# N.I. 43-101 Technical Report & Feasibility Study of the Cerro Blanco Gold Project

## Department of Jutiapa, Guatemala

Effective Date: February 22, 2022

Issue Report Date: April 06, 2022

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on the Cerro Blanco Project

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# 1. SUMMARY

## 1.1 Introduction

This report was prepared by G Mining Services (GMS) for Bluestone Resources Inc. (Bluestone) to summarize the results of the N.I. 43-101 Technical Report and Feasibility Study (FS) on the Cerro Blanco Gold Project (the Project). This report was prepared following the Canadian disclosure requirements of National Instrument 43-101 (N.I. 43-101) and in accordance with the requirements of Form 43-101 F1.

The Cerro Blanco Gold Project was originally studied and permitted for a high-grade underground operation in 2007. The recent engineering development, together with extensive additional drilling and detailed geological studies, significantly enhanced the understanding of the Project and presented an opportunity to capitalize on its near-surface mineralization through open pit development and improved Project economics.

Prior to pivoting to an open pit development scenario, three technical reports were sanctioned by Bluestone Resources and prepared by JDS Energy and Mining Inc. for the Project: a preliminary economic assessment (March 20, 2017), an updated preliminary economic assessment (June 2, 2017) and an FS (January 29, 2019). All of these first three technical reports and resource estimates were studying an underground mining scenario.

Following extensive field investigations and further technical analysis by Bluestone and contracted technical experts, a fourth report (Preliminary Economic Assessment dated February 28, 2021) was prepared for an open pit development scenario by G Mining Services.

All four technical reports are filed on the System for Electronic Document Analysis and Retrieval (SEDAR).

## 1.2 Project Description

A description of the Project is summarized below:

- Mining ........ open pit operation
- Processing ........ crushing-grinding-cyanidation-refining
- Life-of-Mine Mill Feed ........ 53.9 Mt @ 1.64 g/t Au, 7.27 g/t Ag
- Plant Recovery ........ 93.0% Au, 84.3% Ag
- Average Payable Gold ........ 193,000 oz/year
- Mine Life ........ 14 years

The process plant will utilize crushing, grinding, cyanidation leaching and refining processes to recover gold (Au) and silver (Ag). On average, 297,000 oz of gold will be recovered annually during the first four years. The mine life will be 13.7 years based on current defined resources.

Tailings will be filtered and deposited in an engineered dry stack tailings facility (DSTF). Waste rock will be deposited in the waste rock facility or used as construction material for the DSTF.

The site is currently accessible by an all-weather gravel road and a new access road will also be built. Power for the Cerro Blanco Gold Project will be supplied by a transmission line that will be installed prior to start-up.

The Project was granted an exploitation licence through an approved Environmental and Social Impact Assessment (ESIA) that was granted in 2007. The Cerro Blanco Gold Project is fully permitted as an underground mine. However, an amendment to the approved 2007 ESIA will be required due to the proposed change in mining method to an open pit. The amendment to the 2007 ESIA will be completed through an updated permit amendment application. New ESIA's and permits are also required for the new access road and powerline. The amended Environmental and Social Impact Assessment (ESIA) has been submitted to the Ministry of Environment and Natural Resources (Ministerio de Ambiente y Recursos Naturales (MARN) to support the FS design of the open pit scenario.

## 1.3 Property Description & Location

The Project is located in southeast Guatemala, in the Department of Jutiapa, approximately 160 km by road from the capital, Guatemala City (Figure 1-1) and approximately 9 km west of the border with El Salvador. The nearest town to the Project is Asunción Mita, a community of about 18,500 people situated approximately 7 km west of the Project. The exploitation license covers 15.25 $km^2$ and lies entirely in the municipality of Asunción Mita.

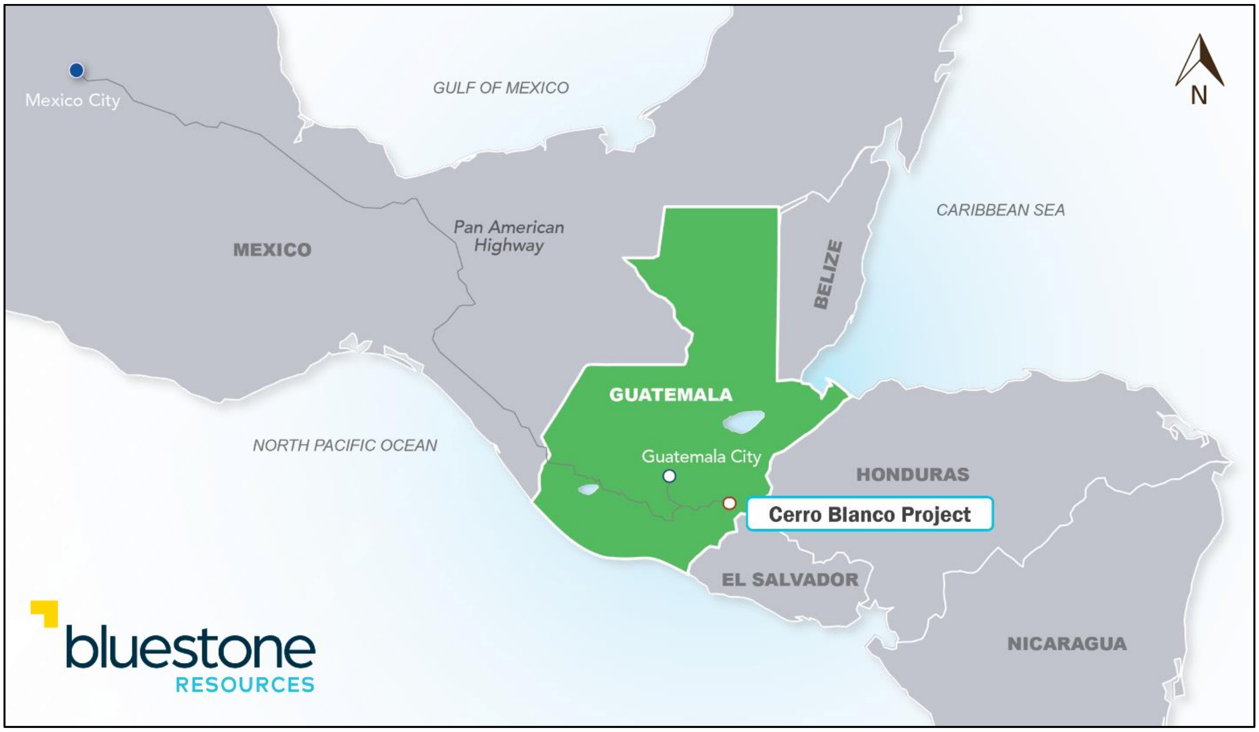
## 1.4 Ownership & Mineral Tenure

The Cerro Blanco Gold Project exploitation concession, which covers an area of 15.25 $km^2$, is owned by Elevar Resources S.A. ("Elevar"). Elevar is an indirect, wholly owned subsidiary of Bluestone Resources.

The Cerro Blanco Gold Project exploration concession was originally awarded to Entre Mares de Guatemala S.A. (Entre Mares) on November 12, 1997 and was converted to an exploitation concession in 2007. Bluestone Resources acquired 100% of Entre Mares from Goldcorp Inc. (Goldcorp) in 2017. In October 2021, Entre Mares underwent a name change to Elevar Resources S.A. as part of a re-branding campaign to reflect the improved Project and the enhanced corporate strategy.

Bluestone has purchased the land over the deposit and for the plant facilities. The majority of the additional land required to optimize the location of the surface infrastructure and serve as a buffer to local communities has been secured.

**Figure 1-1: Property Location**




Source: Bluestone (2022)

## 1.5 Accessibility, Climate, Local Resources, Infrastructure & Physiography

Current road access to site is via the Pan-American Highway (Highway CA1) through the town of Asunción Mita. Existing infrastructure is in place to provide year-round access to the site. The topography is relatively flat with rolling hills.

The climate and vegetation at the Project site are typical of a tropical dry forest environment. The elevation is between 450 and 560 masl. The wet season is typically from May to October. The average annual rainfall is 1,350 mm. Daily temperature highs reach 41°C and lows reach 10°C. The average annual pan evaporation rate is 2,530 mm with an annual average humidity of 62%.

The Project is situated in proximity to a number of communities, the largest one being Asunción Mita, with a population of approximately 18,500 people. The recently constructed La Baranca power substation is located a few kilometers south of Mita. The substation has a capacity to supply up to 20 MW of power.

There is no record of any previous mining in the area; however, with the closure of Goldcorp's Marlin Mine in late 2017 it is anticipated that a significant contingent of Guatemalan trained labour will be available for employment at Cerro Blanco. As such, the Project intends to hire the majority of operations staff locally and has allowed for cost of training programs within the Owner's budget.

A portion of the mine workforce is expected to live at the mine site in a purpose-built permanent camp, while employees living in the surrounding communities will provide their own transportation to and from the mine site. For those employees living in the wider Jutiapa region and in areas further afield from Asuncion Mita, the company will provide transportation to and from the mine site from certain locations. There are several population centers near the Project site.

## 1.6 History, Exploration & Drilling

The Cerro Blanco property was identified by Mar-West by sampling of densely silicified boulders. In October 1998, Mar-West's holdings in Honduras and Guatemala were purchased by Glamis Gold Ltd. In November 2006, Goldcorp Inc. became the sole proprietor of the Project through the purchase of Glamis Gold. Goldcorp undertook a comprehensive exploration program from 2006-2012 including additional surface exploration, over 3.4 km of underground development, and 43,016 meters of surface and underground drilling. On January 4, 2017, Bluestone entered into an agreement with Goldcorp to acquire 100% of the Project.

As of the end of 2021, Bluestone had drilled approximately 267 holes for a total of 45,725 m on the Cerro Blanco property since the acquisition from Goldcorp. Table 1.1 summarizes the historical drilling on the property.

**Table 1.1: Summary of Drilling**

| Year | Company | Holes Drilled | Meters |
|---|---|---|---|
| 1998 | Mar-West | 9 | 1,340 |
| 1999 | Glamis | 48 | 7,074 |
| 2000 | Glamis | 18 | 3,525 |
| 2002 | Glamis | 23 | 6,525 |
| 2004 | Glamis | 42 | 9,370 |
| 2005 | Glamis | 120 | 29,065 |
| 2006 | Glamis | 67 | 15,129 |
| 2007 | Goldcorp | 47 | 12,373 |
| 2008 | Goldcorp | 2 | 586 |
| 2009 | Goldcorp | 1 | 140 |
| 2010 | Goldcorp | 10 | 2,277 |
| 2011 | Goldcorp | 28 | 5,898 |
| 2012 | Goldcorp | 96 | 21,370 |
| 2017 | Bluestone | 8 | 2,324 |
| 2018 | Bluestone | 74 | 13,993 |
| 2019 | Bluestone | 61 | 8,403 |
| 2020 | Bluestone | 74 | 15,172 |
| 2021 | Bluestone | 50 | 5,833 |
| Total | | 778 | 160,397 |

Source: Bluestone (2021)

## 1.7 Geology & Mineralization

The Cerro Blanco Gold Project is a classic hot springs-related, low-sulphidation epithermal gold-silver deposit comprising both high-grade vein and low-grade disseminated mineralization. The Cerro Blanco district forms part of an active volcanic arc of Miocene-Pliocene-aged bimodal volcanism that extends through El Salvador, Honduras, and Nicaragua.

High-grade mineralization is hosted in the Mita unit as two upward-flaring vein swarms comprising over 60 veins (North and South Zones) that converge downwards and merge into basal feeder veins. Low-grade disseminated and veinlet mineralization within and as halos around the high-grade veins is well documented in drilling since discovery of the deposit. Most of the veins are blind to surface, and concealed by the syn-mineral Salinas Unit, a sub-horizontal sequence of volcanogenic sediments and sinter horizons approximately 100 meters thick that form the low-lying hill at the Project. The Salinas cap rocks are host to low-grade mineralization associated with silicified conglomerates and contemporaneous dacite / rhyolite flow domes or cryptodomes.

In profile, the inverted wedge-shape of the high-grade veins (upward flaring arrays) and their low-grade halos overlain by the mineralized Salinas cap rocks render the deposit amenable to exploitation by surface methods with a low strip ratio.

Both high and low-angle banded crustiform / colloform chalcedony veins, locally with calcite replacement textures, make up the deposit, with bonanza-grade gold grades largely confined to the chalcedony-quartz veins, especially where adularia bands are prominent. High-grade mineralization occurs over a vertical profile of 400 m (150 to 450 masl). At depth, calcite dominated veins form the limit to mineralization, nonetheless, very locally, high-gold values are present in calcite-dominated veins and in silicified structures containing only minor quartz veinlets.

The Salinas Group includes thin hot spring deposits, including sinters, which are genetically linked to underlying swarms of epithermal, gold-silver bearing quartz veins. The west and east sides of the Cerro Blanco ridge consist of flat agricultural plains characterized by Quaternary basalts, interbedded with boulder beds and sands. These rocks also appear down-faulted to lower elevations, implying major post-mineral extensional movements on such faults.

The current gold resource occurs under a small hill within an area 400 m by 920 m. Gold-bearing structures in the Cerro Blanco Project area extend 2 km to the northwest of the gold deposit and occur largely confined within the hydrothermal alteration zone. The extensive drilling undertaken to date of the high-grade vein swarms and their surrounding low-grade mineralized envelopes and overlying mineralized cap rocks show impressive intercepts, including 203.8 meters grading 2.3 g Au/t and 8.1 g Ag/t (CB20-420) and 87.2 meters grading 5.9 g Au/t and 32.5 g Ag/t (UGCB18-89).

Vein textures suggest that gold and silver were introduced as one major event of multi-stage finely banded veining (originally amorphous silica) with subordinate bands of platy calcite which is mostly pseudomorphed to cryptocrystalline silica phases. Repetitive “crack and seal” pulses and associated boiling/flashing events very close to the paleosurface are proposed as the main mechanism for precious metal deposition. Very high-grade core intersections with coarser and more abundant sulphides, electrum, and free gold appear to represent an earlier series of events. Deportment studies indicate that approximately 99% of the gold occurs in electrum as free or exposed grains, with lesser amounts as native gold and kustelite. The lack of post-mineral structural displacement of veins and distribution of high grades over a +400 m vertical profile attest to the pristine nature of the veins at Cerro Blanco. Lack of inter-stage hydrothermal brecciation and coarse-grained primary quartz textures suggest that the mineralizing event was a fairly short-lived and occurred very close to the paleosurface.

## 1.8 Sample Preparation & Data Verification

Continued data validation and verification processes have not identified any material issues with the Cerro Blanco sample and assay data.

During Q3 and Q4 2020, the Cerro Blanco drill and assay database was moved to the AcQuire - GMSuite platform hosted by CSA Global, providing an enhanced and more secure standard of data management.

It is the opinion of the QP, Garth Kirkham, P. Geo., that the sampling preparation, security, analytical procedures and quality control protocols used by Bluestone are consistent with generally accepted industry best practices and are therefore reliable for the purpose of resource estimation.

## 1.9 Mineral Processing & Metallurgical Testing

Metallurgical testwork was previously completed at Base Metallurgical Laboratories Ltd. (BaseMet) in Kamloops, Canada on ore samples at depth when the Project was considering an underground mine operation. Recent feasibility metallurgical testwork at BaseMet focused on testing ore samples near surface in the Salinas Zone to support the FS and current proposed open pit mine operation as well as composite samples representing the initial years of mining. Gold deportment studies were also completed by Surface Science Western (SSW), a department at the University of Western Ontario and vendor tailings dewatering tests were completed by Metso:Outotec using their laboratory equipment located at SGS Lakefield.

The primary objectives of the feasibility metallurgical testwork program were as follows:

- assess the ore properties (chemical and mineralogical) of a number of discrete variability samples, primarily from the Salinas zone
- assess the metallurgical performance of several discrete variability composites utilizing cyanide leaching to recover gold and silver
- test composites representing the initial mining years
- conduct specific tests to generate engineering data such as tailings settling, carbon loading, carbon adsorption and cyanide detoxification data
- generate tailings for environmental, geotechnical and vendor dewatering studies.

The test program was organized and completed over two main phases. An additional sub test program was initiated during phase 1 (called Phase 1B) to investigate the poor metallurgical performance of some of the Phase 1 samples:

- Phase 1 – Variability Assessment
- Phase 1B – Investigation of Poor Metallurgical Performance of Phase 1 Samples
- Phase 2 – Mine Plan Composites and Engineering Data

The drill core and assay laboratory coarse rejects were first used to make variability composites. Using the already developed flowsheet, the samples were tested and evaluated for metallurgical response. The variability data was compared to the feed sample characteristics to explain any anomalous metallurgical response. Once the analysis was complete, variability samples and other samples were then used to assemble larger composites for detailed testing to generate advanced engineering data.

Comminution tests were previously completed on various ore lithologies and dilution material to determine sample abrasiveness and energy requirements for comminution circuit design. The average BWi was determined to be 20 kWh/t. There was no distinct variation in the BWi between the various lithological zones tested. RWi tests indicated that the ore was moderately hard, averaging 18 kWh/t. The SMC A x b values were determined when sufficient sample mass was available. This parameter provides a measure of resistance to impact breakage in a SAG mill. A smaller number means that the sample had a higher level of resistance to impact breakage. The average A x b value, as derived from the SMC test, was 39.3 indicating these samples are classified as hard by the JKTech report. When sample mass was not sufficient for an SMC test, a hardness index test (HIT) was conducted on the remaining samples. The HIT test is an abbreviated test using a small sample size, providing an estimate of the A x b value. The HIT A x b value of these samples averaged at 39.2, similar to the SMC derived A x b value. The Bond abrasion indices for a number of samples was measured and averaged 0.47. These samples would be considered moderate to highly abrasive. The comminution results determined there was no specific trend developed relating a physical property or geological lithology of the sample to hardness. Furthermore, there was little variance in the comminution properties of the variability samples tested.

The Phase 1 feasibility metallurgical testwork program evaluated direct cyanide leaching. The previously developed flowsheet was used to evaluate the samples (i.e., samples were ground to a primary grind size of $P_{80}$ of 53 µm); prior to leaching, the ground sample was subjected to a two-hour pre-oxidation step with oxygen. The leach test was a bottle roll test with 2, 6, 24 and 48-hour sampling periods. Lime and cyanide levels were measured at each sampling period and were adjusted to maintain test parameters. The leach solution had 500 ppm NaCN and the pulp pH was maintained at 10.5 with lime. Lead nitrate was added at a dosage of 250 g/t. There was considerable variability in metallurgical performance. Some samples achieved over 93% extraction which was consistent in previous metallurgical testwork programs. Other samples had significantly poorer extractions. Three metallurgical ore domains were identified in the Salinas zone (i.e., Zone 1 (+85% recoveries), Zone 2 (60-85% recoveries) and Zone 3 (less than 60% recoveries).

Phase 1B testwork involved several investigative tests to evaluate the poor recoveries of Salinas Zones 2 and 3 samples and included adjusting process parameters, completing repeat tests, diagnostic leach tests and flotation tests. Detailed gold deportment and geological studies confirmed that the low recoveries of Zone 3 samples were due to very fine (<5um) locked-up gold grains within approximately 1 Mt of predominately strongly silicified and un-oxidized Scgl conglomerates at their deepest level in the South Zone and were consequently designated as waste in the mine plan. The inclusion of a flotation circuit can increase the gold recovery of Zone 2 by approximately 10% but a trade-off study concluded it was not economically feasible to add this circuit. Zone 2 ore occurs deeper in the mine and is only expected to be treated later in the mine life.

For Phase 2, two composite samples were created to represent two mining periods (i.e., Phase 2 Composite 1 representing ore from mining period Years 1 to 3. Phase 2 Composite 2 representing ore from mining period Years 4 to 6). Samples included select Salinas samples and other lithology samples that were tested in previous metallurgical testwork plans. For the initial six years of mining, Salinas ore will predominately originate from Salinas Zone 1. Optimization testing was conducted on Composites 1 and 2, evaluating the effect of primary grind sizing and sodium cyanide concentration on gold and silver extraction. The primary grind size was evaluated at $P_{80}$ of 53 µm and 75 µm. There was no statistically significant difference in gold leach performance between the grind sizes for both composites. Gold extraction was 94% and 92% for Composites 1 and 2, respectively. Gold extractions were completed within 24 hours. The addition of lead nitrate showed a small improvement in initial leach kinetics as the dosage was increased from none to 250 g/t. The results from the Phase 2 test results reconfirmed the selected leach-CIP flowsheet and optimized the design criteria for the process plant design.

Bulk tailings samples from Phase 2 Composites 1 and 2 at $P_{80}$ of 53 µm and 75 µm (four samples in total) from the BaseMet 2022 metallurgical testwork program were further tested by Metso:Outotec at SGS Lakefield. The testwork involved completing tailings thickening and filtration tests on the four samples. All tailings samples were successfully thickened to high densities and achieved minimum thickener underflow densities of 50%. All tailings samples were successfully filtered; Composite 1 samples achieved the target 16% moisture and Composite 2 samples will require longer drying times to achieve 16% moisture. There was no major difference between filtering 53 µm and 75 µm material; however, there was difference between Composite 1 and Composite 2 and may be mineralogy related. This has not been quantified but may be related to an increase of MVO lithology material in the mill feed.

Gold and silver recoveries for the Project were evaluated based on the whole of ore leach test results from the BaseMet test programs (2020b, 2021, 2022). In general, test results showed no difference in gold recoveries between the north and south zones. Gold recovery is more of a function of lithology and grades. The bulk of the metallurgical tests were completed on samples with a gold grade > 4.5 g/t Au and silver grade > 4.4 g/t Ag. For the FS-level metallurgical testwork program, lower grade Salinas ore and two composites representing the initial mining years and used together with the high-grade samples to derive recovery curves. A first order reaction formula was used to develop the equation using all high-grade results, results from Phase 2 (BaseMet 2022), Salinas samples less than 1 g/t Au and Salinas samples that produced greater than 80% recovery. This equation was used to estimate recovery for the first six years of mining. A separate recovery equation was developed for Salinas Zones 1 and 2 which is used together with the main equation for subsequent mining years. There is no recovery relationship with head grade and lithologies for silver. An average of the test results was used for silver recovery.

## 1.10 Mineral Resource Estimate & Methodology

### 1.10.1 Mineral Resource Estimate

This resource is based on a measured, indicated and inferred mineral resource estimate undertaken by Mr. Garth Kirkham, P. Geo., of Kirkham Geosystems Ltd. Mr. Kirkham is an independent qualified person as defined by N.I. 43-101.

Cerro Blanco is a classic hot springs-related, low-sulphidation epithermal gold-silver deposit comprising both high-grade vein and low-grade disseminated mineralization. Most of the high-grade mineralization is hosted in the Mita unit as two upward-flaring vein swarms (north and south zones) that converge downwards and merge into basal feeder veins where drilling has demonstrated widths of high-grade mineralization (e.g., 15.5 m 21.4 g Au/t and 52 g Ag/t). Bonanza gold grades are associated with ginguru banding and carbonate replacement textures. Sulphide contents are low, typically < 3% by volume. Low-grade disseminated and veinlet mineralization in wall rocks around the high-grade veins is well documented in drilling since discovery of the deposit, with grades typically ranging from 0.3 to 3.0 g/t Au.

The Mita rocks are overlain by the Salinas unit, a sub-horizontal sequence of volcanogenic sediments and sinter horizons approximately 100 m thick that form the low-lying hill at the Project. Low-grade disseminated and veinlet mineralization within and as halos around the high-grade vein swarms is well documented in drilling since discovery of the deposit, with grades typically ranging from 0.3 to 1.5 g Au/t. The overlying Salinas cap rocks are also host to low-grade mineralization associated with silicified conglomerates and rhyolite intrusion breccias.

