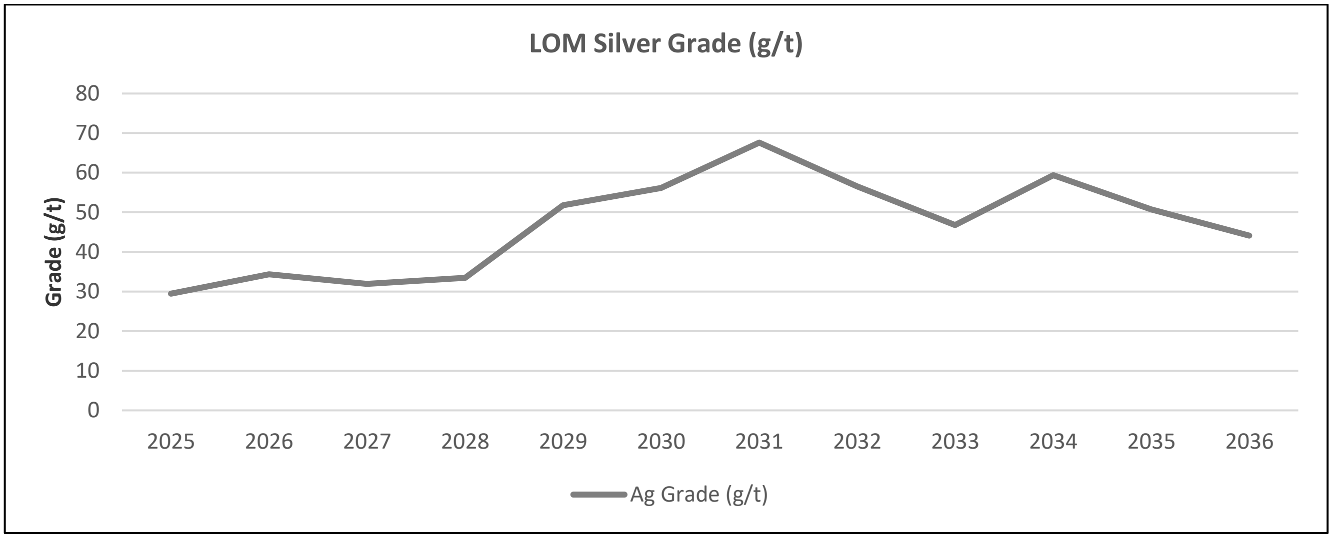


Figure 16-60: LOM Underground Silver Grade

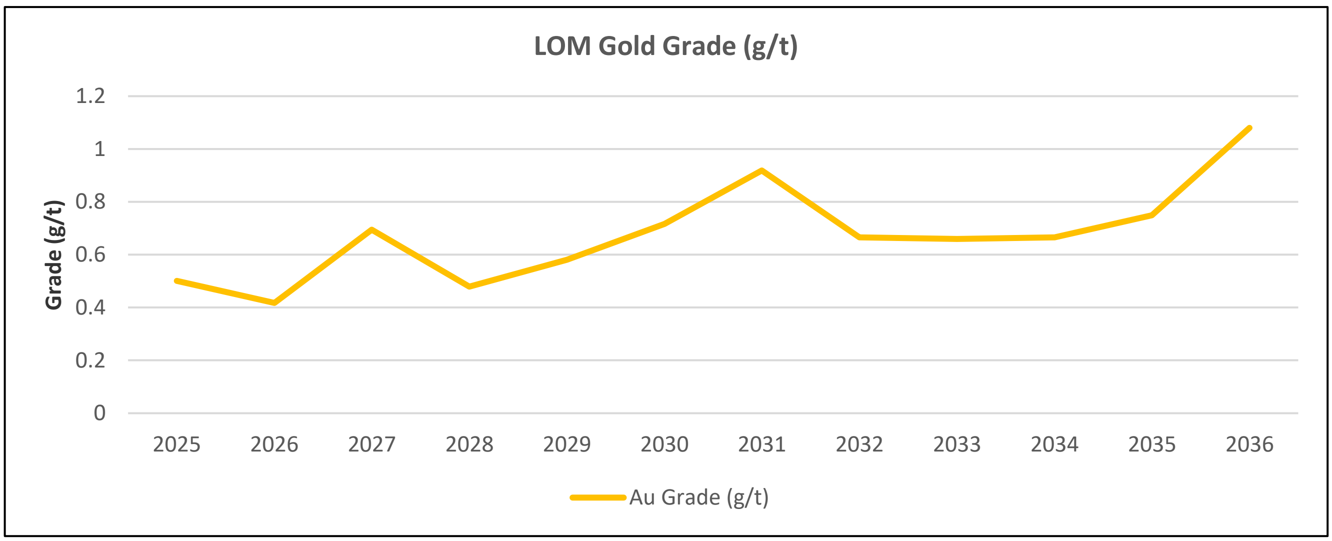


Figure 16-61: Underground Gold Grade

16.5.4.7 LOM Production Schedule

Underground production is scheduled to begin in 2026 (Year -1) with about 150 kdmt of ore being mined from the Main underground deposit. Underground production at Main doubles to about 320 kdmt of ore in 2027 (Year 1) before reducing to about 130 kdmt or ore in 2028 (Year 2).

Esso production is scheduled to begin in 2028 (Year 2) with 80 kdmt mined, supplementing ore from the Main deposit. Production at Esso ramps up to 360kdmt for the next two years (2029 and 2030), peaking at 370 kdmt in 2032. The production rate is scheduled to reduce to to 290 kdmt in 2033, 180 kdmt in 2034, 240 kdmt in 2035, and 30 kdmt in 2036.

Waste will be mined over a four-year period, starting with about 35 kdmt in 2025 (Year -2), about 310 kdmt in 2026 (Year -1), about 250 kdmt in 2027 (Year 1), and about 120 kdmt in 2028 (Year 2). No waste is scheduled to be mined over the remainder of the mine life.

Table 16-41: LOM underground ore tonnage mined (in Mdmt)

Ore mined	Units	Year	-2	-1	1	2	3	4	5	6	7	8	9	10
		LOM	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036
Main	Mdmt	0.60	0.00	0.15	0.32	0.13	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Esso	Mdmt	2.23	0.00	0.00	0	0.08	0.36	0.36	0.33	0.37	0.29	0.18	0.24	0.03

16.5.4.8 Cemented Rockfill

The longitudinal longhole stoping method requires backfill to allow the planned stope extraction sequence. The backfill will not only support the walls of the stopes but will provide a surface for mucking the overlying? stope. Cemented rockfill (CRF) was selected as the backfill material for the Project.

The CRF must remain stable and self-supporting, and different cement and/or binder content may be used depending on required stope dimensions and expected exposure duration. The performance of the CRF is susceptible to many variabilities including moisture content, gradation, geophysical rock properties, temperature, and the quality of the mixing.

Different backfill strengths may be desired depending on the stope dimensions, geotechnical regime, and proximity to nearby workings. This may be achieved by blending different amounts of binder or cement, adjusting the gradation, and/or modifying the installation method, among other things.

Typically, the unconfined compressive strength (UCS) of CRF backfill is used as the benchmark for the limiting strength of a backfill mix, whereby UCS is taken as a proxy for Young's modulus, tensile strength, and cohesion. Since the UCS is a simple test, it can easily be incorporated into a routine fill testing program to ensure that the strength requirements are being achieved (Stone, 1993) (Terrane Geoscience Inc., 2021).

To achieve a σucs = 0.72 MPa based on published values (Stone, 1993) and benchmarking studies at other mine sites it is recommended that 2-3% Portland cement content by weight be used as a binder. A water to cement ratio of 1:1 should be used in the CRF backfill. The aggregate used for the preparation of CRF backfill should be 4" minus material with fines kept in the range of 25-35% passing a 3/8" mesh sieve. (Terrane Geoscience Inc., 2021). Most of the rock fill will be sourced from the ore sorter reject material from the processing plant. This material fits the requirements for backfill from TGI in that it does not contain less than ½" and has a top size of 3". Prior to the ore sorter reject material being available, a mobile crushing unit will be used to produce the product range using waste drive material as the feed stock.

This empirical method is considered sufficient for this phase of work however, more detailed 3D stress modeling should be completed to confirm these results (Terrane Geoscience Inc., 2021)

The summary of CRF parameters, material properties and CRF properties are presented in Table 16-42, Table 16-43 and Table 16-44.

Table 16-42: Summary of CRF parameters

Parameter	Value
UCS for CRF	0.72 Mpa
Portland cement content by weight	2 to 3%
Water:cement ratio	1:1
Aggregate size, 100% passing	4" (100 mm) mesh
Fines range, 25 to 35% passing	3/8" (9.5 mm) mesh

(Source:Terrane Geoscience Inc., 2021)

Table 16-43: Material properties

Material	Specific gravity	Bulking factor
Ore (as per resource block model)	3.6	35%
Waste (as per resource block model)	2.8	35%

Table 16-44: Backfill design parameters

Backfill design parameters	Unit	Value
Fill factor	%	95
Backfill – paste plant capacity	m^3/hr	40
Cure time target	days	21
Cement – backfill (stopes)	%	2.5
Cement – backfill (sill pillars)	%	10.0

16.5.5 *Ventilation Design*

Mine ventilation requirements for the Project were examined in two different sections as suggested by the mine production plan. To attain the required production levels at Main, 75 m^3/s (159 kcfm) of fresh airflow is needed through the main haulage way. This quantity can be supported with ~40 m^3/s (85 kcfm) provided from the Esso network allowing 115 m^3/s (244 kcfm) for Main Deposit ventilation including the East Ramp. The Esso deposit requires 115 m^3/s of fresh air which will be provided through a dedicated single intake and single exhaust ventilation system in the Esso network.

The summary of mobile equipment list for underground operations are presented in Table 16-45.

Table 16-45: Equipment list for underground operations

Equipment	No. of units	Engine power (hp)	Total power (hp)	Availability	Utilization	Net (hp)	Air (cfm)	Air (m^3/s)
Truck (45 t/20 m^3)	3	450	1350	0.85	0.85	975.375	97,538	46.0
LHD (6.7 t/3.0 m^3)	2	220	440	0.85	0.85	317.900	31,790	15.0
LHD (6.7 t/3.0 m^3) Ram Jam	1	285	285	0.85	0.85	205.913	20,591	9.7
LHD (10 t/4.0 m^3)	5	325	1625	0.85	0.85	1174.06	117,406	55.4
Jumbo – Two Boom	2	192	384	0.85	0.2	65.280	6,528	3.1
Longhole Drill – Top Hammer	2	100	200	0.85	0.2	34.000	3,400	1.6
Infill Drill	1	100	100	0.85	0.2	17.000	1,700	0.8
Small Explosives Truck	1	180	180	0.85	0.4	61.200	6,120	2.9
Large Explosives Truck	1	225	225	0.85	0.4	76.500	7,650	3.6
Bolter	2	100	200	0.85	0.3	51.000	5,100	2.4
Shotcrete Sprayer – Mobile	1	135	135	0.85	0.8	91.800	9,180	4.3
Shotcrete Sprayer – Manual	1	135	135	0.85	0.8	91.800	9,180	4.3
Grout Pump	1	110	110	0.85	0.3	28.050	2,805	1.3
Scissor Lift	2	165	330	0.85	0.3	84.150	8,415	4.0
Boom Truck	1	145	145	0.9	0.5	65.250	6,525	3.1
Telehandler	1	120	120	0.9	0.5	54.000	5,400	2.5
Utility Vehicle	1	150	150	0.9	0.7	94.500	9,450	4.5
Transmixer	2	175	350	0.9	0.7	220.500	22,050	10.4
Mobile Rock Breaker	1	145	145	0.9	0.5	65.250	6,525	3.1
Mechanics Truck	1	110	110	0.9	0.5	49.500	4,950	2.3
Fuel/Lube Truck	1	110	110	0.9	0.5	49.500	4,950	2.3

Equipment	No. of units	Engine power (hp)	Total power (hp)	Availability	Utilization	Net (hp)	Air (cfm)	Air (m^3/s)
Electrician Truck	1	110	110	0.9	0.5	49.500	4,950	2.3
Personnel Carrier	3	165	495	0.9	0.5	222.750	22,275	10.5
Supervisor Truck	1	110	110	0.9	0.5	49.500	4,950	2.3
Crew Van/Ambulance	1	185	185	0.95	0.3	52.725	5,273	2.5

Ventilation requirements are based on equipment selection and utilization as illustrated in Table 16-45. To properly estimate the airflow demand and associated heat emissions from the equipment listed, different machine utilization schemes are applied. These utilization rates are selected based on the operational priority and utilization intensity of the equipment. Accordingly, while service equipment and auxiliary machinery is kept in low utilization (at 15–40%), production equipment are considered as highly utilized (at 85–90%) during regular mining practices. The equipment selection was based on the production needs of the mine and will be employed according to the mine planning.

The VentSIM™ software was used for ventilation simulation. Figure 16-62 and Figure 16-63 illustrate the general outline of the critical ventilation path analysis done in the simulation software.

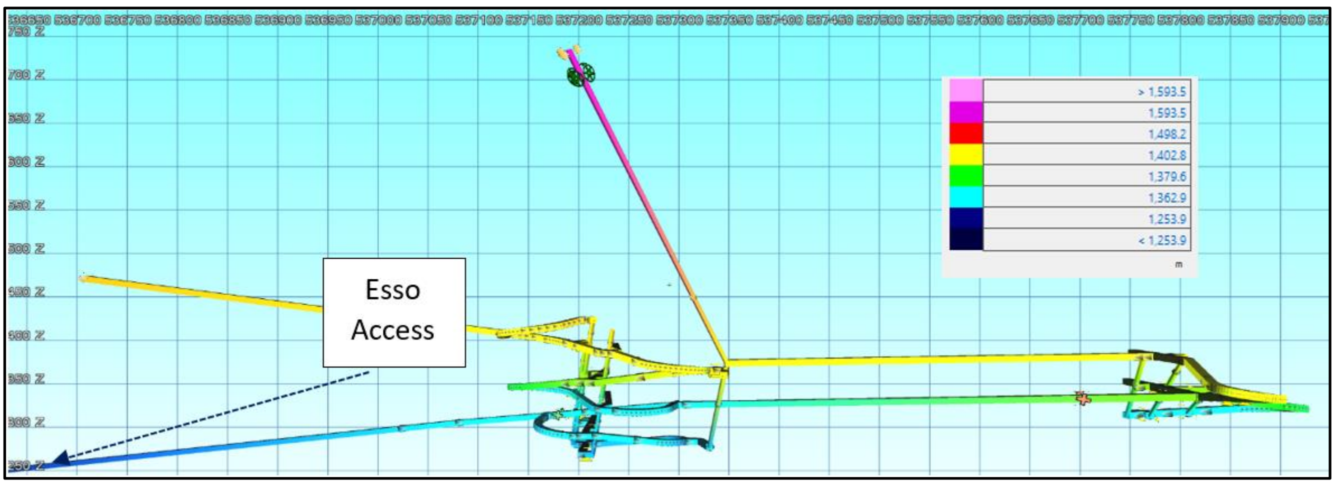


Figure 16-62: Main Deposit Critical Ventilation Path (looking north)

Note: Grid is 50 m square.

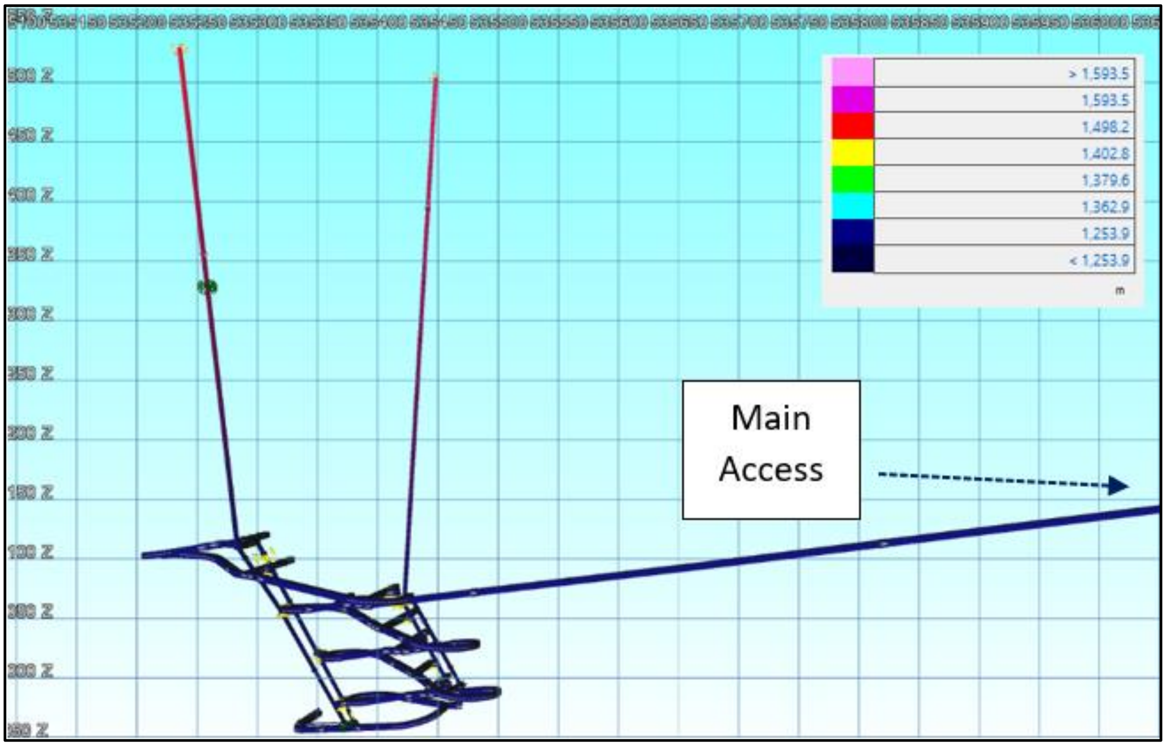


Figure 16-63: Esso Deposit Critical Ventilation Path (looking north)

Note: Grid is 50 m square.

16.5.5.1 Design Criteria

Main design criterion selected for the ventilation design can be listed as follows:

- For each one hp of diesel machinery 100 cfm of fresh air will be delivered
- For active workings at least 40,000 cfm of fresh air to be provided. (1 LHD + 1 Trucking unit)
- Maximum WB temperature is 28°C
- Minimum intake air DB temperature is 3°C
- Maximum air velocity is 3 m/s.

16.5.5.2 Main Deposit

For the Main deposit, 75 m^3/s (159 kcfm) of airflow is required to be accessed from the main haulage way to maintain the desired ore production rate. The negative air pressure at the Main deposit exhaust would create suction at the intake and would provide the necessary airflow for the Main deposit workings in addition to exhausting the return air of the mine. The following figure highlights air flow portals.

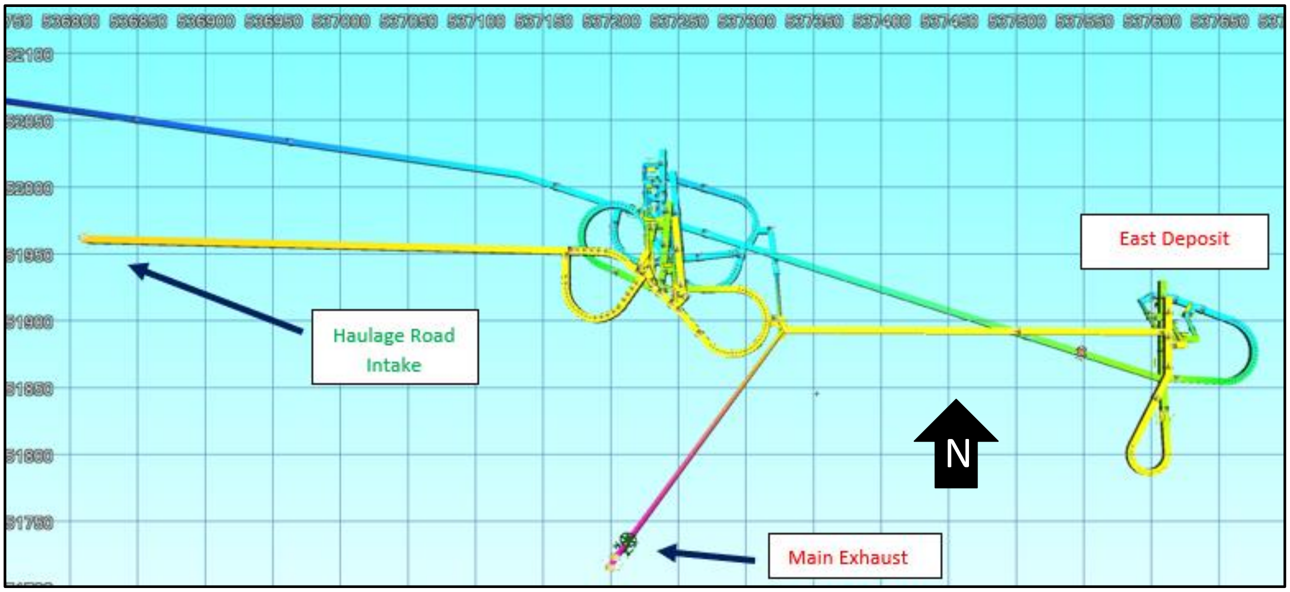


Figure 16-64: Main Deposit Intake and Exhaust Airways (plan view)

Note: Grid is 50 m square.

The 75 m^3/s (159 kcfm) of fresh air provided at the main haulage portal will be utilized during main mine development and production, and as the Esso mine will be accessed from the Main deposit area the crew developing the Esso Mine will also be ventilated with this air. Considering ~19 m^3/s (40 kcfm) flow per crew, this amount of air flow will be able to concurrently support a maximum of three crews in the mine, or two crews in the Main deposit with the third developing the connection drift. The Main deposit fan settings would be set to 75% air flow velocity for this flow in early development. The ventilation capacity was considered in the mine production plan.

Operating the Main Exhaust fan at more than 80% air flow velocity might cause increase airway velocities (+3.5 m/s) at the Main Haulage drift.

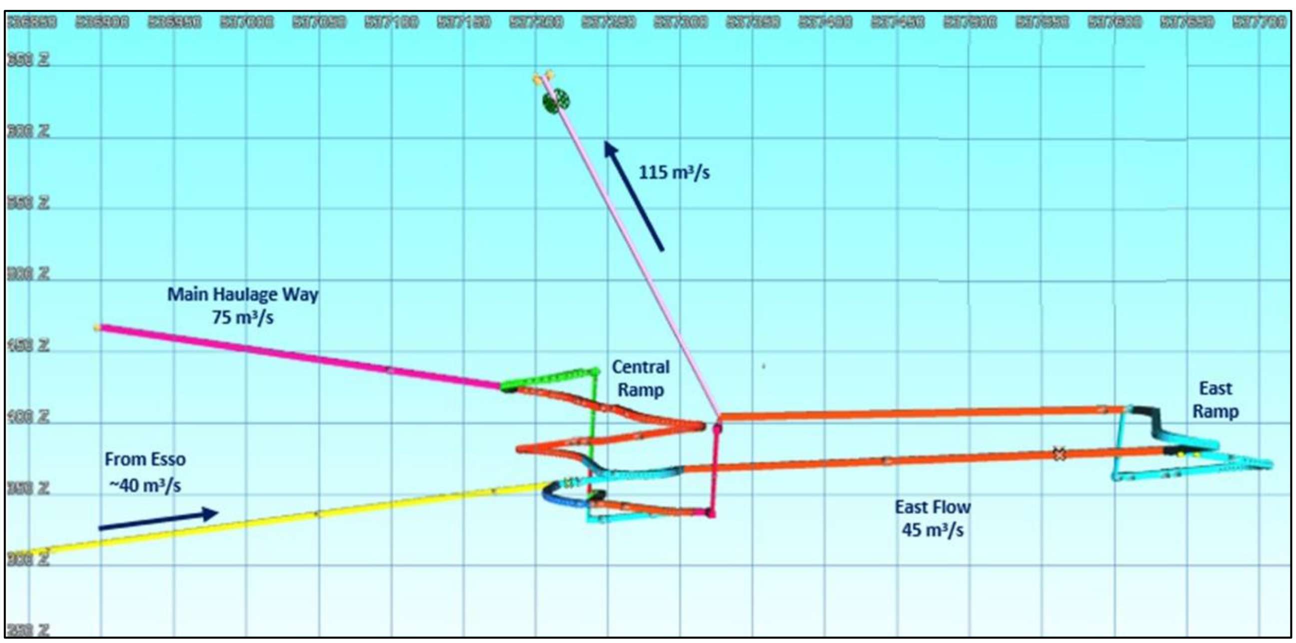


Figure 16-65: Fresh Air Distribution from Haulage Way and Esso Deposit (fully developed – looking north)

Note: Grid is 50 m square.

Any traffic at the connecting drift between two deposits will be ventilated from the Main deposit up until the first shaft of the Esso deposit is developed, and once the Esso deposit ventilation is established the airflow direction will be reversed to maintain the air flow from Esso to Main which will be exhausted through the Main deposit exhaust fan.

Figure 16-66 illustrates the worst-case scenario of the Main deposit ventilation and connection drift. As seen, 75 m³/s (159 kcfm) flow is distributed as 55 m³/s (117 kcfm) for mining activities in the Main deposit whereas 20 m³/s will be provided for developing the connection drift. Note that 80% of VFD setting for the main exhaust fan could be considered to increase total fresh air flow up to 87 m³/s (185 kcfm) which would yield a 3.5 m/s flow limit at the main haulage drift.

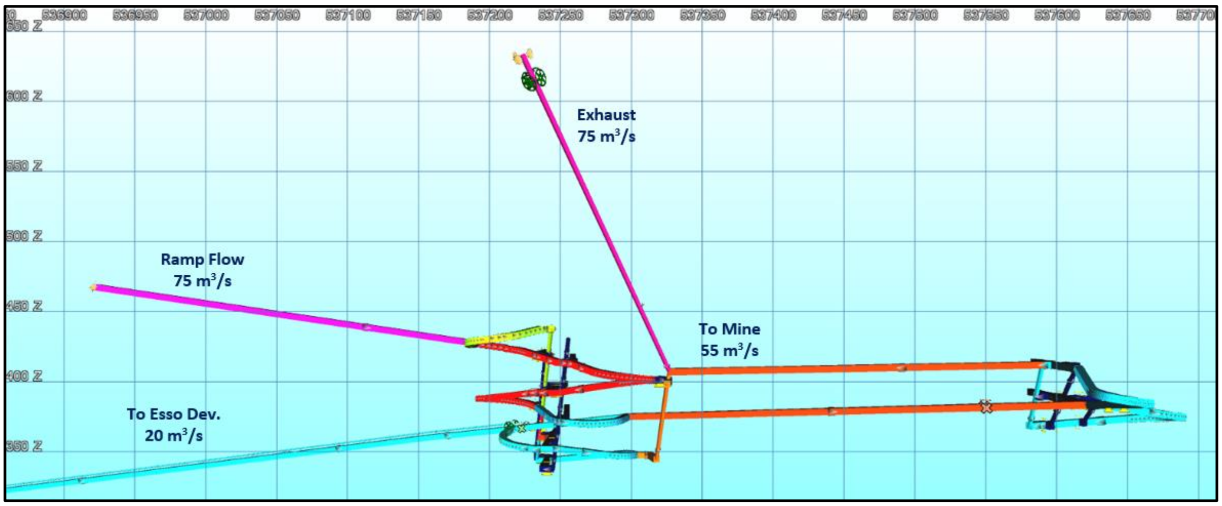


Figure 16-66: Air Delivery During Esso Development (looking north)

Note: Grid is 50 m square.

Figure 16-67 illustrates the preliminary flow assessments across the Main deposit. Central and West ramps are provided with a maximum air flow limit of 70 m³/s (148 kcfm) for a 5 m x 5 m drift size. However, due to its relative size the minimum criterion was kept at 60 m³/s (127 kcfm) for the East ramp which might create a high air velocity in the drift connecting the Central ramp and West ramp. During system operation the flow in this region must be maintained and occasionally bypassed through the East exhaust.

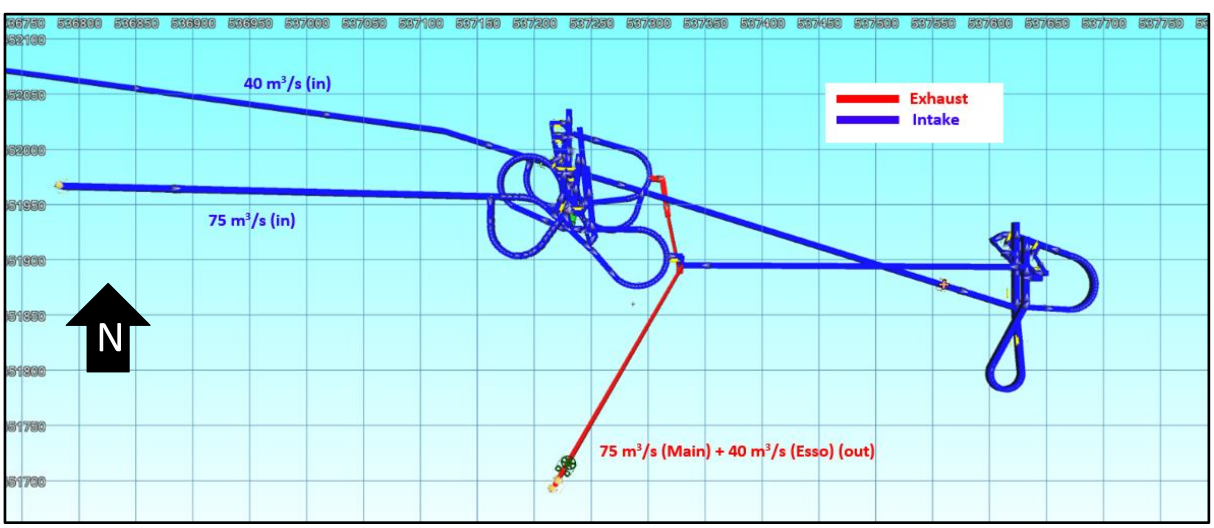


Figure 16-67: Fresh and Exhaust Air Distribution for Main Deposit (plan view)

Note: Grid is 50 m square.

16.5.5.3 Esso Deposit

It is estimated that of a 115 m^3/s (244 kcfm) of fresh air is required which will be provided through a single intake ventilation shaft to achieve planned production in the Esso deposit. This system will be set up in a push configuration to accompany the pull system located in the Main deposit.

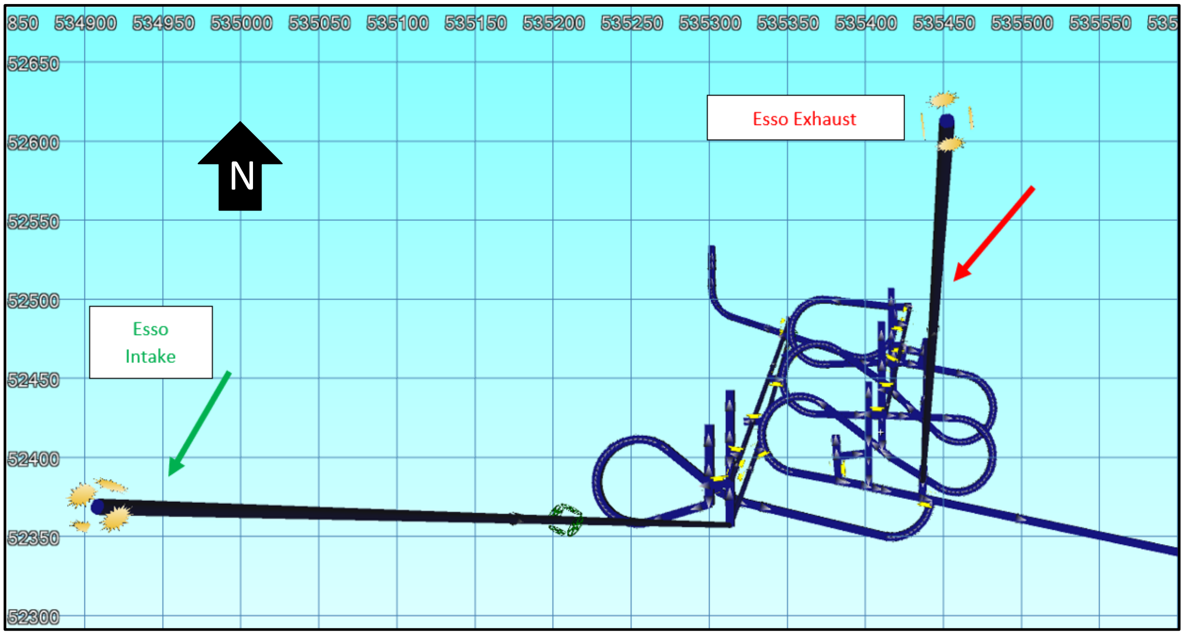


Figure 16-68: Esso Deposit Intake and Exhaust Airways (plan view)

Note: Grid is 50 m square.

The fresh air distribution needs to be done at the 1120 Z level for Esso. A total of 75 m^3/s (159 kcfm) airflow will be provided to the Esso ramp whereas ~40 m^3/s (85 kcfm) flow will be required for ventilating the connection drift between the Esso and the Main deposits. This flow will be enough to provide fresh air for at least two trucks and an LHD (~30 m^3/s) (64 kcfm). Additionally, this flow would also be supporting the Main area in case there is any operational or explorational activity after Esso shaft is fully developed. Figure 16-69 illustrates the air distribution for the Esso deposit.

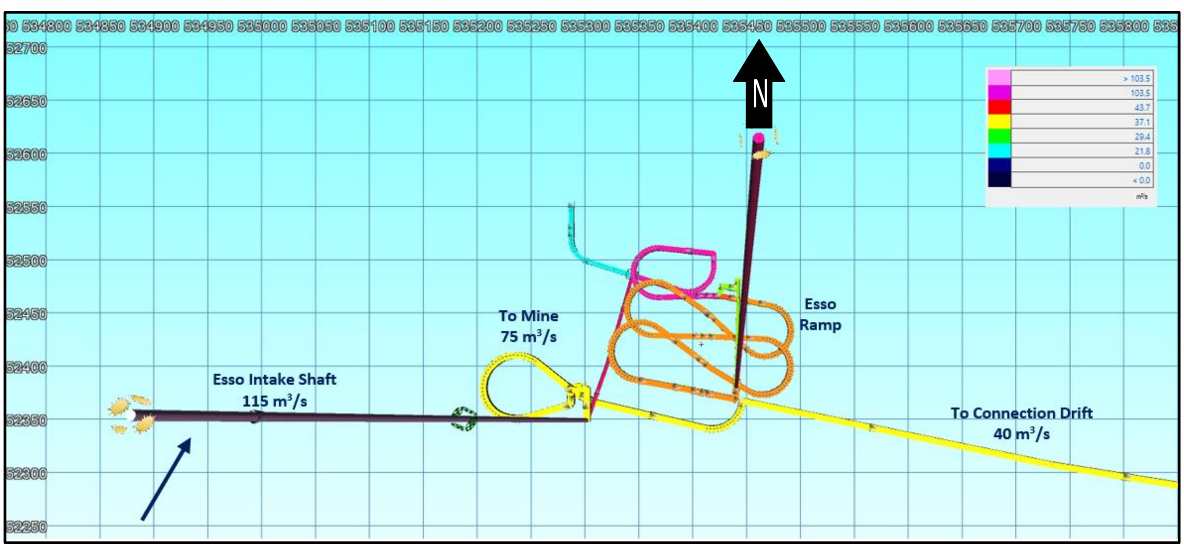


Figure 16-69: Esso Deposit Air Flow Distribution m^3/s (plan view)

Note: Grid is 50 m square.

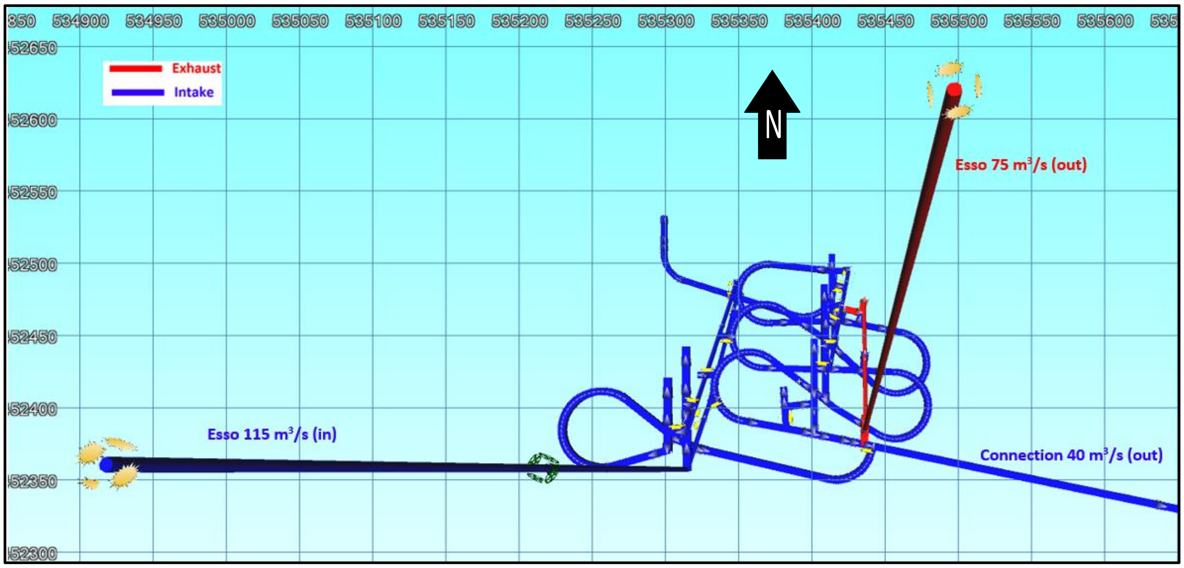


Figure 16-70: Fresh and Exhaust Air Flow Distribution for Esso Deposit m^3/s (plan view)

Note: Grid is 50 m square.

Air distribution over the Esso area is relatively straightforward as the deposit have a dedicated shaft system equipped with fresh and exhaust streams as seen in Figure 16-71. The 40 m^3/s (159 kcfm) of air provided to the connection drift will travel up to the Main Exhaust of the Main deposit for proper exhausting.

16.5.5.4 Emergency Responses

The following emission scenarios are prepared for establishing an understanding of the flow behavior in the mine during an emergency case. In this regard, several hotspots are selected across the mine and an emission of 1000 ppm CO is simulated to highlight the escape scenarios. Affected areas are color coded and legends are shown.

- The inter-level ventilation raise at the Main area (not the exhaust) should have an escape ladderway for allowing egress of the crew to the Main Haulage Way since it accommodates fresh air and allows an easy exit from the mine. This ladderway could be installed in the central raise going across to the ramp distributing the fresh air between the main haulage way and the lower levels.
- A smaller escape ladderway could be installed to provide access to Main Haulage Way during an emergency at the East Ramp or nearby.
- The mine would need two refuge chambers to maintain the safety of the workers, whereas three refuge chambers may diminish the mobile effort of moving refuge chambers to new locations and improving the crew safety. Otherwise, as the Esso deposit mine is developed the chambers are expected to move to westwards along with the connection drift. One refuge station should be located at the Main deposit and one ‘mobile’ chamber should be located at the connection drift with limited access to East part of the ramp.

16.5.6 Underground Mine Equipment

The selection of underground mining equipment is based on the mining method, drift size and stope dimensions, production rates, operating costs, equipment availability, and capital costs. It is assumed that only new equipment would be purchased under lease financing agreements with equipment vendors for major equipment purchases. The list of equipment considered for the project is presented in the table below. It should be noted the equipment is shared between Esso and Main and will be fully utilized at Esso, once Main production is completed.

Table 16-46: Equipment list for underground operations

Equipment	Quantity	Engine power (hp)	Availability	Utilization
Truck (45 t /20 m³)	3	450	0.85	0.85
LHD (6.7 t/3.0 m³)	2	220	0.85	0.85
LHD (6.7 t/3.0 m³) Rammer Jammer	1	285	0.85	0.85
LHD (10 t/4.0 m³)	5	325	0.85	0.85
Jumbo – two-boom	2	192	0.85	0.2
Longhole drill – top hammer	2	100	0.85	0.2
Infill drill	1	100	0.85	0.2
Small explosives truck	1	180	0.85	0.4
Large explosives truck	1	225	0.85	0.4
Bolter	2	100	0.85	0.3
Shotcrete sprayer – mobile	1	135	0.85	0.8
Shotcrete sprayer – manual	1	135	0.85	0.8
Grout pump	1	110	0.85	0.3
Scissor lift	2	165	0.85	0.3
Boom truck	1	145	0.9	0.5
Telehandler	1	120	0.9	0.5
Utility vehicle	1	150	0.9	0.7
Transmixer	2	175	0.9	0.7
Mobile rock breaker	1	145	0.9	0.5
Mechanics truck	1	110	0.9	0.5
Fuel/Lube truck	1	110	0.9	0.5
Electrician truck	1	110	0.9	0.5
Personnel carrier	3	165	0.9	0.5
Supervisor truck	1	110	0.9	0.5
Crew van/ambulance	1	185	0.95	0.3

16.5.7 Underground Mining Personnel

The basis of shift assumptions for underground operations are detailed in Table 16-47.

Table 16-47: Shift assumptions

Shift assumptions	Unit	Value
Roster: 14 days on 14 days off	ratio	0.50
Number of crews	#	4.00
Shifts per day	#	2.0
Shift length	hrs	12.0
Toolbox talk	hrs	0.25
Travel to face + work area preparation	hrs	0.50
Breaks	hrs	0.25
Clear smoke	hrs	0.00
Working time per available hour	min/hr	55
Effective hours per shift	hrs/shift	10.1
Daily effective hours (hourly)	hrs/day	20.17
Worker efficiency (hourly)	%	84.0%
Effective hours per man per year (hourly)	hrs/year	2,160
360 days x 24 hours per day	hrs/year	8,640
Effective hours per year (hourly)	hrs/year	7,260
Effective hours per man per year (hourly)	hrs/year	1,815

A maximum of 88 underground personnel will be required over the course of the mine life. The personnel requirements are listed in Table 16-48 through to Table 16-54.

Table 16-48: Underground mine positions

Underground mine general	Schedule/Shift	No. of positions
Mine Production Superintendent	Day	1
Administrator/Clerk	Day	2
HSE Manager	Day	1
Mine Rescue/Safety Officer/Trainer	Day	1
Mining Superintendent	Day	1
Mine Supervisor/Shift Boss	Shift	8
Maintenance Superintendent	Day	2
Maintenance Supervisor/Shift Boss	Shift	8
Construction Superintendent	Day	2
Mechanical Foreman	Day	4
Electrical Foreman	Day	4
Security Guard	Shift	2
Total		**36**

Table 16-49: Underground mine drill and blast positions

Underground drill and blast	Schedule	No. of positions
Jumbo Operator	Shift	3
Production Drill Operator	Shift	2
Blaster	Shift	3
Drill and Blast Nipper	Shift	4
Total		**12**

Table 16-50: Underground mine load and haul positions

Underground load and haul	Schedule	No. of positions
LHD Operator	Shift	6
Underground Truck Driver	Shift	4
Surface Loader	Shift	2
Surface Truck	Shift	1
Total		**13**

Table 16-51: Underground mine support positions

Underground support services	Schedule	No. of positions
Electrician	Shift	2
Development Service	Shift	1
Ground Support/Bolter/Shotcrete	Shift	5
Utility Vehicle Operator	Shift	3
Services Nipper	Shift	3
Total		**14**

Table 16-52: Underground mine backfill positions

Underground backfill operations	Schedule	No. of positions
Crushing and Screening Plant Operator	Day	1
LHD Operator – Rammer Jammer	Day	1
Total		**2**

Table 16-53 Underground mine maintenance positions

Underground mine maintenance	Schedule	No. of positions
Maintenance Planner	Shift	1
Lube/PM Mechanic	Shift	1
HD Mechanic	Shift	1
Drill Mechanic	Shift	1
Overhaul Mechanic	Shift	1
Welder	Shift	1
Tyreman	Shift	1
Apprentice	Shift	2
Dry/Lapman/Bitman	Shift	2
Total		**11**

Table 16-54: Underground mining technical services positions

Underground mining technical services	Schedule	No. of positions
Senior Mine Engineer	Day	1
Mine Planning Engineer	Day	2
Mine Ventilation/Project Engineer	Day	1
Geotechnical Engineer	Day	1
Mine Technician	Day	2
Surveyor	Day	1
Surveyor Helper	Day	1
Production Geologist and Mine Geologist	Day	1
Geotechnical Technician/Sampler	Day	1
Total		**11**

16.5.8 Underground Mining Services

16.5.8.1 Dewatering

The mine dewatering system is designed to accommodate groundwater inflows from the workings of the Main and Esso deposits, as well as inflows from operating equipment. Sources of water in the underground areas include surface runoff (down the ramp), groundwater seepage, drilling, and diesel fuel combustion.

Total average inflows over the life of mine are estimated to be 0.005 m^3/s at the Main deposit and 0.013 m^3/s at the Esso deposit based on hydrogeological modelling discussed in Section 20. To accommodate for uncertainties in the water inflow model, the design capacity for pumping systems and associated excavations are based on peak inflows of 0.029 m^3/s at the Main deposit and 0.039 m^3/s at the Esso deposit. Updated estimates of third year maximum flows are 0.019 m^3/s at the Main deposit and 0.039 m^3/s at the Esso deposit.

The dewatering plan for the Main and Esso development systems both utilize a network of sumps leading to three settling ponds – one at Main and two at Esso. The sumps utilize submersible pumps located throughout working levels, and contact water collected at each sump is pumped directly to the nearest settling pond. The pumps handle peak groundwater and equipment inflows.

The Main deposit contains one settling pond at the 1360 masl elevation and is used as a means of collecting contact water from the Main workings. Clear water will be pumped up the mine portal to the surface contact water collection system and/or underground water storage to supply equipment based on production requirements.

Contact water will be collected at each of the level sumps at Esso which will be pumped up to a settling pond located at the 970 or 1045 masl elevation. Pumps at the 1045 masl elevation will pump up the decline to the Main clearwater sump at the 1360 masl elevation where the Main deposit pump station will lift the water out of the underground mine and into the contact water collection system on the surface. All water from the underground mine is expected to be treated before release.

Dewatering layouts for Main and Esso sumps and settling ponds are illustrated inFigure 16-71 and Figure 16-72.

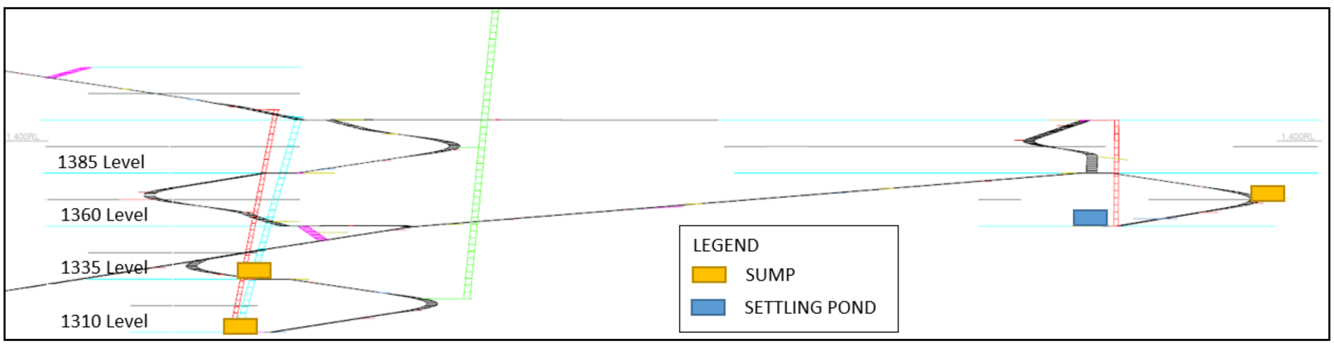


Figure 16-71: Main Deposit Looking North: Dewatering Sumps and Settling Ponds

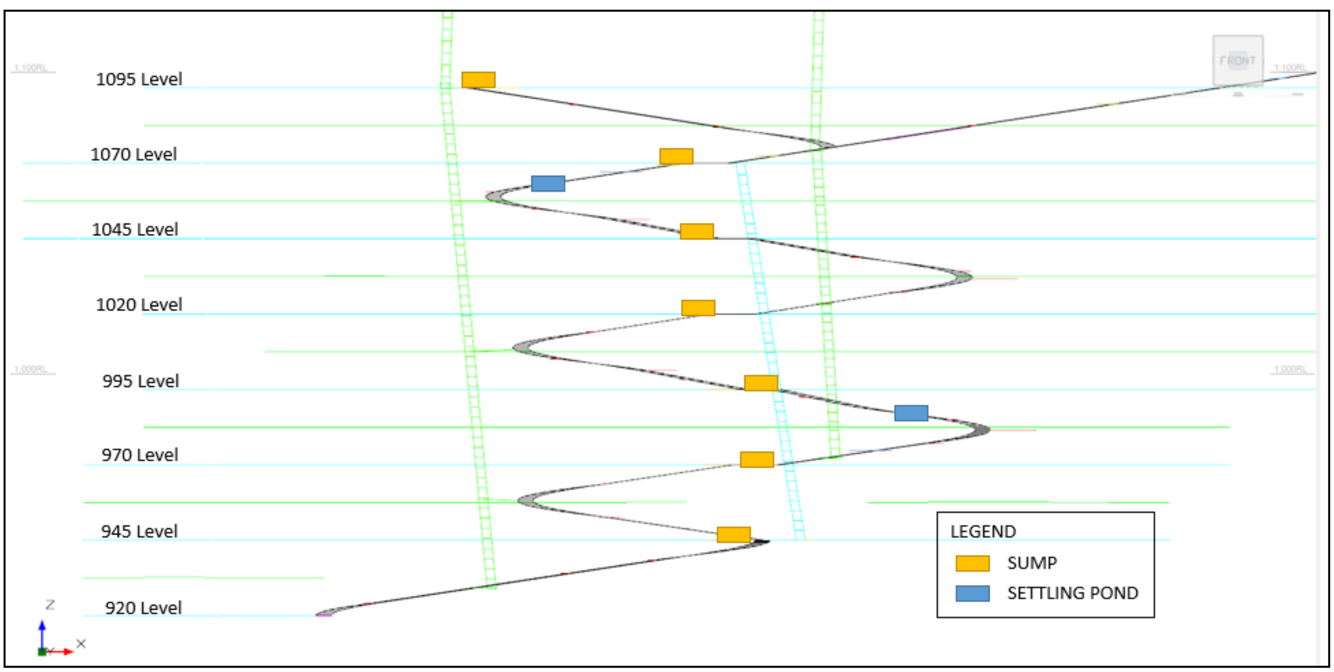


Figure 16-72: Esso Deposit Looking North: Dewatering Sumps and Settling Ponds

16.5.8.2 Compressed Air

Compressed air will be provided by electric compressors strategically placed above ground. No stationary compressors will be installed in the underground workings. The compressed air distribution piping will be installed in ramps and main drifts as needed.

16.5.8.3 Air Heating

The mine air will be heated using LNG burners. The ventilation system considered these units in the design of the air flow. The heating system targeted a working temperature of 5˚C and considered historical monthly temperatures at site.

16.5.8.4 Explosives Storage

Explosives will be stored both above ground and underground. Guidance published by the National Standard of Canada / Bureau de Normalization du Quebec (BNQ) was used for these designs. Regulations defining storage methods, materials, areas, and proximities were gleaned from published documents including:

- Explosives - Magazines for Industrial Explosives: CAN/BNQ 2910-500/2015
- Explosives - Quantity Distances: CAN/BNQ 2910-510/2015 (National Standard of Canada, 2015)

Designs for both surface and underground magazines are provided in Section 18.
A total of 2,115 t of explosives are expected to be consumed over the life-of-mine, and annual consumption of explosives is expected to range between 363 t in Year 2 to 93 t in Year 9. Section 16.2 of this report provides a table of the annual consumption.

16.5.8.5 Power Distribution

Power will be fed to the underground workings through the portal and ventilation raises. Junction boxes will be used to distribute power to underground substations. The C22.1-18 Canadian Electrical Code and Part 5 of the Health, Safety, and Reclamation Code for Mines in British Columbia will be utilized for the design of the power distribution system for the underground mine.

16.5.8.6 Communication

Underground communications will be facilitated through a "leaky feeder" UHF/VHF and/or fibre optic cable as required. Capabilities for video feeds, operational data, and safety monitoring will be integrated.

16.5.8.7 Mine Safety

Fire Prevention

Fire prevention measures in the underground mining areas will consist of proper ventilation, safe explosives storage and handling, fire suppression equipment (fire extinguishers), and fire doors. The appropriate type of fire suppression equipment will be strategically located. Some of the areas that require fire suppression equipment include fuel bays, electrical stations, explosives storage, and Refuge Chambers. All mobile equipment including, but not limited to, haul trucks, jumbos, and LHDs will be equipped with fire extinguishers and an automatic fire suppression system.

Mine Rescue

A Mine Rescue team will be trained to handle a variety of emergency situations. A Mine Rescue team is an integral part of any mining operation. Guidance from the Western Canada Mine Rescue Manual published by the Ministry of Energy and Mines will inform the Mine Rescue team hierarchy and training procedures. A variety of emergency situations can be encountered, including equipment accidents, fall of ground, and chemical leaks or fires.

Refuge Chambers

Stationary or portable 20-person refuge chambers will be strategically located near the main working areas. A total of three refuge chambers (two stationary and one mobile) will be required over the life of the mine. The refuge chambers will be provided with clean water, clean air, medical supplies, and redundant communication methods. A typical refuge chamber is illustrated in Figure 16-73.

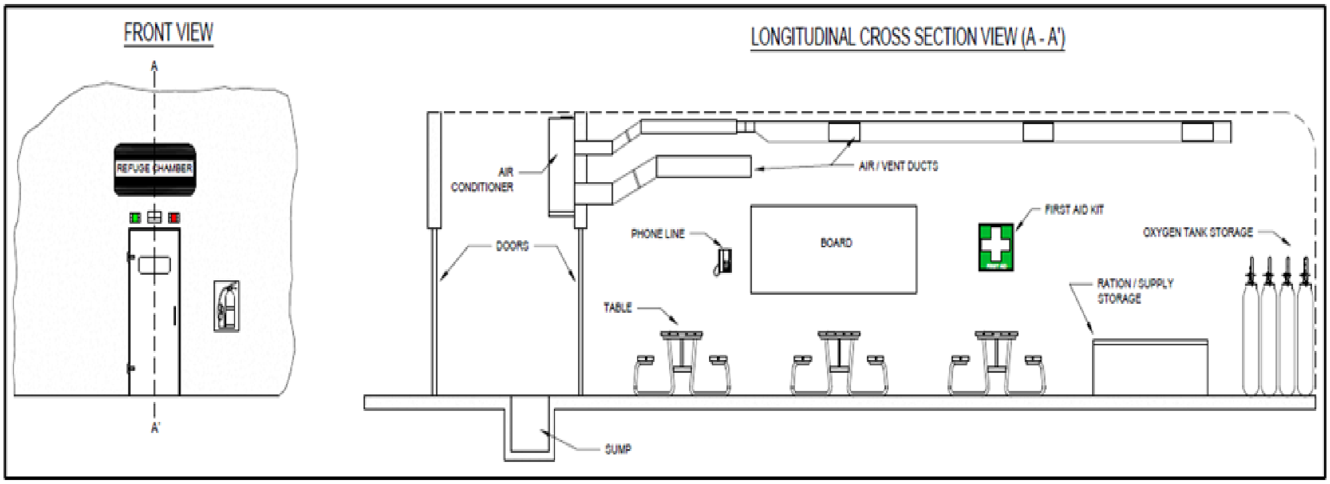


Figure 16-73: Typical Stationary Refuge Chamber (dimensions not to scale)

Emergency Egress

An underground mine must always have at least two paths for personnel to exit the mine in case of emergencies. Access via the Main portal is considered the primary access and egress for the mine. Secondary egress would be via a dedicated manway in the Esso and Main ventilation intakes, and manway raises connect all levels so that a worker is never more than 500 m from a refuge station.

Stench System

Each fresh air intake will be equipped with a stench gas system. A stench gas system is used to alert personnel who may not be able to hear audible alarms or see visual alarms. The system would be used to inform underground workers of an emergency.

Sulphide Dust

Underground mines with high sulphide ores may be susceptible to sulphide dust explosions. As per Section 6.31.1 of the Health, Safety, and Reclamation Code for Mines in British Columbia, the mine manager is required to implement a plan for minimizing the danger from sulphide dust explosions once an explosion has occurred Vigilance is required given the high sulphide content of the ore, and a work safe practice will be established to minimize exposure of workers.

17 Recovery Methods

17.1 Introduction

The results of the metallurgical test work described in Section 13 were used to select the recovery methods for the Project. The resulting design criteria was used to design the process facility described in this section of the report.

The crushing plant will process ore at a rate of 4,500 tpd with LOM average head grades of 1.58% Cu, 2.31% Zn, 28 g/t Ag and 0.39 g/t Au. The ore is amenable to particle ore sorting. A pair of Tomra XRT particle ore sorters will reject an estimated 560 tpd of waste. The rejected waste is estimated to have an average grade of 0.12% Cu, 0.10% Zn, 3 g/t Ag and 0.08 g/t Au. The process plant will process 3,940 tpd after the ore sorting circuit with an average LOM head grade of 1.79% Cu, 2.62% Zn, 31 g/t Ag and 0.43 g/t Au. The two-stage grinding circuit will target a product size of 80% passing (P_{80}) 55 µm, followed by a sequential flotation process to produce copper and zinc concentrates. Based on test work, the overall LOM recoveries of produced metal for both concentrates combined from the flotation circuit are estimated to be approximately 88.6% for copper, 95.4% for zinc, 86.6% for silver and 60.5% for gold. The tailings will be pumped to a tailings management facility (TMF) with water recovered for re-use in the process plant. The crushing and ore sorter circuit will operate at an availability of 70%, while the grinding and flotation circuits will operate 24 hours per day, 365 days per year, at an availability of 92%. The concentrate filtration circuit will operate at an availability of 65%.

The plant will consist of the following unit operations:

- Primary Crushing – A vibrating grizzly feeder and jaw crusher in open circuit, producing a final product P_{80} of approximately 95 mm.
- Secondary Crushing and Ore Sorting – A cone crusher, triple deck screen and two XRT particle ore sorters producing an upgraded final product P_{80} of approximately 50 mm and a waste reject. Material smaller than 12.5 mm bypasses the ore sorter.
- Crushed Material Storage and Reclaim – A 1,800-tonne bin with two reclaim belt feeders feeding the SAG Mill Feed Conveyor.
- Ore Sorter Waste – 730-tonne bin with truck loadout.
- Primary Grinding – A SAG mill in open circuit, producing a transfer size T80 of approximately 600 µm.
- Secondary Grinding – A ball mill in closed circuit with a cyclone cluster, producing a final product P_{80} of 55 µm.
- Copper Rougher and Cleaner Flotation – Rougher flotation cells, rougher concentrate regrind (P_{80} of 15 µm), and cleaner flotation cells.
- Zinc Rougher and Cleaner Flotation – Rougher flotation cells, rougher concentrate regrind (P_{80} of 20 µm), and three stages of cleaner flotation cells.
- Concentrate Dewatering, Filtration and Loadout – Thickeners, stock tanks, and pressure filters for each of the zinc and copper concentrates.
- Final Tailings Thickener and Disposal to the TMF – Thickener and centrifugal pumps to discharge slurry to the TMF, a floating intake system and a shore-based pump reclaims TMF water for treatment before re-use in the process plant or as clean water release.

17.2 Process Design Criteria

Conceptual design criteria for major equipment of the 4,500 tpd crushing and ore sorting circuits, and the 3,940 tpd grinding and flotation circuits are listed in Table 17-1.

Table 17-1: Process design criteria

Description	Unit	Value	Source
Mill Feed Characteristics			
Specific gravity of in situ mineralized rock	g/cm^3	3.59	Test work
Bulk density	t/m^3	2.2	Historical reports
Moisture content (average)	%	5	Mine plan
Operating Schedule			
Shift/Day	#	2	Design
Hours/Shift	h	12	Design
Hours/Day	h	24	Design
Days/Year	d	365	Design
Plant Availability/Utilization			
Crusher availability	%	70	Design
Grinding and flotation plant availability	%	92	Design
Crushing feed rate	t/h	268	Engineering calculation
Grinding and flotation process plant feed rate	t/h	178	Engineering calculation
Average Head Grades	% Cu	1.79	Mine plan
	% Zn	2.62	Mine plan
Recovery (to crusher head): to copper concentrate	% Cu	87.2	Mine plan
Recovery (to crusher head): to zinc concentrate	% Zn	71.5	Mine plan
Average copper concentrate grade	% Cu	25.5	Mine plan
	% Zn	10.4	Mine plan
Average zinc concentrate grade	% Zn	55.1	Mine plan
	% Cu	0.62	Mine plan
Copper concentrate mass recovery	%	6.07	Test work
Zinc concentrate mass recovery	%	3.46	Test work
Crushing			
Static grizzly	mm x mm	800 x 800	Design
Crusher feeder	-	Vibrating	Design
Primary crusher	-	Jaw	Design
Primary screen	-	Double deck vibrating	Design
Secondary crusher	-	Cone	Design
Secondary screen	-	Double deck vibrating	Design
Ore Sorter			
-75 to 25 mm feed rate	tph	184	Estimation
-25 to 12.5 mm feed rate	tph	38	Estimation
Waste rejection of ore sorter feed	% of feed	15	Test work
Ore Sorter Product			
Ore sorter product bin size (live capacity)	t	1,800	Design
Reclaim feeders	#	2 - Belt	Design
Reclaim feed rate (each), operating	t/h	178	Engineering calculation
Estimated crushing circuit product size, P_{80}	mm	50	Bruno simulation
Ore Sorter Waste			
Ore sorter waste bin size (live capacity)	t	730	Design
Ore sorter waste bin feed rate	t/h	33.4	Calculation

Description	Unit	Value	Source
Grinding			
Crushing Work Index, CWi	kWh/t	15.5	Test work
Bond Rod Mill, RWi	kWh/t	9.5	Test work
Bond Ball Mill BWi, (Average at P_{80} 55 µm)	kWh/t	14.1	Test work
Abrasion Index, Ai	g	0.20	Test work
Primary Grinding			
Mill type	-	SAG Mill	Design
Mills	#	1	Design
Feed solids	%w/w	72	Design
Circulating load	%	100	Design
Feed size, P_{80}	mm	50	Bruno simulation
Product size, P_{80}	µm	600	Vendor simulation
Secondary Grinding			
Mill type		Ball Mill	Design
Mills	#	1	Design
Feed solids	%w/w	70	Design
Circulating load	%	300	Design
Feed size, P_{80}	µm	600	Vendor simulation
Product size, P_{80}	µm	55	Design
Recirculation load	%	300	Design
Classification type	-	Cyclones	Design
Flotation Circuit			
Rougher scale-up factor	-	3.0	Design
Cleaner scale-up factor	-	2.5	Design
Copper Rougher Circuit			
Flotation time	min	30	Test work
Slurry pH	-	6.0	Test work
Mass recovery	%	7.48	Test work
Copper Regrind Circuit			
Mill type	-	Vertical Mill	Design
Product size, P_{80}	µm	15	Test work
Copper Cleaner			
Flotation time	min	20	Test work
Slurry pH	-	5.5	Test work
Copper concentrate tonnage	t/h	10.8	Test work
Mass recovery	%	6.07	Test work
Zinc Rougher Circuit			
Flotation time	min	18	Test work
Slurry pH	-	7.0	Test work
Mass recovery	%	7.08	Test work
Zinc Regrind Circuit			
Mill type	-	Vertical Mill	Design
Product size, P_{80}	µm	20	Test work

Description	Unit	Value	Source
Zinc First Cleaner			
Flotation time	min	12.5	Test work
Slurry pH	-	11.0	Test work
Mass recovery	%	3.66	Test work
Zinc Second Cleaner			
Flotation time	min	10.0	Test work
Slurry pH	-	11.0	Test work
Mass recovery	%	3.94	Test work
Zinc Third Cleaner			
Flotation time	min	10	Test work
Slurry pH	-	11.0	Test work
Zinc concentrate tonnage	t/h	6.2	Test work
Mass recovery	%	3.46	Test work
Thickening			
Thickener type	-	High Rate	Design
Thickener Underflow density	% solids	60	Test work
Thickener unit area rate	m^2/(tpd)	0.0246	Design
Thickener underflow filter storage tank capacity	h	6	Design
Thickener diameter: copper concentrate	m	11	Design
Thickener diameter: zinc concentrate	m	11	Design
Filtration			
Filter type	-	Pressure	Design
Filter model	Metso	VPA1530	Vendor data
Filter plate dimensions	m	1.5 x 1.5	Vendor data
Filter volume	m^3	2.0	Vendor data
Chamber volume	m^3	0.055	Vendor data
Filtration cycle time	min / cycle	18	Vendor data
Filter availability	%	65	Vendor data
Concentrate bulk density	t/m^3	2.10 to 2.92	Test work
Filter cake moisture	%	8.0 to 9.0	Test work
No. of filter chambers; calculated/recommended: copper	-	42 / 60	Design
No. of filter chambers; calculated/recommended: zinc	-	24 / 40	Design
Tailings Thickening			
Thickener type	-	High Rate	Design
Thickener underflow density	% solids	65	Test work
Thickener unit area rate	m^2/(tpd)	0.0485	Design
Thickener diameter	m	25	Design

17.3 Plant Design

A simplified process flowsheet is presented in Figure 17-1.

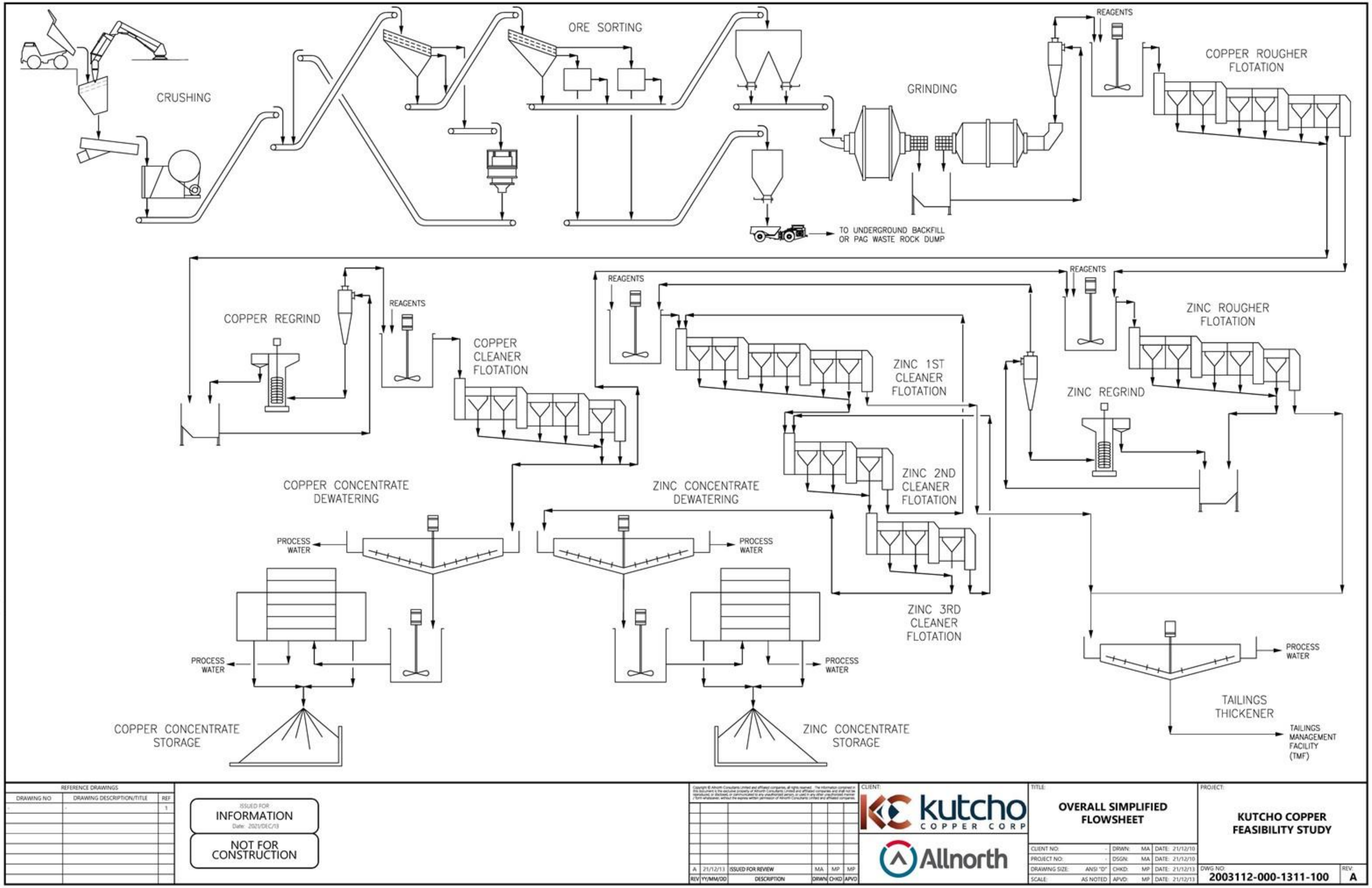


Figure 17-1: Simplified Process Flowsheet

17.4 Process Description

A 4,500 tpd crushing plant for the Project to process the volcanogenic massive sulphide (VMS) mineralization is proposed. The crushing plant will operate at 70% availability. The process plant is assumed to operate 365 days per year at 92% availability at a rate of 3,940 tpd after the ore sorting circuit. The plant feed will be crushed, sorted to remove low grade material, ground and then sequentially subjected to copper and zinc flotation. The metals contributing to revenue in the produced concentrates are copper, zinc, gold, and silver.

The zinc tailings will be pumped to the TMF for disposal. The copper and zinc concentrates will be thickened and dewatered before loading onto semi trailer trucks. Precious metals will be contained in and shipped with the copper and zinc concentrates.

17.5 Crushing Operations

Run-of-mine (ROM) ore will be delivered to the primary crusher by haul trucks from the open pit and underground mine or when direct feed is not available, by a 7 m^3 or 10 m^3 bucket capacity loader from a short-term stockpile located adjacent to the crusher or by truck rehandled from the long-term low-grade stockpile. The ROM material will feed a dump pocket via a stationary grizzly (800 mm x 800 mm) at an average rate of 268 tph. Grizzly oversize will be broken by a rock breaker. Grizzly undersize will be discharged to a dump hopper and fed via a vibrating grizzly feeder to the primary jaw crusher (132 kW) with a closed size setting (CSS) of 85 mm. The primary circuit product size will be approximately eighty percent passing (P_{80}) 95 mm. The jaw crusher discharge and vibrating grizzly undersize combine on the jaw crusher discharge conveyor that feeds the primary screen feed conveyor.

The primary screen feed conveyor feeds the primary double deck screen (1.8 m x 6.1 m). Three size fractions will be produced from the screen: +75 mm, -75 mm +25 mm and -25 mm. The oversize (+75 mm) will be sent to a cone crusher and the middle and undersize fractions combine to feed the secondary double deck screen.

The oversize material feeds the cone crusher feed conveyor and discharges into the cone crusher feed bin. A pan feeder under the bin will feed the cone crusher (132 kW). The cone crusher CSS is set at 40 mm with a product P_{80} of approximately 50 mm. The cone crusher discharges onto a transfer conveyor and then onto the primary screen feed conveyor.

The primary double deck screen middle and undersize fractions feed the secondary double deck screen (1.8 m x 6.1 m). The screen decks divide the material into three sizes: +25 mm, -25 mm +12.5 mm and -12.5 mm. The oversize material (+25 mm) will feed the coarse ore sorter and the middle size material (-25 mm +12.5 mm) will feed the fine ore sorter.

The secondary screen undersize will discharge onto the crushed ore bin feed conveyor. This fine and coarse ore sorter product will also discharge onto this conveyor once the sorting of ore? grade material and waste is complete.

Two streams will be produced from the crushing and ore sorting circuits. The crushed ore, secondary screen undersize and ore sorter product, will feed an 1,800-tonne bin and the ore sorter waste will feed a 730-tonne bin. The crushed ore will be reclaimed from the bin by two belt feeders to feed the SAG mill feed conveyor. The waste discharged into the bin will be removed by trucks.

Crusher operations will require to be cleared for some open pit blasting operations and a blast safety management plan will need to be developed.

Special consideration was given to project start-up operational learning and is discussed in the mine plan in Section 16.

17.5.1 *Ore Sorting*

The secondary double deck screen crusher products will be fed to the sorting circuit which will comprise two XRT ore sorters. The -75 + 25 mm size fraction will be fed to the coarse surge bin and then to the coarse ore sorter using a pan feeder, whereas the -25 + 12.5 mm size fraction will be fed to the fine surge bin and then to the fine ore sorter using a pan feeder. The crusher secondary screen undersize (-12.5 mm) will bypass the ore sorting circuit and is then conveyed directly to the crushed ore bin.

Based on the results obtained from the ore sorting test program, it was established that an XRT based sorter would be suitable for the Project. The TOMRA COM XRT 2.0 with a 2.4 m belt width is designed to process 184 tph of coarse rocks (-75 + 25 mm size fraction). The TOMRA COM Tertiary XRT with 1.2 m belt width is designed to process 38 tph of fine rocks (-25 + 12.5 mm size fraction). The ore sorters are estimated to reject 15% of the sorter feed as waste based on test work and analysis. The upgraded sorter product will be conveyed to the crushed ore bin, and the sorter reject will be conveyed to the ore sorter waste bin.

Ore sorter grade and recovery estimates were based on a bulk ore sorting sample (from drillhole core) and test work results are summarized in Table 17-2. Fines by-passing the ore sorter was based on size fraction modelling of the crushing and screening circuit at 17.2% of the crusher feed. The grade of the fines bypassing the ore sorter is estimated to be 77% of the head grade for copper, 70% for zinc, 100% for silver and 100% for gold.

Table 17-2: Ore sorter recovery parameters

Item	Value
Sorter reject of ore sorter feed	15.04%
Recovery to reject of ore sorter feed	
Copper	1.12%
Zinc	0.62%
Silver	1.36%
Gold	3.22%

17.5.2 *Grinding Circuit Operations*

The crushed ore will be reclaimed from the crushed ore bin by two belt feeders at a controlled rate of 178 tph to feed the semi autogenous grinding (SAG) mill. Test work and metallurgical reports were provided to the Vendors to run their simulations and provide the grinding circuit equipment sizing.

Reclaimed material from the crushed ore bin at a feed size F_{80} of approximately 50 mm will feed a 6.1 m diameter x 3.0 m long, 2,000 kW SAG mill, with a variable frequency drive (VFD) to control the speed, via the SAG mill feed conveyor. A belt-scale on the feed conveyor will monitor feed rate. The SAG mill will operate with 100% circulating load in closed circuit with the cyclone cluster. Process water will be added to the SAG mill to maintain the slurry charge in the mill at a constant density of 72%. Ground slurry from the SAG mill will flow through a trommel to the SAG mill pump box to feed a cyclone cluster.

The SAG mill pump box slurry will be pumped to 4 place-380 mm cyclones (three operating) for classification. The coarse cyclone undersize will flow by gravity back to the SAG mill to combine with the crushed ore new feed. The cyclone overflow with a T_{80} transfer size in the range of 600 µm will flow to the secondary grinding circuit.

Secondary grinding will take place in a 5.0 m diameter x 7.6 m long, 3,250 kW ball mill supplied with a VFD. The ball mill will operate with a 300% circulating load in closed circuit with a cyclone cluster. The ball mill, at a target density of 70%, will discharge through a trommel into the ball mill pump box and pumped to the ball mill grinding cyclones. The cyclone cluster consists of eight place-380 mm cyclones (five operating). The cyclone underflow will report back to the ball mill and combine with the SAG mill grinding cyclone overflow. The cyclone overflow will be directed to the copper flotation circuit at a target P_{80} of 55 µm and 35% solids.

Steel balls will be used as the grinding media. Zinc sulphate will be added to the ball mill pump box.

17.6 Copper and Zinc Flotation Circuits

17.6.1 Discussion Regarding Concentrate Grades over the Life of Mine

The flotation circuit consists of the equipment required for the sequential flotation of the copper minerals, followed by the zinc minerals. Each circuit has a regrind mill complete with a cyclone classification system to regrind the respective rougher concentrates to the required particle sizes.

Table 17-3 shows the anticipated average metal recoveries and grades for the copper and zinc concentrates based on the mine plan plant feed grades.

Table 17-3: Concentrate recovery (to crusher head) and grade values from mine plan

Metal/Mass	Recovery to concentrates (%)		Concentrate grade (% or g/t)	
	Cu concentrate	Zn concentrate	Cu concentrate	Zn concentrate
Copper	87.2	1.2	25.5	0.62
Zinc	24.3	71.5	10.4	55.1
Silver	59.2	9.4	305	87
Gold	36.8	24.3	2.63	3.06
Mass	6.2	3.6	-	-

Recovery and mass pull calculations allow for sub-optimal performance at start-up of operations to allow for operational training and equipment commission in the initial periods of operation as shown in Table 17-4.

Table 17-4: Copper concentrate recovery (to crusher head) and mass pull modification for start-up

Period from start-up →	2027 / Q1	Q2	Q3	Q4	2028 / Q1	Q2+
Copper	65%	85%	95%	97%	99%	100%
Zinc	100%	100%	100%	100%	100%	100%
Silver	65%	85%	95%	97%	99%	100%
Gold	65%	85%	95%	97%	99%	100%
Mass	65%	85%	95%	97%	99%	100%

Table 17-5: Zinc concentrate recovery (to crusher head) and mass pull modification for start-up

Period from start-up →	2027 / Q1	Q2	Q3	Q4	2028 / Q1	Q2+
Copper	100%	100%	100%	100%	100%	100%
Zinc	65%	85%	95%	97%	99%	100%
Silver	65%	85%	95%	97%	99%	100%
Gold	65%	85%	95%	97%	99%	100%
Mass	65%	85%	95%	97%	99%	100%

Partially oxidized floatable ore (termed oxide) comprises about 2.6% of total feed tonnage. The small proportion is not material to the total feed tonnage and will be processed after the payback period. Limited test work was completed on the oxide material. The QP determined that it was appropriate based on their experience that the recovery for oxide material was reduced by 10% compared to sulphide ore and the mass pull was increased by 25%.

Mass pull and recovery were estimated for each quarter of production and vary as a function of head grade (with formulae developed from test work) and duly modified for start-up. Figure 17-2 shows flotation head grade by quarter, which is a key flotation recovery control.

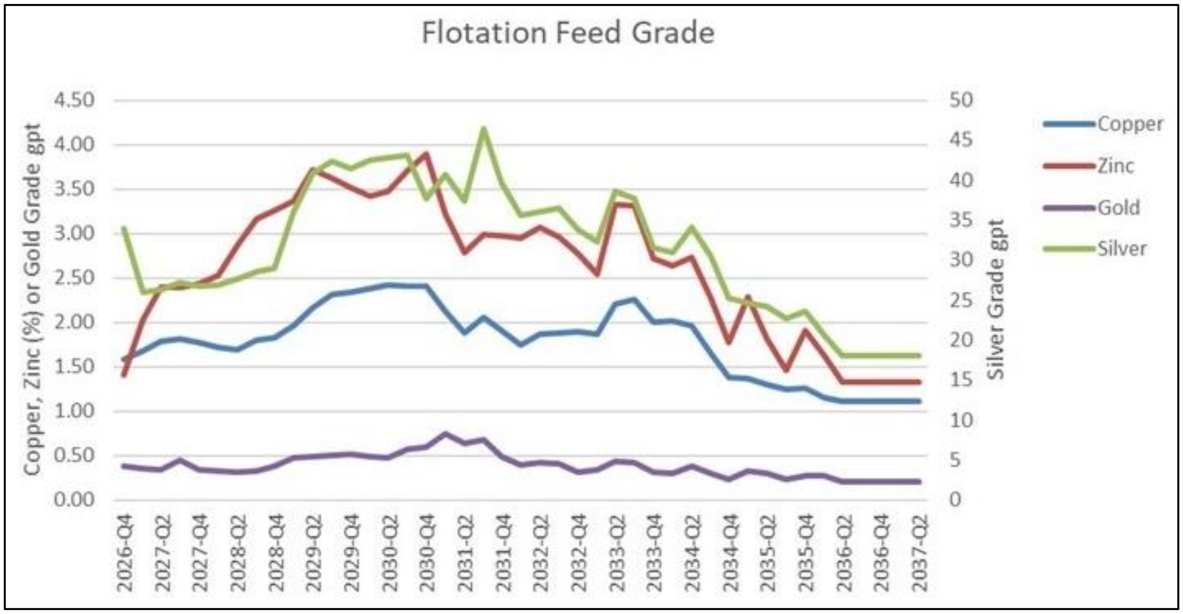


Figure 17-2: Zinc Concentrate Recovery (To Crusher Head) and Mass Pull Modification for Start-Up

Figure 17-3 shows the copper concentrate recoveries for copper, zinc, silver, and gold (as compared to the crusher head grade). Based on test work, zinc recovery to the copper circuit concentrate is predicted to increase with reduced zinc head grade whereas copper, silver and gold recoveries increase with increasing head grade.

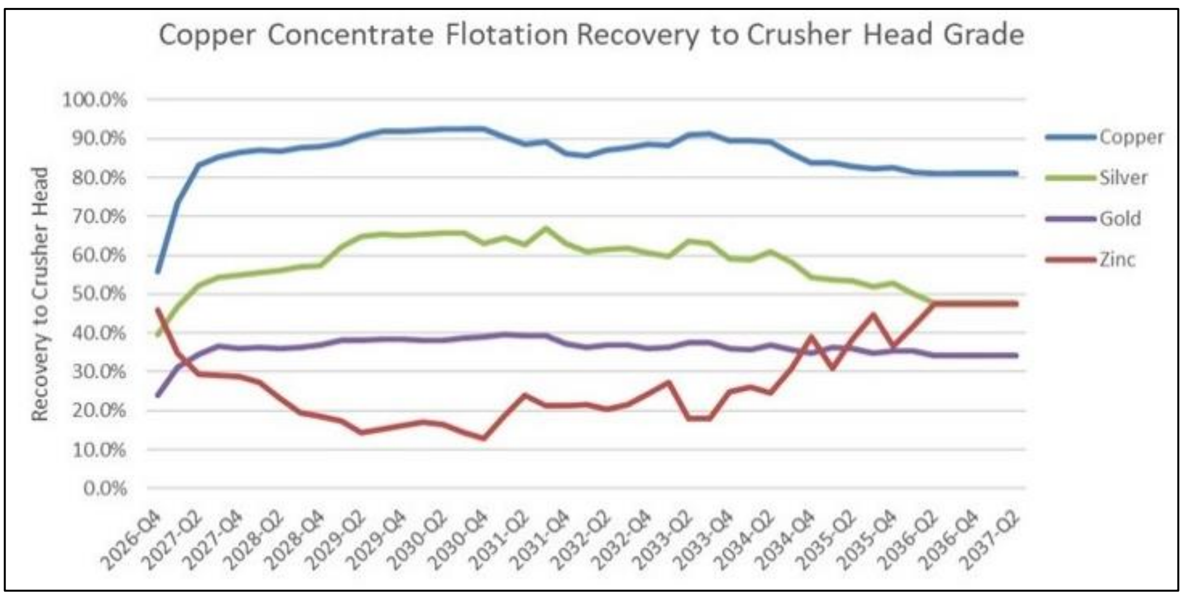


Figure 17-3: Graph of Recoveries of Copper, Zinc, Silver, and Gold to the Copper Concentrate

Figure 17-4 shows the zinc concentrate recoveries for copper, zinc, silver, and gold compared to the crusher head grade. Based on test work, copper recovery to the zinc circuit is predicted to increase to the zinc concentrate with reduced copper head whereas zinc and silver recoveries increase with increasing head grade. Gold recoveries are predicted to be relatively unchanged with head grade.

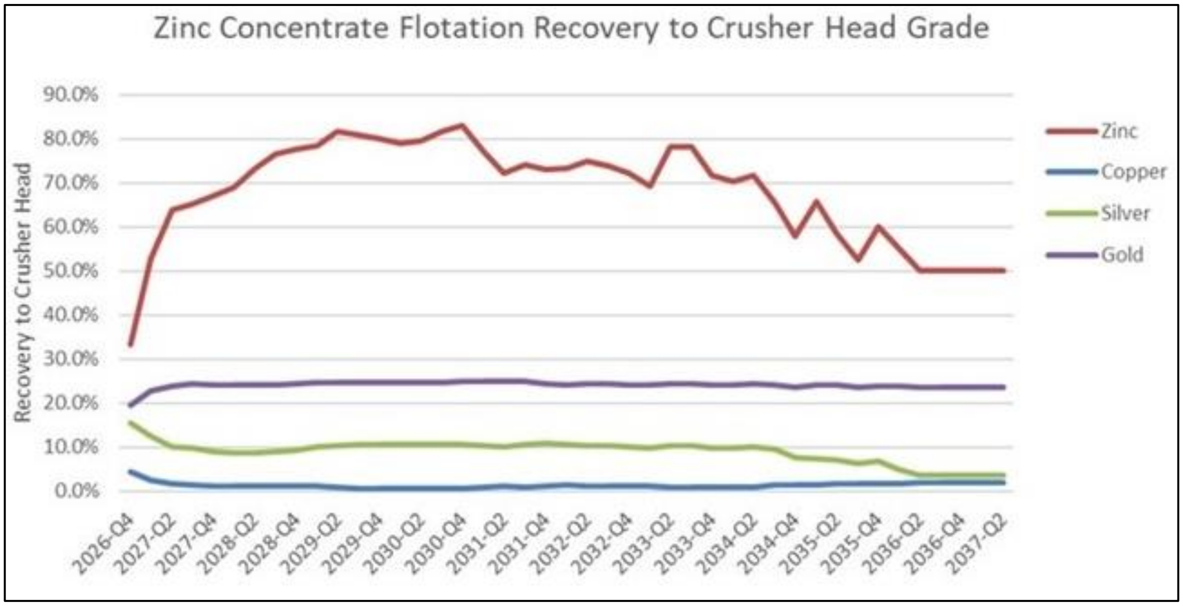


Figure 17-4: Graph of Recoveries of Copper, Zinc, Silver, And Gold to the Zinc Concentrate.

Concentrate grades were predicted for each period based on head grades, and formulae for mass pull and recovery. Final modelled results were reconciled to test work. Penalty grades, other than zinc in copper concentrate, were not estimated but instead were taken from assays of locked cycle final cleaner product from the 2021 metallurgical test work. Details are discussed in Section 19 for concentrate grades and penalties in relation to concentrate marketing.

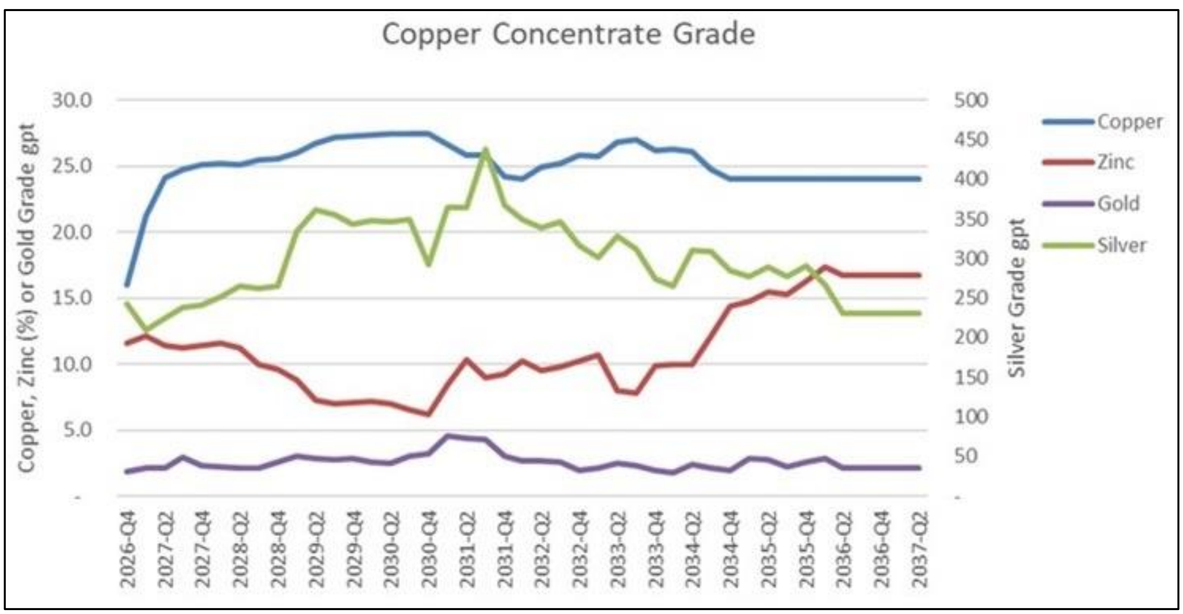


Figure 17-5: Graph of Copper Concentrate Grades for Copper, Zinc, Silver and Gold

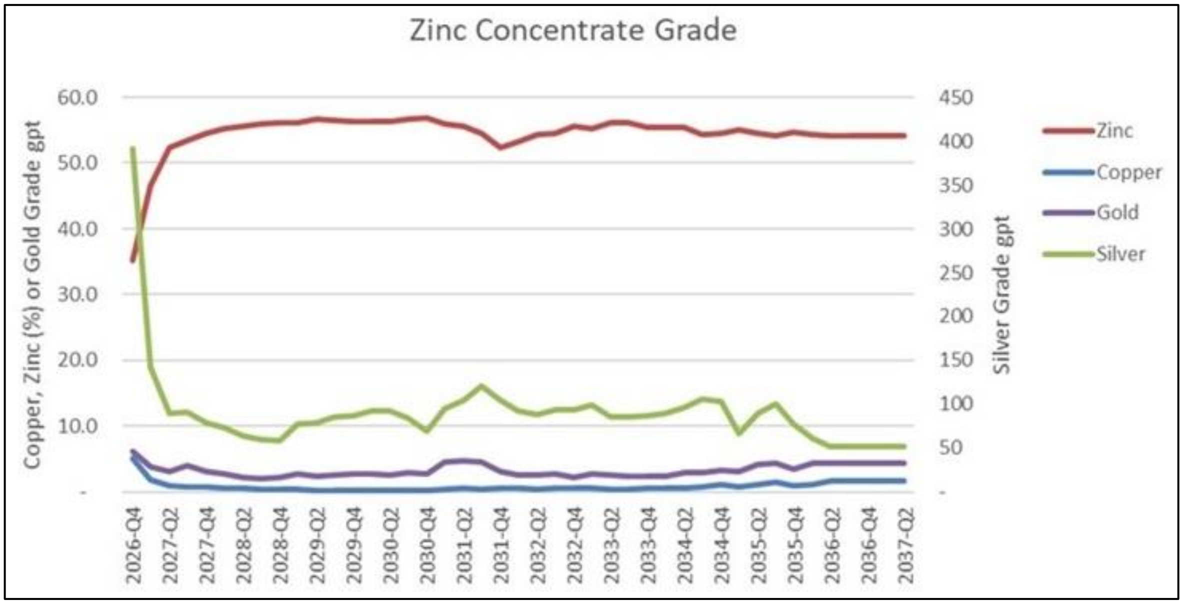


Figure 17-6: Graph of Zinc Concentrate Grades for Copper, Zinc, Silver, and Gold

The flotation plant was designed to produce and handle the following average daily concentrates produced:

- Copper concentrate; 239.2 tpd
- Zinc concentrate; 136.3 tpd.

However, given the variability of the deposits and consequently different flotation responses, as well as the potential influence of the ore sorter in removing barren material, the flotation circuit was designed to accommodate fluctuations in the feed grade of the ore.

17.6.2 Description of the Flotation Circuits

The major equipment in the copper and zinc flotation sections includes the following:

- A flotation feed trash screen at the head of the copper flotation stage
- Two rougher stage conditioning tanks for each circuit
- Flotation reagent addition facilities
- Rougher stage flotation cells
- A regrind mill and cyclone classifier
- A cleaner stage conditioning tank for each circuit
- Cleaner flotation cells, one cleaner stage for the copper circuit, and three cleaner stages for the zinc circuit
- Particle size monitors for each reground rougher concentrate
- A froth camera system for process control
- An on-stream sampling system
- Pump boxes and standpipes
- Sump, slurry and solution pumps
- An overhead crane
- Air blowers
- Safety showers and eyewash stations.

The flotation circuit feed comprises the cyclone overflow from the grinding circuit and contains zinc sulphate added in the grinding circuit. The feed particle size of the slurry will be a P_{80} of 55 µm. Prior to the flotation stage, the slurry will pass over a 2 m x 3 m trash screen with 2 mm slots to remove tramp oversize material and plastics which may have entered the slurry stream.

To facilitate the copper flotation process, the slurry will be conditioned in two sequential conditioning tanks. The slurry will initially be dosed with sulphur dioxide and sodium sulphide in the first conditioning tank, followed by lime for pH control in the second conditioning tank to ensure a slurry pH of 6.0. Together with the zinc sulphate (added in the grinding circuit), these reagents will be added to depress pyrite and sphalerite. The other reagents, namely the frother methyl isobutyl carbinol (MIBC), and the collector reagents Aero 3894 (a dialkyl thiocarbamate) and Aero 3477 (a dialkyl dithiophosphate), will be added to the cells at the head of the flotation stages. The copper flotation rougher circuit will consist of six 40 m^3 cells. The rougher concentrate will be collected and will feed the copper regrind mill and classification cyclone system to produce a P_{80} particle size of 15 µm. Lime will be added to the regrind circuit to control pH to 5.5. The reground rougher concentrate will feed the copper cleaner flotation circuit consisting of five 5 m^3 cells which will feed the copper concentrate thickener. The cleaner circuit also has a conditioning tank for dosing of the reagent sodium meta-bisulphide (SMBS) while the frother and collector reagents will be added directly to the flotation cells. The cleaner stage tailings will be combined with the copper rougher flotation stage tailings which will constitute the feed to the zinc flotation circuit.

An on-stream x-ray fluorescence (XRF) analyser will periodically sample the flotation feed, the rougher concentrate (after regrinding), the cleaner tailings and the cleaner concentrate streams for process control purposes. The combined rougher tailings and cleaner tailings streams, namely the zinc flotation feed stream, will also be sampled. After completion of the analysis, the respective slurries will be returned to the most appropriate part of the process.

Froth will be monitored by the control room operator through a camera system as an aid for controlling the copper flotation circuit.

The zinc flotation circuit will also have two conditioning tanks. The pH of the slurry will initially be adjusted with lime in the first tank to a value of 7.0, while copper sulphate will be added in the second conditioning tank to activate the sphalerite. The other reagents, namely the frother W31 (a glycol-ether), and the collector reagent Aero 3894, will be added to the cells at the head of the flotation stages. The zinc flotation rougher circuit will consist of five 40 m^3 cells. The rougher concentrate will be collected and will feed the zinc regrind mill and classification cyclone system to produce a P_{80} particle size of 20 µm. Lime will be added to the regrind circuit for pH control which will be 11.0 in the cleaner stages. The reground zinc rougher concentrate will feed the zinc 1st cleaner flotation circuit which will consist of a conditioning tank for lime for pH control, for the addition of reagents, and the 2nd cleaner tailings stream. The 1st cleaner circuit will be equipped with six 5 m^3 flotation cells. The 1st cleaner concentrate product will be pumped to the 2nd cleaner stage for further upgrading, while the 1st cleaner tailings will be combined with the zinc rougher stage tailings and will constitute the final flotation plant tailings. The final tailings stream will be thickened and discharged to the -TMF.

The 1st cleaner concentrate will be combined with the 3rd cleaner tailings product and will constitute the feed to the 2nd cleaner stage. The 2nd cleaner stage will consist of three 5 m^3 flotation cells. The pH will be controlled at the value of 11.0 by the addition of lime, while the frother and collector reagents will be added as required. The 2nd cleaner concentrate will be pumped to the 3rd cleaner flotation stage for the final stage of upgrading. The 3rd cleaner flotation stage will also consist of three 5 m^3 flotation cells. Reagents will be added as required while lime will be added to maintain the pH value at 11.0. The 3rd cleaner concentrate will be delivered to the zinc concentrate thickener and dewatered prior to despatch off the mine. The 3rd cleaner tailings will be recycled to the 2nd cleaner stage.

The on-stream x-ray fluorescence (XRF) analyser will periodically sample the zinc flotation feed, the rougher concentrate (reground product), the 1st cleaner tailings and the 2nd and 3rd cleaner concentrate streams for process control purposes. The combined zinc rougher tailings and 1st cleaner tailings streams, namely the final plant tailings stream, will also be sampled and analyzed. The respective slurries will be returned to the process.

Froth will be monitored by the control room operator through a camera system as an aid for controlling the zinc flotation circuit.

17.6.3 ***Concentrate Dewatering and Handling***

The copper and zinc flotation concentrate dewatering circuits are similar, and the major equipment items for each circuit includes the following:

- A concentrate thickener
- A flocculant addition facility
- Concentrate thickener overflow solution handling system
- Concentrate thickener underflow pumps
- A concentrate filter feed stock tank
- A pressure filter system including feed pumps, compressed air, and a filter wash water and filtrate handling system
- A filtered concentrate storage facility
- Sump, slurry and solution pumps
- Under-cover storage of concentrate
- Concentrate load-out area for trucks (weighbridge located at the gatehouse)
- Safety shower and eyewash stations.

The copper and zinc concentrates will be dewatered in the same manner. The final copper and zinc flotation concentrates will be discharged to the respective concentrate thickener, each having a diameter of 11 m. Flocculant will be added to aid the settling process. The thickener overflow solution will be collected and pumped to the process water tank for recycling via the water treatment facility. The thickened concentrate will be pumped to the respective concentrate stock tank using the concentrate thickener underflow slurry pumps at a density of 60% solids. A pressure filter will dewater each concentrate to produce a concentrate with a moisture content of 8–9% (9% moisture was used in the economic analysis). Each filter press will have plate dimensions of 1.5 m x 1.5 m with up to 60 (copper) and 40 (zinc) plates. The number of plates for each concentrate will depend on the rate of concentrate production with the number of plates varying, namely 46 and 27 plates respectively for the copper and zinc filter press under average production conditions. At the end of the filter cycle, the filtered concentrates will be discharged into designated storage areas. Each filtered concentrate will have a storage capacity of 2,000-tonne equivalent to about eight days of copper production, and 14 days of zinc production (under average plant feed grade conditions). The copper and zinc concentrates will be loaded onto 40-tonne capacity trucks, sampled, and weighed prior to dispatch from the mine. The filtrate and wash water solutions will be returned to the process water tank for recycling.

The concentrate dewatering area will be equipped with safety showers and eyewash stations, while the concentrate handling area will also be equipped with a shower and eyewash station as well as a dust collection system.

17.6.4 Tailings Disposal

The flotation tailings from the zinc flotation circuit will be the final plant tailings and will be discharged to the tailings thickener for water recovery and to reduce the volume of the slurry deposited at the TMF. The main items of equipment will be the following:

- A tailings thickener feed tank
- A tailings thickener
- The tailings thickener overflow solution handling system
- The tailings thickener underflow pumps
- Flocculant addition facility
- Sump, slurry and solution pumps
- Safety shower and eyewash station.

The final tailings from the zinc flotation circuit will be discharged to the tailings thickener feed tank and then flow to the thickener. The tailings thickener will have a diameter of 25 m. Flocculant will be added to the tailings slurry to aid the settling process to obtain a density of 65% solids in the thickener underflow. The thickener overflow solution will be collected and pumped to the process water tank for recycling via the water treatment facility. The thickened tailings will be pumped to the TMF.

17.6.5 Fresh Water Supply System

Fresh water will be provided from wells and will supply the potable water requirements of the camp and the mine administrative facilities. Fresh water will also be pumped to the Fresh Water/Fire Water Tank and distributed as make-up water for the plant process and for reagent preparation.

17.6.6 Reclaim Water

Water accumulating in the TMF will be recovered using a floating intake system with a shore based pump to deliver the reclaimed water to the water treatment facility via the contact water pond. The treated water will be recycled to the plant process water tank and re-used.

17.6.7 Water Treatment Plant

Surface, underground, contact, treated sewage, and process plant water are to be treated by a single water treatment plant. For the six ice free months of the year, May to October, the plant is designed to treat average and maximum flow rates of 249 L/s and a 320 L/s, respectively. Approximately 112 L/s of treated water will be returned to the process plant and the remainder discharged to the environment. For the six non ice-free months of the year, November to April, the plant is designed to treat water from the tailings thickener overflow and return the treated water back to the processing facility for re-use.

The water treatment system process steps are:

- Metals removal using ferric sulphate and alkali in reactor tanks
- Clarification to remove suspended solids
- Sludge recirculation to the metals removal stage
- Sludge pumping to the tailings thickener
- Polishing filtration
- Neutralization
- Treated effluent distribution.

Treated effluent is planned to be discharged during ice-free months into Kutcho Creek just north of the confluence with Andrea Creek, a distance of approximately 10 km from the mine site. Treated effluent is to flow through a 400 mm diameter HDPE DR 11 line and discharged through a spray header over Kutcho Creek. The spray header will be supported by two concrete posts, one on either side of the creek, located outside of the highwater zone. A pump will fill the pipeline with treated effluent though the pipeline is designed for gravity flow. The line will be emptied prior to freezing conditions to prevent damage.

The discharge line will run on surface adjacent to the access road and then along a service road to Kutcho Creek.

17.6.8 Services

Compressed air will be generated for filter, instrument, and maintenance purposes. Air blowers will produce the low-pressure air required for the flotation process.

17.6.9 Reagent Preparation

Various chemical reagents will be added during the flotation process to facilitate the production of the two flotation concentrates. The preparation of the reagents will require the following facilities:

- A bulk handling system
- Mix and holding tanks
- Metering pumps
- A flocculant preparation and delivery system
- A lime slaking facility
- Overhead cranes
- Dust collection system
- Safety shower and eyewash stations.

Various chemical reagents will be added to the grinding and flotation circuit to modify the mineral particle surfaces and to enhance the floatability initially of the copper minerals, followed by the zinc minerals, and their recovery into two distinct concentrate products.

The reagents will be received in bulk in palletized bags, chemtainers, drums or bulk bags. Fresh water will be used for making up and/or dilution of the applicable reagents which are supplied in powder/solids form, or which require dilution prior to addition to the slurry. These prepared solutions will be dosed at the flotation circuit addition points using metering pumps. The reagents thus prepared include the flocculant preparation system, copper sulphate and zinc sulphate, SMBS, sodium sulphide, and sulphur dioxide. A lime slaking facility will prepare the lime required for pH control and for use in the water treatment facility. Each reagent will be prepared within its own bunded area in order to contain any spillage and avoid the contamination of other areas. A dedicated sump pump will be installed to return any spillage to the mixing tank of each specific reagent. Each bulk handling preparation facility will be equipped with a dust extractor.

Some reagents will be received in bulk containers and will be added directly to the flotation circuit without dilution and/or preparation. These reagents include MIBC and W31 (frothers), and Aero 3477 and Aero 3894 (collectors). The empty containers will be returned to the supplier or recycled.

17.6.10 Assay and Metallurgical Laboratory

The assay laboratory will be equipped with the necessary analytical equipment and instruments to provide routine sample assays for the process plant, as well as for the mining and the environmental departments. The most important equipment includes the following:

- Preparation, handling and drying facilities required for the bulk handling of shift samples

- Gold and silver assay facilities
- An atomic absorption spectrophotometer
- An inductively coupled plasma spectrometer
- A Leco furnace
- Glassware and equipment required for wet chemical assays, balances and hot plates
- Safety equipment, including all the applicable material safety and data sheets.

The metallurgical laboratory will be equipped with the necessary equipment to conduct metallurgical investigative test work to monitor process performance in order to improve unit operations and efficiencies. The laboratory equipment will include a laboratory-sized crusher, mill, flotation cells, filtering devices, screens, balances, drying ovens, etc.

17.7 Process Control Philosophy

The plant control system will be a programmable logic controller (PLC) system to control and monitor all plant operations and processes. The plant is divided into different process areas. Each process area is controlled by a single PLC system. The PLCs will be linked to provide plant-wide control using an Ethernet communication system.

Process control and monitoring for the facility will be performed in two centralized control rooms. The control rooms will be in the main process plant and in the primary crusher area. Human machine interface (HMI) operator stations will be in the control rooms. These HMIs will display the process equipment and operating parameters graphically. The PLC in conjunction with the HMI will perform all equipment and process interlocks, level control, alarms, and generate reports.

The motor starters and VFDs will be controlled by the PLC via a device net communication system.

17.8 Material Consumptions

17.8.1 Crushing Supplies

Crushing supplies were developed from vendor recommended spares and costs for main components at 0.055 kg/t of ore crushed.

17.8.2 Comminution Supplies

Comminution supplies were developed from abrasion index testing, mill power draw and vendor information. The vendor cost of one set of liners was included per annum for the regrind mills, ball mill and SAG mill. The wear rates for grinding media were developed from the abrasion index and mill power draw. The estimated steel wear rate was 0.28kg/t and 0.80kg/t ore processed for the SAG and ball mill, respectively.

17.8.3 Reagents

The reagent regimen was developed from the BML #714 metallurgical report. The QP considered it appropriate that the test work results be modified to reflect a 25% reduction in consumption based on expected variance that is expected to be developed through operational experience.

Table 17-6: Process plant reagent consumption

Plant reagents	Consumption (kg/t of ore)
Zinc Sulphate	1.500
Copper Sulphate	0.478
Aero 3894/Polyfloat 2979	0.078
Aero 3477	0.078
Na_2S	0.750
SMBS	1.153
Liquid SO_2	2.430
W31 Frother	0.023
MIBC frother	0.030
Lime – Quicklime	2.298
Flocculant – Magnafloc MF351	0.030

17.8.4 Other Operating and Maintenance Supplies

Filter cloth consumption was based on vendor information and was estimated at one complete set per annum per filter. General maintenance parts were estimated based on the equipment listing.

No maintenance and operating supplies were specified other than those listed in this section. The costs for all other items were identified from the equipment list and database factors of equipment capital costs. These costs are summarized in Section 21.

17.8.5 Power

Comminution test work was used to estimate the SAG mill and ball mill annual power consumption. A load list was developed to predict the power consumption for the remainder of the plant based on the installed power and efficiency factors.

The SAG and ball mill motors are conservatively sized compared to the average power consumption calculated using comminution test results. This provides the Project with flexibility for unforeseen variability in localized rock hardness variability or the impact of the ore sorters. The power plant has a maximum predicted loading of 79% of the installed 10 MW generating capacity (one 2.5 MW unit is held as a rotating standby and 2 MW of diesel capacity is used to provide additional torque for start-ups and emergency backup).

Total power consumption per annum is estimated at 62,757 MWhr. The processing plant is the major consumer of power from the LNG power plant which peaks at 68,853 MWhr per annum. A sitewide power consumption table is provided in Section 21.

17.8.6 Liquid Natural Gas

The plant consumes liquid natural gas (LNG) for power generation and heating. Heating requirements were based on a target 5°C working temperature for the crusher, warehouse, and processing buildings, consider burner efficiency, building dimensions, and seasonal temperature.

Since the processing plant is the major consumer of power and the power plant is the major consumer of LNG all other consumption sources of LNG are provided here in Table 17-7.

Description of the power plant is given in Section 18.8 Heating requirements for underground and workshop are discussed in Section 16. Camp heating requirements were estimated based on consumption per person with verification to other regional projects completed by AllNorth.

The power plant LNG consumption was developed with the vendor and was estimated at 8.48 Mj/eKWhr and verified against other installations completed by Allnorth.

Table 17-7: Project LNG consumption per year (LNG, Mm³)

	Total	-2	-1	1	2	3	4	5	6	7	8	9	10	11
Power Plant	155.96	-	2.33	14.91	14.91	14.82	14.70	14.86	14.89	14.84	14.72	14.32	14.12	6.55
Underground heating	6.04	0.02	0.37	0.71	0.75	0.75	0.71	0.74	0.71	0.38	0.56	0.33	-	-
Plant heating	3.19	-	0.13	0.29	0.29	0.29	0.29	0.29	0.29	0.29	0.29	0.29	0.29	0.16
Workshop heating	0.47	0.04	0.04	0.04	0.04	0.04	0.04	0.04	0.04	0.04	0.04	0.04	0.04	0.02
Camp, office	1.12	0.08	0.13	0.11	0.10	0.10	0.09	0.10	0.10	0.09	0.08	0.07	0.06	0.03
Total	**166.78**	**0.14**	**2.99**	**16.06**	**16.09**	**15.99**	**15.82**	**16.02**	**16.02**	**15.64**	**15.69**	**15.04**	**14.51**	**6.76**

17.8.7 Diesel Consumption

Mobile equipment at the plant excluding the stockpile rehandle loader are estimated to consume around 120,000 litres of diesel per annum. The rehandle loader diesel consumption is stated with the mine open pit fleet. Diesel fuel and diesel exhaust fluid is stored and dispensed at the fuel farm and a description is provided in Section 18. A site-wide diesel consumption table is provided in Section 16.

17.8.8 Water Consumption

A plant water balance was developed. The main water streams are summarized in Table 17-8. Some of the make-up water required will come from water reclaimed from the TMF or directly from the process plant thickener overflows. All the water supplied to the process plant will be treated by the water treatment plant prior to use.

Table 17-8: Water inflow and outflow streams

Process Stream		m³/hour	m³/ day
Water IN:	Water contained in Plant Feed	9.87	236.84
Water OUT:	Water contained in Sorter Rejects	1.23	29.49
	Water contained in Concentrates	1.36	32.65
	Water contained in Final Tailings	90.92	2182.03
Make-up water required		83.64	2007.33
Fresh water required		10.61	254.54
Recycled treated water required		73.03	1752.79

17.8.9 Personnel

Personnel requirements were developed on the basis of operational requirements, shift and leave rosters, equipment attendance, safety, training, and maintenance requirements. Allowance was made for start-up training and lower productivity during commissioning. Average annual personnel numbers are provided in Table 17-9.

Table 17-9: Plant personnel requirements

Area/Position	Shift	No.
Mill Management		
Mill Superintendent / Plant Manager	Day	1
Chief Metallurgist	Day	1
Senior Metallurgists	Day	1
Plant Metallurgists	Day	1
Mill Metallurgical Technicians	Day	2
Plant Trainer	Day	1
Mill Clerk	Day	1
Operations Labor		
Mill Foreman / Operations Shift Supervisor	Shift	4
Control Room Operators	Shift	4
Crusher Plant Operators	Shift	4
Grinding/Flotation Operators	Shift	4
Concentrate De-Watering & Loadout Operators	Shift	4
Ore Sorter attendant	Shift	2
Reagents / Utilities	Day	2
Mill Labourers / Trainees	Shift	4
Laboratory		
Chief Chemist / Chief Assayer / Lab Manager	Day	1
Senior Chemist	Day	1
Chemical Analysts	Day	2
Sample Preparation / Buckers	Day	2
Plant Maintenance		
Mill Maintenance Foremen	Day	2
Mill Maintenance Planner	Day	1
Mill Maintenance Clerk	Day	1
Millwrights - Journeyman	Shift	4
Welders - Journeyman	Shift	4
Apprentices	Shift	4
Electricians day shift	Day	2
Electricians	Shift	4
Senior Instrument Technician	Day	1
Instrument Technicians	Shift	4
TOTAL		**69**

Note: Shift is Night/Day.

18 Project Infrastructure

18.1 Introduction

No infrastructure exists at the proposed mine site other than drill-site access trails. The site is currently accessed from Highway 37 via the existing Boulder trail, or via the airstrip. The compact project footprint reflects the constraints imposed by environmental considerations and site topography.

The site plan Figure 18-1 shows the major Project facilities, including the open pit mine, portal entrance, tailings management facility (TMF), waste rock dump (WRD), and crushing and processing facilities. Additional infrastructure to support the Project will consist mainly of access road improvements, onsite civil work including roads pads and water management infrastructure, administration buildings, accommodation camp, maintenance facilities, a water supply system, water treatment and discharge systems, and site electrical.

Site facilities include both mine facilities and process facilities:

- The crushing and process facilities pad includes the administration offices, assay laboratory, warehouse, workshop, reagent storage, and miscellaneous other facilities.

The Project is designed to be serviced by potable water wells located adjacent to the camp, a potable water treatment facility, fire water, on-site power generation system, compressed air, and communications.

The airstrip, bulk explosives storage facilities and the treated effluent discharge site are located up to 12 km away from the main mine site and are shown in Figure 18-2.

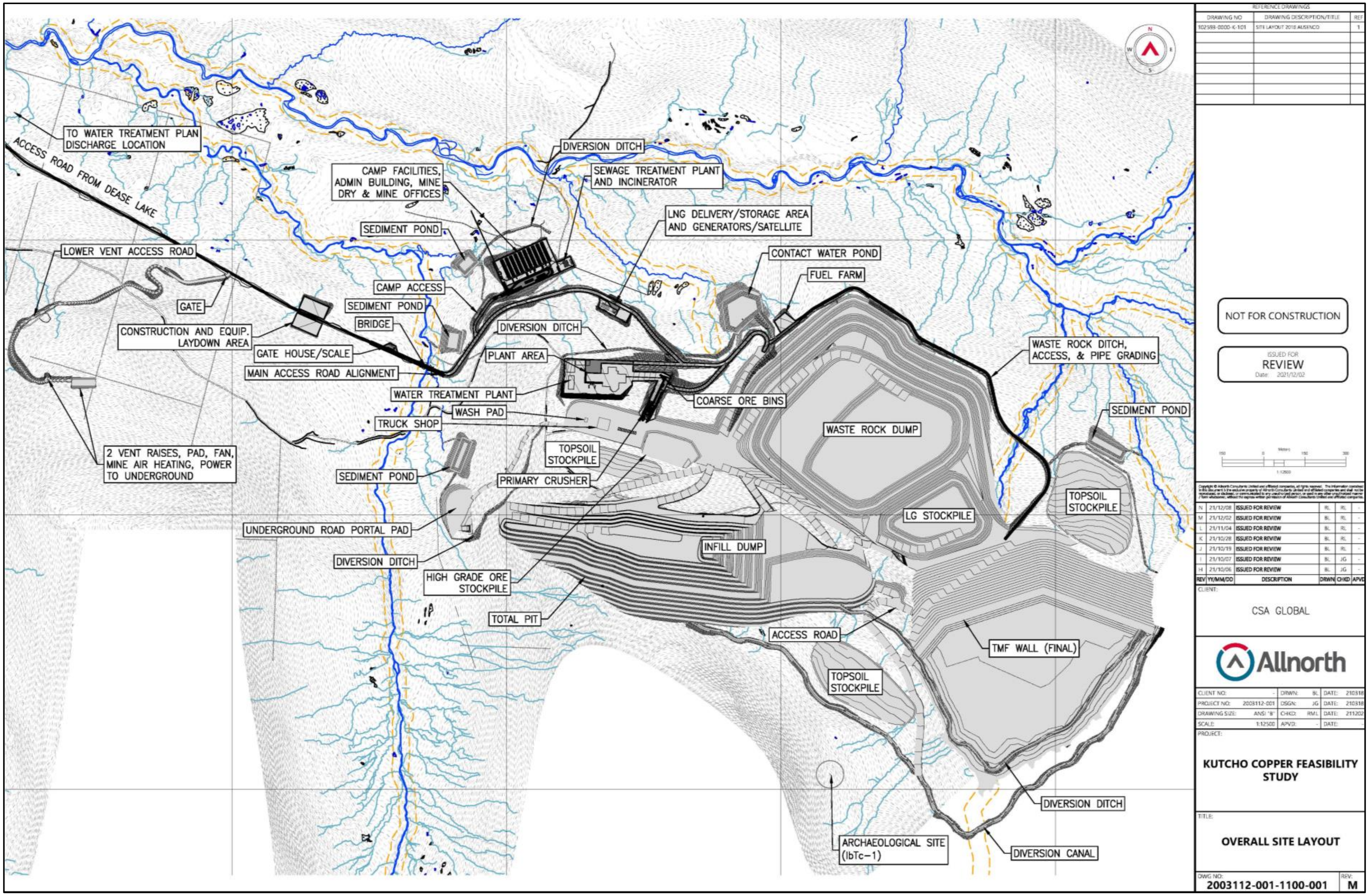


Figure 18-1: Site Layout NAD83 UTM Zone 9N

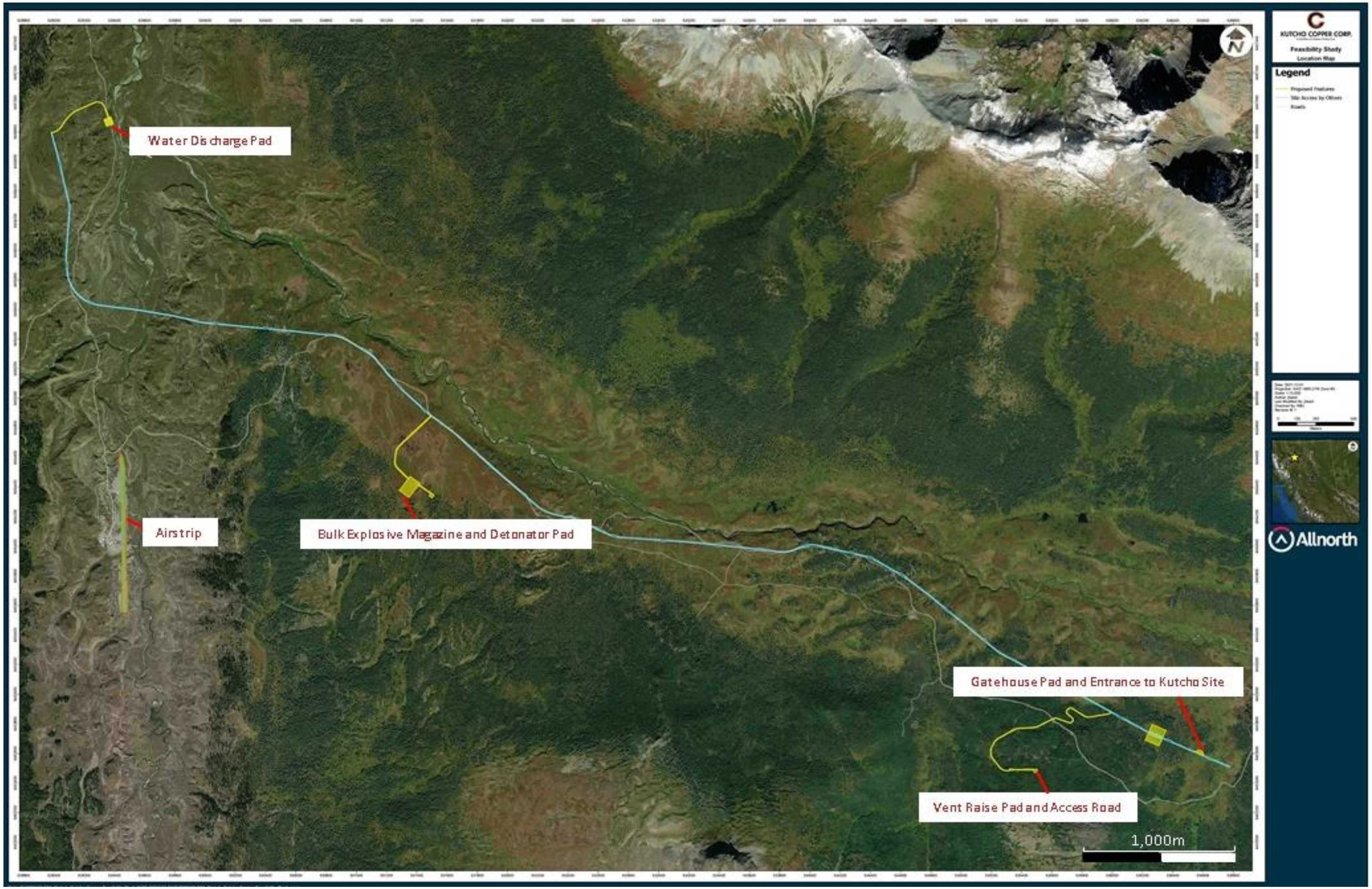


Figure 18-2: Airstrip, Bulk Explosives Storage Facilities, and Treated Effluent Discharge Locations (NAD83 UTM Zone 9N)

18.2 Off-Site Roads

Onsite Engineering Ltd ("Onsite") completed the design of the 119 km long Project access road. This report section summarises the work undertaken by Onsite. Complete details can be found in Onsite's "Kutcho Access Road, Geometric Road Design Report" dated 13 October 2021.

The access road is designed for year-round use by freight trucks, buses and some light vehicles for the duration of the Project. The road is by majority a gravel surfaced, single lane, five-meter width with inter-visible turnouts and radio control with a maximum design speed of 50 kmph. Bridges are designed to accommodate BCL-625 Highway Legal Loading as specified by the Canadian Highway Bridge Design Code. The road mostly upgrades the existing Boulder Trail but has some new detours to decrease wetland interaction and improve trafficability. The construction schedule allows for a period of snow road usage to enable the Project construction schedule.

18.2.1 Design Basis Information

Several information sources were used by Onsite to complete the proposed access road design. Those information sources are described in the following list:

- Manually collected GPS survey of the existing roadway and adjacent ground
- Data collected by hand traversing portions of the alignment that were unable to be GPS surveyed
- High-order GPS points for maintaining horizontal and vertical controls completed by Allnorth
- Review of crossing designs and hydrology previously completed by Allnorth
- Fisheries Assessments completed by Rescan Environmental Services Ltd
- Geology and Surficial Materials and Soils baseline report completed by Rescan Environmental Services Ltd
- Gravel source and terrain hazard mapping completed by Madrone Environmental Services Ltd.

The design is also supported by reference texts such as the Fish-Stream Crossing Guidebook (BC Ministry of Forests, Lands, Natural Resource Operations and Rural Development, 2012), The Engineering Manual (BC Ministry of Forests, Lands, Natural Resource Operations and Rural Development, 2019), 2020 Standard Specifications for Highway Construction (British Columbia Ministry of Transportation and Infrastructure, 2020), amongst others

18.2.2 Layout and Survey

Sections of the original access road alignment location were re-aligned by Onsite, at the request of Kutcho Copper, to a lower design speed with the intent of reducing the road construction costs. Field location of the new road included pre-field and field reconnaissance works by a Professional Engineer followed by field layout and traverse of the revised road centreline taking into account the road design specifications provided by the previous owner, Capstone. Sections where slope stability was assessed to be of concern were also reviewed in the field by a Professional Geoscientist specialized in the assessment of slopes with respect to road construction. Onsite completed detailed hand traversing and real-time kinetic (RTK) survey of the revised road centreline locations tied to previously existing Allnorth control points for coordinate system continuity. The level of detail required from the road survey is tied to the complexity of the end result required by Kutcho Copper. As the end result will be a gravel-surfaced mining haul road, a level 2 survey was deemed to be appropriate for this project. Refer to Figure 18-3 for an overview of the alignment.

For the purposes of this design, it was determined through discussions with Kutcho Copper that the original hand survey methods were the most appropriate to achieve the goal of a Feasibility Study level of design.

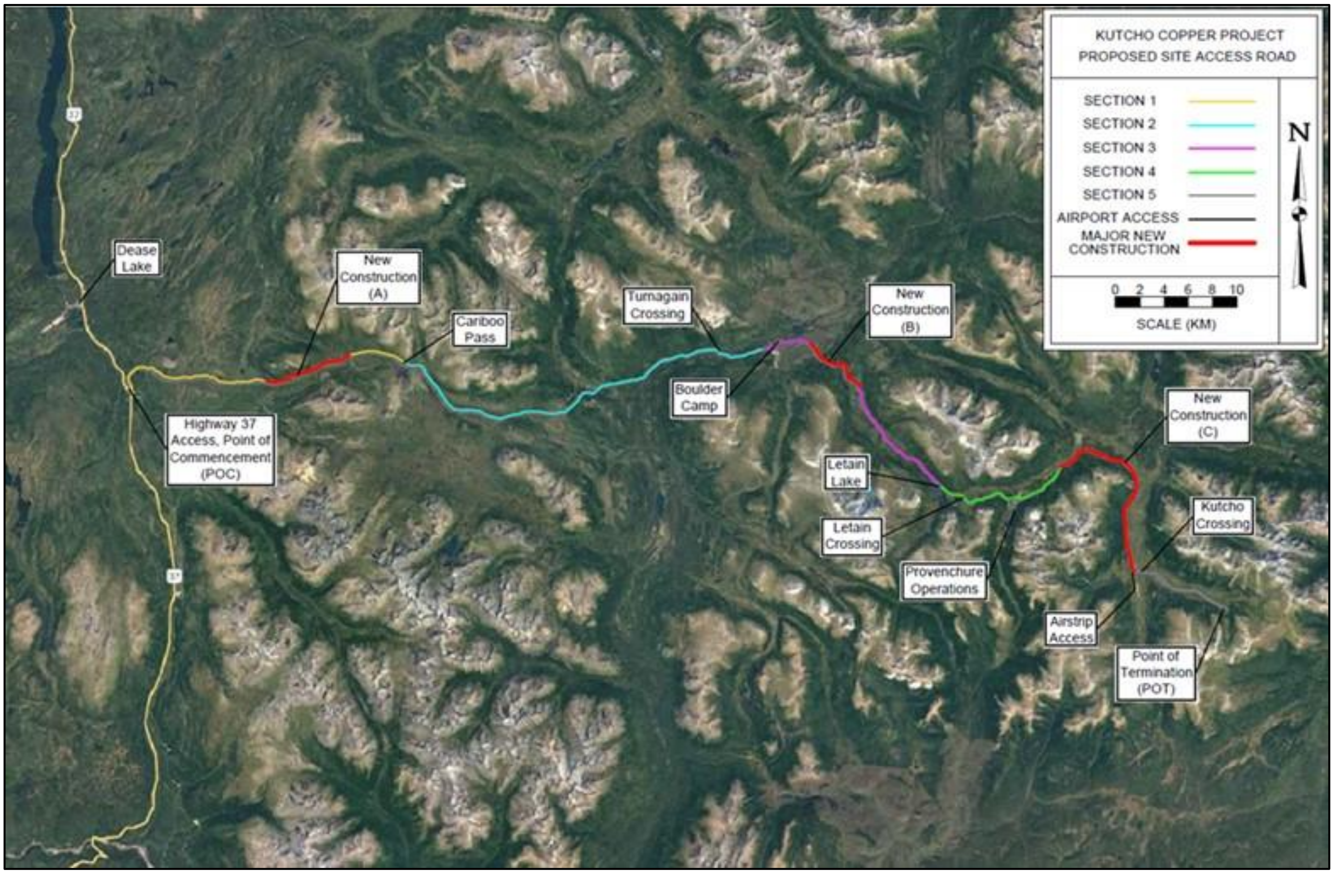


Figure 18-3: Road Overview Map NAD83 UTM Zone 9N

18.2.3 Road Design

The purpose of a road design is to produce specifications for road construction by determining the optimum road geometry that will accommodate vehicle configuration, traffic volume, and user safety, while minimizing the cost of construction and road maintenance. Road design is a process that takes collected survey information and produces a desired road profile showing grade, alignment, excavation and embankment volumes, drainage structure location, turnouts, and surfacing requirements. The optimum road design reduces impacts on other resources by minimizing clearing widths and excavations. Considerations are made for anticipated construction equipment to be used to optimize material movement distances, construction techniques such as rolling grades or end hauling, road widths, cut and fill angles, and horizontal and vertical control angles.

18.2.4 Design Parameters

The following parameters were used to design the alignment and cross-section templates and determine earthworks volumes. These parameters were guided by Kutcho Copper with additional references from the Forest Road Engineering Guidebook (2002) and the Ministry of Forests Engineering Manual (2009).

The MOFLNRORD Engineering Manual specifies an 8.0 m road width to be used for a 50 kmph design speed. This specification assumes the road would be designed to accommodate two-way traffic, capable of passing one another without slowing down or pulling over. There are some cases where deviations have been made from the aforementioned references as per industry standard practice. For example, the road is being designed as a single lane private mine access road with inter-visible turnouts and radio control. As such, a 5.0 m road width is appropriate to accommodate users and road safety.

The road design is based on the most economical route as a function of construction cost estimates, surficial and subsurface material types, and construction difficulty. Additional details can be found in Onsite's "Kutcho Access Road, Geometric Road Design Report" dated 13 October 2021.

Table 18-1: Road design parameters

Design parameter	Comments				
Finished road width	5.0 m.				
Design speed	50 kmph.				
Vehicle loading	BCL-625 Highway Legal Loading as specified by the Canadian Highway Bridge Design Code.				
Road surface	0.15 m of surfacing.				
Sub-base	Minimum 0.35 m of subbase as required in overland construction or in areas with limited subgrade bearing capacity.				
Road fill	All large fills to be well drained granular materials, placed and compacted in maximum 0.3 m thick lifts where stability or settlement are critical. Other fills may be constructed with local materials and compacted with the excavation equipment (i.e. bucket and track packing).				
Ditch	Minimum 0.9 m depth, 0.6 m bottom width. Ditch will be larger as specified in the Geotechnical Report or as required to accommodate culvert cover.				
Crown	2 % crown.				
Turnouts	10 m entry and exit tapers, 15 m long turnout, 7.5 m width from centreline.				
Clearing width	20 m or 3 m from the limits of excavation, whichever is larger.				
Horizontal alignment criteria	**Design speed (kmph)**	**Minimum stopping sight distance (m)**	**Minimum curve radius (m)**		**Maximum road gradient (short pitch)**
	50	135	100		8% (10%)
	40	95	65		12% (14%)
	30	65	35		12% (14%)
Vertical alignment criteria	**Design speed (kmph)**	**Minimum stopping sight distance (m)**	**Minimum K-value**		**Maximum road gradient**
			Sag	**Crest**	
	50	135	12	11	10% (12%)
	40	95	7	5	12% (14%)
	30	65	4	3	12% (14%)

18.2.5 Bridges and Culverts

The access road, from the junction with Highway 37 to the project site at approximately km 119, crosses 134 S-class streams. Of these, 20 are major culvert crossings and 32 are clear-span bridges. There are no existing structures in place along the alignment in these proposed crossing locations.

Fish habitat assessments of drainages along the access road, completed by Rescan from 2006 to 2008, included detailed habitat assessment and stream class identification (Rescan Environmental Services Ltd, 2008). The classifications from the 2006 and 2008 field studies have been used in determining the appropriate crossing method for each crossing. Crossings along the proposed alignment include standard non-fish stream culverts, clear span all steel portable (ASP) bridges on a variety of abutment types, clear span concrete slab girders on integrated ballast walls, and clear span concrete composite bridges on a variety of abutments. It should be noted that there is additional baseline habitat data that was collected in 2013 and 2021. As understood by Onsite, this additional data should not alter the recommended crossing types.

18.2.6 Key Construction Sections

For reference, Km 0+000 is the intersection with Highway 37. There are three main sections requiring a significant amount of continuous new construction. The sections are (A) 14+700 to 21+900, (B) 65+000 to 71+000, and (C) 94+000 to 110+00. Further, the road was broken into five segments for costing and design purposes. Further detail can be found in Onsite's "Kutcho Access Road, Geometric Road Design Report" dated 13 October 2021. Refer to Figure 18-4 for locations and landmarks related to the Project.

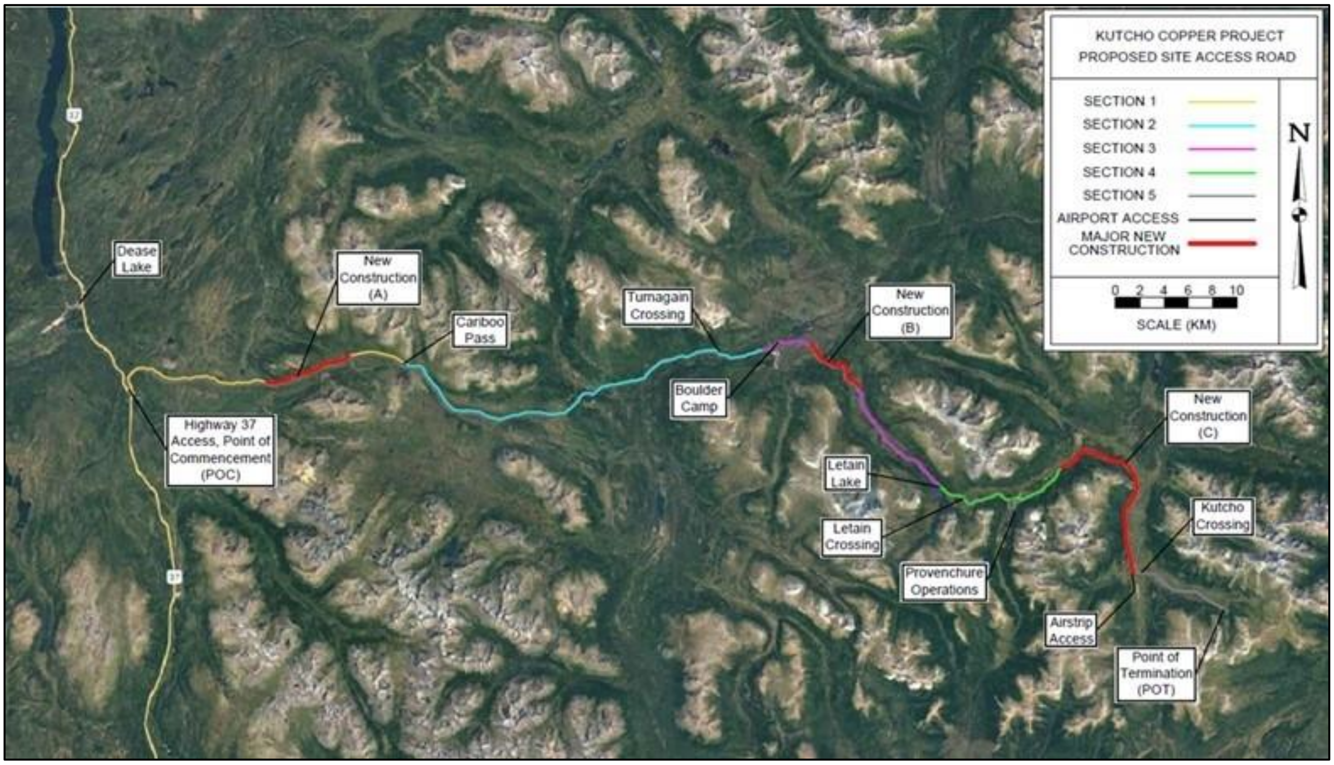


Figure 18-4: Overview Map Showing Design Sections (NAD83 UTM Zone9N)

The first key section (A) is intended to bring the road out of the low-lying drainage paths and provide a well-drained durable road. Most of this section is conventional cut and fill in silty sand and glaciofluvial deposits. There are numerous small unclassified drainages through this section that will require culvert installations. The construction through this section will not be significantly challenging but this section is critical to complete early in the construction phase to avoid the wet, low-lying areas this section bypasses. This section can be built with crews working from either end.

The second key section (B) begins approximately 4 km from Boulder Camp. This section of road cuts off part of the existing trail that follows low-lying ground and eliminates approximately 1.5 km from the existing access trail. This section crosses gentle to moderate side slopes with local areas up to 60%. The alignment alternates between conventional cut-and-fill construction and overland construction through poorly drained low-lying areas. There will be a significant quantity of fill material required to construct the overland portions. There are borrow sites that have been identified on either end of this section between 61+000 and 73+000. The early development of these sites will be critical in enabling the construction of the overland sections.

The third key section (C) begins approximately 4 km after the Provenchure operations. This section of new construction avoids the river flats at the south end of Wolverine Lake and many low-lying sections of the existing access trail along the west side of Kutcho Creek. This segment departs the existing trail at approximately station 94+000 and crosses gentle side slopes with intermittent moderate side slopes. The terrain requires alternating sections of overland construction and conventional cut-and-fill. The alignment transitions into moderate side slopes from 96+300 to 102+300 with frequent wet conditions expected. The alignment transitions back to gentle side slopes at 102+300 with continued intermittent wet subgrade conditions. A significant amount of imported fill will be required for the overland portions of this segment. There are seven identified borrow locations along this segment that require development.

A fourth short section from 82+300 to 85+000 requires relatively simple construction but has some challenging portions near the Letain River crossing. This section of road detours around the low poorly drained areas near Letain Lake.

It is expected that the crossings will be constructed in mostly sequential order starting nearest to Dease Lake. However, the Letain, Turnagain, and Kutcho crossings should be given special consideration as they are larger structures that require more work than most of the other crossings. They also eliminate existing non-engineered fords to avoid issues that would be associated with the increased construction traffic during construction of the road. Constructing these crossings quickly and early will facilitate efficient material movement for road construction purposes.

*18.2.7 **Winter Road***

So that the road can be constructed in one season, a pre-construction winter road is proposed. The winter road will serve several purposes:

- Enabling mine and mill equipment to be mobilized at an early stage in order to partially avoid using the road during the main construction phase
- Enable the entire road to be constructed to a useable and finished state within one construction season by pre-placing all crossing structures at or near their locations
- Enable construction equipment and camps to be staged and established at appropriate locations prior to the construction season
- Enable ROW clearing in the winter season to avoid bird nesting closures.

The winter road is expected to follow the approximate alignment of the existing access trail. It will be critical to have early ground freeze to maximize the useable window of the winter road.

All crossings are expected to freeze to the stream bed, except for the Turnagain River, Letain Creek, and the Kutcho Creek crossings. These three crossings will require an all-steel portable bridge set on temporary timber sills.

It was assumed that once the winter road is established, tractor-trailer combinations will be capable of transporting crossing materials to each site. If the road is not suitable for this configuration, six wheeled low ground bearing pressure, flat-bed cargo trucks, known as Deltas will be required to move materials, this would significantly increase the transport costs.

The winter road design is outside the scope of the access road design, but the winter road is expected to be constructed as a Snow Road as per section 5.10.1 of the Engineering Manual (BC Ministry of Forests, Lands, Natural Resource Operations and Rural Development, 2019).

18.3 On-Site Roads

The mine site roads include haul roads, two lane access roads, and single lane roads.

The haul roads connect the fuel farm pad with the WRD, the crusher pad, warehouse pad, TMF, topsoil stockpiles, and ore stockpiles. The haul roads are approximately 5.2 km in total length and designed to be 25 m wide. Further design details of the haul roads are provided in Section 16.

The two-lane roads are designed to provide access to all the major site facilities and infrastructure. The on-site roads tie into the main site access road approximately 190 m east of the gatehouse. The on-site roads have a total length, including the camp and plant access roads, of approximately 2.3 km. The road section is designed to be 9.0 m wide to accommodate two-way, light vehicle traffic.

Single lane roads are designed to provide access to critical infrastructure during construction and periodic maintenance. They are typically designed to be four metres wide except for the diversion structure roads which are 3.0–3.5 m wide. The single lane roads make up a total length of approximately 8.7 km. The minor roads are:

- The WRD road which is to provide access between the fuel farm area and the lower TMF. The access road is designed to have a larger shoulder, which will allow for piping to be placed adjacent to the road.
- The ventilation raise pad access road.
- The water discharge access road.
- The bulk explosives and detonator pads access road.
- The diversion ditch and diversion canal roads which are to provide access during construction and for maintenance to the diversion structures.

18.4 Tailings Management Facility

18.4.1 Tailings Management Facility Background

There have been several tailings management studies for the Project dating back more than a decade. However, most of the sites evaluated are no longer under consideration due to conflicts with other mine facilities, impacts on sensitive habitat, or technical limitations (such as inadequate storage capacity). These studies were performed at levels of detail ranging from preliminary assessments to Pre-feasibility. The most important geotechnical study was completed by AMEC in 2007 (Kutcho Project 2007 Geotechnical Investigation Factual Report, April 2008) and this work included limited investigation of the Playboy Creek TMF site. In 2018, Ausenco was engaged to perform a Feasibility Study but that study, which focused exclusively on one location in Rusty Creek, was terminated shortly after the geotechnical fieldwork commenced. In 2020 the current Feasibility Study was commissioned. This study included additional geotechnical investigation work in Playboy Creek in 2021.

The governing design criteria for the facility are the Canadian Dam Association (CDA) Dam Safety Guidelines. In accordance with the CDA dam classification methodology the proposed TMF is provisionally classified as having a "high" consequence of failure, based on the potential impacts to the environment. The design of the TMF was carried out to exceed the minimum allowable factors of safety under static and pseudo-static conditions recommended by the CDA.

18.4.2 Options Evaluation

Ausenco's Tailings Management Facility Trade-off Study (July 2018) considered four disposal technologies (low density and thickened tailings slurry, paste, and filtered tailings) and two TMF locations (Site 1 and Site 2, both on Rusty Creek) for a smaller project (storage for 4.3 Mt of tailings) that considered underground disposal of some tailings. Ausenco ranked the options and sites using a weighted matrix approach, based on a range of technical considerations from downstream impacts and ease of closure, capital cost, operating cost, and net present value. From this they ranked low-density slurry disposal at Site 2 as the preferred option.

In 2020, Piteau Engineering Associates performed an updated trade-off study (TSF Disposal Options Analysis, November 2020), considering changes to the design concept to optimize costs (especially for the rock fill used for most of the dam construction) and to remove Site 2 from consideration due to encroachment on sensitive fish habitat. Thus, only Site 1 was considered. This study used a similar ranking matrix as Ausenco's and the recommended option was thickened tailings disposal at Site 1.

In early 2021, the project was reconfigured from an underground mine only to principally open pit extraction. Consequently, the size of the reserve increased, requiring a much larger TMF storage capacity. The economics of Site 1 become very unattractive at this size and the decision was made to move the TMF to Playboy Creek, immediately east of the pit. This also allowed the embankment design to rely on waste rock mined from the pit in place of quarried rock or borrowed soil. Further, the storage efficiency (storage volume/embankment volume) of the larger TMF at the Playboy Creek site was better than the smaller TMF at Site 1. The combination of lower cost waste rock and improved storage efficiency significantly reduced the unit disposal cost. The Playboy Creek site was further away from the sensitive fish habitats, thus improving environmental and closure issues.

18.4.3 Design Criteria

The TMF was designed to safety store 13 Mt of tailings along with site contact water (up to 1,008,000 m^3). The dam will be constructed as a starter dam in the pre-production period and then three downstream raises, using principally compacted nPAG waste rock generated from the open pit operation. The TMF was designed based on the following criteria:

- Tailings discharge rate = 1.31 Mtpa
- Minimum ultimate storage capacity = 11.7 Mt
- Tailings specific gravity = 3.69 (Main ore zone) and 3.19 (Esso ore zone)
- Particle size distribution, P_{80} = 59 µm (Main) and 67 µm (Esso)
- Tailings discharge slurry solids content = 65% (mass basis)
- Settled tailings dry density = 2.1
- Tailings slope = 1% above the pool, 2.5% submerged
- Embankment slopes = 2h:1v downstream, 2.5h:1v upstream
- Downstream embankment slope at closure = 3h:1v
- Minimum factors of safety for slope stability: End of construction, static = 1.3; Long term, static = 1.5; and Pseudo-static = 1.0.

18.4.4 Design and Construction

The overall design objective of the TMF is to protect the regional ground and surface water resources during both operations and mine post-closure, and to allow an effective closure of the facility. A staged dam construction using downstream raises and high-quality nPAG waste rockfill was selected. The upstream face of the embankment and the entire impoundment will be lined with a composite system consisting of a geomembrane and locally sourced low permeability soil. The TMF will additionally be used to seasonally store contact water as required and based on the length of each year's open-water season.