In profile, the inverted wedge-shape of the high-grade veins (upward flaring arrays) and their low-grade halos overlain by the mineralized Salinas cap rocks to surface, render the deposit amenable to exploitation by surface methods with a low strip ratio.

### 1.10.2 Methodology

The mineral resource estimate reported herein was prepared by Mr. Garth Kirkham, P. Geo. The mineral resources have been estimated in conformity with generally accepted CIM "Estimation of Mineral Resource and Mineral Reserves Best Practices" and are reported in accordance with NI 43-101 guidelines. There are 130,307 gold assays or 153,078 m total which average 0.68 g/t and 130,238 silver assays or 153,003 m total which average 3.75 g/t. Specific gravities were assigned to individual rock types and assigned on a block-by-block basis using measurement data by lithology.

The estimate was completed using MineSight™ software using a 3D block model (5 m by 5 m by 5 m). Interpolation parameters have been derived based on geostatistical analyses conducted on 1.5 meter composited drill holes. Block grades have been estimated using ordinary kriging (OK) methodology and the mineral resources have been classified based on proximity to sample data and the continuity of mineralization in accordance with CIM's "Definition Standards and Estimation Best Practices".

There is 3.09 Moz of gold and 13.4 Moz silver contained in the measured and indicated mineral resources, along with of 0.031 Moz of gold and 0.112 Moz silver contained in the inferred mineral resources. The mineral resource estimate is reported at a base case above a 0.4 g Au/t cut-off, as tabulated in Table 1.2.

**Table 1.2: Mineral Resource Statement**

| Resource Category | Tonnes (kt) | Au Grade (g/t) | Ag Grade (g/t) | Contained Gold (koz) | Contained Silver (koz) |
|---|---|---|---|---|---|
| Measured | 40,947 | 1.8 | 7.9 | 2,382 | 10,387 |
| Indicated | 22,595 | 1.0 | 4.2 | 706 | 3,058 |
| **Measured & Indicated** | **63,542** | **1.5** | **6.6** | **3,089** | **13,445** |
| | | | | | |
| Inferred | 1,672 | 0.6 | 2.1 | 31 | 112 |
| **Below Pit (Indicated) *** | **189** | **5.7** | **13.4** | **35** | **82** |
| **Stockpile (Measured)** | **30** | **5.4** | **22.6** | **5** | **22** |

Notes: The mineral resource statement is subject to the following: (1) All mineral resources have been estimated in accordance with Canadian Institute of Mining and Metallurgy and Petroleum (CIM) definitions, as required under National Instrument 43-101 (N.I. 43-101), with an effective date of December 31, 2020. (2) Mineral resources reported demonstrate reasonable prospect of eventual economic extraction, as required under N.I. 43-101; mineral resources are not mineral reserves and do not have demonstrated economic viability. (3) * Underground mineral resources are reported at a cut-off grade of 3.5 g Au/t. Cut-off grades are based on a price of US$1,600/oz gold, US$20/oz silver and a number of operating cost and recovery assumptions, plus a contingency.
(4) Numbers arerounded. (5) The mineral resources may be affected by subsequent assessment of mining, environmental, processing, permitting, taxation, socio-economic and other factors. (6) An inferred mineral resource has a lower level of confidence than that applying to an indicated mineral resource and must not be converted to a mineral reserve. It is reasonably expected that the majority of inferred mineral resources could be upgraded to indicated mineral resources with continued exploration. (7) Mineral resources are inclusive of mineral reserves. Source: Kirkham (2021).

## 1.11 Mineral Reserves

Mineral reserves for the Cerro Blanco Gold Project are estimated at 53.9 Mt at an average grade of 1.64 g/t of gold for 2,846,000 ounces, and 7.27 g/t of silver for 12,602,000 ounces, as summarized in Table 1.3. The mineral reserve estimate (MRE) was prepared by G Mining Services Inc. (GMS).

**Table 1.3: Mineral Reserves**

| Reserve Category | Tonnage (kt) | Gold (g/t) | Gold (koz) | Silver (g/t) | Silver (koz) |
|---|---|---|---|---|---|
| Proven | 37,618 | 1.89 | 2,286 | 8.33 | 10,084 |
| Probable | 16,279 | 1.07 | 560 | 4.81 | 2,518 |
| **Proven & Probable** | **53,896** | **1.64** | **2,846** | **7.27** | **12,602** |

Notes: (1) CIM definitions were followed for mineral reserves. (2) Effective date of the estimate is Nov 1, 2021. (3) Mineral reserves are estimated at a cut-off grade of 0.50 g/t Au Eq. (4) Mineral reserves are estimated using the following long-term metal prices (Au = US$1,550/oz and Ag = US$20/oz). (5) Bulk density of ore is variable but averages 2.70 t/m3. (6) The average strip ratio is 2.7:1. (7) The average mining dilution factor is 6.7%. (8) Other costs and factors used for gold cut-off grade determination were process, G&A, and other costs of $21.17/tonne, a royalty of $31.60 /oz Au and gold and silver metallurgical recoveries of 91% and 85% respectively. (9) Tonnages are rounded to the nearest 1,000 tonnes; metal grades are rounded to two decimal places. 7) Tonnage and grade measurements are in metric units; contained gold and silver are reported as thousands of troy ounces 8) The mineral reserves may be affected by subsequent assessment of mining, environmental, processing, permitting, taxation and socio-economic factors 9) Mineral resources are inclusive of mineral reserves.

The mine design and MRE were completed to a level appropriate for feasibility studies. The MRE stated herein is consistent with CIM definitions and is suitable for public reporting. As such, the mineral reserves are based on measured and indicated mineral resources, and do not include any inferred mineral resources. The inferred mineral resources contained within the mine design are classified as waste.

## 1.12 Mining

The Cerro Blanco Gold Project will be mined by conventional open pit mining techniques using trucks and diesel-powered hydraulic excavators supported by production wheel loaders. Trucks will haul the ore to the primary crusher and to a run-of-mine (ROM) pad where it will be rehandled and processed through the concentrator.

Mining is planned with eight phases split between the main pit and a smaller satellite pit as shown in Figure 1-2 below. The objective of pit phasing is to improve the economics of the Project by feeding the mill with higher-grade material during the earlier years and/or delaying waste stripping until later years. Internal phases are designed to have a lower stripping ratio than the subsequent phases. The Cerro Blanco deposit is split into two separate pits: the main pit and the satellite pit. The main pit is split into seven phases; three are in the north end, three are in the south end, and one is the final pit that includes both the north and south ends. The satellite pit has one phase.

Drill and blast specifications are established to effectively drill and blast a 10 m bench with a single pass. Loading in the pit will be performed by two 15 $m^3$ face shovels and two 11 $m^3$ wheel loaders. The loading units will be matched with a fleet of 90-tonne payload capacity mine haul trucks.

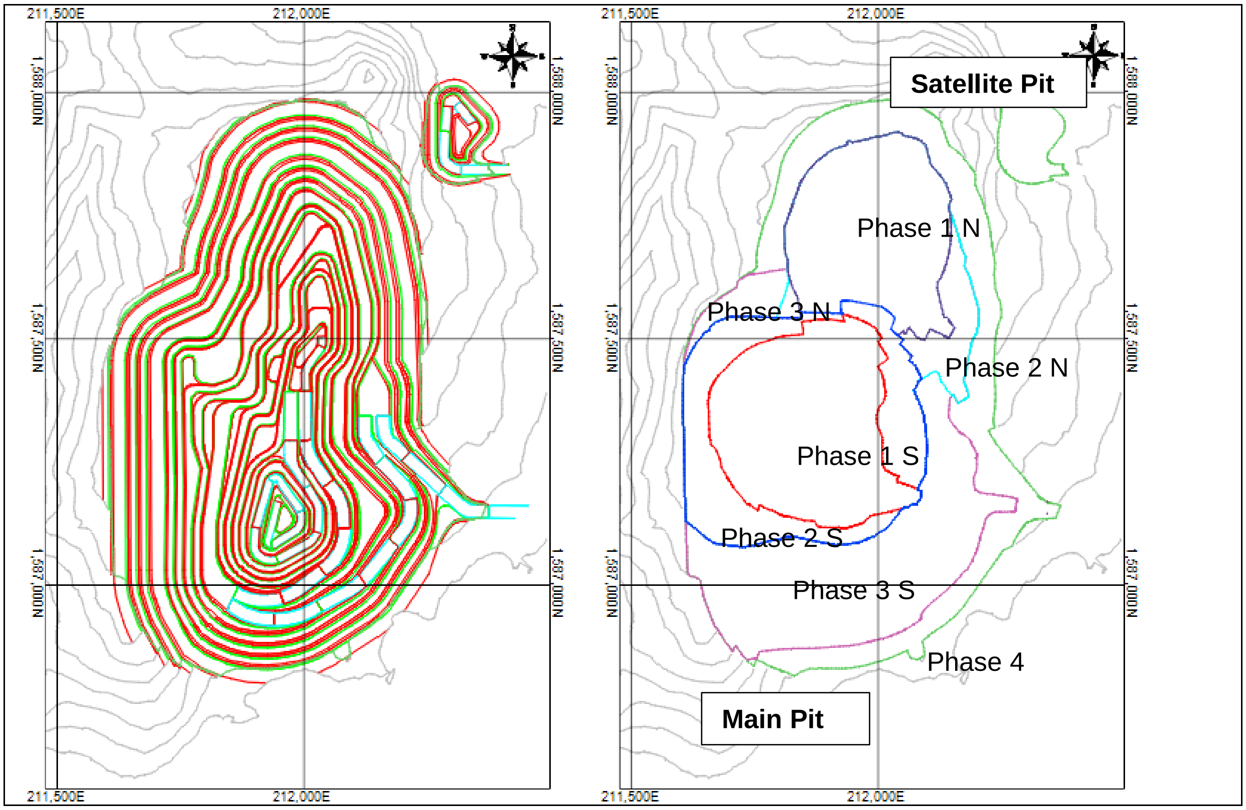
A strategic optimization exercise was carried out to optimize the mine production rate and mill throughput. This exercise resulted in an increased mine production rate and decreased mill throughput. Following this, an open pit optimization was conducted to determine the optimal economic shape of the open pit to guide the pit design process. This task was undertaken using the Whittle software which is based on the Lerchs-Grossmann algorithm.

Over the mine life, the Project will produce 53.9 Mt of mill feed and 145.4 Mt of waste at an overall stripping ratio of 2.70 (see Table 1.3). The total in-situ gold and silver ounces contained in the ultimate pit design are 2.85 Moz and 12.60 Moz, respectively, including the existing stockpile from the previous underground workings.

Mining activities for the Project take place over ~12 years. This includes a ramp-up period of two years; eight years of mining at peak capacity; and a two-year post-peak production ramp down. The peak mining rate is 21 Mt/a (57,500 t/d) at an average stripping ratio of 2.1:1 (waste to ore).

Figure 1-3 depicts the production schedule by material type and the strip ratio. Mill feed is consistent throughout the mine life with no periods of significant stripping required to meet mill requirements. Mine and mill production details are provided in Table 1.3 and Table 1.4, respectively.

**Figure 1-2: End-of-Mine Pit Layout & Phase Limits**




Notes: End of mine life (left); phase limits (right). Source: GMS (2021).

**Figure 1-3: Mine Production Schedule**


Source: GMS (2021).

**Table 1.3: Detailed Mine Production**

| Description | Unit | Total | Y-2 | Y-1 | Y1 | Y2 | Y3 | Y4 | Y5 | Y6 | Y7 | Y8 | Y9 | Y10 | Y11 | Y12 | Y13 | Y14 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Total Tonnage | Mt | 199.3 | 1.7 | 17.1 | 21.0 | 21.0 | 21.0 | 21.0 | 20.6 | 20.0 | 19.6 | 18.4 | 13.3 | 4.6 | 0.0 | 0.0 | 0.0 | 0.0 |
| Total Waste | Mt | 145.4 | 1.6 | 14.0 | 13.8 | 14.4 | 14.3 | 16.2 | 16.4 | 14.8 | 14.4 | 12.9 | 9.3 | 3.2 | 0.0 | 0.0 | 0.0 | 0.0 |
| Strip Ratio | W:O | 2.7 | 50.7 | 4.4 | 1.9 | 2.2 | 2.1 | 3.3 | 3.9 | 2.8 | 2.8 | 2.4 | 2.4 | 2.4 | 0.0 | 0.0 | 0.0 | 0.0 |
| Ore Tonnage | Mt | 53.9 | 0.0 | 3.2 | 7.2 | 6.6 | 6.7 | 4.8 | 4.2 | 5.2 | 5.2 | 5.5 | 4.0 | 1.4 | 0.0 | 0.0 | 0.0 | 0.0 |
| Gold Grade | g/t | 1.64 | 0.0 | 1.0 | 2.9 | 2.0 | 2.8 | 2.2 | 1.1 | 1.5 | 1.8 | 2.1 | 2.0 | 1.7 | 0.6 | 0.6 | 0.6 | 0.6 |
| Silver Grade | g/t | 7.3 | 0.0 | 4.5 | 14.6 | 11.8 | 13.9 | 9.8 | 6.3 | 7.4 | 6.1 | 5.3 | 5.0 | 5.0 | 3.7 | 3.7 | 3.7 | 3.7 |

Source: GMS (2021).

**Table 1.4: Detailed Milling Schedule**

| Description | Unit | Total | Y-2 | Y-1 | Y1 | Y2 | Y3 | Y4 | Y5 | Y6 | Y7 | Y8 | Y9 | Y10 | Y11 | Y12 | Y13 | Y14 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Ore Milled | Mt | 54 | - | 1 | 4 | 4 | 4 | 4 | 4 | 4 | 4 | 4 | 4 | 4 | 4 | 4 | 4 | 1 |
| Gold Grade | g/t | 1.64 | 0 | 1.04 | 2.87 | 1.95 | 2.83 | 2.16 | 1.14 | 1.52 | 1.75 | 2.07 | 2.04 | 1.73 | 0.56 | 0.56 | 0.56 | 0.56 |
| Silver Grade | g/t | 7.27 | 0 | 4.51 | 14.55 | 11.75 | 13.88 | 9.81 | 6.29 | 7.36 | 6.08 | 5.32 | 5.02 | 4.96 | 3.66 | 3.66 | 3.66 | 3.66 |
| Gold Recovery | koz | 2,645 | 0 | 29 | 347 | 235 | 344 | 261 | 135 | 183 | 209 | 248 | 245 | 207 | 63 | 63 | 63 | 12 |
| Silver Recovery | koz | 10,617 | 0 | 133 | 1,580 | 1,271 | 1,512 | 1,064 | 674 | 798 | 665 | 579 | 547 | 538 | 395 | 394 | 394 | 74 |

Source: GMS (2021).

## 1.13 Recovery Methods

The process plant design for the Project is based on a robust metallurgical flowsheet to treat gold- and silver-bearing ore to produce doré. The flowsheet is based on metallurgical testwork, industry standards and conventional unit operations.

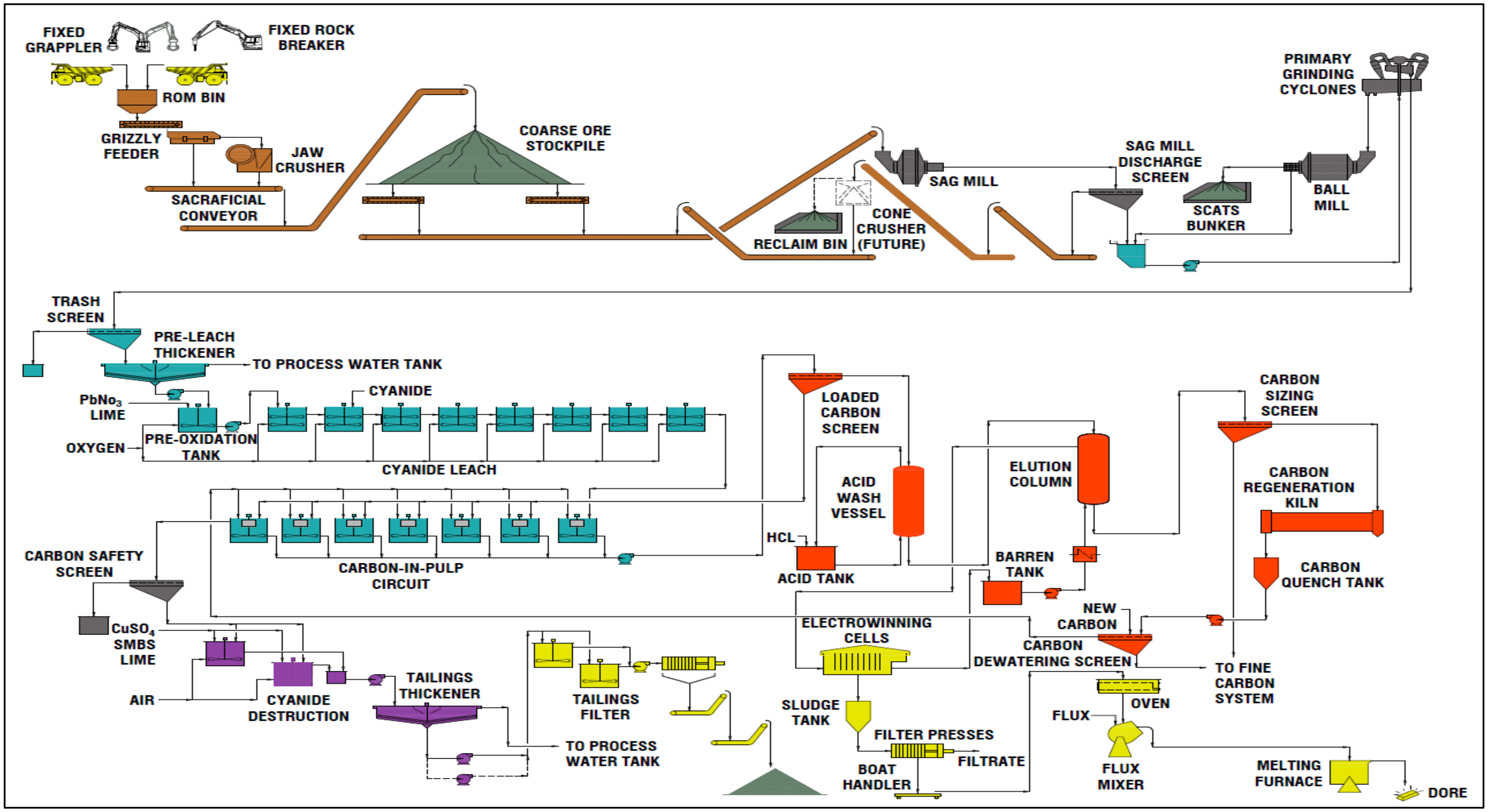
The process plant will treat 4.0 million tonnes per year (Mt/a) of ore and will consist of comminution, cyanide leach, carbon adsorption via carbon-in-pulp (CIP), carbon elution and gold recovery circuits. CIP tailings will be treated in a cyanide destruction circuit and dewatered to produce a filtered tailings product.

The key Project design criteria for the process plant are listed below:

- nominal throughput of 4.0 Mt/a of ore
- crushing plant availability of 75%
- grinding, leach, gold recovery availability of 92% and tailings filtering circuit availability of 85% through the use of standby equipment in critical areas, inline crushed ore stockpile and reliable power supply
- comminution circuit to produce a particle size of 80% passing ($P_{80}$) 53 µm (design)
- leach residence time of 36 hours to achieve optimal gold and silver extraction
- cyanide destruction circuit to produce weak acid dissociable (WAD) cyanide levels of less than 10 ppm
- sufficient process plant control to minimize the need for continuous operator interface and to allow for manual override and control if and when required
- equipment selection based on suitability for the required duty, reliability, and ease of maintenance
- plant layout that provides ease of access to all equipment for operating and maintainability, while facilitating concurrent construction activities in multiple areas of the plant.

An overall process flow diagram depicting the various unit operations is presented in Figure 1-4.

**Figure 1-4: Overall Process Flow Diagram**




Source: GMS (2021).

## 1.14 Infrastructure

The proposed site layout has been designed to minimize environmental and social impacts, provide security-controlled site access, minimize construction costs, and optimize operational efficiency. Primary buildings have been located to allow easy access for construction and to utilize existing topography to minimize bulk earthworks volumes.

The Cerro Blanco Gold Project infrastructure is designed to support a nominal 4 Mt/a processing plant with an annual mining rate of 21 Mt/a that is operating 24 hours per day, 7 days per week. The Project infrastructure is also designed for local conditions and topography.

The main infrastructure items include the following:

- 5 km new site access road (including a 110 m long bridge)
- new 138 kV powerline
- on-site 138 kV to 13.8 kV substation
- 500-person permanent camp with kitchen and laundry
- surface water management facilities including diversion channels, ditches, and collection ponds
- process plant site pad and buildings
- administration and mine office building
- warehouse
- assay and metallurgical laboratory
- primary crusher pad
- emergency power generator(s)
- one dewatering well (seven future wells deferred to sustaining capital)
- reagent warehouse and cyanide storage pad
- mine service center
- mine maintenance facility and wash bay
- fresh and fire water tanks
- process water tank
- fuel and gasoline storage and distribution bay
- contact water treatment plant
- sewage treatment plant
- domestic water treatment plant

- solid waste disposal facilities (including topsoil deposit, temporary waste storage pad and overburden deposit)
- dry stack tailings facility (DSTF)
- waste rock storage facility
- mine haul roads
- on-site access roads for plant and facilities
- additional security facilities including site access control station

The open pit is considered the reference point of the Project with the DSTF positioned on its east side. The process plant is located on the northwest side of the DSTF, and the primary crusher is located to the northeast of the process plant.

The service mining facilities (mine service center, maintenance facility, wash, and fuel bay) are north of the process plant and allow easy access for heavy equipment. A traffic management plan governs light-vehicle circulation in this sector and facilitates entry to the administration area, change room, and office building. The laboratory and warehouse are also located in this area.

Most of the existing infrastructure is within the blast radius and will be decommissioned. Existing electrical infrastructures will initially be used for temporary power.

## 1.15 Environment & Community

### 1.15.1 Permitting

The approved ESIA and permits allow the Project to proceed with underground mine development and construction of the process facilities, provided future operations adhere to the existing permit requirements. Since the design has been updated and optimized to be developed as an open pit operation, the ESIA has been amended and submitted for approval with the updated Project design. Table 1.5 provides a summary of main permit amendments, new permits required, and their status. The Cerro Blanco permit register indicates all applicable permit commitments have been fulfilled to date.

**Table 1.5: Main Permit Amendments & New Permit Required**

| Project Component | Action Required |
|---|---|
| Water Management (including Treatment Plant) | ESIA Amendment – In evaluation process in the Ministry |
| Increase of exploitation rate (Change of Mining Method): Open Pit, Processing Plant and Waste Rock Management | ESIA Amendment – In evaluation process in the Ministry |
| DSTF New Location | ESIA Amendment – In evaluation process in the Ministry |
| New Access Road | ESIA Amendment – In evaluation process in the Ministry |
| New Bridge | New EIA – Environmental License granted |
| New Powerline | New EIA – In preparation |

Source: Bluestone (2021).