Embankment construction would be coordinated with mining operations so that waste rock is direct hauled from the pit to the dam. As there is anticipated to be a surplus of waste rock in the early years, the sequencing of the dam includes extending the downstream toe to the ultimate (Phase 4) location as part of the Phase 2 raise. This additionally creates an unusually robust dam in Phases 2 and 3

The Phase 1 design must be completed in the initial capital period. Phases 2, 3 and 4 would be completed during operations and regarded as sustaining capital expenditure. Phases 1 and 2 incorporate the use of a temporary water diversion structure along with the main water diversion structure to limit runoff water ingress

Emergency overflow water is designed to utilize the water diversion structure (see Section 18.7).

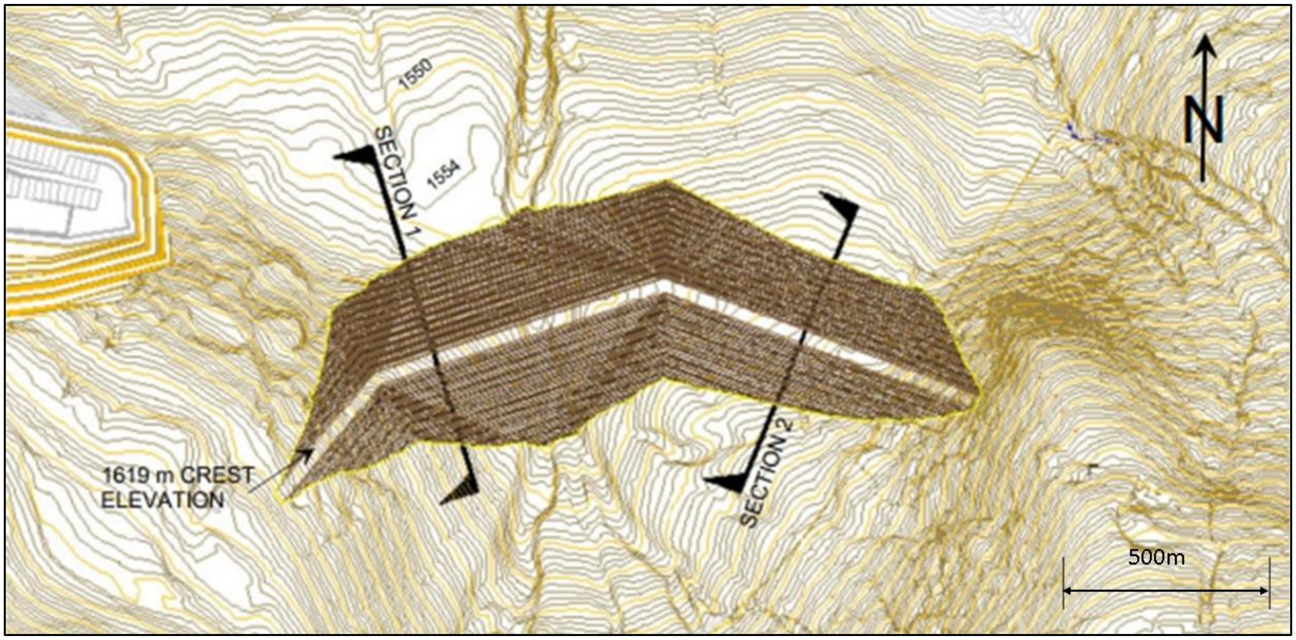


Figure 18-5: Plan of Ultimate TMF Configuration
(Source Piteau, 2021)

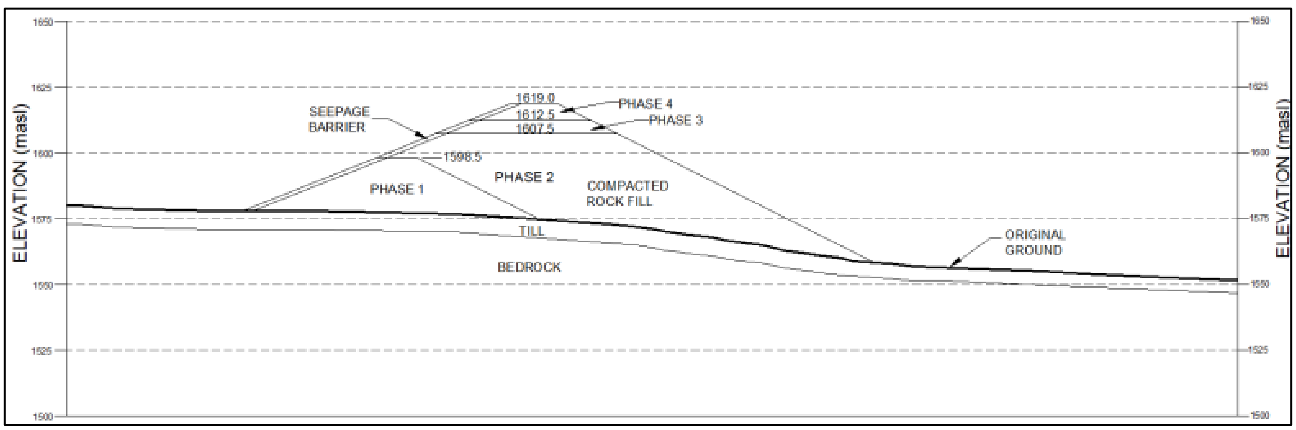


Figure 18-6: Cross section 1 of the design phases
(Source: Piteau, 2021)

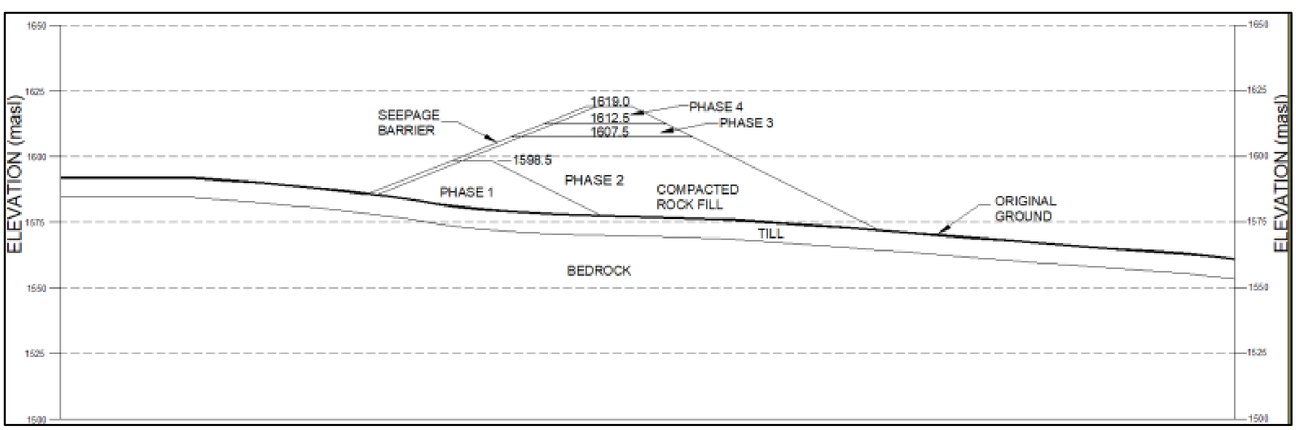


Figure 18-7: Cross section 2 of the Design Phases
(Source: Piteau, 2021)

The design process included utilization of existing data where appropriate but also included new data. The design process included but was not limited to the following:

- Completion of a site-specific Seismic Hazard Analysis (Anddes, 2021)
- Stability analysis utilizing guidance provided in the Canadian Dam Association (CDA) Dam Safety Guidelines – 20017 (20013 Edition)
- Tailings density and rheology testing (RDL 2021)
- Dam break analysis (Anddes 2021)
- Climate, hydrogeology and hydrological analysis combined with site water balance for use in estimation of water storage requirements, overflow and other water control measures (CSA Global, 2021)
- Soil and sub-soil test pitting was most recently completed by Ausenco in 2018 and recent drilling completed under the site supervision of Terrain Geoscience in 2021 in the vicinity of the TMF to get a better understanding of foundation conditions
- Bulk density, shear test, Brazilian tensile strength tests, unconfined compressive strength, permeability, liquid limit of soils, plastic limit and plasticity of soils, amongst others of core from foundation-oriented core drillholes (Golder, 2021)
- Bulk density, shear testing and other tests of rock from core identified for use in Phase 1 of the TMF bulk fill (Golder, 2021)
- Hydraulic conductivity testing of drillholes in the foundation vicinity (Terrane Geoscience 2021).

Mine operations planning includes the addition of a further rock buttress at 3H:1V slope and ramp accesses that will be revegetated as part of mine closure.

18.4.5 Operation of the Tailings Management Facility

Tailings would be discharged principally via spigots from the dam crest and at strategic points around the impoundment. The sequencing and location of the spigots are designed to be adjusted on a periodic basis to control the operating pool location to the centre or far southwest portion of the TMF. Open-pipe discharge would be used to prevent freezing during the winter months. The pool must be maintained at a sufficient distance from the exposed liner system to prevent ice damage to the liner.

Reclaim water would be pumped from the operating pool via a floating intake and delivered to the process plant via the contact water pond. Site contact water will be treated and discharged from the site, but only during open water season. The TMF will be used to seasonally store contact water for later treatment and discharge.

The anticipated TMF schedule of construction Table 18-2 and storage capacity are described in Table 18-3.

Table 18-2: Planned bulk nPAG rock supply from open pit for wall construction (Mt waste rock)

Year->	Total	-1	1	2	3	4	5	6	7	8	9	10
Phase 1	0.3	2.5	-	-	-	-	-	-	-	-	-	-
Phase 2	7.3	-	3.4	3.9	-	-	-	-	-	-	-	-
Phase 3	2.6	-	-	-	2.4	-	0.5	-	-	-	-	-
Phase 4	0.4	-	-	-	-	-	-	-	0.4	-	-	-
Total	**10.5**	**2.5**	**3.4**	**3.9**	**2.1**	**-**	**0.5**	**-**	**0.4**	**-**	**-**	**-**

Table 18-3: Planned tailing storage tonnages (Mt tailings)

Year->	Total	-1	1	2	3	4	5	6	7	8	9	10
Phase 1	3.3	0.2	1.2	1.3	0.6	-	-	-	-	-	-	-
Phase 2	3.8	-	-	-	0.7	1.2	1.3	0.6	-	-	-	-
Phase 3	2.5	-	-	-	-	-	0.5	0.7	1.3	0.5	-	-
Phase 4	3.5	-	-	-	-	-	-	-	-	0.8	1.3	1.3
Total	**13.0**	**0.2**	**1.2**	**1.3**	**1.3**	**1.2**	**1.3**	**1.3**	**1.3**	**1.3**	**1.3**	**1.3**

18.5 Waste Rock Dumps

Waste rock from the open pit and underground mines will be stored at four separate locations: the underground portal, the main waste rock dump (WRD), the in-pit dump, and the TMF wall. All dump access roads are designed with maximum 10% gradient and are to be 25 m-wide dual lane. Stockpiles slopes are constructed with an overall slope of 2.75H: 1V in 20 m maximum height lifts. The 20 m batter slopes are designed at a 35˚ angle of repose.

The foundation area of the WRD prior to dumping commences is planned to be cleared of vegetation. Topsoil and any unsuitable materials will be stripped and stockpiled for use in rehabilitation. Three main topsoil stockpiles are allowed for. Three small topsoil stockpiles will be required for external areas at the Esso ventilation raises, the laydown area and the explosives magazine.

The main WRD dump is designed to store both PAG and nPAG materials. The PAG WRD area is designed to be underlain by a truck-dumped layer of nPAG rock. The area is also closest to the CWP for best management of potentially affected waters.

As soon as practical in the mining schedule, Phase 1 of the open pit will be utilized as a backfill location for preferential long-term storage of nPAG rock. At the completion of phase 2 open pit mining, all remaining nPAG rock is planned to be rehandled to the open pit back-fill. The nPAG in-pit backfill will be overlain by a low permeability soil layer and finally capped by nPAG rock. At closure the in-pit WRD and the remaining external WRD will be land-formed, covered with a layer of topsoil, and revegetated

The ore sorter reject rock from the processing plant is expected to be PAG. This is planned to be used preferentially for underground stope backfill. Any excess is planned to be stored either in the WRD PAG section or the pit backfill.

A small waste rock dump (228 kt) of nPAG material is planned to be utilized as a platform for the mobile waste crusher that will be used for construction materials and waste rock if utilized in backfill.

Figure 18-9 shows the locations of the PAG waste (light green), nPAG waste (dark green), and topsoil stockpiles (dark brown). Figure 18-8 illustrates the quarterly cumulative development of stockpiles inclusive of tailings. Note that approximately 43 Mt of waste rock is placed in the in-pit storage area either directly or as re-handle.

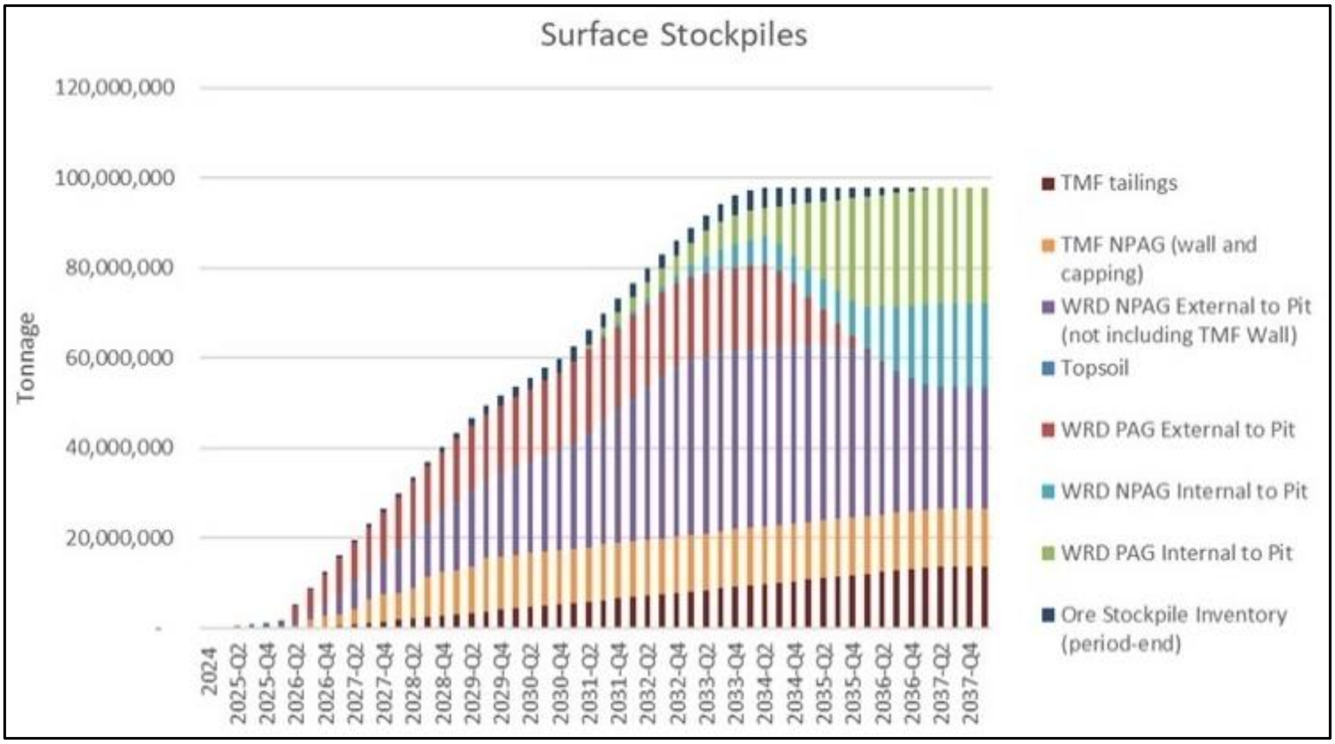


Figure 18-8: Stockpile Storage Capacity

18.6 Ore Stockpiles

Two ore stockpiles are required:

- A high-grade stockpile of around 250 kt is located near to the crusher. The high-grade stockpile will vary in capacity during the project life and the cut-off grade utilized to allocate ore to this stockpile this stock will be constantly varied according to mine plan and economics dictates as well as to ensure that the stockpile size is maintained. The stockpile pad comprising nPAG waste rock, will be graded to maintain water flow within the contact water control system.
- The low-grade stockpile pad is designed to contain up to 4.3 Mt of material. The stockpile will be segregated by oxide and sulphide ore to aid the metallurgical recovery and phased processing of these materials. The area is designed to be underlain by nPAG material and all drainage is directed to the contact water control system.

Figure 18-9 shows the locations of the high-grade stockpile (red) and the low-grade stockpile (orange) in year 2033 (Year 7) of the operation.

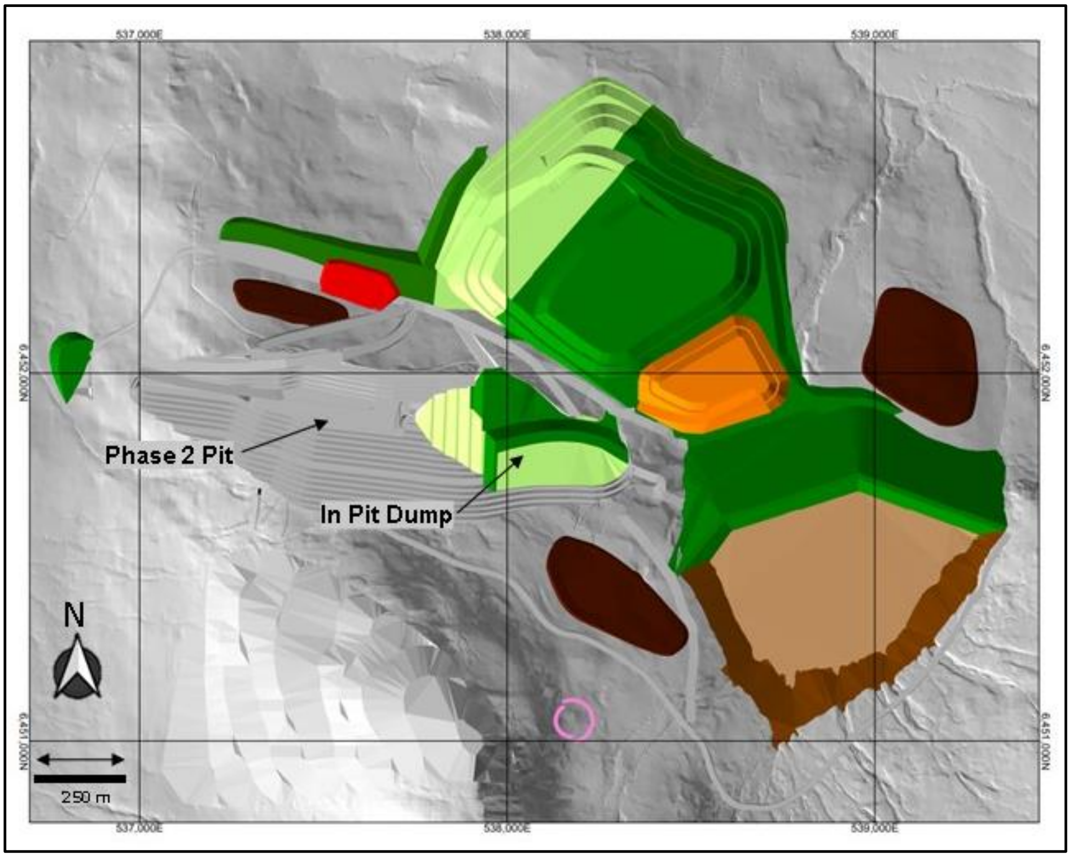


Figure 18-9: Plan Showing the Location of Various Stockpiles and Waste Dump Locations in Year 7 (2033)

18.7 Surface Water Management

The primary objective of the surface water management plan is to manage on-site surface water during the life of the mine and to maintain environmental and operational performance for the Project. The mine layout is designed to separate surface water into contact water and non-contact water catchments. The system captures, conveys, and treats all surface contact water within the active mine area (open pit, plant, waste rock dump and ore stockpile areas) which has come into contact with PAG rock. All non-contact surface water is designed to be collected and conveyed to a series of sedimentation ponds before being released to the receiving environment. The surface water management system is designed to limit surface water from entering the site by utilizing several diversion ditches and canals to divert surface water around the mine site and discharge to the downstream receiving environments. All contact water is designed to report to the contact water pond (CWP) and the TMF. Surplus water not required for use as process water is to be treated in the water treatment plant, sampled, and analyzed for compliance with a discharge permit prior to being pumped to the water discharge location.

CSA Global hired ERM to undertake water modelling, develop a water balance model and to determine the required sizing for the contact water pond. Allnorth designed the water management infrastructure including diversion canals, ditching, pipelines, the sedimentation ponds, and the CWP.

18.7.1 Design Criteria

Surface water runoff from each catchment area (Figure 18-10) was estimated using site-specific hydrology data developed by Lorax Environmental Services (Hydromet Metrics Kutcho Copper Project, 2021). Water management structures are designed to safely handle a 1 in 100-year storm event with climate change factors applied over the life of the project and snowmelt included.

18.7.2 Site Infrastructure

The critical water management structures described in this Technical Report are listed below:

- Contact Water Pond
- Contact Water Collection System
- Diversion Structures
- Non-Contact Water Collection System and Sediment Ponds
- Treated water discharge system and discharge outlet.

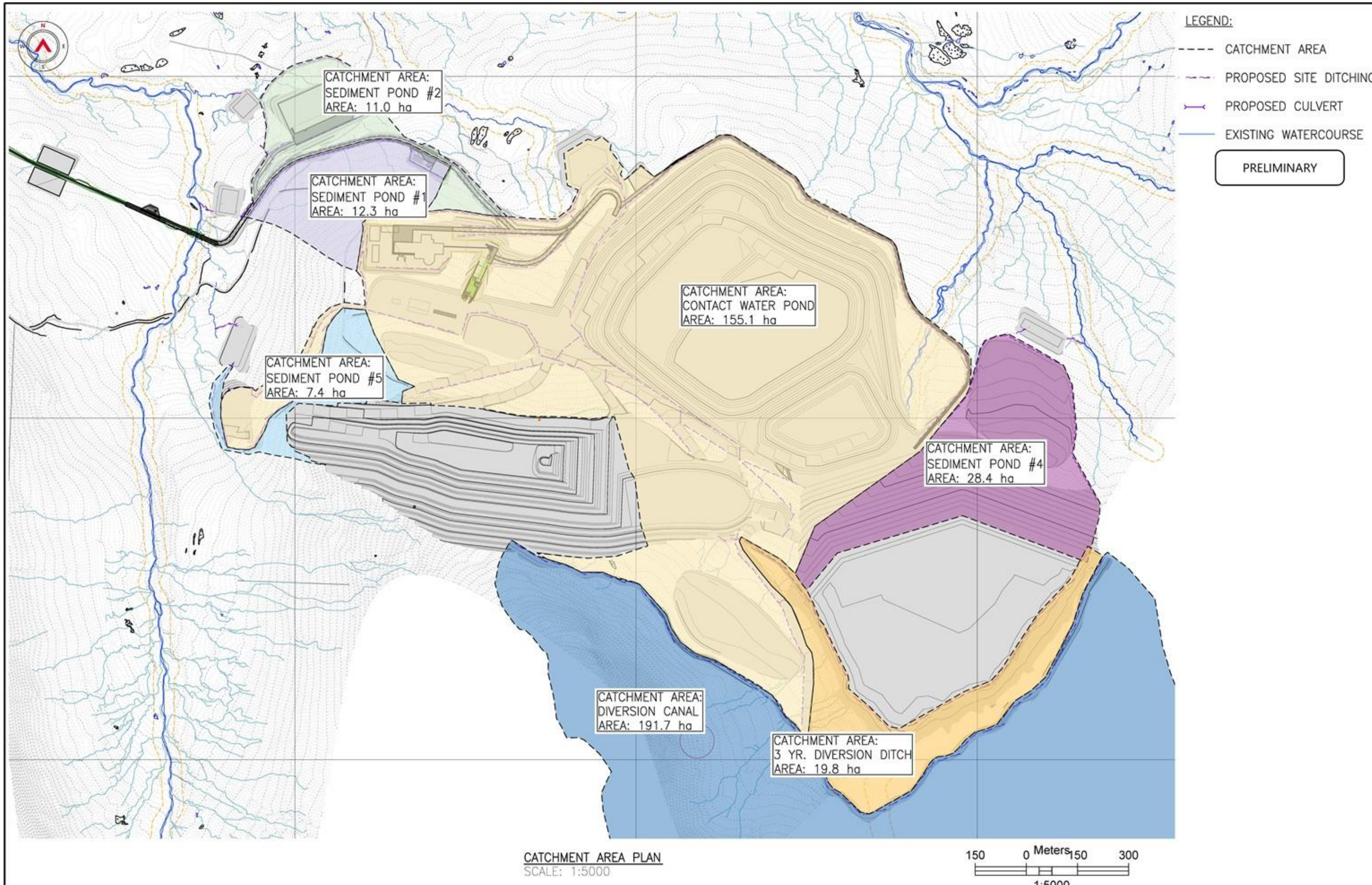


Figure 18-10: Mine Site Catchment Areas with Project Infrastructure

Source: Allnorth Consultants, 2021

18.7.2.1 Contact Water Pond

The designed contact water pond is located downslope of the proposed active mining area, which allows for gravity inflows of captured contact water. The proposed CWP is to have a capacity of 50,000 m^3 and be constructed using earth berms at 3H:1V side slopes in cut-and-fill sections. The basin and berms of the CWP will be lined with 1.5 mm thick, high density, polyethylene (HDPE), overlaying a 100 mm thick sand layer. The minimum top width of the CWP berm is to be 3.0 m with two small pad sections to allow for pump staging. The CWP is designed to be 7.0 m deep which allows for 1.0 m of sediment storage, 1.0 m of freeboard and 5.0 m of active storage.

18.7.2.2 Contact Water Collection System

The designed contact water collection system consists of three main ditches. The first ditch follows the toe of the WRD and collects surface water from the WRD and low-grade ore stockpile catchment areas. An access road is designed to follow the contact water ditch and provide the dual benefit of housing the pipe alignments running between the CWP and the TMF and providing maintenance for the collection ditch. A second ditch is designed to follows the north and west edges of the Process Plant pad and collects surface water from the crusher pad, warehouse pad and plant pads. The final designed contact ditch follows the main haul roads within the site and collects surface water from most of the active mine site, and culverts allow for transmission of contact water below roads. Site ditching was designed using riprap and rock drop-structures/minor check dams to control water velocity and reduce erosion.

The proposed mine portal pad and the open pit base are the only two areas within the active proposed mine site that are located lower than the CWP in elevation and will require contact water to be pumped. Surface water and ground water which collect in the proposed mine pit would be pumped north into the nearest contact water ditch. Surface water which collects on the portal pad and portal pad access road, would be directed to a sump where it could be pumped to the process plant pad contact water ditch.

18.7.2.3 Main Water Diversion Infrastructure

The main diversion canal is designed to be constructed to divert surface water around the TMF to limit the amount non-contact water that enters the system. The main diversion canal is designed to share the TMF's overflow structure to discharge freshwater to Caribou Creek. The main diversion canal is to include a 3.5 m wide access road, primarily for construction and maintenance. The road would contain a protection berm on the downhill side. The canal is to have side slopes of 2H:1V, base of 2 m and a height of 1.5 m. The diversion canal is designed to be cut into the hillside. Non-woven geotextile and riprap are designed to be used to control water velocity and erosion of the canal. Once the main diversion canal is constructed, a second smaller ditch is to be constructed within the main diversion canal. The diversion ditch is designed to prevent stormwater from entering the TMF for the first three years of the mine life but will eventually be flooded as the TMF water level increases after Year 3. The smaller diversion ditch is to be constructed with a 3.0 m wide access road and protection berm on the downhill side. The diversion ditch is designed to have side slopes of 2H:1V, 0.6 m wide at the ditch base and a height typically of 1.0 m. No geotextile or riprap is required for this structure.

18.7.2.4 Non-Contact Water Collection System and Sediment Ponds

Four sediment ponds and collection ditches are proposed to collect non-contact water prior to release to the receiving environments. All sediment ponds are sized using "Method A" from MOE (Technical Guidance 7, 2015). As per Technical Guidance 7, the sediment ponds are designed to capture 10 micron soil particles for the 10-year, 24-hour runoff event. Sediment ponds will be constructed with 3H:1V earth berms in cut-and-fill sections with a minimum berm top width of 3.0 m. All ponds are designed to allow for 1.0 m of freeboard, 1.5 m active water storage, and 1.0 m of sediment storage volume. Three of the sediment ponds are designed to have a 3.0 m tall HDPE baffle placed lengthwise down the centre of the pond to increase the effective length of the ponds.

The pond locations and structures are outlined below:

- The first sediment pond, with a volume of 4,336 m^3, is located east of the gate house pad and collects surface water from the internal roadside ditches the liquid natural gas pad, and a catchment of undisturbed ground which drains into the roadside ditches. The proposed sediment pond will be approximately 40 m wide by 60 m in length at the base and will be constructed with an HDPE baffle to increase its effective length.
- The second sediment pond with a volume of 3,378 m^3, is located northwest of the accommodation complex and is to collect surface water from the accommodation complex pad, sewage treatment plant pad and adjoining access road. The stormwater ditching is designed to minimize freshwater entering the sediment pond. This sediment pond is designed to be approximately 35 m wide by 52 m long and will be constructed with an HDPE baffle to increase its effective length.
- The third sediment pond with a volume of 5,672 m^3 s located north of the east topsoil stockpile area which collects surface runoff from the face of the TMF and the topsoil stockpile. Surface water is to be collected by a ditch which follows the toe of the topsoil stockpile. The sediment pond is designed to be approximately 25 m wide and 124 m long at the base.
- The fourth sediment pond with a volume of 4,383 m^3 is located north of the portal pad. Surface water from a catchment above the portal pad is to be captured by a diversion ditch and directed to the sediment pond. The sediment pond is designed to be 21 m wide and 105 m long.

18.8 Power Generation and Electrical Distribution

Electrical power is designed to be generated on-site by five modular LNG generator units In a N + 1 configuration, where N is the number of units required to meet the maximum power demand at any time (i.e., four). Each generator is capable of 10% extra power generation to cover momentary surges in power demand. Backup emergency power is provided by a 2 MW diesel generator.

The LNG generator power plant will feed power to a 13.8 kV metal-clad switchgear located inside an electrical room in the process plant. The 13.8 kV switch gear will distribute the power throughout the site using overhead power lines.

Annual power demand is expected to be 68,943 MWhr though this will vary during operations.

18.9 Communications

The mine site communication system is designed to be linked to the nearest accessible internet provider using satellite receivers supporting both data and voice communications. An internet protocol (IP) phone system and wireless network is designed to be available throughout the mine site. A 2-way radio system will provide the infrastructure to enable handheld and mobile radio sets to communicate around the site.

18.10 Site Earthworks

Overburden depth varies from 0 to 3.7 m below existing grade with a site wide average overburden depth of 0.95 m as found in the Piteau Geotechnical Report (Piteau, 2021). Stripping depth for each development area was interpolated based on the available borehole and test pit logs found in the AMEC Geotechnical report (AMEC, 2008). For areas with no known boreholes or test pits, an average overburden stripping depth of 0.95 m was assumed. Site pads and roads are generally designed to be constructed in both cut and fill to minimize material movement throughout the site. All pads are to have a minimum grade of 1% and direct water away from critical infrastructure to stormwater swales and ditches. All pads are designed to be constructed with 450 mm of granular subbase material. High traffic areas are to include an additional 150 mm surface crush. Rock cuts are designed to be constructed as per MOTI Technical Bulletin GM02001. Bulk fills are designed to be compacted to

95% Modified Proctor Dry Density (MPDD), while granular sub-base and surfacing crush are to be compacted to 98% MPDD.

18.10.1 ***Plant***

The plant pad is designed to be primarily located in cut. Stripping depth varies between 0.2 m and 0.6 m with fractured bedrock estimated at 2.6 m below grade. The pad is 250 m long and 126 m wide and will be graded towards its perimeter at 1% to ensure surface water drainage.

18.10.2 ***Crusher and Ore Sorter Pad***

The crusher and ore sorter pad is located directly east of the plant and houses the crusher, ore sorters, conveyors, and two storage bins, one for sorted reject rock and the other for ore. The crusher tipping wall will be constructed using a mechanically stabilized earth (MSE) wall. The crushing pad has a designated access road, which provides access to the storage bins. Stripping depth at the crushing pad is estimated to be between 0.3 m and 0.6 m.

18.10.3 ***Fuel Farm***

The fuel farm is to be located primarily on fill. The stripping depth is estimated to be 0.2 m. The fuel farm pad is to be 50 m wide and 50 m long. Surface water is designed to be directed to the north and east perimeters of the pad and directed to the contact water system.

18.10.4 ***Power Generation and Distribution***

The power generation and distribution pad is designed primarily in cut. The stripping depth for the pad is estimated to be 0.3 m, and fractured bedrock is estimated to be 2.7 m below existing grade. The designed pad is approximately 120 m long by 40 m wide. It has ditches along the south perimeter to direct surface water off the pad.

18.10.5 ***Accommodation Complex***

Approximate stripping depth for the accommodation complex pad and its access road is 0.3 m below existing grade. The pad is 213 m long and 133 m wide. The wastewater treatment plant is designed to be located adjacent to the accommodation complex pad on a 37 m by 55 m pad extension. Ditches are designed along the south and east perimeter of the pad to collect and direct surface water to a sediment pond.

18.10.6 ***Laydown***

A laydown area is to be constructed just outside of the main mine site adjacent to the main access road. It is designed to be constructed with a balanced cut-and-fill. The stripping depth for the laydown area is estimated to be 0.95 m. The laydown area is 120 m wide and 100 m long. Ditches are designed to collect surface water on the south edge of the laydown pad and to direct it to the main site access road ditches.

18.10.7 ***Vent Raise Pad***

The vent raise pad is designed to be primarily in a cut section. The stripping depth is approximately 0.3 m. Stormwater ditches are designed to collect surface water along the south edge of the pad and direct it to the Vent raise access road surface water ditches. The ventilation raise pad design is presented in Figure 18-11.

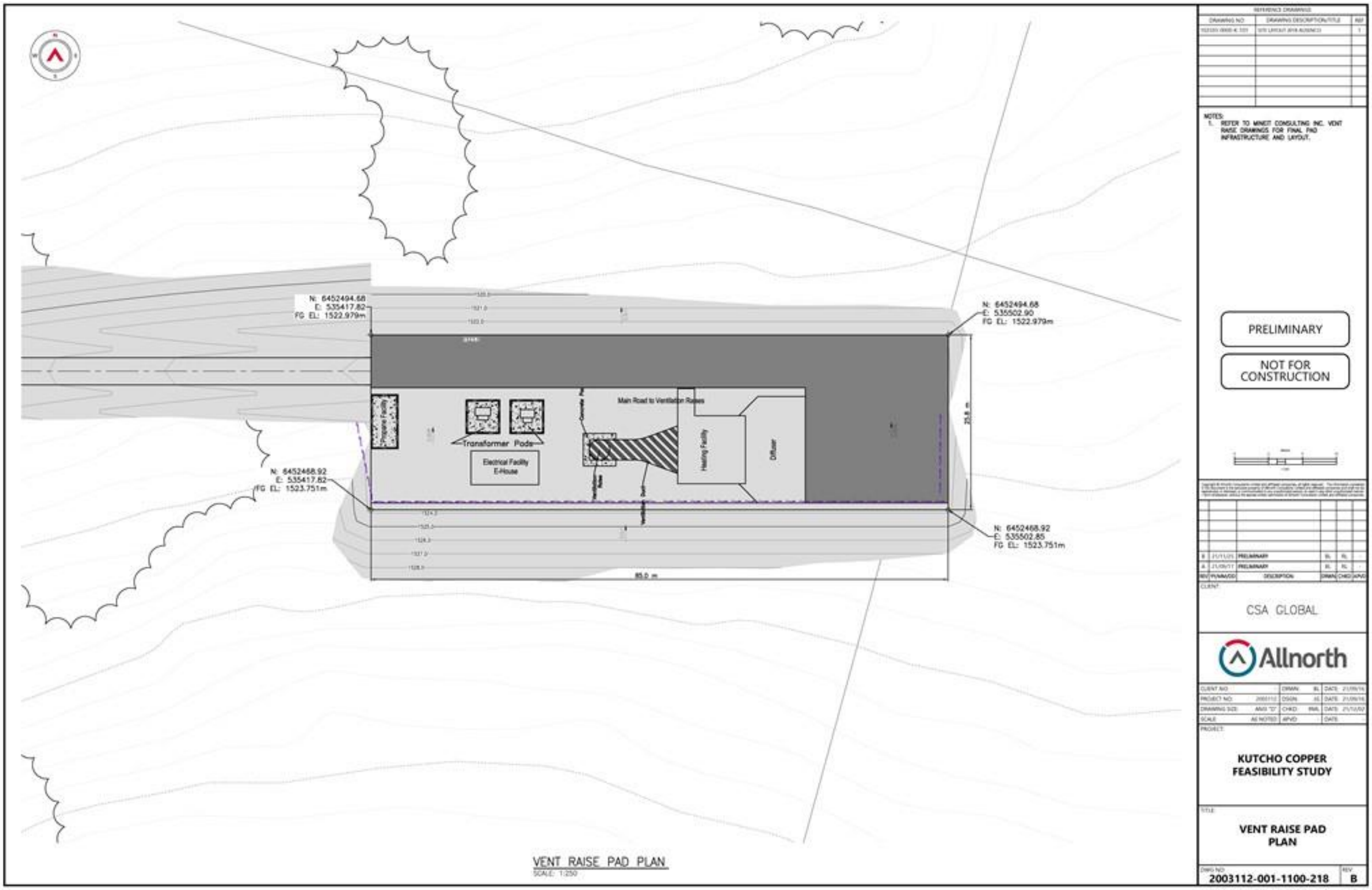


Figure 18-11: Vent Raise Pad Design

18.10.8 Bulk Explosives and Detonator Magazines Pad

The bulk explosives magazine pad is located primarily in cut. The stripping depth is assumed to be 0.95 m at this location as there were no boreholes or test pits at this location. The bulk explosives and detonator magazines pad design is presented in Figure 18-12.

18.10.9 Treated Water Discharge Pad

The stripping depth at the treated water discharge dad is estimated to be 0.95 m. The treated water discharge pad is approximately 50 m x 50 m.

18.11 Buildings and Layout

The overall site plan (see Figure 18-1) shows the major Project facilities, including the open pit mine, underground portal, TMF, WRD, and crushing and processing facilities. Infrastructure to support the Project is to consist mainly of facility pads, roads, contact and non-contact water management infrastructure, administration buildings, accommodation complex, maintenance facilities, a water supply system, a water treatment and discharge system, and site electrical generation and distribution.

The crushing and process facilities includes the administration offices, assay laboratory, warehouse, reagent storage, and miscellaneous facilities.

The Project is designed to be serviced with potable water extracted from wells, potable water treatment facility, fire water, on-site power generation, compressed air, and communications.

The camp facility includes a change room and canteen.

18.11.1 Crusher Building

The Crusher building is designed as a pre-engineering steel building measuring 60 m long x 28 m wide. The building is designed to be fitted with a 10-tonne overhead crane for construction and maintenance activities. The building will be fabricated using insulated metal panels (IMPs) for the roofing and siding systems and include roll-up equipment doors, and man doors for access and egress of vehicles and personnel. Concrete footings and slab on grade will be founded on suitable native grade or compacted granular fill.

A control room for the operation of the jaw crusher will include plant process cameras.

The Crusher building will have a liquid natural gas fired heating and ventilation system.

See Figure 18-13 for plan and elevation views of the Crusher building.

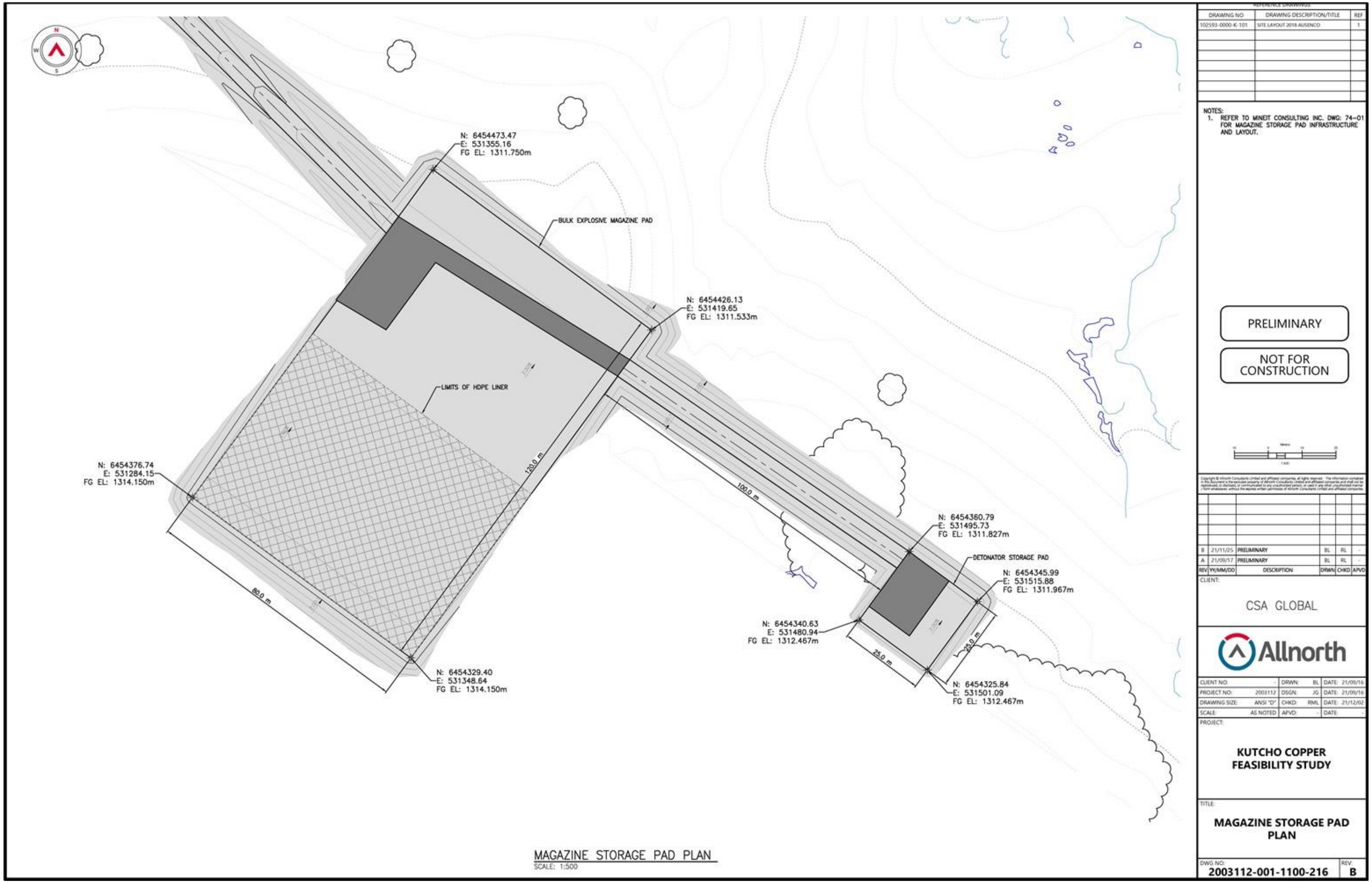


Figure 18-12: Bulk Explosives and Detonator Magazines Pad

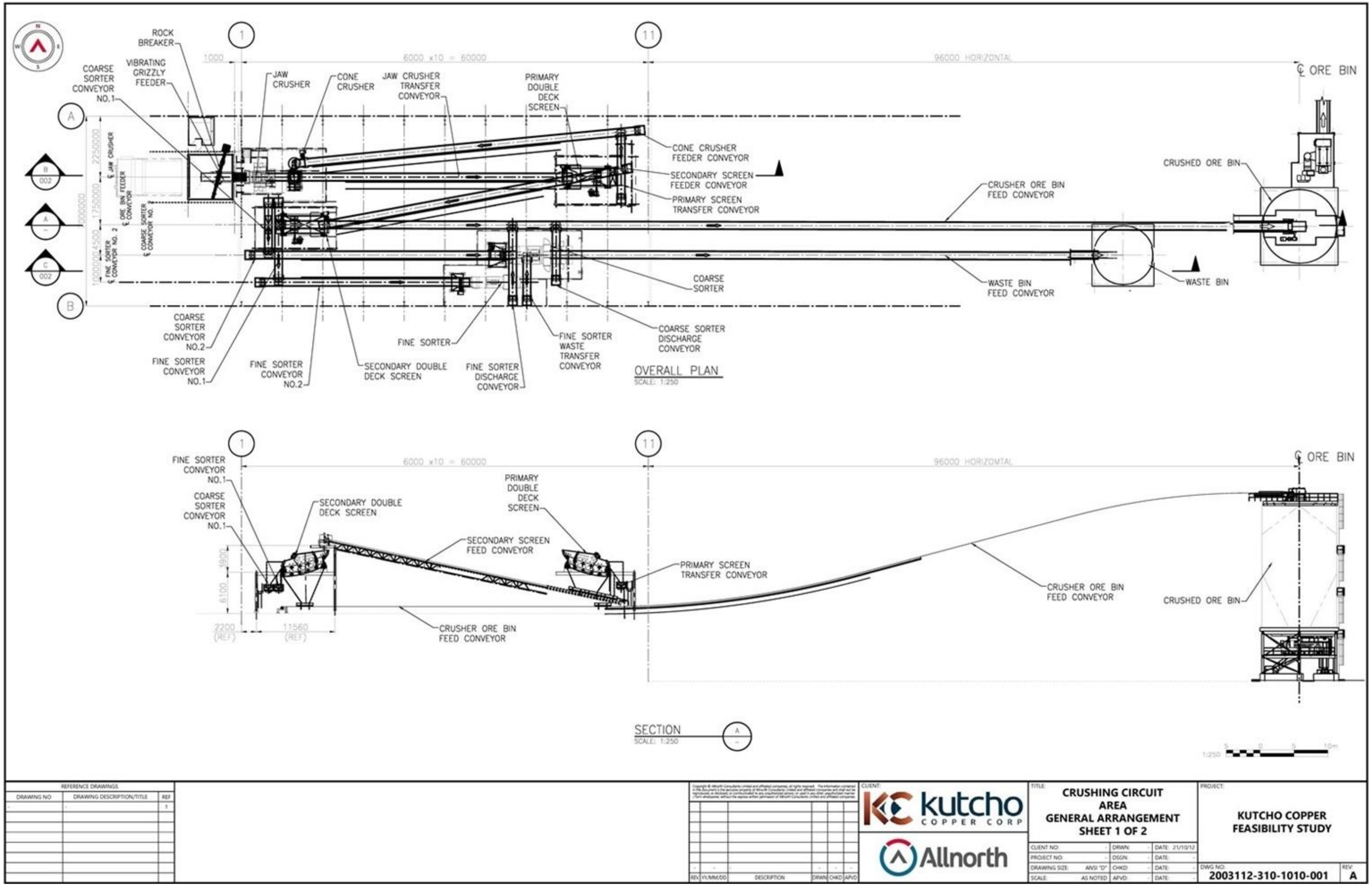


Figure 18-13: Crusher Building Plan and Elevation View

18.11.2 Process Plant Building

The Process Plant building is designed as a pre-engineered steel building measuring 132 m long x 31.5 m wide. A pre-engineered steel building measuring 30 m long x 24 m wide adjacent to the Process plant is planned to be utilized for reagent storage. The buildings are to be fabricated using insulated metal panels (IMPs) for the roofing and siding systems and include roll-up equipment doors, and man doors for access and egress of vehicles and personnel. Intermediate elevated platforms with steel grating will provide access to elevated equipment.

The Process Plant building includes a 50-tonne overhead crane for construction and maintenance activities. A 5-tonne overhead crane will be situated in the flotation area.

The copper, zinc, and tailings thickeners are located outside and adjacent to the Process Plant. The design includes removable insulated metal panels on the base of the thickeners for weather protection. A fabric covered roof structure over the zinc and copper concentrate thickeners is included in the design to retain heat. An external tailings pump enclosure is located adjacent to the tailings thickener.

Concrete footings and slab on grade are designed to be founded on suitable native grade or compacted granular fill. The SAG and Ball mill foundations are estimated to have 1,700 m^3 and 1,190 m^3 of concrete for each of their respective foundations.

A control room is provided for in the crushing and grinding area.

The Process Plant building is designed to include a liquid natural gas fired heating and ventilation system.

See Figure 18-14 for plan view of the Process Plant building.

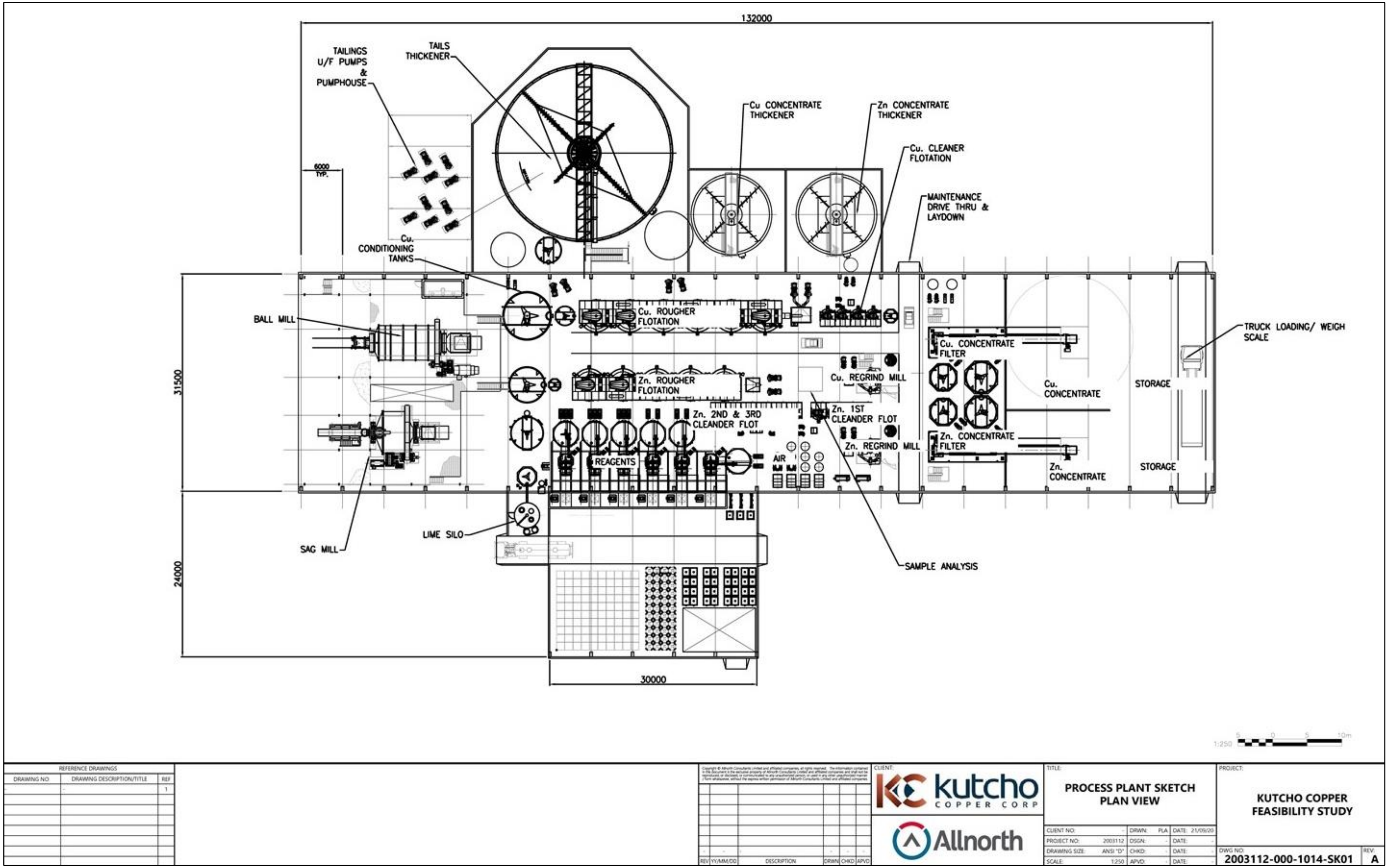


Figure 18-14: Process Plant Plan View

18.11.3 *Administration, Open Pit, Plant, and Maintenance Offices*

A 14-unit single story modular office complex is designed to be located on the process plant pad to house the administration, mining, plant, and maintenance personnel. The facility is designed to have 35 offices (single and double occupancy), a reception area, two meeting rooms, an IT/communications room, printing area, kitchen area, and a bull pen.

The facility is planned to be used both during construction and operations.

18.11.4 *Assay Lab*

The modular assay laboratory is designed to be located adjacent to the process plant and will consist of four modular buildings:

- A 10’ x 40’ assay building
- A 10’ x 32’ metallurgical preparation building
- A 10’ x 24’ firing building
- A 10’ x 16’ fire wash building.

The buildings are planned to be arranged in a U-shape with the central area used for sample drop-off.

18.11.5 *Warehouse*

The warehouse building serves mining and plant requirements and will be located on the process plant pad. The warehouse building has dimensions of 30 m wide x 44 m long x 11 m high at the sides. The building is to be steel framed and fabric covered system and includes a concrete floor. The building includes racking shelves, heating, lighting, and communication services. An external graded area accommodates large and bulk items. Reagents are planned to be stored in designated areas in the process plant.

18.11.6 *Truck Shop and Wash Bay*

The truck shop is a 40 m wide x 42 m long x 16 m high, steel frame, insulated fabric covered building. It is designed to have six overhead doors, three at each end of the building, for vehicle access to the six bays. Each overhead door is planned to be 8.5 m wide x 7 m high to allow for mining fleet and support equipment access. The six-bay truck shop is to provide sufficient area and facilities for the maintenance of the open pit and underground mining and ancillary fleets.

The truck shop design includes a compacted gravel floor with a 1.5 mm HDPE liner beneath it to contain any hydrocarbon spills. No overhead crane was allowed for, and any lifting will be carried out using the on-site mobile service cranes, either inside the building or outside.

The wash bay is adjacent to the truck shop. It is planned to be a 10 m wide x 15 m long x 11 m high steel frame, insulated fabric covered building. It is designed to have two overhead doors, one at each end of the building, for drive through vehicle access. Each overhead door is to be 8.5 m wide x 7 m high to allow for mining fleet and support equipment access. The wash bay is designed to have a concrete floor and a main water collection sump to capture wash water.

18.11.7 *Mine Dry and Underground Mining Offices*

The mine dry and underground mine offices are designed to be located at the accommodation complex and connected to it by arctic corridors. The mine dry and underground mining offices are to consist of modular units fabricated off site and connected in a single storey configuration. The mine dry design includes separate clean and dirty side facilities for 20 women and 142 men, hanging baskets, lockers, individual showers, and washroom facilities.

The underground mine offices are designed to be connected to the mine dry. The underground mining offices consist of modular units fabricated off site and connected in a single-floor configuration. The facility includes ten offices, a boardroom, IT and communications room, a lamp charging and repair room, shift change meeting room, and a general assembly/line up area.

The mine dry and underground mining offices are planned be used both during construction and operations.

18.11.8 Accommodations Complex and Associated Buildings

The accommodations complex (the camp) is designed to be located north of the Main open pit mining area. The camp is planned to consist of modular units fabricated off site and connected in a single storey configuration. The camp is planned to consist of:

- Sleeping facilities for up to 348 people
- Kitchen and dining complex
- Recreational facilities including games, television, and fitness areas
- Sewage treatment facility and solid waste incinerator
- Potable water storage and treatment facility
- Mine dry and offices for construction and operations
- Parking area.

The design includes four 49-room modules and four 38-room modules for a total capacity of 348 people. All modules are to be installed at the start of construction to allow for peak camp loading. Each room in the 49-room module is designed to include a bed, desk, and closet, with washrooms provided in a wash car unit in each module. Each room in the 38-room modules is designed to include a bed, desk, closet, and private three-piece bathroom. The camp is planned to include a kitchen/dinning/prep building, a recreation building, the underground mining office/line up building, the mine dry building, and interconnecting arctic corridors. After construction is completed, it is planned that two of the 49-room modules will be removed from service and sold. The remaining 250 rooms is to accommodate operations staff and for short-term workforce increases during TMF dam raises, shutdown occurrences and other visitors.

Potable water is designed to be supplied to the camp via a well located within 100 m of the camp. The well design is based on a groundwater depth ranging from slightly artesian to 4.88 m below surface (Piteau Associates, 2021). A package water treatment plant is planned to treat the water to potable standards.

After treatment the camp pad groundwater will be stored in individual firewater and potable water tanks, each with a volume of 63.5 m^3. The firewater tank is planned to come equipped with a fire water skid containing a 46 HP pump. The potable water tank has a pump with a 15 HP motor capacity. Potable and fire water pipe mains are to be DR17 high-density polyethylene.

Sewage is planned to be collected from the individual camp buildings via a gravity system. Lift stations are to transport sewage to the treatment plant located on the southeast corner of the camp pad. Grey water from the kitchen is to flow through a grease trap prior to entering the general sewage collection system. Sewage waste is planned to be treated to permitted discharge standards and then pumped to the contact water pond.

Natural gas to power forced air furnaces and hot water tanks is designed to be fed by a 50 mm medium density polyethylene pipes from the main LNG storage area. It is planned during the operational phase for electricity to the camp and associated infrastructure to come from the LNG generators and power distribution system. The camp is designed to receive back up power from the 2 MW diesel genset. During construction, power is planned to be supplied by rented stand-alone diesel gensets.

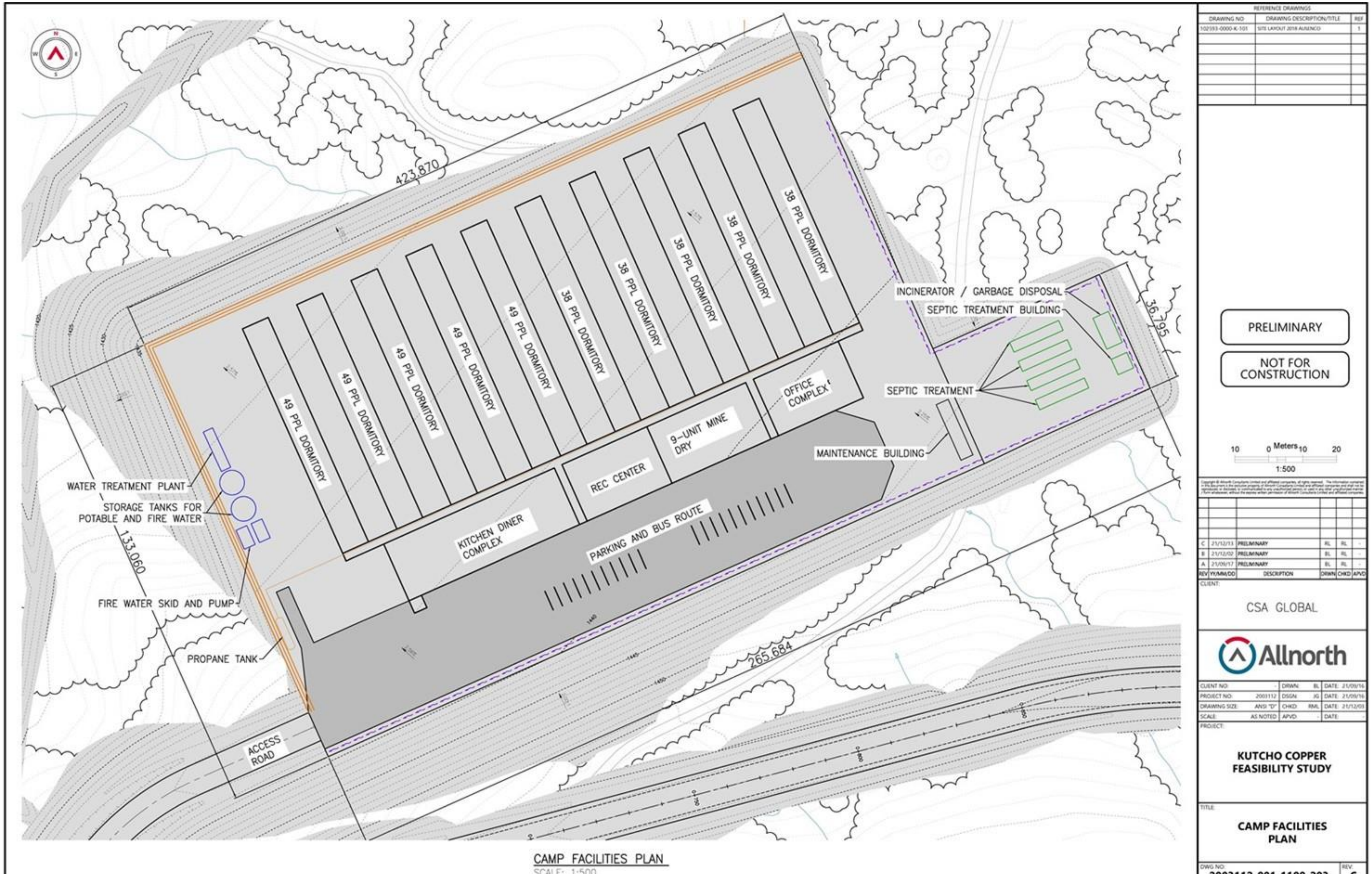


Figure 18-15: Plan View of Camp and Offices
(Source: Allnorth Consultants, 2021)

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

Vehicle access is designed to be provided to the camp pad via a two-lane road connected to the main access road. Light vehicle parking spaces, with electrical outlets, and a bus loading/unloading area are planned to be located at the front of the camp area. Camp pad roads are designed to accommodate highway charter buses and semi-trailers.

18.11.9 ***Diesel Fuel***

Diesel to supply fuel to light vehicles, the mining fleet and mobile plant and equipment is designed to be stored at the fuel farm. The fuel farm is designed to have two 100,000 L double-walled horizontal storage tanks, each with receiving/dispensing capabilities, an oils/coolants/grease dispensing module, and a diesel exhaust fluid system. The fuel and diesel exhaust fluid facilities are to be located within a HDPE lined containment area, and the fuel farm is designed to allow for filling of two mining vehicles simultaneously. The storage tank capacity is designed to provide approximately seven days of fuel usage during the first year of operation and 10–14 days storage for the remainder of operations.

Fuel will be dispensed by means of electric pumps with low- and high-flow capabilities (100 and 400 litres per minute, respectively) and an automatic shut-off dispensing nozzle. All fuel dispensed will be monitored with flow meters showing both a total and event values.

All fuel required at the mine site is designed to be delivered in B-train tanker trucks by commercial suppliers. Minor quantities of gasoline are to be stored in a specific tank for this purpose.

18.11.10 ***Liquified Natural Gas System***

The on-site storage and vaporization facility will be located on a pad adjacent to the power generators. The liquid natural gas system uses an N+1 approach for all critical gas supply equipment. The system, including instrumentation, is designed to withstand temperatures up to -40°C. The control room is to be housed in an insulated enclosure. The vaporization system is skid mounted and rated for outdoor climate. The system is to be unmanned on a continuous basis. The primary vaporization mode includes the use of waste heat from the natural gas generators. In the event there is a failure of the waste heat system, a fully redundant gas-fired vaporizer will be in hot standby mode. This is to ensure that in the event of a system trip, the gas fired vaporizer can seamlessly continue vaporization of natural gas.

Liquified natural gas is planned to be supplied by specially built B-train units and the on-site system includes five-day on-site storage.

18.11.11 ***Sewage and Waste***

Sewage from the camp and the other mine site buildings is to be treated in a sewage treatment plant to be located next to the camp pad. Treated effluent is to be pumped to the CWP.

The sewage treatment system is designed to use a moving bed bioreactor system and ultraviolet disinfection to treat the waste. Flow equalization tanks will manage variations in sewage flow rates. The sewage system is sized to handle the construction camp population (348 persons) and thus will allow for extended treatment times during operations when the camp complement is reduced.

Food waste, general office waste, cardboard packaging, and wood waste is to be collected on site and burned in incinerators to be located on the same pad as the sewage treatment system. The incinerators are designed to be housed in a non-insulated metal building containing an animal-proof metal waste storage bin. Two incinerators are to be initially installed to meet construction peak demands and one incinerator is to be removed after construction is completed. The incinerators are planned to be dual-chamber, batch-operation systems.

Incinerator ash will be collected in sealed bins and transported off site for disposal at an approved facility.

A waste stream segregation program is planned to be used to separate inert and recyclable wastes that cannot be incinerated (metals, rubber, and plastics as an example) to allow for transport off site or for disposal in an onsite landfill.

Hazardous and hydrocarbon wastes are planned to be collected and transported off site for appropriate reprocessing and/or disposal.

*18.11.12 **Gatehouse***

The gatehouse will be located on the access road west of the main mine site. The gatehouse is to be a modular trailer with an office, washroom, lunchroom, and communications. The gatehouse is planned to be the first point of control for vehicles entering and the checkout point for vehicles leaving the mine site.

The 50-tonne capacity concentrate truck scale, 3.1 m x 18 m, is planned to be located at the gatehouse and will have an automatic data collection system. This truck scale is to be the official recording point of concentrate truck outbound weight.

*18.11.13 **Water Treatment and Discharge***

18.11.13.1 General

Surface, underground, contact, treated sewage, and process plant water are to be treated by a single water treatment plant. For the six ice free months of the year, May to October, the plant is designed to treat average and maximum flow rates of 249 L/s and a 320 L/s, respectively. Approximately 112 L/s of treated water will be returned to the process plant and the remainder discharged to the environment. For the six frozen months of the year, November to April, the plant is designed to treat water from the tailings thickener overflow and return the treated water back to the processing facilities for re-use.

18.11.13.2 Process

The treatment system process steps are:

- Metals removal using ferric sulphate and alkali in reactor tanks
- Clarification to remove suspended solids
- Sludge recirculation to the metals removal stage
- Sludge pumping to the tailings thickener
- Polishing filtration
- Neutralization
- Treated effluent distribution.

18.11.13.3 Discharge

Treated effluent is planned to be discharged during ice-free months into Kutcho Creek just north of the confluence with Andrea Creek, a distance of 12 km from the mine site. Treated effluent is to flow through a 400 mm diameter HDPE DR 11 line and discharged through a spray header over Kutcho Creek. The spray header will be supported by two concrete posts, one on either side of the creek, located outside of the highwater zone. A pump will fill the pipeline with treated effluent though it is designed for gravity flow. The line will be emptied prior to predicted freezing climate to prevent damage.

The discharge line will run on surface adjacent to the access road and then along a service road to Kutcho Creek. Animal crossing points comprising soil mounded over the discharge line will be 50 m in extent every 200 m. The final 2,500 m stretch prior to discharge at Kutcho Creek will be mounded over entirely as the line intersects an animal travel corridor.

18.11.13.4 Location and Building

The water treatment plant is to be located on the process plant pad adjacent to the concentrate load-out area. A 1,000 m^2 tensioned-fabric building is designed to house the water treatment plant equipment and reagents. The building's concrete footings and slab on grade are to be founded on suitable native grade or compacted granular fill. The slab on grade floor will have sumps for spill and washdown management so that no untreated effluent escapes.

18.11.14 Vent Raises

The ventilation raise pad is designed to be accessed via a 1.3 km long single-lane, gated access road connected to the main site access road approximately 650 m northwest of the gatehouse. The ventilation raise pad is designed to be approximately 85 m long x 26 m wide and accommodates the ventilation raise, ducting, power generation and distribution equipment, and mine air heating facilities. The ventilation raise pad includes a turnaround to accommodate a cement truck or equivalent sized vehicle for construction and routine maintenance.

A pullout located west of the ventilation raise pad will house the underground mine exhaust vent. The access road is expected to be low-traffic and so designed as a single lane.

18.11.14.1 Vent Raise Facilities

Vent raise facilities include the ventilation duct, ventilation fan, heating facility, air intake/diffuser, and power generation and distribution equipment. The mine ventilation system was designed to provide 100 cfm of fresh air for each hp of diesel-powered equipment operating underground. For active workings, this equates to a minimum requirement of 18.9 m^3/s (40 kcfm) of fresh air to be provided. Additional criteria including maximum wet bulb temperature (28°C), minimum intake dry bulb temperature (3°C), and maximum air velocity (3 m/s). The ventilation system requirements informed the fan selection and facilities design process.

The fans selected for ventilation are summarized in Table 18-4 while additional details can be found in Section 16.4.3.

Table 18-4: Fan selections for underground mine ventilation

Fan selection	Quantity	Brand	Model	Series	Stage	Combined kW
Main exhaust	2	Howden	Vane Axial Fan	3150	Single stage	285
Esso vent raise	2	Howden	Vane Axial Fan	3150	Single stage	295
Connection drift	2	Howden	Vane Axial Fan	2700	Two stage	240

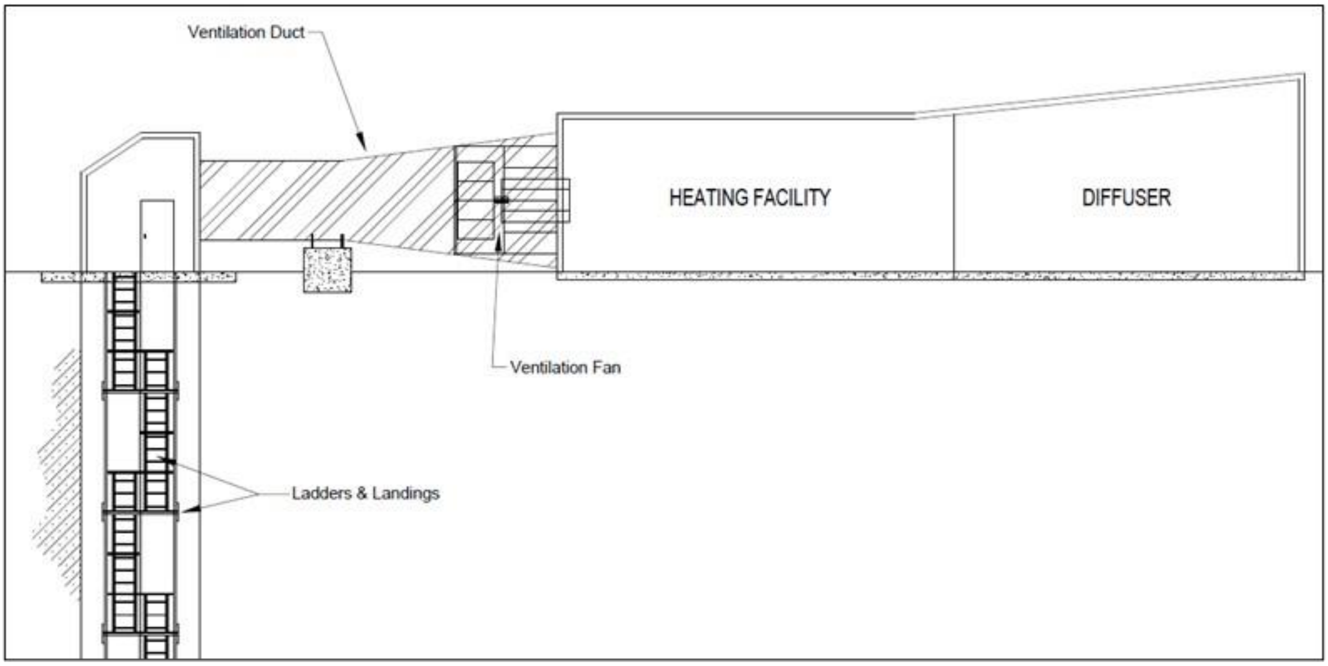


Figure 18-16: Vent Raise Design Long Section – Not for Construction

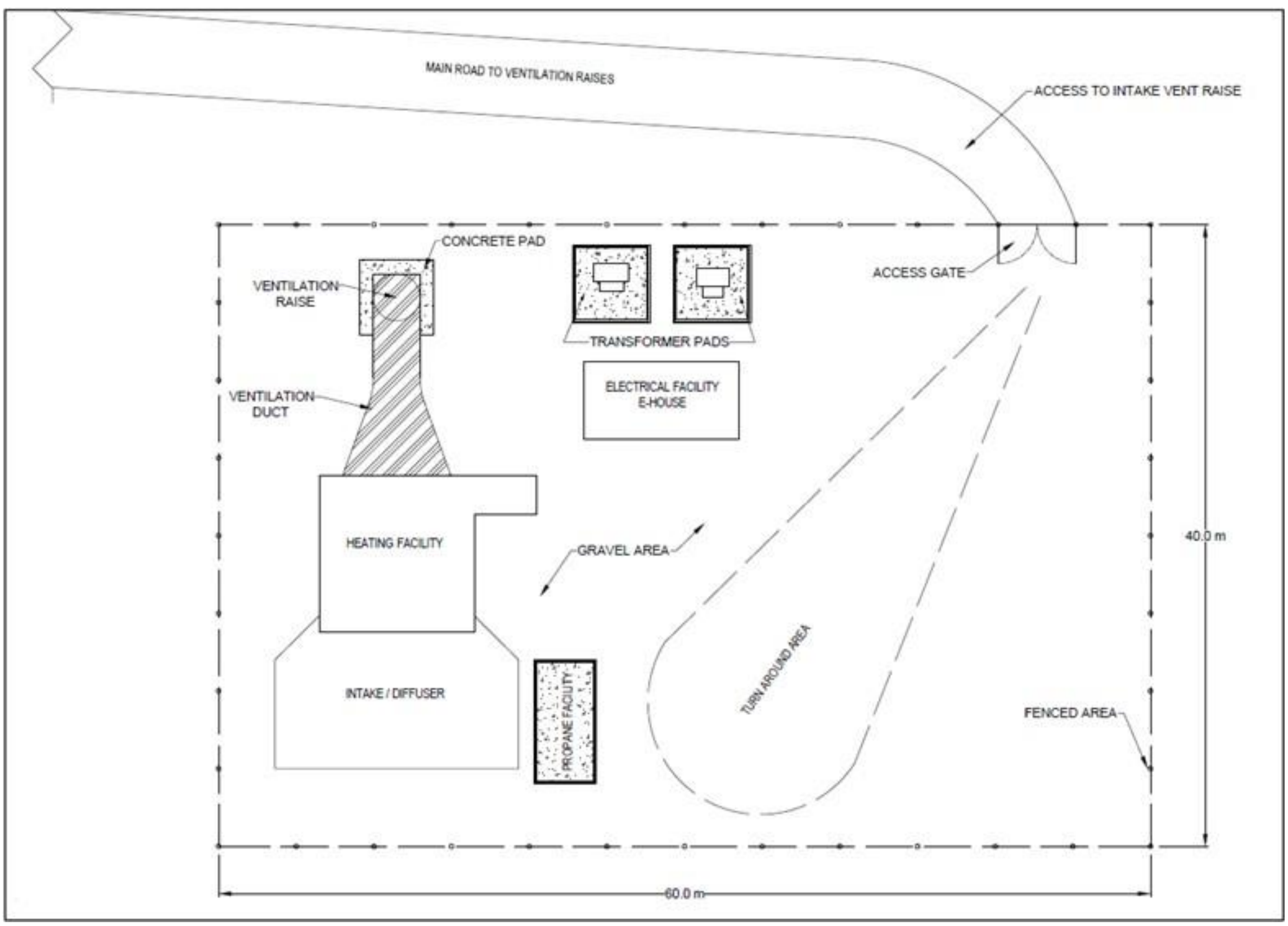


Figure 18-17: Vent Raise Pad Plan View – Not for Construction

*18.11.15 **Bulk Explosives and Detonator Magazines***

The bulk explosives magazine and detonator magazine pads are designed to be accessed via a 554 m long single-lane access road located off the main site access road, approximately 6.3 km west of the gatehouse. The proposed location of the facilities adheres to the required separation distance (as per National Standard of Canada CAN/BNQ 2910-510/2015) to the main access road and ventilation raises, at 375 m and 2,080 m respectively. See Figure 18-18.

The bulk explosives magazine pad is 80 m wide x 120 m long. It is to house the bulk explosives silos, packaged explosives storage, a vehicle wash bay, an HDPE lined collection pond, power generation, lighting, and parking. The south half of the pad, which houses the explosive silos, wash pad and collection pond is designed to be lined with a 60 mm HDPE liner, which will prevent surface water from entering the environment and will direct water to the collection pond. The collection pond is designed to require periodic pumping. Surface water is prevented from entering or existing the pad via a perimeter berm. A security gate will prevent un-authorized access to the pad.

The detonator magazine pad is 25 m x 25 m in extent and is to be located approximately 250 m east of the bulk explosive magazine pad. It is designed to house the detonator storage magazine. A security gate will prevent unauthorized access to the pad.

Lighting towers will be located adjacent to the bulk explosive magazine and detonator magazine pads along the access road.

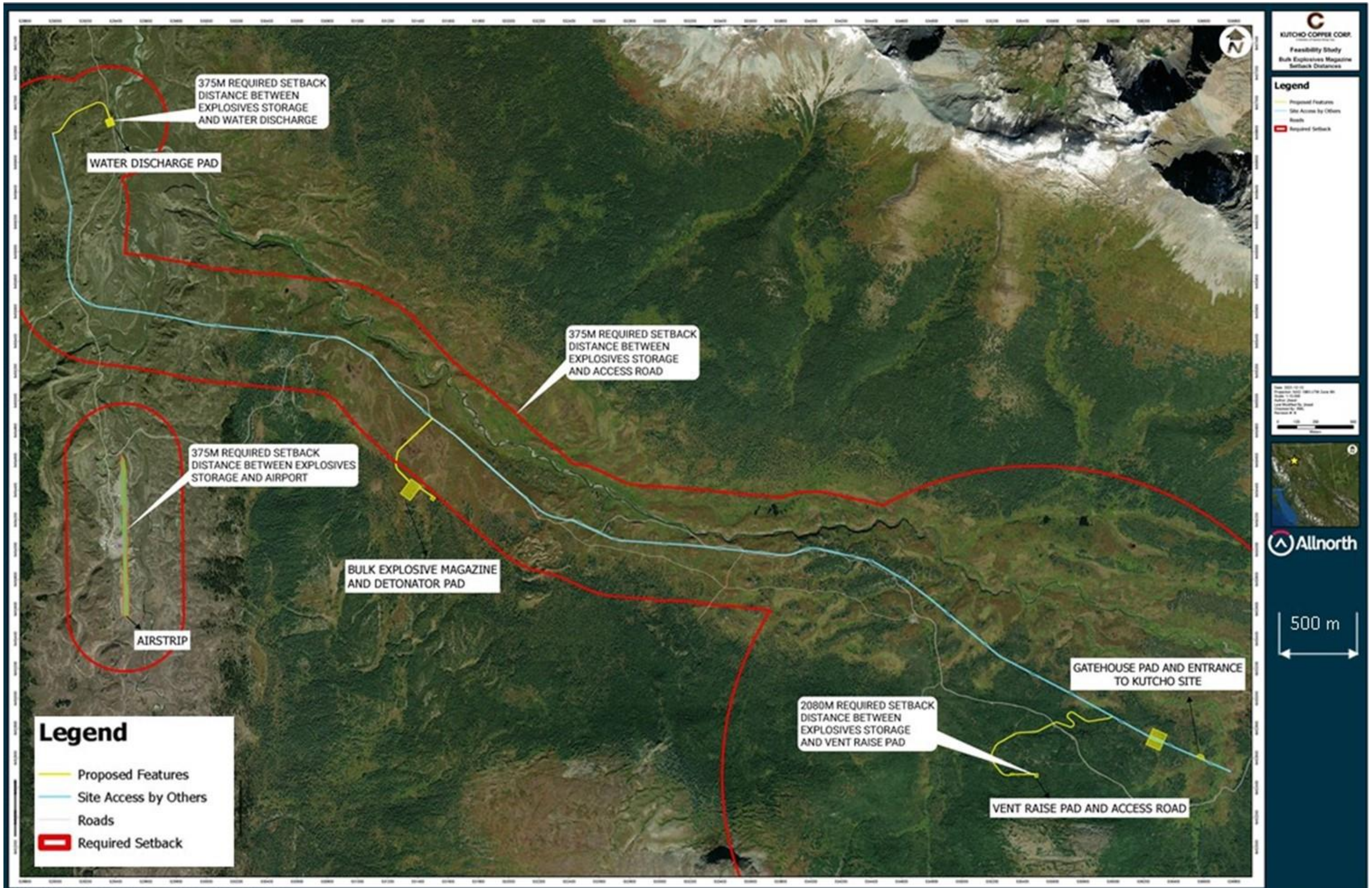


Figure 18-18: Bulk Explosives and Magazine Setback Distance Requirements

18.11.15.1 Magazine Facilities

Explosives will be stored both above ground and underground. Guidance published by the National Standard of Canada/Bureau de Normalization du Quebec (BNQ) was used for these designs. Regulations defining storage methods, materials, areas, and proximities were gleaned from published documents including:

- Explosives - Magazines for Industrial Explosives: CAN/BNQ 2910-500/2015
- Explosives - Quantity Distances: CAN/BNQ 2910-510/2015.

Underground, the magazines will be accessible from a separate drift, parallel to the main drift. This is to provide a safe area for personal to stop and load or unload explosives, detonators, caps, and accessories. Each magazine will be separated from the other by a minimum of 15 m of rock. As illustrated in Figure 18-19 and Figure 18-20, the magazines will feature concrete or concrete block walls and will be 5 meters high and 5 m wide.

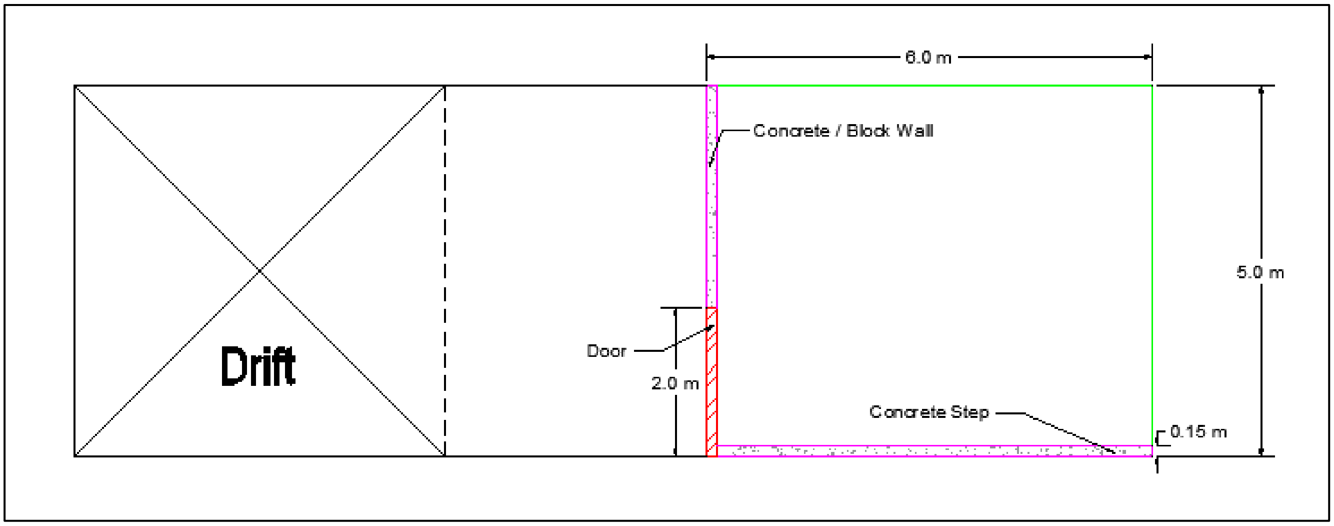


Figure 18-19: Typical Section View – Bulk Explosives Magazine – Not for Construction

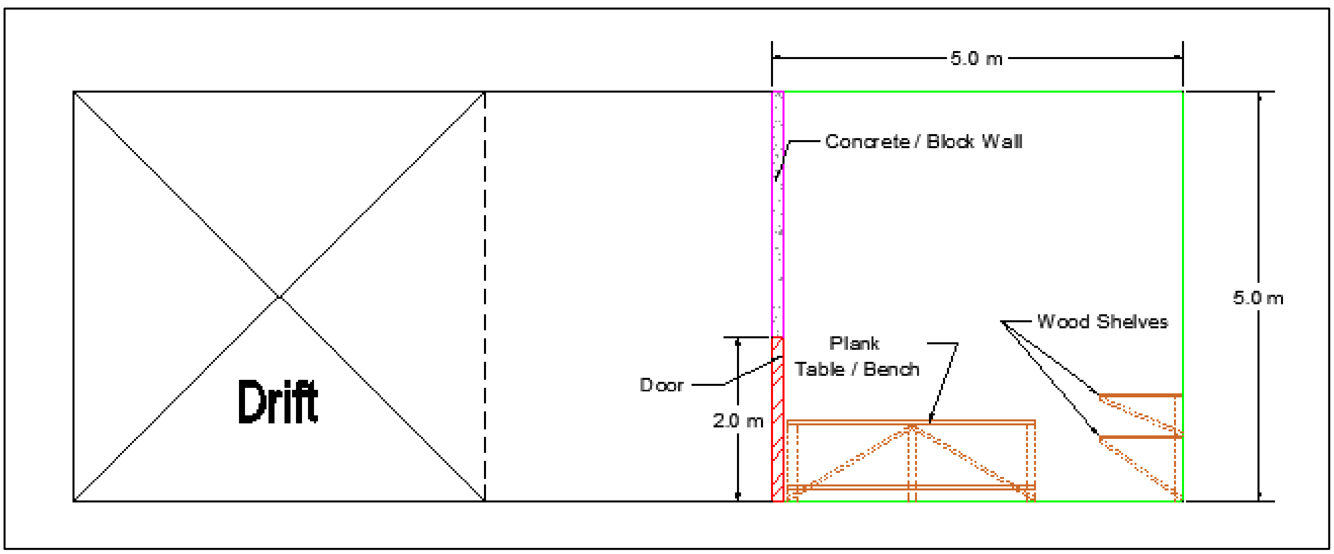


Figure 18-20: Typical Section View – Detonator/Cap Magazine – Not for Construction

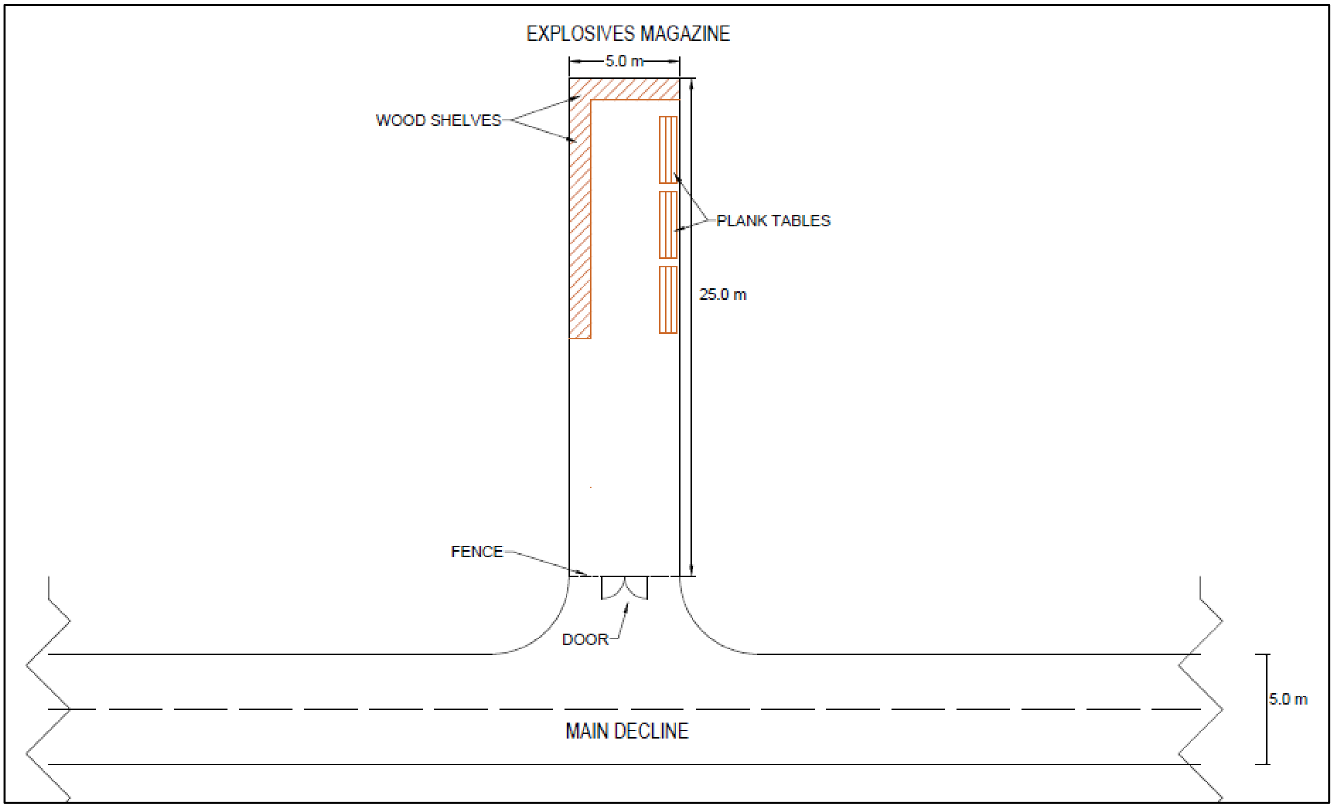


Figure 18-21: Magazine Facilities Plan View – Not for Construction

18.11.16 ***Infrastructure Within Open Pit Blast Radius***

Figure 18-22 identifies the proposed site infrastructure components located within the blast radius of the open pit mine workings. A Blast Management Plan will be required during the construction and operations phase of the mine to provide administrative controls on how blasting will be carried out in proximity to personnel and structures. It is anticipated that key outcomes of the required Blast Management Plan will be to evacuate the mine workshop and the crusher building prior to blasting and to perform an inspection of the equipment and facility after personnel return.

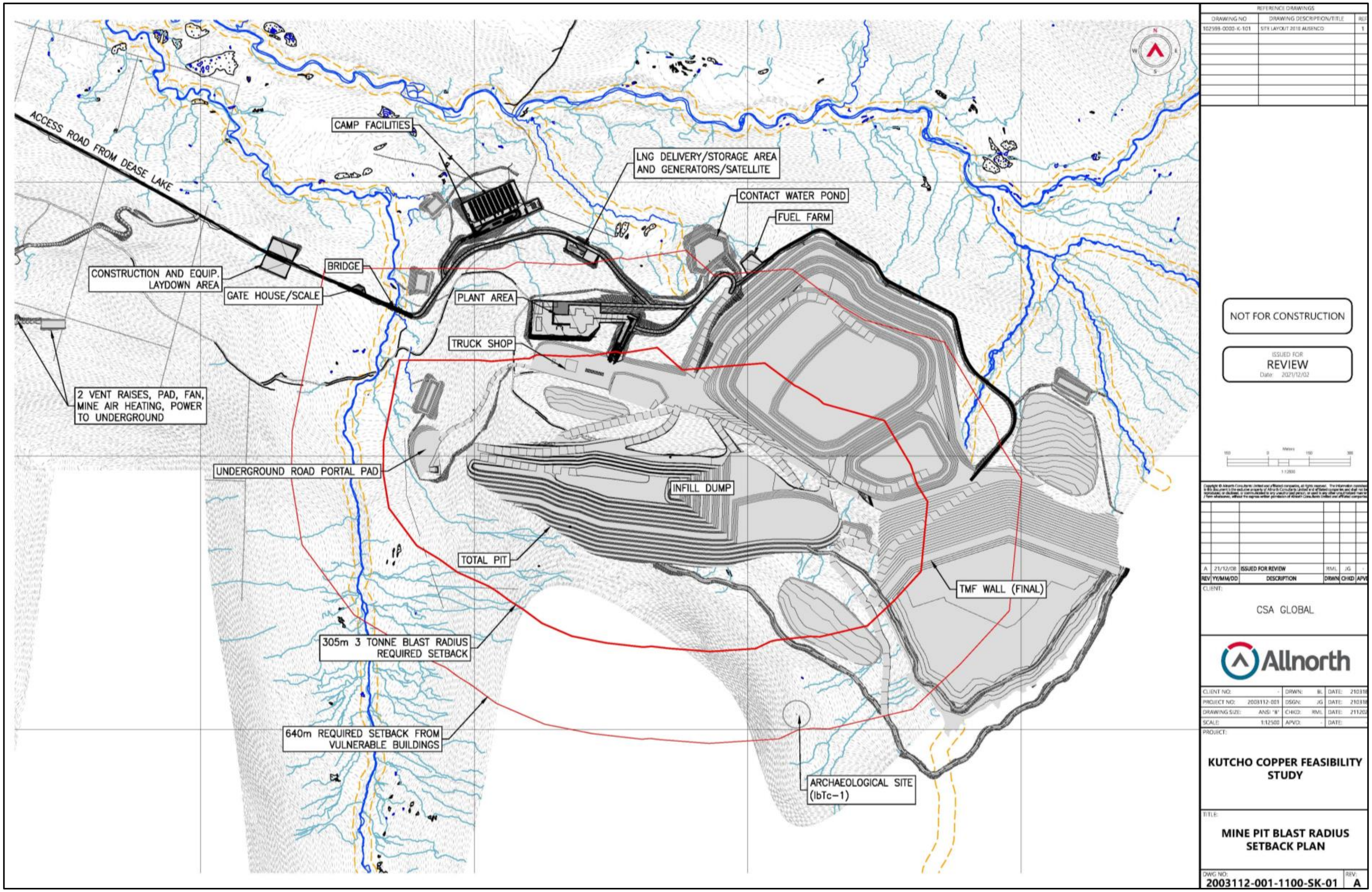


Figure 18-22: Blast Clearance Plan

18.12 Logistics

18.12.1 Personnel Transportation

Personnel that will construct and operate the mine will be primarily sourced from northwestern and northern BC, and Yukon, with some personnel sourced from southern BC. The communities of Dease Lake and Whitehorse are to be used as the primary transportation hubs with Vancouver planned as a secondary hub for personnel from southern BC. Personnel are planned to arrive at the mine either by bus or by air depending on their point of origin.

Personnel from southern BC, via Vancouver, and Yukon and northern BC, via Whitehorse, are to travel by air charter to the airstrip located 10 km from the mine site. These personnel are planned to be met by bus and transported to the site. Personnel from southern BC, Yukon and northern BC are to depart site on the return leg of the air charter.

Personnel from local northwestern and northern BC communities are planned to travel from Dease Lake to site by charter bus.

Personal vehicles will not be allowed at site.

18.12.2 Existing Camp

The existing camp, located adjacent to the airstrip, is leased from Continental Jade by Kutcho Copper for use during exploration activities. The camp can accommodate approximately 50 people and is planned to be used as an early-works construction camp prior to accommodations being installed at the site.

18.12.3 Airstrip

The existing airstrip is 900 m long and located approximately 10 km west of the mine site. Access to the airstrip from the camp is via a spur road connected to the main site access road. Except for equipment to power landing lights, no permanent power or infrastructure will be located at the airstrip.

18.12.4 Goods and Material Transportation

Goods and materials delivered to site are will initially be construction materials and equipment during the construction phase, and consumables during the operations phase of the project. During operations, the primary consumables will be:

- Diesel fuel
- Liquid natural gas
- Cement
- Ammonium nitrate (prill)
- Processing reagents
- Food and camp consumables
- Grinding media.

Diesel fuel is to be trucked in from Prince George, BC, or Edmonton, Alberta. Each super B tanker is anticipated to carry approximately 62,000 L per truck.

Liquid natural gas is planned be trucked in from northern BC or Alberta in specially design super B tankers. Each super B will carry approximately 1,550 GJ of LNG per trip.

Cement is planned to be trucked in sealed bulk units that will allow pneumatic offloading at the site. Cement will be sourced from suppliers located in BC and Alberta.

Ammonium nitrate is to be trucked in prill form to the site in bulk trucks.

Process reagents and consumables are planned to be transported in solid or liquid form depending on the specific consumable.

The planned total volume of trucks per week for each of the consumables listed above is shown Table 18-5.

Table 18-5: Planned truck volumes for consumables

	Unit	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11
Ammonium nitrate	dmt	2,947	3,116	2,496	1,806	2,700	2,676	2,198	409			
	Tr/wk	2	2	2	1	2	2	2	1			
Cement	dmt	5,825	4,726	6,815	6,483	6,506	6,711	7,380	1,601	-	-	-
	Tr/wk	3	3	4	4	4	4	4	1			
Diesel	ML	10.1	7.8	7.1	6.1	7.8	7.5	6.8	4.8	5.8	4.5	1.5
	Tr/wk	4	3	3	2	3	3	3	2	2	2	1
LNG	TJ	630	631	627	621	628	628	613	615	590	569	265
	Tr/wk	8	8	8	8	8	8	8	8	8	8	4
Other[1]	Tr/wk	10	10	10	10	10	10	10	10	10	10	10
Total	**Tr/wk**	**27**	**26**	**27**	**25**	**27**	**27**	**27**	**22**	**20**	**20**	**15**

Other[1] includes process reagents, grinding media, maintenance parts, general freight, and camp supplies.

18.12.5 *Concentrate Haulage*

It is planned that both the zinc and copper concentrates are to be trucked from the mine site to the port of Stewart, BC and shipped to Asia via sea going bulk carriers in 5,000 dmt lots. Bulk concentrate storage and loading facilities at the port of Stewart are planned to be utilized to ship concentrate year-round. The bulk terminal operator currently has available and suitable concentrate storage facilities at the port to store the project's concentrate based on the production schedule.

Concentrate is designed to be transported throughout the year to the port facility by trucks with a 48.65-tonne payload. The distance from the mine site to Stewart is approximately 505 km and the one-way trip is projected to take 10 hours. The transportation company is planned to provide camp lodging at Stewart and Dease Lake for the drivers (each end of the transportation leg). The concentrate trucking schedule is shown in Table 18-6.

Table 18-6: Planned concentrate truck volumes

	Unit	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11
Copper	kdmt	83.5	88.0	108.0	117.7	100.0	93.6	102.9	87.6	64.7	54.6	25.3
	kwmt	91.0	95.9	117.7	128.3	109.0	102.0	112.2	95.5	70.5	59.5	27.5
	Tr/annum	1,871	1,972	2,420	2,638	2,242	2,097	2,305	1,963	1,450	1,224	566
	Tr/wk	36	38	47	51	44	41	45	38	28	24	11
	Tr/day (1)	6	6	8	8	7	7	7	6	5	4	2
Zinc	kdmt	38.2	56.7	72.7	74.5	58.5	57.3	57.3	41.3	29.5	19.1	8.2
	kwmt	41.7	61.8	79.3	81.2	63.7	62.5	62.5	45.0	32.2	20.8	8.9
	Tr/a	857	1,270	1,630	1,669	1,310	1,284	1,284	925	661	429	184
	Tr/wk	16	24	31	32	25	25	25	18	13	8	4
	Tr/day (1)	3	4	5	5	4	4	4	3	2	2	1

(1) A 345-day operating year was used to account for an estimated 20 lost transportation days due to weather and road closures.

The concentrate haulage contractor will be considered to be based in Dease Lake. The contractor will build an office, camp, and maintenance facility for the fleet of 10 trucks. Personnel will fly-in and out of Dease Lake. Concentrate storage facilities exist for the type of truck planned, and the contract includes contractor supply of services. Consequently, no additional concentrate haulage infrastructure costs are anticipated.

19 Market Studies and Contracts

19.1 Introduction

The Kutcho Project will produce a copper concentrate and a zinc concentrate. The copper concentrate will obtain value mainly from copper, but also contained silver and gold content. The zinc concentrate will obtain value mainly from zinc, but also from gold and sometimes silver. Kutcho Copper intends to sell the concentrates to refineries and does not intend to directly produce metals. The mostly likely end-user of the concentrates will be refineries based in China, and possibly Korea or Japan.

Roskill (2021) Consulting Group Ltd ("Roskill") were contracted to complete a market overview of the relevant metals, price range recommendation, and concentrate sales terms. In addition, Kutcho Copper provided consensus forecast metal prices from S&P Global IQ. The discussion within this section on concentrates and metal pricing summarises these provided data and gives additional context to the financial analysis that is fully reported in Section 22.

Having reviewed the studies and analyses, the QP concludes that the results support the assumptions in the technical report.

Table 19-1: Exchange rate and metal prices used in the financial analysis

Parameter	Value
Exchange rate	0.76:1 (USD:CAD)
Copper price	US$3.52/lb (US$7,750/t)
Zinc price	US$1.15/lb (US$2,525/t)
Silver price	US$20.00/oz
Gold price	US$1,600/oz

Kutcho Copper has entered into a material contract with Wheaton Precious Metals Corp. (Wheaton) that if not terminated upon delivery of a Feasibility Study will allow Wheaton to purchase silver and gold produced on the Project on certain terms and conditions. This contract is discussed in Section 19.7.

19.2 Exchange Rate

All metal prices must be considered in the context of the United States dollar Canadian dollar exchange rate. Industrial metal prices in 2020 and 2021 have been underpinned and boosted by a weakening US dollar in combination with disruptive market forces associated to the COVID-19 situation and the low carbon economy transition. Based on data from the International Monetary Fund, CSA Global determined the average three-year trailing exchange rate to the end of September 2021 was 0.765:1 and the five-year was 0.762:1. The exchange rate on 8 October 2021 was 0.80:1.00. Current US dollar weakness is expected to ease and return to the long-term average by the time the project enters production.

The exchange rate utilized in the Feasibility Study financial analysis and in construction of all underlying cost structures where foreign equipment or services are required was therefore selected as 0.76:1.00 (USD:CAD). The USD:CAD exchange rate is one of the more dominant and geared parameters of the Kutcho Project economics since it directly influences metal prices and revenue in terms of Canadian dollar value. Sensitivity analysis of the exchange rate is provided in Section 22.5 of this report.

19.3 Metal Prices

19.3.1 Copper Price

Roskill (2021) reported on data from the International Copper Study Group (ICSG) who estimated that global production of copper metal in 2019 was 20.5 Mt, of which 81% was produced through ore concentration and smelting (the remainder was through recycling). The Kutcho Project total copper production represents about 1/1000th of the annual global production and is therefore not likely to influence market prices.

Roskill (2021) reported that the world copper consumption is expected to continue to grow by a compound rate of 2.5% annually to 2030 as governments and industries continue to advance the transition to a low carbon economy (often termed the Low Carbon Economy Transition or LCET).

Copper is a key component the LCET. Additional demands created through this transition period and moving toward a new economy are creating longer term buoyancy in metal pricing, as evidenced by the three-year and five-year price rolling averages increasing from US$3.01/lb to US$3.12/lb. Current increases in metal prices are believed to be a product of the unwinding of COVID-19 issues such as supply chain constraints and financial incentives, thus generating abnormal market responses. The spot copper price reported on Kitco.com on 15 October 2021 was US$4.71/lb. As such, short-term (one year) and mid-term (two to five years) pricing is well above expected long-term (five plus years) pricing. However, it is important to look beyond the near term and consider forecast pricing in the production period for the project (2027–2038). For the longer-term period Roskill reported an expectation for copper prices in the range of US$3.18/lb (US$7,000/t) to US$4.08/lb (US$9,000/t). Another forecasting service, S&P Global IQ, gave consensus long term forecast median consensus pricing of a more conservative US$3.20/lb with a high consensus price of US$3.50/lb.

The QP considers it appropriate, based on market information available, to consider copper pricing at the higher range of market forecasting. A price of US$3.52/lb (US$7,750/t) was chosen for the Feasibility Study financial analysis. This is in the mid-range of forecasting from Roskill (US$3.18/lb to US$4.08/lb). It should be noted that copper is the most important determinant of project value. Sensitivity analysis of copper price over a range from US$2.64/lb to US$4.39/lb (±25%) is provided in Section 22 of this report (the sensitivity analysis includes a range of other parameters).

19.3.2 Zinc Price

Roskill (2021) reported on data from the International Lead/Zinc Study Group (ILZSG) who estimated that world production of mined zinc metal in 2019 was 12.9 Mt, falling to 12.5 Mt in 2020 due to COVID-19 restrictions. Mine supply is expected to rebound but demand is predicted to similarly return as Chinese stimulus measures feed through to stronger steel production. While prices have been pushing above US$1.19/lb in recent tight markets, in the long term this tightness is expected to weaken and stabilize. The Kutcho Project zinc metal supply represents around 2/1000th of the annual global production and is therefore not likely to influence market prices.

For the longer term, Roskill's (2021) expectation is for a zinc price of US$1.13/lb (US$2,500/t). CSA Global determined the three-year rolling average spot price, based on data from the International Monetary Fund to 30 September 2021, was US$1.15/lb and the five-year rolling average to the same date was US$1.22/lb. The LME spot price on 15 October was US$1.61/lb; this is considered unusually high in the context of previous months.

A price of US$1.15/lb for zinc (US$2,525/t) was chosen for the Feasibility Study financial analysis, which is the same as the three-year and lower than the five-year rolling average spot price. It should be noted that that zinc is a moderately important parameter for determining project value. Sensitivity analysis of zinc price over a range from US$0.86/lb to US$1.44/lb (±25%) is provided in Section 22 of this report (the sensitivity analysis includes a range of other parameters).

19.3.3 Gold Price

Gold will be sold as a contained metal within both the copper and zinc concentrates. The amount of gold contained within the concentrates is insignificant in relation to the global market and therefore will not influence global prices.

Roskill (2021), based on broker consensus forecasts for gold from January 2020, forecasted annual average prices to track downwards from the US$1,900/oz level to just above US$1,600/oz by 2025. Roskill (2021) recommended a price of US$1,600/oz for gold for the production period of the Kutcho Project.

CSA Global determined that the three-year rolling average spot price, based on data from the International Monetary Fund to 30 September 2021, was US$1,606/oz. The LME spot price on 15 October 2021 was US$1,772/oz.

A gold price of US$1,600/oz was chosen for the Feasibility Study financial analysis, consistent with the three-year rolling actual price and the Roskill (2021) recommendation. The Project has limited sensitivity to the gold price; sensitivity data is provided in Section 22 of this report.

19.3.4 Silver Price

Silver will be sold as a contained metal within the copper concentrates and, at times, within the zinc concentrate (the silver grade in the zinc concentrate does not normally achieve saleable grades). The amount of silver contained within the concentrates is insignificant in relation to the global market and therefore will not influence global prices.

Roskill (2021) reports that silver markets straddle both the investment and industrial arenas and shares some controls within both markets. Coming out of 2019, silver underperformed gold with the fundamental market balance remaining in a slight surplus according to the Silver Institute. As gold prices continued to move upwards, through 2019, investment into silver exchange traded products helped push silver prices above the US$16/oz and beyond the US$20/oz level through 2020 and 2021.

Roskill (2021) determined that broker consensus forecasts for silver from January 2020 were for annual average prices to track successively downwards from the US$25/oz level to just above US$20/oz by 2025. Roskill recommended US$20/oz for silver for the production period of the Kutcho Project.

CSA Global determined that the three-year rolling average spot price, based on data from the International Monetary Fund to 30 September 2021, was US$19.99/oz. The LME spot price on 15 October 2021 was US$23.24/oz.

A silver price of US$20/oz was chosen for the Feasibility Study financial analysis, consistent with the three-year rolling average price and the Roskill (2021) recommendation. The Project has limited sensitivity to the silver price; sensitivity data is provided in Section 22 of this report.

19.4 Concentrate Product Specifications and Market Acceptance

19.4.1 Copper Concentrate Specifications and Market Acceptance

Kutcho, as part of the 2020/2021 metallurgical test work program undertaken by Base Metallurgical Laboratories Ltd, (BML report 714, 2021) included detailed analysis of the copper and zinc concentrates produced from locked cycle tests of four composites of the Main/Esso ore (descriptions of the composites can be found in Section 13 of this report). The assays of these concentrates and the expected range of contained elements in deliveries to the market were provided to Roskill for their evaluation. These are shown in Table 19-2.

Table 19-2: Copper concentrate average expected content performance

Element	Unit	Locked cycle test number				Test range and expected			
		17	33	71	73	Minimum	Maximum	Average	Expected
Be	ppm	< 0.5	< 0.5	< 0.5	< 0.5	< 0.5	< 0.5	< 0.5	< 0.5
Sc	ppm	< 1	< 1	< 1	< 1	< 1	< 1	< 1	< 1
Y	ppm	< 1	< 1	< 1	< 1	< 1	< 1	< 1	< 1
B	ppm	< 10	< 10	< 10	< 10	< 10	< 10	< 10	< 10
Ba	ppm	< 10	< 10	< 10	< 10	< 10	< 10	< 10	< 10
Ga	ppm	< 10	< 10	< 10	< 10	< 10	< 10	< 10	< 10
La	ppm	< 10	< 10	< 10	< 10	< 10	< 10	< 10	< 10
U	ppm	< 10	< 10	< 10	< 10	< 10	< 10	< 10	< 10
Th	ppm	< 20	< 20	< 20	< 20	< 20	< 20	< 20	< 20
Sr	ppm	5	2	2	2	2	5	3	3
Tl	ppm	5	< 2	3	2	< 2	5	3	3
Au	ppm	2.76	4.57	2.11	2.11	2.11	4.57	2.89	3.24
V	ppm	10	5	4	4	4	10	6	6
Hg	ppm	9	< 1	15	15	< 1	15	10	10
Bi	ppm	19	< 2	8	7	7	19	11	11
W	ppm	25	< 10	19	18	< 10	25	17	17
Mo	ppm	23	27	17	10	10	27	19	19
Se	ppm	90	0.002	0.010	0.009	0	90	23	23
Ni	ppm	53	19	22	20	19	53	29	29
Co	ppm	33	< 1	38	24	24	38	32	32
Cr	ppm	81	36	40	34	34	81	48	48
Te	ppm	74	68	48	64	48	74	64	64
Sb	ppm	18	206	34	36	18	206	74	74
Mn	ppm	230	41	83	73	41	230	107	107
Zr	ppm	129	89	110	129	89	129	114	114
Cd	ppm	207	193	459	447	193	459	327	327
As	ppm	403	315	413	365	315	413	374	374
Ag	ppm	347	565	285	280	280	565	369	378
Ti	%	< 0.01	< 0.01	< 0.01	< 0.01	< 0.01	< 0.01	< 0.01	< 0.01
F	%	< 0.01	< 0.01	< 0.01	< 0.01	< 0.01	< 0.01	< 0.01	< 0.01
K	%	< 0.01	< 0.01	< 0.1	< 0.1	< 0.01	< 0.1	< 0.1	< 0.1
Na	%	0.013	0.015	0.014	0.014	0.01	0.02	0.01	0.01
Cl	%	< 0.01	< 0.01	< 0.01	0.02	< 0.01	0.02	0.02	0.02
P	%	0.062	0.073	0.071	0.076	0.06	0.08	0.07	0.07
Mg	%	0.22	0.16	0.09	0.08	0.08	0.22	0.14	0.14
Al	%	0.04	0.24	0.18	0.16	0.04	0.24	0.16	0.16
Ca	%	0.36	0.06	0.16	0.11	0.06	0.36	0.17	0.17
Si	%	0.92	0.51	0.28	0.24	0.24	0.92	0.49	0.49
Pb	%	1.87	1.01	0.501	0.513	0.50	1.87	0.97	0.97
Zn	%	4.65	4.45	11.0	11.8	4.45	11.80	7.98	9.83
Fe	%	19.4	28.2	22.3	22.5	28.20	19.40	23.10	23.10
Cu	%	26.6	27.9	28.0	25.2	25.20	28.00	26.93	27.50
S	%	34.6	35.5	35.4	34.2	34.20	35.50	34.93	34.93
Total	**%**	**88.9**	**98.3**	**98.1**	**95.0**	**93.1**	**98.6**	**95.1**	**97.5**

(Source: BML 2021)

Roskill (2021) undertook a comparison of the Kutcho copper concentrate main elements versus major market concentrate specifications and found the Kutcho copper concentrate to compare well. The comparison is shown in Table 19-3. If not blended with other concentrates from other mines, the concentrate will face penalty payments for zinc, lead and cadmium. The quantity of Kutcho copper concentrate in the total market is small and plenty of opportunity exists due to its other qualities for market product blending should these items become an issue for marketing. It is an assumption of this Feasibility Study that the concentrates do not benefit from blending from third party producers.

Table 19-3: Copper concentrate main elements specifications vs major market concentrates

		Kutcho	Mine A	Mine B	Mine C	Mine D	Mine E	Mine F	Mine G	Mine H
H_2O	%	9	8	8	1	10		8	8	9
Cu	%	27.5	30	28	36	27.6	30	31	24.9	23
Ag	ppm	378	60	67	125	45	54	40	141	373
Au	ppm	3.24	1.7	0.5	3	0.4	0.43	0.8	14.2	1.4
Zn	%	9.83	1.5	0.3	0.05	0.2		2.0	0.2	5.5
Pb	%	0.97		0.1	0.1				0.4	1.0
As	%	0.03	0.17	0.11	0.07	0.25		0.01	0.4	0.04
Sb	%	0.002						0.1		
Bi	%	0.001			0.02	0.01				0.08
Hg	ppm	9.9	1.4	3	5	2.2		5.3		
Fe	%	23.1	24.5	26.5	16.5	22	26	18	22.1	26.5
S	%	34.9	35	32	23	32	32.5	31.5	26.1	

(Source: Roskill, 2021)

Roskill (2021) determined that the amount of copper concentrate being sold is small compared to the capacity of the larger smelters, possibly a limiting factor for sale to some smelters. Additionally, any smelter purchasing the concentrate will require the ability to recover the precious metals, similarly limiting available smelters. Nevertheless, it was Roskill's opinion that, based on the copper grades, the concentrate should be able to find a ready market. The QP agrees with this assessment.

Roskill (2021) also completed a detailed assessment of 20 elements contained within the copper concentrate in relation to Chinese National Impurities Tolerances and proposed changes to those tolerances. Roskill determined that the concentrate would be determined as a "clean" concentrate, with acceptable levels of arsenic and maximum penalty element concentrations all falling within the thresholds of China's import requirements.

There is the possibility that proposed Chinese tolerance levels may be applied that will cause some concern with respect to cadmium and thallium. The current permissible level for cadmium is 500 ppm, and the proposed limit is reported to be 300 ppm. The current measured copper concentrate grade for cadmium is 327 ppm, and whilst presently acceptable, may indicate that blending or modified plant processes may be required to reduce the cadmium content if the new limits were applied. The proposed thallium limit of 1.2 ppm is lower than the measured value in the test material of 3 ppm. No test work was conducted on the thallium content controls, its source or means of control (including blending with other concentrates), and this would be investigated further if the proposed limit were to be applied. Further, it is noted that the values being measured are close to detection limits and the accuracy is thus questionable. More precise measurements will be required to accurately determine if and by how much this element poses as a future risk. Because the limit is only a proposal, and with greater knowledge on thallium distribution, Kutcho may be able to develop adequate control strategies.

The QP considers that the current assumption of saleability is considered appropriate for the Kutcho copper concentrate.

19.4.2 Zinc Concentrate Specifications and Market Acceptance

The 2020/2021 metallurgical test work program (BML report 714, 2021) included detailed analysis of the copper and zinc concentrates produced from locked cycle tests of four composites of the Main/Esso ore (descriptions of the composites can be found in Section 13 of this report). The assays of these concentrates and the expected range of contained elements in deliveries to the market were provided to Roskill for their evaluation. These are shown in Table 19-4.

Table 19-4: Zinc concentrate average expected content performance

Element	Unit	Locked cycle test number				Test range and expected			
		17	33	71	73	Minimum	Maximum	Average	Expected
Be	ppm	< 0.5	< 0.5	< 0.5	< 0.5	< 0.5	< 0.5	< 0.5	< 0.5
Sc	ppm	< 1	< 1	< 1	< 1	< 1	< 1	< 1	< 1
Y	ppm	< 1	< 1	< 1	< 1	< 1	< 1	< 1	< 1
B	ppm	< 10	< 10	< 10	< 10	< 10	< 10	< 10	< 10
Ba	ppm	< 10	< 10	< 10	< 10	< 10	< 10	< 10	< 10
La	ppm	< 10	< 10	< 10	< 10	< 10	< 10	< 10	< 10
U	ppm	< 10	< 10	< 10	< 10	< 10	< 10	< 10	< 10
W	ppm	< 10	< 10	< 10	< 10	< 10	< 10	< 10	< 10
Th	ppm	< 20	< 20	< 20	< 20	< 20	< 20	< 20	< 20
Tl	ppm	< 2	< 2	< 2	3	< 2	3	2	2
Sr	ppm	3	2	2	1	1	3	2	2
V	ppm	4	2	4	3	2	4	3	3
Au	ppm	3.51	2.95	0.64	2.67	0.64	3.51	2.44	2.89
Ga	ppm	< 10	10	< 10	< 10	< 10	10	10	10
Se	ppm	30	< 0.001	0.004	0.004	< 0.001	30	10	10
Zr	ppm	30	8	5	6	5	30	12	12
Co	ppm	12	< 1	21	20	< 1	21	13	13
Bi	ppm	12	10	17	19	10	19	15	15
Te	ppm	39	21	< 1	< 1	< 1	39	15	15
Mo	ppm	49	12	4	5	4	49	18	18
Sb	ppm	13	114	4	5	4	114	34	34
Hg	ppm	96	23	12	16	12	96	37	37
Mn	ppm	80	78	152	112	78	152	106	106
Ag	ppm	77	96	74	103	74	103	88	109
As	ppm	274	181	3	6	3	274	116	103
Ni	ppm	18	11	291	247	11	291	142	142
Cr	ppm	24	16	493	423	16	493	239	239
Cd	ppm	2750	2960	3000	3000	2750	3000	2928	2928
Ti	%	< 0.01	< 0.01	< 0.01	< 0.01	< 0.01	< 0.01	< 0.01	< 0.01
F	%	< 0.01	< 0.01	< 0.01	< 0.01	< 0.01	< 0.01	< 0.01	< 0.01
K	%	< 0.01	< 0.01	< 0.1	< 0.1	< 0.01	< 0.1	< 0.1	< 0.1
P	%	0.007	0.007	0.004	0.003	0.003	0.007	0.005	0.01
Cl	%	0.01	0.01	< 0.01	< 0.01	< 0.01	0.01	0.01	0.01
Na	%	0.011	0.014	0.014	0.013	0.01	0.01	0.01	0.01
Al	%	0.01	0.07	0.07	0.05	0.01	0.07	0.05	0.05
Mg	%	0.08	0.07	0.07	0.05	0.05	0.08	0.07	0.07

Element	Unit	Locked cycle test number				Test range and expected			
		17	33	71	73	Minimum	Maximum	Average	Expected
Pb	%	0.18	0.06	0.060	0.068	0.06	0.18	0.09	0.09
Ca	%	0.19	0.01	0.15	0.10	0.01	0.19	0.11	0.11
Si	%	0.19	0.16	0.15	0.11	0.11	0.19	0.15	0.15
Cu	%	0.88	0.62	1.34	2.12	0.62	2.12	1.24	1.19
Fe	%	2.2	4.48	6.66	7.0	2.20	7.00	5.09	6.00
S	%	34.1	35.3	35.6	35.1	34.10	35.60	35.03	35.03
Zn	%	59.7	58.2	54.0	54.2	54.00	59.70	56.53	55.80
Total	**%**	**97.9**	**99.4**	**98.5**	**99.2**	**91.5**	**100.0**	**98.7**	**98.9**

(Source: BML 2021)

Roskill (2021) completed a comparison of the Kutcho zinc concentrate main elements versus major market concentrates specifications and found the Kutcho zinc concentrate to compare favourably, particularly regarding zinc grade. The comparison is shown in Table 19-5.

Table 19-5: Zinc concentrate main elements specifications vs major market concentrates

		Kutcho	Mine A	Mine B	Mine C	Mine D	Mine E	Mine F	Mine G
H_2O	%	9	9			6			
Zn	%	55.8	52	50	51.8	53.5	48.5	50.8	46
Pb	%	0.1	3.8	1.5	2.1	0.8	0.2	0.6	3.1
Cu	%	1.2	0.2	0.1	0.3	2	0.6	3.1	1.8
Ag	ppm	109	150	238	381	53	110	65	452
Au	ppm	2.89	0.1	0.2	6.1		0.7	0.1	6.8
Cd	%	0.29	0.4	0.2	0.3		0.2	0.1	0.27
Fe	%	6	5	3.5	3.5	7.8	11	7.5	4.3
S	%	35.0	31.6	33.0	30.3		34.1	31.5	24.6
SiO_2	%		4.5	2.5	6		0.5	2.2	5.4
As	%	0.01		0.1	0.1		0.1		0.15
Sb	%	0.003	0.1	0.1	0.1		0.005		

(Source: Roskill 2021)

Roskill (2021) determined that the amount of copper concentrate being sold is small compared to the capacity of the larger smelters, possibly presenting a limiting factor for sale to some smelters. Additionally, any smelter purchasing the concentrate will require the ability to recover the precious metals, similarly limiting available smelters. Nevertheless, it was Roskill's (2021) opinion that, based on the zinc grades, this does not discount salability but does create a more limited demand. The QP agrees with this assessment.

China is reliant on imports to satisfy zinc demand and is home to approximately 47% of global zinc smelting capacity. Given the structural need to import zinc concentrate to satisfy the large number of end-consumers, China is the likely destination for the smelting of Kutcho zinc concentrate. The expected zinc concentrate specification is compliant with Chinese import controls as set out currently, but individual shipment grades may vary.

Roskill (2021) also completed an assessment on 10 elements contained within the zinc concentrate in relation to Chinese National Impurities Tolerances. Roskill (2021) determined that the concentrate would be termed a "First Grade" concentrate for sale into China, with acceptable levels of all elements reviewed. Cadmium content is the most sensitive penalty element for the zinc concentrate for sales into China. Strategies for blending ore supply and concentrate handling will be required as the average concentrate grade is at 97.5% of the current 0.3% upper limit for cadmium. Great care will need to be applied in relation to this element in order to avoid concentrate rejection and thereby create a solid First Grade sale product.

Roskill (2021) determined that the Kutcho zinc concentrate would be well suited to the electrolytic smelting methods with lead, iron, copper, manganese, and silica content all within acceptable limits.

19.5 Concentrate Contract Terms

Roskill (2021) provided a guide on what they considered to be standard payment terms based on the observed concentrate qualities and the likely buyers. These guides were adopted for the Kutcho Project. A summary of the concentrate contract terms (other than metal payability) are provided in Table 19-6.

Table 19-6: Concentrate contract terms

Concentrate items	Units	Costs per tonne or unit in/of concentrate	
		Copper	Zinc
Treatment charge	US$/dmt	80.00	200.00
Refining charge – copper	US$/lb	0.08	0.00
Refining charge – silver	US$/oz	0.50	0.00
Refining charge – gold	US$/oz	5.00	0.00
Penalties	US$/dmt	21.78	0.00
Transport to Port (1)	US$/dmt	77.10	77.10
Port charges	US$/dmt	15.00	15.00
Insurance	US$/dmt	5.00	5.00
Ocean freight	US$/dmt	40.00	40.00
Total Charges	**US$/dmt**	**351.71**	**443.56**

(1) Estimated by CSA Global.

19.5.1 Copper Concentrate Contract Terms

19.5.1.1 Payable Terms

Roskill determined the payable terms for the zinc concentrate and those are summarized as:

- Copper – pay for 96.6% subject to a minimum deduction of 1% where the concentrate grade is >20%
- Copper – pay for 96.5% subject to a minimum deduction of 1.1% where the concentrate grade is 17–20%
- Copper – pay for 96.5% subject to a minimum deduction of 1.2% where the concentrate grade is <17%
- Silver – pay for 90% of silver content subject to a minimum content of 30 g/dmt
- Gold – pay for 90% of gold content given a concentrate gold grade of 1–3 g/dmt.

For the financial analysis in Section 22 of this report, the payment terms were applied to quarterly concentrate grades.

19.5.1.2 Penalties

Roskill (2021) report that individual smelters have their own tolerances for impurities or deleterious elements and will apply costs to overcome their treatment or control. Therefore, exact penalty terms will differ between smelters and are often modified depending on each concentrate's particular qualities. Roskill developed a list of typical penalties for copper concentrate smelting based on the provided specification of the copper concentrate shown in Table 19-3. Common penalty elements for copper processing include arsenic, antimony, zinc, lead, mercury, bismuth, nickel, cobalt, chlorine, fluorine, selenium, and cadmium. Concentrations of deleterious elements in the expected zinc concentrate exceed the minimum thresholds for zinc, lead, and mercury. Cadmium may sometimes exceed the minimum payable limit. Most of the penalty is associated with zinc contained in the copper concentrate.

Penalty terms for the problematic elements in copper concentrate are:

- Zinc – penalty of US$1.50 for each 1.0% Zn over 3.0% per dmt concentrate
- Lead – penalty of US$0.20 for each 0.1% Pb over 0.6% per dmt concentrate
- Mercury – penalty of US$2.00 for each 10 ppm Hg over 10 ppm per dmt concentrate
- Cadmium – penalty of US$3.00 for each 0.01% Cd over 0.03% per dmt concentrate.

From the penalty terms provided by Roskill, it was estimated that the average zinc penalty is about US$0.80/t ore crushed or US$14.50/t of copper concentrate. The lead and cadmium penalties amount to US$1.50/t of copper concentrate. The mercury levels result in no penalty payment but has potential in some shipments to have a small penalty levied.

19.5.1.3 Treatment Charges

Roskill (2021) noted that there is no benchmark data for copper concentrate treatment or refining charges. Roskill developed a list of annual negotiations between major counterparties. Their five-year treatment charge averages US$75.5/dmt with spot contracts in 2021 in the low US$50's/dmt. Roskill (2021) consider that the current low spot contract treatment charge reflects a shortage of domestic and imported concentrate in China, exacerbated by COVID-19 related mine closures. They recommend a long-term forecast charge of US$80/dmt, which was adopted for the financial analysis reported in Section 22.

19.5.1.4 Refining Charges

Roskill (2021) noted that there is no benchmark data for copper concentrate treatment or refining charges. Roskill developed a list of annual negotiations between major counterparties. Their five-year treatment charge averages US$0.076/lb copper with spot contracts in 2021 as low as US$0.03/lb. Roskill (2021) consider that the current low refining charge reflects a shortage of domestic and imported concentrate in China, exacerbated by COVID-19 related mine closures. They recommended a long-term forecast charge of US$0.08/lb copper, which was adopted for the financial analysis reported in Section 22.

Roskill (2021) report that refining charges for silver and gold are more opaque but they report that they can vary between US$3.00/oz to US$7.00/oz for gold and US$0.30/oz and US$0.40/oz for silver. Based on their experience they recommended a gold refining charge of US$5.00/oz and a silver refining charge of US$0.50/oz. These recommendations were adopted for the financial analysis reported in Section 22.

*19.5.2 **Zinc Concentrate Contract Terms***

19.5.2.1 Payable Terms

Payable terms for the zinc concentrate are summarized as:

- Zinc – pay for 85% subject to a minimum deduction of 8%
- Silver – pay for 70% of silver content after a minimum deduction of 93.1 g/dmt concentrate
- Gold – pay for 70% of gold content after a minimum deduction of 1 g/dmt concentrate.

The payment terms were applied to quarterly grades. It is noted that silver grades are on average very close to the minimum deduction and attract payment in only about half of the shipments. The timing and quantity of the Esso ore is the main control on payability of silver as this is the main source of high-grade silver in the feed. There may be strategies for campaigning Esso ore that may marginally increase project value. These strategies were not trialed for the Feasibility Study and instead a most conservative "process when available" strategy was applied.

Gold is commonly not payable in zinc concentrate by many smelters, which limits buyer competition. Sales strategies will be required to maximize concentrate sales value, balancing smelter competition with distance to the smelter, payment on gold revenue, and treatment charges. The Feasibility Study includes the cost of a commercial manager to administer the sales process.

19.5.2.2 Penalties

Common penalty elements for zinc processing include arsenic, silica antimony, iron, mercury, manganese, bismuth, fluorine, and cadmium. Expected concentrations of all deleterious elements in the zinc concentrate fall below the minimum penalty thresholds. Roskill does not expect any penalties to be levied.

19.5.2.3 Treatment Charges

Roskill (2021) noted that there is a high degree of volatility in spot treatment charges and therefore a high degree of volatility of premium or discount rates versus benchmark rates. They also noted that contracts have, until recently, included price escalation/de-escalation clauses that, as a broad guide, provide some price participation to the miner. With volatile zinc prices, the 2020 setting of benchmark terms did not include price escalators/de-escalators. Roskill (2021) considers it appropriate moving forward to not include such a term, as does the QP.

Roskill (2021) reported 2020 contracts traded on Open Mineral were in the range of US$80/dmt to US$100/dmt. However, there are indications of lower rates in 2021 but Roskill (2021) is forecasting a return to higher treatment charges by 2027 when processing commences for the Project.

19.5.2.4 Refining Charges

No refining charges are predicted for the zinc concentrate for zinc, silver, or gold.

19.6 Concentrate Freight Rates

Both the zinc and copper concentrates will be assumed to be shipped to China by the same transportation methods. Therefore, transportation rates apply equally to both. The bulk concentrate handling facilities at the port of Stewart, BC will be utilized. The port has concentrate storage facilities suitable for both concentrates that have been verified by Allnorth as available for the project timeframe. Concentrate is planned to be continuously transported to the port facility by a contractor and budget contract terms were received by a locally based capable haulage company with appropriate capacity.

It is estimated that the haulage fleet will comprise a total of 10 truck/trailers (inclusive of spares) capable of 48.65 wmt per load for a total cost of C$4,656/load or US$79.29/dmt. Contract details include:

- 131,000 wmt annually (11-year average)
- 23 full-time drivers and six standby drivers
- 19.5-hour round trip
- 685 litres of diesel per trip at C$1.25/L
- Driver wages were applied inclusive of taxes and support costs and include a premium for the Dease Lake to site leg of the journey
- Contractor provides air transport for drivers in and out of Dease Lake
- Fully inclusive driver camp and office provided by contractor at Dease Lake.

The port facility will store up to 10,000 dmt of concentrate allowing sea shipment of 5,000 dmt lots of either type or both.

The financial model also includes a 2,000 dmt storage of concentrate at the Kutcho plant prior to first overland trucking. Therefore, the financial model includes a 7,000 dmt delay prior to first sales.

Off-site concentrate transport costs are estimated as:

- For inland freight Allnorth estimated a transport cost of US$79.29/dmt
- Allnorth determined that port charges will amount to US$15/dmt
- Ocean freight was determined by Roskill to be US$40/dmt for transport to smelters in China
- Roskill assessed that insurance and assaying charges as US$5/dmt of concentrate.

CSA Global considered that marketing costs were covered by wages associated to the commercial manager, associated staff, and office costs. These charges are included in the site administration operating cost.

19.7 Major Contracts

A major contract, the Wheaton Precious Metals Corp Purchase Agreement, is in place. In August 2017, Kutcho Copper Corp and Wheaton Precious Metals Corp. (Wheaton) entered into an Early Deposit Precious Metals Purchase Agreement (PMPA) whereby Wheaton will be entitled to purchase 100% of the silver and gold production from the Project until 5.6 Moz of silver and 51,000 ounces of gold have been delivered, at which point the stream will decrease to 66.67% of the silver and gold production for the life of the mine. Wheaton will make ongoing production payments to Kutcho equal to 20% of the applicable spot prices of silver and gold delivered. An additional payment of up to US$20 million will be payable by Wheaton should the processing capacity increase to 4,500 tpd or more within five years of attaining commercial production.

Under the terms of the proposed PMPA, Kutcho Copper was advanced US$7 million as an Early Deposit to fund the Feasibility Study. The balance of the US$65 million is payable during construction of the Kutcho project, subject to certain conditions being met. Wheaton will have the option to terminate the PMPA at any time after the Early Deposit payment and before delivery of the Feasibility Study documentation, or at any time on or after 14 December 2019 if the Feasibility Study documentation has not been delivered to Wheaton by such date, or at any time within 90 days after the delivery of the Feasibility Study documentation, or at any time there is a default.

Should Wheaton choose not to terminate the contract after delivery of the Feasibility Study, it is estimated that the PMPA will reduce the after-tax net present value (at 7% discount rate) of the Kutcho Project from C$461 million (as per Section 22) to C$417 million. Conversely, due to the timing and value of the upfront payment relative to the repayments, the IRR increases from 24.8% to 25.9%. The impact of adding the PMPA does not alter the Mineral Reserve. The Mineral Reserve was developed based on operational cut-offs that are in excess to the incremental breakeven cut-off and the difference is greater than the added impact of the PMPA. Adding the cost of the PMPA does not therefore result in ore being required to be re-designated as waste. There are other features of the mine plan development documented in Section 15 concerning the robustness of the open pit and underground mine designs in respect of the PMPA.

The PMPA contract terms are uncommon in the mining industry and in particular to BC. A discussion on industry precedents is therefore not applicable. Furthermore, due to the same novelty of the contract, certain tax implications are open to some interpretation. In evaluation of the impact of adding the PMPA, CSA Global has assumed that BC Mineral Tax deductions of the PMPA streaming payments are allowed.

Royal Gold Inc. holds “back-in” rights to certain claims as described in Section 4.5. Royal Gold is entitled to a royalty of 2% of net smelter returns on certain claims if it does not elect to exercise the “back-in” rights). Royal Gold acquired this royalty interest effective 1 October 2008, as part of the acquisition of a royalty portfolio from Barrick Gold Corporation. The economic assumption of this Technical Report is that Royal Gold will elect to not “back-in” and therefore the only effect applied to the Project from this agreement is that the Royal Gold royalty is allocated to the economic analysis. This is considered a reasonable assumption for the Project. The terms of Royal Gold choosing to “back-in” are described in Section 4.5.1.

20 Environmental Studies, Permitting and Social or Community Impact

20.1 Introduction

The Project is located approximately 100 km east of Dease Lake in northern British Columbia, within the traditional territories of the Kaska Dena Nation and Tahltan Nation. Liard First Nation is of the view that the Kaska Dena Council (Kaska) represents their members, including Daylu Dena Council, and this was communicated to the Environmental Assessment Office (EAO). Consultation with Indigenous Nations, the public, stakeholders, and government agencies commenced in 2005 when a previous proponent originally submitted the Project as a proposed open pit mining operation. The EAO has identified to Kutcho Copper which First Nations should be consulted as part of these consultation activities that would continue through the environmental assessment (EA) and permitting process.

The closest community to the mine site is Dease Lake, located approximately 100 km to the west and accessed by a 119 km seasonal access road known as the "Jade Boulder Road". The closest Indian Reserve is the Tahltan Dease Lake Indian Reserve No. 9, located adjacent to Dease Lake. The closest Indigenous community to the mine site is the Iskut Nation community, located on Iskut 6, approximately 83 km south of Dease Lake on Highway 37.

The Project is on Crown Land, but does not overlap with any private, provincial, or federal titled or surveyed lands, nor does it encroach on any provincial or national parks, protected areas, or historic sites. There are no existing or past commercial developments, industrial facilities, or known permanent, seasonal, or temporary residences overlapped by the Property. No zoning bylaws apply to lands required for the Project. There are no agricultural land reserves near the Project and there are no water licenses in the vicinity of the Project.

The mine site, airstrip and a portion of the access road are located within the Dease-Liard Sustainable Resource Management Plan (DLSRMP). Mining is an acceptable land use under the DLSRMP. The resource management zone overlapped by the Project is designated as wildlife/wilderness/back-country recreation/tourism, with a resource management emphasis on wildlife, cultural, tourism and recreation values. The Province and Tahltan signed a framework agreement in June 2021, which funds Tahltan to advance land use plans. This agreement will influence future EA and permitting activities for the Kutcho Project.

A portion of the mine site, the airstrip, and the access road are located in an area designated as high-value thinhorn sheep habitat. The access road passes through a "large river corridor" tourism resource management zone. In addition, the DLSRMP provides specific management strategies for bull trout, a blue-listed species, which are present in Andrea Creek.

There is no forestry or agriculture in the region of the Project. Placer and jade mining, guide outfitting, and trapping are the primary economic activities. Kaska Dena Nation and Tahltan Nation have advised that mineral exploration, jade, and placer mining are significant economic activities in the area and are concerned with the environmental impacts of these activities. They have provided information that over 1,100 active claims and over 100 exploration permits are found in the Jade Boulder Road area. Their concerns relate to the incremental pace of development in an area of cultural significance to both Nations. Recreational pursuits in the region include hunting, horseback riding, hiking, camping, back-country skiing, rafting, canoeing, and snowmobiling. Traditional use of lands and resources in the vicinity of the Project was documented for both the Kaska Dena Nation and Tahltan Nation.

In summary, and as presented in detail below, there are currently no known environmental issues that are likely to preclude or materially impact Kutcho Copper's ability to construct, operate and successfully close the proposed mine. All environmental issues identified can be expected to be reasonably mitigated.

20.2 Regulatory Approval Process

20.2.1 Federal Environmental Assessment

Kutcho Copper does not anticipate that the Project would trigger a federal impact assessment under the Federal Impact Assessment Act (S.C. 2019, c. 28, s. 1). This is because the Project's production rate of up to 4,500 tpd is below the threshold of 5,000 tpd for a new metal mine identified under the Regulations Designating Physical Activities (SOR/2019-285). However, the Project does have several potential triggers, as detailed in Table 20-1 that could prompt more federal involvement with the EA process.

Table 20-1: Summary of CIA Act triggers and Kutcho Copper potential to trigger a Federal EA

Potential trigger	Reason project is not anticipated to trigger CIA Act
1. The Project is federally sponsored.	Not applicable
2. The federal government grants money or financial assistance to the proponent to enable a project to proceed.	Not applicable
3. The federal government grants an interest in federal land to enable a project to be carried out.	Not applicable
4. A federal agency or department exercises a regulatory duty in relation to a project. The following Acts have been considered:	
5. Fisheries Act – the authorization is required for the harmful alteration, disruption or destruction of fish habitat. *6. Metal and Diamond Mine effluent Regulation (MDMER)* Authorization is required under this act for the deposition of deposit of, an effluent containing any deleterious substance that is prescribed in section 3 in any water or place referred to in subsection 36(3) of the Act	HADD – Potential trigger MDMER – Potential trigger
7. Navigable Waters Protection Act – if the Project potentially affects navigability through the construction or alteration of works on, over, under, through or across a navigable waterway.	Potential trigger – though only two streams remain if the access road is removed from the EA.
8. Migratory Birds Conservation Act and Species at Risk Act – requires evidence that no impact on migratory birds or any species at risk by the Project.	Potential trigger as there is the possibility for migratory birds to use the site.
9. Species at Risk Act – requires evidence that there would be no impact on any species at risk by the Project.	Potential trigger as species at risk do occur at the site.
10. Explosives Act – requires federal approval to locate explosives factory on project site.	Potential trigger. The site would have an explosive storage area. Underground explosives would be pre-mixed and normally a Powergel-type explosive. The open pit would use ANFO prill, mixed on-site, and loaded directly into the blast hole. Both would require detonators and cord to be stored at the site.
11. Canadian Transportation Act – applies to certain projects where a rail line crossing or relocation is contemplated.	Not applicable
12. Indian Act and Natural Resources Act – covers projects that are located on or require access through federal lands such as national parks, First Nation reserves, or national defence bases.	Not applicable

20.2.2 Provincial Environmental Assessment

The Project Description for the Project was submitted under the British Columbia Environmental Assessment Act (2002) and the Project is reviewable under this Act. An EA is needed as the Project’s proposed production capacity exceeds the threshold of 75,000 tpa of mineral ore for a new mineral mine, as set out in the Reviewable Projects Regulation (BC Reg. 370/2002). After submitting the Project Description, the Environmental Assessment Act was updated to the BC Environmental Assessment Act (2018) and the project would now follow the 2018 Act process with an entry point just prior to Process Planning as shown in Figure 20-1.

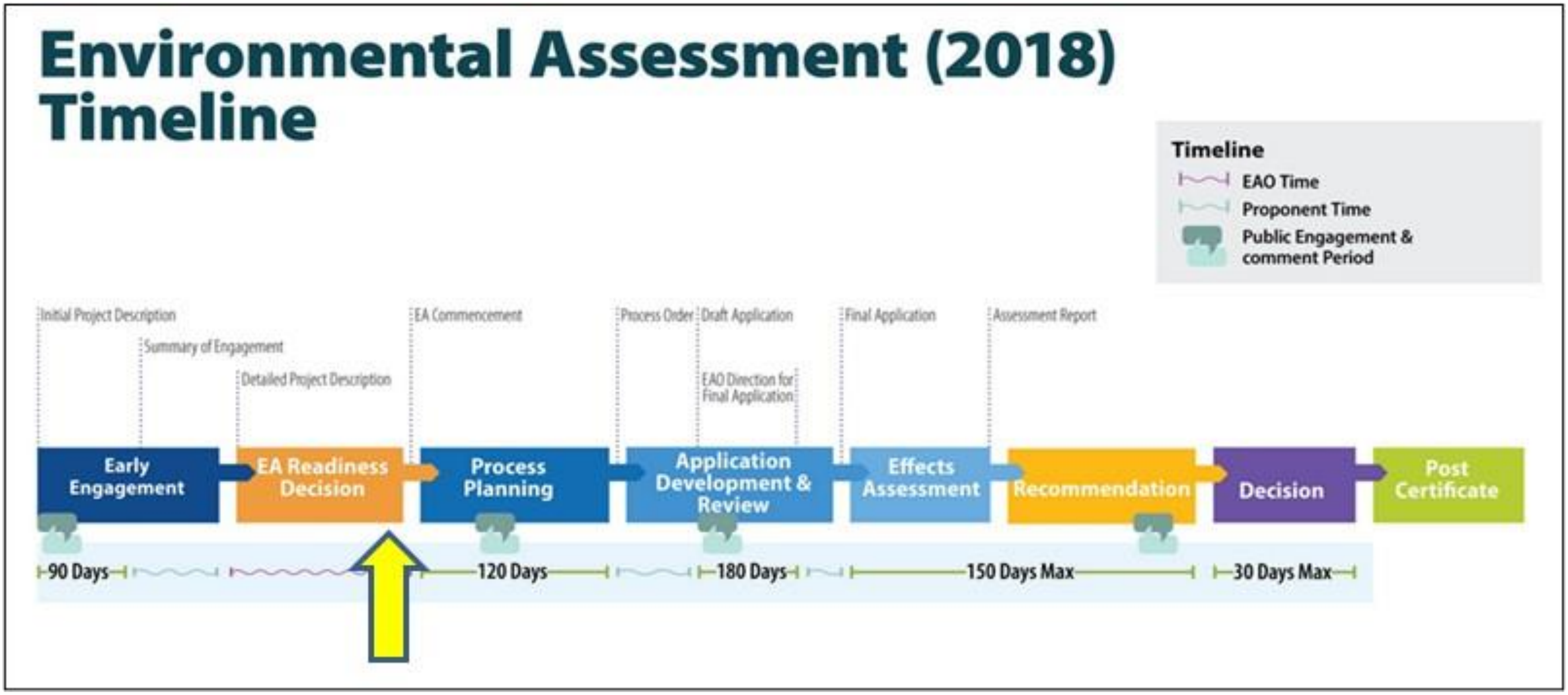


Figure 20-1: The 2018 Environmental Assessment Process

Note: Arrow indicates Kutcho Copper progress at issue of the Feasibility Study.

The EAO has notified Kutcho Copper that the new process is to be followed to completion of the Project's EA. Since submitting the 2018 Project Description, Kutcho Copper has updated the mine plan for the Feasibility Study. Consequently, an amendment to the Project Description would be required. The EAO has indicated that Kutcho Copper would be required to commence immediately prior to the "Process Planning" step (Yellow arrow in Figure 20-1). It is anticipated that this would occur towards the end of 2021 or within the first quarter of 2022. As part of re-entry into the Process Planning step, Kutcho Copper would be submitting the following documents:

- Amendment to the Project Description
- Draft Application Information Requirements
- The initial list of Valued Components that would be taken into consultation
- An Updated Consultation Plan.