### 1.15.2 Social Aspects & Stakeholder Engagement

Bluestone is committed to developing the Cerro Blanco Gold Project with best international practices in line with its corporate sustainability and transparency objectives. In 2021, Bluestone completed an update to the Economic and Social Baseline Study to provide a current socio-economic context for the Project. The Social Baseline Study (SBS) has been updated and key information has been included in the amended ESIA recently submitted to the authorities for approval. The SBS update included the most recent primary data obtained in each household, along with face-to-face meetings with representatives of local organizations and communities. In addition, social management plans, stakeholder engagements and grievance mechanism procedures were reviewed and updated based on international best practices and the relevant IFC Performance Standards.

In 2021, Bluestone developed and initiated the implementation of a new social management system that includes a database to register all community engagements, activities, and key information related to the community relationships. The social management system is used to track and monitor performance against key Community Social Responsibility (CSR) objectives.

### 1.15.3 Rehabilitation & Closure

The approved ESIA (2007) includes a conceptual mine closure and reclamation plan. The recently submitted permit amendment also includes a conceptual closure plan. Under Guatemalan regulations, a mine closure plan does not need to be submitted to the authorities until three years before the projected mine closure.

## 1.16 Capital & Operating Cost Estimates

### 1.16.1 Capital Cost Estimate

Table 1.6 presents the capital estimate summary for the pre-production phase and sustaining capital costs in US dollars with no escalation or VAT included.

Life-of-mine project capital costs are estimated at $749.9 million over three distinct phases, as follows:

- Pre-production Capital Costs – This phase includes all costs to develop the property to an average of 11,747 kt/d mill production rate. Initial capital costs total $571.5 million (including $60.7 million for contingency and $47.5 million in pre-production revenue), which will be expended over a 25-month pre-production design, construction, and commissioning period.
- Sustaining Capital Costs – This phase includes all costs related to the acquisition, replacement, or major overhaul of assets during the mine life required to sustain operations. Sustaining capital costs are estimated to be $140.3 million and do not include contingency. Sustaining costs are expended in operating Years 1 through 14.
- Closure Costs – This phase includes all costs related to the closure, reclamation, and ongoing monitoring of the mine after operations. Closure costs total $38.1 million and do not include contingency. Closure costs are all incurred in Years 14 through 18.

**Table 1.6: Capital Cost Summary**

| WBS Area | WBS Description | Pre-Production Cost (US$M) | Sustaining/Closure Cost (US$M) | Project Total Cost (US$M) |
|---|---|---|---|---|
| **100** | **Infrastructure** | **39.6** | **7.0** | **46.6** |
| 110 | Roads, Bridges & Fencing | 6.9 | 1.9 | 8.8 |
| 120 | Mine Infrastructure | 7.3 | 1.3 | 8.7 |
| 130 | Support Infrastructure | 8.3 | - | 8.3 |
| 140 | Camp Facilities | 7.2 | - | 7.2 |
| 160 | Process Plant Infrastructure | 6.8 | - | 6.8 |
| 170 | Fuel Systems Storage | 0.9 | - | 0.9 |
| 180 | Stockpile Pads | 2.1 | 3.8 | 5.9 |
| **200** | **Power & Electrical** | **38.8** | **0.3** | **39.1** |
| 210 | Main Power Generation | 10.8 | - | 10.8 |
| 220 | Secondary Power Generation | 1.5 | - | 1.5 |
| 230 | Water Management Electrical Room | 1.5 | 0.2 | 1.7 |
| 240 | Service Electrical Room | 0.1 | - | 0.1 |
| 260 | Process Plant Electrical Rooms | 18.8 | - | 18.8 |
| 270 | Overhead Distribution Line | 1.9 | 0.1 | 2.1 |
| 280 | Automation Network | 1.9 | - | 1.9 |
| 290 | IT Network & Fire Detection | 2 | - | 2 |
| 202 | Electrical Duct Bank | 0.2 | - | 0.2 |
| **300** | **Water Management** | **52** | **39.9** | **91.9** |
| 310 | Fresh water Intake / Wells | 0.3 | - | 0.3 |
| 320 | Water Ponds & Water Management | 13.1 | 5.3 | 18.4 |
| 330 | Domestic Water (Cost Code Account) | 0.8 | - | 0.8 |
| 340 | Sewage (Cost Code Account) | 1.9 | - | 1.9 |
| 350 | Fire Protection (Cost Code Account) | 3.6 | - | 3.6 |
| 360 | Effluent Water Treatment | 24.2 | 16.8 | 41 |
| 370 | Dry Stack Tailings Facility | 1.1 | 3 | 4 |
| 380 | Dewatering & Injection Wells | 7 | 14.9 | 21.9 |
| **400** | **Surface Operations** | **14.4** | **-** | **14.4** |
| 410 | Surface Operation Equipment | 11.5 | - | 11.5 |
| 430 | Concrete Batch Plant | 0.1 | - | 0.1 |
| 480 | Aggregate Plant | 2.8 | - | 2.8 |
| **500** | **Mining** | **42.3** | **40.3** | **82.6** |
| 540 | Mine Infrastructure | 0.5 | 3.9 | 4.4 |
| 550 | Mine Equipment (129) | 41.8 | 36.4 | 78.2 |
| **600** | **Process Plant** | **136.9** | **-** | **136.9** |
| 602 | Pipe Rack | 2.2 | - | 2.2 |
| 603 | Underground Services | 0.2 | - | 0.2 |
| 604 | Run-of-Mine Pad & Mechanical Stabilized Earth (MSE) Wall | 1.3 | - | 1.3 |
| 605 | Final Grading | 0.3 | - | 0.3 |
| 610 | Comminution | 53.2 | - | 53.2 |
| 640 | Lixiviation | 29.5 | - | 29.5 |
| 650 | Reagents | 3.6 | - | 3.6 |
| 660 | Refinery | 9.7 | - | 9.7 |
| 680 | Tailings Management | 29.6 | - | 29.6 |
| 690 | Process Plant Services | 7.4 | - | 7.4 |
| **700** | **Construction Indirect** | **66.3** | **-** | **66.3** |
| 710 | Engineering, Construction Management, Project Management | 20.1 | - | 20.1 |
| 720 | Construction Offices, Facilities & Services | 1.7 | - | 1.7 |
| 730 | Shops | 0.7 | - | 0.7 |
| 740 | Construction Equipment & Tools | 15.1 | - | 15.1 |
| 760 | Energy | 10.6 | - | 10.6 |
| 790 | External Engineering | 18 | - | 18 |
| **800** | **General Services – Owner's Costs** | **77.8** | **-** | **77.8** |
| 810 | G&A Departments | 21.3 | - | 21.3 |
| 820 | Logistics / Taxes / Insurance | 28.3 | - | 28.3 |
| 830 | Operating Expenses | 15.1 | - | 15.1 |
| 840 | Environmental | 6.4 | - | 6.4 |
| 850 | Health & Safety | 3.2 | - | 3.2 |
| 860 | Site Insurance | 3.4 | - | 3.4 |
| **900** | **Pre-Production, Start-up, Commissioning** | **90.2** | **52.9** | **143** |
| 910 | Mining Pre-Production | 52.6 | 52.9 | 105.5 |
| 920 | DSTF Preprod | 1.8 | - | 1.8 |
| 950 | Process Plant Pre-Production | 29.2 | - | 29.2 |
| 960 | First Fill, Spares & Consumables (8% of Process Plant Mechanical Equipment) | 6.6 | - | 6.6 |
| **955** | **Pro-production Revenue** | **-47.5** | **-** | **-47.5** |
| **991** | **Project Contingency** | **60.7** | **-** | **60.7** |
| | Closure Costs | - | 38.1 | 38.1 |
| | Project Total Cost | 571.5 | 178.4 | 749.9 |

Source: GMS (2021).

The project capital cost estimate includes all costs to develop and sustain the project at a commercially operable status. The estimate does not include costs related to operating consumables inventory purchased before commercial production; these costs are considered within the working capital estimate. Sunk Costs and Owners Reserve accounts are not considered in the FS estimates or economic cash flows.

### 1.16.2 Operating Cost Estimate

A summary of the operating cost estimate is provided in Table 1.7. The estimate includes the costs to mine and process the mineralized material to produce doré, along with site services to maintain the site, and general and administrative (G&A) expenses. The costs are expressed in U.S. dollars with no allowance for inflation. The target accuracy of the operating cost estimate is -10/+15%.

Operating costs are estimated to be US$29.56/t processed.

**Table 1.7: Summary of Operating Cost Estimate**

| Operating Costs | $/t Milled | Life of Mine ($M) |
|---|---|---|
| Mining | 10.42 | 550 |
| Processing | 12.97 | 685 |
| Site Services | 2.73 | 144 |
| G&A | 3.43 | 181 |
| **Total** | **29.56** | **1,560** |

Source: GMS (2021).

## 1.17 Economic Analysis

An economic model was developed to estimate annual cash flows and project sensitivities. Pre-tax estimates of project values were prepared for comparative purposes, while after-tax estimates were developed and are likely to approximate the true investment value. It must be noted, however, that tax estimates involve many complex variables that can only be accurately calculated during operations and, as such, the after-tax results are only approximations.

Sensitivity analyses were performed for variations in metal prices, head grades, operating costs, capital costs, and discount rates to determine their relative importance as project value drivers.

This technical report contains forward-looking information regarding projected mine production rates, construction schedules, and forecasts of resulting cash flows. The mill head grades are based on sufficient sampling that is reasonably expected to be representative of the realized grades from actual mining operations. Factors such as the ability to obtain or amend permits to construct and operate a mine, or to obtain major equipment or skilled labour on a timely basis to achieve the assumed mine production rates at the assumed grades, may cause actual results to differ from those presented in this economic analysis.

A summary of the mine plan and payable metals produced is provided in Table 1.8. Other economic factors are listed below.

- A discount rate of 5% is applied.
- The analysis is performed in nominal 2021 dollars.
- Revenues, costs, and taxes are calculated for each period in which they occur rather than actual outgoing/incoming payment.
- Working capital is calculated as three months of consumables, one week of plant inventory, and two weeks of accounts receivable.
- Results are presented based on 100% ownership.
- No management fees or financing costs (equity fund-raising was assumed) are applied.
- The model excludes all pre-development and sunk costs (i.e., exploration and resource definition costs, engineering fieldwork and studies costs, environmental baseline studies costs, financing costs, etc.).

**Table 1.8: Life-of-Mine Summary**

| Parameter | Unit | Value |
|---|---|---|
| Total Material Mined | Mt | 199.3 |
| Ore Processed | Mt | 53.9 |
| Strip Ratio | waste:ore | 2.7 |
| Mill Average Daily Production | kt | 11,747 |
| Mill Average Annual Production | Mt/a | 4.00 |
| Average Gold Mill Grade | g/t | 1.64 |
| Average Silver Mill Grade | g/t | 7.27 |
| Gold Contained | koz | 2,846 |
| Silver Contained | koz | 12,602 |
| Gold Recovered | koz | 2,645 |
| Silver Recovered | koz | 10,617 |
| Gold Recovery | % | 93% |
| Silver Recovery | % | 84% |
| Peak Gold Production | koz/year | 347 |
| Average Gold Production Years 1-4 | koz/year | 297 |
| Average Silver Production Years 1-4 | koz/year | 1,356 |
| Initial Capital Cost | US$M | 572 |
| Sustaining Capital Cost | US$M | 140 |
| Closure Capital Cost | US$M | 38 |
| Total Life of Mine Capital | US$M | 750 |

Source: GMS (2022).

Table 1.9 outlines the metal prices and exchange rate assumptions used in the economic analysis. The gold and silver price is in line with recently released comparable technical reports.

The reader is cautioned that the metal prices and exchange rates used in this study are only estimates based on recent historical performance. There is no guarantee that they will be realized if the project is taken into production. The metal prices are based on many complex factors and there are no reliable long-term predictive tools.

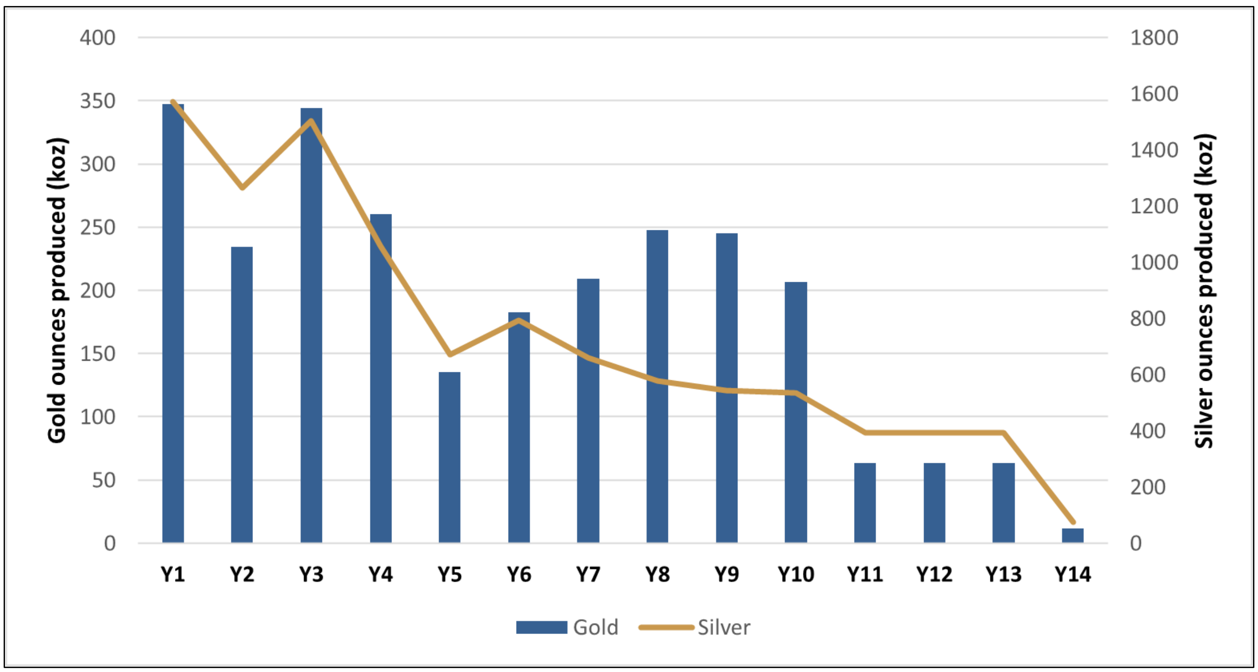
**Table 1.9: Metal Prices & Exchange Rates**

| Assumptions | Unit | Value |
|---|---|---|
| Gold Price | US$/oz | 1,600 |
| Silver Price | US$/oz | 20.00 |
| FX Rate | GTQ:USD | 7.69 |
| | CAD:USD | 1.32 |

Source: GMS (2021).

Figure 1-5 shows the grade and the amount of gold and silver recovered during the mine life. A total of 2,645 koz of gold and 10,617 koz of silver is projected to be produced over the life of mine. Gross project revenues are divided into 95% and 5% for gold and silver, respectively.

**Figure 1-5: Life-of-Mine Payable Gold & Silver**




Source: GMS (2022).

The Project has a post-tax IRR of 30% and a net present value using a 5% discount rate (NPV5%) of $1,047 million using the metal prices described in Table 1.10 summarizes the economic results.

The life-of-mine, all-in sustaining costs (AISC) and AISC (net of by-product) are US$709/oz and US$629/oz, respectively. The straight AISC cost is calculated by adding the refining, transport, royalty, operating, and sustaining and closure costs together and dividing by total payable ounces of gold. This calculation does not consider the value of silver. The AISC (net of by-product) is a similar calculation. It adds the refining, transportation, royalty, operating and sustaining and closure costs, but subtracts the value of the silver before dividing by total payable ounces of gold.

**Table 1.10: Summary of Results**

| Parameter | Unit | Result |
|---|---|---|
| AISC* | US$/oz | 709 |
| AISC (Net of By-product)** | US$/oz | 629 |
| **Capital Costs** | | |
| Pre-Production Capital | $M | 619.0 |
| Pre-Production Contingency | $M | 60.7 |
| Pre-Production Revenue | $M | 47.5 |
| Sustaining and Closure Capital | $M | 178.4 |
| Sustaining and Closure Contingency | $M | 0 |
| Total Sustaining and Closure Capital | $M | 178.4 |
| Working Capital | $M | 32.2 |
| Pre-tax Cash Flow | LOM $M | 2,571 |
| | $M/a | 185.1 |
| Taxes | LOM $M | 286.3 |
| After-Tax Cash Flow | LOM $M | 2,349 |
| | $M/a | 169.1 |
| **Economic Results** | | |
| Pre-Tax NPV 5% | US$M | 1,265.0 |
| Pre-Tax IRR | % | 37.30% |
| Pre-Tax Payback | years | 1.7 |
| After-Tax NPV 5% | US$M | 1,047 |
| After-Tax IRR | % | 30.30% |
| **After-Tax Payback** | **years** | **2.2** |

Notes: *All-in sustaining cost is calculated as (refining & shipping costs + royalties+ operating costs + sustaining and closure capital)/payable gold ounces. **All-in sustaining cost (net of by-product) is calculated as (refining & shipping costs + royalties+ operating costs + sustaining capital + closure capital - payable silver value) / payable gold ounces. Source: GMS (2022).

A univariate sensitivity analysis was performed to examine which factors most affect the project economics when acting independently of all other cost and revenue factors. Each variable evaluated was tested using the same percentage range of variation, from -25% to +25%, although some variables may actually experience significantly larger or smaller percentage fluctuations over the life of mine. For instance, the metal prices were evaluated at a ±25% range to the base case, while the recovery and all other variables remained constant. This may not be truly representative of market scenarios, as metal prices may not fluctuate in a similar trend. The variables examined in this analysis are those commonly considered in similar studies - their selection for examination does not reflect any particular uncertainty.

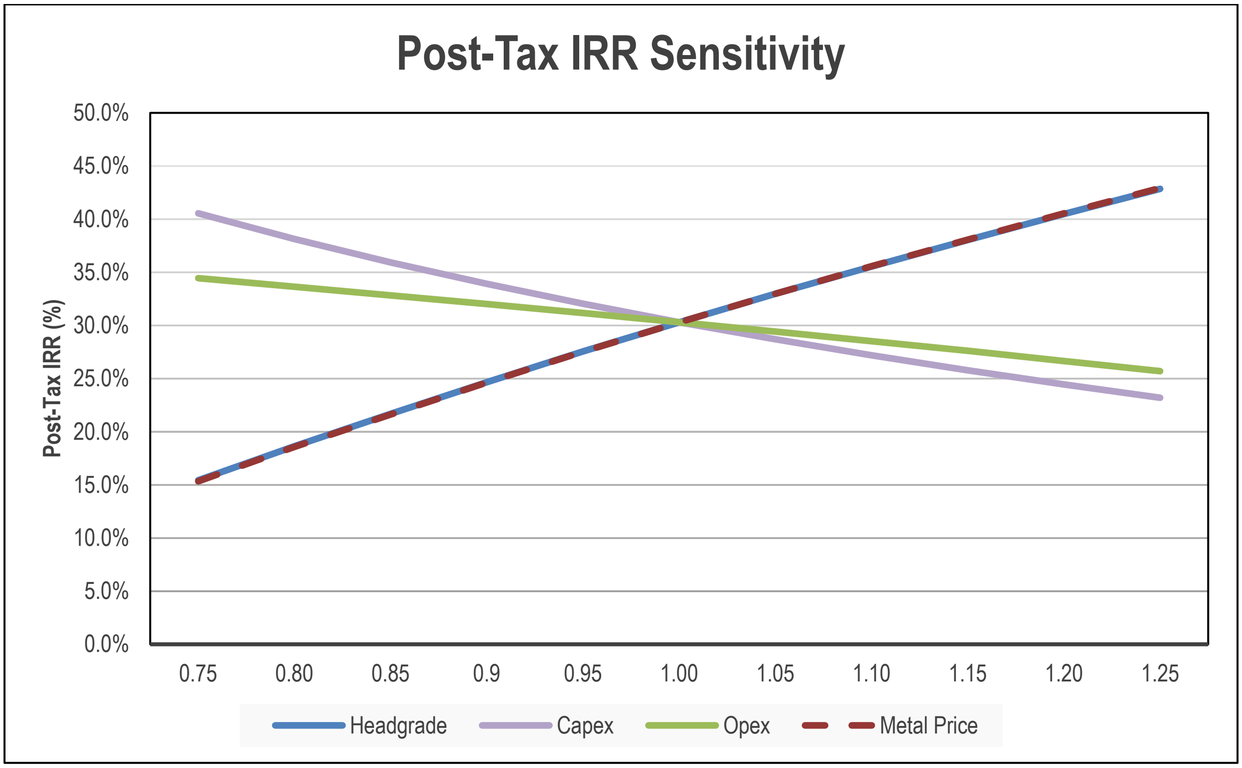
Notwithstanding the above limitations to the sensitivity analysis, which are common to studies of this nature, the analysis revealed that the project is most sensitive to metal prices and head grade. The project showed the least sensitivity to capital costs. Table 1.11 and Figure 1-6 show the results of the sensitivity tests.

**Table 1.11: Pre-Tax & After-Tax Sensitivity Results on NPV @ 5%**

| Variable | After-Tax NPV5% ($M) | | |
|---|---|---|---|
| | -25% Variance | 0% Variance | 25% Variance |
| Metal Price | $355 | $1,047 | $1,739 |
| Mill Head Grade | $359 | $1,047 | $1,735 |
| Operating Cost | $1,297 | $1,047 | $788 |
| Capital Cost | $1,222 | $1,047 | $872 |

Source: GMS (2022).

**Figure 1-6: After-Tax IRR Sensitivity**



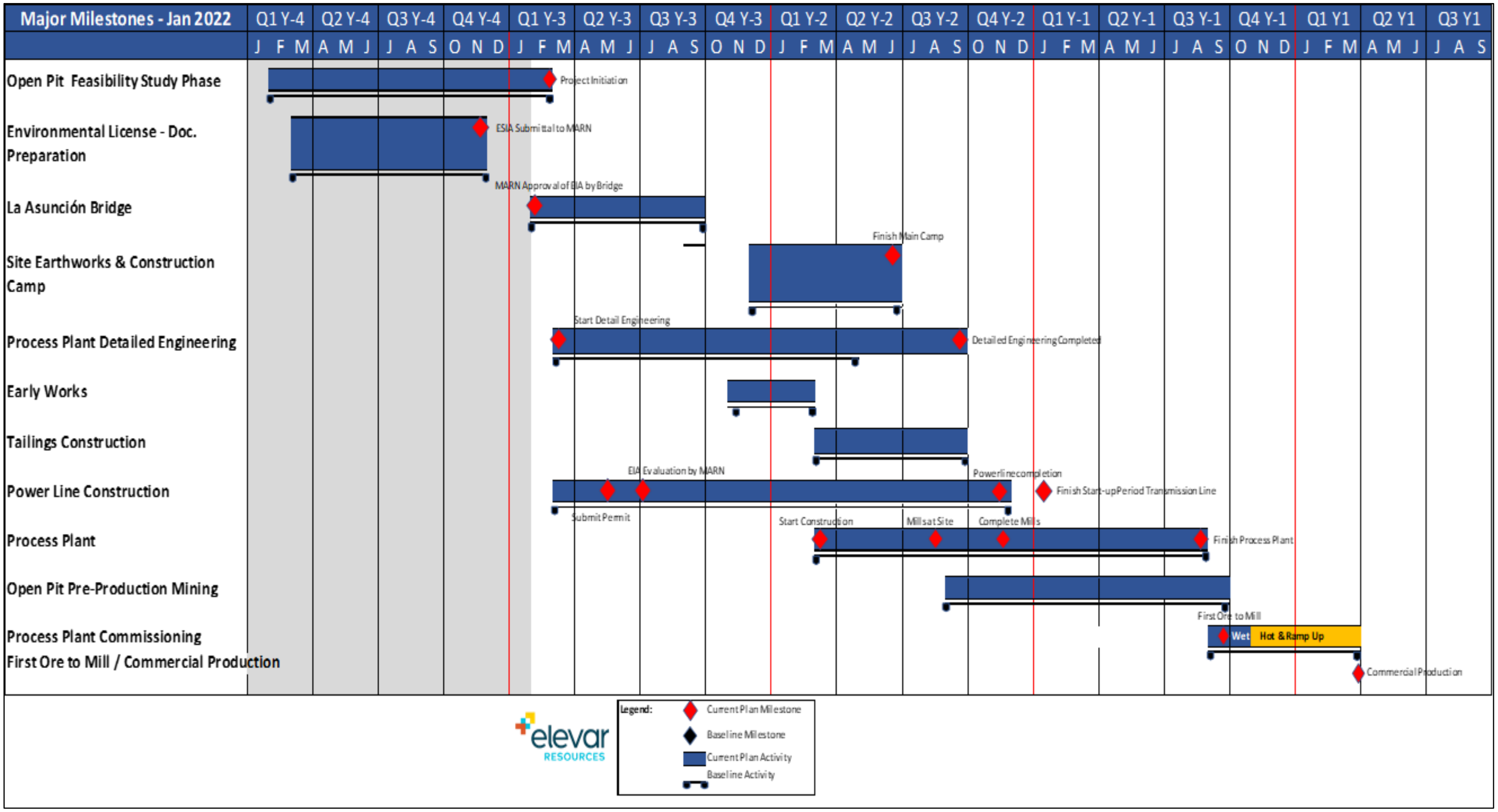

Source: GMS (2022).

## 1.18 Project Development

The integrated project management team (IPMT) will be created to lead the execution of the Project using a self-perform approach (Owner personnel and a contracted project management team). The plan is for the IPMT to lead the project execution and construction of all on-site infrastructure and the process plant. Mine development will also be self-performed by the Elevar mine team. Off-site infrastructure, including the access road, bridge and powerline, will be built by a contractor under the supervision of the IPMT.

A Level 1 schedule is shown in Figure 1-7. The overall construction and commissioning period for the project is estimated to be approximately 20 months to commissioning followed by 5 months of ramp-up to commercial production reached.

**Figure 1-7: Level 1 Schedule**




Source: GMS (2021)

## 1.19 Conclusions

The Cerro Blanco Gold Project has been optimized and updated as demonstrated by the results and findings provided in this technical report. This FS shows the results of the project for the current long-term gold price, exchange rate, final pit design and reserves, life-of-mine production schedule, process plant, infrastructure and support facilities design and execution.

This N.I. 43-101 Technical Report reconfirms the technical feasibility and economic viability of the Project based on an open pit mining operation with a life of mine production of 2.6 Moz of gold over an initial 14-year life of mine, it is recommended to advance the Project to the execution phase.

## 1.20 Risks & Opportunities

### 1.20.1 Risks

The most significant project risks are summarized below:

- Permitting – While the project is fully permitted and has an approved EIA in place, permit amendments are required for some of the proposed modifications, including the change to an open pit operation, increase in processing rate and discharge flow to the environment. New EIAs and permits are also needed for the powerline and access road. Although several of these amendments have been submitted, potential delays in approval of permit amendments and/or new permits could result in increased duration of assumed project development schedule.
- Powerline Right of Ways – Negotiation for the powerline ROW is set to begin and until all right of ways have been acquired, this remains a risk to the timely completion of the powerline.
- Power Availability – The availability of power from the national grid and ability of the National utilities and the transmission line contractor to deliver it on time needs to be confirmed.
- Power Cost – The cost of grid power is based on a market projection and not a power supply agreement. A higher power cost would result in increased operating costs.
- Site Land Negotiations – There is a potential for schedule delays due to finalization of negotiated purchases of land and/or rights-of-way for areas impacted by the expanded footprint of the operation.
- COVID-19 – The pandemic may impact the project schedule and cost (restrictions to get to site, availability of personnel, permitting process, procurement delays, equipment costs).
- Commodity Prices (Au, Ag) – Lower commodity prices will change size and grade of the potential targets. Conversely, increased commodity prices will improve economics and resources.
- Groundwater – It has been assumed that groundwater dewatering would be achieved using horizontal drains. Pit inflows and dewatering deep wells flows have been estimated based on an understanding of the groundwater pathways gained during the 2018 Feasibility Study. Further testwork and modelling is required to better understand the groundwater flows and optimize the dewatering approach. Increases in the actual amount of groundwater encountered would impact development costs and/or

the schedule. Drilling for drainage, and operational definition drilling included in the mine plan, will help to identify specific water-bearing zones with higher-than-expected flows to establish control and/or management procedures. Initiating well development earlier in the mine life should allow time to better understand mine dewatering requirements.

- Project Schedule – Project components are tightly coupled within the schedule; many must be completed without delay to maintain the desired development targets.
- Inflation – The current market conditions are experiencing abnormal inflation. If the trend continues costs may be increased beyond budgetary predictions.
- Local Procurement – The obligation to purchase in country may inflate project costs.
- Socio-Political Risk –There is a potential risk of socio-political opposition in the local communities as well as at the national level, which could adversely impact the project schedule.

### 1.20.2 Opportunities

The main opportunities identified for the project are listed below:

- Mineral Resources – There is the potential for an increase in mineral resources with increased exploration drilling along trend of the deposit.
- Grade Control – As with most vein deposits, effective grade control and proper mining execution to maintain minimal unplanned dilution will minimize potential impacts on grade, throughput, and operating costs. RC definition drilling ahead of blasting will improve the definition of grade boundaries between high-grade veins and low-grade disseminated ore. In combination with geological mapping and sampling at the bench, definition drilling will help minimize unplanned dilution and negative impacts during mining.
- Acid Rock Drainage and Metal Leaching – Developing a waste rock facility design and PAG material handling plan could reduce the acidity and concentration of metals released from the waste rock, thereby reducing treatment costs and environmental footprint. The benefits of such a waste rock facility design will be evaluated.
- Cyanide Destruction – The current process uses the addition of sodium metabisulphite as a source of sulphur dioxide ($SO_2$) to destruction cyanide. This process results in high concentrations of sulphate in the process effluents, which creates the need for a costly sulphate removal system prior to discharge to the environment. It is recommended to investigate alternative cyanide destruction options to minimize the introduction of sulphate in the effluent and eliminate a costly removal system.
- Nitrogen Treatment – The current process assumes a costly nitrogen treatment; a trade-off study should be conducted to examine passive treatment through wetlands.
- Administrative Services – Relocating administrative services in Asuncion Mita would reduce the number of offices required on site.
- Infrastructure – The next phase will allow the Project to explore more local procurement opportunities which may reduce costs and help build relationships.

## 1.21 Recommendations

At a gold price assumption of $1,600/oz, the results of this FS demonstrate that the Cerro Blanco Gold Project warrants advancement due to its positive economics and strong cash flows.

Additional engineering work and mine planning, as well as continuous engagement with local stakeholders, have all improved the understanding of the project and reduced its risk profile.

The path forward for the Project is to obtain all the required permits and licenses and advance detailed engineering for an efficient project construction mobilization.

Refer to Section 26 for detailed project recommendations.

# 2. INTRODUCTION

Bluestone Resources Inc. (Bluestone and/or the Company) is a mineral exploration and development company that is focused on advancing its 100%-owned Cerro Blanco Gold and Mita Geothermal projects located in southeast Guatemala approximately 160 kilometers by road from the capital, Guatemala City. The Company is primary focused on the development of the Cerro Blanco Gold Project that is located in southeastern Guatemala. Elevar Resources S.A., the Company's wholly owned subsidiary, is the 100% owner of the Cerro Blanco Gold Project. Ministry of Environmental and Natural Resources (MARN) granted the environmental license for Cerro Blanco Gold Project back in 2007 for an underground operation and up to 1,000 tons per day The Company also owns a 100% interest in Mita Geothermal through its wholly owned subsidiary, Geotermia. Mita Geothermal is a geothermal energy resource located adjacent to Cerro Blanco and is seven kilometers from the Pan American Highway near the town of Asuncion Mita, in the Department of Jutiapa in Guatemala. In November of 2015, the Government of Guatemala granted Geotermia a 50-year license to build and operate a 50-megawatt geothermal plant.

This report was prepared by G Mining Services (GMS) for Bluestone to summarize the results of the N.I. 43-101 Technical Report for the Feasibility Study of the Cerro Blanco Gold Project. This report was prepared following the guidelines of the Canadian disclosure requirements of National Instrument 43-101 (N.I. 43-101) and in accordance with the requirements of Form 43-101 F1.

The Cerro Blanco Gold Project was originally studied and permitted for a high-grade underground operation in 2007. The recent engineering development, together with extensive additional drilling and detailed geological studies, significantly enhanced the understanding of the Project and presented an opportunity to capitalize on its near-surface mineralization through open pit development and improved Project economics.

The change of method from underground to open pit mining represented a significant divergence from previous technical work included in the 2019 Feasibility Study and first published as a Preliminary Economic Assessment in 2021 requiring a complete review of the Project. This technical report offers an in-depth, FS-level view of the open pit Project.

## 2.1 Scope of Work

The scope of the report was to complete an FS on the Cerro Blanco Gold Project on behalf of Bluestone utilizing an open pit mining method. The work in this report was carried out by several parties, as summarized in Table 2.1. This work constitutes the total Project scope of work.

## 2.2 Qualifications & Experience

The Qualified Persons (QPs) responsible for specific report sections are listed in Table 2.2. The QPs preparing this 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.

**Table 2.1: Scopes of Work**

| Company/Organization | Scope of Work |
|---|---|
| Bluestone Resources Inc. (Bluestone) | Evaluating status of current permits and starting new permits as required<br>Preparing a financial model and conducting an economic evaluation including sensitivity and Project risk analyses |
| G Mining Services Inc. (GMS) | Mine engineering, design and scheduling<br>Estimating mining, process plant, G&A and site services operating costs<br>Estimating mining, G&A and site services capital costs<br>Compiling the technical report including information provided by other consulting companies<br>Development of conceptual flowsheet, detailed flowsheets, specifications and selection of process equipment<br>Design oversight related to site infrastructure, access road, powerline, plant facilities and other ancillary facilities<br>Establishing gold and silver recovery values for doré production on site<br>Interpreting the results and making conclusions that lead to recommendations to improve Project value and reduce risks<br>Development of site plans and detail engineering / material take-offs (MTOs) associated with the processing facilities<br>Development of an execution plan for Project development |
| Kirkham Geosystems Ltd. (KGL) | Deposit geology and mineralization<br>Mineral resource estimate |
| Stantec Consulting Inc. (Stantec) | Geochemistry work<br>Pit inflow estimates |
| Environmental Resources Management (ERM) | Water management and water quality modelling |
| BQE Water | Water treatment design |
| SGS Canada | Geochemical testwork |
| Newfields Mining Design & Technical Services LLC (NewFields) | Tailings management<br>Tailings characterization<br>Tailings design and cost estimates<br>Site geotechnical work |
| Base Metallurgical Laboratories Ltd. (BaseMet) | Metallurgical testwork |

**Table 2.2: Responsibilities of Qualified Persons**

| Qualified Person | Company | QP Responsibility / Role | Site Visit | Report Section |
|---|---|---|---|---|
| Mathieu Gignac, P. Eng. | GMS | Introduction, Project Description, Property Description & Location, Accessibility, Climate, Local Resources, Infrastructure, & Physiography, History, Mineral Reserves Estimate, Mining Methods, Market Studies and Contracts, Capital & Operating Costs, Economic Analysis, Adjacent Properties, Other Relevant Data and Information, Interpretations & Conclusions, Recommendations, References | Nov. 13-26, 2020<br>Feb. 15-25, 2021<br>Feb. 10-12, 2022 | 1.1 to 1.6, 1.11, 1.12, 1.14, 1.16.1 to 1.17,1.19, 1.20.1 to 1.21, 2, 3, 4, 5, 6, 12.0, 12.1, 15 except for 15.3.1. 16 except 16.2.2, 19, 21.0 to 21.8, 21.8.2 to 21.8.7, 21.10 to 21.13.8, 22, 23, 24, 25.1, 25.3, 25.4, 25.6 to 25.10, 26.0, 26.4, 26.5, 26.7, 27 |
| Joël Lacelle, P. Eng. | GMS | Rehabilitation & Closure, Project Development, Project Infrastructure, Capital and Operating costs, Closure Cost Estimate | Mar. 30, 2021 | 1.15.3, 1.18, 12.5.1, 18.1, 18.2, 18.4 to 18.7, 18.8.2 to 18.12, 21.9 |
| Neil Lincoln, P. Eng. | GMS | Mineral Processing & Metallurgical Testing, Metallurgy, Recovery Methods, Process Plant Operations, Operating Cost Estimate, Processing | No | 1.9, 1.13, 12.3, 13, 17, 21.8.1, 25.5, 26.2, 26.3 |
| Garth Kirkham, P. Geo. | KGL | Geological & Mineralization, Deposit Types, Sample Preparation & Data Verification, Mineral Resource Estimate, Drilling, Analysis and Security, Exploration, Geology & Resources | May 8, 2017<br>Sep. 21-22, 2017<br>Apr. 24-28, 2018<br>Feb. 16-22, 2020<br>Jan. 10-15, 2021 | 1.7, 1.8, 1.10, 7, 8, 9, 10, 11, 12.2, 14, 25.2, 26.1 |
| Rolf Schmitt, P. Eng. | ERM | Environmental Studies, Permitting and Social or Community | No | 1.15.1, 1.15.2, 12.5.5, 20 except 20.1.3 |
| Carl J. Burkhalter, P. Eng. | NF | Tailings Management & Geotechnical Work | Oct. 13-14, 2020 | 12.5.2, 18.3, 18.13, 18.13.1, 18.13.2, 18.13.4, 18.13.5, 26.6 |
| Hong-Chang Liang, PhD, P. Chem | BQE | Water treatment Plant Process Selection and Design | No | 12.5.3, 18.8.1 |
| Jim B. Finley, PhD, P. Geo. | Stantec | Geochemical characterization, prediction of waste rock and tailings seepage chemistry | No | 12.5.4, 18.13.3, 18.14, 18.14.1, 18.14.2, 20.1.3 |
| Nolberto Contador, P. Eng. | EMT | Open Pit Design Criteria | Sep. 22-24, 2021 | 12.4.1,15.3.1,16.2.2 |

None of the QPs or any associates employed in the preparation of this report have any beneficial interest in Bluestone and none are 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 Bluestone and the QPs. The QPs are being paid a fee for their work in accordance with normal professional consulting practices.

The QPs, by virtue of their education, experience, and professional association, are considered to be Qualified Persons as defined by N.I. 43-101 requirements and are members in good standing of their appropriate professional associations and institutions.

## 2.3 Site Visits

### 2.3.1 Mr. Mathieu Gignac

Mathieu Gignac, P. Eng., visited the property from November 13 to 26, 2020, February 15 to 25, 2021 and from February 10 to 12, 2022. The site visits included an inspection of the property, offices, underground vein exposures, core storage facilities, water treatment plant, stockpiles and a tour of major centers and surrounding villages most likely to be affected by any potential mining operation. Mining data was verified during multiple site visits. Any studies referred to were thoroughly reviewed, updated and revised as required to align with the FS mine design and mine plan. All mining data was verified and is adequate for the FS Technical Report as required by NI 43101 guidelines.

### 2.3.2 Mr. Joël Lacelle

Joël Lacelle, P. Eng., visited the site March 30, 2021 and confirmed the quality and location of the different roads, buildings and services (such as electrical and water distribution). Mr. Lacelle also observed available equipment and their status at site.

### 2.3.3 Mr. Neil Lincoln

Neil Lincoln, P. Eng, did not visit the site. Mr. Lincoln determined it was not necessary to visit site given the nature of the project and the expertise required to complete the work for the relevant section of the technical report.

### 2.3.4 Mr. Garth Kirkham

Garth Kirkham, P. Geo., first visited the property on May 8, 2017 to satisfy the site visit requirements related to the 2017 Technical Report. The site visit included an inspection of the property, offices, underground vein exposures, core storage facilities, water treatment plant, stockpiles and a tour of major centers and surrounding villages most likely to be affected by any potential mining operation.

Since 2017, Mr. Kirkham has visited the property numerous times for extended periods to develop and implement data gathering and sampling method and procedures. He also worked with Bluestone geologists to develop drill programs and to supervise interpretation and wireframe modelling, in addition to vetting and review of QA/QC procedures. In September 21 to 22, 2017, Mr. Kirkham inspected the progress with the recommended historic drill core rehabilitation program and to initiate the structural studies. In April 24 to 28, 2018, the site visit focused on the advancement of the planning and advancement of sampling and drilling along with supporting lithological and structural modelling. In February 16 to 22, 2020, Mr. Kirkham assisted with the planning and development of advanced drilling and sampling and provided guidance on lithology and high-grade vein modelling for resource estimation. In January 10 to 15, 2021, Mr. Kirkham validated drill and sample data, refined high-grade models, reviewed low-grade models, and provided guidance for finalizing the open pit bulk tonnage resource scenario.

### 2.3.5 Mr. Rolf Schmitt

Rolf Schmitt, P. Geo, has not visited site. Mr. Schmitt determined it was not necessary to visit site given the nature of the project and the expertise required to complete the work for the relevant section of the technical report.

### 2.3.6 Mr. Carl J. Burkhalter

Carl Burkhalter, P.E., visited the site on October 13 and 14, 2020 and reviewed the past infrastructure geotechnical field and laboratory work. Mr. Burkhalter also reviewed the current site conditions and gathered additional data as needed.

### 2.3.7 Mr. Hong-Chang Liang

Hong-Chang Liang, P. Chem, has not visited site. Mr. Liang determined it was not necessary to visit site given the nature of the project and the expertise required to complete the work for the relevant section of the technical report.

### 2.3.8 Mr. Jim B. Finley

Jim B. Finley, P. Geo, has not visited site. Mr. Finley determined it was not necessary to visit site given the nature of the project and the expertise required to complete the work for the relevant section of the technical report.

### 2.3.9 Mr. Nolberto Contador

Norberto Contador, P.E. visited site on September 22 to 24, 2021, Mr. Contador and reviewed the current site conditions and gathered additional data for the pit stability study that E Mining Technology designed. Based on this study and his review, he proposed the design parameters for the open pit. E Mining approved the final mine design for the Cerro Blanco Gold Project used in this FS.

## 2.4 Sources of Information & Data

This report is based on information collected by the QPs during site visits and on additional information provided by Bluestone throughout the course of GMS's investigations. Other information was obtained from the public domain. GMS has no reason to doubt the reliability of the information provided by Bluestone. This technical report is based on the following sources of information:

- discussions with Bluestone's on-site personnel, including the site General Manager and Environmental Manager
- inspection of the site, including underground, surface facilities and drill core
- review of exploration data collected by Bluestone
- previous studies completed by Goldcorp and Bluestone
- additional information from public domain sources

## 2.5 Units of Measure, Abbreviations & Nomenclature

All units of measurement in this report are metric and all costs are expressed in United States dollars (US$ or USD) unless otherwise stated. Contained gold metal is expressed as troy ounces (oz). All material tonnes are expressed as dry tonnes (t) unless stated otherwise.

Lists of the main units of measure and abbreviations used throughout this report are presented in Sections 2.5.1 and 2.5.2, respectively.

### 2.5.1 Units of Measure

Above mean sea level ........ amsl
Annum (year) ........ a
Cubic centimeter ........ $cm^3$
Cubic meter ........ $m^3$
Cubic yard ........ $yd^3$
Day ........ d
Days per week ........ d/wk
Days per year (annum) ........ d/a
Degree ........ °
Degrees Celsius ........ °C
Dollars per ounce ........ $/oz
Dollars per tonne ........ $/t
Dry metric ton ........ dmt
Foot ........ ft
Gallon ........ gal
Gallons per minute ........ gal/min
Gram ........ g
Grams per litre ........ g/L
Grams per tonne ........ g/t
Greater than ........ >
Hectare (10,000 $m^2$) ........ ha

Horsepower ........ hp
Hour ........ h
Hours per day ........ h/d
Hours per week ........ h/wk
Hours per year ........ h/a
Inch ........ "
Kilo (thousand) ........ k
Kilogram ........ kg
Kilograms per cubic meter ........ kg/m$^3$
Kilograms per hour ........ kg/h
Kilograms per square meter ........ kg/m$^2$
Kilometer ........ km
Kilometers per hour ........ km/h
Kilopascal ........ kPa
Kilovolt ........ kV
Kilovolt-ampere ........ kVA
Kilovolts ........ kV
Kilowatt ........ kW
Kilowatt hour ........ kWh
Kilowatt hours per tonne (metric ton) ........ kWh/t
Kilowatt hours per year ........ kWh/a
Less than ........ <
Litre ........ L
Litres per minute ........ L/m
Megabytes per second ........ Mb/s
Megapascal ........ MPa
Megavolt-ampere ........ MVA
Megawatt ........ MW
Meter ........ m
Meters above sea level ........ masl
Meters per second ........ m/s
Metric ton (tonne) ........ t
Micrometer (micron) ........ µm
Milligram ........ mg
Milligrams per litre ........ mg/L
Millilitre ........ mL
Millimeter ........ mm
Million ........ M
Million tonnes ........ Mt
Million pounds ........ Mlb
Minute (plane angle) ........ '
Minute (time) ........ min
Month ........ mo
Ounce ........ oz
Ounces per tonne ........ oz/t
Parts per billion ........ ppb
Parts per million ........ ppm
Pascal (newtons per square meter) ........ Pa
Pascals per second ........ Pa/s
Percent ........ %
Phase (electrical) ........ Ph
Pound(s) ........ lb
Second (time) ........ s

Specific gravity ........ SG
Square centimeter ........ $cm^2$
Square foot ........ $ft^2$
Square inch ........ $in^2$
Square kilometer ........ $km^2$
Square meter ........ $m^2$
Thousand ounces ........ koz
Thousand tonnes ........ kt
Tonne (1,000 kg) ........ t
Tonnes per day ........ t/d
Tonnes per hour ........ t/h
Tonnes per year ........ t/a
Volt ........ V
Week ........ wk
Weight/weight ........ w/w
Wet metric ton ........ wmt
Yard ........ yd
Year (annum) ........ a
Year ........ y

### 2.5.2 Acronyms & Abbreviations

Acid base accounting ........ ABA
Acid rock drainage ........ ARD
All-in sustaining cost ........ AISC
Alternating current ........ AC
Atomic absorption ........ AA
Autogenous grinding ........ AG
Ball mill work index ........ BWI
Base Metallurgical Laboratories ........ BaseMet
Bed volume ........ BV
Bluestone Resource Inc. ........ Bluestone
Canadian ........ CDN
Canadian Dam Association ........ CDA
Canadian dollars ........ C$ or CAD
Canadian Electrical Manufacturers Association ........ CEMA
Canadian Institute of Mining, Metallurgy and Petroleum ........ CIM
Canadian Standards Association ........ CSA
Carbon in leach ........ CIL
Carbon in pulp ........ CIP
Conveyor Equipment Manufacturer’s Association ........ CEMA
Daily maximum precipitation ........ DMP
Diamond drilling ........ DD
Direct current ........ DC
Distributed control system ........ DCS
Dry Stack tailings facility ........ DSTF
Engineering and procurement ........ EP
Engineering Institute of Canada ........ EIC
Engineering, procurement, and construction management ........ EPCM
Environmental and Social Impact Assessment ........ ESIA
Environmental impact assessment ........ EIA
Environmental Management Plan ........ EMP
Environmental Resources Management ........ ERM