It is anticipated that the Environmental Assessment Application would be submitted in 2023 and that a decision by EAO would be made in late-2023/early-2024. Kutcho Copper intends to run a separate and parallel process for permitting under the *Mines Act*, leading to the start of construction in 2024. The Major Mines Office (MMO) oversees the permitting process independent of, but in collaboration with the EAO.

The Crown (federal, provincial, and territorial governments) has a legal duty to consult with and, where appropriate, accommodate Indigenous interests when considering projects that might adversely affect asserted or established Aboriginal or treaty rights. The Crown may delegate procedural aspects of consultation to proponents.

The Federal and BC governments have committed to a renewed relationship with Indigenous Peoples based on recognizing Indigenous Rights. Both governments have endorsed the United Nations Declaration on the Rights of Indigenous Peoples (UNDRIP; United Nations 2007) and recognize the need to consult with Indigenous Nations to secure their free, prior, and informed consent. Kutcho Copper would work with Kaska Dena Nation and Tahltan Nation with the intent of securing their free, prior, and informed consent that respects each Nation's unique decision-making processes based on all sources of knowledge.

Under the new BC Environmental Assessment Act, an impact assessment would be required to assess impacts on Indigenous rights, and decision making would include consideration of indigenous knowledge. The new BC environmental assessment process is designed to ensure that any decision taken on the question of consent by an Indigenous Nation is free, prior, and informed. The BCEAO would work together and seek consensus with Indigenous Nations at a technical level throughout the EA process. There would also be key decision points where Indigenous Nations may express their consent, lack of consent, or abstain from deciding on behalf of their communities. Kutcho Copper intends to abide by the intent of the new BC legislation.

Under Section 25(2)(g) of the new Environmental Assessment Act, an environmental assessment would be required to consider any regional or strategic assessment conducted under the Act. Kutcho Copper is unaware of any such current or planned studies in the vicinity of the Project.

20.2.3 Potential Exclusion of the Access Road

A portion of the Jade Boulder Road is located within the Dease-Liard Sustainable Resource Management Plan and a section within the Cassiar Iskut-Stikine Land Resource Management Plan. The Province has issued a special use permit for the road (S26726) to the Boulder Trail User Association; Kutcho Copper is not part of the Association. Kaska and Tahltan Nations have expressed concern related to the road allowing uncontrolled access to areas for the purposes of hunting and fishing.

Kutcho Copper initiated discussions to improve the road alignment, surface, and associated creek crossings in partnership with Tahltan and Kaska. Based on discussions with Tahltan and Kaska, the EAO has indicated that the access road could potentially be removed from the EA process. In the partnership proposed, the road would be owned and maintained by First Nations as a restricted access toll road that would provide Kutcho Copper unfettered access to the Project site at a fee to cover maintenance and operation of the road. Although discussions are at an early stage, EAO has presented the matter to the BC Province for guidance. Kutcho Copper would participate in Tahltan, Kaska, EAO and Province meetings to further discuss the road partnership.

20.2.4 Kaska Dena Nation and Tahltan Nation Authorizations

Kaska Dena Nation and Tahltan Nation authorizations would include consent-based decisions provided to Kutcho Copper, and the Government of British Columbia.

The authorizations would provide sufficient direction on the Project's potential effects to each Nation and the range of mitigations, monitoring, and contingencies required if the Project is approved.

Consent would be based on the potential effects, mitigations, and contingencies of Kaska Dena and Tahltan Nations criteria, methods, scale, magnitude, analysis, and mitigations of each Nation.

20.3 Environmental Authorizations and Permits

20.3.1 Existing Permits

Kutcho Copper currently holds a Mines Act multi-year area-based permit (MYAB) permit for exploration (Permit MX-1-612). This permit is valid for five years; it was issued in 2018 and would expire in March 2023. An annual MYAB permit notification of work is being prepared for review by First Nations and the provincial regulator to accommodate the 2022 exploration.

20.3.2 Federal and BC Permit Requirements

Table 20-2 and Table 20-3 present the list of potential Federal and BC authorizations, licenses and permits that may be required for the Project. It would be updated as the Project moves through the EA and regulatory approval process.

Table 20-2: Potential federal authorizations, licences and permits

Authorization	Legislation	Agency	Purpose
Fish Collection Permit	*Fisheries Act*	Fisheries and Oceans Canada	Authorization for fish salvage for data collection
Authorization for Works or Undertakings Affecting Fish Habitat	*Fisheries Act*	Fisheries and Oceans Canada	Authorization(s) if the Project Harmful Alteration, Disruption or Destruction (HADD) (e.g. watercourse crossings and clearing riparian vegetation)
Various Operational Statements	*Fisheries Act*	Fisheries and Oceans Canada	Operational statements for installing clear-span bridges, temporary stream crossings, bridge and culvert maintenance, overhead line construction, and riparian maintenance
Authorization	*Navigation Protection Act*	Transport Canada	Authorizes work built in, on, over, under, through or across any navigable water that may interfere with navigation
Radio Licences	*Radio Communication Act*	Industry Canada	License for radio frequencies for the Project
Explosives Magazine Storage and Use Permit	*Explosives Act;* Health, Safety, and Reclamation Code for Mines in British Columbia	Natural Resource Canada; Ministry of Energy and Mines	Authorization for explosive use and storage during construction
Explosives User Magazine Licence	*Explosives Act*	Natural Resource Canada	Authorization for storage of blasting explosives and other types of industrial explosives (required permits and licenses to be obtained by explosives vendor)

Table 20-3: Potential provincial authorizations, licences and permits

Authorization	Legislation	Agency	Purpose
Environmental Assessment Certificate	*Environmental Assessment Act*	EAO	To minimize or avoid adverse environmental, heritage, health, social and economic effects.
Mine Permit	*Mines Act*	Ministry of Energy, Mines and Low Carbon Innovation	Authorizes development, operations, closure, reclamation, and abandonment.
Mining Lease	*Mineral Tenure Act*	Ministry of Energy, Mines and Low Carbon Innovation	The correct term is to "replace" mineral claims with a mining lease. Allows use of the surface of lands for the production of minerals.
Effluent Discharge Permit	*Environmental Management Act*	BC Ministry of Environment and Climate Change Strategy	Authorizes discharge of liquid effluent to the environment (sediment, tailings and sewage, filter plant).
Air Discharge Permit	*Environmental Management Act*	BC Ministry of Environment and Climate Change Strategy	Authorizes discharge of airborne emissions to the environment (crushers, refuse, incinerators).
Hazardous Waste Registration	*Environmental Management Act* Hazardous Waste Regulation (BC Reg. 63/88)	Ministry of Environment and Climate Change Strategy	Authorizes temporary storage of hazardous waste.
Open Burning Permit	*Environmental Management Act*	Ministry of Environment and Climate Change Strategy	Authorizes burning due to land clearing activities.
Notifications	*Water Sustainability Act*	Ministry of Environment and Climate Change Strategy	Notifications are used for works that do not involve water diversion, may be completed within a short period and would have minimal impact on the environment or third parties.
Approval for Works in and about a Stream	*Water Sustainability Act*	Ministry of Environment and Climate Change Strategy	Permission to work in and about a stream (i.e. stream crossings).
Short Term Water Use Approval	*Water Sustainability Act*	Ministry of Environment and Climate Change Strategy	Authorizes short term water use.
Water Use Licence	*Water Sustainability Act*	Ministry of Environment and Climate Change Strategy	Authorizes storage, use or diversion of surface water, including installation of works.
Inspection Permit	*Heritage Conservation Act*	Ministry of Forests, Lands and Natural Resource Operations and Rural Development	Authorizes field study to assess the archaeological potential.
Investigative Permit	*Heritage Conservation Act*	Ministry of Forests, Lands and Natural Resource Operations and Rural Development	Authorizes mitigation of impacts to sites (should any be identified) through systematic data recovery after an impact assessment was completed.
Site Alteration Permit	*Heritage Conservation Act*	Ministry of Forests, Lands and Natural Resource Operations and Rural Development	Authorizes alteration or removal of a site (should any be identified and impacted by the Project.
Fish Collection Permit	*Wildlife Act*	Ministry of Forests, Lands and Natural Resource Operations and Rural Development	Authorizes fish salvage for data collection.
Wildlife Salvage Permit	*Wildlife Act*	Ministry of Forests, Lands and Natural Resource Operations and Rural Development	Authorization for amphibian/small mammal capture and release.

20.3.3 Other Future Authorizations

The Project may require additional federal, provincial, and municipal licenses, permits, and authorizations depending on the final project design. Table 20-4 sets out the authorizations, licences and permits that may be required.

Table 20-4: Other potential authorizations, licences and permits for the Project

Other Approvals and Licences	Enabling Legislation
IAAC Screening to exclude the Project from the Federal EA process	*Canadian Impact Assessment Act*
Metal and Diamond Mining Effluent Regulations (MDMER)	*Fisheries Act* / Environment Canada
Fish Habitat Compensation Agreement	*Fisheries Act*
Dangerous Occurrence Report	Transportation of Dangerous Goods Regulation
Radioisotope Licence (Nuclear Density Gauges/X-ray Analyzer)	*Atomic Energy Control Act*
SARA approvals(1) to work around listed species	*Species at Risk Act*
Permit to work near or alter archaeological sites	*Heritage Conservation Act*

(1) Species at Risk Act (SARA) Approvals: Although the broader project area falls within the range of several species at risk, these have not been encountered on or near the site that would be disturbed. ERM was retained to evaluate habitat suitability, undertake surveys, and develop management plans before any mine construction and operational activities occur.

20.4 Consultation

20.4.1 First Nations Considerations

Traditional use of lands and resources in the vicinity of the Project was documented for both the Kaska Dena Nation and Tahltan Nation. Kutcho Copper has engaged in formal meetings, regular phone calls, letters, and email throughout the project. This contact continues to be maintained, and a formal consultation plan would be developed for the BCEAO with Input from both the Kaska and Tahltan Nations. Kutcho Copper remains committed to working with First Nations through a jointly developed Consultation Plan.

20.4.2 Consulted First Nations

In May 2018, Kutcho Copper was advised that the Dease River First Nation (DRFN), on behalf of the Kaska Dena Nation, which includes the Kwadacha First Nation, Daylu Dena Council, and the Kaska Dena Council, were the lead to engage with Kutcho Copper on the Project. Since that time, DRFN and the Tahltan Nation were consulted on the Project Description and the determination of appropriate mechanisms to facilitate meaningful engagement and participation within the EA process.

In a further update from DRFN in June 2021, DRFN confirmed with EAO that they would also represent the Liard First Nation on the Kutcho Copper Project.

Kutcho Copper continues the consultation undertaken by previous proponents of the Project. Between 2006 and 2011, previous proponents engaged with Kaska and Tahltan through meetings, community open houses, community meetings, a socio-cultural effects assessment workshop and a community researcher training event. Topics discussed included the project overview and updates, the environmental assessment process, employment opportunities, Indigenous Knowledge research and protocols, capacity funding, and socio-cultural effects assessment and business opportunities.

20.4.3 Informed First Nations

Kutcho Copper has communicated with the Treaty 8 Tribal Association and all Treaty 8 Nations several times (June and July 2017, January 2018) during the acquisition of the Project. In addition, Kutcho Copper met with West Moberly First Nation in January 2018 to provide an overview of the Project. Kutcho Copper would continue to provide ongoing information sharing as required. It is not anticipated that the Project would impact on Treaty 8 First Nations' current use of lands and resources for traditional purposes.

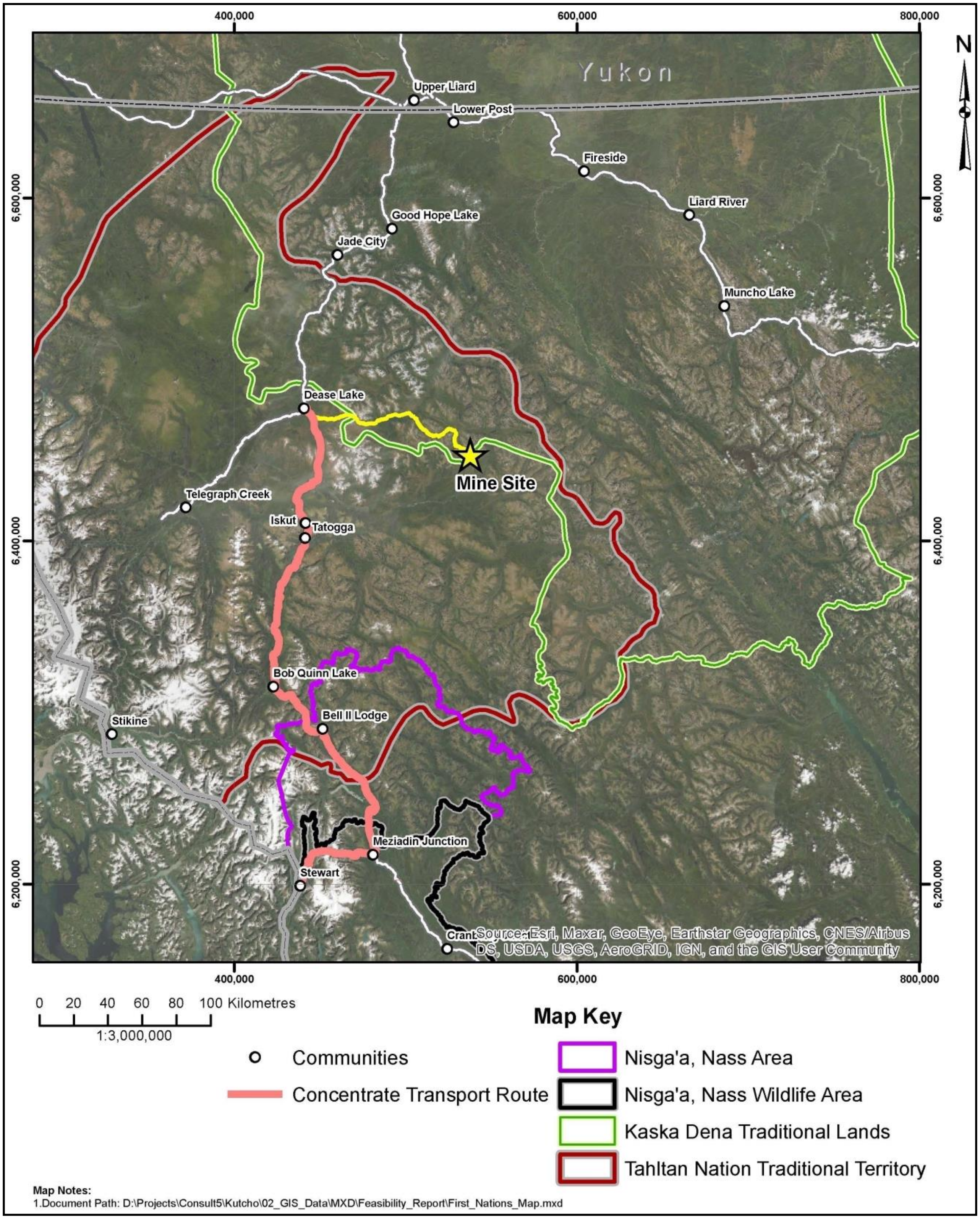


Figure 20-2: First Nations

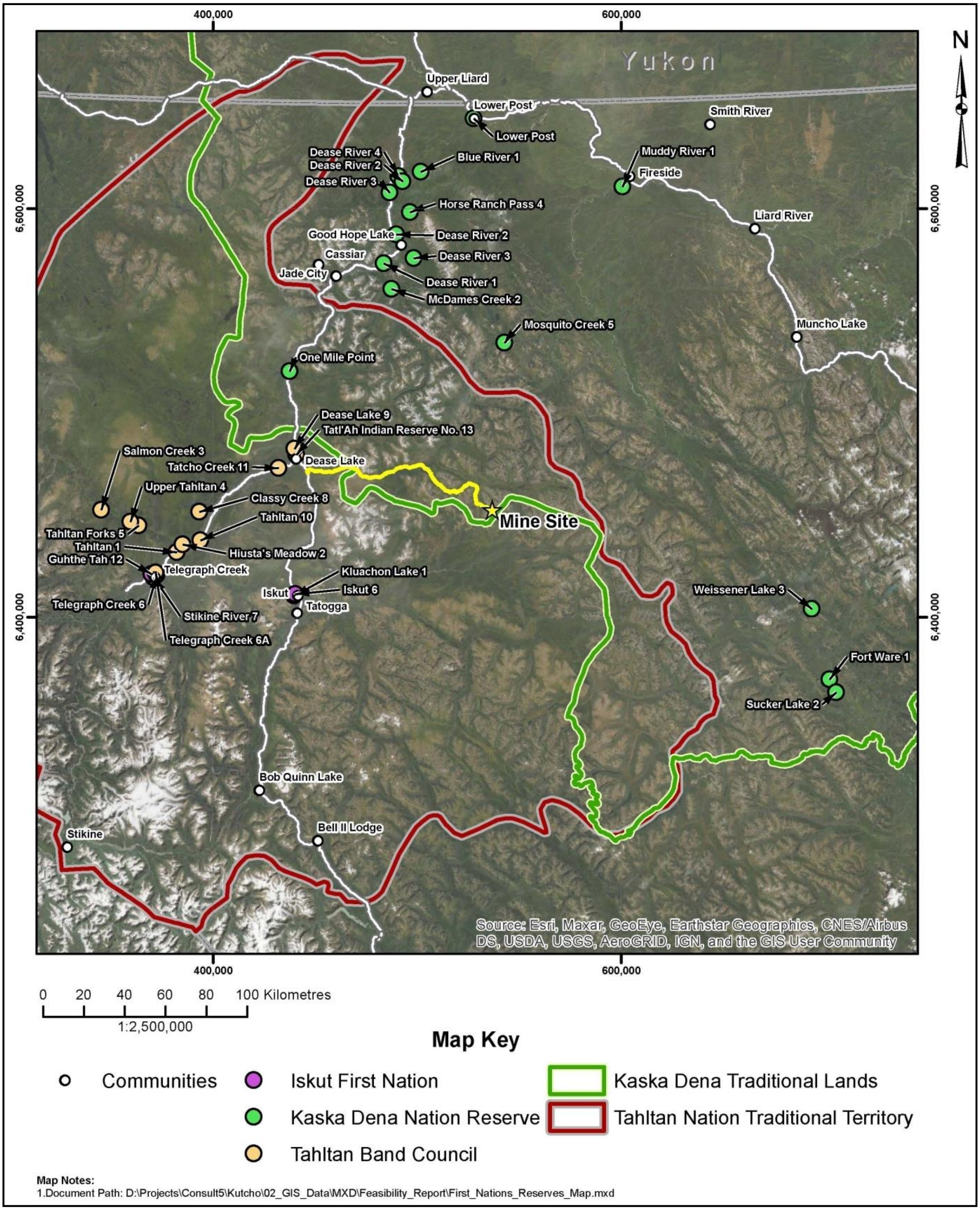


Figure 20-3: First Nations Reserves

20.4.3.1 Agreements with First Nations

Kutcho Copper has entered into several agreements with the Kaska Dena, and Tahltan Nations as set out in Table 20-5.

Table 20-5: Agreements with First Nations

Agreement	Kaska	Tahltan
Existing Agreements		
Opportunities Sharing Agreement	Existing	Existing
Communication Agreement	-	Existing
Exploration	Existing	Existing
Anticipated Future Agreements		
Economic Participation Agreement (EPA)	Initial discussions about the process	Initial discussions
Access Road Partnership	Initial discussions	Initial discussions

*20.4.4 **Community Considerations***

The Project is connected to various towns by road and to the Dease Lake airport and may affect these communities and services. The nearest towns and the airport are described in more detail below.

20.4.4.1 Dease Lake

Dease Lake is an unincorporated place located on Highway 37, approximately 100 km from the Property (Figure 4-1). The town includes a school, small businesses, a hotel, and a Northern Lights College campus. According to the 2016 Canadian census, the population of Dease Lake was 335, up 10% from 2011. Economic activity is generated by tourists travelling to the Alaska Highway, ecotourism, hunting, wilderness activities, and mineral exploration.

20.4.4.2 Dease Lake Airport

Dease Lake airport is the nearest airport to the Project and is frequently used as a base by fixed-wing aircraft and helicopters attending the project site. Airport construction and upgrades were undertaken in 2021 to renew the runway, expand the apron, and revise the lighting system. Kutcho Copper maintains close contact with airport personnel during project operational periods.

20.4.4.3 Stewart

Stewart is a district municipality located at the head of the Portland Canal, approximately 390 km from Dease Lake on Highway 37. The community includes one school, hotels, and small businesses. According to the 2016 Canadian census, the population of Stewart was 401, down approximately 19% since 2011. The district's economy is supported by mining, logging, port operations and tourism.

20.4.4.4 Smithers

Smithers is located approximately 600 km southeast of Dease Lake along Highway 37. Smithers is the regional service centre for the Bulkley Valley, and it provides a range of commercial, business, administrative, recreational, and cultural services. In 2016, the population of Smithers was 5,401, virtually unchanged from the 2011 census.

20.4.4.5 Terrace

Terrace is located on the Skeena River, approximately 572 km by road (Highway 37) southwest of Dease Lake. Terrace is a retail, service, and transportation hub for northwestern BC. Terrace has a rail service, highways, a regional airport, and a hospital. The city's population increased by a few hundred to 13,363 in the 2016 census.

20.4.5 Regulator Considerations

20.4.5.1 Federal

Kutcho Copper met with the Canadian Environmental Assessment Agency (CEA Agency) (now Impact Assessment Agency Canada (IAAC)) on 24 August 2018, to introduce the company, the management team, and to re-introduce the Project. A follow-up meeting on 28 November 2018, provided an update on the Project. In addition, the CEA Agency was part of a call with Kutcho Copper, BC EAO and the Kaska and Tahltan Nations to discuss aspects of the Project Description. It is anticipated that future consultation would also include the following federal government ministries:

- Natural Resources Canada
- Environment Canada
- Department of Fisheries and Oceans Canada
- Health Canada
- Transport Canada.

20.4.5.2 Provincial

Kutcho Copper met with BC EAO on February 15, 2018, and again on 23 August 2018, to re-introduce the Project. Correspondence has continued through email and telephone discussions since that time to receive direction. An additional meeting was held on 8 February 2019 to discuss the amended legislation and how it may affect the Project's process. BC EAO was also part of a call with Kutcho Copper, the CEA Agency and the Kaska and Tahltan Nations on 8 March 2019, to discuss aspects of the Project Description.

In 2021, Kutcho Copper restarted monthly meetings with EAO and is working to restart the EA process towards the end of 2021 or the beginning of 2022. Documentation requested by EAO to restart the process is being prepared to meet this timeline.

It is anticipated that ongoing consultation would also include the following provincial government agencies:

- Ministry of Energy, Mines and Low Carbon Innovation
- Ministry of Indigenous Relations and Reconciliation (MIRR)
- Ministry of Forests, Lands, Natural Resource Operations and Rural Development (FLNRORD)
- Ministry of Environment and Climate Change Strategy
- Ministry of Transportation and Infrastructure (MOTI)
- Ministry of Municipal Affairs and Housing.

20.4.5.3 Regional and Local Government

The previous Project proponents held several meetings in 2007 and 2012 with local and regional government parties to discuss and understand their issues and concerns, including highway traffic volumes and safety, and zoning. Kutcho Copper has met with several local governments to date and would maintain regular contact via meetings, phone, and email communications with local and regional government representatives, including the Regional District of Kitimat Stikine (RDKS), throughout the environmental assessment process.

20.4.6 Consultation Plan

The description of the Consultation Plan used by Kutcho Copper was prepared with input from the Kaska and Tahltan Nations as presented in the Project Description.

Kutcho Copper intends to work with Kaska Dena Nation and Tahltan Nation to prepare an Indigenous Consultation Plan that meets the Nations' engagement requirements as well as the obligations of achieving "free prior and informed consent" in consideration of activities specific to the environmental assessment process. The plan would be consistent with the EAO's "Guide to Involving Proponents when Consulting First Nations in the Environmental Assessment Process". The plan would embrace the spirit of Indigenous reconciliation embodied in:

- The federal "Principles Respecting the Government of Canada's Relationship with Indigenous Peoples"
- BC's "Draft Principles that Guide the Province of British Columbia's Relationship with Indigenous Peoples"
- The "United Nations Declaration on the Rights of Indigenous Peoples"
- BC's *Declaration on the Rights of Indigenous Peoples Act*.

The plan would reflect any commitments that come out of any historical or new agreements between Kutcho and Indigenous nations. Kutcho Copper is currently engaging with Kaska and Tahltan Nations to determine appropriate mechanisms to facilitate meaningful engagement and participation within the EA process.

A concept outline of the plan for the basis of discussion with Kaska and Tahltan Nations is being prepared. The plan is one of the documents required by the BC EAO to restart the environmental Assessment Process.

20.5 Notable Features of the Project Environment

20.5.1 Baseline Investigations

The Property has changed ownership several times since the initial discovery in 1968. Environmental and socio-economic investigations have been conducted on the Project area in different periods and at differing levels of rigour. The historical data, pre-1980s, is largely excluded from consideration as the methods and data validation do not meet current standards. The Project's most recent environmental and socio-economic baseline studies were initiated in 2005 and updated in 2012 and 2013 by Capstone Mining Corp. After acquisition from Capstone in 2017, Kutcho Copper started additional baseline studies in 2018, and these have continued through to 2021. These studies would complete the dataset required to support the preparation of an application to the BCEAO for an environmental assessment certificate and for permitting and represent a substantial dataset with a long period for the baseline.

20.5.1.1 Previous Baseline Studies

In 2005, a comprehensive environmental and socio-economic baseline study program was initiated to support an EA Application. Studies considered both the biophysical and human environment, including meteorology, air quality, hydrology, hydrogeology, metal leaching and acid rock drainage, aquatic ecology, fish and fish habitat, soils, vegetation, ecosystem mapping, wildlife, wetlands, archaeology, socioeconomics, land use, country foods and human health, and traditional use and traditional ecological knowledge. Baseline studies had a broad regional study area (RSA) and smaller local study area (LSA). The RSA covered approximately 305,000 ha and was located within the Stikine Ranges, which form a part of the larger Cassiar Mountains. The LSA covered approximately 23,000 ha and included the proposed mine site area and a 2 km wide corridor (1 km on either side of the centre line) surrounding the proposed access road.

20.5.1.2 Current Baseline Studies

Table 20-6 outlines ongoing baseline studies.

Table 20-6: List of 2018–2021 baseline studies and status

Baseline discipline	Status
Physical context	
Climate and meteorology	Ongoing. Data table and summary complete.
Air quality	Ongoing.
Groundwater resources	Ongoing.
Surface water resources	Ongoing.
Terrain, soils and surficial materials	Ongoing.
Noise and vibration	Draft complete.
Geochemistry	Draft complete, ongoing for open pit.
Biological context	
Fish and aquatics	Draft complete.
Terrestrial ecosystems and vegetation	Draft complete.
Wetlands	Ongoing.
Wildlife and wildlife habitat	Draft complete, additional winter survey in 2022 to confirm long-term trends. Habitat Suitability modelling in 2022.
Socio economic context	
Economics	Scheduled for 2022, to use new census data.
Social	Scheduled for 2022, to use new census data.
Land and resource use	Scheduled for 2022, to use new census data.
Archaeology	Additional TMF area to be surveyed in 2022. First Nations have approved application.
Human health	Baselines complete, modelling 2022.
Visual quality	Draft complete.

20.5.2 Description of the Project Environment

20.5.2.1 Geographical Setting

The Kutcho property is located approximately 100 km east of Dease Lake in the Liard mining division of northern BC. The property is located within the Cassiar Mountains, just to the north of the continental divide between the Arctic and Pacific catchments. The mine site and mine-related discharges will all be located within the Kutcho Creek catchment, a major tributary of the Turnagain River. The Turnagain River generally flows east and north to join the Kechika River. The Kechika River is a tributary of the Liard River, which flows to the Mackenzie River, emptying into the Arctic Ocean in Canadian waters along the coast of the Northwest Territories.

The area is moderately rugged with elevations ranging from 1400 masl to 2200 masl. Most of the area is alpine with the tree line at approximately 1500 masl. Snow cover can persist for nine months of the year, particularly on shady north-facing slopes. Winters are long, cold and dry, while summers are short, cool and moist. Some of the valleys contain permafrost, especially above 1200–1400 masl. There are no glaciers or permanent ice fields within the mine site area.

20.5.2.2 Climate and Meteorology

The climate is characterized by prevailing westerly winds, rain-shadow effects and evenly distributed precipitation throughout the year. Winters are long and cold while the summer growing season is cool and short, with a mean annual temperature range between -0.7°C and -3°C. The warmest and coolest months on average are July (10.0°C) and January (-12.6°C). Measured monthly average air temperatures were as low as -18.6°C in December 2008 and as high as 14.0°C in August 2008. The extreme minimum hourly temperature of 41.0°C was recorded in January 2008, while the extreme maximum temperature of 28.2°C was recorded in July 2009. With sub-zero temperatures from October to May, precipitation accumulates predominantly as snow, where it remains until spring. Temperature inversions are known to occur at the site.

Meteorological monitoring was conducted on-site using two automated weather stations (Kutcho Airstrip, 1247 masl; Kutcho Deposit Station, 1425 masl) between 2005 and 2015, supplemented by manual snow surveys between 2005 to 2015. A monitoring station was re-established at the airstrip in May 2018. These two automated weather stations measure air temperature and humidity, wind speed and direction, snow depth, total precipitation and evaporation, and have operated with periodic interruptions since 2005. Data from the local climate stations are supplemented by regional stations operated by the BC Government, BC Hydro and others.

The measured mean annual precipitation at the site is 816 mm (for the Deposit station), comprised of approximately 50% rainfall and 50% snowfall. Both snowfall and rainfall increase locally with elevation at the Project site due to orographic cooling. Annual precipitation is estimated to range from 538 mm in a 1 in 100 dry year to 1,065 mm in a 1 in 100 wet year. The maximum snow depths recorded at the Deposit Station were 161 cm on 6 February 2009, and 160 cm on 11 April 2008. Snowmelt at the Project site typically begins in late April or early May and lasts for four to six weeks. Annual open water or lake evaporation was estimated to be in the range of 288 to 319 mm for the Andrea Creek catchment.

The mean annual temperature at the site is -1.5°C. July is the warmest month on average (10.0°C) and December is the coldest (-12.6°C). Measured mean daily extremes temperatures have ranged from -42.2°C to 33.4°C.

Between October 2008 and September 2009, the average wind speed observed at the Deposit Station was 2.1 m/s with maximum instantaneous wind gusts of approximately 18 m/s (65 kmph).

20.5.2.3 Hydrology

The Project is situated in the Stikine Ranges of the Cassiar Mountains in north-central BC, A freshet snowmelt-dominated streamflow regime characterizes local drainages, with peak flows typically occurring in late May or early June. Streamflow then recedes to summer low flow values that sporadic rainfall event-driven peaks may punctuate. Finally, stream discharges reduce to their annual minimums in March or early April before rising rapidly again with the arrival of warmer temperatures and freshet flow conditions.

The main Project development area lies south of, and drains to, Andrea Creek or its tributaries the Sumac, Playboy, Twenty and Rusty creeks. Andrea Creek in turn discharges to Kutcho Creek about 6 km (measured in straight line) downstream of the Project area. Kutcho Creek flows ultimately discharge to the Arctic Ocean via the Turnagain, Kechika, Liard and Mackenzie rivers (and not west to the Pacific Ocean).

Local flow data are provided by a mixture of continuous and spot flow measurement stations at eight locations on Kutcho Creek, Andrea Creek and its tributaries (Figure 20-4). The earliest flow data for the site is from 2006 and continues to present, though there are gaps in data collection at all stations. The most complete records available are from the continuously monitored Andrea and Kutcho Creek stations. Local flow data are supplemented by data from regional stations operated by Environment Canada.

Mean annual runoff is 741 mm (at the AC-7.4 Andrea Creek station). Spring freshet typically starts in May and peaks in June. Winter flows are typically negligible to low, with some near-zero flows recorded in Andrea Creek tributaries. At the confluence of the Kutcho and Andrea creeks, the catchment area of Kutcho Creek is 174 km^2, or approximately five times larger than the area of Andrea Creek catchment in the vicinity of the proposed mine area (approximately 35 km^2 at the confluence between the Twenty and Andrea creeks).

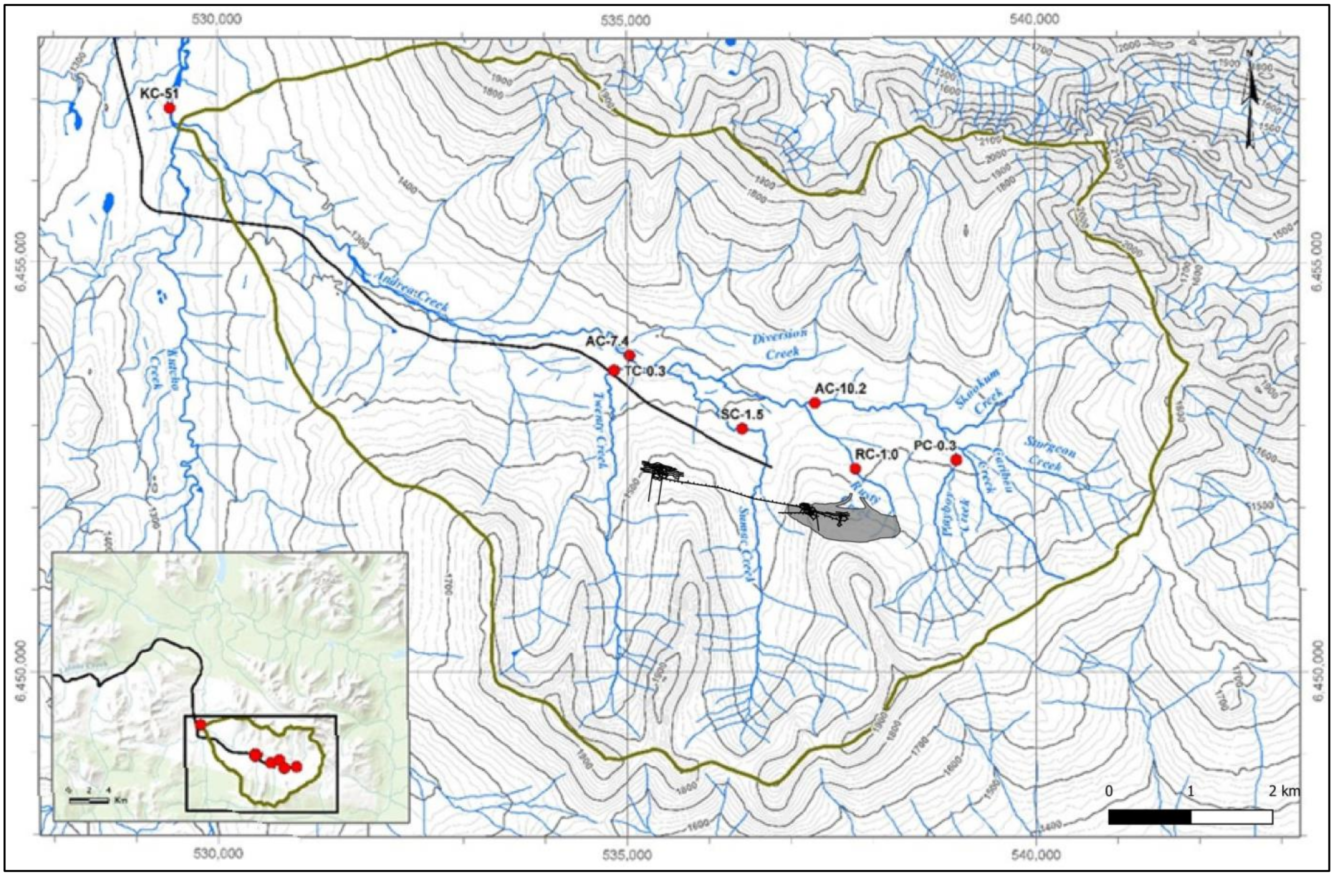


Figure 20-4: Kutcho Project Flow Monitoring Stations (adapted from Lorax, 2018)

20.5.3 Water Quality

Baseline water quality in both Andrea Creek, and Kutcho Creek below the confluence with Andrea Creek, is relatively pristine and reflective of an unimpacted headwater tributary (Table 20-7 and Table 20-8). Stream pH values exhibit near-to-neutral, to slightly alkaline signatures. Dissolved organic carbon (DOC) concentrations are typically low at the Project site (i.e. 1–2 mg/L; refer to Table 20-7 and Table 20-8), although they increase during the open water period (up to 4 mg/L). While DOC has the potential to ameliorate metal toxicity in lotic environments, baseline DOC measurements at the Project site screen as being 'Low' on the water quality guideline classification scheme for copper.

Hardness values in Andrea Creek vary seasonally, being as low as 30–40 mg/L during freshet months (May, June), but values increase in magnitude as flows decrease through summer and into winter, as groundwater contributes proportionally more to total flow in the stream. Sulphate and metals such as calcium and magnesium show similar seasonal patterns to hardness; for example, sulphate values in Andrea Creek range from 2 mg/L to 8 mg/L during freshet, but values as high as 16 mg/L are evident during the winter low flow period.

Table 20-7: Average monthly water quality for Upper Andrea Creek (AC 10.2; all values in mg/L)

Parameter	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec
Hardness	85	93	96	100	46	35	52	60	58	63	78	86
TSS	1.5	0.9	1.5	1.1	2.6	1.8	1.2	1.5	2.3	1.6	2.7	1.5
TDS	101	112	110	108	56	49	65	73	73	79	85	94
Chloride	0.3	0.3	0.3	0.6	0.3	0.3	0.3	0.3	0.3	0.3	0.3	0.3
Sulphate	12	13	14	14	4	5	8	8	9	10	10	13
Ammonia	0.0025	0.0025	0.0025	0.0188	0.0043	0.0029	0.0030	0.0025	0.0040	0.0085	0.0037	0.0025
Nitrate	0.038	0.030	0.043	0.037	0.013	0.003	0.003	0.003	0.003	0.004	0.111	0.024
Nitrite	0.00050	0.00050	0.00127	0.00050	0.00050	0.00050	0.00050	0.00050	0.00050	0.00050	0.00050	0.00050
D-As	0.00049	0.00054	0.00043	0.00048	0.00028	0.00035	0.00046	0.00046	0.00042	0.00044	0.00051	0.00058
T-As	0.00050	0.00053	0.00041	0.00052	0.00032	0.00051	0.00049	0.00046	0.00047	0.00045	0.00048	0.00117
D-Cd	0.000008	0.000008	0.000016	0.000014	0.000013	0.000020	0.000013	0.000013	0.000012	0.000017	0.000010	0.000008
T-Cd	0.000008	0.000007	0.000007	0.000012	0.000014	0.000021	0.000015	0.000011	0.000018	0.000015	0.000011	0.000007
D-Co	0.00005	0.00005	0.00008	0.00005	0.00005	0.00007	0.00005	0.00007	0.00006	0.00007	0.00005	0.00005
T-Co	0.00005	0.00005	0.00008	0.00005	0.00007	0.00016	0.00006	0.00007	0.00006	0.00007	0.00005	0.00005
D-Cu	0.000238	0.000250	0.000343	0.000270	0.000937	0.000830	0.000576	0.000542	0.000658	0.000543	0.000550	0.000300
T-Cu	0.000317	0.000250	0.000370	0.000250	0.001227	0.001198	0.000695	0.000689	0.000796	0.000571	0.000460	0.000250
D-Fe	0.010	0.005	0.010	0.005	0.019	0.023	0.009	0.012	0.014	0.010	0.009	0.005
T-Fe	0.010	0.005	0.010	0.005	0.074	0.204	0.032	0.022	0.037	0.023	0.009	0.005
D-Mn	0.00009	0.00015	0.00013	0.00021	0.00177	0.00081	0.00032	0.00041	0.00041	0.00029	0.00024	0.00010
T-Mn	0.00013	0.00024	0.00024	0.00021	0.00458	0.00749	0.00178	0.00077	0.00162	0.00097	0.00046	0.00012
D-Hg	0.0000038	0.0000025	0.0000038	0.0000025	0.0000033	0.0000034	0.0000036	0.0000035	0.0000042	0.0000041	0.0000035	0.0000025
T-Hg	0.0000038	0.0000025	0.0000038	0.0000025	0.0000033	0.0000039	0.0000037	0.0000044	0.0000042	0.0000041	0.0000035	0.0000025
D-Ni	0.00025	0.00025	0.00033	0.00025	0.00045	0.00049	0.00027	0.00029	0.00041	0.00033	0.00030	0.00025
T-Ni	0.00025	0.00025	0.00033	0.00025	0.00059	0.00088	0.00039	0.00042	0.00047	0.00035	0.00025	0.00025
D-Se	0.00022	0.00024	0.00034	0.00028	0.00016	0.00015	0.00016	0.00013	0.00021	0.00023	0.00023	0.00023
T-Se	0.00023	0.00026	0.00030	0.00028	0.00016	0.00014	0.00017	0.00017	0.00022	0.00024	0.00023	0.00022
D-Zn	0.0011	0.0010	0.0023	0.0010	0.0026	0.0027	0.0019	0.0020	0.0018	0.0019	0.0012	0.0009
T-Zn	0.0013	0.0015	0.0018	0.0015	0.0030	0.0030	0.0023	0.0025	0.0025	0.0025	0.0020	0.0015

Notes: Assumed discharge window for Upper Andrea Creek highlighted light grey (i.e. May-November). Data for monitoring stations AC-12.1, AC-10.2 and AC-9.7 used to compute receiving stream.

Table 20-8: Average monthly water quality for Kutcho Creek below Andrea Creek confluence (KC-51; all values in mg/L)

Parameter	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec
Hardness	84.2	96.0	98.2	102.3	49.9	57.4	66.3	87.0	83.4	87.2	96.0	96.6
TSS	1.5	1.2	1.5	0.8	5.6	1.5	1.6	1.5	1.5	1.5	1.5	1.5
TDS	101.8	119.7	108.3	110.3	65.5	70.5	76.3	104.7	104.3	103.9	107.3	108.5
Chloride	0.25	0.25	0.25	0.25	0.25	0.25	0.25	0.25	0.25	0.25	0.25	0.25
Sulphate	11.4	15.1	13.5	15.1	5.7	8.5	10.1	12.7	13.0	12.9	12.6	15.2
Ammonia	0.00323	0.00333	0.00250	0.00250	0.00250	0.00250	0.00250	0.00397	0.00423	0.00250	0.00423	0.00250
Nitrate	0.0586	0.0641	0.0786	0.0679	0.0218	0.0038	0.0034	0.0046	0.0025	0.0122	0.0466	0.0451
Nitrite	0.0005	0.0005	0.0005	0.0005	0.0005	0.0005	0.0005	0.0005	0.0005	0.0005	0.0005	0.0005
D-As	0.000110	0.000120	0.000153	0.000130	0.000117	0.000205	0.000175	0.000190	0.000200	0.000174	0.000157	0.000120
T-As	0.000113	0.000133	0.000155	0.000147	0.000147	0.000203	0.000205	0.000203	0.000213	0.000180	0.000150	0.000145
D-Cd	0.0000078	0.0000033	0.0000075	0.0000034	0.0000082	0.0000103	0.0000062	0.0000101	0.0000074	0.0000124	0.0000103	0.0000025
T-Cd	0.0000088	0.0000025	0.0000059	0.0000025	0.0000090	0.0000081	0.0000071	0.0000069	0.0000128	0.0000065	0.0000110	0.0000042
D-Co	0.000050	0.000067	0.000083	0.000050	0.000050	0.000100	0.000050	0.000079	0.000075	0.000072	0.000050	0.000050
T-Co	0.000050	0.000050	0.000083	0.000050	0.000103	0.000100	0.000050	0.000079	0.000075	0.000072	0.000050	0.000050
D-Cu	0.000218	0.000287	0.000333	0.000247	0.001128	0.000777	0.001093	0.000737	0.000983	0.000821	0.000557	0.000300
T-Cu	0.000413	0.000250	0.000357	0.000250	0.001433	0.001077	0.001220	0.000777	0.001008	0.000883	0.000617	0.000250
D-Fe	0.0125	0.0067	0.0117	0.0050	0.0162	0.0168	0.0115	0.0119	0.0243	0.0117	0.0117	0.0050
T-Fe	0.0173	0.0050	0.0200	0.0050	0.1310	0.0570	0.0458	0.0186	0.0253	0.0140	0.0190	0.0240
D-Mn	0.002258	0.000250	0.001498	0.000480	0.000935	0.000910	0.000584	0.000646	0.000757	0.000648	0.001104	0.000140
T-Mn	0.002717	0.000273	0.002706	0.000500	0.007833	0.002448	0.002132	0.001375	0.001015	0.001286	0.001912	0.001330
D-Hg	4.38E-06	2.50E-06	4.17E-06	2.50E-06	4.17E-06	4.17E-06	3.75E-06	3.93E-06	4.38E-06	4.17E-06	4.17E-06	2.50E-06
T-Hg	4.38E-06	2.50E-06	4.17E-06	2.50E-06	4.17E-06	4.17E-06	3.75E-06	3.93E-06	4.38E-06	4.17E-06	4.17E-06	2.50E-06
D-Ni	0.00025	0.00033	0.00053	0.00025	0.00061	0.00129	0.00044	0.00077	0.00082	0.00077	0.00049	0.00025
T-Ni	0.00025	0.00025	0.00055	0.00025	0.00094	0.00157	0.00070	0.00090	0.00087	0.00077	0.00045	0.00039
D-Se	0.000348	0.000261	0.000275	0.000262	0.000169	0.000234	0.000154	0.000180	0.000266	0.000222	0.000222	0.000266
T-Se	0.000206	0.000248	0.000287	0.000282	0.000168	0.000306	0.000104	0.000189	0.000296	0.000257	0.000214	0.000225
D-Zn	0.00050	0.00067	0.00117	0.00050	0.00095	0.00185	0.00063	0.00124	0.00175	0.00094	0.00150	0.00050
T-Zn	0.00075	0.00150	0.00150	0.00150	0.00162	0.00183	0.00150	0.00150	0.00148	0.00128	0.00083	0.00150

Notes: Assumed discharge window for Kutcho Creek is January to December.

20.5.4 Hydrogeology

Hydrogeological investigations have been carried out within the Kutcho Project area since 1986. The primary hydrogeological units in the project area comprise surficial unconsolidated soils, fractured bedrock and deeper bedrock. Hydraulic test datasets are available for the different lithologies across the Project area.

There is a groundwater monitoring network across the Project area and groundwater levels have been measured across the Project area since 2006. Most of the groundwater monitoring data are captured within annual baseline hydrogeology reports between 2006 and 2014. However, no measurements were undertaken in 2015, 2016 and 2017; and the more recent data (from 2018 and 2019) were not captured in annual reports.

The bedrock in the project area (Figure 20-5) is separated into five hydrogeological units (argillite and limestone, conglomerate, tuff-argillite, metagabbro and rhyolitic lapilli tuff) within four geographical areas: the North Valley, Infrastructure Area, Kutcho Deposit Area and West Valley. Measured hydraulic conductivity values ranged over six orders of magnitude (5.1×10^{-11} m/s to 1.0×10^{-5} m/s). Overburden deposits, consisting of glacial till, had a narrower range of measured hydraulic conductivities (generally ranging between 1.0×10^{-8} mis to 9.0×10^{-6} m/s) with an average (geometric mean of 2.7×10^{-7} m/s) that was comparable to the overall average of bedrock K (geometric mean of 1.25×10^{-7} m/s). The conglomerate, tuff-argillite and quartz feldspar crystal tuff units of the Infrastructure Area and West Valley showed K values approximately one order of magnitude higher than the argillite and limestone unit and rhyolitic tuff units of the North Valley and Kutcho Deposit areas, respectively.

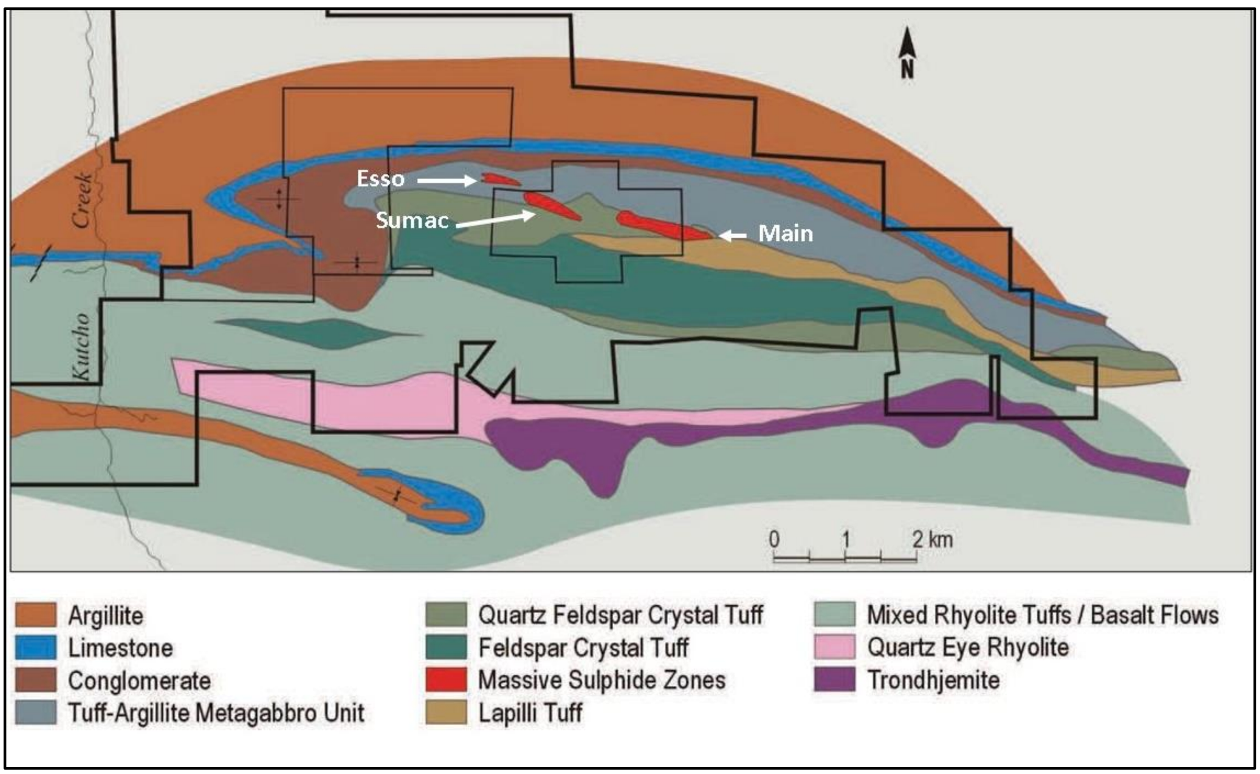


Figure 20-5: Surficial Geology of the Kutcho Project Area
Source: Wardrop, 2007

Depth to groundwater across much of the Project area is relatively shallow (from near surface/artesian to less than 15 mbgl). However, depth to groundwater appears to extend as deep as 30–40 m beneath sections of the Main deposit, depending on topography. Groundwater levels are seasonally variable, with the lowest groundwater elevations occurring just prior to freshet in late April/early May, while the highest groundwater elevations generally occur in late May/June during freshet.

There are notably large fluctuations in groundwater levels at some locations following precipitation events. Figure 20-6 illustrates groundwater elevations between March 2012 and October 2014 (from ACL-KUT-01 at Esso deposit). While these appear somewhat anomalous given the relatively low hydraulic conductivities documented across the stratigraphy, there is no solid evidence to counter the validity of the data. If these relatively large fluctuations are indeed indicative of aquifer response, there is potential for significantly elevated seasonal groundwater inflows into mine workings, and this possibility should be incorporated into the dewatering designs.

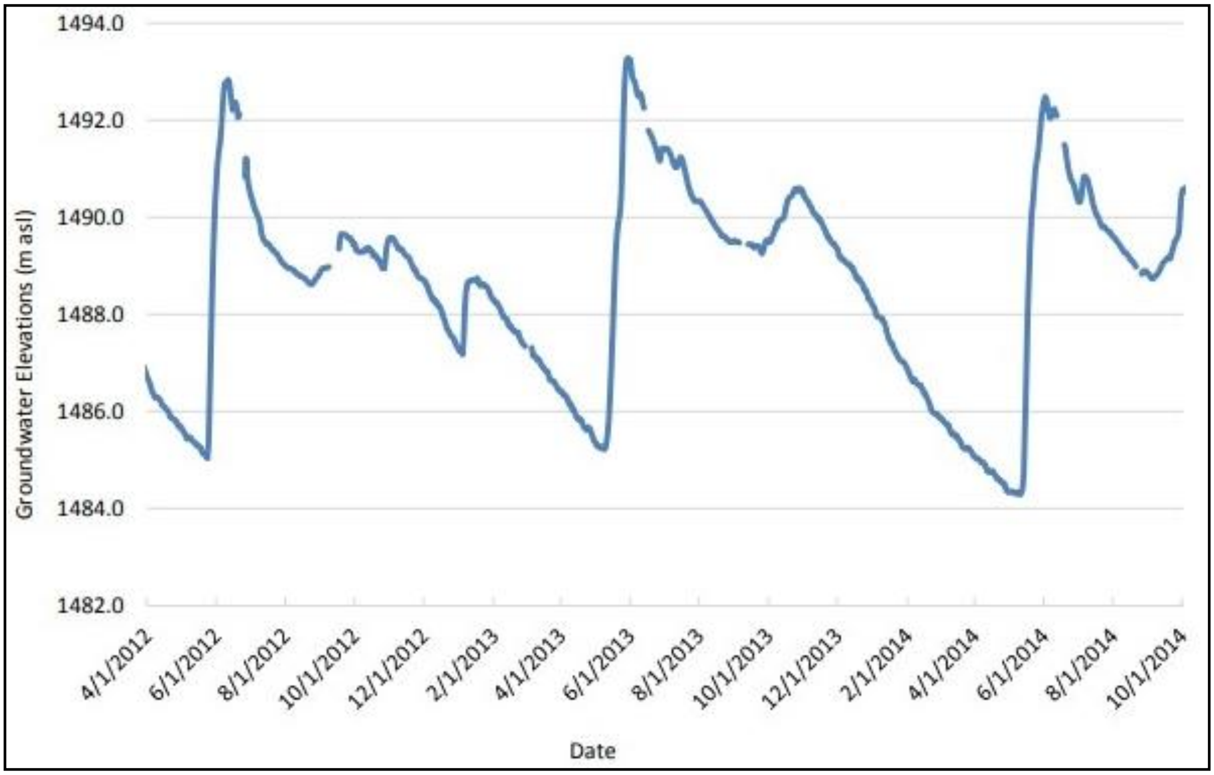


Figure 20-6: Groundwater elevation time series at monitoring well ACL-KUT-1
(Source: Cassiar Geoscience, 2014)

The extent to which structural lineaments facilitate hydraulic connection between surface water and the deep bedrock aquifers is uncertain. The hydraulic test data, on average, do not suggest significantly higher hydraulic conductivities across the fault zones where tested. It is, however, noted that there are many drillholes within the deposit areas and these may play a role in facilitating snowmelt/surface water infiltration to depth. The hydraulic connection between surface waters and deep bedrock aquifers is also reasonably supported by the relatively low TDS of the groundwater, suggesting either periodic recharge of the deep bedrock aquifer with fresh surface water or relatively short groundwater flow pathways.

Groundwater quality data for the site are collated in baseline reports from 2006 to 2014 (Table 20-9); while raw data are also available (but not captured in reports) for 2018 and 2019. The groundwater quality can generally be classified as a calcium-magnesium-bicarbonate (Ca-Mg-HCO_3) type. In general, the groundwater has near-neutral pH and relatively low concentrations of total and dissolved metals. Groundwater from the deposit area showed influence of sulphate and magnesium, perhaps due to the host geology (rhyolitic lapilli tuff) that is common to these well locations. Dissolved metal concentrations are generally low and below the freshwater aquatic guideline. There are some naturally occurring exceedances specifically for cadmium and chromium. Elevated sulphate concentrations are believed to result from oxidation of sulphide minerals although only a few wells showed sulphate concentrations above guideline values.

Table 20-9: Physical parameters, concentrations of major anions and nutrients, and cyanides in groundwater

Parameter	Units	No. of samples	No. of samples below detection	Minimum	Maximum	Average(1)	Standard deviation
Physical parameters							
Hardness (as $CaCO_3$)	mg/L	112	0	28.4	2810	196.5	336
Colour, true	CU	82	57	<5	28.7	4.1	4.25
Conductivity	uS/cm	82	6	<2.0	992	347	196
pH	pH	112	0	6.52	9.44	8.08	0.427
Total dissolved solids	mg/L	112	4	<1	751	211	139
Total suspended solids	mg/L	112	4	<3	2120	252	415
Turbidity	NTU	112	11	<0.1	2640	255.5	476
Anions and nutrients							
Ammonia (as N)	mg/L	112	68	<0.005	7.0	0.30	1.14
Acidity (as $CaCO_3$)	mg/L	112	26	<1	202	18	46.9
Alkalinity, bicarbonate (as $CaCO_3$)	mg/L	58	11	<1	257	121	72.1
Alkalinity, carbonate (as $CaCO_3$)	mg/L	58	51	<1	37.8	2.4	6
Alkalinity, hydroxide (as $CaCO_3$)	mg/L	58	44	<1	202	31	64
Alkalinity, total (as $CaCO_3$)	mg/L	112	5	<0.005	257	127	59
Bromide (Br)	mg/L	112	109	<0.05	0.25	0.04	0.050
Chloride (Cl)	mg/L	112	95	<0.5	13.6	0.9	2.14
Fluoride (F)	mg/L	112	31	<0.02	1.04	0.17	0.211
Sulfate (SO_4)	mg/L	112	11	<0.005	384	37.0	71
Nitrate (as N)	mg/L	112	75	<0.001	0.176	0.015	0.0345
Nitrite (as N)	mg/L	82	64	<0.001	0.43	0.014	0.0576
Total Kjeldahl nitrogen	mg/L	112	43	<0.005	9.52	0.457	1.42
Total nitrogen	mg/L	112	37	<0.0066	9.52	0.455	1.42
Total phosphate (as P)	mg/L	112	2	<0.002	420	12.0	55.4
Total organic carbon	mg/L	112	14	<0.5	490	11.6	56.4
Cyanides							
Total cyanide	mg/L	41	20	<0.001	0.0088	0.0015	0.0020
Weak cyanide	mg/L	32	30	<0.001	0.001	0.001	0.0001
Free cyanide	mg/L	32	32	<0.005	<0.005	<0.005	4.43887E-19

(1) Average is calculated using half of the detection limit when the result was below it.

Groundwater recharge occurs primarily during the freshet (snowmelt period) and large rainfall events during the summer months. Recharge to the bedrock groundwater regime is mainly from infiltration through permeable faults and associated fractures transecting the site (particularly where the correlate with drainage channels) and leakage from the overlying overburden. Groundwater from across the Project area discharges to the Andrea and Sumac Creeks as well as to the various tributaries on the surrounding hills.

20.5.5 Air Quality

Air quality at the proposed mine site is typical of a pristine environment with background concentrations expected to be below all relevant provincial and federal air quality objectives. The highest dust fall measured was 0.31 mg/dm^2/day, the only reading above the detectable level. Dustfall levels at the other stations were generally below the detection limit of 0.10 mg/dm^2/day. Results from all the dustfall samples collected to date were well below the 1979 BC Pollution Control Objectives for Dustfall (1.7–2.9 mg/dm^2/day). Based on the dustfall monitoring results to date and recognizing the absence of any human activities in the area, it is confirmed that the air quality in the Project area is pristine.

20.5.6 ***Wetlands***

A total of 2,677 ha of wetlands, approximately 10% of the local study area, have been identified and mapped in the vicinity of the Project. Approximately 65% of the wetlands were mapped along the access corridor and 35% in the mine site area. A total of 22 wetland ecosystems have been identified. Most of the wetland classes identified are fens and alpine seepage sites.

20.5.7 ***Vegetation***

The mine site and airstrip are situated within the spruce-willow-birch biogeoclimatic zone, a subalpine zone characterized by white spruce and subalpine fir. Shrub-dominated ecosystems ranging from swamps and fens to dry colluvial scrub are well represented in the zone. At upper elevations, the landscape of this zone is characterized by scrub/parkland dominated by tall scrub birch and willows. Wetlands in the zone include white spruce and tall willow swamps, sedge fens, and sedge marshes.

Ten plant species identified as being of cultural and/or traditional significance to members of the Tahltan Nation have been identified. The majority are berry-producing species (e.g. blueberries, crowberry, stoneberry). No plants tracked by the BC Conservation Data Centre (BC CDC) or listed under the Species at Risk Act were identified during field surveys. Scrub birch/water sedge communities currently blue-listed in BC have been observed within the regional and local study areas; no other listed ecological communities with the potential to occur in the area have been observed during field surveys.

20.5.8 ***Wildlife***

Mountain caribou appear to be the most abundant mountain ungulate in the vicinity of the Project. The mine site and airstrip are located within the Spatsizi caribou herd range. The mine site access road runs along the boundary between the Tsenaglode and Horseranch caribou herds. Stone’s sheep, mountain goats and moose have also been observed. Mountain caribou, Stone’s sheep, and mountain goats are provincially blue-listed species of conservation concern.

Breeding songbird surveys have identified four provincial blue-listed species of conservation concern: short-eared owl (*Asio flammeus*), barn swallow (*Hirundo rustica*), red-necked phalarope (*Phalaropus lobatus*), and rusty blackbird (*Euphagus carolinus*). All but the red-necked phalarope are designated under the Species at Risk Act.

The mine site area appears to support minimal habitat for waterfowl, but the wetlands and lakes along the proposed access corridor support large numbers of migrating waterfowl and provide important breeding habitats. One waterfowl species of concern was detected: the provincial, blue-listed surf scoter (*Melanitta perspicillata*). The surf scooter is not designated under the Species at Risk Act.

20.5.9 ***Fisheries***

Fish communities consist mainly of bull trout (*Salvelinus confluentes*), rainbow trout (Oncorhynchus mykiss), mountain whitefish (*Prosopium williamsoni*), Arctic grayling (*Thymallus arcticus*), lake trout (*Salvelinus namaycush),* and longnose sucker (*Catostomus catostomus*). Bull trout dominate the fish communities of streams in the proposed mine site and receiving environment, and account for half of all fish captured from streams, lakes, and wetlands in the vicinity of the Project. Bull trout are blue-listed in BC, which means they are considered to be vulnerable to human activities and natural events. Bull trout have been identified as a species of special concern by the Committee on the Status of Endangered Wildlife in Canada but are not listed under the Species at Risk Act. Genetic studies showed there are no Dolly Varden (*Salvelinus malma*) in the study area.

20.5.10 *Species at Risk*

Table 20-10 provides a summary of the SARA species whose ranges coincide with the Property.

Table 20-10: List of Species at Risk

Scientific name	English name	Species at Risk Act Designation	BC status
Mammals			
Gulo gulo luscus	Wolverine, luscus subspecies	SC	Blue
Martes pennanti	Fisher	Not Assessed	No Status
Ursus arctos	Grizzly bear	SC	Blue
Rangifer tarandus pop. 15	Caribou (northern mountain population)	SC	Blue
Myotis lucifugus	Little brown myotis	E	Yellow
Myotis septentrionalis	Northern myotis	E	Blue
Ochotona collaris	Collared Pika	SC	Blue
Birds			
Asio flammeus	Short-eared owl	SC	Blue
Contopus cooperi	Olive-sided flycatcher	T	Blue
Phalaropus lobatus	Red-necked phalarope	Not Assessed	Blue
Euphagus carolinus	Rusty blackbird	SC	Blue
Hirundo rustica	Barn swallow	T	Blue
Melanitta perspicillata	Surf scoter	Not Assessed	Blue
Falco peregrinus anatum	Peregrine falcon, anatum subspecies	SC	Red
Chordeiles minor	Common nighthawk	T	Yellow
Podiceps auritus	Horned grebe	SC	Yellow
Phalaropus lobatus	Red-necked phalarope	SC	Blue
Fish, Amphibians and Reptiles			
Anaxyrus boreas	Western toad	SC	Yellow
Salvelinus confluentus	Bull Trout	SC	Blue

(Source: BC CDC 2018)

E = Endangered Species = Species facing imminent extirpation or extinction.

SC = Special concern = Species which may become threatened or endangered because of a combination of biological characteristics and identified threats.

T = Threatened = Species which are likely to become endangered if nothing is done to reverse the factors leading to their extirpation or extinction

Rare plants have been encountered in the broader project study area. None have federal status or are being considered for SARA status. Most of these species have occurrences in the LSA only, away from the proposed infrastructure areas. Some changes to the list are likely, as several species are not as rare as originally thought. The tracked or otherwise rare plant and lichen species reported from baselines studies include (the BC risk rank follows each species):

- Lichens: *Collema ceraniscum* (Blue), *Hypogymnia dichroma* (Blue), *Phaeophyscia adiastola* (Blue), Umbilicaria cinereorufescens (Red), Umbilicaria decussata (Blue)
- Liverworts: Asterella lindenbergiana (Red), Cephalozia pleniceps (Blue), Lophozia quadriloba (Blue), Tritomaria exsectiformis (Blue), Tritomaria polita (Blue)
- Mosses: Bryum calobryoides (Red), Cynodontium schistii (Blue), Orthotrichum pylaisii (Blue), Psilopilum cavifolium (Red), Ptychostomum schleicheri (Blue), Schistidium atrichum (Red), Scorpidium cossonii (Blue), Tortula leucostoma (Blue)
- Vascular plants: *Festuca brevissima* (newly discovered for BC, possibly rare. The BC CDC is not aware of this species in the province).

20.6 Environmental Considerations

Although Kutcho Copper has tried to eliminate as many potential environmental effects as possible by refining the mine design, minimizing the project footprint, and not encroaching on fish-bearing watercourses, some effects are inevitable. These potential effects are described in this section, along with proposed management measures. The key features that have been addressed through design changes include:

- Upgrading the site access road to minimize impact on water quality and natural resources, including run-off control and clear span bridge crossings
- Use of liquified natural gas for power generation as opposed to diesel, significantly reducing the generation of greenhouse gases and reducing the potential for fuel spills
- The tailings management facility (TMF) is designed to be a fully lined impoundment with the rockfill embankment comprising non-potentially acid generating (nPAG) waste rock and built using downstream construction methods
- On closure, the TMF would be capped with nPAG waste rock and revegetated
- Waste rock not required for construction would be placed on a stockpile, most of which would be backfilled into the pit during operations and revegetated
- Potentially acid generating (PAG) waste rock would be placed on temporary stockpiles, all of which would be rehandled back into the pit at the end of the mine life to reduce the risk of future acid generation or metal leaching
- No infrastructure is located within fish-bearing streams
- The water management system would divert non-contact water around proposed mine facilities
- Contact water would be collected for operational re-use or treated in a water treatment plant prior to discharge to the environment to ensure adherence to provincial and federal water quality guidelines
- At closure all buildings would be removed, disturbed lands rehabilitated, and the property returned to functional use according to as yet to be developed and approved reclamation plans and accepted practices at the time of closure.

20.6.1 Potential Effects and Management

20.6.1.1 Air Quality Management

The meteorological monitoring results indicate the seasonal presence of a temperature inversion at the mine site, inhibiting mixing and dilution of air contaminants. Potential effects from the mine site, access road and airstrip on air quality include elevated airborne particulates, sulphur dioxide, nitrogen oxide, fugitive dust, and greenhouse gases. Most of the dust and particulates would be generated from open-pit mining and rock and stockpile development.

Fugitive dust would be controlled by applying best management practices that are typical for a small mine site. The Project would also mitigate air quality effects via regular vehicle and equipment maintenance, high-efficiency diesel engines, and LNG for the power plant.

20.6.1.2 Surface Water Management

Water for the mine would be supplied from wells drilled within or near the mine site for potable water requirements. Potable water would be treated and stored separately at a facility located near the accommodation and office complex. To minimize water usage, dewatering of the underground workings and surface contact water collected within the mine site footprint would be the major water source for use in the process plant circuit, fire, fresh, and pump gland seals. Re-use of plant water would be maximized.

In the Feasibility Study design, contact water from within the mine site area would be captured and directed to small water management structures and then stored in the tailings management facility before treatment followed by re-use or release into the environment. The design removes the need for a large containment structure, thus reducing the footprint of the Project, and allows for treated water to be released in Kutcho Creek, below the confluence with Andrea Creek (some 10 km from the mine site). Where possible, water would be recycled and used in the process plant. Excess water would be directed to a water treatment plant, where it would be treated and then discharged to the Kutcho Creek catchment.

This would impact the timing of discharges to Kutcho Creek and may change streamflow quantity as discharge water volumes would be greater than during baseline flow conditions.

Potential effects on surface water quality include increases in total suspended solids, hydrocarbons and metal concentrations, and nutrient enrichment. Changes in total suspended solids could arise from dust-generating activities, construction activities and changes in creek flow regimes. Potential spills during fuel transport and handling could increase hydrocarbon concentrations. Explosives use and camp sewage effluent have the potential to result in nutrient enrichment. Groundwater seepage and ML/ARD from Project source areas and mine effluent can increase metal concentrations.

The Project would limit alteration to the pre-development drainage network and the volume of contact water to be managed. Mine design would consider reduction in water use, as well as the reduction in the volume and footprint area of reactive mine waste stored on surface. Underground mine stopes would be expeditiously backfilled with cemented rock backfill to reduce the length of exposure of mine walls and mine waste to oxygen. All contact water would be collected and used in mineral processing. Excess water would be treated and discharged as mine effluent to the receiving environment within the Kutcho Creek catchment in compliance with applicable federal and provincial discharge standards. An effluent discharge point within Kutcho Creek below the confluence with the Andrea Creek catchment was identified and would be confirmed through ongoing engineering work and the pending environmental assessment.

The general water management plan during mine operations includes the following components:

- Constructing non-contact or freshwater diversion ditches and berms around the proposed mine waste rock and tailings management facilities
- Building a collection system for mine contact water, including a contact water dam
- Pumping contact water to the tailings management facility and then to the water treatment plant
- Recycling a portion of the treated or untreated site contact water and process water for ore processing
- Discharging the treated water to the receiving environment within the Kutcho Creek catchment after the water quality meets the discharge criteria.

20.6.1.3 Groundwater

Project-related impacts to groundwater quantity may include alteration of groundwater levels, thus resulting in changes to groundwater flow paths and the spatial distribution and volume of groundwater recharge and discharge to surface streams and water bodies near the mine. In addition, these activities may increase or decrease water levels, thereby altering the extents of wetland areas and the baseflow regime of nearby creeks.

Potential impacts to groundwater quality may include increased concentrations of dissolved constituents, including metals, sulphate, and nitrogen species, and changes in pH and redox conditions. Explosives use may lead to the release of nitrogen compounds and the increase of nitrogen concentrations above background groundwater qualities observed in baseline data. Exposure of mine workings, tailings and waste rock to air causes mineral oxidation, lowering pH and solubilizing metals in surface water and groundwater. The operation of sewage and fuel facilities may introduce nutrients and hydrocarbons to the groundwater system. However, these impacts would be minimized by the implementation of best management practices. The site-wide groundwater monitoring network is positioned to establish baseline groundwater quality and quantity and monitor effects during operations. All water-related effects would be mitigated well upstream of provincial boundaries.

20.6.1.4 Metal Leaching and Acid Rock Drainage (ML/ARD)

Tailings Management Facility

The Kutcho tailings would be derived from the massive sulphide ore body. Based on mineralogical statistics derived from borehole core samples taken from the Main and Esso deposits, the ore contains a mix of sulphides, metals, silicates, and carbonates. Tailings would comprise approximately 80% pyrite and are strongly net-acid generating. Geochemical studies of rock samples undertaken to date have characterized mineralogy, sulphur content, acid-generating potential, and mineral leaching/acid rock drainage (ML/ARD) production. Further testing would be undertaken before the start of construction.

The tailings from the process plant would be dewatered and pumped to the tailings management facility for disposal.

Mine Rock Management

The waste rock from the mine operation includes PAG and nPAG waste rock generated from the open pit and underground mines. nPAG waste rock would be utilized as on-site construction material, including for the construction of the TMF embankment.

A temporary waste rock facility is required to store the PAG waste rock generated during the mining operation. This PAG waste rock would be gradually reduced over the mine life as it is used for backfilling of the open pit.

20.6.1.5 Fish and Fish Habitat Protection

The Project has the potential to impact fish and fish habitat through in-stream works, sedimentation, liquid discharges, blasting and vegetation clearing. Changes in contaminant and nutrient concentrations, water flow and water temperature could affect fish growth, survival, and reproduction. Project-related vegetation clearing, sedimentation and footprint could result in fish habitat alteration and loss. Fish mortality could arise from heavy equipment use, dewatering, blasting, liquid discharges and recreational fishing.

The current mine plan is designed to reduce the harmful alteration, disruption, or destruction of fish habitat. The TMF location was moved into the upper Playboy Creek catchment to avoid placing a conventional flooded impoundment on the mainstem of Andrea Creek and the fish-bearing reaches of Playboy Creek. Kutcho Copper would treat all mine site-impacted water to quality levels sufficient to achieve BC water quality guideline criteria in the receiving environment.

*20.6.2 **Terrestrial Ecosystems and Wetland Protection***

Project-related land clearing and mine construction are anticipated to result in the loss or alteration of terrestrial ecosystems, vegetation, and wetlands. Changes in air, water, and soil quality from dust deposition can potentially change ecosystem and wetland function. The Project will mitigate potential impacts on ecosystems, vegetation, and wetlands by applying appropriate management practices and environmental management plans and by avoiding infrastructure placement within wetlands where possible.

*20.6.3 **Wildlife Protection***

The Project would mitigate potential impacts on wildlife by applying appropriate management practices and wildlife management plans. This would include measures to minimize Project interactions with wildlife, conducting vegetation clearing within suitable timing windows, observing applicable best management practices for wildlife, and implementing a no-hunting policy for mine employees and contractors. No hunting would be allowed in or near the mine site or access road for the safety of both the public and mine personnel.

20.6.4 Topsoil Management

Subsoil and topsoil would be salvaged and stockpiled for reclamation of the mine site area. Areas affected by the overburden storage and disposal would be prepared by clearing and soil removal. Sediment control for disturbed areas would be established where necessary. Soils, including identifiable topsoil, would be removed from areas affected by mining activities before disturbance. The salvaged soil would be stockpiled for use as growth media in progressive and final reclamation of the mine site. Vegetation cleared for mine development would be incorporated into the salvaged soil to provide organic content. Soil storage facilities would be in areas away from traffic and protected from wind and water erosion by mulching and promoting vegetative growth. Runoff from these storage facilities would be directed to sediment ponds. Soil from roads and perimeter ditches may be stockpiled in adjacent windrows suitably protected from traffic and further disturbance.

20.6.4.1 Sewage Management

A modular, pre-packaged sewage treatment plant would be utilized on site. Treated sewage water would be discharged directly to the receiving environment in the Kutcho Creek catchment in compliance with Environmental Management Act permit limits.

20.6.4.2 Waste Management

A proposed on-site landfill may be constructed to dispose non-salvageable and non-hazardous solid waste that cannot be incinerated. In addition, the operation may generate waste products considered hazardous and these would be segregated and transported off-site for appropriate disposal at an approved disposal site.

A solid waste incinerator may be used on-site to incinerate materials such as paper, wood, and food waste. This would eliminate materials from the landfill waste stream and help control odours and food that may attract wildlife.

A land-farm may be constructed for the treatment of any hydrocarbon contaminated soil.

20.6.5 Potential Social and Economic Effects and Management

20.6.5.1 Land Use

Traditional use of lands and resources in the vicinity of the Project was documented for both the Kaska Dena Nation and Tahltan Nation. There are no communities, all-season roads, or permanent residences currently in the vicinity of the Project. The Project does not overlap with any provincial or national parks, protected areas, or historic sites, or any private, provincial, or federal titled or surveyed lands, nor overlap with any active or pending forestry tenures or forestry recreation sites. There are no Land Act tenures in the vicinity of the mine site and airstrip. A section of the access road overlaps a recreation reserve (map reserve designated under Section 16 of the Land Act, Crown Land File #8000849) covering Wheaton Lakes along the Turnagain River. There are no recreation sites or trails in the vicinity of the Project. However, the Project study area is a popular moose hunting area among resident and Indigenous hunters, a favourite destination for snowmobilers, a prime fishing area for both subsistence and sport anglers, and a source for mining high-quality jade.

Potential effects on the use of lands and resources include changes to the use of, access to tenures and public recreation (e.g. hunting, fishing, snowmobiling) and the quality of experience for the public recreationists due to noise from the Project and increased vehicular traffic.

Mitigation for non-traditional land use would include measures such as:

- Following recognized best practice standards for control of noise and dust
- Seeking input on use and access of tenures and the public and end land use objectives
- Implementing a reclamation and closure plan consistent with the end land use objective.

20.6.5.2 Human Health

Recreational users, hunters and trappers and Indigenous peoples may have the potential to experience adverse health effects resulting from consuming water, fish, wildlife, or vegetation that was affected by the Project. Water and country foods (fish, wildlife, plants) may be affected by changes to surface water quality and from metal-laden dust deposition on vegetation and soils in the vicinity of Project components. It is important to note that the assessment results conducted in 2006 and 2007 indicated no unacceptable risks to human receptors (toddlers or adults) from the consumption of moose, caribou, grouse, snowshoe hare, caribou weed, and crowberry.

Mitigation for human health impacts includes comprehensive management and mitigation measures for air quality, dust, noise, and water quality. Human health would be protected through the Project's compliance with relevant laws, regulations, permits, policies, and guidance.

20.6.5.3 Employment

Northwestern BC exhibits a more extensive dependence on primary resource industries, including mining, forestry, and fishing, than the rest of the province. The region is defined by several small, predominantly Indigenous communities, generally located along the north-south corridor of Highway 37. The larger centres of Smithers and Terrace, located along the east-west corridor of Highway 16, provide services, and supplies to much of the region. The region is further characterized by its remoteness. Communities are dispersed, and power supply, transportation and communication options are limited.

The regional mining industry (including jade and placer) currently constitutes a significant source of employment for the Highway 37 communities, Smithers, and Terrace. Current significant operating mines in the region include Brucejack and Red Chris.

Construction of the Kutcho Project would employ up to 550 workers. During operations, the Project is estimated to employ approximately 370 personnel and, in the stockpile, rehandle period approximately 230 personnel. The Project would provide local employment and local, provincial and federal tax revenues over a 10-to-12-year mine life.

The Project would provide direct and indirect employment and economic development opportunities for local communities, members of the Kaska Dena Nation, the Tahltan Nation, and residents located elsewhere in BC and Canada.

Potential project-related economic effects may include:

- Creation of employment and income opportunities during construction and operation
- Loss of employment and income opportunities at mine closure
- Creation of training, skills development, and work experience opportunities.

Potential project-related effects to the social environment may include:

- Changes in demographic characteristics of primary and secondary communities
- Changes in quality of life and health due to changes in income, work schedules, and social roles
- Pressure on community infrastructure and services due to changing population demands.

Changes in the availability of traditional foods, income and work schedules can impact the quality of life, social stressors, and health. Company policies would include equal opportunity, anti-discrimination, and health and safety measures to promote well-being. Examples include employee codes of conduct; zero tolerance for drug and alcohol use on the mine site; and ongoing education in areas such as financial skills, infectious diseases, family planning and parenting, cultural practices, and language. Kutcho Copper would work with local communities concerning employment opportunities, contract opportunities and business and training opportunities. Culturally appropriate counselling services may be sponsored on-site and in communities. Project shift rotations and flexibility around traditional activities, especially harvesting, may be developed with local guidance.

20.6.5.4 Housing and Accommodation

The Project would provide on-site accommodation, and it is not expected to directly affect accommodation or housing availability or prices in local communities. Indirectly, the Project may increase local and regional populations, increasing pressure on the use of, or access to, existing housing, infrastructure, and social services.

20.6.5.5 Heritage Protection

An Archaeological Impact Assessment (AIA) was conducted along the access road and within the proposed mine site footprint between 2006 and 2008 following the Heritage Conservation Act. In addition, in 2011, following a revision to the proposed road alignment in 2010, a Preliminary Field Reconnaissance (PFR) was conducted along the road corridor to assess areas not captured in the AIA conducted for the Project. Archaeological surveys conducted between 2006 and 2008 identified 11 archaeological sites within the vicinity of planned Project infrastructure. A new Heritage Permit was applied for in 2021 to accommodate the new Feasibility design, particularly for the area below the proposed Tailings Management Facility. This area was not completely surveyed in the past, nor was the route for the proposed discharge pipeline.

Project activities associated with the movement, excavation, or disturbance of soil have the highest potential for interactions between the Project and archaeological sites. The potential impact to archaeological and culturally significant sites is the loss of recorded and unrecorded cultural material because of development activities.

Archaeological sites would be protected either through site avoidance or mitigation where avoidance is not possible. Mitigation measures would be determined in consultation with the British Columbia Archaeological Branch, Kaska Dena Nation, and Tahltan Nation and carried out by a Project Archaeologist under an HCA permit. Mitigation may include systematic data recovery, detailed mapping, construction monitoring, site capping, and other means deemed appropriate. Once mitigation and associated reporting are completed, approval would be given by the BC Archaeology Branch and the Kaska Dena Nation and the Tahltan Nation to allow for impacts within the site boundaries. In addition, archaeological assessments may be required if Project developments are altered or expanded to include areas that have not been previously assessed.

The protection of as-yet-unknown protected archaeological resources, if present, from Project activities, would involve the implementation of the Project’s Chance Find Procedure and education of Project personnel regarding the protections afforded to archaeological sites. The Project’s Chance Find Procedure was developed for the exploration phase of the Project and would be updated in collaboration with Kaska Dena Nation and Tahltan Nation and implemented to assist construction teams in avoiding the loss of unrecorded artifacts and sites. The Chance Find Procedure would also take into consideration the Tahltan Archaeological Standards.

In addition to potential effects on physical heritage, the Project can interfere with the intergenerational transmission of knowledge, skills, stories, and indigenous identity. Measures used to mitigate effects to archaeological sites, biophysical resources, air, and water would also reduce potential effects on Indigenous peoples’ physical and cultural heritage. Additional measures may be considered through consultation with Indigenous Nations, such as the use of Indigenous place names in Project materials, employee cultural awareness training, and support for Indigenous-led cultural programs.

20.7 Closure and Reclamation

*20.7.1 **Closure and Reclamation Plans***

The Project would be developed, operated, and closed in such a manner as to leave the property in a condition that would mitigate potential environmental effects and restore the land as closely as is practical to its pre-mining land use and capability. Closure and reclamation activities would be carried out concurrent with mine operations wherever possible and consistent with a Reclamation and Closure Plan that would be developed. Final closure and reclamation measures required by First Nations and regulators would be implemented at the time of mine closure.

20.7.1.1 Progressive Closure and Reclamation Plan

During the mining operation waste rock and ore sorter rejects would be used as cemented back fill in the underground mining operation. This would reduce the amount of material stored on surface, reduce the storage capacity of the defunct mine workings and speed their flooding. The flooding would create anaerobic conditions that would significantly limit metal leaching and acid rock drainage (ML/ARD).

Kutcho Copper would start progressive closure during the operational life of the mine with a particular focus in the last three years of operation. During this time mining would have ceased, processing of the stockpile would be ongoing, and backfilling of the open pit would continue. It is anticipated that the majority of the nPAG rock would be used in the closure of the TMF and the backfilling of the open pit. The PAG rock would be placed into the open pit and enclosed by nPAG waste rock during the backfilling process.

20.7.1.2 Closure Plan

Closure after processing finishes is anticipated to take 12 months. Primary activities during closure would include:

- Removal of surface structures and concrete foundations and their placement in the open pit before backfilling. Hazardous materials would be removed from site to a certified disposal site
- Ripping and scarification of compacted soils and placement of overburden and topsoil covers
- Remediation of any contaminated soils
- Re-vegetation with native or approved non-native plant species selected for individual ecosites across the site
- Deactivation of site powerlines, mine site roads, removal of any bridges and culverts that have no further use, pipelines, fences, and communication lines
- nPAG rock placement underlying a multi-layer soil cover to promote revegetation over the TMF, open pit and waste rock facilities
- Water treatment and environmental monitoring
- Maintenance and repair of eroded or failed landforms and water management structures.

It is anticipated that Indigenous monitors would support environmental monitoring and implementation of mitigation measures during the closure phase. After mine closure, water from the reclaimed/rehabilitated TMF would be collected in water collection ponds and then pumped to the water treatment plant. The water would be discharged directly to the environment when the water quality meets the discharge criteria. The small water collection ponds may be decommissioned after a monitoring period specified in the associated authorizations and management plans.

20.7.1.3 Reclamation Plan

After closure, reclamation is expected to take a further 24 months. Under the BC Mines Act and the Health, Safety, and Reclamation Code for Mines in British Columbia, the primary objective of the reclamation plan would be to return, where practical, areas disturbed by mine operations to their pre-mining land use and capability. The Project area supports hunting, guide outfitting, trapping, and outdoor recreation. End land-use objectives would be determined in consultation with the Kaska Dena Nation and Tahltan Nation. The reclamation and closure plan would include the following goals:

- Long-term preservation of water quality within and downstream of mine operations
- Long-term stability of engineered structures

- Removal and proper disposal of structures and equipment that would not be required after the end of the mine life
- Long-term stabilization of exposed erodible materials
- The natural integration of disturbed areas into the surrounding landscape, to the extent practicable
- Integrated passive reclamation systems to eliminate long-term monitoring
- Establish a self-sustaining cover of vegetation consistent with the natural range and diversity of local ecosystems and wildlife habitat.

20.7.1.4 Current Environmental Liability

Kutcho does not own or operate the airstrip at the site and is not a current user of the Jade Boulder Road. Therefore, it has no liabilities associated with either. Kutcho does, however, operate a small camp at the airstrip, makes use of site roads, and has several exploration sites that would need to be remediated if the site were to be closed at this time. The MYAB permit issued for exploration over the site specifies what facilities are allowed and what activities Kutcho is permitted to undertake. It makes financial provision for closure and reclamation of the exploration facilities, sites, and roads. This bond is currently fully funded to the value of C$159,600. Additional funding of C$25,000 is required to remove structures left in 2021 exploration drilling and historical geochemical test work cribs.

No other environmental liability is currently evident or provided for at the site.

*20.7.2 **Mine Closure, Reclamation and Rehabilitation Cost Estimate***

This section presents the key aspects associated with the mine closure reclamation and rehabilitation approach and associated cost estimate prepared as part of the EIA. At this stage, the proposed actions are presented at a conceptual level, considering existing information and assumptions. The mine closure rehabilitation process should be seen as a dynamic process, that should continuously evolve and mature with the development of the site as well as with and best practices and technical improvements on the mining industry. As the mine activities evolve, closure and rehabilitation planning and approach would be refined and explored, gradually increasing in detail and complexity.

With evolution and refinement of closure and rehabilitation key objectives that support the definition of strategies and main activities would be considered. For the Project, the following objectives are considered key:

- Compliance of laws and regulations
- Management of potential hazards considering avoidance, elimination and mitigation measures
- Geochemical and geophysical stability of the area
- Recovery of the environment characterized by wildlife habitat, revegetation, water and air quality to the levels required
- Transition of the area to the land use agreed with stakeholders through a participative process towards the achievement of self-sustained process.

The mine closure rehabilitation process is structured in three main stages, identified as:

- Pre-closure Stage: Characterized by the performance review and design of detailed programs, plans, studies and inventories, required for the reclamation and rehabilitation of the area.
- Closure Stage: Characterized by the implementation of the main activities associated with the decommissioning, deactivation and adequacy of assets, management of potential hazards, implementation of actions to support both safety, environmental and social aspects towards closure and transition to the defined land use.

- Post -closure monitoring and maintenance Stage: Characterized by activities associated with the maintenance of the area as well as short and long-term actions required as well as the performance of monitoring and associated evaluation of the success of the measures implemented.

As part of the best practices, progressive reclamation and rehabilitation measures will be promoted during the operations, through the Life of Mine. Additionally, the site will document the development of the site in terms of updates in disturbance areas, inventories, background and additional information monitoring and studies developed through the life of mine and progressive reclamation measures implemented.

The information would be updated on a yearly basis and the results presented in the Annual Reclamation Reports. The Mine Closure Plan and its associated cost estimate would be updated every five years.

For the Pre-closure stage, the project considers the review of background information and performance of characterization studies such geotechnical and hydrogeochemical investigation, soil and groundwater investigation, modelling, development of programs required to support closure and social related studies. The cost estimate to these activities were based on an assumption in terms of hours required and market consultancy rates.

For reclamation planning, the Project is broadly split into key reclamation units.

20.7.2.1 Open Pit

The open pit would be backfilled during operations as part of the operational mine activity. During that operation life all PAG waste stored outside of the pit will be returned to the pit and further covered with nPAG. During the reclamation period the backfilled pit would be contoured and surface slope drainage control added. The waste rock would then be covered with subsoil, topsoil and revegetated. The PAG backfilled waste rock would be placed below the pit's decant point to ensure it remains anaerobic to minimise ML/ARD production. Water from the reclaimed area would be diverted to the CWP to allow for sediment separation, and then if the water is of suitable quality, it would be discharged to the environment. If the water requires treatment, it would be sent to the water treatment plant before release to the environment.

20.7.2.2 Underground Mine

Once disposal of underground equipment is complete, the portal entrance and surface raise openings would be permanently sealed with concrete bulkheads. Monitoring data collected during operations would confirm groundwater quality and guide mitigations associated with seepage from underground workings during closure.

20.7.2.3 Tailings Management Facility

A multiple-layer soil cover would be placed over the top of the thickened tailings at mine closure to minimize water infiltration and maximise runoff and to provide an oxygen diffusion barrier to minimize the influx of oxygen. The cover system, from first placed to surface, consists of:

- A low permeability nPAG waste rock layer. This would be placed over the tailings and the key design objective of the low-permeability layer is to maintain a high degree of saturation under all conditions. This objective appears achievable based on the proposed current cover design and given the Project site's meteorological, hydrological, hydrogeological, and ground conditions.
- Multiple impermeable soil layers. These layers provide a hydraulic barrier between the topsoil cover and the underlying low permeability waste rock layer. The impermeable layers would minimize upward movement of moisture in the waste rock layer during summer drying and early winter freezing and would provide physical protection to the waste rock layer against potential damage from root penetration and burrowing animals. This layer would also help drain any infiltrated water away from the top surface of the low-permeability waste rock layer.

- A native topsoil layer that would be revegetated. The topsoil would provide physical protection for the underlying layers and facilitate the establishment of a sustainable vegetation cover. The topsoil cover also provides water storage capacity to prevent the low-permeability soil layer drying during dry seasons.

20.7.2.4 Waste Rock Facility

During the early years of mining a temporary nPAG waste rock storage facility is required and would gradually be reduced over the mine life as it is used to construct site facilities and back fill the open pit. It is anticipated that a 26 Mt nPAG waste rock stockpile would remain external to the open pit at closure. This would be re-contoured and have a soil cover placed over it.

20.7.2.5 Water Treatment Plant

Water treatment was also considered for the post-closure stage and will be in place for the time that it will be required. For the purpose of costing the cost estimate was developed considering market costs for operation of a water treatment facility, contemplating chemicals, diesel power generation, sludge removal and other suppliers and materials needed as well as labor to operate the plant through a time scale of 25 years with decreasing treatment requirements over time. Transport and disposal costs were included for the sludge to an external approved site. The water treatment plant used during the operational period of the mine will be refurbished at the end of operating life to serve for the closure period.

During the operational life of the mine, alternative treatment methods would be assessed and considered for the post-closure phase. Water management strategies would also be evaluated and optimized to minimize the volumes of water requiring treatment. Potentially passive water treatment may be used during the post closure monitoring period.

20.7.2.6 Mine Site Facilities

Mine site facilities including the process plant, camp, administration, maintenance shop, laboratory, site roads, magazine, and fuel storage would be dismantled or demolished at closure. Salvageable materials would be removed from the site and sold. Hazardous wastes would be removed from the site and disposed of in approved facilities.

Non-hazardous, inert building materials would be disposed in the site landfill or buried in the open pit during backfilling. Concrete footings would be broken up and buried during the backfilling of the open pit. Metal-contaminated soils would be removed and placed below the recharge water table in the open pit that would be backfilled. Hydrocarbon-contaminated soils would be excavated and either treated on-site in the land farm or transported off site for appropriate disposal. If treated on-site, these treated soils would be placed in the landfill or in the open pit that would be backfilled.

Following removal of the facilities and any associated contamination, the disturbed areas would be re- graded, capped with topsoil (if necessary) and revegetated with native species where possible. Mine site roads would be scarified and revegetated. Stream crossings would be returned to their pre-mining condition. Finally, the landfill would be closed using best management practices consistent with the approved reclamation and closure plan.

20.7.2.7 Access Road

Under the current access road partnership plan, Kaska Dena Nation and Tahltan Nation would be the road owners. It is anticipated that the access road would remain in place to offer access to other mining companies and for traditional hunting and fishing activities.

20.7.2.8 Airstrip

The airstrip is not owned by Kutcho Copper and Kutcho has no responsibility for its removal or reclamation.

20.7.2.9 Post-Closure Monitoring

Post-closure monitoring is expected to last between five and ten years after closure with requirements confirmed during operations and at closure. Monitoring requirements would be identified in the effluent discharge permit, and in consultation with the relevant government authorities and Indigenous nations. It is expected that the closure plan, developed in conjunction with the First Nations, to have Indigenous environmental monitors from the Kaska Dena and Tahltan Nations throughout the life cycle of the mine.

Monitoring activities would include:

- Water-quality monitoring of key post-closure water management facilities
- Environmental effects monitoring of water quality, sediment quality, benthos, and fish populations to assess effects on the aquatic receiving environment, including downstream flows in the Andrea and Sumac creeks
- Engineering inspections by qualified persons of the water collection ponds, water treatment plant, landfill, and all remaining engineered structures
- Implementing follow-up measures and repair if required
- Annual reporting to government and First Nations.

Closure of the water treatment plant and associated facilities would occur when monitoring indicates that direct discharge of untreated flows meets Environmental Management Act permit limits, as determined by the Ministry of Environment and Climate Change Strategy. Monitoring requirements would decrease or cease once water quality meets discharge criteria.

For the cost estimate purpose, the scope of work associated for the programs were developed applying analytical/sampling costs and consultancy market rates. The scope of work of each program was scale down through the years to reflect review of the monitoring programs that will be done through the time.

20.7.2.10 Cost Estimate

The cost estimate for these activities were developed considering expected quantities and effort required, applying market labor and equipment rates. The costs also considered a 15% contingency, indirect, camp and management costs.

A cost model for the closure and rehabilitation plan at the conceptual level was estimated considering a timeframe of 30 years after final ore processing. The total estimated cost of the activities corresponds to approximately C$34.5 million (undiscounted). Table 20-11 to Table 20-13 present the summary of the cost.

When the Mines Act Permit is issued for the construction and operation of the mine a reclamation security bond will be established for the initial five-year detailed mine plan. Thereafter, the permit will require the reclamation liability to be updated at least of every five years throughout the life of mine. Several years (approximately three years) before mine operations are planned to stop. The mine owner will be required to prepare a final closure and reclamation cost estimate and plan. The regulator will require full bonding of the final reclamation mine closure liability.

Table 20-11: Reclamation and rehabilitation cost estimate

Item	Total cost (C$M)
Topsoil load, haul and spread	1.9
TMF impermeable material load haul and spread	3.4
Pit impermeable material load haul and spread	2.1
Bulk land forming	1.4
Steep terrain drainage controls	0.9
General land forming (scarifying, ripping and land forming)	0.5
Fringe, clearing, grubbing, land forming	0.5
Revegetation	1.4
Facility reclamation	1.9
Underground facility closure	0.2
Water treatment plant capital	1.6
Deconstruction and rehabilitation post-closure accommodation	1.5
Deconstruction and rehabilitation post-closure management	1.0
Post operations water treatment cost	5.0
Post operations indirect during rehab period	0.8
Monitoring and studies	5.7
Contingency	4.5
Total Cost (CAD)	**34.5**

Table 20-12: Reclamation and rehabilitation cost schedule (2024 to 2045)

Item	Cost (CAD) - years 2024 to 2045																					
	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038	2039	2040	2041	2042	2043	2044	2045
Topsoil load, haul and spread						126,445			14,560		118,249	9,855		452,144	509,682	451,896	190,570					
TMF impermeable material load haul and spread															3,361,400							
Pit impermeable material load haul and spread											412,972	412,972	412,972	825,944	-							
Bulk land forming											137,449				549,795	710,005						
Steep terrain drainage controls						57,393			6,609		53,673	36,790		229,074	231,342	227,163	91,184					
General land forming (scarifying, ripping and land forming)						34,299			3,949		32,076	15,908		126,189	138,256	122,581	51,694					
Extension areas, clearing, grubbing, land forming						-			-			112,635		173,721	-	208,243	44,251					
Revegetation						90,560			10,428		84,690	50,358		347,899	365,035	341,762	140,336					
Facility reclamation														1,876,578			50,000					
Underground facility closure												220,604										
Water treatment plant capital															1,390,000							
Deconstruction and rehabilitation post closure accommodation														463,628	752,734	237,090	65,324					
Deconstruction and rehabilitation post closure management														302,366	490,913	154,624	42,603					
Post operations water treatment cost															279,109	279,109	279,109	279,109	279,109	237,230	237,230	237,230
Post operations Indirect during rehabilitation period															584,187	184,002	50,697					
Monitoring and studies											692,800	692,800	204,480	-	650,500	625,100	325,280	325,280	325,280	168,060	210,760	142,660
Contingency	-	-	-	-	-	46,305	-	-	5,332	-	229,786	232,788	92,618	719,631	1,395,443	531,236	199,657	90,658	90,658	60,794	67,199	56,984
Total Cost (CAD)	**-**	**-**	**-**	**-**	**-**	**355,002**	**-**	**-**	**40,877**	**-**	**1,761,694**	**1,784,710**	**710,070**	**5,517,174**	**10,698,396**	**4,072,811**	**1,530,706**	**695,048**	**695,048**	**466,084**	**515,189**	**436,874**

Table 20-13: Reclamation and rehabilitation cost schedule (2046 to 2067)

Item	Cost (CAD) - years 2046 to 2067																					
	2046	2047	2048	2049	2050	2051	2052	2053	2054	2055	2056	2057	2058	2059	2060	2061	2062	2063	2064	2065	2066	2067
Topsoil load, haul and spread																						
TMF impermeable material load haul and spread																						
Pit impermeable material load haul and spread																						
Bulk land forming																						
Steep terrain drainage controls																						
General land forming (scarifying, ripping and land forming)																						
Fringe, clearing, grubbing, land forming																						
Revegetation																						
Facility reclamation																						
Underground facility closure																						
Water treatment plant capital																						195,000
Deconstruction and rehabilitation post-closure accommodation																						
Deconstruction and rehabilitation post-closure management																						14,625
Post operations water treatment cost	237,230	237,230	196,601	196,601	196,601	196,601	196,601	156,909	156,909	156,909	156,909	156,909	138,273	138,273	138,273	138,273	138,273					
Post operations Indirect during rehabilitation period																						
Monitoring and studies	142,660	210,760	107,640	82,240	82,240	82,240	82,240	107,640	39,890	39,890	39,890	39,890	32,645	32,645	32,645	32,645	32,645	32,645	32,645	32,645	32,645	32,645
Contingency	56,984	67,199	45,636	41,826	41,826	41,826	41,826	39,682	29,520	29,520	29,520	29,520	25,638	25,638	25,638	25,638	25,638	4,897	4,897	4,897	4,897	36,341
Total Cost (CAD)	**436,874**	**515,189**	**349,877**	**320,667**	**320,667**	**320,667**	**320,667**	**304,231**	**226,319**	**226,319**	**226,319**	**226,319**	**196,555**	**196,555**	**196,555**	**196,555**	**196,555**	**37,542**	**37,542**	**37,542**	**37,542**	**278,611**

21 Capital and Operating Costs

21.1 Introduction

This section summarises capital and operating costs, with the major components set out in tabular form. Explanations and justifications for the basis of the cost estimates are included.

21.1.1 ***Base Date and Escalation Rates***

The costs are based on Q4 2021 pricing. All costs are real; escalation factors have not been applied in the cost estimates.

21.1.2 ***Exchange Rate and Base Currency***

All costs are expressed in Canadian dollars unless otherwise stated. Unless otherwise stated a rate of exchange of US$0.76 to C$1.00 was used for currency conversions.

No allowances were made for future fluctuations in exchange rates.

21.1.3 ***Accuracy***

The accuracy of the capital and operating cost estimates is ±15%. This is considered appropriate for Feasibility Study requirements.

21.1.4 ***Data Compilation***

The capital cost estimate was compiled and approved by CSA Global with major contributions from Mineit and Allnorth. The capital costs were obtained either from suppliers directly, assembled and verified from other contributors by CSA Global, or from reference databases.

The following entities either provided cost estimates that were applied directly, or inputs that were used for the estimation of costs:

- Piteau provided all quantities and EPCM costs for the TMF
- Allnorth provided quantities and costs for structures and services, the process plant, contractor indirects, and associated EPCM costs
- ABH provided costs estimates for the ore sorters and related equipment
- Mineit provided quantities for ventilation raises and explosives storage infrastructure
- Mineit provided estimates for underground equipment and access development
- CSA Global provided open pit mining-related quantities, pre-stripping estimates, topsoil removal and initial pit road construction costs, ownership, and administration costs
- Onsite provided cost estimates for the construction of the access road.

Some costs, such as open pit mine pre-stripping and owner's cost were developed in detail in the operating cost estimate and then transferred and reported during the construction period as capital costs. Whilst these amounts are included as capital in this section, the cost development is provided in the operating cost section of this report.

21.1.5 ***General Estimating Methodology***

Overall plant layout and equipment sizing was prepared with sufficient detail to permit an assessment of the engineering quantities for majority of the facilities, including earthworks, concrete, steelwork, and mechanical items. The layouts enabled estimates of quantities to be derived for all areas and interconnecting items.

A design allowance was included to cover inaccuracies in quantity estimates and the uncertainties associated with engineering design and construction. The quantum of the design allowance is based on the level of design detail and the quality of the cost information received and was specified against each line item.

For construction, unit rates for labour were derived from Allnorth's database and from sub-contractors with local regional experience in the scale and type of work specified. For capitalized operating costs, labour rates were derived from Mineit's database according to the type and scale of work in the region.

Pricing for key pieces of production equipment (for all areas of the plant, mine, or administration) were obtained from reputable suppliers with experience in the region, except for low-value items which were costed from Allnorth's or Mineit's database of recent project costs.

21.2 Capital Cost Estimation

The capital cost estimate for the Kutcho Project is summarized in Table 21-1.

Table 21-1: Summary of capital costs

Component (1), (2)		Units	Initial	Sustaining	Closure	Total
Direct costs	Mining costs	C$M	132.9	44.8	0.0	177.7
	Process plant	C$M	106.0	15.1	0.0	121.2
	On-site infrastructure	C$M	54.2	15.8	0.0	70.1
	Off-site infrastructure	C$M	31.7	0.0	0.0	31.7
EPCM and indirect costs		C$M	75.3	0.0	0.0	75.3
Owner's costs (including working capital)		C$M	35.9	5.6	0.0	41.5
Total capex without contingency		C$M	436.1	81.3	0.0	517.5
Contingency (3)		C$M	46.7	8.2	0.0	54.9
Salvage		C$M	0.0	0.0	-18.0	-18.0
Mine closure bond		C$M	10.0	9.3	-19.3	0.0
Closure and rehabilitation (4)		C$M	0.0	10.2	24.3	34.5
Total capex with rehabilitation		C$M	492.8	109.0	-13.0	588.9

Notes:
(1) All values stated are undiscounted.
(2) No inflation or depreciation of costs were applied; all costs are in 2021 money values. Major underground mobile equipment, all open pit mobile equipment and the power gensets are leased.
(3) Includes average contingency of 10.6%.
(4) Includes contingency of 15%.

21.2.1 Capital Cost Estimation Methodology

21.2.1.1 Plant Site Earthworks

Quantities for plant site bulk earthworks were estimated from the layout drawings.

The following activities are excluded from the capital cost estimate as they are captured in mining development costs:

- Clearing and grubbing.
- The removal and stockpiling of the top 300 mm of topsoil.
- Delivery of run-of-mine nPAG waste rock to the WRD or for construction site bulk fill construction.
- Mobilization and operation of a crushing plant to produce sized granular material from mining waste. It is assumed that this material will be available from a stockpile close to the construction site and will meet the required specification.

Rates for the wet hire of an earthworks construction fleet were built-up from first principles (equipment hire, fuel and lubrication, maintenance, labour) with allowances for miscellaneous labour, equipment, and supervision.

21.2.1.2 Concrete

Quantities for concrete works were established using:

- Material take-offs from layouts prepared for the current study
- Benchmarking of concrete volumes against general plans for similar sized projects completed by Allnorth
- Geotechnical reporting provided by Piteau (2021).

Whilst geotechnical test pits and drillholes over the site exist (as reported in Kutcho Surface Facilities Geotechnical Assessment and Data Review (Piteau, 2021), no geotechnical test pits nor drillholes are located in the specific areas of the SAG and ball mill foundations or in other process plant locations. As such, depth to bedrock was extrapolated from nearby test pit and drillhole logs. Designs were based on this data but a conservative contingency factor on thickness was used for concrete foundations. Recommendations for additional geotechnical investigations prior to detailed engineering are set out in Section 26.

Rates for this estimate were based upon those from a regional subcontractor's quotations with experience and capacity to perform the works as set out in Allnorth's database.

Quantities and composite rates were developed on a per cubic metre basis inclusive of concrete, rebar, formwork, and embedment.

A crushing plant, costed as part of the underground mining operation, will be provided for the construction period. The operating cost of the crushing plant, estimated from first principles assuming a crew of two people for one year, is included in the plant and site infrastructure capital cost.

An allowance for winter concreting and snow clearance was included in the estimate under contractor's indirect costs.

21.2.1.3 Steelwork

Quantities for structural steel were established using:

- The layout and equipment elevation drawings
- Benchmarking against detailed drawings for similar sized projects completed by Allnorth.

Rates for steel supply were based on the Allnorth database of fabricators. Particular care was taken to ensure that pricing projected for the construction period was not loaded with short term market conditions.

Installation hours were estimated using Allnorth's database of regional subcontractors with experience in this type of work.

21.2.1.4 Platework and Tankage

Platework and tankage quantities were determined using the dimensions provided in the mechanical equipment list prepared for the Feasibility Study. The tonnage of plate steel was estimated from the layout drawings and compiled in the platework list. Lining materials, where applicable, were quantified separately.

Supply costs for platework for this estimate were based on data from fabricators with experience on global supply. Budget pricing was obtained from reputable suppliers for the majority of the tanks.

Site installation hours were derived from estimates supplied by regional subcontractors with experience in this type of work.

21.2.1.5 Mechanical Equipment

The mechanical equipment list prepared for the Feasibility Study provided the quantities and sizing for the cost estimate.

Budget pricing was obtained from reputable suppliers for the majority of mechanical equipment.

Equipment installation hours were estimated based on data from contractors.

For each individual item of equipment due allowances were made for transportation from the source location, handling, placing, installation, and commissioning.
A list of the major mechanical equipment is provided in Table 21-2.

Table 21-2: Major mechanical equipment list

Equipment identification	Technical	Power		Capital cost
Description	**Equipment description**	**Power installed (kW)**	**Horsepower**	**Total (C$M)**
Rock breaker		56	75	0.16
Jaw crusher		132	177	0.30
Primary double deck screen	1,830 mm x 6,090 mm 6202-32CS	30	40	0.34
Cone crusher		185	248	0.34
Secondary double deck screen	1,830 mm x 6,090 mm 6202-32CS	30	40	0.34
Fine sorter	COM Tertiary XRT1200/B	15	20	0.86
Coarse sorter	COMXRT2400/B2.0	23	31	1.72
Waste bin feed conveyor	735 mm x 92,200 mm x 9,100 mm	15	20	0.35
Crushed ore bin feed conveyor	735 mm x 149,300 mm x 16,660 mm	22.4	30	0.70
SAG mill	6,100 mm X 3,500 mm (with VFD)	2,000	2681	5.36
Ball mill	5,030 mm X 7,620 mm (with VFD)	3,250	4357	5.09
Sag mill liner handler		30.0	40.2	1.44
Copper regrind mill		337	452	1.18
Zinc regrind mill		224	300	0.78
Copper concentrate thickener	8 m diameter x 2 m SWH, HRT	2.57	3	0.34
Copper concentrate filter press		38.5	52	1.39
Zinc concentrate thickener	6 m diameter x 2 m SWH, HRT	7.5	10	0.26
Zinc concentrate filter press		36.5	49	1.24
Tailings thickener	20 m diameter x 3 m SWH, HRT	6.05	8	0.59
Fire/fresh water tank	9,000 mm X 10,000 mm		0.0	0.56
Fire water jockey pump		22	29	0.32
Process water tank	9,000 mm X 10,000 mm		0.0	0.56
Plant assay laboratory load		282	378.0	2.20

21.2.1.6 Plant Pipework

The size and specification for each of the major pipe runs in the plant were based on Allnorth's engineering specifications. Piping layout design drawings were developed based on the plant layout and material take offs and quantities were derived from these design drawings.

Supply and installation costs were based on Allnorth's database of contract prices for similar projects and are inclusive of heat tracing, elbows, tees, and combination valves as required. A provision is made for hangers and supports, hydrostatic tests, shop drawings, and reworks.

The supply and installation for small-bore piping and bulk materials was estimated using factors derived from previous projects. These factors are applied as a percentage of the mechanical equipment costs and are calculated for each plant area. Bulk materials include items such as gaskets, bolts, and tubing.

21.2.1.7 Overland Pipework

Overland piping was quantified based on material take-offs derived from site layout drawings.

Supply and installation costs were based on Allnorth's database of similar projects.

21.2.1.8 Electrical and Instrumentation

An electrical and instrumentation list was developed from engineering design and site layout drawings. Supply and installation costs were based on a mix of budget quotes and Allnorth's database.

Initially, construction requires 2,000 kW of installed diesel generation capacity (four units of 500 kW capacity each). However, once commissioning commences the power plant will provide 10,000 kW of installed capacity from LNG generators in an N+1 configuration (while a 2,000 kW diesel generator will be maintained for added start-up plant torque).

The cost of the natural gas power-generation facility with partial diesel power-generation backup, switchgear, and an e-house was provided by reputable suppliers. Evaluation of proposals from these suppliers determined that the most-cost effective and capital-efficient route was to lease. Consequently, 20% of the total cost is assumed as a deposit and a further three months lease payments are included as capital costs. The remainder of the lease costs are included as an operating cost in the form of a power cost per kW hour and assigned to the operating area of power consumption.

The cost of the 13.8 kV overhead powerline was based on a construction distance of 4.1 km.

The plant control system cost was quoted and is inclusive of a phone and satellite communication system, local wireless network and a two-way radio installation.

The supply and installation cost for electrical and instrumentation 'bulks' was estimated using factors derived from previously built projects. "Bulks" include such items as power and control cabling, cable tray, instrument stands, and termination lugs.

21.2.1.9 Architectural and Buildings

Building dimensions were estimated from the layout drawings and similar projects of comparable scale. The style of building used in each instance was selected using a "fit for purpose" process to identify the most cost-effective building style which also met functional requirements. Factors considered were: building life requirement, functional description, and capital costs.

Budget pricing for pre-engineered buildings were sourced from reputable suppliers based on layout drawings. A provision was included for HVAC and fire protection system, sewage collection, and fit-outs.

The permanent camp was sized for 348 beds including recreation facilities, arctic corridors, kitchen, mine dry and indirect costs. The camp supply, delivery, and installation costs were sourced from reputable suppliers based in western Canada.

*21.2.2 **Mining Costs***

Mining costs are estimated to total C$177.7 million. Table 21-2 summarizes the mine capital costs.

Table 21-3: Summary of mining costs

Component	Units	Initial	Sustaining	Total
Underground mining equipment and infrastructure	C$M	17.9	14.0	31.9
Underground development	C$M	15.8	18.2	33.9
Pre-production mining	C$M	81.1	0.0	81.1
Mining ancillary	C$M	1.1	0.0	1.1
Site development	C$M	17.0	0.3	17.3
Mine sustaining	C$M	0.0	12.3	12.3
Total mining costs	**C$M**	**132.9**	**44.8**	**177.7**

Note: Table figures may not sum due to rounding.

21.2.2.1 Underground Mining Equipment and Infrastructure

Underground mining equipment and infrastructure capital costs are estimated to total C$31.9 million. Table 21-4 summarizes the cost components.

Table 21-4: Underground mining equipment and infrastructure summary

Component	Units	Initial	Sustaining	Total
Underground service infrastructure	C$M	3.7	0.1	3.8
Water management	C$M	1.2	0.8	2.0
Ventilation	C$M	0.2	1.5	1.7
Mobile equipment	C$M	9.8	6.4	16.2
Electrical	C$M	2.5	2.7	5.2
Technical and safety	C$M	0.8	2.3	3.0
Total	**C$M**	**17.9**	**14.0**	**31.9**

Note: table figures may not sum due to rounding.

The underground mine infrastructure component includes the underground access portal, compressed air supply (exclusive of piping), and the cemented rock fill (CRF) mixing plant. Total LOM costs for surface infrastructure are summarized in Table 21-5. The underground infrastructure includes the cost of two 20-person underground refuge stations.

Table 21-5: Underground service infrastructure LOM costs

Underground service infrastructure	Total LOM (C$M)
Portal	0.3
Compressed air plant	0.2
CRF mixing plant	3.0
Refuge stations	0.3
Total	**3.8**

The water management component includes sumps, pumps, booster pump stations, and sediment management (sludge pumps). Total LOM capital costs for water management are summarized in Table 21-6.

Table 21-6: Water management LOM costs

Water management	Total LOM (C$M)
Fresh water supply system	0.1
Sumps	0.5
Booster stations	0.4
Pumping skids	0.2
Face pumps	0.1
Sediment management	0.3
Rebuilds and replacements	0.3
Total	**2.0**

The ventilation component includes fans, ducting, airlocks, fire doors, bulkheads, regulators, and miscellaneous. Total LOM capital costs for ventilation are summarized in Table 21-7.

Table 21-7: Ventilation LOM costs

Ventilation	Total LOM (C$M)
Fans	0.8
Bulkheads and regulators	0.5
Airlocks and fire doors	0.3
Ducting	0.1
Stench gas equipment	0.1
Miscellaneous other	0.0
Total	**1.7**

Note: Table figures may not sum due to rounding.

The mobile equipment component includes deposits and initial payments for leased equipment, equipment mobilization, rebuilds, assembly and first fills. Mobile equipment LOM capital costs are detailed in Table 21-8 and additional equipment cost parameters are provided in Table 21-9.

Table 21-8: Mobile equipment LOM costs

Mobile equipment	Total LOM (C$M)
Lease initial payments	11.7
Mobilization	1.7
Rebuilds	1.8
Assembly	0.8
First fills	0.2
Total	**16.2**

Table 21-9: Equipment lease terms

Equipment lease terms	Unit	Value
New or used		New
Preferred vendors		Sandvik/CAT/Komatsu
Lease - deposit payment	%	25
Lease – origination fee	%	0.6
Lease – contract fee	C$	1,580
Lease – APR	%	5.5
Lease – payment periods	months	48.0

The electrical capital estimate includes surface electrical distribution from the final pole of the overland powerline (overland powerlines are covered in overall site power distribution), underground electrical distribution (including transformers, gate boxes and cabling), and communication systems. Communication systems include a UHF/VHF leaky feeder system, fibre cables, and associated communications equipment. Total LOM capital costs for electrical are summarized in Table 21-10.

Table 21-10: Electrical LOM costs

Electrical	Total LOM (C$M)
Surface electrical distribution	1.4
Underground electrical distribution	3.2
Communication systems	0.6
Total	**5.2**

The technical and safety capital estimate includes software and computer equipment, mine rescue, first fills of personal protection equipment (PPE), signage, barriers, and technical studies. Items not included that have a safety function are refuges (included in underground infrastructure), and ladderways (included in vertical development). Technical and safety cost are summarized in Table 21-11.

Table 21-11: Technical and safety LOM costs

Technical and safety	Total LOM (C$M)
Software and computer equipment	0.4
Mine rescue	0.3
PPE, signage, and barriers	1.0
Technical studies	1.3
Total	**3.0**

21.2.2.2 Underground Development

Underground development costs included in capital total C$33.9 million of which C$15.8 million is incurred in the initial construction timeframe (pre-production) and the remainder is spent in the following two years. These costs are mostly for drive development and include:

- All ramps including drill chambers, re-muck bays, by-pass bays and safety cut-outs
- All ventilation raises and associated accesses
- Dewatering sumps
- Electrical stations.

Drive development costs have been estimated through estimation of productivities, mine support costs, ventilation, power, fuel consumption, equipment maintenance costs, maintenance labour, supplies management and labour. The costs are estimated based on an owner-operator basis. Table 21-12 summarizes the underground development costs.

Table 21-12 : Underground development costs

Underground Development	Total LOM (C$M)
Permanent waste drives	18.4
Vent raises	5.0
Escape way	1.3
Ore pass	1.4
Material handling	1.8
Mine maintenance	1.3
Mine general	4.6
Total	**33.9**

21.2.2.3 Pre-Production Mining

Pre-production mining costs total C$81.1 million. These costs were developed through the operating costs estimation process and more details are included in Section 21.3. In summary, the pre-production mining costs include:

- Topsoil stripping and stockpiling by the operations mine fleet, and supervision
- Open pit mining of sufficient nPAG waste for construction of TMF Phase 1 prior to commencement of plant operations
- Open pit mining ramp-up of ore production sufficient to enable a rapid achievement of steady state plant throughput at the start of Year 1
- Open pit and plant manning in the start-up period, including three months training for 25% of the crew and additional trainer time
- Developing underground mining to full ore capacity on the Main deposit
- Plant commissioning operating costs (for tonnages produced during this timeframe – see Sections 17 and 22 for details).

The production mine plan estimates that during the pre-production period 503 kt of ore is to be mined from the open pit (190 kt of sulphide high-grade ore, 213 kt of oxide ore that is stockpiled for later processing, and 100 kt of low-grade sulphide ore that is placed on the long-term stockpile). A further 150 kt of high-grade sulphide ore is planned to be mined from the Main underground. It is planned that 204 kt of ore will be processed in the plant during the pre-production commissioning period.

The pre-production period is planned to allow both the underground and open pit mines to attain full mining rate capacity by the start of the operating period. It is planned that 11.5 Mt of waste will be mined in the open pit during this period and will contain 4.4 Mt of nPAG waste of which 2.5 Mt is planned to be directed to the TMF Phase 1 embankment. The rest of the open pit nPAG waste will be used to as an under-liner pad at the PAG WRD and the remainder placed on the WRD.

21.2.2.4 Mining Ancillary

Mining Ancillary costs are estimated at C$1.1 million and include:

- Open pit dewatering pipelines (pumps are covered in operating costs)
- Surface explosives storage pad area and access road
- Surface storage facilities mobilization, detonator magazine purchase and delivery, and packaged explosives magazine purchase and delivery
- ANFO silos will be rented; the rental cost is included in operating costs for open pit mining.

21.2.2.5 Site Development

Site development capital costs for the Project total C$17.3 million (Table 21-13).

Table 21-13: Mine capital summary

Component	Units	Initial	Sustaining	Total
Bulk earthworks	C$M	6.6	0.0	6.6
On-site infrastructure roads	C$M	3.0	0.0	3.0
Surface water management	C$M	5.1	0.0	5.1
Clear and Grub	C$M	1.2	0.3	1.5
Initial mine roads	C$M	1.5	0.0	1.5
Total	**C$M**	**17.0**	**0.3**	**17.3**

NB: Table values may not sum due to rounding.

Topsoil stripping costs for the entire site is included in operating costs for open pit mining.

Clearing and grubbing costs were obtained from database values provided by Allnorth for locations with limited harvestable timber. Less than 35% of the site to be impacted has shrub or forest cover.

Initial mine road construction costs include the use of equipment that is shared with mining operations to develop the initial 3,270 m of mine haul roads that connect the workshop, pit, TMF, WRD and topsoil stockpiles.

21.2.2.6 Mine Sustaining

Mine sustaining costs total C$12.3 million for both the underground and open pit mines. Based on previous experience in this type of operation, mine sustaining costs are estimated at 2% of annual mining mobile equipment capital cost in use, and 3% of underground mining capital. At the cessation of underground and open pit mine operations, starting in 2035 (Year 9) and for each subsequent year, the underground and open pit sustaining cost estimate was halved.

*21.2.3 **Process Plant Capital Cost***

The process plant direct capital is estimated at C$121.2 million. Table 21-14 details the component costs.

Table 21-14: Direct processing plant capital summary

Component	Units	Initial	Sustaining	Total
Crushing	C$M	17.3	0.0	17.3
SAG and ball mills	C$M	35.5	0.0	35.5
Copper flotation	C$M	11.9	0.0	11.9
Zinc flotation	C$M	7.1	0.0	7.1
XRF sample analysis	C$M	0.5	0.0	0.5
Copper concentrate dewatering	C$M	3.7	0.0	3.7
Zinc concentrate dewatering	C$M	3.5	0.0	3.5
Tailings thickening	C$M	5.2	0.0	5.2
Water services and water treatment	C$M	13.5	0.0	13.5
Reagents	C$M	5.4	0.0	5.4
Plant utilities, building and general	C$M	2.4	0.0	2.4
Sustaining	C$M	0.0	15.1	15.1
Total	**C$M**	**106.0**	**15.1**	**121.2**

The crushing circuit includes the ore sorters and ore sorters reject storage bin. Plant mobile equipment is leased, and that cost is included in the plant operating cost.

The water treatment plant is sized to handle a peak flow of 320 L/s which meets maximum flow as set out in the site-wide water balance and TMF water containment volumes. The water treatment plant is designed to remove metals by the addition of ferric sulphate followed by precipitation after the addition of lime. The water treatment system capacity may be increased by the addition of modular units capable of treating 70 L/s each should water volumes exceed the ranges predicted by the water balance.

21.2.4 On-Site Infrastructure Capital

The direct on-site infrastructure capital is estimated at C$70.1 million. Table 21-15 details the component costs.

Table 21-15: Direct on-site infrastructure capital summary

Component	Units	Initial	Sustaining	Total
Power generation and distribution	C$M	13.3	0.0	13.3
Bulk fuel storage and distribution	C$M	0.9	0.0	0.9
Camp complex and accommodations	C$M	15.2	0.0	15.2
IT and communications	C$M	6.5	0.0	6.5
Waste management	C$M	0.8	0.0	0.8
Ancillary buildings	C$M	7.5	0.0	7.5
Surface mobile equipment	C$M	0.0	0.0	0.0
Gate house	C$M	0.1	0.0	0.1
Tailings management facility	C$M	9.9	15.8	25.7
Total	**C$M**	**54.2**	**15.8**	**70.1**

A cost trade-off analysis was completed to determine if the power plant and the LNG storage facility would be leased or purchased. The analysis determined that it was beneficial to lease the power plant but purchase the LNG storage. The power generation estimate includes 20% of the purchase cost of the power plant, the remainder is leased, and the cost is provided in the operating cost estimate.

Camp accommodation and associated infrastructure is sized to meet the peak personnel demands during the construction phase and will be installed as early as possible in Year -2 of construction. The peak camp loading of 348 was determined through identifying direct and indirect construction hours based on the material take-offs, quotes from vendors, and the estimating methodology, and then time phasing the activities based on the two-year construction schedule. The camp accommodations and associated modules proposed are a combination of new and lightly used modules. After completion of construction, two modules will be removed from the project and sold, leaving a total of 250 beds for the operational phase. This number of beds is sufficient accommodate regular operating personnel and additional contractors that may be required for the TMF dam raises and other maintenance and engineering activities. Supporting infrastructure such as LNG supply for heating, sewage treatment, and power distribution are designed to meet the peak construction camp loading demands and a rental incinerator will be used to supplement the incinerator sized for operations.

Surface mobile equipment is included in the plant and open pit mine operating costs set out in Section 22.

21.2.4.1 Tailings Management Facility

The bills of quantities and technical specifications for the TMF were provided by Piteau. The scope of work encompasses the foundation of the storage facility containment wall or embankment, the storage area impoundment, underdrain system, embankment liner, access road, diversion and perimeter channels, dam-toe canal, sediment control ponds, and instrumentation.

Rates were derived from the Allnorth database for the hire (wet) of an earthworks fleet built-up with allowances for miscellaneous additional labour and equipment, and supervision.

The cost of delivery of open pit mine waste for the bulk fill of the embankment wall was estimated in the mine operating cost. It was determined that the delivery cost and handling of waste to the TMF or to the WRD was equal and thus no additional haulage cost was applied.

21.2.4.2 EPCM and Indirects

The EPCM and indirect costs are estimated at C$75.3 million. Table 21-16 details the major components of the EPCM and indirect capital costs.

Table 21-16: EPCM and indirect costs

Component	Units	Initial
Heavy equipment	C$M	3.9
Contractor field indirects	C$M	23.4
Construction support indirects	C$M	7.9
Vendors representatives	C$M	0.5
Capital spares	C$M	4.1
Start-up and commissioning	C$M	1.0
First fills	C$M	0.6
Freight and logistics	C$M	10.0
Engineering and procurement	C$M	6.5
Construction and project management	C$M	17.5
Total	**C$M**	**75.3**

21.2.4.3 Heavy Equipment

A 200-tonne heavy lift crane and operator for 10 months will be required to support construction activities. Support equipment for the construction phase, such as 10-tonne forklift, telehandler, and flatbed trailer and truck are included in owner's costs. They will remain as part of the site equipment fleet during operations.

Heavy mobile equipment for the open pit, administration and underground is proposed to be leased and is described in Sections 21.3.2 and 21.3.3.

Contractor field indirect allowances were made for major construction cranage and equipment, and construction costs such as site establishment, construction waste management, medicals, emergency response, security services, third party inspection, surveying, a car wash, and fuel.

The following services have been included under Owner's Costs:

- Site communication
- Camp snow removal
- Catering and janitorial costs
- Camp LNG consumption
- Transportation of the direct and indirect workforce to and from points of pick up and the site.

21.2.4.4 Contractor Distributable

Contractors' indirect costs were estimated by first principals based on the assessed scope of work. Mobilization, demobilization, and indirect costs were identified and listed separately for each contract based on the following contract packages:

- Site earthworks
- Concrete
- Structural, mechanical, piping, and overlanding piping

- Electrical and instrumentation
- Tailings management facility construction

Contractors indirect costs include costs excluded in the contractor unit rates as per the following:

- Mobilization and demobilization
- Off-site preliminary costs including corporate costs, off-site management and administration and off-site overheads
- On-site indirect labour
- Temporary site facilities and associated services/equipment
- Health, safety and environmental management
- Personnel transport
- Site signage and barricades
- Scaffolding.

21.2.4.5 Vendor Representatives

Vendor representatives' costs were based on a combination of budget pricing from vendors and historical database pricing.

21.2.4.6 Capital Spare Parts

Price estimates for spares parts were derived from a combination of vendor quotes and factored equipment supply based on grouping equipment from the equipment list into equipment packages (e.g. all pumps were grouped into a pump package). The estimates for each equipment package were benchmarked against spares expenditures on projects of a comparable scale.

21.2.4.7 Start-Up and Commissioning Services

A provision was included for commissioning services, assuming a crew of four training personnel and one commissioning expert. It also includes contractor assistance during commissioning (contractor support of tradespeople for 2.5 months).

21.2.4.8 First Fills

A consumable list was prepared based on the project design criteria. Quantities for first fill have been assembled from basic principles assuming a provision of consumables for one week of operations.

21.2.4.9 Freight and Logistics

Freight was estimated on a line-by-line basis of equipment and materials identified in the capital cost estimate. Pricing was based on a combination of budget pricing from equipment vendors, budget pricing from trucking companies, and factors applied against equipment and material supply costs.

21.2.4.10 EPCM

The Engineering, Procurement and Construction Management (EPCM) cost was estimated from first principle according to the scope of the project.

*21.2.5 **Owner's Costs (Capitalized General and Administration Costs)***

The capitalized general and administration costs total C$41.5 million.

Owner's costs were developed as part of the mine operating costs but modified for the construction period. Details of the estimation methodology and full list of items covered is included in Section 22. Capitalized operating costs include:

- Camp operations for construction and operations personnel
- Transportation for construction and operations personnel
- Site snow removal
- Off-site access road maintenance
- Site general maintenance
- Camp LNG consumption
- Hire of construction-period diesel power generators
- General and administration labour and costs
- Water treatment costs.

Working capital is estimated at C$5.6 million. This is estimated on a monthly basis and is projected to occur in the first quarter of the first production year. It is estimated as the difference between income and operating cost. Delays in concentrate build-up and sales timing was considered.

*21.2.6 **Contingency***

The purpose of contingency is to make specific provision for uncertain elements of cost within the project scope and thereby reduce the risk of cost over-run. Contingencies do not include allowances for scope changes, escalation, or exchange rate fluctuations.

Contingency reflects the measure of the level of uncertainties related to the scope of work and was applied to all parts of the estimate, i.e., direct costs, indirect costs, services costs, etc.

The estimation of contingency was made by assessing the level of confidence in:

- Material take-off definition
- Scope definition
- Supply cost basis
- Installation cost basis.

The weighted average of each input or cost element was used to calculate the contingency, and those varied between 9% and 11.5% for each discipline. The contingency estimate totals C$54.9 million or 10.6% of the total direct and indirect capital costs. Contingency for the site rehabilitation is estimated separately at 15% and included within the site rehabilitation estimate cost in order to determine the mine closure bond directly.

*21.2.7 **Salvage***

Salvage values for mechanical and electrical equipment were estimated by contacting used equipment vendors to establish salvage value factors. These salvage factors (ranging from 1% to 15%) were applied to each line item of the capital cost estimate to derive a salvage value. Only items that have an original purchase price of at least C$100,000 were considered for salvage. The salvage value of plant and infrastructure equipment, estimated by Allnorth, is C$9.6 million.

Site surface mobile equipment salvage value was factored at 10% of the initial investment and estimated at C$3.8 million, inclusive of transport. Underground equipment and mobile equipment salvage value was estimated by Mineit at C$4.5 million. Total salvage value to be realized at closure was estimated at C$18.0 million. The salvage value is regarded as slightly conservative as some equipment would be sold prior to site closure as open pit and underground mining operations terminate a few years prior to full mine closure.

21.2.8 Site Closure, Rehabilitation and Bond

Site rehabilitation costs are detailed in Section 20.7. They are estimated to total C$34.5 million, and majority of the costs occur after plant closure. Closure and rehabilitation cost include a15% contingency. A mine closure bond is paid progressively and one year in advance as a function of surface disturbance. It is calculated as the cost to rehabilitate the site at that time. The long-term post closure bond is estimated prior to closure. Full recovery of the bond is assumed to occur five years after rehabilitation is determined as successfully completed. Costs after closure are estimated in present value terms to 2038. The bond recovery assumes full and successful rehabilitation.

21.2.9 Capital Cost Exclusions

The following items are specifically excluded from the capital cost estimate:

- Corporate financing costs or interest costs during construction
- Schedule delays exceeding four weeks and associated costs
- Scope changes
- Unidentified ground conditions
- Extraordinary climatic events
- Exchange rate variations
- Force majeure
- Labour disputes
- Receipt of information beyond the control of EPCM contractors
- Schedule recovery or acceleration
- Research and exploration drilling
- Escalation.

21.3 Operating Costs

Operating costs were development on the basis of equipment productivities, contractor quotations or supplier costs for machinery, consumable and services. The location of the Project in northern BC was also considered.

Labour costs across all activities were estimated from file data and benchmarking exercise undertaken by Mineit.

The mining operating costs includes the leasing of all mining equipment, and all activities operated and managed by the owner. Only very specialized services are provided from contractors or consultants.

Process plant operating costs include all consumables (balls for the ball mill, reagents, and chemicals) power, external services, TMF operating costs, and plant maintenance to allow for the production of two concentrates. No operating contingency is included. However, costs estimated in the capital construction period (and reported as capital costs) include a contingency. The battery limit for plant operating is at the point of production of the concentrate at site. Concentrate transport, port storage and onward costs are considered off-site costs and are detailed in Section 19 and Section 22

General and administration costs were developed from file data of similar operations in Canada and include some zero-based cost estimation for items such as water treatment costs. Costs not developed by first principles are covered by budget estimates provided by regionally based companies.

The average LOM mine operating cost is estimated to be C$65.89/t of ore crushed. Table 21-17 summarises the LOM operating costs.

Table 21-17: LOM operating costs exclusive of pre-production costs

Component	Units	Early production	Steady state	Stockpile rehandle	LOM (2)
		Year 1 (1)	Years 2 to 8	Years 9 to 11	Years 1 to 11
Open pit mining cost	C$/t OP ore mined	27.68	20.55	NA (3)	24.75
Underground mining cost	C$/t UG ore mined	62.17	57.88	25.61	55.38
Plant processing cost	C$/t ore crushed	28.06	28.48	27.53	28.21
General and administration	C$/t ore crushed	10.01	9.36	6.69	8.79
Total	**C$/t ore crushed**	**79.24**	**70.51**	**47.67**	**65.89**

Notes:
(1) Year 1 includes pro-rated adjustments for working capital.
(2) Pre-production tonnages and costs are not included in the LOM operating cost summary (these are years -2 and -1 and are capitalized).
(3) No ore mined, rehandle period.

21.3.1 Labour Costs

Labour costs used in all elements of the operating costs for open pit, underground and administration were developed by Mineit using a database of regional labour rates. Payroll on-costs (benefits) varied from 15% to 35% and averaged 27%. Payroll on-costs include:

- Canadian pension plan
- Employment insurance
- Workers compensation benefit
- Vacation
- Extended health coverage
- Bonuses.

Costs for travel and camp accommodation were included in General and Administration. Most personnel were considered to work a two-weeks on, two-weeks off shift rotation. A shift length of 10 hours was assumed.

Some administration personnel were considered as based in Whitehorse and a few administration staff are to be based at the mine and in Whitehorse. Some positions are allocated as day shift only, thus necessitating two personnel per position and one pax for accommodation purposes. Most positions, however, were for a day-night shift roster necessitating four personnel per position and two pax for accommodation purposes.

21.3.2 Open Pit Mine Operating Costs

The operating cost model utilized is an activity and area-based model that was developed from first principles and estimated on a quarterly basis and a cost per tonne mined. The key drivers in the production schedule and thus the operating cost model included material movement by type (PAG waste, nPAG waste, oxide ore, low-grade sulphide ore and high-grade sulphide ore) and haul route distances and gradients per period. Table 21-18 reflects the LOM open pit operating costs.

Table 21-18: LOM operating costs exclusive of pre-production costs

Component	Units	Pre-production (2)	Early (2) production	Steady state	Stockpile rehandle	LOM (2)
		Years -2 to -1	Year 1	Years 2 to 8	Years 9 to 11	Years -2 to 11
Ore mined	Mt	0.5	1.6	12.4	0.0	14.5
Waste mined	Mt	11.5	12.1	58.2	0.0	81.8
Strip ratio	W/O	22.8	7.7	4.7	0.0	5.6
Ore trucked rehandle	Mt	0.0	0.0	0.9	3.8	4.7

Component	Units	Pre-production (2)	Early (2) production	Steady state	Stockpile rehandle	LOM (2)
		Years -2 to -1	Year 1	Years 2 to 8	Years 9 to 11	Years -2 to 11
Waste rehandle	Mt	0.0	0.0	5.1	26.5	31.6
Total movement	Mt	12.0	13.7	76.6	30.3	132.5
	tpd	16,400	37,562	29,978	27,645	27,934
Mining cost	C$/t moved	4.94	3.33	3.33	1.57	3.07
	C$/t ore mined	117.28	28.93	20.55	na	28.10
Mine life		12.5 years including pre-production and rehandle				

Notes:
(1) 1.25 years for Pre-Production and Stockpile Rehandle periods.
(2) Table values will differ to operating costs exclusive of pre-production period.

Open pit mine operating costs vary mostly (but not entirely) due to:

- The volume being moved (larger volumes per period have reduced costs per tonne moved through distribution of fixed overheads)
- Length of the haul and the mining bench elevation in the pit (longer hauls from deeper parts of the are more costly per tonne moved)
- Timing (after Year 4, most lease repayments cease and this reduces the operating cost)
- The proportion of 5 m and 10 m benches (10 m benches being lower cost)
- Once the pit terminates, rehandle operations have reduced management requirements and no drill and blast component
- Overheads being higher in the pre-production period for additional training and low movement
- Volumes.

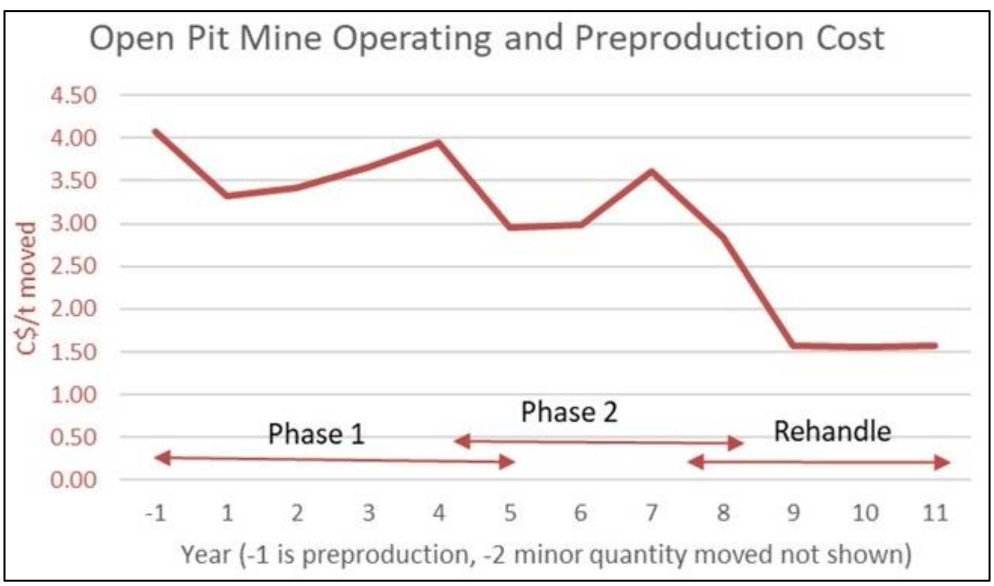


Figure 21-1: Total Open Pit Mine Operating Cost per Tonne Moved by Year

The open pit mining costs were estimated on the basis of activities:

- Drilling
- Blasting
- Loading

- Hauling
- Auxiliary equipment
- Mine management labour and general (including grade control).

The unit costs of open pit mining are presented in Table 21-19 below.

Table 21-19: LOM total cost and cost per tonne moved by open pit mining activity

Open pit mining activity	LOM C$M	C$/t moved
Drilling	28.2	0.21
Blasting	59.9	0.45
Loading	42.3	0.32
Hauling	150.8	1.14
Auxiliary	87.9	0.66
Mine management	37.9	0.29
Total	**407.0**	**3.07**

Operating costs are also based on the following cost areas:

- Fuel
- LNG
- Power
- Lubricants
- Operating supplies and miscellaneous
- Tyres
- Repair cost
- Lease costs
- Labour.

The open pit operating costs for the study were mostly derived from supplier quotations. Where these were not available CSA used internal data sources and industry benchmarked prices.

The operating costs for the primary production fleet (haul trucks, hydraulic excavators, blast-hole drilling rigs and front-end loaders) were obtained from the original equipment manufacturers (OEMs) and developed alongside the equipment performance criteria and expected fuel, oil and lubricant consumption rates were taken from suppliers or database values.

The mine plan is premised on an "owner-operated" mining operation. All mining equipment costs include complete coverage of lease purchase as an operating cost. The lease cost is applied firstly as a 25% lump sum on initial purchase followed by equal quarterly sums for the entire 48-month lease term. The lease recovery includes a 5.55% APR interest rate. The initial lump sum also includes equipment assembly costs (where not explicitly stated by the OEM, 8.8% of total price), initial capital spares (6% of total price) and freight (2.8% of total price). About 90% of all plant and open pit mobile equipment were based on supplier estimates. The remaining 10% were derived from file data from Allnorth and Mineit databases. The proportion of data from quotations is considered appropriate for a feasibility estimate.

The estimation does not include construction of the initial mine access roads (temporary roads within the mining area are covered by ancillary equipment costs) and clearing and grubbing. These are included as capital costs.

Lubricants, fuel and tire costs are derived from supplier quotations or file information as a consumption or cost per operating hour for each equipment type and moderated for equipment duty.

The mine operating cost battery limit is at the point where the ore is fed to the crusher (but does include ore stockpile rehandle by a front-end loader).

Equipment operating costs for open pit and ancillary equipment were developed from file data, quotations from OEMs and modified for duty where appropriate. Operating costs for each type of equipment are included in Table 21-20.

Table 21-20: Equipment operating costs

Equipment and operating costs C$/operating hour	Maintenance parts	Diesel	Lubricants	Tyres
Drill Rig 328 kw 90-165 mm	43.00	58.74	7.05	-
Wheel Loader 672 kw 10 m^3 bucket capacity	77.52	115.71	11.62	5.89
Off-Highway Rigid body Haul Truck 90 t capacity	63.85	79.29	6.34	5.89
Tracked Dozer 264 kw	34.45	38.65	4.64	-
Wheel Loader - straight edge 7 m^3 bucket capacity	30.45	49.55	4.96	4.12
Motor Grader 221 kw	13.35	16.08	2.09	4.12
Wheel Loader 263kw multi-purpose	21.00	49.34	-	4.20
Water truck articulated (35,000 l)	18.13	34.54	2.76	4.12
Tracked hydraulic excavator 4 m3 capacity bucket	46.75	70.48	10.57	-
Vibratory compactor 117kw	1.04	2.35	0.35	0.03
Utility Backhoe - 2.0 m^3	0.75	1.76	0.26	1.25
Tire handler 263kw	6.00	11.75	1.76	1.75
Tractor & Lowboy - 50 t	6.25	11.75	1.76	1.50
Mobile field fuel & lube truck - 11,300 l	0.75	1.76	0.26	1.80
Mechanics Service Truck- 20.4 t	0.23	2.94	0.44	0.07
Mechanics/Welder Truck - 3.9 t	0.75	1.76	0.26	0.38
Crew bus	1.25	2.94	0.44	0.16
Crew Cab Pickup Truck - 4x4, Heavy Duty	1.25	0.75	0.11	0.19
Pickup Trucks - 3/4 ton, Heavy Duty	1.25	0.75	0.11	0.24
Wheel Loader fitted as Field Forklift	1.25	2.94	0.44	1.54
Shop forklift	1.25	2.94	0.44	0.24
Light Plant/Towers 8kw	0.50	0.88	0.13	-
Mine Pump 35 L/s	1.25	3.52	0.53	-
Haul Truck Articulated 35t capacity	29.16	34.54	1.30	2.55
Tracked Dozer 177 kw	43.58	34.18	0.97	-
Wheel Loader 119 kw multipurpose	12.64	14.80	1.23	3.50
Snowplough/sanding truck	4.90	12.33	1.40	3.50
Fire truck	30.00	52.86	7.93	4.92
Ambulance	30.00	52.86	7.93	4.92
Hydraulic crane w/ Telescopic Boom - 40 t	15.00	21.15	3.17	0.07
Trucks c/w picker arms	3.00	7.05	1.06	1.54

Over 90% of all equipment purchase costs were quotations from OEMs. Assembly costs were either supplied by the OEMs or are applied at 8.8% of the purchase price. Capital spares are either supplied by the OEM or are applied at 6% of the purchase price. Freight costs from the point of sale to site are either supplied by the OEM or are at 2.8% of the purchase price.

The lease costs were initially estimated based on a 25% down payment of the combined purchase, assembly, spares, and freight charge. The lease cost after purchase was based on a 48-month payment plan at 5.55% annual effective interest.