Factor of Safety ........ FOS
Flow coefficient ........ CV
Front-end loader ........ FEL
G Mining Services ........ GMS
Gearless mill drive ........ GMD
General and administrative ........ G&A
General arrangement ........ GA
Global positioning system ........ GPS
Gravity recoverable gold ........ GRG
Hardness index tester ........ HIT
Heating, ventilating, and air conditioning ........ HVAC
High density polyethylene ........ HDPE
High pressure ........ HP
High voltage ........ HV
High-intensity grinding ........ HIG
Hydrochloric acid ........ HCl
Integrated project management team ........ IPMT
Intermediate bulk containers ........ IBCs
Internal rate of return ........ IRR
International Building Code ........ IBC
Inverse distance squared ........ ID2
Kirkham Geosystems Ltd. ........ KGL
Low pressure ........ LP
Material take-off ........ MTO
Million dollars ........ $M
Motor control center ........ MCC
Nearest neighbour ........ NN
Net present value ........ NPV
Net smelter return ........ NSR
Not potentially acid-generating ........ NPAG
Ordinary kriging ........ OK
Original equipment manufacturer ........ OEM
Overall slope angle ........ OSA
Paterson & Cooke ........ P&C
Plant control system ........ PCS
Potentially acid generating ........ PAG
Preliminary Economic Assessment ........ PEA
Probable maximum flood ........ PMF
Probable maximum precipitation ........ PMP
Process flow diagram ........ PFD
Project Execution Plan ........ PEP
Qualified Person ........ QP
Quality assurance and quality control ........ QA/QC
Quantitative evaluation of materials by scanning electron microscopy ........ QEMSCAN
Quantitative X-ray diffraction ........ QXRD
Reverse circulation ........ RC
Rock mass rating ........ RMR
Rock quality designation ........ RQD
Rod mill work index ........ RWI
Roll-on, roll-off ........ RoRo
Run of mine ........ ROM
SAG and ball ........ SAB
Semi-autogenous grinding ........ SAG

Social Baseline Study ........ SBS
Social Management Plan ........ SMP
Sodium metabisulphite ........ SMBS
Stantec Consulting ........ Stantec
Supervisory control and data acquisition ........ SCADA
Synthetic precipitation leaching procedure ........ SPLP
System for Electronic Document Analysis and Retrieval ........ SEDAR
Total cyanide ........ $CN_T$
Underflow ........ U/F
Undersize ........ U/S
Uninterruptible power supply ........ UPS
US dollars ........ US$ or USD
Value added tax ........ VAT
Valued ecosystem component ........ VEC
Variable frequency drive ........ VFD
Variable speed drive ........ VSD
Water treatment plant ........ WTP
Weak acid dissociable ........ WAD
Weak acid dissociable cyanide ........ $CN_{WAD}$
Wide area network WAN
Waste-to-ore ratio ........ W:O
Work breakdown structure ........ WBS

# 3. RELIANCE ON OTHER EXPERTS

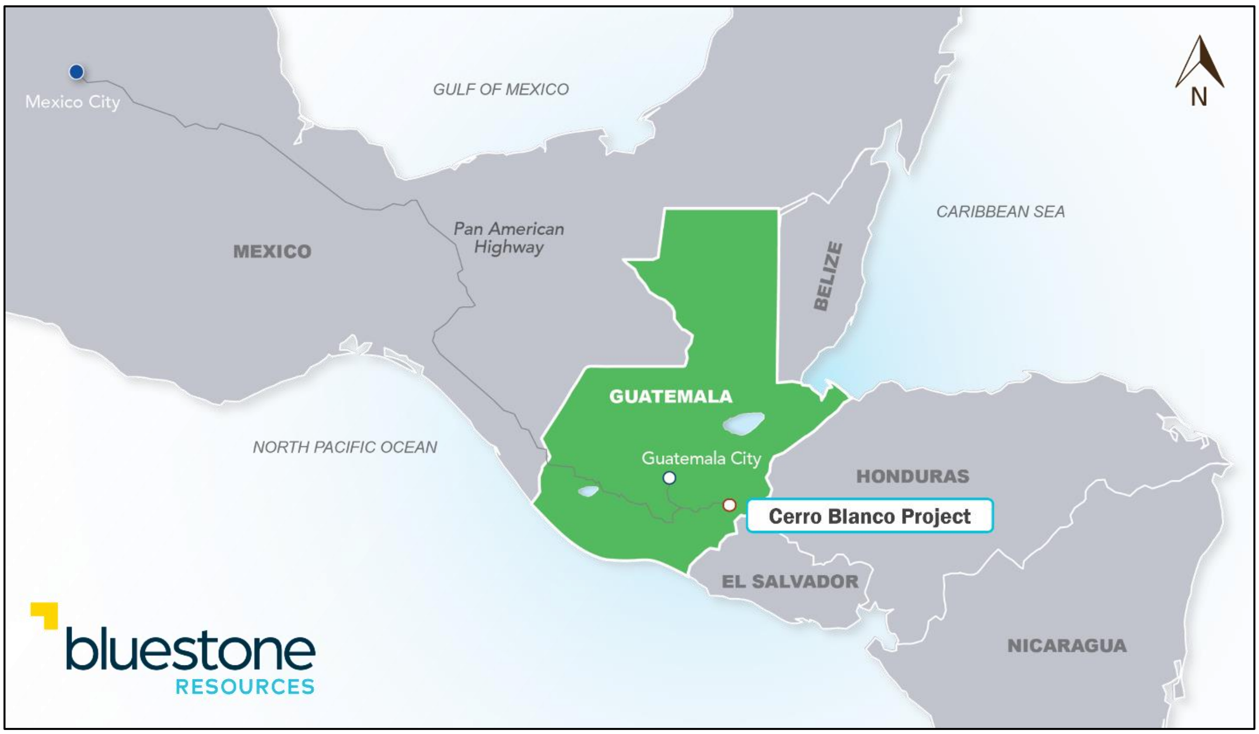
Not applicable.

# 4. PROPERTY DESCRIPTION & LOCATION

## 4.1 Location

The Cerro Blanco Gold Project is located in southeast Guatemala, in the Department of Jutiapa, approximately 160 km by road from the capital, Guatemala City (Figure 4-1) and approximately 9 km west of the border with El Salvador. The nearest town to the Project is Asunción Mita, a community of approximately 18,500 people situated approximately 7 km west of the Project. The exploitation license covers 15.25 km$^2$ and lies entirely in the municipality of Asunción Mita.

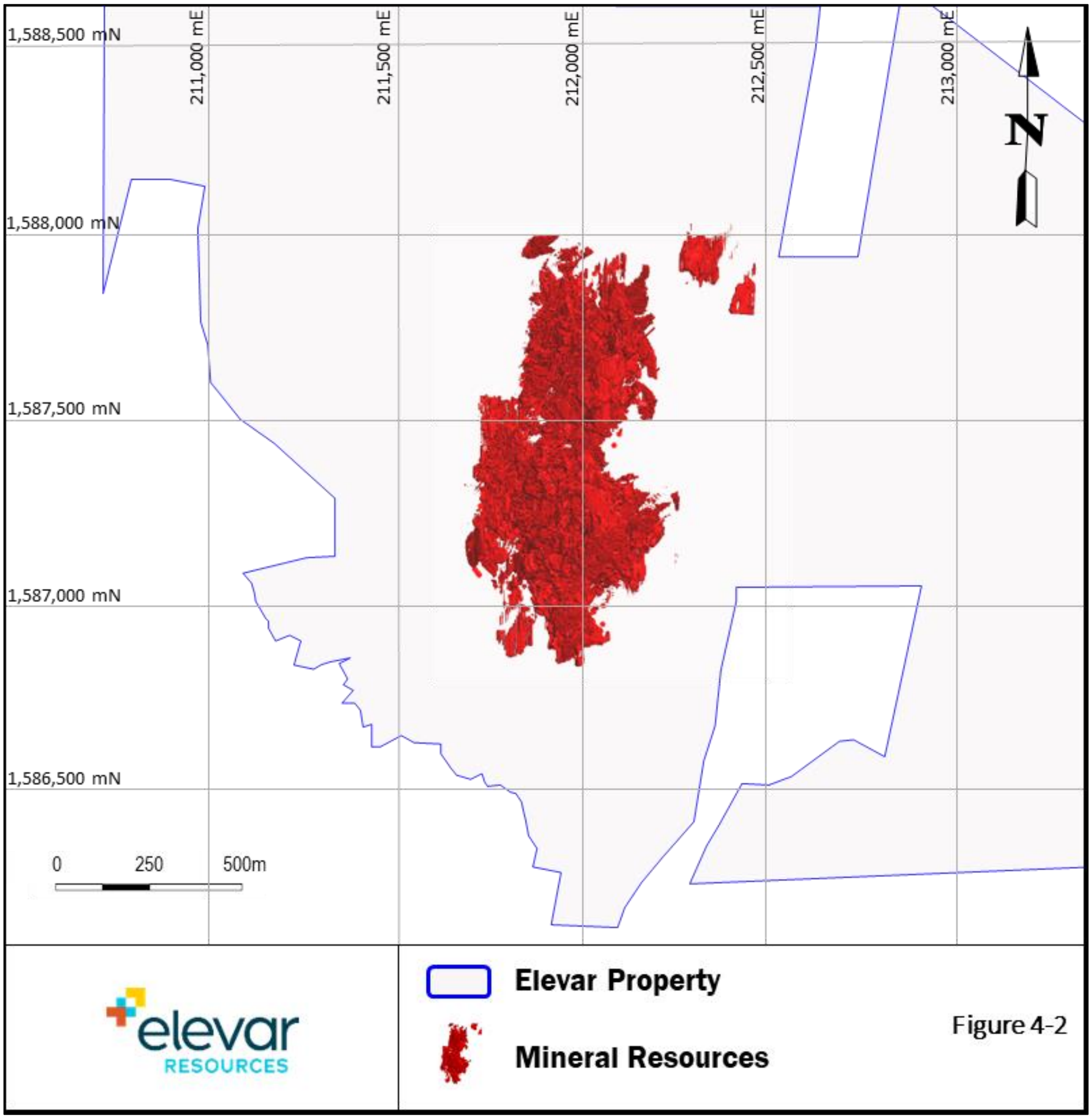
**Figure 4-1: Project Location Map**




Source: Bluestone (2021).

The location of the mineral resources relative to the property boundary is shown in Figure 4-2.

**Figure 4-2: Location of Mineral Resources Relative to Property Boundary**




Source: Bluestone (2022).

## 4.2 Mineral Tenure

### 4.2.1 Land Strategy & Property Ownership

Elevar is an indirect, wholly-owned subsidiary of Bluestone Resources, who acquired 100% of Elevar (formerly called Entre Mares de Guatemala S.A.) from Goldcorp Inc. (Goldcorp) in 2017. Goldcorp originally acquired the Project from Glamis Gold in 2006 as an exploration property. Elevar has purchased the land over the deposit and for the plant facilities. Additional land is being purchased to extend the footprint and serve as a buffer to local communities

#### 4.2.2 Underlying Agreements Specific to the Resource & Land Tenure

In Guatemala, natural resources are property of the State. Consequently, mining exploitation only requires the resolution of the Ministry of Energy and Mines and the Ministry of the Environment before an exploitation license can be granted. Land tenure in Guatemala is regulated by the constitutional law which states that all citizens have the right to obtain land as private property. Access to land during exploration and exploitation by mining companies can be obtained by outright purchase of surface rights from landowners or by lease agreements under civil and constitutional law.

#### 4.2.3 Mineral Rights

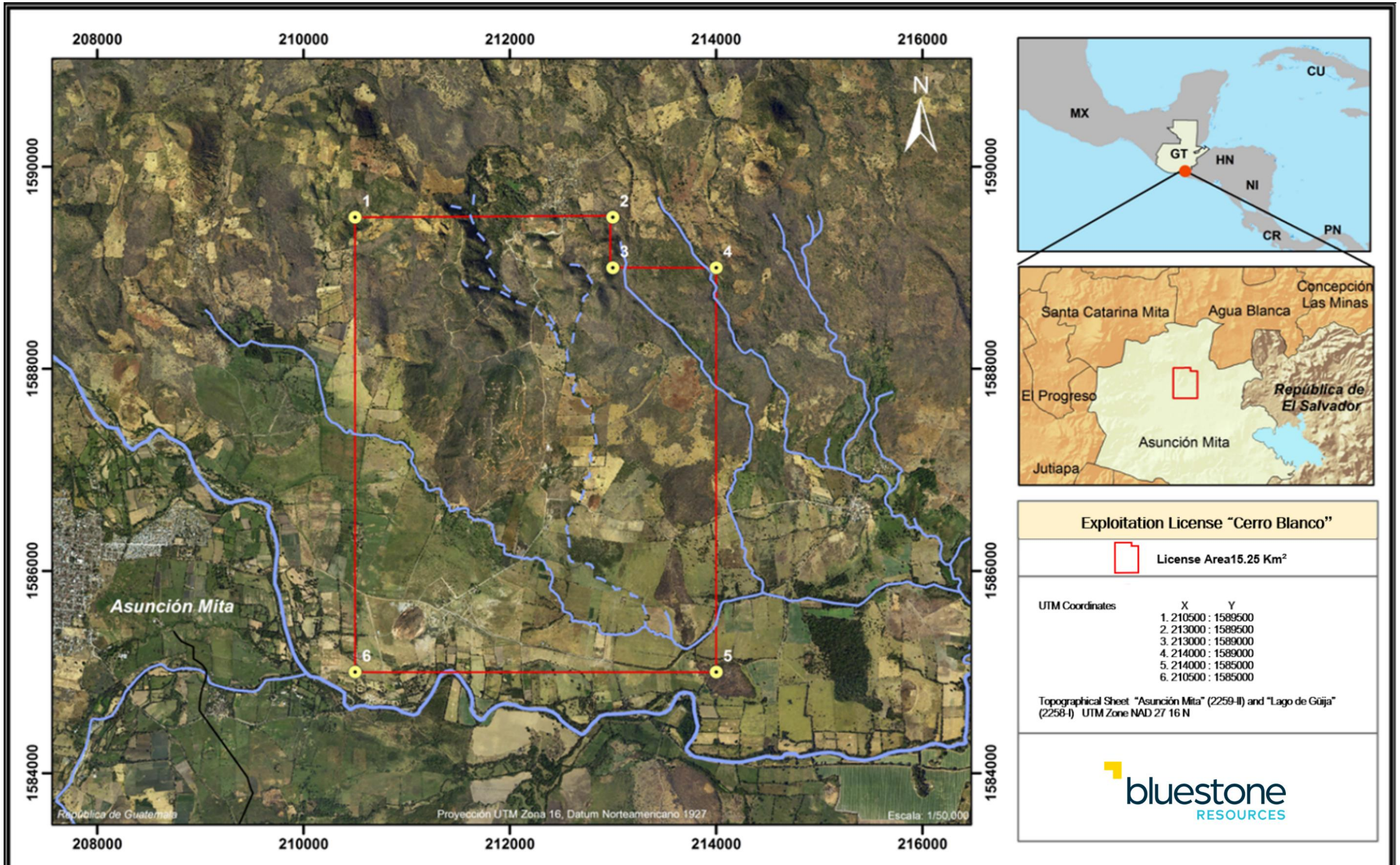
Guatemalan mining law provides for three types of licenses specifically for reconnaissance, exploration and exploitation. A reconnaissance license has a term of six months with the possibility of renewal for an additional six-month period. The reconnaissance license can cover an area from 500 to 3,000 km$^2$ and, upon application, can be converted to an exploration license. An exploration license covers areas up to 100 km$^2$, has a term of three years, and can be extended for two additional two-year periods. With each extension, the surface area of the exploration license must be reduced by 50% unless a request is made for an extension two months prior to expiration of the term. An exploration license is succeeded by an exploitation license, which has a 25-year term that can be extended for a second 25-year period. It covers a maximum of 20 km$^2$. Surface right fees are payable with all types of licenses and royalties of 1% are payable on commercial production at the exploitation stage, shared between the State and the municipality where the Project is situated. An additional 1% royalty exists to a third party as part of the acquisition of the Project.

The original Cerro Blanco exploration concession was awarded to Elevar Resources in 1997 and covered an area of 39 km$^2$. The exploitation license and Environmental and Social Impact Assessment (ESIA), covering 15.25 km$^2$ were granted in 2007 for a fully permitted underground mine. This approved ESIA will require an amendment due to the proposed change in mining method to an open pit. EIA’s and permits are also required for the new access road and powerline. The amended Environmental and Social Impact Assessment (ESIA) has been submitted to the Ministry of Environment and Natural Resources (Ministerio de Ambiente y Recursos Naturales (MARN)) to support the FS design of the open pit scenario.

### 4.3 Mining Rights

The coordinates of the 15.25 km$^2$ exploitation license are recorded in Decree DIC-CM-158-05 and are shown in Figure 4-3. The perimeter of the area is described as having the UTM X and Y coordinates shown on Figure 4-3.

**Figure 4-3: Cerro Blanco Exploitation License Coordinates**




Source: Bluestone (2019).

The Cerro Blanco Gold Project will be developed as an open pit mining operation. Elevar owns the mining rights for Cerro Blanco but will require the permits and water rights to develop the property as an open pit operation.

## 4.4 Project Agreements

In 2007, the government of Guatemala (under the Mining Law, Decree Number 114-97) awarded Elevar an exploitation license "Proyecto Minero Cerro Blanco". This granted the Company exclusive rights to mine gold and silver inside the boundaries of the approved concession for a period of 25 years from the day the notification was received.

The liens, charges and/or encumbrances described below are applicable to the Project.

- Authorization Canon: In accordance with paragraph a) of Article 66 of the Mining Law, the title holder must pay in advance, in Quetzals at the time of the resolution, an authorization payment of 1,300 Quetzals.
- Surface Area Canon: In accordance with Paragraph d) of Article 66 of the Mining Law, the title holder must pay in advance, in January of each year, a surface area canon of 12 units per square kilometer or fraction thereof. The payment in the first year is made at the time of notification of the resolution; this amount is proportional to the time remaining in the year from the approval date to December 31.
- Royalties: The title holder must pay royalties to the State and the Municipality of Asunción Mita for the extraction of mineral products. The royalties are determined according to Article 63 of the Mining Law, Decree 48-97 and Article 33 of the Mining Law, Agreement 176-2001. The royalties must be paid to the State and the Municipality within 30 days after the end of the calendar year. The title holder must attach a photocopy of the receipts of payment to his annual report. Failure to pay will attract interest payments at the current average rate in the banking system.
- Title Holder Obligations: The title holder of the exploitation license is obliged to:
  1. Start exploitation work on the "orebody" within 12 months of the date of the approval resolution; however, this period will be extended if the characteristics of the orebody require it, or when other circumstances justify it.
  2. Technically develop the orebody.
  3. Within the stipulated periods, pay the surface area canon and royalties.
  4. Comply with the terms of the approved Environmental Impact Study and carry out the recommendations indicated in resolution 2613-2007/ECM/LP, August 14, 2007, issued by the Ministry of Environment and Natural Resources.
  5. Compensate third parties for any damages and harm that may be caused by the operation.
  6. Present a written annual report to the Directorate General of Mining within three months of the end of each calendar year.

## 4.5 Environmental Liabilities & Considerations

The Cerro Blanco Gold Project is not subject to environmental liabilities other than the obligations attached to the existing permits. The Cerro Blanco Gold Project Permits Register shows that all applicable permits obligations have been fulfilled.

## 4.6 Permit Requirements

Bluestone will require all necessary permits to proceed with developing the open pit mine and construction of the process facilities, subject to future operations adhering to the conditions of such permits.

## 4.7 Property Risks

The pivot to an open pit operation requires an increased footprint for surface infrastructure. Bluestone is finalizing the acquisition of additional land necessary for various surface infrastructure. Until all land access has been acquired, this represents a risk to the Project.

# 5. ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE & PHYSIOGRAPHY

## 5.1 Accessibility

Current road access to site is via the Pan-American Highway (Highway CA1) through the town of Asunción Mita. Existing infrastructure is in place to provide year-round access to the site. The topography is relatively flat with rolling hills.

Guatemala has 400 km of coastline and claims its territorial waters extend 22 km outward, plus an exclusive economic zone of 370 km offshore. Hurricanes and tropical storms sometimes affect coastal regions.

The five main ports in Guatemala and their main activities are listed below:

- Atlantic Ports:
  - Puerto Santo Tomás de Castilla (containers)
  - Puerto Barrios (containers)
- Pacific Ports:
  - Puerto San José (liquids)
  - Puerto Quetzal (multi-use)
  - Puerto Champerico (fishing)

Puerto Santo Tomás de Castilla is the most important port on the Atlantic coast of Guatemala. This cargo terminal can handle a variety of cargo (e.g., containers and roll-on, roll-off (RoRo)), as well as general and liquid bulk cargo, passenger ships, vehicle carriers, and barges. The port facilities are approximately 290 km northeast of Guatemala City. The total distance from Santo Tomás de Castilla to the Cerro Blanco Gold Project site is approximately 440 km.

Puerto Quetzal, which is the most important port on the Pacific Coast, has the most modern installations. It is mainly a dry bulk cargo terminal; however, it also handles containers, RoRo, general bulk cargo, and liquid bulk cargo. The port facilities are about 100 km South of Guatemala City. The distance from Puerto Quetzal to the Project site using the coastal highway is approximately 310 km. Puerto Quetzal is 2,050 nautical miles from Los Angeles.

These two ports handle nearly 80% of the sea traffic to Guatemala. Guatemala's Empresa Nacional Portuaria is a state-owned corporation of the Guatemalan port facilities.

## 5.2 Climate & Physiography

The Cerro Blanco Gold Project is located on a hill with two peaks. The surrounding areas are relatively flat with minimal undulation. A photo showing the typical landscape around the mine property is included as Figure 5-1.

**Figure 5-1: Typical Landscape in the Project Area, Looking South**

Source: Bluestone (2020).

The climate and vegetation at the Cerro Blanco Gold Project site are typical of a tropical dry forest environment. The elevation is between 450 and 560 masl. The wet season is typically from May to October. The average annual rainfall is 1,350 mm. Daily highs reach 41°C and lows reach 10°C. The average annual pan evaporation rate is 2,530 mm with an annual average humidity of 62%.

Classified as Zona Oriental, the principal characteristics of the region are a deficiency of rain for much of the year with high ambient daytime temperatures. Most of the vegetation in the Project area loses its foliage because of a lack of precipitation to support growth during the winter months of November through April.

The Project occurs within a south-southwest trending ridge that extends from higher ground to the north, outward into the basin and floodplain deposits of the Rio Ostua. The elevation of the upper part of the ridge is in excess of 600 masl. The elevation of the basin and flood plain deposits is about 460 to 490 masl.

The west side of the ridge is flanked by a south-southeast-trending perennial drainage called Rio Tancushapa. The east side of the ridge is flanked by a seasonal drainage called Quebrada El Tempisque which also trends to the south-southeast. These drainages join to the south-southeast of the Project area and flow into the Rio Ostua about 4 km down gradient.