The purchase costs of the mobile equipment fleet are presented in Table 21-21.

Table 21-21: Equipment capital costs

Equipment purchase costs C$	Purchase	Assembly	Capital spares	Freight
Drill Rig 328 kw 90-165 mm	920,000	-	55,200	-
Wheel Loader 672 kw 10 m3 bucket capacity	2,379,149	152,152	142,749	56,489
Off-Highway Rigid body Haul Truck 90 t capacity	1,841,322	219,008	110,479	59,946
Tracked Dozer 264 kw	667,000	58,658	40,020	18,401
Wheel Loader straight edge 7 m3 bucket capacity	892,000	78,445	53,520	24,609
Motor Grader 221 kw	1,560,000	-	93,600	-
Wheel Loader 263kw multi-purpose	623,000	54,788	37,380	17,187
Water truck articulated (35,000 l)	820,000	68,156	49,200	22,622
Tracked hydraulic excavator 4 m3 capacity bucket	836,000	73,520	50,160	23,064
Vibratory compactor 117kw	280,000	-	16,800	-
Utility Backhoe - 2.0 m3	250,000	21,986	15,000	6,897
Tire handler 263kw	744,000	65,429	44,640	20,526
Tractor & Lowboy - 50 t	650,000	57,163	39,000	17,932
Mobile field fuel & lube truck - 11,300 l	170,000	14,950	10,200	4,690
Mechanics Service Truck- 20.4 t	200,000	17,589	12,000	5,518
Mechanics/Welder Truck - 3.9 t	180,000	15,830	10,800	4,966
Crew bus	180,000	15,830	10,800	4,966
Crew Cab Pickup Truck - 4x4, Heavy Duty	60,000	5,277	3,600	1,655
Pickup Trucks - 3/4 ton, Heavy Duty	50,000	4,397	3,000	1,379
Wheel Loader fitted as Field Forklift	350,000	-	21,000	-
Shop forklift	15,897	1,398	954	439
Light Plant/Towers 8kw	23,000	-	1,380	-
Mine Pump 35 L/s	130,000	-	7,800	-
Haul Truck Articulated 35t capacity	775,000	-	46,500	-
Tracked Dozer 177 kw	800,000	-	48,000	-
Wheel Loader 119 kw multipurpose	350,000	-	21,000	-
Snow plough/sanding truck	158,950	13,979	9,537	4,385
Fire truck	150,000	13,191	9,000	4,138
Ambulance	180,000	15,830	10,800	4,966
Hydraulic Crane w/ Telescopic Boom - 40 t	350,000	30,780	21,000	9,656
Drill Rig 328 kw 90-165 mm	250,000	21,986	15,000	6,897

Equipment operating life hours were estimated. Main production equipment was estimated to have a useful operating life of about 55,000 hours and ancillary equipment 35,000 hours. These operating hours are close to the estimated LOM operating hours for the open pit mine (10 years). No equipment replacements or rebuilds are considered. Utilization and timing of avoids equipment over-use and avoids low capital-efficiency equipment purchases.

21.3.2.1 Drilling

Drilling costs were based on machinery productivities for four activities:

- Blasting 5 m benches for ore

- Blasting 10 m benches for waste
- Hangingwall pre-spit holes
- Footwall dewatering holes.

OEMs supplied suitable drill rigs whose specifications matched the operational duty range for presplit and production holes. The selected units have a motor capacity of 328 kW and can drill a range of hole diameters of 90-165 mm using down the hole hammers. Two units are considered adequate. The use of 10 m benches for hangingwall waste removal is a key consideration for drill productivity.

A schedule was developed for timing of availability of 5 m and 10 m benches, hangingwall pre-split holes and footwall dewatering holes. The drill rigs were scheduled on the timing of the quantities required. Drilling costs summarized in Table 21-22.

Table 21-22: Drilling cost summary

Drilling cost element	C$ per tonne moved	C$ per tonne mined
Fuel	0.04	0.05
Oil and grease, coolant, filters	0.00	0.01
Operating supplies	0.06	0.09
Repairs	0.02	0.03
Leasing	0.02	0.03
Maintenance labour	0.02	0.03
Labour	0.04	0.06
Total Drilling Costs	**0.21**	**0.30**

21.3.2.2 Blasting

Blasting costs were based on three activities:

- Blasting 5 m benches for ore
- Blasting 10 m benches for waste
- Hangingwall pre-spit blast holes.

Blasting assumes the use of ANFO for all production blasts. This will require blast hole water management (to obtain dry hole conditions), the use of ANFO as a dry-hole product and hole clearing. ANFO has advantages over emulsion in maintaining a shot product with low nitrate levels which reduces water contamination.

The blasting equipment was estimated as fully rented and includes two 50-tonne AN Prill silos, a blasting truck, universal front end loader/forklift, a loader for stemming, explosives truck and pick-ups. Labour includes a crew of eight blasters, four helpers, an engineer and a foreman. Blasting costs are summarized in Table 21-23.

Table 21-23: Blasting cost summary

Blasting cost element	C$ per tonne moved	C$ per tonne mined
Fuel	0.00	0.00
Oil and grease, coolant, filters	0.00	0.00
Anfo	0.14	0.20
Ancillary explosives and products	0.18	0.25
Repair	0.00	0.00
Equipment rental	0.03	0.04
Maintenance labour	0.00	0.00
Labour	0.09	0.13
Total	**0.45**	**0.63**

21.3.2.3 Ore and Waste Material Loading

Loading costs are based on the use of three types of loaders:

- Two main production loaders with 10 m^3 buckets for ore and waste, and stockpile rehandle
- One loader with 6 m^3 bucket for crusher feed and concentrate loading
- One excavator with 4 m^3 bucket for wall trimming and occasional pit loading where detailed work is required.

Equipment productivity estimates are included in Section 16. LOM loading costs are summarized in Table 21-24.

Table 21-24: Loading cost summary

Loading cost element	C$ per tonne moved	C$ per tonne mined
Fuel	0.08	0.11
Oil and grease, coolant, filters	0.01	0.01
Operating supplies	0.00	0.01
Repairs	0.05	0.07
Leasing	0.05	0.07
Maintenance labour	0.06	0.09
Labour	0.06	0.09
Total	**0.32**	**0.44**

21.3.2.4 Hauling

All open pit ore and waste haulage will be undertaken with 90-tonne capacity, fixed-body trucks. The fleet will have a maximum of 8 units. The site is compact, which will help the fleet to achieve high productivity. The haulage software FPC was used to estimate the equipment productivity for each bench source and destination. Production details are available in Section 16 of this report. LOM hauling costs are summarize in Table 21-25.

Table 21-25: Hauling cost summary

Hauling cost element	C$ per tonne moved	C$ per tonne mined
Fuel	0.26	0.36
Oil and grease, coolant, filters	0.02	0.03
Operating supplies	0.02	0.03
Repairs	0.21	0.29
Leasing	0.16	0.22
Maintenance labour	0.22	0.31
Labour	0.25	0.34
Total	**1.14**	**1.58**

21.3.2.5 Ancillary Equipment

Costs for the following ancillary equipment were estimated:

- Road maintenance equipment including graders, compactors, snow ploughs, water trucks
- Tracked and wheel dozers
- Maintenance support equipment including low-boy truck; service trucks, fuel truck, cranes, forklifts
- Pit pumps
- Lighting towers
- Light vehicles
- Ambulance.

A list of the equipment types and numbers is provided in Section 16.4.2.5.

Equipment operating hours were developed based on experience and standard reference tables. LOM ancillary equipment costs are summarized in Table 21-26.

Table 21-26: Ancillary equipment cost summary

Ancillary cost element	C$ per tonne moved	C$ per tonne mined
Fuel	0.14	0.20
Oil and grease, coolant, filters	0.01	0.01
Operating supplies	0.01	0.01
Repairs	0.06	0.09
Leasing	0.14	0.20
Maintenance labour	0.08	0.11
Labour	0.22	0.31
Total	**0.66**	**0.92**

21.3.2.6 Mine Maintenance

Open pit mine maintenance labour comprises all management and workers to sustain the open pit mine mobile equipment in good working order. The estimate does not include initial equipment assembly, which is included as supplied by the OEM upon delivery and is included in the equipment lease cost. Mine maintenance labour was determined based on ratios of equipment operating hours to required labour hours. A maintenance management structure was developed appropriate to the team requirements. Maintenance labour costs were summed then pro-rated to each activity based on the promotion of equipment hourly maintenance and parts cost for each activity (drilling, blasting, etc.).

21.3.2.7 Mine Management

Mine management costs include all open pit mine management labour including:

- General and department management
- Production crew supervision
- Mining engineering
- Surveying
- Geology
- Geotechnical
- Crew training
- Grade control technicians.

Management costs include grade control analyses at an external laboratory and mining software and computers. Open pit mine management costs are summarized in Table 21-27.

Table 21-27: Open pit mine management costs

Mine management cost element	C$ per tonne moved	C$ per tonne mined
Operating supplies	0.00	0.01
Mine assays	0.03	0.04
Software and computers	0.02	0.02
Labour	0.24	0.33
Total	**0.29**	**0.40**

Note. LNG for heating and power were estimated but are insignificant values.

21.3.3 *Underground Mine Operating Costs*

Underground mine operating costs were developed on the basis of using company-employed personnel and leased equipment. The operating cost model is an activity and area-based model that was developed from first principles basis. The operating cost was estimated for all periods including pre-production. The pre-production costs were re-allocated as capital. Table 21-28:details the underground mine operating costs for the various mining periods.

Table 21-28: Underground operating cost by period (including pre-production)

Component	Units	Pre-production	Early production	Steady state	Stockpile rehandle	LOM
		Years -2 to -1	Year 1	Years 2 to 8	Years 9 to 11	Years -2 to 11
Underground ore mined	Mt	0.15	0.32	2.09	0.27	2.83
Underground ore mined[1]	tpd	327	865	817	602	738
Underground mining cost	C$/t ore mined	92.4	67.7	57.9	25.6	57.66
Underground mine life		10.5 years (including pre-production)				

(1) 1.25 years for Pre-Production and Stockpile Rehandle periods.

Table 21-29 summarizes the underground mine operating costs by activity per tonne of ore processed. The activities are broad collections of mining actions, such as stoping, which in itself includes blast-hole drilling, cable bolting, blasting, mucking, etc.). The costs are shown based on the LOM cost and tonnages produced which includes the pre-production period.

Table 21-29: Total operating cost by activity (LOM)

Total mining operating cost	C$ per tonne mined
Temporary waste drives	0.39
Development ore drives	3.91
Longitudinal longhole stoping	10.42
Material handling	7.44
Backfill	13.20
Mine maintenance	2.51
Mine general	19.80
Total	**57.66**

Table 21-30: Total operating cost by area (LOM)

Total operating cost by area	LOM C$M
Labour	65.0
Equipment	30.8
Fuel	10.9
Power	5.5
Materials	24.0
Contractor services	1.5
Equipment lease payments	25.3
Total	**163.0**

Note Total may not sum due to rounding.

Table 21-31 provides a breakdown of underground mining operating costs by area, including LOM costs related to temporary waste drives, development ore drives, longhole stoping, material handling, backfill, mine maintenance, and mine general categories.

Table 21-31: Breakdown of operating costs by area (LOM)

Operating cost by area	LOM (C$M)	Unit cost per tonne mined (C$/t)
Temporary Waste Drives		
Labour	0.5	0.15
Equipment	0.2	0.07
Fuel	0.0	0.02
Material	0.3	0.10
Total	**1.0**	**0.32**
Development Ore Drives		
Labour	6.0	2.29
Equipment	1.4	0.60
Fuel	0.4	0.14
Material	2.4	0.95
Total	**10.0**	**3.50**
Material Handling		
Labour	6.4	2.25
Equipment	7.4	2.61
Fuel	7.0	2.45
Total	**20.7**	**7.31**
Longhole Stoping		
Labour	15.5	5.47
Equipment	4.9	1.73
Fuel	2.3	0.81
Material	6.8	2.41
Total	**29.5**	**10.42**
Backfill (CRF)		
Labour	7.0	2.48
Equipment	14.9	5.28
Fuel	0.8	0.27
Material	14.6	5.18
Total	**37.3**	**13.20**
Mine Maintenance		
Labour	6.4	2.23
Equipment	0.3	0.10
Fuel	0.2	0.08
Total	**6.9**	**2.41**
Mine General		
Labour	22.3	7.93
Equipment	1.6	0.57
Fuel	1.3	0.46
Material	5.7	2.02
Infill drilling	1.5	0.54
Equipment lease payments	25.3	9.00
Total	**57.7**	**19.80**
GRAND TOTAL	**163.0**	**57.66**

Table 21-32 illustrates the input costs of major consumables. The diesel price includes carbon tax based on British Columbia's April 2021 carbon tax rate of C$45 per tonne of emissions or C$0.1171 per litre. The diesel was sourced from vendor quotations and regional suppliers and includes tax and haulage costs. Power costs were developed from first principles for the entire site. Cement was quoted by regional vendors.

Table 21-32: Major consumables

Major consumables	Unit	Value
Diesel price	C$/litre	1.17
Power price (operational - LNG)	C$/kWhr	0.18
Power price (pre-production - diesel)	C$/kWhr	0.28
Cement price (FOB mine site)	C$/tonne	370.00
Consumable delivery cost	% of opex	10%

21.3.3.1 Equipment Operating Costs

Operating costs for equipment were sourced from the Mineit's database and supplier quotes. Costs include power, wear, maintenance, tyres, oil, and lubrication. Usage costs are based on operating hours needed to fulfill the LOM plan. Table 21-33 summarises the equipment lease terms.

Table 21-33: Equipment lease terms

Equipment lease terms	Unit	Value
New or used		New
Down payment	%	25
Origination fee	%	0.6
Contract fee	C$	1580
Apr	%	5.5
Payment periods	months	48.0

21.3.3.2 Underground Mine Workforce

Estimates of the workforce complement were based on mining activities, equipment fleet operating hours, shift requirements, and maintenance requirements. These were verified against in-house data and similar operations within Canada. Annual costs considered bonuses, overtime, and benefits in addition to base salaries and hourly wages. A total average labour burden including bonuses of approximately 29.6% was calculated. Rotation schedules are assumed to be two weeks on followed by two weeks off (2/2) and two shifts per day. It is assumed that 40% of the personnel will be sourced from nearby towns, 50% from Whitehorse, and 10% for Vancouver or Edmonton.

21.3.3.3 Consumables

Costs for consumables and supplies, including fuel, power, cement, and explosives, were obtained from suppliers' quotations. Other consumable costs were generated using Mineit's database.

21.3.3.4 Lateral and Vertical Development

Most of the underground access development in waste rock is assumed to be carried out by Kutcho using leased equipment. Excavation dimensions are the same for Main and Esso. Development costs related to stope production are considered an operating cost. Main ramps, ventilation, dewatering, and electrical developments are considered capital costs. Direct costs exclude supervision, general services, maintenance, power, capital, backfill material. Lateral dimensions vary to adjust for ground control and equipment variance. Operating lateral development dimensions and cost estimates are presented in Table 21-34.

Table 21-34: Lateral development lengths and total cost

Lateral development	Main deposit (m)	Esso deposit (m)	Total (m)	Total cost (C$M)
Temporary waste drives	199	290	489	1.0
Development ore drives	1,915	3,307	5,222	11.0
Total	**4,765**	**7,637**	**12,402**	**12.0**

21.3.4 Process Plant Operating Cost

The operating costs for the processing plant are summarized in Table 21-35. The costs are separated by the major cost centres. Operating supplies (mostly reagents and grinding media), labour and power dominate the cost estimate.

Table 21-35: Plant operating cost summary by cost centre (LOM, so including pre-production)

Major cost element	LOM C$M	C$/t crushed
Labour	101.2	5.85
Power	114.4	6.61
Diesel	1.5	0.09
Maintenance supplies	19.6	1.13
Operating supplies	192.3	11.11
Freight	52.8	3.05
Miscellaneous	3.9	0.22
Mobile equipment leasing	7.1	0.41
Total	**492.8**	**28.47**

The process plant costs can also be considered by area and are shown in Table 21-36.

Table 21-36: Plant operating cost summary by area (including pre-production)

Major cost activity	C$/t crushed
Crushing	1.51
Sorting	0.64
Milling	7.72
Flotation	13.74
Concentrate handling	1.14
Tailings Thickening and Operations	0.62
Management	2.24
Laboratory	0.85
Total	**28.47**

Labour costs include operating, maintenance, maintenance, analytical and management requirements of the process plant. Total complement is estimated at 69. The plant commissioning period carries extra manning due to training requirements. Table 21-37 shows the complement by grouped task.

Table 21-37: Processing plant labour count

Processing plant labour count	Number of positions
Management	8
Operations	28
Laboratory	6
Maintenance	10
Total	**69**

Process plant power consumption was estimated by developing a list of all equipment required in the plant based on duty load and losses. The plant power consumption for the SAG and ball mills was based on test work discussed in Section 13.5. Total power consumption is estimated at 62,757 MWh per annum at steady state production rate at a LOM cost of C$0.175/kWhr. Table 21-38 outlines the power consumption by major activity.

Table 21-38: Process plant power consumption and cost by activity

Major cost activity		kWhr/t	C$/t crushed
Crushing	C$/t crushed	1.84	0.32
Sorting	C$/t crushed	0.64	0.18
Milling	C$/t milled	22.15	3.37
Flotation	C$/t milled	10.11	1.54
Concentrate handling	C$/t milled	1.17	0.18
Tailings thickening and operations	C$/t milled	2.74	0.45
Management	C$/t crushed	2.56	0.45
Laboratory	C$/t crushed	0.85	0.12
Total	**C$/t crushed**	**38.0**	**6.61**

Note. note that some items are per tonne crushed and others per tonne milled reflecting the cost driver.

Power consumption for the site includes open pit mining, underground mining processing, maintenance, camp, and administration. The estimate is dominated by plant requirements, especially comminution. Consumption details are summarized in the Table 21-39.

Table 21-39: Site-wide power consumption

Power consumption per annum, GWh														
	Total	**-2**	**-1**	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**	**11**
Underground	24.3	0.7	3.904	5.682	2.385	2.184	1.893	1.975	1.836	1.663	1.740	0.5	0	0
Open pit	1.7	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1
Plant	658.4	0	7.7	58.9	62.1	62.1	62.1	62.4	62.8	62.8	62.8	62.8	62.8	29.3
Admin	48.9	4.3	6.2	4.3	4.3	4.1	3.8	4.2	4.1	4.0	3.4	2.8	2.4	0.9
Total	**733.5**	**5.0**	**18.0**	**68.9**	**68.9**	**68.5**	**68.0**	**68.7**	**68.8**	**68.6**	**68.1**	**66.2**	**65.3**	**30.3**

Power supply until the commencement of process plant commissioning is planned to be supplied by rental diesel generators. Power requirements in construction are estimated to require up to 2,000 kW of installed diesel generation capacity (four units of 500 kW capacity each). Once commissioning commences the power plant will have an installed capacity of 10,000 kW of LNG generators in an N+1 configuration. A 2,000 kW diesel generator will be available for added start-up plant torque. Average plant loading is estimated to be around 78–80% of installed capacity. The power plant cost estimate includes:

- Lease costs for 80% of the repayment terms (initial 20% payment is included in capital for the initial up-front sum) Lease costs at a 6% APR and 72-month term
- LNG and diesel consumption
- Diesel unit rent during construction
- Power plant maintenance
- LNG storage and vaporization maintenance.

LNG costs were estimated at C$17.45/Gj including commodity pricing, liquefaction, freight to site, carbon tax and provincial taxes. Diesel costs were estimated at C$1.175/L and include commodity pricing, freight to site, and federal and provincial taxes and carbon taxes. Power costs vary during the life of operation based on whether diesel or LNG generating units are in operation, and the termination of lease costs. Table 21-40 outlines the cost of power generation per annum.

Table 21-40: Unit power cost over the life of the project

Power cost C$/kWhr														
	Total	**-2**	**-1**	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**	**11**
Total	**0.176**	0.31	0.25	0.19	0.19	0.19	0.19	0.19	0.18	0.15	0.15	0.15	0.15	0.15

Diesel for the processing plant includes the mobile ancillary fleet, excluding the loader used in the crusher feed and concentrate loading. The costs for these activities are covered in the open pit mobile fleet cost estimation. The remaining plant specific mobile units include:

- 40 tonne crane
- 100 tonne crane
- Truck with lifting arm
- Pick-ups
- Forklift
- Front-end loader (general clean-up – separate to that used for crusher and concentrate loading).

Plant mobile equipment was considered leased but the cost of lubricants, tyres, and maintenance costs is included. Cranage is considered a shared service with the truck workshop.

Maintenance supply costs were developed based on a percentage (12%) of the mechanical capital cost. The applied rate was developed from other operating facilities.

Plant operating supplies were developed mostly from quotations from suppliers for steel wear parts, grinding media, liners, and filter cloths. Dewatering supplies were based on a percentage (4%) of the mechanical capital. The steel wear parts and regrind media consumption rates were based on abrasion test work. Regrind media consumption rates and costs were developed from anticipated average consumptions. Regrind consumption costs are not based on specific test work.

The costs of process plant consumables are presented in Table 21-41.

Table 21-41: Process plant operating supplies unit cost

Operating supplies	Unit	Cost
Jaw crusher parts	C$/t crushed	0.03
Jaw crusher liners	C$/t crushed	0.03
Conveyor Idlers, etc.	C$/t crushed	0.03
Cone crusher liners	C$/t crushed	0.05
Sag mill balls	C$/t milled	0.41
Sag mill liners	C$/t milled	0.29
Ball mill balls	C$/t milled	1.09
Ball mill liners	C$/t milled	0.11
Regrind mill grind media	C$/t milled	0.20
Regrind mill liners	C$/t milled	0.28
Filter cloths	C$/t milled	0.29
Dewatering supplies	C$/t milled	0.03
Reagents	C$/t milled	9.56
Laboratory supplies	C$/t crushed	0.16
Management	C$/t crushed	0.01

Operating supplies	**Unit**	**Cost**
Ore Sorter supplies	C$/t crushed	0.10
Total	**C$/t crushed**	**11.11**

The cost of process plant consumable reagents is presented in Table 21-42.

Table 21-42: Processing plant reagents costs

Reagents	**kg/t of ore**	**C$/kg**	**FOB Point**	**C$/t milled**
Zinc Sulphate	1.500	1.60	Delta, BC	2.40
Copper Sulphate	0.478	3.42	Delta, BC	1.64
Aero 3894/Polyfloat 2979	0.078	4.61	Delta, BC	0.36
Aero 3477	0.078	3.28	Delta, BC	0.26
Na2S	0.750	1.45	Delta, BC	1.09
SMBS	1.153	0.72	Delta, BC	0.83
Liquid SO_2	2.430	0.77	Prince George	1.87
W31 Frother	0.023	5.07	Delta, BC	0.12
MIBC frother	0.030	3.36	Delta, BC	0.10
Lime – Quicklime	2.298	0.33	Delta, BC	0.76
Flocculant – Magnafloc MF351	0.030	5.07	Delta, BC	0.15
Total				**9.56**

Reagent consumption rates were based on 2021 test work conducted by BML. The consumption rates were reduced to 75% of the test work values based on experience of the QP.

Freight was estimated for each reagent from the point of supply and totals C$3.05/t crushed. Freight rates were developed from quoted rates for regional suppliers at C$0.185/t/km. Some freight charges were estimated at 10% of the reagent direct cost. About 80% of the freight cost is associated with reagent supply and the bulk of the remainder to steel wear parts.

An amount of C$30,000 per month was allocated for miscellaneous costs associated for consulting and specialized maintenance charges.

The annual plant operating cost varies due to the plant start up inefficiencies and variations in plant power costs (due to lease charges and diesel v LNG generation). Table 21-43 shows the annual variation in cost per tonne crushed per annum.

Table 21-43: Annual processing plant operating unit costs per tonne crushed (LOM)

Process operating cost, C$/t crushed per annum													
	Total	**-1**	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**	**11**
Labour	5.85	14.49	6.01	5.70	5.70	5.70	5.70	5.70	5.70	5.70	5.70	5.70	6.08
Power	6.61	8.26	7.26	7.26	7.27	7.28	7.30	6.94	5.75	5.75	5.75	5.75	5.76
Diesel	0.09	0.17	0.09	0.09	0.09	0.09	0.09	0.09	0.09	0.09	0.09	0.09	0.09
Maintenance supplies	1.13	1.72	1.14	1.12	1.12	1.12	1.12	1.12	1.12	1.12	1.12	1.12	1.16
Operating supplies	11.11	11.27	11.12	11.11	11.11	11.11	11.11	11.11	11.11	11.11	11.11	11.11	11.12
Freight	3.05	3.07	3.05	3.05	3.05	3.05	3.05	3.05	3.05	3.05	3.05	3.05	3.05
Miscellaneous	0.22	0.44	0.23	0.22	0.22	0.22	0.22	0.22	0.22	0.22	0.22	0.22	0.23
Leasing	0.41	0.80	0.42	0.40	0.40	0.40	0.40	0.40	0.40	0.40	0.40	0.40	0.43
Total	**28.47**	**40.21**	**29.32**	**28.94**	**28.95**	**28.97**	**28.99**	**28.62**	**27.43**	**27.43**	**27.43**	**27.44**	**27.92**

The plant operating cost is based on the processing of sulphide ore. Around 2.5% of the ore is oxidized or partially oxidized. It is considered appropriate to apply the sulphide ore process costs to the oxide material for the Feasibility Study.

No operating contingency is applied to the plant operating cost estimate. However, it must be noted that the plant operating cost in the commissioning period, defined as a capital cost, attracts an 11% contingency.

21.3.5 General and Administration

General and administration operating costs are summarized in Table 21-44.

Table 21-44: General and administration costs (including the pre-production period, Years -2 and -1)

			-2	-1	1	2	3	4	5	6	7	8	9	10	11
Camp	C$M	57.2	6.5	8.6	4.8	4.8	4.5	4.1	4.6	4.5	4.3	3.4	3.1	2.8	1.1
Transportation	C$M	7.6	0.8	1.1	0.6	0.7	0.6	0.5	0.6	0.6	0.6	0.5	0.4	0.4	0.2
Freight	C$M	5.1	0.5	0.7	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.3	0.3	0.3	0.1
Administration	C$M	12.6	0.6	1.1	1.2	1.2	1.2	1.1	1.2	1.2	1.2	1.1	0.8	0.6	0.2
IT	C$M	1.9	0.1	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.1	0.1	0.0
Safety	C$M	2.7	0.1	0.2	0.3	0.3	0.3	0.3	0.3	0.3	0.3	0.2	0.1	0.1	0.0
Environmental	C$M	1.9	0.1	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.1	0.1	0.0
Hr	C$M	2.0	0.1	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.1	0.1	0.0
Water treatment	C$M	36.5	1.9	3.3	3.4	3.4	3.4	3.4	3.4	3.4	3.3	3.3	2.3	1.7	0.5
Site services	C$M	24.3	1.5	2.5	2.3	2.3	2.1	1.9	1.9	1.9	1.9	1.8	1.8	1.7	0.0
Labour	C$M	35.3	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	0.0
Total	**C$M**	**186.9**	**15.0**	**20.9**	**16.3**	**16.4**	**15.8**	**15.0**	**15.7**	**15.6**	**15.3**	**13.9**	**12.0**	**10.6**	**0.0**
Total C$ per tonne crushed		**10.80**	**-**	**102.14**	**10.46**	**9.96**	**9.62**	**9.11**	**9.55**	**9.48**	**9.30**	**8.48**	**7.33**	**6.47**	**15.79**

21.3.5.1 Camp

Camp costs include:

- Power
- Room and catering charges
- Camp labour and management.

Power consumption per person (C$1.75 kW/person in camp) for the camp and catering was based on data from other operations and similarly located projects.

Room and catering charges were based on data from similar projects and budget quotes at C$45 per person per day in camp, excluding catering, cleaning and camp management wages, heating, power, and freight.

Camp personnel requirements for cleaning, catering, maintenance, and management were based on data from similar projects and quotations. Camp labour complements fluctuate as a function of the number of people in camp.

The numbers of people in camp were derived from detailed personnel requirements estimated for the open pit mine, the underground mine, the processing plant, and administration. Construction personnel complements were mostly based on detailed estimates of working hours as well as some factoring for less well-defined works. Some allowances were made for visitors, occasional vendor maintenance, maintenance personnel, unplanned haulage stop-overs, and TMF construction personnel. Total camp numbers fluctuate from a maximum of 350 during construction to a more regular 190 during operations, to 120 during the end-of-life stockpile reclaim period. During operations the camp has a maximum of 250 beds.

21.3.5.2 Personnel Transportation

Transportation costs include:

- Flights from Whitehorse to site. Each flight is assumed to deliver 18 people. Budget quotes were obtained. These flights are assumed to account for half of the workforce.
- Commercial flights from Vancouver to Whitehorse are assumed applied to 10% of the workforce.
- Bus or car trips to Dease Lake from surrounding towns are considered to cover 30% of the workforce.
- Around 10% of the workforce were assumed local at Dease Lake.
- Around 40% of the workforce would be bused from Dease Lake to site.

The mining cost also covers two manned crew buses for transport to the airport and transport within the mine site. The mine operations, plant operations, and administration costs also cover light vehicle hire and operational costs.

21.3.5.3 Administration Freight

Based on quotes and estimations for food and general freight, freight costs were estimated for road haulage from Prince George. Freight costs in this category vary proportionally to the number of people present in the camp.

21.3.5.4 Administration

Administration costs were mostly determined from comparison to other projects and operations. Heating costs (LNG) were determined based on a standard consumption per quarter per person in camp. The cost areas were determined as:

- LNG for heating offices
- Training
- Administration, light vehicles, leasing maintenance and fuel
- Office supplies
- Whitehorse building rent and utilities
- Mining lease fees
- Legal counsel
- Auditing and tax advice services
- Bank charges and fees
- Community and public relations
- Insurance
- External consultants.

21.3.5.5 Information Technology (IT)

IT costs were mostly determined from comparison to other projects and operations. The mobile phones and satellite communications cost were obtained from a quotation. The cost areas were determined as:

- Computers
- Software and support contracts
- Telephone land-line services

- Mobile phones and charges
- Satellite telecommunications.

21.3.5.6 Safety

Safety costs were mostly determined from comparison to other projects and operations. Uniforms were based on the number of employees per annum. The cost areas were determined as:

- Safety operating materials and tools (sitewide)
- Administration group safety supplies
- Uniforms and safety supplies (sitewide)
- Safety incentive programs and personnel events (site-wide).

21.3.5.7 Environmental

Environmental costs were mostly determined from comparison to other projects and operations. Environmental costs do not cover any rehabilitation cost or associated investigations (these are instead estimated as a capital cost). The costs shown here would allow the environmental personnel to attend to minor spills, regular site monitoring and inspections. The cost areas were determined as:

- Environmental operating materials & tools
- Environmental external services.

21.3.5.8 Human Resources Management

Human resources costs were mostly determined from comparison to other projects and operations. The cost areas were determined as:

- Recruitment and relocation
- Medical expenses
- Human resources external services
- Incentive programs and personnel events.

21.3.5.9 Water Treatment

Water treatment costs were developed with quotations obtained through the water treatment equipment vendor. The plant is designed to treat up to 320 L/s for six months during months where water could be released (ice-free) and 112 L/s during the other six months when water would be stored in the TMF. Annual flow through the unit is estimated to be 5.7 million m^3 per annum. Total costs for water processing are estimated to be C$0.59/$m^3$ or C$0.19/tonne ore crushed. Power and reagent costs vary throughout the year. The water treatment cost includes:

- Power
- Reagents
- Reagent freight
- Maintenance.

Reagents include coagulants, precipitants, anionic polymers, sodium hydroxide, sulfuric acid and micro-sand. Reagents and reagent freight represents 83% of the water treatment cost.

The plant capacity is designed to match the Project water balance. However, should a year be wetter than expected or the release period shorter, the water treatment capacity can be augmented through the addition of modular rental units. The rental units are capable of processing 70 L/s. No costs were allocated for rental units.

21.3.5.10 Site Services

Site service costs include a maintenance team that will complete site and access road maintenance, site electrical maintenance, rubbish collection and incineration. Along with operators assigned to specific equipment this team includes:

- Foremen (two)
- Electricians (four)
- Welders/Journeymen (four)
- Apprentices (two).

The costs for this area cover the equipment and power for service buildings not covered elsewhere such as:

- Warehouse
- Diesel fuel
- LNG
- Gatehouse
- Sewage
- Incinerator
- Mine office complex
- Emergency response team (ERT) equipment.

21.3.5.11 Administrative Personnel

Labour rates for administration were developed as noted in other sections of this report. The administration rates consider that some personnel will be stationed in Whitehorse. The following table (Table 21-45) outlines the number of positions during an average production year.

Table 21-45: Administrative positions

Position	Number
OHS&T Manager	1
Commercial and Administration Manager	1
Environmental and Community Manager	1
IT Officer/Business systems	2
Site Accountant	1
Accounts Payable Clerk	2
Payroll Clerk	1
Administration Clerk	1
Store Manager and Procurement Supervisor	1
Purchasing Officer	1
Stores Personnel	2
Gatehouse Security	4
Environmental Officer	1
Environmental Technician	2
Community liaison/Mentor	1
Site Nurse/ERT	2
Emergency Services Officer/ERT	2

22 Economic Analysis

The results of the economic analysis represent a deterministic calculation of the NPV, IRR, and payback period of the project using acceptable and probable values for variables that are subject to a number of known and unknown risks, uncertainties and other factors that may cause actual results to differ from those presented here. The outcome of this economic analysis may vary from the actual future outcome of the Project due to the inherent uncertainty in many variables which include, but are not limited to, the future price of metals, the estimation of Mineral Reserves and Mineral Resources, the realization of the Mineral Reserve estimate, the timing and amount of estimated future production, costs of production, capital expenditures, results of the permitting process, currency exchange rate fluctuations, requirements for additional capital, government regulation of mining operations and taxation, environmental risks, unanticipated reclamation expenses, title disputes or claims and limitations on insurance coverage.

Additional risks affecting the economic outcome of the project can come from changes in the project parameters as plans are refined, variations in Mineral Reserves, grade or recovery rates, failure of plant, equipment or processes to operate as anticipated, accidents, labour disputes and other risks of the mining industry, including potential delays in obtaining regulatory approvals.

The financial evaluation presents the determination of the key economic performance indicators for the Project, including the Net Present Value (NPV), the payback period (time in years to redeem the initial capital investment), and the Internal Rate of Return (IRR) for the project. The discounted cash flow (DCF) model is reported at 100% attributable equity. Annual cash flow projections are estimated over the life of the mine based on the estimates of capital expenditures, production costs and sales revenues. The sales revenue is based on the production of a copper concentrate and a zinc concentrate that are planned to be sold to refineries based in the East Asia region.

The estimates of initial and sustaining capital expenditures and site production costs were developed specifically for this project and are presented in earlier sections of this report. Total initial and sustaining capital is C$493 million and C$99 million, respectively (see Section 21 – Capital and Operating Costs for details and Table 22-18.

The NPV at assumed long term metal prices using a 7% discount rate is C$461 million and the internal rate of return is 25% (Table 22-18). Payback of the initial capital occurs 3.4 years after commercial production commences (Table 22-18). The plant produces a payable total of 504 Mlb of copper, 531 Mlb of zinc, 8.3 Moz of silver, and 107 koz of gold over a 10.5-year processing life (Table 22-3 and Table 22-7). The processing life is preceded by a two-year on-site construction period and commissioning period, which is itself preceded by a two-year off-site road construction period. The processing life is followed by a period for closure and rehabilitation.

Table 22-1: Key project overview and metrics

Component	Early production	Steady state	Stockpile rehandle	LOM
	Year 1	Year 2 to 8	Year 9 to 11	Year 1 to 11
Total Payable Copper Metal	41 Mlb	385 Mlb	77 Mlb	504 Mlb
Total Payable Zinc Metal	31 Mlb	435 Mlb	65 Mlb	531 Mlb
Total Payable Silver Metal	0.6 Moz	6.6 Moz	1.1 Moz	8.3 Moz
Total Payable Gold Metal	9 koz	81 koz	20 koz	107 koz
Average Operating Costs	C$123 M	C$116 M	C$64 M	C$102 M
Annual Average Sustaining Capital plus Operating Costs	C$149 M	C$126 M	C$69 M	C$112 M
Initial Capital - including 10.6% contingency incl. Mine Closure Bond	C$493 M			
Sustaining Capital incl. Mine Closure Bond but excl, rehabilitation	C$99 M			

Component	Early production	Steady state	Stockpile rehandle	LOM
	Year 1	Year 2 to 8	Year 9 to 11	Year 1 to 11
Closure/rehabilitation Capital	C$34 M			
Salvage	C$(18) M			
Mine Closure Bond Reclamation	C$(19) M			
Total Capital (Initial, sustaining, salvage and closure/rehabilitation)	C$589 M			
After-Tax Net Present Value - 7% discount rate	**C$461 M**			
Annual Average Net Revenue (NSR total less Royalties)	C$221 M	C$332 M	C$143 M	C$270 M
Annual Average Taxation	C$2 M	C$49 M	C$24 M	C$38 M
Annual Average After Tax Free Cash Flow	C$70 M	C$157 M	C$50 M	C$158 M
After Tax Internal Rate of Return	**24.8%**			
After Tax Payback Period	**3.4 years**			

Table Notes

- *Base case prices US$3.50/lb, zinc US$1.15/lb, silver US$20/oz, gold US$1,600/oz.*
- *Tonnages are reported in metric tonnes, copper and zinc in pounds (lbs), silver and gold reported in troy ounces (oz).*
- *"M" = million, "k" = thousand.*
- *All tables report rounded figures and may not sum precisely.*
- *The financial model is based on 100% of the project is being financed through equity (no equity from other projects can be used to offset the cost of capital). No debt or equity schedule is included.*
- *The highlights refer to the Feasibility Study base case. The Wheaton Precious Metals Purchase Agreement (PMPA) is not applied to the Feasibility Study base case.*
- *Method of discounting for NPV – all capital years are undiscounted, discounting commences in Year 1.*
- *The after tax NPV for commencing discounting in Year -3 is C$360 million, the NPV for commencing discounting in Year -2 is C$388 million and the NPV for commencing discounting in Year -1 is C$431 million.*
- *Cash costs are operating expenses for mining, plant operations and administration to the point of production of the concentrate at the Kutcho site. It excludes off-site concentrate costs, sustaining capital, closure/rehabilitation and royalties.*
- *All-in Sustaining Costs includes all cash costs and adds to this all capital expenses to support ongoing operations such as TMF construction, major plant equipment replacement and repair. It does not include closure/rehabilitation.*
- *No inflation or depreciation of costs were applied, all costs are in 2021 money terms.*
- *Major underground mobile equipment, all open pit mobile equipment and the power plant are leased.*
- *Metal prices are held constant.*

The economic analysis and supporting financial information, including capital and operating costs, were developed in constant dollar terms. The economic analysis uses the Probable and Proven Mineral Reserves as described in the Mineral Reserve Estimate of this report. Cash flow forecasts on an annual basis using the Mineral Reserves for the base case metal price are included in Table 22-7.

Sensitivity analysis charts are presented as Figure 22-4 and Figure 22-5.

22.1 DCF, Exchange Rate, Funding, Corporate Costs and NPV

The DCF method of financial valuation used for the Project considers that all capital years are undiscounted, discounting commences in Year 1. The after-tax NPV for commencing discounting in Year -3 is C$360 million, the NPV for commencing discounting in Year -2 is C$388 million and in Year -1 is C$431 million. The DCF model is reported at 100% attributable equity. Key inputs to the financial valuation such as the ore production profile, operating costs and capital costs are described in detail in the preceding sections of this report.

The DCF model utilized Canadian dollars (C$) as the base currency as the majority of capital and operating cost estimates are based in C$.

Cash flows are discounted at 7% to obtain an NPV of the Project. The choice of discount rate was based on using a capital asset pricing model (CAPM) with the weighted average cost of capital (WACC) method.

To calculate WACC, the cost of equity is multiplied by the % of equity in the company's capital structure and adds to it the cost of debt multiplied by the % of debt on the company's structure. Because interest in debt is a pre-tax expense, the cost of debt is reduced by the tax rate (effectively tax deductible).

The formula is:

$$WACC = \frac{Ve}{Ve + Vd} \, x \, Ke \quad + \quad \frac{Vd}{Ve + Vd} \, x \, Kd \, x \, (1 - T)$$

Where:

- Ke = the cost of equity.

The basic CAPM formula for Ke is:

$$Ke = Rf + \beta * (Rm - Rf) + \alpha$$

Where:

- Rf = Risk free rate of return
- B (Beta) = Sensitivity of the expected stock return to the market return.
- Rm = Market rate of return
- A (Alpha) = Additional risk
- Kd = cost of debt. This is the average interest rate on the company's debt.
- T = corporate tax rate.
- Ve = value of equity. Company market capitalization less cash plus debt.
- Vd = value of debt.

It was considered appropriate to use financial parameter estimates for the company more likely to be in effect as the company transitions to construction rather than present conditions. The following assumptions were made, based mostly on Canadian mining projects:

- Cost of equity 8.7% (Country risk of 4.7%, risk free rate of 1.9% and additional risk of 1.5%)
- Market variability (Beta) of 1.9
- Debt to equity weighting of 1:1.

Other than in the development of the discount rate, the financial model does not include any consideration of funding or funding costs. There are no corporate administration overheads applied. It is important to state that the precious metals purchase agreement (PMPA), should it be affirmed by Wheaton after the disclosure of this Feasibility Study, will alter the values for cash flow, NPV, and IRR. It must also be understood that the PMPA delivers some of the capital funding requirements, so partly de-risking the Project in that respect. Specific details of the PMPA are supplied in Section 19.

22.1.1 ***Working Capital***

A working capital requirement is estimated for the first quarter of operations. The value is estimated based on the net monthly difference between revenue and operating cost. Adjustments are applied in operating cost for taxation equity. The quantity amounts to C$5.4 million.

22.1.2 Taxes and Royalties

Investment and new mine allowances have been applied against the BC Mineral Tax. A 2% provincial minimum tax payable on net current proceeds, which is credited against the Mineral Tax, is calculated based on operating profit less applicable capital cost deductions. The mining tax is deductible in computing provincial and federal income tax. Canadian Development Expenses and Canadian Exploration Expenses have been applied to the tax model. Total taxes payable over the life of the Project (undiscounted) are estimated at C$422 million after allowing for tax credits of C$30 million accumulated by Kutcho Copper to date. It is an assumption of the Project that tax deductions associated to the new mines provision will continue to apply.

Table 22-2: Total taxation and royalty estimates

Item	Unit	Value
Net Present Value Discount Rate	%	7.0
BC Mineral Tax Rates	%	13.0
BC Provincial Income Tax Rates	%	12.0
Federal Income Tax Rates	%	15.0
Opening Balance Federal Investment Tax Credits	C$M	0.0
Opening Balance Canadian Development Expenses	C$M	10.2
Opening Balance Canadian Exploration Expenses	C$M	18.7
Foreign Resource Expense Tax Pool	C$M	1.3
BC Mineral Tax	C$M	136.1
BC Income Tax	C$M	127.3
Federal Income Tax	C$M	159.1
Total Taxes Estimated	C$M	422.5
Royal Gold Royalty	C$M	26.1

The Royal Gold Royalty applies to part of the Main deposit Mineral Reserve and all the Esso deposit Mineral Reserve as described in Section 4.5.1. Royalty estimates were completed based on the NSR calculation for ore within the applicable Royal Gold Royalty lease boundaries. Figure 22-1 shows the areas attributable to the Royal Gold Royalty. The royalty was applied to ore crushed. Therefore, ore mined but placed on stockpile does not incur the royalty payment until it is crushed. Stockpiles are, however, undifferentiated, and so average royalty payments were applied to stockpiled ore. Checks of original Reserve estimates of the royalty and final scheduled values were undertaken to verify that the distribution was correctly applied in total.

The royalty rights of Sumac Mines Inc. were extinguished in an agreement signed between Sumac Mines Inc. and Kutcho Copper in 2021 and reported in a press release dated 6 July 2021. Certain payment terms within the agreement are contemplated to be completed after the publication of this report. However, for the purposes of this study, it was assumed that the Sumac royalty was extinguished.

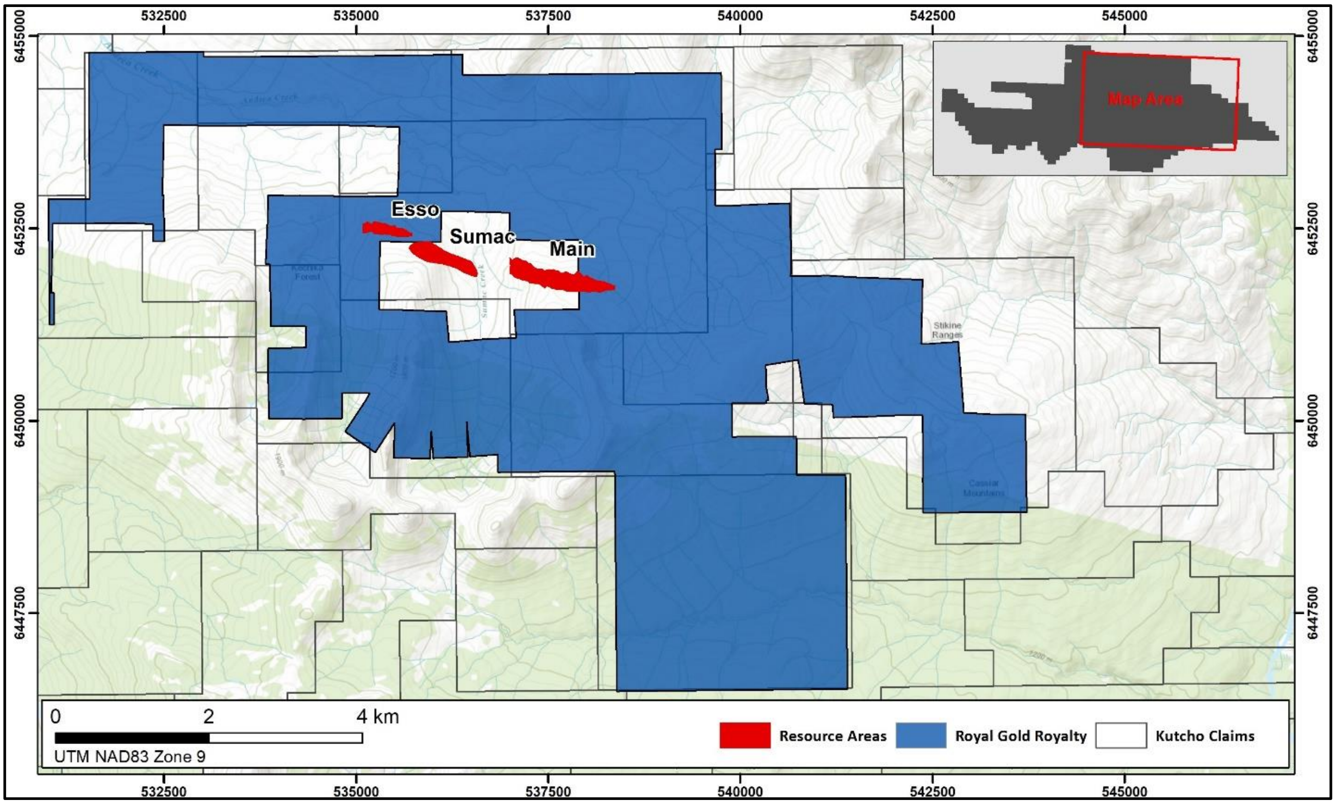


Figure 22-1: Leases Attributable to the Royal Gold Royalty

22.1.3 Revenue

The DCF schedule allows for build up of 2,000 wmt of concentrate for each concentrate at the processing plant at start-up (to allow for fill of thickeners and plant storage) and a further build up of 5,000 wmt of stock for each concentrate prior to first sales revenue. The delay in concentrate sales versus production is carried through all periods until the final period where the additional 7,000 wmt for each concentrate are added to revenue. The quantity allows for fill of the concentrate thickeners and storage at the plant plus a 5,000 wmt lot for sale at the port of Stewart. The quantity is reclaimed at end of processing plant life.

Metal prices and sales terms are assumed remain unchanged through the life of the project.

22.2 Key Financial Assumptions and Indicators

Key financial assumptions are presented in Table 22-3 to Table 22-6.

Table 22-3: Key project mining statistics

Component	Units	Pre-production	Early production	Steady state	Stockpile rehandle	LOM
		Year -2 to -1	Year 1	Year 2 to 8	Year 9 to 11	Year -2 to 11
Underground Ore Mined	Mt	0.15	0.32	2.09	0.27	2.83
Underground Ore Mined	tpd	327	865	817	602	738
Underground Mining Cost	C$/t ore mined	92.4	67.7	57.9	25.6	57.7
Underground Mine Life including pre-production		10.5 years				
Open Pit Ore Mined	Mt	0.5	1.6	12.4	0.0	14.5
Open Pit Waste Mined	Mt	11.5	12.1	58.2	0.0	81.8
Open Pit Strip Ratio	W/O	22.8	7.7	4.7	0.0	5.6

Component	Units	Pre-production	Early production	Steady state	Stockpile rehandle	LOM
		Year -2 to -1	Year 1	Year 2 to 8	Year 9 to 11	Year -2 to 11
Open Pit Ore Rehandle	Mt	0.0	0.0	0.9	3.8	4.7
Open Pit Waste Rehandle	Mt	0.0	0.0	5.1	26.5	31.6
Open Pit Total Movement	Mt	12.0	13.7	76.6	30.3	132.5
	tpd	16,400	37,562	29,978	27,645	27,934
Open Pit Mining Cost	C$/t moved	4.94	3.33	3.33	1.57	3.07
	C$/t ore mined	117.28	28.93	20.55	na	28.10
Open Pit Mine Life – incl. pre-prod and rehandle		12.5 years				

Table 22-4: Key project plant statistics

Component		Units	Pre-production	Early production	Steady state	Stockpile rehandle	LOM
			Year -2 to -1	Year 1	Year 2 to 8	Year 9 to 11	Year -2 to 11
Processing life (1)		years	10.75 years				
Ore crushed	Ore crushed	kt	205	1,556	11,498	4,053	17,311
	Ore crushed (2)	tpd	2,275	4,264	4,500	4,037	4,312
Concentrate produced	Cu concentrate	kt	10	83	698	145	936
	- Cu grade	%	16.0	24.0	26.1	24.0	25.5
	- Zn grade	%	11.6	11.5	9.0	16.2	10.4
	- Ag grade	g/t	242.5	229.5	325.1	257.5	305.3
	- Au grade	g/t	1.8	2.4	2.7	2.4	2.6
	Zn concentrate	kt	2	38	418	57	516
	- Cu grade	%	5.2	1.0	0.5	1.3	0.6
	- Zn grade	%	35.2	52.2	55.6	54.4	55.1
	- Ag grade	g/t	390.8	96.8	87.2	68.1	87.2
	- Au grade	g/t	6.1	3.5	2.9	4.0	3.1
Metal produced	Copper	Mlb	4	45	406	78	533
	Zinc	Mlb	4	65	652	120	841
	Silver	Moz	0.1	0.7	8.5	1.3	10.6
	Gold	koz	1.1	10.8	99.2	18.6	129.7
Produced metal recovery (3)	Copper	%	60.1%	84.2%	90.3%	83.6%	88.4%
	Zinc	%	79.0%	92.8%	95.6%	96.7%	95.4%
	Silver	%	54.6%	62.4%	72.0%	55.6%	68.6%
	Gold	%	43.1%	58.0%	61.5%	58.2%	60.5%
Payable metal (4)	Copper	Mlb	1	41	385	77	504
	Zinc	Mlb	0	31	435	65	531
	Silver	Moz	0.0	0.6	6.6	1.1	8.3
	Gold	koz	0.1	8.8	81.5	16.6	107.0

Table notes:

(1) 10.75 years after pre-production, plus 0.25 years in pre-production start-up.

(2) 0.25 years for Pre-Production and 2.75 for Stockpile Rehandle periods.

(3) Recovery is of metal produced and includes effect of loss of metal to sorter reject and tails and is compared to crusher head; it does not include concentrate transportation losses.

(4) Payable Metal includes timing delays on concentrate allotments, inventory builds and 0.25% tonnage loss in transport and handling.

Table 22-5: Summary of LOM operating costs (per tonne of ore processed)

Cost component	Units [2]	Early production	Steady state	Stockpile rehandle	LOM
		Year 1 [1]	Year 2 to 8	Year 9 to 11	Year 1 to 11
Open pit mining	C$/t OP ore mined	27.68	20.55	NA [3]	24.75
Underground mining	C$/t UG ore mined	62.17	57.88	25.61	55.38
Processing	C$/t ore crushed	28.06	28.48	27.53	28.21
General and administration	C$/t ore crushed	10.01	9.36	6.69	8.79
Total	**C$/t ore crushed**	**79.24**	**70.51**	**47.67**	**65.89**

Table notes:

(1) Year 1 includes pro-rated adjustments for working capital.

(2) Pre-production tonnages and costs are not included in the LOM operating cost summary (these are Years -2 and -1 and are capitalized).

(3) No ore mined in rehandle period.

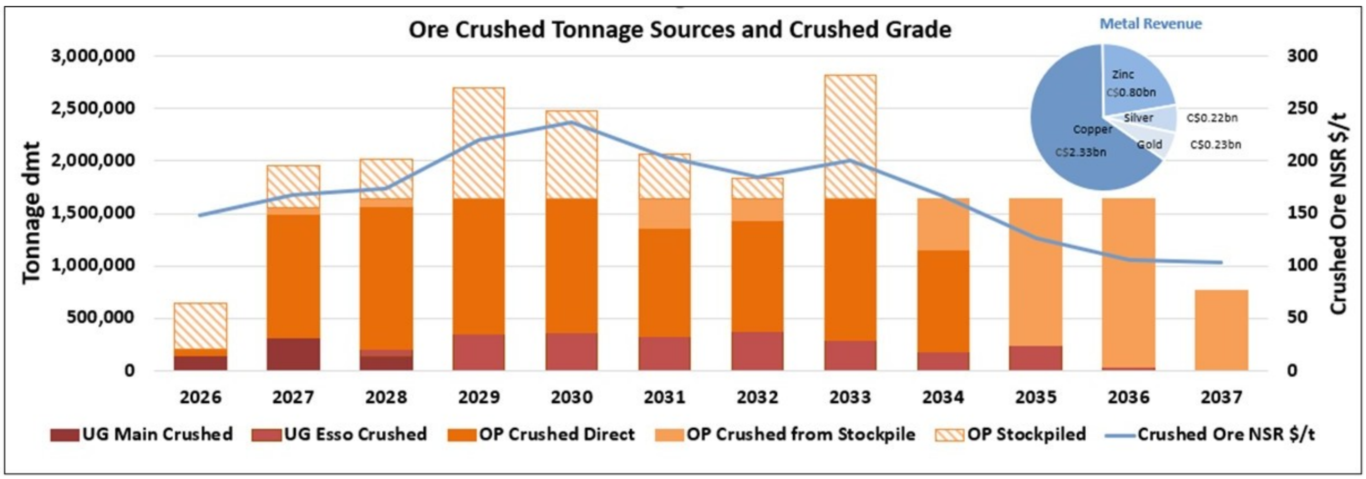


Figure 22-2: Crushed Tonnage Sources and Crushed Grade Reflected as NSR C$/t

Note: Metal revenue per metal pie chart.

Table 22-6: Summary of total capital costs

Component		Units	Initial	Sustaining	Closure	Total
Direct Costs	Mine Costs	C$M	132.9	44.8	0.0	177.7
	Processing Plant	C$M	106.0	15.1	0.0	121.2
	On-Site Infrastructure	C$M	54.2	15.8	0.0	70.1
	Off-Site Infrastructure	C$M	31.7	0.0	0.0	31.7
EPCM and Indirect Costs		C$M	75.3	0.0	0.0	75.3
Owner's Costs (including working capital)		C$M	35.9	5.6	0.0	41.5
Total Capex without Contingency		**C$M**	**436.1**	**81.3**	**0.0**	**517.5**
Contingency		C$M	46.7	8.2	0.0	54.9
Salvage		C$M	0.0	0.0	-18.0	-18.0
Mine Closure Bond		C$M	10.0	9.3	-19.3	0.0
Rehabilitation, Monitoring and Closure Costs		C$M	0.0	10.2	24.3	34.5
Total Capex with Rehabilitation		**C$M**	**492.8**	**109.0**	**-13.0**	**588.9**

22.3 Cash Flow Forecasts

The DCF schedule and analysis is presented in Table 22-7 to Table 22-17.

Table 22-7: Full discounted cashflow analysis – mining data

Mining	Units	Total	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038
OPEN PIT PRODUCTION																	
Ore mined	Mt	14.5	0.0	0.0	0.5	1.6	1.7	2.3	2.1	1.5	1.3	2.5	1.0	0.0	0.0	0.0	0.0
Copper grade	%	1.50	0.00	0.52	1.21	1.44	1.39	1.65	1.74	1.39	1.34	1.49	1.44	0.00	0.00	0.00	0.00
Zinc grade	%	2.06	0.00	0.74	1.58	2.02	2.32	2.43	2.38	1.78	1.86	1.80	1.73	0.00	0.00	0.00	0.00
Silver grade	g/t	23.5	0.0	11.6	15.7	20.2	21.6	25.7	25.8	25.5	23.7	23.7	22.8	0.0	0.0	0.0	0.0
Gold grade	g/t	0.33	0.00	0.19	0.21	0.28	0.29	0.38	0.39	0.48	0.28	0.27	0.24	0.00	0.00	0.00	0.00
Throughput ore	tpd	3,200	0	0	1,400	4,300	4,700	6,400	5,800	4,000	3,400	6,900	2,700	0	0	0	0
Waste mined	Mt	81.8	0.0	0.6	10.9	12.1	11.9	9.2	6.0	11.3	11.4	7.6	0.9	0.0	0.0	0.0	0.0
Strip ratio	(W/O)	5.6	0.0	575.7	21.7	7.7	6.9	3.9	2.8	7.7	9.1	3.0	0.9	0.0	0.0	0.0	0.0
Total mined	Mt	96.2	0.0	0.6	11.4	13.7	13.6	11.5	8.1	12.7	12.6	10.1	1.8	0.0	0.0	0.0	0.0
Throughput total	**tpd**	**21,500**	**0**	**2,000**	**31,300**	**37,600**	**37,300**	**31,500**	**22,200**	**34,900**	**34,600**	**27,800**	**5,100**	**0**	**0**	**0**	**0**
UNDERGROUND PRODUCTION																	
Ore mined – Main	Mt	0.6	0.0	0.0	0.1	0.3	0.1	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Ore mined – Esso	Mt	2.2	0.0	0.0	0.0	0.0	0.1	0.4	0.4	0.3	0.4	0.3	0.2	0.2	0.0	0.0	0.0
Ore mined – Total	**Mt**	**2.8**	**0.0**	**0.0**	**0.1**	**0.3**	**0.2**	**0.4**	**0.4**	**0.3**	**0.4**	**0.3**	**0.2**	**0.2**	**0.0**	**0.0**	**0.0**
Copper grade	%	1.99	0.00	1.14	1.39	1.40	1.59	1.67	2.17	2.34	2.36	2.21	2.40	2.09	1.42	0.00	0.00
Zinc grade	%	3.58	0.00	1.19	1.13	1.62	2.47	2.97	3.82	4.41	4.68	4.67	4.23	4.43	4.41	0.00	0.00
Silver grade	g/t	50.1	0.0	29.5	34.4	32.0	33.5	51.8	56.2	67.6	56.6	46.8	59.4	50.7	44.2	0.0	0.0
Gold grade	g/t	0.68	0.00	0.50	0.42	0.69	0.48	0.58	0.72	0.92	0.67	0.66	0.67	0.75	1.08	0.00	0.00
Ore throughput	tpd	770	0	40	400	860	580	970	980	890	1,010	800	480	880	0	0	0
Waste mined	Mt	0.7	0.0	0.0	0.3	0.2	0.1	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
TOTAL MINED (Underground Plus Open Pit)																	
Waste mined	Mt	82.5	0.0	0.6	11.2	12.4	12.0	9.2	6.0	11.3	11.4	7.6	0.9	0.0	0.0	0.0	0.0
Ore mined	Mt	17.3	0.0	0.0	0.6	1.9	1.9	2.7	2.5	1.8	1.6	2.8	1.1	0.2	0.0	0.0	0.0
Total mined	**Mt**	**99.8**	**0.0**	**0.6**	**11.9**	**14.3**	**13.9**	**11.9**	**8.5**	**13.1**	**13.0**	**10.4**	**2.0**	**0.2**	**0.0**	**0.0**	**0.0**
REHANDLE (Trucked)																	
Waste rehandled	Mt	31.6	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	5.1	13.1	10.7	2.7	0.0
Ore rehandled (1)	Mt	4.7	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.2	0.2	0.0	0.5	1.4	1.6	0.8	0.0
TOTAL MOVED (Open Pit with Rehandle)																	
Waste moved	Mt	113.4	0.0	0.6	10.9	12.1	11.9	9.2	6.0	11.3	11.4	7.6	6.0	13.1	10.7	2.7	0.0
Ore moved	Mt	19.2	0.0	0.0	0.5	1.6	1.7	2.3	2.1	1.6	1.5	2.5	1.5	1.4	1.6	0.8	0.0
Total moved	**Mt**	**132.5**	**0.0**	**0.6**	**11.4**	**13.7**	**13.6**	**11.5**	**8.1**	**12.9**	**12.8**	**10.1**	**7.5**	**14.5**	**12.3**	**3.5**	**0.0**

(1) From long term stockpile to ROM or crusher.

Table 22-8: Full discounted cashflow analysis – process plant data

Process plant	Units	Total	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038
ORE CRUSHED																	
Tonnage	**Mt**	**17.3**	**0.0**	**0.0**	**0.2**	**1.6**	**1.6**	**1.6**	**1.6**	**1.6**	**1.6**	**1.6**	**1.6**	**1.6**	**1.6**	**0.8**	**0.0**
Copper grade	%	1.58	0.00	0.00	1.40	1.56	1.55	1.94	2.13	1.76	1.63	1.84	1.55	1.14	0.98	0.98	0.00
Zinc grade	%	2.31	0.00	0.00	1.23	2.05	2.60	3.13	3.19	2.63	2.59	2.62	2.07	1.65	1.23	1.17	0.00
Silver grade	g/t	27.9	0.0	0.0	30.0	23.5	24.8	35.6	36.8	36.3	31.4	31.0	26.7	21.0	16.5	15.9	0.0
Gold grade	g/t	0.39	0.00	0.00	0.37	0.37	0.34	0.49	0.53	0.64	0.38	0.38	0.30	0.29	0.22	0.21	0.00
Copper	Mlb	602.4	0.0	0.0	6.3	53.4	56.3	70.3	77.1	63.8	59.1	66.7	56.1	41.3	35.7	16.5	0.0
Zinc	Mlb	881.3	0.0	0.0	5.6	70.2	94.1	113.3	115.4	95.3	93.8	94.9	74.9	59.6	44.6	19.7	0.0
Silver	Moz	15.5	0.0	0.0	0.2	1.2	1.3	1.9	1.9	1.9	1.7	1.6	1.4	1.1	0.9	0.4	0.0
Gold	koz	214.4	0.0	0.0	2.5	18.6	17.7	26.1	27.9	33.6	20.2	19.9	16.1	15.1	11.8	5.1	0.0
ORE SORTER REJECT (1)																	
Tonnage	**Mt**	**2.2**	**0.0**	**0.0**	**0.0**	**0.2**	**0.2**	**0.2**	**0.2**	**0.2**	**0.2**	**0.2**	**0.2**	**0.2**	**0.2**	**0.1**	**0.0**
FLOTATION FEED																	
Tonnage	**Mt**	**15.2**	**0.0**	**0.0**	**0.2**	**1.4**	**1.4**	**1.4**	**1.4**	**1.4**	**1.4**	**1.4**	**1.4**	**1.4**	**1.4**	**0.7**	**0.0**
Copper grade	%	1.79	0.00	0.00	1.59	1.76	1.76	2.19	2.41	1.99	1.85	2.08	1.75	1.29	1.11	1.10	0.00
Zinc grade	%	2.62	0.00	0.00	1.40	2.32	2.95	3.55	3.62	2.99	2.94	2.98	2.35	1.87	1.40	1.32	0.00
Silver grade	g/t	31.5	0.0	0.0	33.9	26.6	28.0	40.2	41.5	41.0	35.5	35.0	30.2	23.8	18.6	18.0	0.0
Gold grade	g/t	0.43	0.00	0.00	0.41	0.41	0.37	0.55	0.59	0.71	0.43	0.42	0.34	0.32	0.25	0.23	0.00
Copper	Mlb	596.6	0.0	0.0	6.3	52.9	55.7	69.6	76.3	63.2	58.5	66.0	55.5	40.9	35.3	16.4	0.0
Zinc	Mlb	876.5	0.0	0.0	5.5	69.8	93.6	112.7	114.8	94.7	93.2	94.4	74.5	59.2	44.3	19.6	0.0
Silver	Moz	15.3	0.0	0.0	0.2	1.2	1.3	1.9	1.9	1.9	1.6	1.6	1.4	1.1	0.9	0.4	0.0
Gold	koz	208.7	0.0	0.0	2.4	18.1	17.2	25.4	27.2	32.7	19.7	19.4	15.6	14.7	11.5	4.9	0.0
TAILINGS (2)																	
Tonnage	**Mt**	**13.7**	**0.0**	**0.0**	**0.2**	**1.2**	**1.3**	**1.3**	**1.2**	**1.3**	**1.3**	**1.3**	**1.3**	**1.3**	**1.4**	**0.6**	**0.0**

Table notes:
(1) Ore sorter reject grades average 0.12% copper, 0.10% zinc, 3 g/t silver and 0.08 g/t gold.
(2) Tailings grade average 0.21% copper, 0.12% zinc, 11 g/t silver and 0.18 g/t gold.

Table 22-9: Full discounted cashflow analysis – concentrate produced data

Concentrate produced	Units	Total	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038
COPPER CONCENTRAT PRODUCEDd																	
Tonnage (1)	**kdmt**	**935.9**	**0.0**	**0.0**	**10.0**	**83.5**	**88.0**	**108.0**	**117.7**	**100.0**	**93.6**	**102.9**	**87.6**	**64.7**	**54.6**	**25.3**	**0.0**
Copper grade (2)	%	25.5	0.0	0.0	16.0	24.0	25.3	26.8	27.4	25.6	25.0	26.5	25.4	24.0	24.0	24.0	0.0
Zinc grade	%	10.4	0.0	0.0	11.6	11.5	10.6	7.5	6.7	9.2	9.9	9.0	11.4	15.4	16.9	16.7	0.0
Silver grade	g/t	305.3	0.0	0.0	242.5	229.5	261.2	348.5	334.0	384.3	337.5	304.7	291.9	283.2	239.6	230.4	0.0
Gold grade	g/t	2.6	0.0	0.0	1.8	2.4	2.3	2.9	2.8	4.1	2.5	2.2	2.0	2.6	2.3	2.1	0.0
Copper	Mlb	525.6	0.0	0.0	3.5	44.1	49.2	63.9	71.2	56.5	51.6	60.1	49.0	34.2	28.9	13.4	0.0
Zinc	Mlb	214.4	0.0	0.0	2.6	21.2	20.5	17.8	17.5	20.3	20.5	20.4	21.9	22.0	20.4	9.3	0.0
Silver	Moz	9.2	0.0	0.0	0.1	0.6	0.7	1.2	1.3	1.2	1.0	1.0	0.8	0.6	0.4	0.2	0.0
Gold	koz	79.0	0.0	0.0	0.6	6.5	6.4	10.0	10.7	13.1	7.4	7.4	5.8	5.4	4.1	1.7	0.0
ZINC CONCENTRATE PRODUCED																	
Tonnage (3)	**kdmt**	**515.8**	**0.0**	**0.0**	**2.4**	**38.2**	**56.7**	**72.7**	**74.5**	**58.5**	**57.3**	**57.3**	**41.3**	**29.5**	**19.1**	**8.2**	**0.0**
Copper grade	%	0.6	0.0	0.0	5.2	1.0	0.5	0.4	0.3	0.5	0.5	0.5	0.7	1.0	1.5	1.7	0.0
Zinc grade	%	55.1	0.0	0.0	35.2	52.2	55.7	56.4	56.5	54.6	54.4	55.8	55.0	54.6	54.2	54.1	0.0
Silver grade	g/t	87.2	0.0	0.0	390.8	96.8	63.3	82.4	84.3	106.1	92.1	88.8	98.1	81.0	55.0	52.3	0.0
Gold grade	g/t	3.1	0.0	0.0	6.1	3.5	2.3	2.7	2.8	4.3	2.6	2.6	2.8	3.7	4.4	4.4	0.0
Copper	Mlb	7.1	0.0	0.0	0.3	0.9	0.7	0.6	0.5	0.7	0.7	0.6	0.6	0.7	0.6	0.3	0.0
Zinc	Mlb	626.7	0.0	0.0	1.9	44.0	69.7	90.5	92.8	70.4	68.7	70.5	50.1	35.5	22.8	9.8	0.0
Silver	Moz	1.4	0.0	0.0	0.0	0.1	0.1	0.2	0.2	0.2	0.2	0.2	0.1	0.1	0.0	0.0	0.0
Gold	koz	50.7	0.0	0.0	0.5	4.3	4.2	6.3	6.7	8.1	4.8	4.7	3.8	3.5	2.7	1.2	0.0
TOTAL METAL PRODUCED																	
Copper	Mlb	532.6	0.0	0.0	3.8	45.0	49.8	64.4	71.7	57.2	52.2	60.7	49.7	34.9	29.5	13.7	0.0
Zinc	Mlb	841.1	0.0	0.0	4.4	65.2	90.2	108.3	110.3	90.7	89.2	90.9	72.0	57.5	43.2	19.1	0.0
Silver	Moz	10.6	0.0	0.0	0.1	0.7	0.9	1.4	1.5	1.4	1.2	1.2	1.0	0.7	0.5	0.2	0.0
Gold	koz	129.7	0.0	0.0	1.1	10.8	10.6	16.2	17.5	21.2	12.2	12.1	9.5	8.9	6.8	2.9	0.0

Table notes:

(1) Copper concentrate mass pull 6.2%.

(2) Includes 9% moisture.

(3) Zinc concentrate mass pull 3.6%.