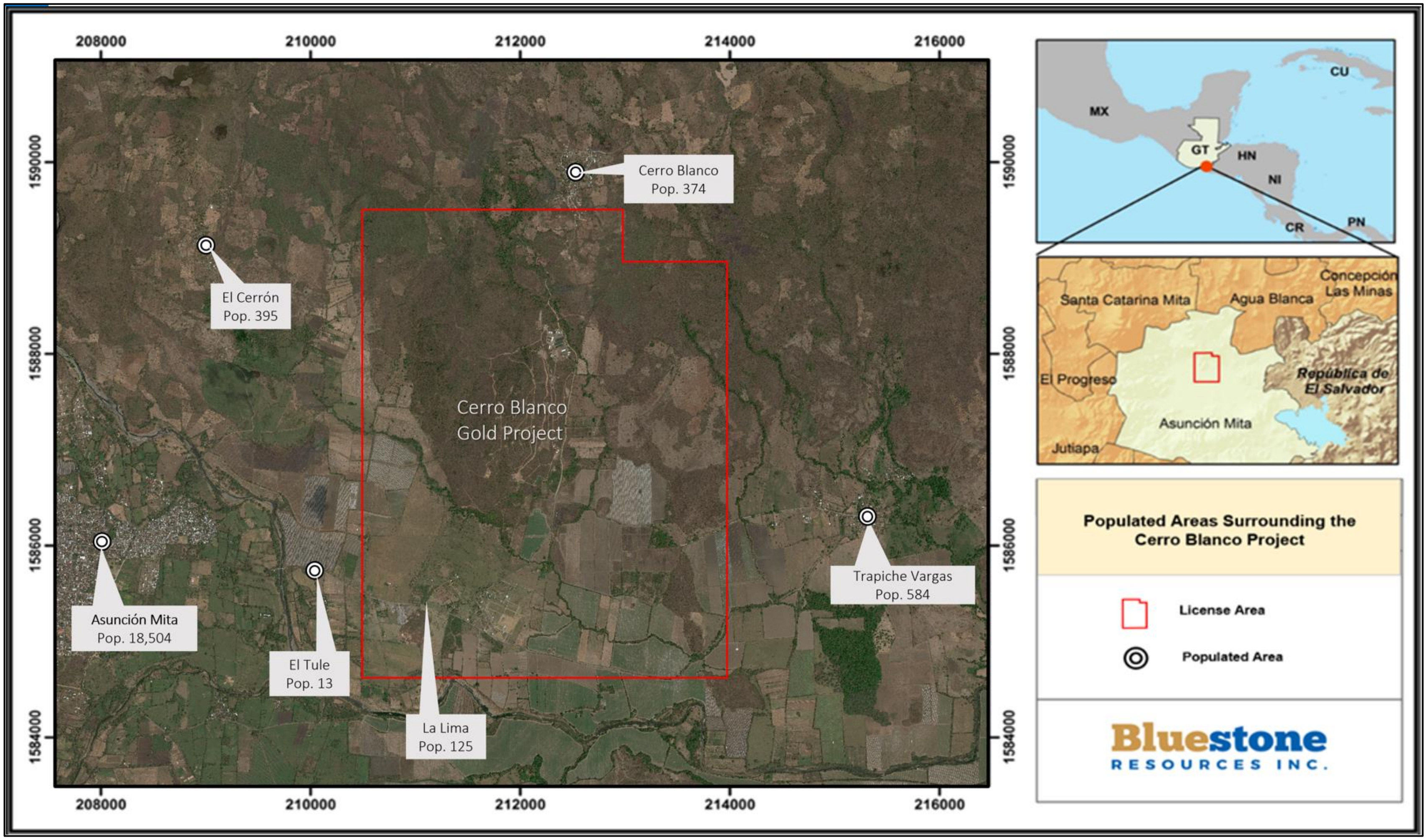
The regional area is generally hilly to mountainous with broad flood plains formed by some of the larger streams and rivers. Three dormant volcanoes are within sight of the Project area: Ixtepeque to the north, Suchitan to the northwest, and Las Viboras to the southwest.

## 5.3 Local Resources & Infrastructure

The Cerro Blanco Gold Project is situated in proximity to a number of communities, the largest one being Asunción Mita, with a population of approximately 18,500 people. The recently constructed La Baranca power substation located a few kilometers to the south of Mita. The substation has a capacity to supply up to 20 MW of power.

There is no record of any previous exploitation in the area; however, with the closure of Goldcorp's Marlin Mine in late 2017 it is anticipated that a significant contingent of Guatemalan trained labour will be available for employment at Cerro Blanco. As such, the Project intends to hire the majority of operations staff locally and has allowed for cost of training programs within the Owner's budget. The local mine workforce is expected to live in the surrounding communities and provide their own transportation to and from the mine site due to proximity of the population centers relative to the Project site (Figure 5-2). Employees from distant areas further than Jutiapa and expatriate employees will be housed in the on-site camp.

**Figure 5-2: Population Centers near the Project Area**




Source: Bluestone (2022).

# 6. HISTORY

## 6.1 Management & Ownership

There is no evidence of exploration activity on the Cerro Blanco property prior to 1997. Mar-West Resources Ltd. (Mar-West), a Canadian exploration company, had been working in adjacent Honduras since 1995 and expanded their gold prospecting activities into southern Guatemala in 1997. The Cerro Blanco property was identified by Mar-West by sampling densely silicified boulders, in some cases cut by chalcedonic veinlets, during an initial reconnaissance evaluation of an area known for active hot springs. Traverses over the hill at Cerro Blanco yielded surface rock assays of 1 to 3 g Au/t. An exploration concession was subsequently applied for and granted in late 1997. Mar-West drilled nine reverse circulation (RC) holes from April to June 1998 which tested near-surface potential to shallow depths of 100 to 150 m. At least seven holes contained one or more intercepts of 5 to 15 m grading 1 to 5 g Au/t, with the occasional 10 to 20 g Au/t interval and were sufficient to justify continued exploration on the property.

In October 1998, Mar-West's holdings in Honduras and Guatemala were purchased by Glamis Gold Ltd. (Glamis) primarily to acquire the San Martin deposit in Honduras. Mar-West geologists continued to manage the Cerro Blanco exploration program through March 1999. The sinter area was soil sampled and trenched, and drilling was advanced to hole 19 when geophysical orientation surveys were undertaken. A further 331 drill holes were completed up until 2006.

Goldcorp became the sole proprietor of the Cerro Blanco Gold Project through the purchase of Glamis in November 2006. Goldcorp undertook a comprehensive exploration program from 2006 to 2012 including additional surface exploration, over 3.4 km of underground development, and 43,016 m of surface and underground drilling. Exploration activities at the Cerro Blanco property by Goldcorp included the following:

- surface soil geochemistry
- surface rock geochemistry
- surface geological mapping
- construction of the north and south ramp access and ventilation raises
- underground geological mapping
- underground chip sampling
- surface and underground diamond drilling.

Several unpublished feasibility studies were completed by Goldcorp from 2011 to 2014. Kappes, Cassidy & Associates (KCA) and Golder Associates (Golder) completed an FS for the Project in May 2012. After this initial FS, Goldcorp issued a new geological model and requested KCA and Golder to update the FS in 2013 using a revised mine design, mine development, mine operation and capital costs. In 2014, an internal updated FS was produced with optimized mine stope parameters, and the mine schedule and costing information updated by Maptek.

## 6.2 Historical Resources

The updated FS (internal report) by Goldcorp in 2014 included the in-situ resource estimate shown in Table 6.1. In-situ mineral resources do not consider mineral availability by underground or open pit mining methods.

**Table 6.1: Indicated & Inferred In-Situ Mineral Resource Estimate (2014)**

| Cut-off Au g/t | Tonnes (Mt) | Au Grade (g/t) | Ag Grade (g/t) | Contained Gold (Moz) | Contained Silver (Moz) |
|---|---|---|---|---|---|
| Indicated Resources | | | | | |
| 1.0 | 19.0 | 3.38 | 13.9 | 2.06 | 8.5 |
| 4.0 | 3.85 | 9.71 | 35.2 | 1.20 | 4.4 |
| Inferred Resources | | | | | |
| 1.0 | 2.3 | 3.1 | 8.5 | 0.22 | 0.6 |
| 4.0 | 0.33 | 10.8 | 19.8 | 0.11 | 0.2 |

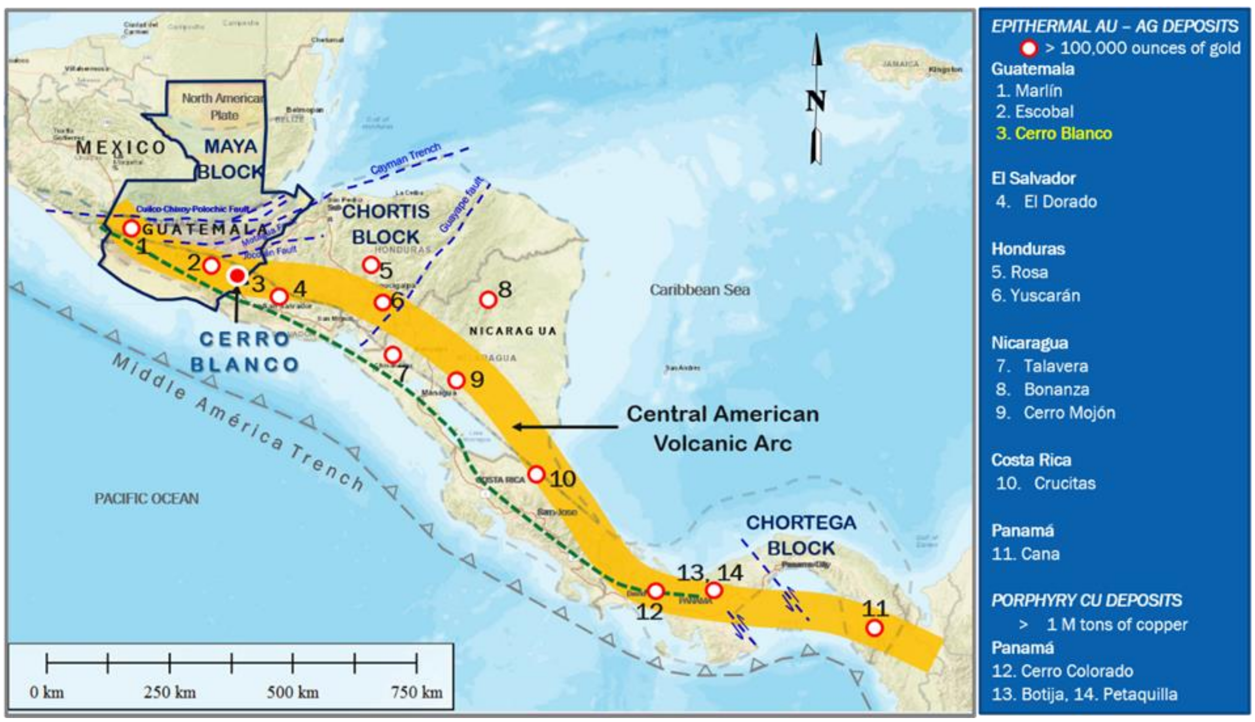
Note: The indicated and inferred resources are historical estimates and use the categories set out in N.I. 43-101. These resources are effective as of May 2014 and are disclosed in Goldcorp's updated FS (May 2014). Given the source of the estimates, Bluestone considers them reliable and relevant for the further development of the Project; however, a qualified person has not done sufficient work to classify the historical estimates as current mineral resources or mineral reserves, and the Company is not treating the historical estimates as current mineral resources or mineral reserves.

# 7. GEOLOGICAL HISTORY & MINERALIZATION

## 7.1 Introduction

The geology of Guatemala comprises rocks that are divided into two tectonic terrains due to the collision between the North American, Caribbean, and Cocos tectonic plates during the Upper Cretaceous, 70 to 90 million years ago. The Maya Block to the north is characterized by igneous and metamorphic basement rocks overlain by late Palaeozoic metasediments. Mesozoic red beds, evaporites and marine limestones overlie these rocks, and a karst landscape formed in the thick limestone units across the north of the country. By contrast, southern Guatemala, south of the Motagua Valley, belongs to the Chortis Block, representing the northern part of the Caribbean Plate. This region forms an active volcanic arc termed the Central American Volcanic Arc (Figure 7-1) which continues from the Guatemala-Mexico border along the Pacific side of Central America into central Costa Rica, with most of the major eruptive events having occurred in the Tertiary and Quaternary.

**Figure 7-1: Location of Cerro Blanco & Other Deposits in the Central American Volcanic**



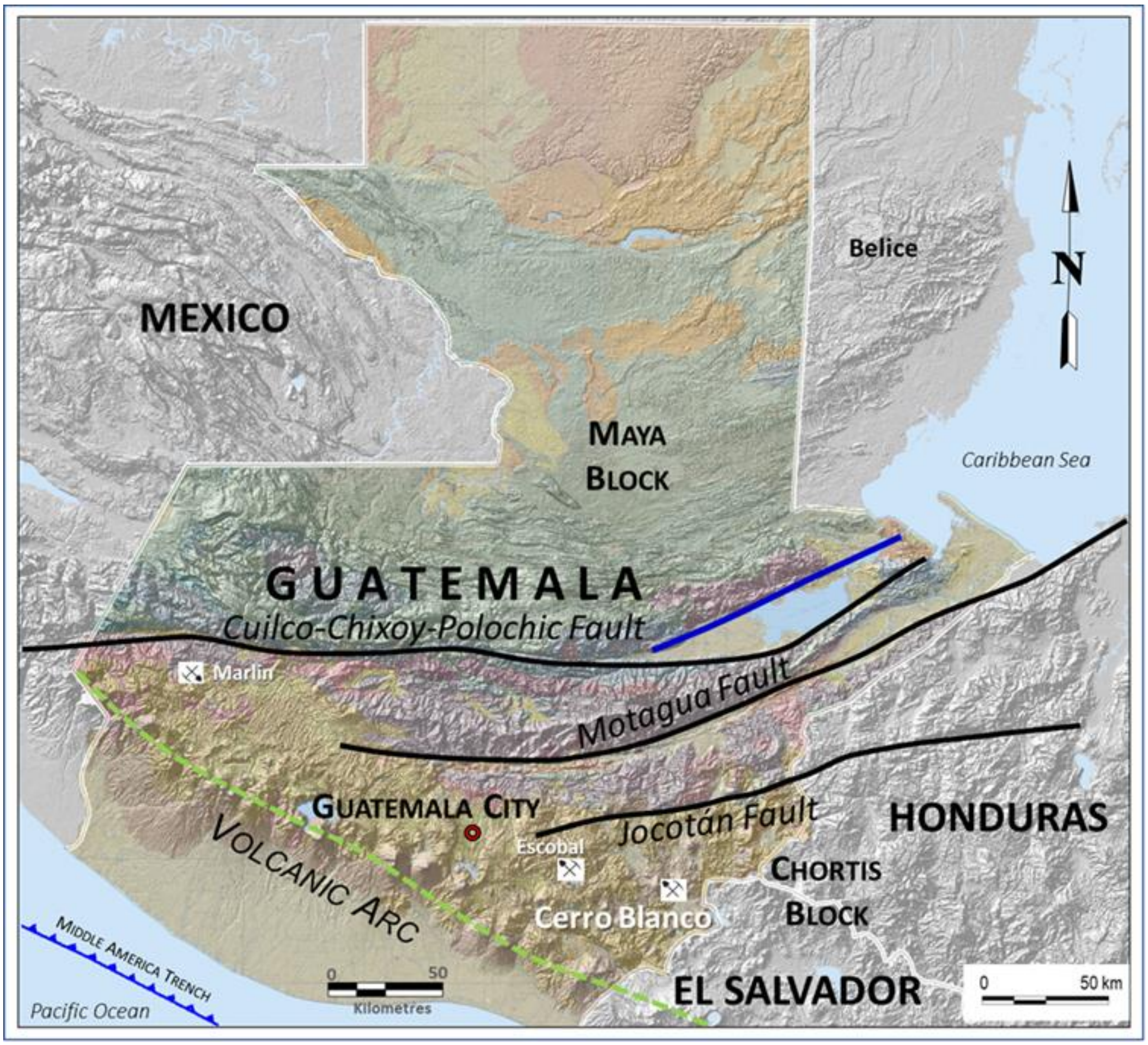

Source: Bluestone (2020).

## 7.2 Regional Geology of Southern Guatemala

Southern Guatemala, El Salvador, Honduras, and Nicaragua are located within the Chortis continental crustal block. The tectonic event that sutured the Chortis block to the North American craton took place between 66 and 70 million years ago along the east-west-striking Polochic-Montagua fault system that crosses southern Guatemala (Figure 7-2). Three regional east-west trending, left-lateral transform faults form the plate collision boundary, defined by the Polochic, Motagua and Jocotan fault systems from north to south. Nearer the Cerro Blanco deposit, other major regional structures that strike north-northeast, such as the Jalpatagua and Ipala faults, are important local structures.

A large group of granitic stocks and batholiths intruded the suture zone south of the Polochic-Montagua fault with ages of 35 to 85 million years. These broadly bracket, both temporally and spatially, the collision event (Donnelly, et al., 1990).

**Figure 7-2: Regional Structural Map of Guatemala**




Source: Bluestone (2021).

The Jocotan Fault is generally considered the southern-most major suture-related fault. It is an east-west fault with considerable Late Cretaceous dip-slip movement (south side down), but it had little or no Tertiary transcurrent movement. Cerro Blanco is located about 50 km south of the Jocotan Fault.

The ancestral Middle America Trench developed at this time. The Pacific Oceanic plate is subducted beneath Central America and is the principal driving force for volcanic and intrusive igneous activity throughout Central America along this boundary trench. The earliest documented volcanic outpouring on the Chortis block was Paleocene (about 55 to 65 million years ago) (Pindell and Barrett, 1990).

In Costa Rica and Panama, a series of west-northwest-trending (arc-parallel) back-arc basins developed. These accumulated tuffaceous sediments continuously from Eocene (about 55 million years) to the present (Donnelly, et al., 1990). The principal periods of Andean-style calc-alkaline volcanism in the Chortis block include the Paleocene-Eocene (relatively minor), Oligocene (major), and Miocene-Pliocene (the biggest) (Pindell and Barrett, 1990).

The Polochic-Montagua suture was reactivated as a sinistral (left-lateral) transform fault that displaced the Chortis block 130 km eastward with respect to the North American craton. Movement took place from 6 to 10 million years ago (Deaton and Burkart, 1984). Associated extension was accommodated by a series of north-south grabens across southern Guatemala and western Honduras. Back-arc rift basins developed adjacent to northwest-striking normal faults all along western Central America. The Nicaraguan Rift began to form about 7 million years ago and continues to subside today. Bimodal, rhyolite-basalt volcanism began during this event and by 7 million years ago was widespread throughout the western half of the Chortis block.

A large number of Central American gold deposits, including Marlin and Cerro Blanco, occur within a narrow belt parallel to the western Central American coast from southern Guatemala through to Panama. The geology of Guatemala comprises rocks that are divided into two tectonic terrains due to the collision between the North American, Caribbean, and Cocos tectonic plates during the Upper Cretaceous, 70 to 90 million years ago. The Maya Block to the north is characterized by igneous and metamorphic basement rocks overlain by late Palaeozoic metasediments. Mesozoic red beds, evaporites and marine limestones overlie these rocks, and a karst landscape formed in the thick limestone units across the north of the country. By contrast, southern Guatemala, south of the Motagua Valley, belongs to the Chortis Block, representing the northern part of the Caribbean Plate. This region forms an active volcanic arc termed the Central American Volcanic Arc (Figure 7-1) which continues from the Guatemala-Mexico border along the Pacific side of Central America into central Costa Rica, with most of the major eruptive events having occurred in the Tertiary and Quaternary.

This metallogenic belt follows the volcanic arc, and precious metal deposits are clearly related in space and time to Miocene-Pliocene extensional tectonics and associated bimodal basalt-rhyolite volcanism. Published age dates cluster between 4 and 8 million years. Argon-argon dating (40Ar-39Ar) of vein adularia from Cerro Blanco returned a date of 4.93 ± 0.47 Ma.

## 7.3 Local Geology

The Cerro Blanco Gold Project deposit is a classic hot springs-related, low-sulphidation quartz-chalcedony-adularia-calcite vein system. It was localized along a structural corridor created during late Miocene-Pliocene tectonic extension within the active Central American Volcanic Arc. Deep penetrating faults and local bimodal igneous activity drove the Cerro Blanco hydrothermal system and formation of the gold deposit.

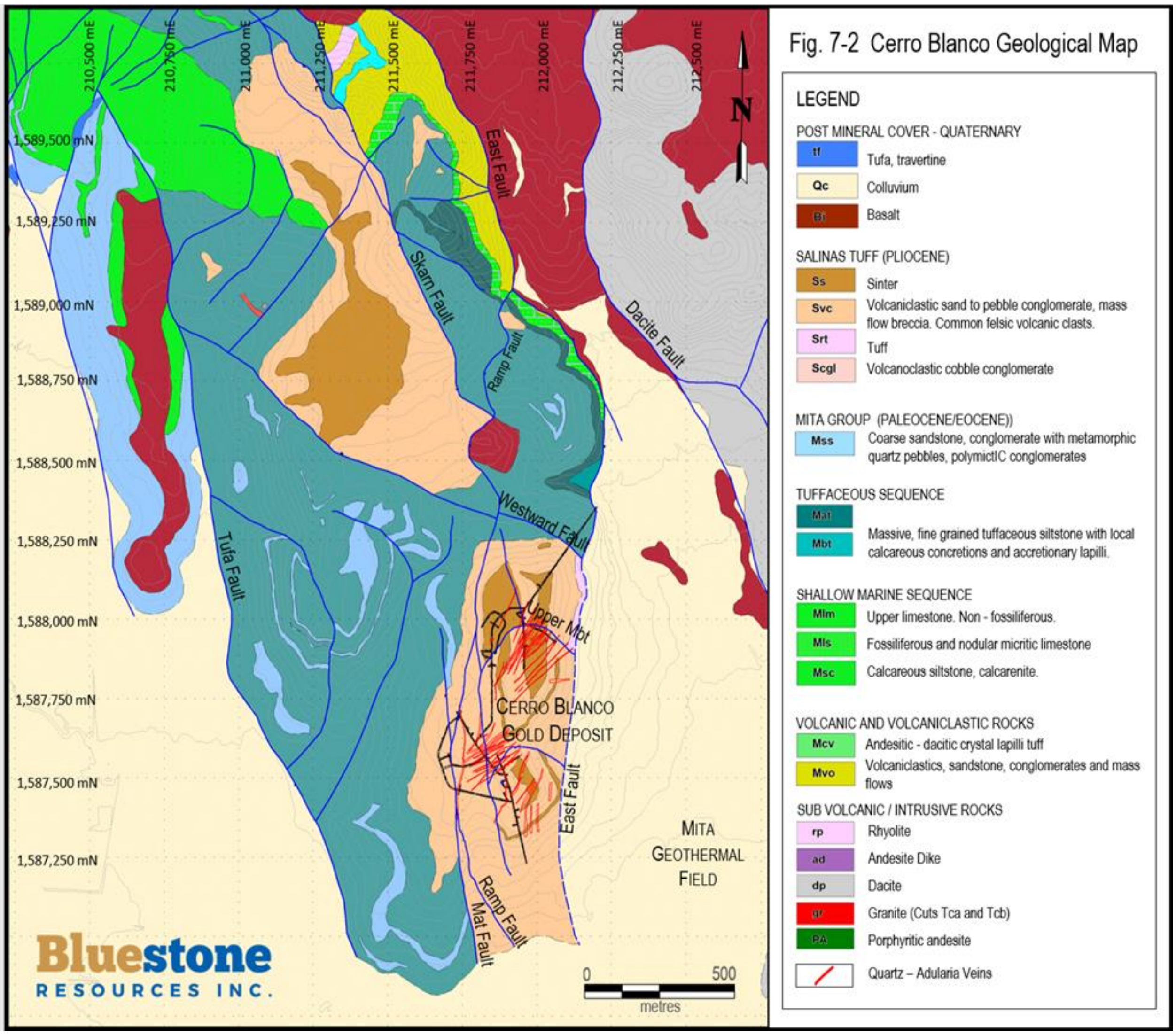
The Cerro Blanco Gold Project lies within the volcanic province with the principal rock units being Tertiary volcanics, volcaniclastics, and sediments, including ignimbrites, siltstone, limestones and conglomerates, that are intruded by andesitic and rhyolitic dykes. Recent basalt lava flows form the youngest rocks in the area in addition to locally derived volcanic sediments.