Table 22-10: Full discounted cashflow analysis – process plant recoveries

Recovery	Units	Total	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038
Copper Circuit Recovery to Flotation Head																	
Copper	%	88.1%	0.0%	0.0%	56.4%	83.4%	88.3%	91.8%	93.2%	89.5%	88.1%	91.0%	88.3%	83.7%	81.8%	81.7%	0.0%
Zinc	%	24.5%	0.0%	0.0%	46.0%	30.3%	21.9%	15.8%	15.2%	21.5%	22.0%	21.6%	29.4%	37.1%	45.9%	47.5%	0.0%
Silver	%	59.9%	0.0%	0.0%	39.9%	52.9%	57.1%	65.1%	65.8%	65.2%	61.9%	62.2%	58.9%	53.6%	48.9%	48.2%	0.0%
Gold	%	37.9%	0.0%	0.0%	24.6%	35.8%	37.3%	39.3%	39.6%	40.0%	37.6%	38.0%	36.9%	36.6%	35.5%	35.2%	0.0%
Zinc Circuit Recovery to Flotation Head																	
Copper	%	1.2%	0.0%	0.0%	4.4%	1.7%	1.2%	0.8%	0.7%	1.0%	1.2%	0.9%	1.2%	1.6%	1.8%	1.8%	0.0%
Zinc	%	71.5%	0.0%	0.0%	33.4%	63.0%	74.4%	80.3%	80.9%	74.3%	73.7%	74.8%	67.2%	60.0%	51.5%	50.0%	0.0%
Silver	%	9.4%	0.0%	0.0%	15.4%	10.2%	8.9%	10.4%	10.5%	10.5%	10.3%	10.1%	9.3%	7.0%	3.9%	3.6%	0.0%
Gold	%	24.3%	0.0%	0.0%	19.7%	23.8%	24.2%	24.6%	24.7%	24.8%	24.3%	24.4%	24.1%	23.9%	23.6%	23.5%	0.0%

Table 22-11: Full discounted cashflow analysis – concentrate revenues

	Units	Total	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038
COPPER CONCENTRATE REVENUE																	
Dry tonnage (1)	kdmt	**933.6**	0.0	0.0	2.5	83.3	87.8	107.7	117.4	99.8	93.4	102.6	87.4	64.5	54.5	32.7	0.0
Wet tonnage (2)	kwmt	**1,025.9**	0.0	0.0	2.7	91.5	96.5	118.4	129.0	109.7	102.6	112.8	96.0	70.9	59.9	35.9	0.0
Copper grade	%	**25.5**	0.0	0.0	16.1	23.1	25.3	26.7	27.4	25.9	24.9	26.5	25.6	24.0	24.0	24.0	0.0
Zinc grade	%	**10.4**	0.0	0.0	11.6	11.5	10.7	7.7	6.8	9.0	9.9	9.0	11.0	15.2	16.8	16.7	0.0
Silver grade	g/t	**305.3**	0.0	0.0	243.1	229.6	259.2	343.2	337.1	378.8	341.5	307.8	291.0	282.6	247.8	230.4	0.0
Gold grade	g/t	**2.6**	0.0	0.0	1.8	2.4	2.2	2.9	2.8	4.1	2.5	2.2	2.0	2.5	2.4	2.1	0.0
Payable copper (3)	Mlb	**503.6**	0.0	0.0	0.8	40.6	47.1	61.1	68.4	54.7	49.1	57.6	47.3	32.7	27.6	16.6	0.0
Payable zinc	Mlb	**0.0**	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Payable silver	Moz	**8.2**	0.0	0.0	0.0	0.6	0.7	1.1	1.1	1.1	0.9	0.9	0.7	0.5	0.4	0.2	0.0
Payable gold	koz	**74.1**	0.0	0.0	0.1	6.0	6.0	9.3	10.0	12.3	7.2	6.9	5.4	4.9	3.9	2.1	0.0
Revenue copper	C$M	**2,319.3**	0.0	0.0	3.8	187.1	216.7	281.2	314.9	252.0	226.1	265.3	218.0	150.7	127.2	76.3	0.0
Revenue zinc	C$M	**0.0**	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Revenue silver	C$M	**217.0**	0.0	0.0	0.5	14.6	17.3	28.2	30.1	28.8	24.3	24.1	19.4	13.9	10.3	5.7	0.0
Revenue gold	C$M	**156.0**	0.0	0.0	0.3	12.6	12.6	19.6	21.0	25.9	15.1	14.5	11.4	10.3	8.3	4.4	0.0
Revenue total	**C$M**	**2,692.3**	**0.0**	**0.0**	**4.5**	**214.2**	**246.6**	**328.9**	**366.1**	**306.7**	**265.5**	**303.9**	**248.8**	**174.9**	**145.8**	**86.5**	**0.0**
ZINC CONCENTRATE REVENUE																	
Dry tonnage (1)	kdmt	**514.5**	0.0	0.0	0.0	33.0	56.6	72.6	74.3	58.3	57.2	57.2	41.9	28.7	22.2	12.6	0.0
Wet tonnage (2)	kwmt	**565.4**	0.0	0.0	0.0	36.3	62.2	79.7	81.6	64.1	62.8	62.8	46.1	31.5	24.4	13.8	0.0
Copper grade	%	**0.6**	0.0	0.0	0.0	1.4	0.6	0.4	0.3	0.5	0.5	0.5	0.6	1.1	1.3	1.7	0.0
Zinc grade	%	**55.1**	0.0	0.0	0.0	50.5	55.5	56.4	56.5	55.2	54.0	55.8	55.2	54.6	54.3	54.1	0.0
Silver grade	g/t	**87.2**	0.0	0.0	0.0	121.9	66.0	79.6	86.1	101.6	93.6	89.5	95.4	87.2	63.0	52.3	0.0
Gold grade	g/t	**3.1**	0.0	0.0	0.0	3.8	2.4	2.6	2.8	4.3	2.7	2.5	2.7	3.7	4.1	4.4	0.0
Payable copper	Mlb	**0.0**	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Payable zinc (4)	Mlb	**531.0**	0.0	0.0	0.0	30.9	58.9	76.6	78.6	60.3	57.8	59.8	43.4	29.3	22.6	12.7	0.0
Payable silver	Moz	**0.1**	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Payable gold	koz	**32.9**	0.0	0.0	0.0	2.8	2.5	3.8	4.3	5.6	3.1	2.8	2.3	2.4	2.0	1.2	0.0
Revenue copper	C$M	**0.0**	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Revenue zinc	C$M	**803.5**	0.0	0.0	0.0	46.8	89.1	116.0	119.0	91.3	87.4	90.5	65.6	44.4	34.2	19.3	0.0
Revenue silver	C$M	**1.6**	0.0	0.0	0.0	0.8	0.0	0.0	0.0	0.5	0.1	0.0	0.1	0.0	0.0	0.0	0.0
Revenue gold	C$M	**69.3**	0.0	0.0	0.0	5.9	5.3	8.0	9.1	11.8	6.6	6.0	4.8	5.0	4.3	2.6	0.0
Revenue total	**C$M**	**874.5**	**0.0**	**0.0**	**0.0**	**53.5**	**94.4**	**124.0**	**128.0**	**103.6**	**94.1**	**96.5**	**70.5**	**49.4**	**38.5**	**21.9**	**0.0**
COMBINED REVENUE	**C$M**	**3,566.8**	**0.0**	**0.0**	**4.5**	**267.7**	**341.0**	**452.9**	**494.1**	**410.3**	**359.6**	**400.4**	**319.3**	**224.3**	**184.3**	**108.4**	**0.0**

Table notes: (1) Includes 0.25% transportation loss. (2) 9% moisture. (3) Copper concentrate payability – copper 96.1%, zinc 0%, silver 90%, gold 94%. (4) Zinc concentrate payability – copper 0%, zinc 84.9%, silver 4%, gold 65%.

Table 22-12: Full discounted cashflow analysis – concentrate treatment and refining charges

Concentrate charges	Units	Total	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038
COPPER CONCENTRATE TREATMENT AND REFINING CHARGES																	
Treatment Charge	C$M	98.3	0.0	0.0	0.3	8.8	9.2	11.3	12.4	10.5	9.8	10.8	9.2	6.8	5.7	3.4	0.0
Refining Charge - Copper	C$M	40.3	0.0	0.0	0.1	3.2	3.8	4.9	5.5	4.4	3.9	4.6	3.8	2.6	2.2	1.3	0.0
Refining Charge - Silver	C$M	5.4	0.0	0.0	0.0	0.4	0.4	0.7	0.8	0.7	0.6	0.6	0.5	0.3	0.3	0.1	0.0
Refining Charge - Gold	C$M	0.5	0.0	0.0	0.0	0.0	0.0	0.1	0.1	0.1	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Penalties	C$M	15.5	0.0	0.0	0.0	1.6	1.5	1.2	1.1	1.4	1.4	1.4	1.5	1.7	1.6	1.0	0.0
Transport to Port	C$M	98.2	0.0	0.0	0.3	8.8	9.2	11.3	12.4	10.5	9.8	10.8	9.2	6.8	5.7	3.4	0.0
Port charges	C$M	18.4	0.0	0.0	0.0	1.6	1.7	2.1	2.3	2.0	1.8	2.0	1.7	1.3	1.1	0.6	0.0
Insurance	C$M	6.1	0.0	0.0	0.0	0.5	0.6	0.7	0.8	0.7	0.6	0.7	0.6	0.4	0.4	0.2	0.0
Ocean freight	C$M	49.1	0.0	0.0	0.1	4.4	4.6	5.7	6.2	5.3	4.9	5.4	4.6	3.4	2.9	1.7	0.0
Total Charges	**C$M**	**331.8**	**0.0**	**0.0**	**0.8**	**29.3**	**31.2**	**38.0**	**41.4**	**35.4**	**33.0**	**36.4**	**31.1**	**23.4**	**19.9**	**11.9**	**0.0**
ZINC CONCENTRATE TREATMENT AND REFINING CHARGES																	
Treatment Charge	C$M	135.4	0.0	0.0	0.0	8.7	14.9	19.1	19.6	15.3	15.0	15.0	11.0	7.5	5.8	3.3	0.0
Refining Charge - Silver	C$M	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Refining Charge - Gold	C$M	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Penalties	C$M	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Transport to Port	C$M	54.1	0.0	0.0	0.0	3.5	5.9	7.6	7.8	6.1	6.0	6.0	4.4	3.0	2.3	1.3	0.0
Port charges	C$M	10.2	0.0	0.0	0.0	0.7	1.1	1.4	1.5	1.2	1.1	1.1	0.8	0.6	0.4	0.2	0.0
Insurance	C$M	3.4	0.0	0.0	0.0	0.2	0.4	0.5	0.5	0.4	0.4	0.4	0.3	0.2	0.1	0.1	0.0
Ocean freight	C$M	27.1	0.0	0.0	0.0	1.7	3.0	3.8	3.9	3.1	3.0	3.0	2.2	1.5	1.2	0.7	0.0
Total Charges	**C$M**	**230.1**	**0.0**	**0.0**	**0.0**	**14.8**	**25.3**	**32.5**	**33.2**	**26.1**	**25.6**	**25.6**	**18.8**	**12.8**	**9.9**	**5.6**	**0.0**
COMBINED CHARGES	**C$M**	**562.0**	**0.0**	**0.0**	**0.8**	**44.1**	**56.5**	**70.5**	**74.6**	**61.5**	**58.6**	**62.0**	**49.9**	**36.2**	**29.8**	**17.5**	**0.0**

Table 22-13: Full discounted cashflow analysis – revenue net of charges and royalty

Concentrate NSR and net revenue	**Units**	**Total**	**2024**	**2025**	**2026**	**2027**	**2028**	**2029**	**2030**	**2031**	**2032**	**2033**	**2034**	**2035**	**2036**	**2037**	**2038**
COPPER CONCENTRATE																	
Revenue	C$M	2,692.3	0.0	0.0	4.5	214.2	246.6	328.9	366.1	306.7	265.5	303.9	248.8	174.9	145.8	86.5	0.0
Charges	C$M	331.8	0.0	0.0	0.8	29.3	31.2	38.0	41.4	35.4	33.0	36.4	31.1	23.4	19.9	11.9	0.0
NSR	**C$M**	**2,360.5**	**0.0**	**0.0**	**3.7**	**184.9**	**215.4**	**290.9**	**324.7**	**271.2**	**232.4**	**267.5**	**217.6**	**151.5**	**125.9**	**74.6**	**0.0**
ZINC CONCENTRATE																	
Revenue	C$M	874.5	0.0	0.0	0.0	53.5	94.4	124.0	128.0	103.6	94.1	96.5	70.5	49.4	38.5	21.9	0.0
Charges	C$M	230.1	0.0	0.0	0.0	14.8	25.3	32.5	33.2	26.1	25.6	25.6	18.8	12.8	9.9	5.6	0.0
NSR	**C$M**	**644.3**	**0.0**	**0.0**	**0.0**	**38.8**	**69.1**	**91.5**	**94.8**	**77.5**	**68.5**	**70.9**	**51.8**	**36.6**	**28.6**	**16.3**	**0.0**
TOTAL CONCENTRATE																	
Revenue	C$M	3,566.8	0.0	0.0	4.5	267.7	341.0	452.9	494.1	410.3	359.6	400.4	319.3	224.3	184.3	108.4	0.0
Charges	C$M	562.0	0.0	0.0	0.8	44.1	56.5	70.5	74.6	61.5	58.6	62.0	49.9	36.2	29.8	17.5	0.0
NSR	**C$M**	**3,004.8**	**0.0**	**0.0**	**3.7**	**223.6**	**284.5**	**382.4**	**419.5**	**348.7**	**301.0**	**338.5**	**269.4**	**188.1**	**154.5**	**90.9**	**0.0**
NSR per tonne ore crushed	C$/t	173.6	0.0	0.0	17.9	143.7	173.2	232.8	255.4	212.3	183.3	206.1	164.0	114.5	94.1	118.4	0.0
ROYALTIES																	
Royal Gold	C$M	26.1	0.0	0.0	0.0	2.4	3.4	5.0	3.6	2.8	2.4	1.6	1.3	2.1	1.1	0.4	0.0
Total Royalties	**C$M**	**26.1**	**0.0**	**0.0**	**0.0**	**2.4**	**3.4**	**5.0**	**3.6**	**2.8**	**2.4**	**1.6**	**1.3**	**2.1**	**1.1**	**0.4**	**0.0**
NET REVENUE																	
Net Revenue	C$M	2,978.7	0.0	0.0	3.6	221.2	281.1	377.4	415.9	345.9	298.6	336.9	268.1	186.1	153.4	90.4	0.0
Net Revenue (1)	**C$/t**	**172.1**	**0.0**	**0.0**	**17.8**	**142.1**	**171.1**	**229.7**	**253.2**	**210.6**	**181.8**	**205.1**	**163.2**	**113.3**	**93.4**	**117.8**	**0.0**

Table notes: (1) Net revenue per tonne crushed.

Table 22-14: Full discounted cashflow analysis – operating costs showing redirection to capital

Operating costs	Units	Total	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038
Open pit mining	C$M	407.0	0.0	12.6	46.5	45.6	46.5	42.1	32.0	38.2	38.2	36.6	21.2	22.8	19.2	5.5	0.0
Underground mining	C$M	163.0	0.0	1.2	12.6	21.4	23.5	21.2	18.0	18.5	16.9	9.8	12.8	7.0	0.0	0.0	0.0
Process	C$M	492.8	0.0	0.0	8.2	45.6	47.5	47.6	47.6	47.6	47.0	45.1	45.1	45.1	45.1	21.4	0.0
General and administration	C$M	186.9	0.0	15.0	20.9	16.3	16.4	15.8	15.0	15.7	15.6	15.3	13.9	12.0	10.6	4.4	0.0
Working capital	C$M	-5.6	0.0	0.0	0.0	-5.6	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Pre-production capital (8)	C$M	-117.1	0.0	-28.8	-88.3	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Total Operating Costs (7)	**C$M**	**1,127.2**	**0.0**	**0.0**	**0.0**	**123.3**	**133.9**	**126.7**	**112.6**	**120.0**	**117.8**	**106.7**	**93.0**	**87.0**	**74.9**	**31.3**	**0.0**
Open pit mining	C$/t[1,6]	3.07	0.00	22.73	4.07	3.33	3.41	3.66	3.95	2.96	2.98	3.61	2.84	1.58	1.56	1.58	0.00
Open pit mining	C$/t[2,6]	4.23	0.00	22.73	4.07	3.33	3.41	3.66	3.95	3.00	3.03	3.61	11.49	0.00	0.00	0.00	0.00
Open pit mining	C$/t[3,6]	28.10	0.00	13,109.74	92.50	28.93	26.90	17.97	15.09	26.15	30.50	14.49	21.79	0.00	0.00	0.00	0.00
Underground mining	C$/t[4,6]	57.66	0.00	344.55	86.42	62.17	66.06	57.97	52.46	52.15	50.16	59.12	48.84	47.58	0.00	0.00	0.00
Process	C$/t[5,6]	28.47	0.00	0.00	40.21	29.32	28.94	28.95	28.97	28.99	28.62	27.43	27.43	27.43	27.44	27.92	0.00
General and administration	C$/t[5,6]	10.80	0.00	0.00	102.14	10.46	9.96	9.62	9.11	9.55	9.48	9.30	8.48	7.33	6.47	5.79	0.00
Total (7)	**C$/t[5,6]**	**65.89**	**0.00**	**0.00**	**0.00**	**79.24**	**81.51**	**77.12**	**68.54**	**73.08**	**71.69**	**64.98**	**56.63**	**52.96**	**45.60**	**40.81**	**0.00**

Table notes:
(1) Cost per tonne open pit material moved.
(2) Cost per tonne open pit material mined.
(3) Cost per tonne open pit ore mined.
(4) Cost per tonne underground ore mined.
(5) Cost per tonne crushed.
(6) Includes pre-production.
(7) After redirection of operating costs to capital for pre-production period.
(8) Operating costs re-directed to capital.

Table 22-15: Full discounted cashflow analysis – operating costs excluding redirections to capital and excluding pre-production tonnages

	Units	Total	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038
OPERATING COSTS																	
Open pit mining	C$M	346.0				43.6	46.5	42.1	32.0	38.2	38.2	36.6	21.2	22.8	19.2	5.5	0.0
Underground mining	C$M	148.3				20.4	23.5	21.2	18.0	18.5	16.9	9.8	12.8	7.0	0.0	0.0	0.0
Process	C$M	482.6				43.7	47.5	47.6	47.6	47.6	47.0	45.1	45.1	45.1	45.1	21.4	0.0
General and administration	C$M	150.3				15.6	16.4	15.8	15.0	15.7	15.6	15.3	13.9	12.0	10.6	4.4	0.0
Total Operating Costs	**C$M**	**1,127.2**				**123.3**	**133.9**	**126.7**	**112.6**	**120.0**	**117.8**	**106.7**	**93.0**	**87.0**	**74.9**	**31.3**	**0.0**
Open pit mining	C$/t[1]	2.87				3.18	3.41	3.66	3.95	2.96	2.98	3.61	2.84	1.58	1.56	1.58	0.00
Open pit mining	C$/t[2]	4.11				3.18	3.41	3.66	3.95	3.00	3.03	3.61	11.49	0.00	0.00	0.00	0.00
Open pit mining	C$/t[3]	24.75				27.68	26.90	17.97	15.09	26.15	30.50	14.49	21.79	0.00	0.00	0.00	0.00
Underground mining	C$/t[4]	55.38				62.17	66.06	57.97	52.46	52.15	50.16	59.12	48.84	47.58	0.00	0.00	0.00
Process	C$/t[5]	28.21				28.06	28.94	28.95	28.97	28.99	28.62	27.43	27.43	27.43	27.44	27.92	0.00
General and administration	C$/t[5]	8.79				10.01	9.96	9.62	9.11	9.55	9.48	9.30	8.48	7.33	6.47	5.79	0.00
Total	**C$/t[5]**	**65.89**				**79.24**	**81.51**	**77.12**	**68.54**	**73.08**	**71.69**	**64.98**	**56.63**	**52.96**	**45.60**	**40.81**	**0.00**
NET OPERATING INCOME																	
Net Revenue	C$M	2,978.7	0.0	0.0	3.6	221.2	281.1	377.4	415.9	345.9	298.6	336.9	268.1	186.1	153.4	90.4	0.0
Total Operating Costs	C$M	1,127.2	0.0	0.0	0.0	123.3	133.9	126.7	112.6	120.0	117.8	106.7	93.0	87.0	74.9	31.3	0.0
Net Operating Income	C$M	1,851.5	0.0	0.0	3.6	97.9	147.2	250.7	303.3	225.9	180.8	230.2	175.1	99.1	78.5	59.1	0.0
Net Operating Income	C$/t[5]	106.95	0.00	0.00	17.80	62.90	89.61	152.63	184.68	137.54	110.10	140.13	106.59	60.32	47.81	77.01	0.00

Table notes:

(1) Cost per tonne open pit material moved.
(2) Cost per tonne open pit material mined.
(3) Cost per tonne open pit ore mined.
(4) Cost per tonne underground ore mined.
(5) Cost per tonne crushed.

Table 22-16: Full discounted cashflow analysis – capital

Capital	Units	Total	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038
UG Mine Equipment and Infrastructure	C$M	31.9	0.0	6.9	11.0	3.3	6.2	1.4	0.5	1.6	0.2	0.2	0.2	0.3	0.0	0.0	0.0
UG Capital Development	C$M	33.9	0.0	2.0	13.8	10.0	8.2	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Pre-Production	C$M	117.1	0.0	28.8	88.3	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Working Capital	C$M	5.6	0.0	0.0	0.0	5.6	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Access Road	C$M	31.7	30.9	0.8	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Mining Ancillary	C$M	1.1	0.2	0.4	0.5	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Site Development	C$M	17.3	0.2	16.6	0.2	0.1	0.0	0.0	0.1	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Mineral Processing	C$M	106.0	6.4	43.5	56.2	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Tailings Management Facility	C$M	9.9	0.0	6.5	3.5	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Site Infrastructure	C$M	44.3	2.7	18.2	23.5	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Indirects	C$M	51.3	0.0	23.1	28.2	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
EPCM	C$M	24.0	3.6	10.8	9.6	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
TMF sustaining	C$M	15.8	0.0	0.0	0.0	2.0	7.9	0.6	2.4	0.0	0.0	2.9	0.0	0.0	0.0	0.0	0.0
Plant and Mine Sustaining Capital	C$M	27.4	0.0	0.0	0.0	1.6	3.1	3.3	3.3	3.3	3.3	3.4	3.1	2.0	0.8	0.2	0.0
Subtotal	**C$M**	**517.5**	**43.9**	**157.5**	**234.8**	**22.5**	**25.4**	**5.4**	**6.4**	**5.0**	**3.6**	**6.6**	**3.3**	**2.3**	**0.8**	**0.2**	**0.0**
Contingency (1)	C$M	55.0	4.6	17.4	24.8	1.9	2.7	0.6	0.7	0.5	0.4	0.7	0.4	0.2	0.1	0.0	0.0
Salvage	C$M	-18.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	-18.0
Site Rehabilitation (2)	C$M	34.5	0.0	0.0	0.0	0.0	0.0	0.4	0.0	0.0	0.0	0.0	1.8	1.8	0.7	5.5	24.3
Bond (3)	C$M	0.0	4.3	2.9	2.7	1.4	1.0	1.0	1.1	1.1	0.7	0.3	0.3	0.3	0.1	2.0	-19.3
TOTAL CAPITAL	**C$M**	**588.9**	**52.8**	**177.8**	**262.3**	**25.8**	**29.1**	**7.3**	**8.2**	**6.6**	**4.8**	**7.6**	**5.7**	**4.6**	**1.6**	**7.7**	**-13.0**

Table notes:
(1) Average 11% contingency.
(2) 15% contingency included, all costs in last year are inclusive to end of closure.
(3) Bond in last year is net of bond reclamation.

Table 22-17: Full discounted cashflow analysis – determination of NPV and IRR

Discounted Cashflow (DCF)	Units	Total	Pre-production	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038
Pre-Tax															
Production year indicator	years	10.50		1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	0.50	0.00
Net Pre-Tax Cash Flow	C$M	1,263.7	(489.2)	72.1	118.1	243.4	295.2	219.3	176.1	222.6	169.3	94.5	76.9	51.4	14.2
Cumulative Pre-Tax CF	C$M	1,263.7	(489.2)	(417.1)	(299.0)	(55.7)	239.5	458.8	634.8	857.4	1,026.8	1,121.3	1,198.2	1,249.6	1,263.7
Pre-Tax Payback	years	3.2	-	1.0	1.0	1.0	0.2	-	-	-	-	-	-	-	-
Pre-Tax NPV	C$M	737.2													
Pre-Tax IRR	%	31.3%													
Average Annual Pre-Tax Cash Flow	C$M pa	163.2													
After-Tax															
Taxes	C$M	422.5	0.5	2.4	3.4	18.1	72.1	60.2	57.0	76.8	58.2	28.7	25.0	18.4	1.7
Net After-Tax Cash Flow	C$M	841.2	(489.7)	69.7	114.7	225.3	223.1	159.1	119.1	145.8	111.1	65.8	51.9	33.0	12.5
Cumulative After-Tax CF	C$M	841.2	(489.7)	(420.0)	(305.3)	(80.0)	143.0	302.1	421.1	566.9	678.0	743.8	795.8	828.8	841.2
After-Tax Payback	years	3.4	-	1.0	1.0	1.0	0.4	-	-	-	-	-	-	-	-
After-Tax NPV	C$M	461.3													
After-Tax IRR	%	24.8%													
Average Annual Post Tax Cash Flow	C$M pa	124.0													

Table notes: 10.5 production years. 7% discount rate.

22.4 NPV, IRR and Payback

Key financial outcomes are presented in Table 22-18.

Table 22-18: Summary of economic results

Parameter	Unit	Pre-tax results	After-tax results
NPV - no discount	C$M	1,264	841
NPV - 7% discount [(1)]	C$M	737	461
Annual average net operating income [(2)]	C$M	158	120
IRR	%	31.3%	24.8%
Payback period	production years	3.2	3.4

(1) *Method of discounting for NPV – all capital years are undiscounted; discounting commences in year 1. The after tax NPV for commencing discounting in Year -3 is C$360 million, the NPV for commencing discounting in Year -2 is C$388 million and the NPV for commencing discounting in year -1 is C$431 million.*

(2) *Covers production years 1-11 only, pre-production and year 12 are excluded.*

The Project after tax undiscounted annual cashflows, cumulative undiscounted and cumulative discounted cashflows are shown in Figure 22-3.

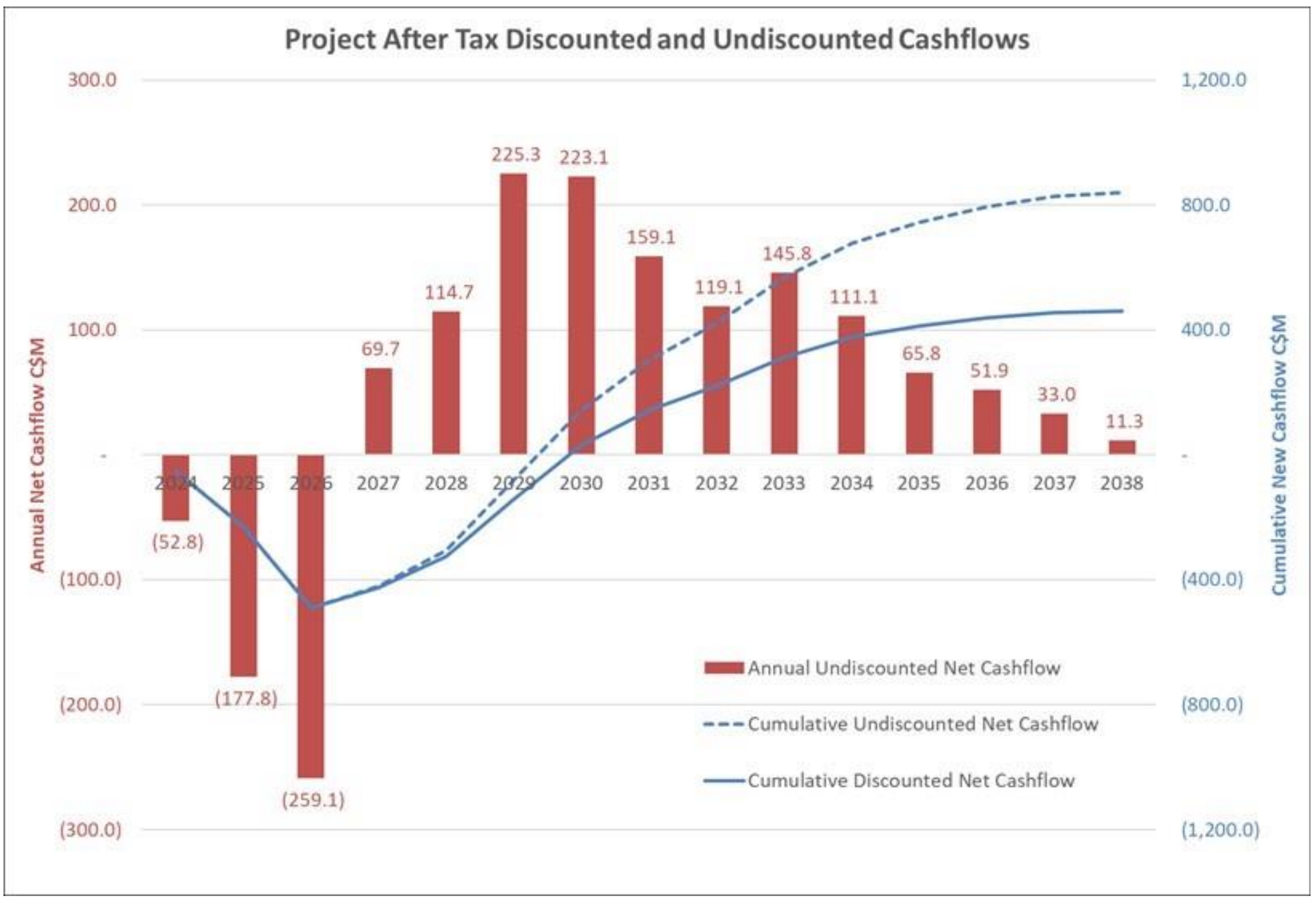


Figure 22-3: Chart Of Project After-Tax Undiscounted Annual Cash Flows, Cumulative Undiscounted and Cumulative Discounted Cash Flows

22.5 Sensitivity

The sensitivity of the Project to a number of parameters were estimated. Neither the cut-off grade, mine plan nor the processing plan were altered.

The Project NPV is most sensitive to changes in metal price and factors such as exchange rate, head grade, and metallurgical recovery. Of the four metals, the project is most sensitive to variations associated to copper. After metal price, IRR is equally sensitive to changes in operating and capital expenditure. NPV on the other hand is more sensitive to operating expenditure than capital costs. A number of standard financial sensitivities are listed in Table 22-19 and represented as graphs in Figure 22-4 for NPV and Figure 22-4 for IRR. More detail on sensitivity to metal price Is shown as Figure 22-6.

The project NPV is robust to changes in capital and operating expense. The IRR is estimated to be more sensitive to capital expense. The NPV and IRR sensitivity of the project is estimated to be four times greater to metal price than capital or operating costs. Further details on metal price and metal costs are provided in Section 19.

Table 22-19: Key sensitivity analyses for the Project

Item	Sensitivities	After-tax IRR (%)	After-tax NPV (C$M)
	-25%	32.0	511
CAPEX	**Base Case**	**24.8**	**461**
	+25%	19.4	410
	-25% (5.3%)	24.8	540
Discount rate	**Base Case (7.0%)**	**24.8**	**461**
	+25% (8.8%)	24.8	391
	-25%	31.8	615
OPEX	**Base Case**	**24.8**	**461**
	+25%	18.3	306
	-25%	8.8	43
Revenue	**Base Case**	**24.8**	**461**
	+25%	38.6	869

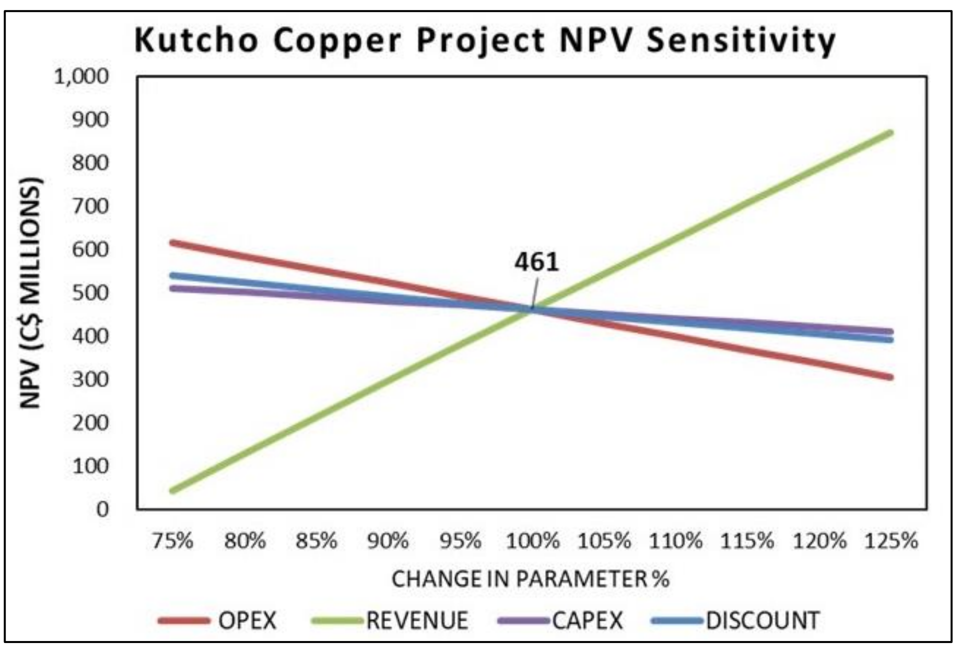


Figure 22-4: Project Sensitivity Analysis NPV

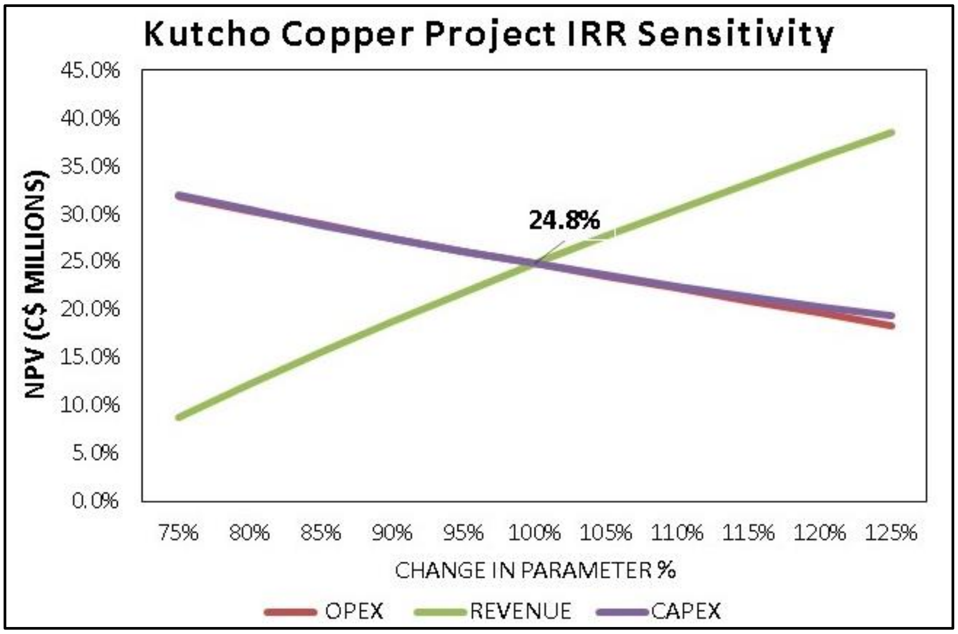


Figure 22-5: Project Sensitivity Analysis IRR%

Note: Discount rate not shown as this is independent of IRR.

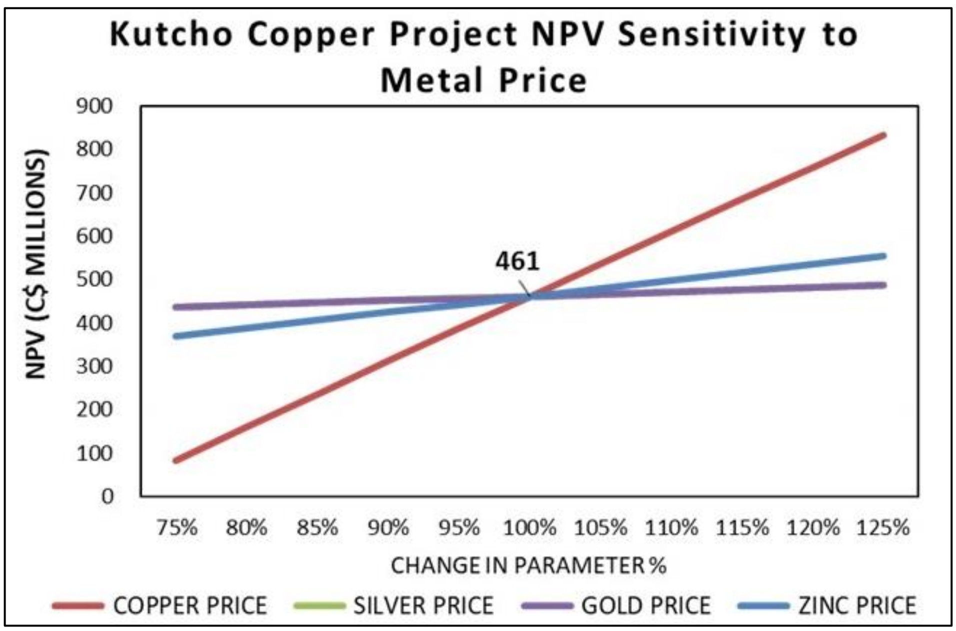


Figure 22-6: Project Sensitivity Analysis of Various Metal Prices

Note: Silver and gold are similar values and so do not show separately.

23 Adjacent Properties

There are no significant or relevant mineral resource properties immediately adjacent to the Project.

24 Other Relevant Data and Information

It is considered that there is no additional information or explanation necessary to make this Technical Report understandable and not misleading.

25 Interpretations and Conclusions

25.1 Overall Consideration

The Kutcho Project is located in northern British Columbia, Canada, approximately 100 km east of Dease Lake within the traditional territories of the Tahltan Nation and Kaska Dena Nation. The Project contemplates mining the Main and Esso deposits to extract mainly copper and zinc, with minor quantities of gold and silver. The Main deposit is designed to be mined primarily as a conventional shovel and truck open pit operation, with a deeper remnant mined by underground longitudinal longhole open stoping (LLHOS) with cemented rock fill (CRF). The underground Esso deposit is also designed to be mined using LLHOS with CRF. A total of 17.3 Mt is planned to be mined over an 11-year mine life, with 14.5 Mt coming from the open pit and 2.8 Mt from the underground mines. A steady-state processing rate of 4,500 tpd of ore crushed is expected be achieved by the end of the first year of operations.

The property has had multiple owners through its exploration history. Drilling first started in 1974 and has continued to the present day. An updated Mineral Resource estimate was prepared for the Project with drilling and assay information incorporated to 9 November 2018. Various geotechnical drilling and surface pitting campaigns have been completed at the site; the most recent drilling being completed in 2021. An ordinary kriging estimation technique was used to model the poly-metallic mineralization hosted in the Main, Sumac and Esso deposits. The Main open pit resources are constrained by an optimized pit shell. Cut-off grades appropriate for the economic input parameters and cost parameters representative of underground mining methods recommended in this report are applied beneath the Main optimized pit shell. The Mineral Resource Estimate has an effective date of 30 July 2021 and are presented in Table 14-14.

A PFS, completed by Wardrop for Western Keltic Inc. in 2007, assessed the project as an open pit mine for the Main deposit and an underground mine for the Esso deposit with a 6,000 tpd processing plant. In 2010 the Project's new owners, Capstone Mining Corp., completed a PEA for a 2,500 tpd mining operation comprising predominantly underground mining of the Main and Esso deposits, with a much smaller Main open pit. This was advanced to a PFS by JDS Engineering in 2011. This PFS was refreshed in 2017 by JDS Engineering under the direction of the then new owners, Desert Star Resources Ltd.

In 2021, CSA Global was engaged by Kutcho Copper to complete a Feasibility Study (the subject of this Technical Report). In a preliminary set of studies, CSA Global concurrently examined a predominantly open pit mining scenario against a 100% underground mine. Underground mining methods, stope heights and backfill methods were evaluated. The studies also examined new locations and methods of construction for the tailings. It was concluded during the process that the predominantly open pit mine delivered improved project value compared to an underground only mine, with no encroachment on fish-bearing streams (which was a limitation of the 2007 study). Further improvements were also applied during the study, including minimization of the open pit site footprint through in-pit waste rock dumping, final storage of all PAG waste into the pit, and a rehabilitated pit topography that maximized surface water runoff and leaves no standing body of water in the former pit area. The addition of an XRT particle sorter enabled smaller milling and flotation circuits, reduced tailings storage requirements and created of a supply of appropriately sized rock for underground backfill. Underground mine production is mainly from the Esso deposit, but some early underground production is provided from the Main deposit.

The Mineral Reserve Estimate is presented in Table 15-16.

The conversion of Mineral Resources to Mineral Reserves was made using industry-standard methods of determining the mining methodology, applicable recovery and loss estimates, and the application of operating costs, capital costs, mining rate, and plant performance. The evaluation considered orebody geometry and continuity, hydrogeological/hydrological conditions, geotechnical conditions, and environmental considerations, amongst others. The evaluation methods applied are considered to represent envisaged operational conditions of the proposed mining project. This report was prepared with the latest information regarding environmental and closure requirements and has set out the type and extent of work required.

The recommended development plan for the Project commences with the construction of a 119 km road from Dease Lake to the Project site approximately two years prior to the commencement of on-site construction activity. Construction, including commissioning, is expected to take two years. On-site development would include an open pit (strip ratio 5.6:1) with ore mined at an average rate of 5,200 tpd and an 800 tpd underground mine. The combined mine production rate would allow for low-grade ore stockpiling and processing at the end of mine life to feed the crusher at 4,500 tpd.

A XRT ore sorter rejects about 600 tpd of low-grade ore (about 13% of feed) and the remaining 3,900 tpd passes to a grinding circuit and two-stage flotation plant. The flotation plant produces copper and zinc concentrates that are planned to be transported by truck to the existing port facilities at Stewart, Alaska, a distance of 520 km. The concentrates are expected to be sold to and processed in smelters in the eastern Asian region. Thickened tailings from the plant are to be stored in a fully lined, downstream-constructed tailings management facility. Ore sorter reject materials will mostly be utilized as underground stope backfill.

The open pit and underground mines are planned to operate concurrently for nine and ten years respectively, including an initial two years of construction and pre-stripping. Processing of the low-grade stockpile follows for a period of three years. The plant operates for almost 11 years inclusive of the start-up period. The mine is planned to utilize primary production equipment under an owner-operator arrangement with all mobile equipment being leased. The primary production equipment selected are commonly available and were chosen for their ability to selectively extract ore at the dimensions of the modelled SMUs. Development of the Project includes training, ore control systems, costs, manning and management that are considered adequate to provide the necessary control to ore delivery to the primary crusher at the specified grade and at a rate adequate to feed the plant at the design throughput rate. The open pit mine is planned to have a peak combined ore plus waste mining rate of 37,600 tpd, delivering 3,400–6,900 tpd of ore to direct crusher feed and low-grade stockpiles. The waste not used for the construction of the TMF embankment will report to the waste dumps. Some low-grade ore designated as waste in the mine plan will be segregated for possible future processing should economics allow.

Oxidized and low-grade ore are planned to be stockpiled and then processed mostly during the last three years of the operation after open pit mining is completed. Most mine pre-stripping activities are aimed at the production of waste rock for the site infrastructure, and in particular the TMF Phase 1 embankment. The mine plan provides for back-filling of the open pit with waste rock to minimize the project footprint, reduce environmental impacts, and to reduce operating costs.

The flotation process is considered to be conventional for the separation of copper and zinc minerals from a high-pyrite content ore. The processes comprise two-stage crushing, XRT ore sorting, SAG and ball milling, and two-stage concentration by rougher-cleaner flotation. Regrinding of the copper and zinc concentrate rougher streams is required in order to achieve design mineral liberation and recoveries. The reagent regimen was selected on the basis of extensive metallurgical testing and comprises a sulphur-dioxide based reagent scheme to depress the pyrite and the sphalerite during the copper flotation stage. The sphalerite is subsequently re-activated prior to the zinc flotation stage. Conventional collector and frother reagents are used in both the copper and zinc flotation stages. The construction of the flotation plant and site infrastructure, exclusive of the access road, will take one and three-quarter years. The flotation plant tailings are to be stored in a purpose-built TMF that utilizes downstream embankment construction. The embankment is lined with a 1.5 mm HDPE or equivalent bituminous geomembrane and a low-permeability soil under-liner. Once mining is complete the low-grade ore will be processed, and the pit will be filled with PAG waste from stockpiles. The PAG fill in the pit would be covered with an impermeable layer and a cap of nPAG waste rock. The tailings would similarly be covered by an impermeable layer followed by revegetated topsoil.

The selected ore processing methodology is backed by extensive historical metallurgical test work; including more recent work carried out under the direction of Kutcho Copper between 2018 and 2021. There is a small amount of oxidized ore (2.5% of total tonnage) for which there is no complete test work. The lack of direct test work on this proportion of the ore is not considered material to the project economics. The oxidized ore is not processed during the payback period (it stockpiled for processing at the end on mine life) and recovery factors (derived from sulphide-ore test work) are discounted. Recovery and concentrate grade formulas for the sulphide ore were developed based on the test work data. Expected recoveries and concentrate qualities decreased through experience-based factoring during the start-up period. Adequate test work data is available to provide operating parameters for flowsheet design and major equipment sizing within the contingency allowances normally associated with a FS.

The copper concentrate contains moderately high levels of zinc and some minor contaminants (lead, mercury, and cadmium) that may attract penalty payments. The concentrate quality would require careful management (mostly through blending) to avoid excessive penalties. Market analysis suggests that changes to smelter specifications may see further constraints placed on concentrate sales with respect to potential contaminants (cadmium and thallium, in particular) which may dictate further metallurgical analysis and/or modified blending strategies. Based on market analysis, the zinc concentrate is free of penalty elements, but the cadmium level is close to the concentrate importation limit and would require careful management to avoid occasional shipment rejection despite the concentrate being otherwise considered “First Grade” by Chinese National standards.

The Project’s copper and zinc concentrates are considered at this time to be acceptable; marketable products and sales parameters were selected to reflect the concentrate qualities being sold into the east Asian smelter market. Over the life of the Project the produced concentrate will contain payable metals totalling 504 Mlb of copper, 531 Mlb of zinc, 8.3 Moz of silver, and 107 koz of gold.

The Project has a comprehensive water management plan that integrates precipitation, surface flows, groundwater inflows, and water re-use with operational water requirements and seasonal discharge limitations. The site water balance indicates that the project has a net water surplus, and water will need to be treated and discharged on an annual basis. The project includes a system of contact and non-contact water separation and management that will serve to minimise the Project’s water treatment requirements. Water will be stored in the TMF over the winter season and discharged after treatment during spring, summer, and autumn.

The inclusion of the XRT particle ore sorter has a significant impact on the Project economics and introduces environmental benefits as well. Foremost of these benefits is the decrease in the size of the milling and flotation circuits of the processing plant, thus decreasing project capital and processing costs along with a commensurate decrease in GHG emissions due to the Project’s reduced power demands.

The ore sorter reject material will be used for underground backfill. Although untested for specific Kutcho materials but based on industry knowledge, the reject particle size and low sulphide content are considered benefits for CRF strength. The availability of the ore sorter reject for CRF also reduces the underground mining cost and the size of the TMF.

It is important to note that permitting of the Project is not complete. Kutcho has initiated the environmental and social assessment process, including the permitting procedures to meet British Columbian provincial and Canadian federal regulations. At the time of writing, environmental data is being compiles to meet statutory federal and provincial requirements.

The initial capital cost over a four-year construction period is C$493 million, inclusive of the access road construction and the commissioning period. Sustaining capital, at C$99 million, covers TMF expansions and major repairs for the mining equipment fleets and plant equipment. A further C$34 million will be required for closure and rehabilitation, and C$37 million is recovered from environmental bonds and salvage. At base case metal pricing the project generates a cumulative, after-tax NPV (discount rate of 7%) of C$461 million and an after-tax IRR of 25%. The project has a payback period of just over three years after commercial production commences.

The project includes an adequate provision for the closure and rehabilitation of the site in an environmentally sound and sustainable manner. The project will provide temporary employment for up to 350 construction workers and 400 permanent jobs during operations.

25.2 Risks

There are no known mining, metallurgical, infrastructure, permitting or other factors that could materially affect the valuation of the Project. However, the Project must not be considered to be without risk.

Risks to the mine plan not achieving the specified technical and financial parameters set out in the Feasibility Study include but are not limited to those set out below.

25.2.1 *Geotechnical*

- Paucity of drilling along the alignment of the decline between Esso and Main could lead to additional support requirements and may delay Esso ore production.
- Rock mass strength in the footwall rocks for the open pit are classified as fair to poor. Anomalous structures or local variations in rock quality may lead to pit wall instabilities. A potential outcome of an unfavorable local variation in geotechnical conditions may result in wall pushback and increased stripping requirements, the need for real-time pit wall monitoring, or a disruption to ore production. The Project is buffered against direct ore supply disruption from the pit by having a large, low-grade stockpile developed early in the project life. Additionally, the pit ramp is built on the better quality hangingwall, and a high-grade ore supply from Esso underground mine is available.

25.2.2 *Hydrogeological/Hydrological*

- Detailed post-closure groundwater modelling is required to confirm water quality in the backfilled Main pit for optimal prediction of long-term water treatment requirements. Whilst pit backfilling combined with capping layers of impermeable materials are considered likely to achieve a rapid rehabilitation of the area, further data and modelling is recommended to reduce this potential risk. It must also be noted that the operation will have other means of developing acceptable discharge qualities. These can be refined and adapted through the operating life of the mine.
- Additional geochemical characterization data and monitoring data is required to reduce uncertainty in geochemical loading rates. There is degree of uncertainty in water treatment operating costs during operations (though likely only minor changes and most likely reductions) and closure (if needed, and a much larger uncertainty).

25.2.3 *Open Pit Mining*

- Inadequate grade control practices and poor mine planning could lead to lower than predicted mining and plant throughput rates, and lower head grades. The key to overcome this threat is having a well-trained crew with good technical leadership and clear, well documented protocols. The Feasibility Study includes industry average pay levels and working conditions to enable attraction of personnel of the required quality but tightness in the labor market at the time of commencement may reduce that expected availability.
- It is considered practical and achievable to only use ANFO explosives, which has cost savings for both drilling and blasting as well as indirectly through water treatment cost (emulsion has a higher nitrate loading to the contact water run-off from the waste dumps). The mine plan includes a passive water drainage system and active in-pit drainage, and the potential that the underground mine will serve as a gallery drain for the open pit (this is evident in water modelling and included in underground pumping requirements). The mine operations costing also incudes provision for blast hole water clearance. However, it may still be possible that the rock mass does not prove drainable in the timeframe of the mining cycle from some presently

unforeseen reason. Some mines where the rock drainage cannot be overcome utilize a 70:30 Ratio of ANFO to emulsion. Should this outcome occur at the Project it is estimated that this would add C$0.05/t to the open pit mine operating cost.

25.2.4 *Underground Mining*

- Stope wall rock dilution was developed through geotechnical data collection and analysis to fully support the feasibility design. There is however still risk associated to the methods of estimation. Additional dilution due to poorer than expected stability the footwall and hangingwall rocks may occur. Increased dilution may be caused by more faulting than indicated from drilling and modelling, and this would then reduce the ore head grade.
- Whilst the mining recovery have data that correctly support the Feasibility Study based on experience, and geotechnical data. The next level of certainty will arise with actual mining. Lower mining recoveries, and in particular associated to lower recovery of the temporary rib pillars than the projected 90%, will reduce project valuation. As a guide based on other mines where geotechnical conditions are on average worse than that currently measured at the Project, using those parameters applied to the Project may reduce the NPV by 2%. The variance even if applied is not material to the Project.

25.2.5 *Processing*

- The QP for the plant flotation considered it appropriate, based on personal experience, that the flotation reagent regimen would be 25% lower in quantity as compared to the test work. The potential threat is that should 100% of the test regimen be required, this will add approximately C$3.10 to the plant operating cost (+11%) and decrease the NPV to C$431 million.
- Testing of the flotation circuit performance was based on set proportions of feed from the Main and Esso deposits. Based on these tests, plant performance is modelled to vary as feed proportions from Main and Esso vary. However, dynamic plant feed source variations may prove more challenging than the fixed proportion tests indicate. Variation of NPV to metal recovery and concentrate grades are given in Section 22.
- Uncertainty attributable to changes in the nature of the grinding circuit after the introduction of ore sorting to the project is a weakness with respect to operating cost projections for the process plant. Whilst the plant SAG and ball mill motor selections give capital contingency, more knowledge of the combined circuit in relationship to the specific energy required in grinding and its response to the ore sorter may improve cost prediction of this element of the operation.
- Comminution test work was completed on core from the 2008 and 2010 drill programs which may be oxidized to an unknown degree, particularly sulphide ore. Comminution test work on a larger fresh core sample or set of variability composites would provide more confidence in ore hardness.
- No concentrate fine-grinding comminution testing was undertaken to refine regrind mill selection. As the type of equipment is now more clearly defined, specific testing will de-risk equipment selection for detailed engineering.

25.2.6 *Tailings Management Facility*

- There are no geotechnical test pits to the west of Playboy Creek, which comprises 50% of the area of the TMF. Whilst the observed and extrapolated geology (also observed in drilling) does not change across the creek, test pits will be required for detailed design. Should the test pits find different overburden depths than those assumed in the Feasibility Study design, costs for development of the TMF may vary depending on those findings. The potential variations based on other test pits across the project area suggest that this would not result in a material change to the design.

25.2.7 Infrastructure

- Variations in construction costs may be incurred as no geotechnical investigations have been carried out for the diversion canals, explosive storage area, and treated water discharge access road. Whilst utilizing average expected depths is a reasonable choice for these minor cost centres, there is still a risk present in not completing test pits in these specific locations. A doubling of civil costs associated to a variant outcome could add an estimated C$1.0 million to the Project's costs and as such is not material to the project.
- Open-source topography data was used for the design of the treated water discharge access road and the explosives storage structures. Conservative parameters have been used in these locations. There is a risk that final conditions might result in higher (or lower) construction costs. A doubling of civil costs associated to a variant outcome could add up to an estimated C$1.0 million and halving would result in C$0.5 million savings. The size of the variation is not material to the project.

25.2.8 Concentrate

- The absence of assumed concentrate storage space at Stewart may result in Kutcho incurring costs for the storage of concentrate in a warehouse to be built, leased or rented.
- Whilst the concentrates currently meet Chinese quality constraints, it is possible that there may tightening of controls. Strategies to overcome this issue that could be applied include further test work to refine the understanding of deleterious metal concentration, using a different flotation reagent scheme, and concentrate blending strategies with other suppliers.

25.2.9 Financial

- Project value is strongly dependent on copper pricing. The project pricing method utilizes the upper range of consensus pricing. The pricing assumes that global copper demand is greater than supply due primarily to the transition to a low carbon economy. Whilst this is considered an appropriate stance to take, a copper price below US$3.50/lb enduring for the entire duration of the project is still a possibility. The project has a relatively low production cost after construction and therefore the operational life is moderately robust to downward pricing. However, return on investment is more sensitive. Charts reflecting metal price sensitivities are included in Section 22 as they have a material bearing on the project.
- The Feasibility Study base case assumes that the Wheaton PMPA is not realized. Should Wheaton choose to continue with the agreement, this will reduce the after-tax project NPV to C$417 million from C$461 million. The PMPA does increase the after-tax IRR of the Project to 26% and reduce the total funding requirements. The open pit and underground mine plans were developed with a cut-off and optimization strategy that assumes the PMPA is realized. Consequently, there are marginal gains that may be attributed to the base case (discussed elsewhere in this section). Whilst this is material to cashflow is not material to the Mineral Reserve (Section 15) nor the mine plan.
- The Project valuation assumes that Royal Gold will not exercise back-in rights, which is considered the most likely outcome, but is not assured.
- The Feasibility Study assumes that the Project will be able to claim the prescribed allowance for new mines for BC Mineral Tax purposes. One third of the cost of capital assets and costs are added to the pool of deductible expenditures incurred for the purpose of bringing the Project into commercial production. If the Project is unable to claim the new mine allowance (should the allowance not be extended beyond 31 December 2025 or the Project is unable to meet the conditions for claiming it) the after-tax NPV would reduce by C$12 million and the after-tax internal rate of return would reduce by 0.3%. It is considered appropriate to apply the allowance given that it was extended twice previously.

- Higher inflation than the average over 2010–2020 would change the valuation. Labour market restrictions would also lead to higher costs. However, inflation must also be considered in relation to metal prices which respond to the same pressures.

25.2.10 Permitting

- There is a material threat to the Project's realization and profitability due to uncertainty pertaining to potential regulatory delays in obtaining the necessary permits.
- Treated water discharge is an important part of the project. Should end of pipe discharge criteria become more stringent as a result of the environmental assessment process, more intensive water treatment may be required.

25.3 Opportunities

Opportunities for improving the performance of the Project include but are not limited to those set out below.

25.3.1 Geological

- Further delineation drilling could upgrade portions of the current Inferred Mineral Resources to Indicated or Measured categories.
- The project tenure has not been fully explored. Further exploration may yield other presently unknown Mineral Resources.

25.3.2 Geotechnical

- A Project strength is that, with the exception of the altered footwall domain and discrete fault zones within the Main open pit, the majority of the rock masses encountered at Kutcho (both Esso and Main) are in the fair to good range. Pit accesses were designed within these stronger materials.
- Whilst there is infrastructure-related geotechnical data, these are not available in the exact locations of the proposed site infrastructure. This has led to a conservative selection of civil work design criteria (in particular, concrete thickness). Increased geotechnical testing will allow for less conservative designs and therefore potentially decreased capital costs.

25.3.3 Hydrogeological/Hydrological

- It is considered that the assessment of the contact-water qualities influent to the water treatment plant resulted in a conservative water treatment plan. The findings suggest that the proposed discharge plan is expected to result in acceptable water qualities within the receiving environment. The conservative approach has led to a robust system with the potential flexibility to accommodate more capacity than the predicted peak treatment rate.
- Stormwater management infrastructure was completed to a Feasibility Study level design, but a more efficient and cost-effective design may be achieved during the detailed design phase of the project.

25.3.4 Open Pit Mining

- The cut-off applied to the Mineral Reserve is above the breakeven cut-off. This marginally profitable material, if processed could increase the $NPV_{7\%}$ by C$4 million. Alternatively, stockpiling low-grade ore serves to leverage potentially higher future metal prices.
- Open pit mining: the grade control method selected does not allow for any benefits that in-field control of mining by a geological professional or excavation equipment ore sensor units could add. These methods of decreasing waste and adding ore were not considered in the Feasibility Study but may be applicable to this style of mineralization.

25.3.5 *Underground Mining*

- The mine plan does not consider the recovery of the Main crown pillar. It is estimated that if this were recoverable at the end of mine life up to 0.25 Mt may be added to the reserve base and increase the $NPV_{7\%}$ to C$480 million.

25.3.6 *Processing*

- The plant comminution motor sizing was selected above the needs indicated by the selected average power consumption. This gives robustness against factors such as the impacts of the ore sorter which may remove harder or softer material at times preferentially. Further comminution test work and design may reveal other plant configuration of the comminution circuit that may reveal reductions in capital or operating cost.
- The ore sorter is based on a single bulk composite sample. Whilst that composite is considered representative of the overall mineralization, it does not give insight into how specific ore types may perform. It is noted that the ore sorter, on the basis principally of density, endeavours to reject rocks that contain little sulphide material. This fact may lead to a mining method strategy that perhaps mines more material at the edges of the open pit ore than is designed for at present, thus reducing mining losses and allowing for the added dilution to be removed by the ore sorter. Without extensive and expensive tests, this feature is unlikely to be resolved before operations commence. Similarly, this feature would potentially make the mine more robust to dilution impacts. Note that the open pit mine mining loss for the Mineral Reserve amounts to 0.79 Mt at 0.91% Cu, 0.98% Zn, 15 g/t Ag and 0.53 g/t Au.
- The ore utilizes XRT technology that sorts the rock essentially based on density (through general sulphide opaqueness to X-rays). Concentrating differentiation based on mineral-type discrimination rather than total sulphide content has the ability to further increase performance. One of the most common sulphide mineral present in the ore is pyrite, which does not contribute to concentrate value. Consequently, there could be additional value derived if rocks that contained only pyrite could be rejected. Near Infra Red technology was tested for this purpose but was not found to be effective. However, there are many other sensor types and further research may add economic benefits to the project.

25.3.7 *Infrastructure*

- The embankment height is easily raised with a commensurate increase in tailings storage capacity. The storage efficiency of the facility improves with added height. The uppermost 6.5 m of the embankment facilitates an additional 3.5 Mt or 27% of the total storage. There is a large source of buttress material available at site and location geometries are favourable to add further height to the embankment.
- Access road funding could be provided, either partially or totally, by third parties.
- The installation of an LNG powerplant is the best economic choice for the Project at the time of the Feasibility Study. Connection to the BC Hydro grid would provide additional power cost saving if a connection point closer than those currently available (at the Red Chris mine for example) become available. Notably, grid power also comes with GHG emission benefits whereas the BC grid is primarily hydro powered

25.3.8 *Financial*

- Higher metal prices and in particular higher copper prices have the strongest impact on project value. The relative impacts are discussed in Section 21 of this report. The analysis in Section 21 was completed without altering the cut-off or mining shapes. This approach is conservative. If the cut-off were altered and the designs altered in response to lower or higher prices, the resulting NPV would be higher with higher pricing and higher with lower pricing. The amount of leverage would require knowledge of the precise conditions over which this would occur – for instance a decision taken after capital construction is quite different to prior. Given these vagaries, the conservative simplistic approach was taken for project advice.

- The feasibility considers the use of only new equipment. There may be some non-critical applications where the used of good quality, second-hand equipment may have favourable benefits.

26 Recommendations

26.1 Introduction

CSA Global believes the technical outcomes and economic results of the Feasibility Study support the statement of the Mineral Reserve estimate. Based on these conclusions CSA Global recommends that Kutcho continue with the environmental approval process and initiate detailed engineering. Outside of the environmental process and not including the cost of detailed engineering, the programs as set out below are recommended. If adopted could enhance the accuracy of project planning and economic outcomes and decrease project technical risk.

Recommendations specific to each of the report sections are set below.

26.2 Mineral Resources and Exploration

Further exploration success is not critical to the next stages development. However, it is recommended that the currently available exploration data be evaluated in order to develop a strategic plan of exploration aligned to Project needs.

26.3 Regulatory Processes

It is recommended that the environmental and social assessment process, including the permitting procedures to meet British Columbian provincial and Canadian federal regulations, be completed.

26.4 Geotechnical/Hydrogeology

Further geotechnical investigative and confirmatory work is recommended to support the detailed design:

Main open pit:

- Additional geotechnical drilling at azimuths not well represented in the current geotechnical model will need to be completed to assess the potential for orientation bias in the structural database.
- Further hydrogeology data (especially packer tests) should be collected from any additional geotechnical holes relating to assess the potential effectiveness of a passive drainage system for the footwall rocks and inclusive of the underground mine as a gallery draw system.

Main underground and portal:

- Geotechnical drillholes at the portal location will be required.
- Re-evaluation of the Main ventilation raise given its short life. Relocation of the raise to a hangingwall position may be favourable.

Ramp from Main underground to Esso underground:

- At present no geotechnical data exists along the interconnecting Main-Esso ramp alignment. Some geotechnical holes along the alignment will reduce the risk to unforeseen ground conditions and may allow refinement of ground support requirements.

Esso underground:

- Borehole breakout analysis to improve the understanding of the in-situ stresses at Esso and may lead to a re-evaluation of stope dimensions.
- Undertake uniaxial compressive strength (UCS) testing of CRF to optimise cement content (tests to be conducted on ore sorter reject material).
- Geotechnical drilling targeting the major ventilation raise positions and further hydrogeological evaluation.

26.5 Mining

No further work is recommended to be undertaken for the open pit before construction unless geotechnical or hydrological studies alter the pit wall angles or there are other material changes in environmental, metallurgical, or financial parameters.

Undertake detailed manual planning of the mining sequence using the stope optimizer results as a template but adding local structural geotechnical planning detail for individual stopes, accesses, and the location of temporary rib pillars. The study should also incorporate detailed planning around the use of remote wireless detonation and explosives optimization to ensure smooth operational start-up.

26.6 Processing Recommendation:

While the copper and zinc recovery process is considered to be robust and reliable for the majority of the Main and Esso deposits, some additional ore sorter and comminution optimization studies are recommended as are some rougher test work of the partially or weakly oxidized material. Some equipment selection refinement may be possible through added testing and analysis. The recommended work includes:

- Three bulk tests for ore sorter are recommended. These should be selected from the ore planned to be mined in the first three years of mine life, which is 90% dependent on the Main deposit. The samples should comprise high, medium, and low-grade ore. Whilst medium grade ore would be a near repeat of the already completed bulk test (the original included an average proportion from the Esso deposit), this would provide repeatability information and material that would subsequently be used for comminution testing.
- Comminution testing should evaluate the impact that ore sorting has on the ball mill work index results with “before ore sorter” and “after ore sorter” work index tests. These would be compared to previous comminution testing which is “before” ore sorting. A program for comminution testing would then be finalized based on the measured impact. If no discernible impact is observed (which is the basis of the cost estimate), then the recommended comminution program would be:
    - An initial comminution test work program comprising three global composites representing the proposed mine plan for Year 0 to 1, 1 to 3 and 3 to end of mine life
    - A subsequent variability test of 30 to 45 discrete samples representing discrete continuous intervals of mineralization
    - The program should include CWi, SMC, RWi, BWi, fine grinding and abrasion index test work
    - Program costs include provision for new drillholes since existing core samples of ore for testing are few and those that do exist have aged, which may affect comminution results.
- A review of comminution testing benefits for the flotation concentrate re-grind units should be completed.
- Geometallurgical data compilation to refine the understanding of oxidized ore behaviours with the undertaking of a set of rougher tests to confirm impacts on mass pull and recoveries. Some shallow drillholes will be required.

26.7 Infrastructure

The TMF requires geotechnical test pits and drilling as part of detailed design. Delaying this work to detailed engineering will allow modifications to the design and storage capacity as a result of outcomes from the ore sorter, water treatment and hydrogeology programs. Other work that will reduce risk and improve infrastructure design and construction, and are independent of other processes that include:

- Further geotechnical investigations in the locations of the proposed pads, canals, access roads and foundations in order to optimize the designs and complete detailed design
- Open-source topography data was used for design of the explosives storage area and its access roads. Completion of a LiDAR survey in these areas will optimize the design and reduce risk

- Drilling a potable water test well within 100 m of the proposed camp site and undertake yield testing to ensure sufficient potable water is available.

26.8 Environmental Studies

The final Project design is yet to be determined, but by initiating the EIA process early, outcomes of the EIA can be used to improve the Project design and maximise the benefits of the EIA without incurring excessive costs. There are a few potential improvements that the Project should evaluate:

- Progress humidity cell testing of PAG and nPAG waste rocks from the pit. Combine the results with investigations for control of higher risk elements through rock stockpile management.
- Complete a process water treatment test to prove effectiveness of the selected process.

26.9 Summary

The recommendations set out above cover the period up commencement on detailed engineering. Detailed engineering costs are not estimated nor are any outcomes of the exploration strategic assessment.

Table 26-1: Estimated cost of studies and activities prior to detailed engineering

Recommended activity/study	Estimated cost (C$k)
Exploration strategic assessment	50
Regulatory Processes	In progress
Geotechnical/Hydrogeological	1000
Mining	400
Metallurgical	1,000
Infrastructure	400
Environmental	100
Total	**2,950**

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28 Abbreviations and Units of Measurement

%	percent
°	degree(s)
°C	degree(s) Celsius
3D	three-dimensional
a	annum/year
ABA	acid based accounting
ARD	acid rock drainage
AAS	atomic adsorption spectroscopy (metal analyzer)
AIA	archaeological impact assessment
Ag	silver
AMSL	above mean sea level
AP	acid potential
ARMC	American Reserve Mining Corporation
ATV	acoustic televiewer
Au	gold
bcm	bank cubic meter
BCWQG	BC water quality guidelines
BC CDC	BC Conservation Data Centre
BM	ball mill
BML	Base Metal Laboratories
BVC	Bureau Veritas Canada
CAD$	Canadian dollars
CCME	Canadian Council of Ministers of the Environment
CEA	Canadian Environmental Assessment Agency
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
cm	centimetre(s)
Conc or C	concentrate (stream)
Corg	concentration carbon organic
CRF	cemented rock fill
CSA Global	CSA Global Consultants Canada Limited
Ctot	concentration carbon total
Cu	copper
$CuSO_4$	copper sulphate
CWP	contact water pond
DCF	discounted cash flow
DGPS	differential global positioning system
dmt	dry metric tonne
DRA	DRA Global Consultants
EA	environmental assessment
EAO	environmental assessment office

EML	Esso Minerals Limited
ELOS	horizontal dilution
FA	fire assay
Flot	flotation (process)
FS	Feasibility Study
g	gram(s)
GHG	greenhouse gas
g/L	gram(s) per litre
g/t	grams per tonne
GW	giga watt
HCA	enviro section20.6.55
HDPE	high density polyurethane
hpa	hours per annum
ha	hectares
IAAC	Impact Assessment Agency Canada ()
IBECO	incremental breakeven cut-off grade
IBC	intermediate bulk container
ICP	Inductive Coupled Plasma (multi element analyzer)
ISA	in-stream analysis
IRR	internal rate of return
IRA	Inter-ramp angle
JV	joint-venture
KBX	potassium butyl xanthate
kcfm	thousands of cubic feet per minute
kg	kilogram(s)
kg/t	kilograms per tonne
km	kilometre(s)
km^2	square kilometre(s)
koz	thousands of ounces (troy)
KSA	King Salmon Allochthon
kt	thousands of tonnes
ktpd	kilotonnes per day
ktpa	thousands of tonnes per year
Kutcho Copper	Kuthco Copper Corp.
kWh	kilowatt hour
l	litre(s)
L/s	litres per second
LCT	locked cycle test
LLHOS	longhole open stoping
LG	Lerch-Grossman
LOM	life of mine
LSA	local study area
$/lin m	dollars per linear metre

m	metre(s)
m^2	square metre(s)
m^3	cubic metre(s)
m^3pa	cubic metre(s)/annum
mbgl	metres below ground level
MIBC	methyl isobutyl carbinol
ML/ARD	mineral leaching acid generating
mm	millimetres(s)
Mm^3	million cubic metre(s)
Moz	millions of ounces (troy)
Mt	million tonnes
Mtpa	million tonnes per annum
MYAB	multi year area based permitting
MW	mega watt
MWh	megawatt hour
NI 43-101	National Instrument
NIR	near infrared
nPAG	not potentially acid generating
NP	neutralising potential
NSR	net smelter return
O_2	oxygen (gas)
ophpa	operating hours per annum
OREAS	certified reference material, a brand of
OSA	overall slope angle
OTV	optical televiewer
oz	troy ounce(s)
MIRR	Ministry of Indigenous Relations and Reconciliation
MDMER	Metal and diamond Mine effluent regulation
MOTI	Ministry of Transportation and Infrastructure
MYAB	multi-year area based permitting
PAG	potentially acid generating
PAX	potassium amyl xanthate
PEA	preliminary economic assessment
pH	scalar measure of acidity logarithmic
PFR	preliminary field reconnaissance
PFS	Pre-Feasibility Study
ppm	parts per million
PMPA	precious metals purchase agreement
Project	Kutcho Copper Project
pH	scalar measure of acidity logarithmic
ppm	parts per million
QAQC	quality assurance and quality control (for sampling and assaying)
QEMSCAN	quantitative evaluation of materials by scanning electron microscopy

RDKS	Regional District of Kitimat Stikine
RSA	regional study area
RWD	raw water dam
RO	reverse osmosis
ROCE	return on capital employed
ROM	run of mine
RTK	differential geographic positioning system
SAG	semi-autogenous grinding
SARA	Species at Risk Act
SEM	scanning electron microscopy
SG	specific Gravity
SML	Sumac Mines Ltd
SMU	selective mining unit
SO_2	sulphur dioxide
Ss	concentration sulphur as sulphide
Stot or S	concentration sulphur total
t	tonne(s) metric
TAU	Tuff-Argillite Unit
tails or T	tailings (stream)
TCL	Teck Cominco Ltd
TDS	total dissolved solids
TLHOS	transverse longhole open stoping
TMF	tailings management facility
tpa	tonnes per annum
tpd	tonnes per day
tph	tonnes per hour
TRP	temporary rib pillar
µm	micrometer
UTM	Universal Transverse Mercator
US$	US dollars
VFD	variable frequency drive
WGS	world geodetic system
WKM	Western Keltic Mines
wmt	wet metric tonne
WRD	waste rock dump
w/v	number of grams of an ingredient in 100 mL of solution
w/w	weight by weight
XRD	x-ray diffraction
XRF	x-ray fluorescence
XRT	x-ray transmission
Y or CY	year

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