The gold- and silver-bearing veins and upper unit of silicified sediments (Salinas unit) occupy a north-trending graben bounded by a fault (termed the East Fault) representing a major structural feature that separates the main Cerro Blanco gold deposit from the Mita geothermal field immediately to the east.

To the north, the graben is concealed beneath Quaternary basalt flows, and to the south it is concealed by recent alluvium. Rhyolite/dacite domes underlie the extreme northeast potion of the district. Active hot springs occur immediately south of the Cerro Blanco hill.

Figure 7-3 shows a simplified geological map for Cerro Blanco.

**Figure 7-3: Geological Map of Cerro Blanco**

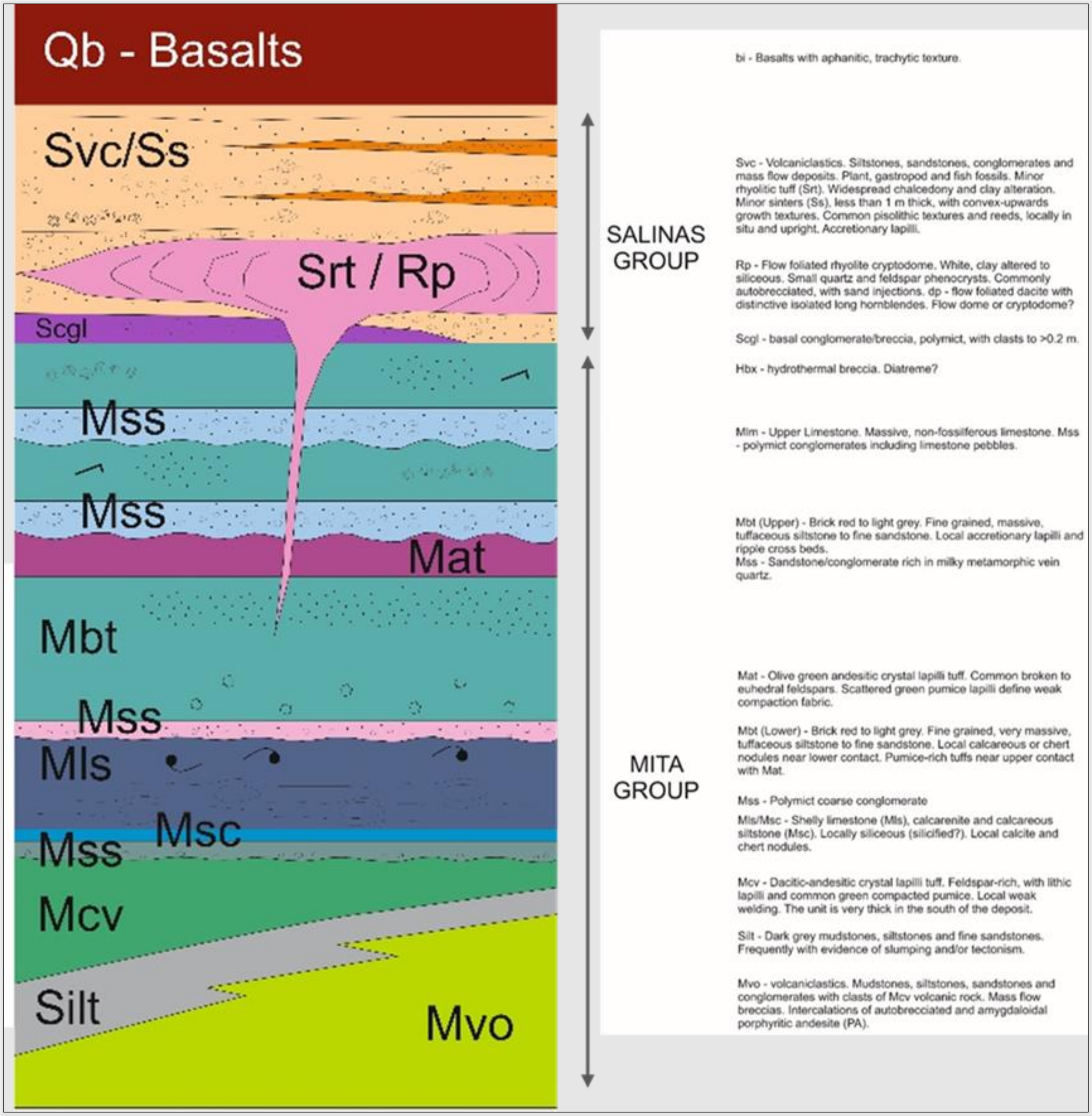



Source: Pratt and Gordon (2019).

### 7.3.1 Lithology

The oldest rocks at Cerro Blanco Gold Project, intersected in deep drill holes, belong to the Mita Group (Pliocene-Miocene). This group exhibits a great variety of volcanic and sedimentary rocks with important marker beds that are crucial for understanding the complex structural geology. Thicknesses seem fairly constant, with little evidence of growth faulting or internal unconformities during their accumulation (Figure 7-4).

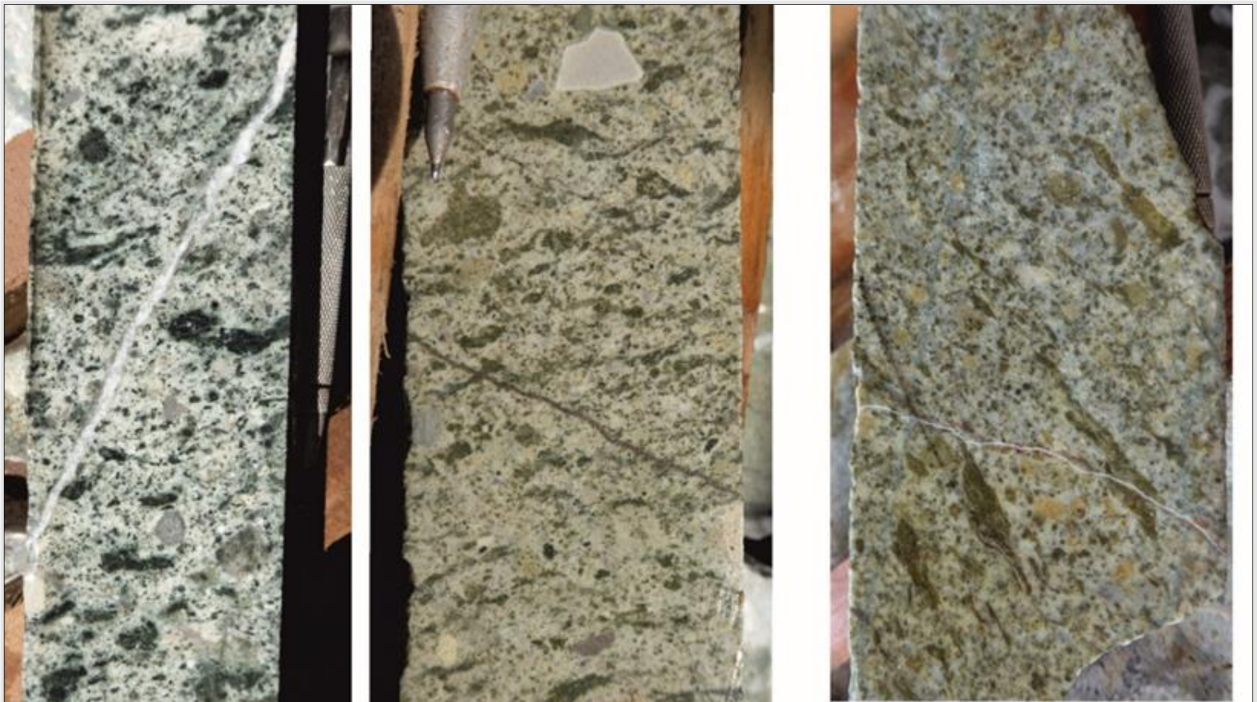
**Figure 7-4: Lithostratigraphy & Lithology at Cerro Blanco**




Source: Pratt and Gordon (2019).

The deeper parts of the Mita Group are dominated by volcaniclastic rocks (Mvo, mass flow deposits, conglomerates) with intercalated autobrecciated and amygdaloidal porphyritic andesites (lithology code PA). There is a distinctive unit of dark grey siltstones and fine sandstones (Silt), frequently with syn-sedimentary disruption. The sequence is capped by a major unit of andesitic-dacitic tuff (Mcv) (Figure 7-5), erupted in a single event. This is at least 50 m thick and rich in broken crystals and small pumice lapilli. It shows a weak compaction fabric or welding (refer to the photographs in Figure 7-5).

**Figure 7-5: Examples of Andesitic Lapilli Tuff (Mcv)**



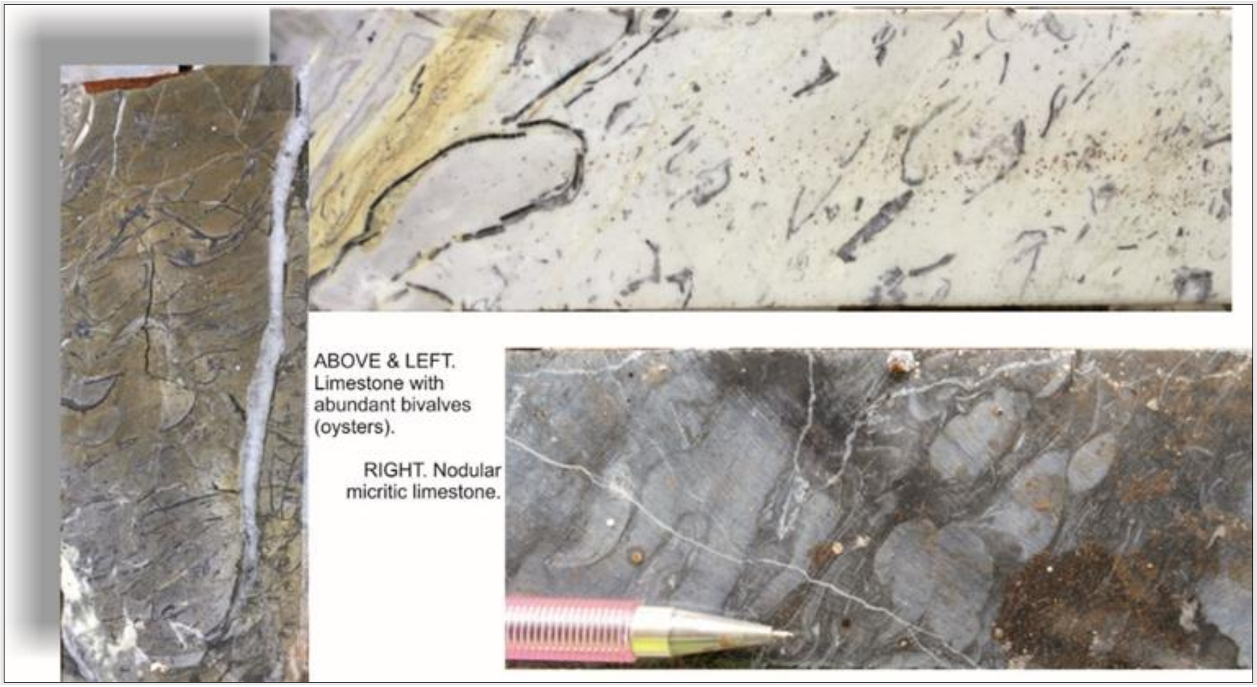

Source: Pratt and Gordon (2019).

The tuff is overlain by sandstones (Mss), followed by a nodular micritic to shelly, oyster-rich limestone (Mls, see Figure 7-6) which is the most distinctive rock at Cerro Blanco. This limestone sequence is about 20 m thick and includes calcarenites (Msc).

The limestone is overlain by a thick sequence of relatively massive, brick red to light grey siltstone and fine sandstone (Mbt). This distinctive rock has local accretionary lapilli, horizons of flaser and ripple cross-bedded fine sandstone, and local calcareous concretions. The Mbt sequence is divided into lower and upper parts by an andesitic crystal tuff (Mat). It is also punctuated by intervals of clean, well-sorted, fine-grained conglomerate (Mss). These can be rich in metamorphic vein quartz pebbles and dark grey schist, indicating a metamorphic hinterland. In the north part of the property there is a second major package of limestone (Mlm) (Figure 7-3), in turn overlain by further massive siltstones (Mbt).

The Mita Group is overlain by the Salinas Group (Svc). This is a complex sequence of interbedded plant-rich siltstones, mudstones, sandstones, conglomerates, mass flow deposits, phreatic breccias, and hot spring sinters. The Salinas unit, of probable Pliocene age, was previously considered to unconformably overlie the Mita unit, which was then assigned to the Eocene-Oligocene. The presence of the unconformity is certainly suggested by the structural culmination defined by the Mita limestone. However, thin sinter horizons are observed interbedded with siltstone at the top of the Mita unit, a situation that requires that the Mita and Salinas are part of a single, uninterrupted succession. This interpretation implies that the Mita part of the succession was in place before the mineralization commenced, whereas the overlying Salinas part accumulated during the mineralization event (Sillitoe, 2018).

**Figure 7-6: Examples of Limestones (Mls)**




Source: Pratt and Gordon (2019).

The syn-mineral Salinas unit is believed to have accumulated progressively in a low-relief graben characterized by a shallow groundwater table. The Salinas conglomerate was presumably derived by erosion of the flanking horst blocks as relief was created during the active faulting. The topographic inversion required to explain the current prominent position of the graben fill is ascribed to the silicic character of the Salinas unit and its consequent resistance to erosion.

Where the paleo-groundwater table intersected the paleosurface, siliceous sinter was precipitated - a situation that must have prevailed on several occasions for relatively protracted time intervals to produce the main sinter horizons. The presence of abundant reed casts in the sinter shows that its formation encroached on marshy ground (Figure 7-7). Where the paleo-groundwater table was several meters below the paleosurface, conglomerate in its immediate vicinity was silicified and the vadose zone above it was subjected to steam-heated alteration. The steam-heated alteration, containing cristobalite, kaolinite and possible alunite (an advanced argillic assemblage), was the product of acidic solutions formed by the condensation of ascendant $H_2S$-bearing steam into downward-percolating groundwater. The overall result is an interlayered sequence of sinter, silicified conglomerate and steam-heated alteration (Figure 7-7).

The Salinas Group is characterized in the mineralized area by widespread chalcedonic alteration, which can make identifications difficult, and elsewhere by strong clay alteration. In some places rock fragments have concentric chalcedony coats (pisoliths), implying they accumulated in a hot spring pool. Silicified reed fragments are common and locally upright, in original growth position. Rare gastropods were observed.

**Figure 7-7: Silicified Reed Fragments**

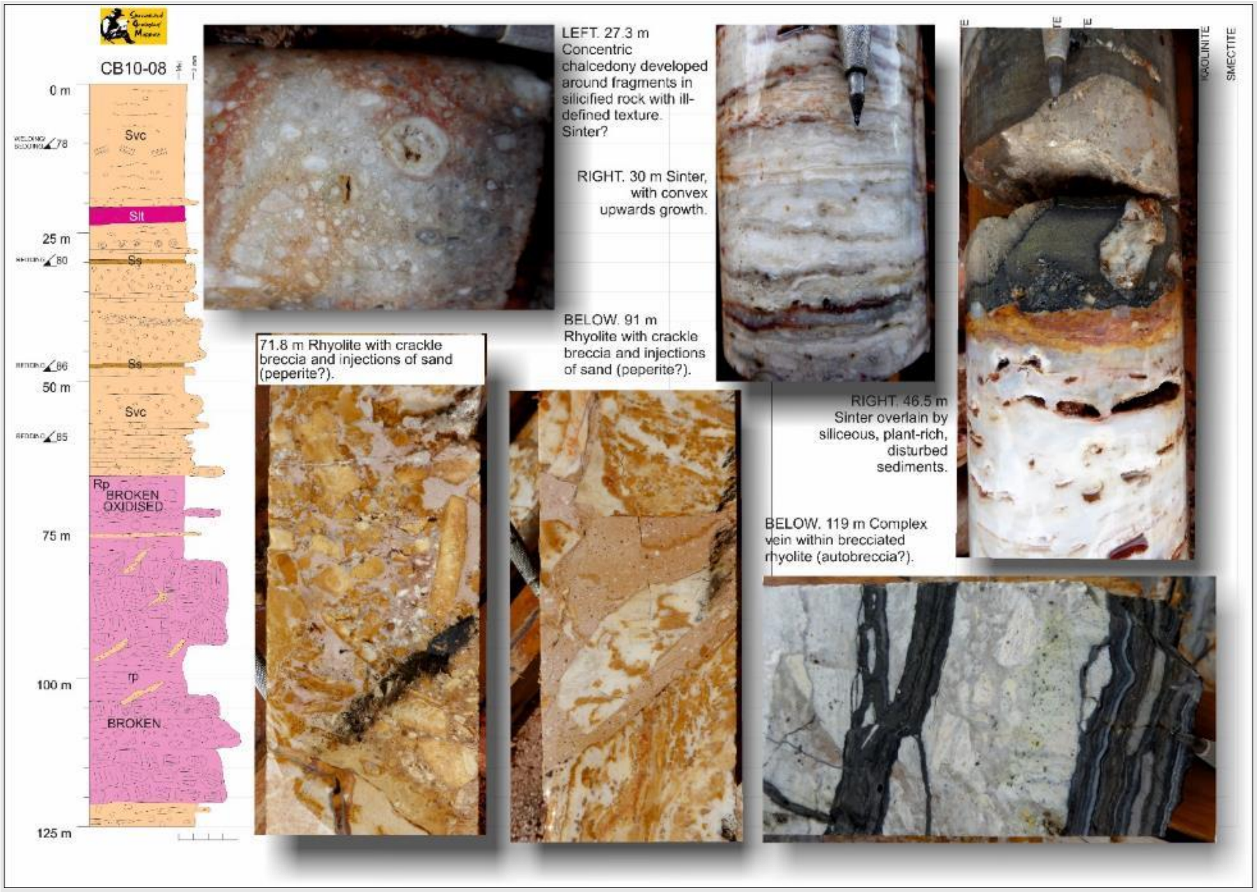
Source: Pratt and Gordon (2019).

The sequence also includes rhyolitic tuffs and a rhyolite cryptodome / flow dome (Rp), both with bipyramidal, embayed quartz crystals. A dacite cryptodome or flow dome (dp) also crops out around the Cerro Blanco village and is observed in drill holes in the hanging wall of the East Fault (Figure 7-4). It has no quartz crystals, but distinctive, isolated, long hornblende phenocrysts. Sediment dykes, common in geothermal districts, where they form the feeders to sand and mud volcanoes, are common in the Salinas Group.

A typical log of the Salinas Group, shown in the photographs in Figure 7-8, includes a body of rhyolite, possibly a cryptodome since probable peperites were seen at the contacts.

The highest stratigraphic part of the Salinas Group, at least 60 m thick and above the sinters, is cut in the graben in the hanging wall of the East Fault. It comprises lacustrine siltstones and volcaniclastic sandstones. The rocks are plant-rich and contain rare fish fossils and brine shrimp/ostracods (e.g., drill hole CB332).

**Figure 7-8: Example Drill Log from the Salinas Group**




Source: Pratt and Gordon (2019).

The Salinas Group includes common mass flow or hydrothermal breccias. Their geometry is frequently unclear; it is uncertain if they are dykes or aprons of phreatic (explosion) breccia ejected from hot springs. Some contain sinter clasts, confirming phreatic eruptions. Underground, the South Ramp is dominated by hydrothermal breccias (Hbx), with polymict clasts up to 0.5 m diameter. This may be the north margin of a south-dipping diatreme. Successive cross-sections show it extending progressively deeper towards the south.

Quaternary basalts (bi), with a felted, trachytic texture, crop out in the north of the Cerro Blanco property and occur in the low graben either side of the horst. They are clearly lava flows. Around the village of Cerro Blanco, they in-fill the paleo-topography formed by a large dacite flow dome. It is unclear if this topography is erosional or is the original hummocky shape of the dacite flow. The basalts include flow foliated and autobrecciated types.

The youngest rocks comprise alluvium and, in a few places, modern travertine and tufa occur at springs around the flanks of Cerro Blanco hill (Figure 7-9). The tufa cements colluvial blocks of the siliceous sinter (Salinas Group) are modern and should not be confused with sinter. They imply probable karst formation and dissolution of limestone.

**Figure 7-9: Recent Travertine Exposure**

Source: Pratt and Gordon (2019).

Discordant igneous intrusions are rare at Cerro Blanco, but a few thin rhyolite (Rp) and aphanitic andesite (ad) dykes are observed.

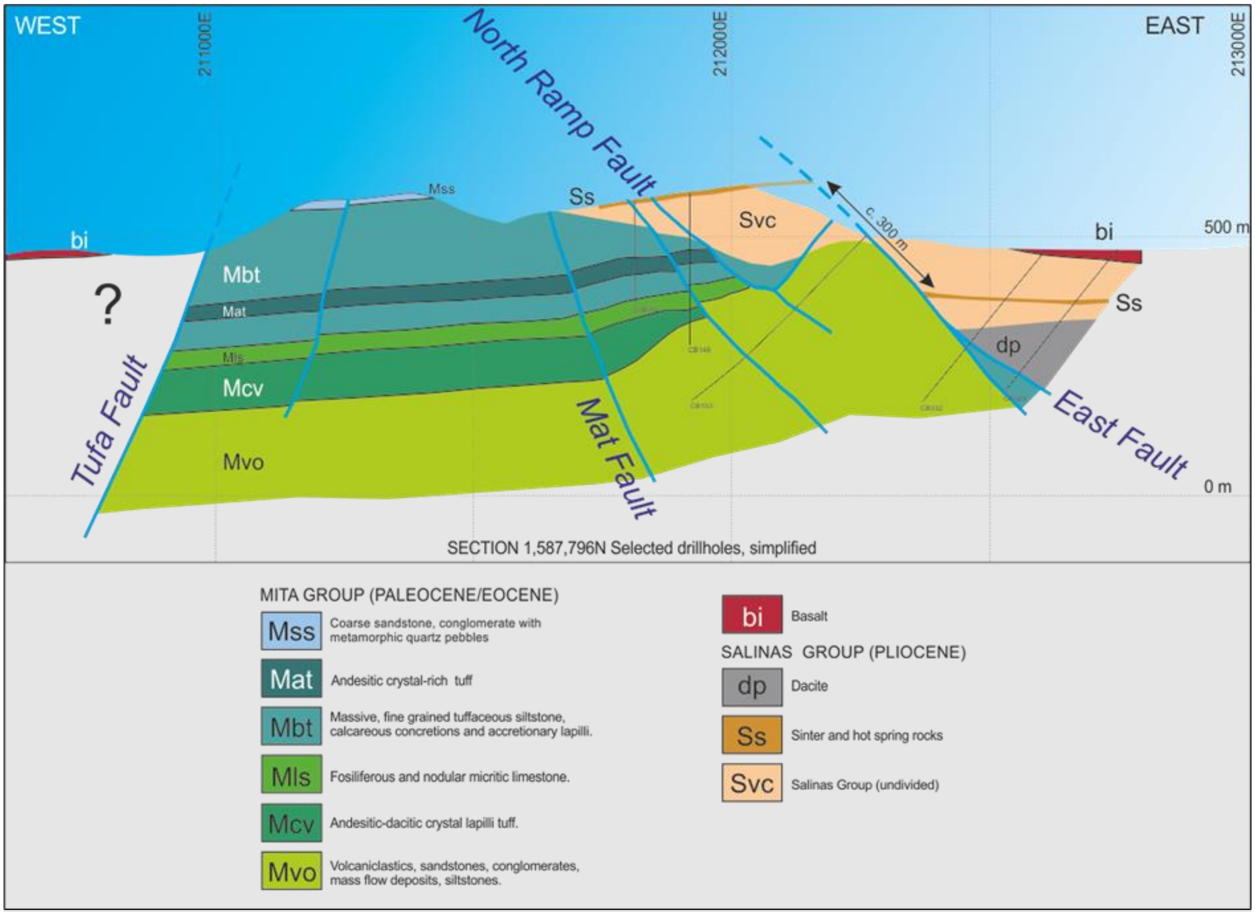
### 7.3.2 Structure

The gold mineralization at Cerro Blanco Gold Project is hosted within a broadly north-south-striking graben. The East Fault (Figure 7-10), also referred to as the “East Horst Fault” in previous studies, is cut by several drill holes and observed in drill core as a broad zone of post-mineral cataclasite developed in Mita siltstone; however, the structure appears to control a linear rhyolite body, suggesting that it was also active during the mineralizing event. This fault may be listric and made up of several strands. Holes CB332 and CB329, in the section below, show narrow wedges of ‘exotic’ lithologies along the fault zone, including limestone

(Mls) and conglomerate (Mss). The apparent displacement, shown by the offset of the sinter (Ss), is about 300 m.

The immediate footwall of the East Fault, hosts to the gold-bearing quartz veins, is structurally complex. A deep geothermal drill hole (MG-07) shows gold mineralization in the probable down-dip extension of the East Fault, at 634-640 m downhole depth.

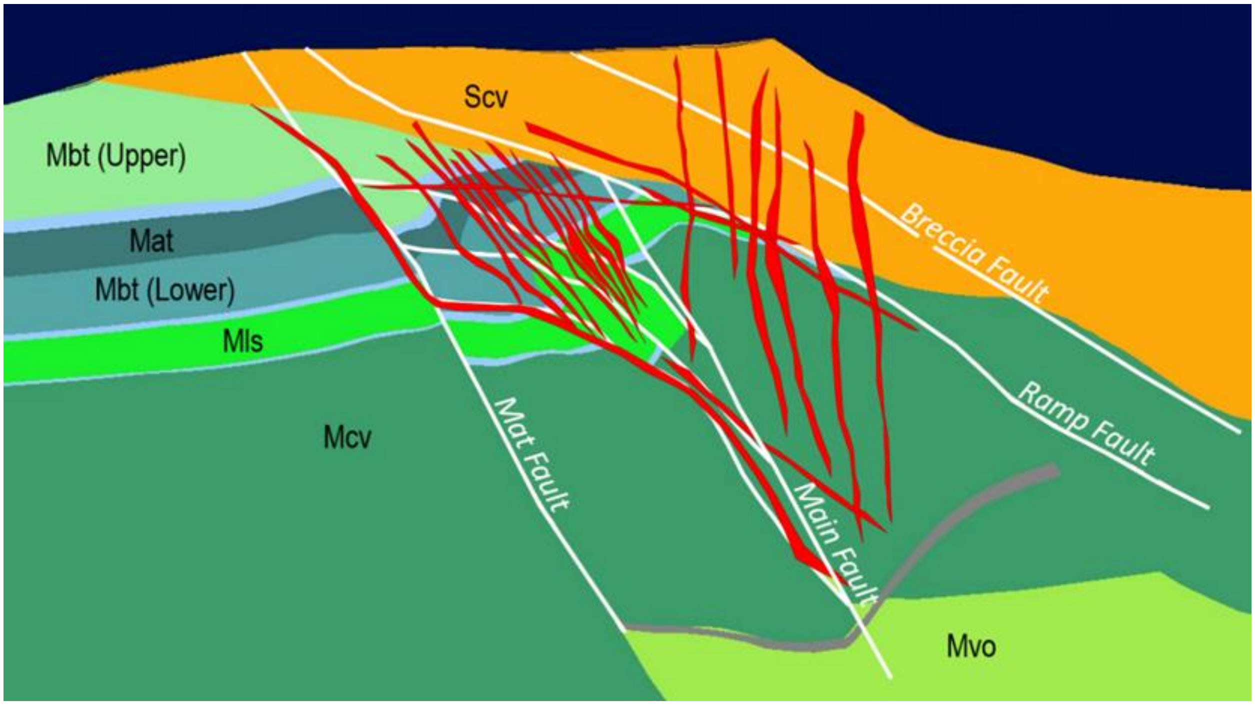
**Figure 7-10: Simplified West-West Cross-Section across Cerro Blanco**




Note: Many drill holes and some lithostratigraphic units and faults were omitted to conserve clarity. Source: Pratt and Gordon (2019).

The Cerro Blanco property has a complex history of faulting. The structural control on mineralization is unusual for a low-sulphidation epithermal vein deposits, which normally comprise a single, relatively continuous vein. At Cerro Blanco there are sheeted vein swarms that resemble a duplex. Figure 7-11 indicates the typical complexity in an east-west section. Note that the thickness of the Mcv west of the Mat Fault and the Mvo to the east is an artifact of Leapfrog software and is overstated. Veins are shown in red, faults in white.

**Figure 7-11: East-West Cross-section of the South Zone, Cerro Blanco looking North**




Source: Bluestone (2021).

Simplistically, the structural history is comprised of the following:

1. Sedimentation of the Mita Group in a basinal to shelf environment, with periodic incursions of calc-alkaline volcanism (mostly waterlain andesitic tuffs and andesite flows, and their volcaniclastic equivalents). Some of the beds appear turbiditic (silt), implying moderate water depth. Some metamorphic clasts imply a metamorphic hinterland.
2. A compressive episode formed a series of broadly north-south-striking, west-verging folds, cored by Mita Group rocks, in particular, the Mvo and Mcv. These folds were associated with west-verging reverse faults and resulted in local overturned limbs. There may have been a component of strike-slip, with development of a positive flower structure at the restraining bend in a major north-south strike slip fault. There is evidence that most of the gold-bearing veins developed at this stage. The controlling structures for the vein swarms are in the footwall of the East Fault and apparently steeper (e.g., the Main Fault, see Figure 7-10).
3. Major extensional faulting with downthrows to the east of up to several hundred meters. These include the Ramp and East faults (see above). These faults may have been active during deposition of the Salinas Group (Svc), possibly growth faults. Metamorphic clasts in the Salinas Group implies continued input from a metamorphic hinterland. Offset of Quaternary basalts implies that the faults may still be active (neotectonic). These faults have the greatest surface expression, reflected in the modern topography by the Cerro Blanco ridge and flanking low relief alluvial plains.

Most of the gold-bearing veins are constrained between the Mat Fault in the west and the East Fault and evidence suggests that most veins at this stage developed along early pre-mineral faults. The Mat fault is interpreted to be a major early structure and hosts the principal footwall vein (VS-101) in the South Zone for some of its length. Lack of continuity of major veining up into the Salinas suggests that much of the faulting had ceased by the time of the Salinas deposition, except for the Cross, Ramp and East Faults.

Some of these faults may represent syn-volcanic growth faults typical of near-surface epithermal settings that represent shallow, low displacements that manifest as larger pre-mineral faults at depth with increased displacements.

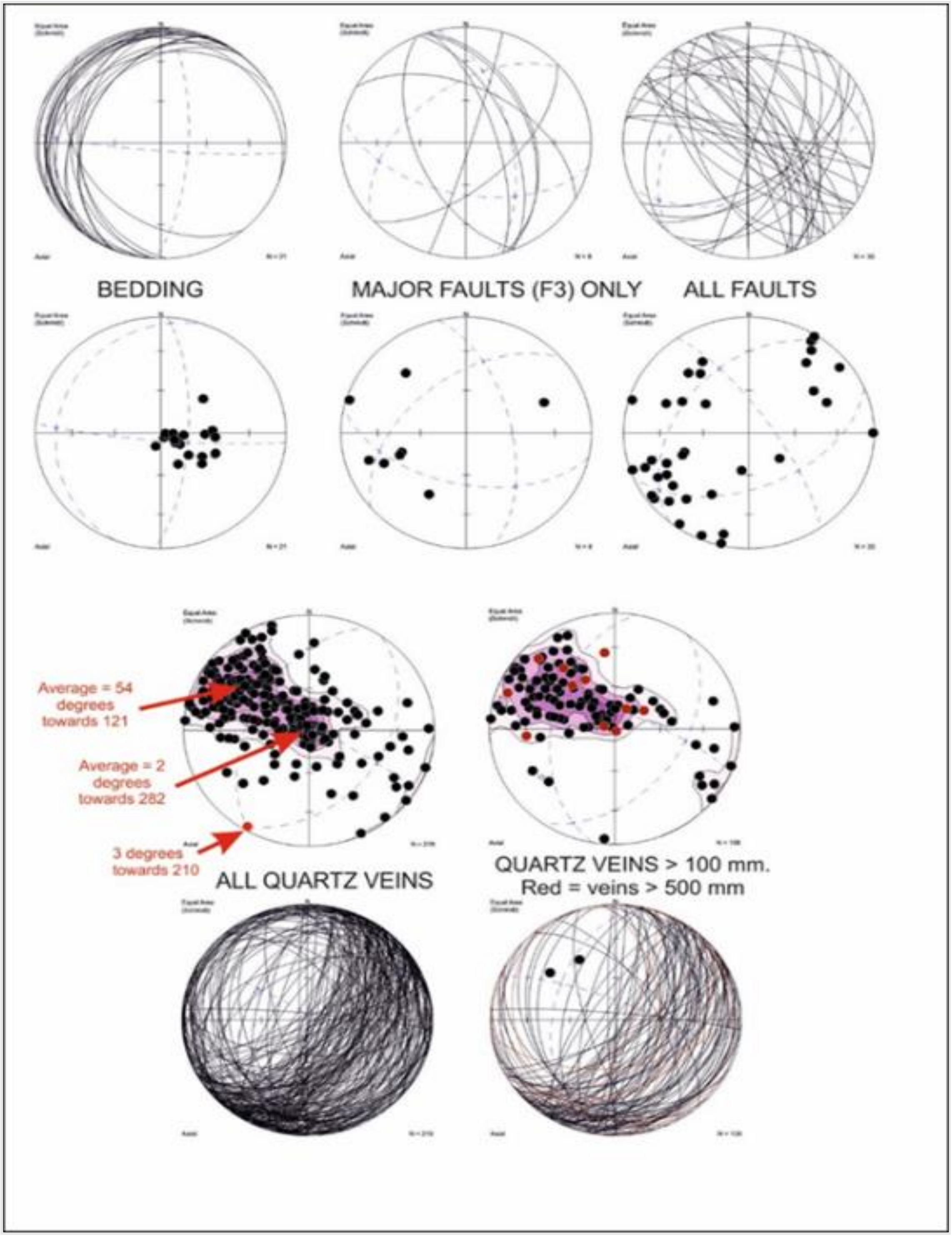
In the southeast portion of the South Zone, narrow sub-vertical gold-bearing veins extend into the Salinas and possibly represent progression from the early compressive to more extensional conditions by the end of the Salinas deposition. Drilling demonstrates that a large chunk of stratigraphy is missing in the area separating the north and south zones of the deposit. This comprises the Mls + Mbt (lower) and Mat. A northwest-striking, southwest-dipping fault ("Upper Mbt Fault") is inferred. It is unclear if this terminates into the major Ramp Fault or vice versa. The throw on the Upper Mbt Fault seems to decline towards the north and the stratigraphy is increasingly preserved in the footwall. Together, the Ramp and Upper Mbt faults define a triangular-shaped block that seems to have slid out southwards. Explaining the geometry, in terms of tectonic regime, is difficult, but a reactivated, extensional flower structure is one possible explanation.

Faults are difficult to map underground and in drill core because they are largely quite narrow (centimeter scale) and 'sealed' by silica; they generally do not form the zones of poor rock quality that typify post-mineral faults (though there are exceptions, for example along the East and Cross faults). This is reflected underground by the general lack of wall rock support. Figure 7-12 shows structural measurements from the underground workings for faults and veins. However, most understanding of the principal faults comes from 3D modelling, based on offsets of the lithostratigraphy and the marker beds.

The underground workings display numerous swarms of quartz veins. There are examples of conjugate veins and veins refracting through different lithologies (competency control). Examples are shown in Figure 7-13.

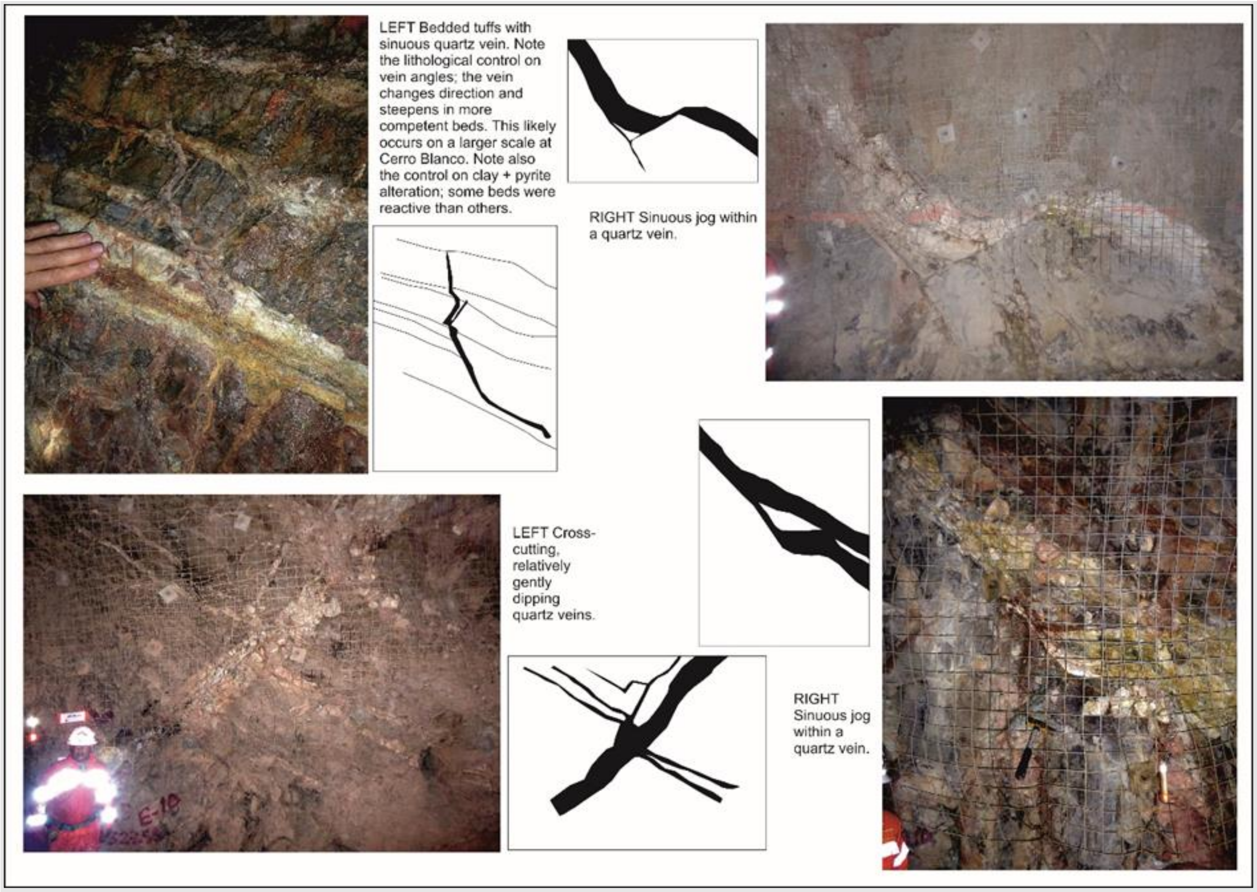
The gold-bearing veins at Cerro Blanco are focused in the footwall, to the west, of the steep Main Fault (also referred to as the Main Zone); in particular, they are concentrated in the uplifted blocks and west-verging folds of basement volcanic rock (Mcv and Mvo). The Upper Mbt lithostratigraphic unit seems to have been less favourable for veining, explaining the relative gap in veining between the North and South ramps. Likewise, the veins tend to pinch out in the Salinas Group (though some do make it to surface and carry low grades).

**Figure 7-12: Stereograms (equal area) Showing Poles & Great Circles for Faults & Veins**




Note: All measured underground. Dots on the great circle plots represent slickensides. Source: Pratt and Gordon (2019).

**Figure 7-13: Photographs with Sketches of Veins Exposed Underground**



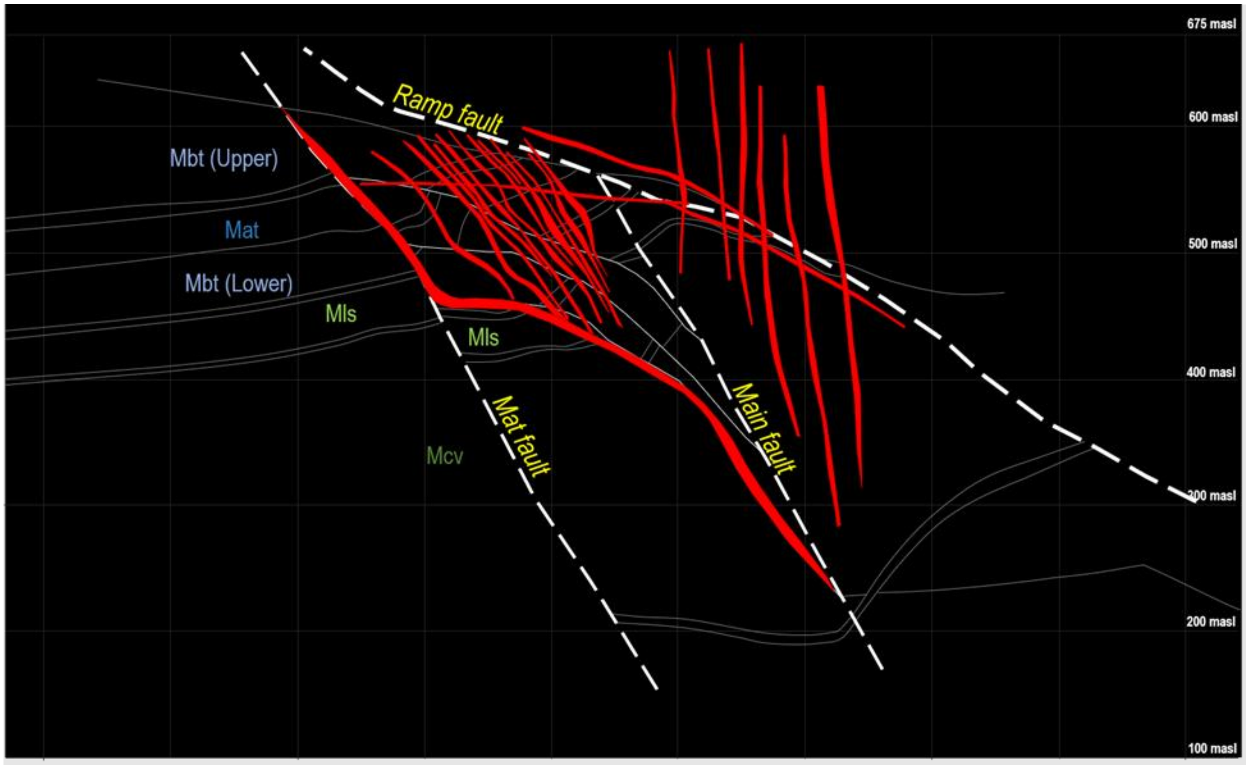

Source: Pratt and Gordon (2019).

In section view, the veins clearly form lozenge-like duplexes and sheeted swarms, one in the South Ramp, the other in the North Ramp. Figure 7-14 is a cross-section across the South Ramp. Vein wireframes were generated in Leapfrog using core logging, alpha angles (angle between core axis and vein) in non-oriented drill core, assay data, and underground mapping. They show a distinct branching and converging of relatively shallow veins into a steeper zone (Main Fault). Most veins are also constrained to the footwall of the Ramp Fault and the hanging wall of the steeper Mat Fault.

Sheeted veins and lozenge-shaped duplexes are also obvious in map (plan) view. Figure 7-15 shows a series of horizontal slices at different elevations. The gap between the South and North resource areas mostly comprises the triangular wedge Upper Mbt stratigraphy between the Upper Mbt and Ramp faults. This seems to have been unfavourable for veining.

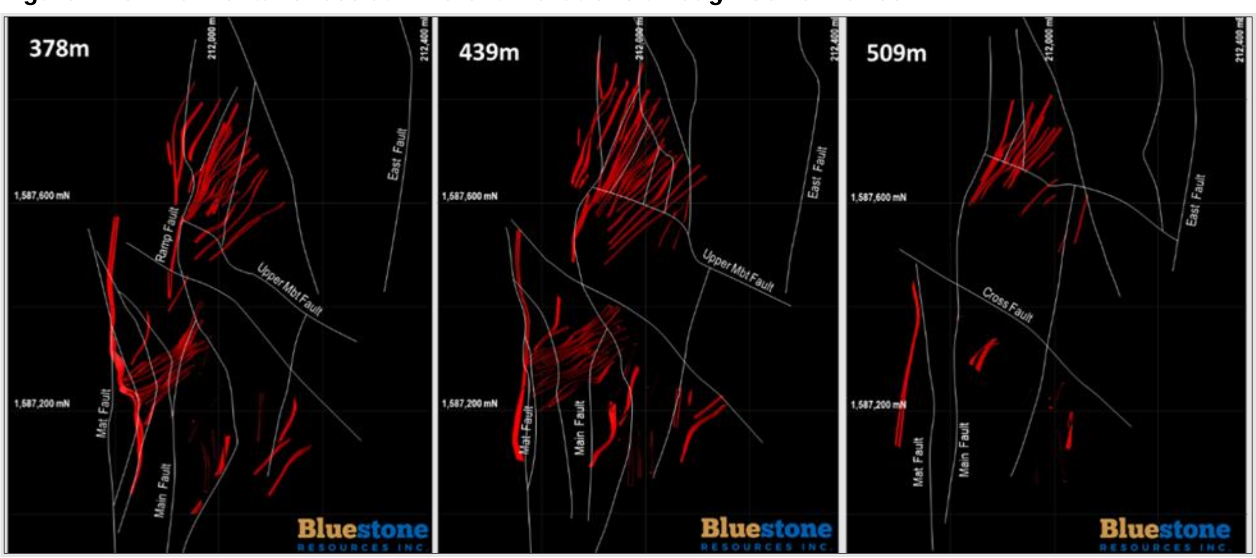
Underground mapping supports the 3D modelling; it shows a similar steepening and converging of veins into the Main Fault / Zone. Individual veins become thicker and more closely spaced along the Main Fault. The way individual veins swing into and intersect with the Main Fault creates ore shoots that plunge approximately 30° south.

**Figure 7-14: Annotated, Vertical East-West Cross-Section across the South Ramp (looking North)**




Source: Bluestone (2020).

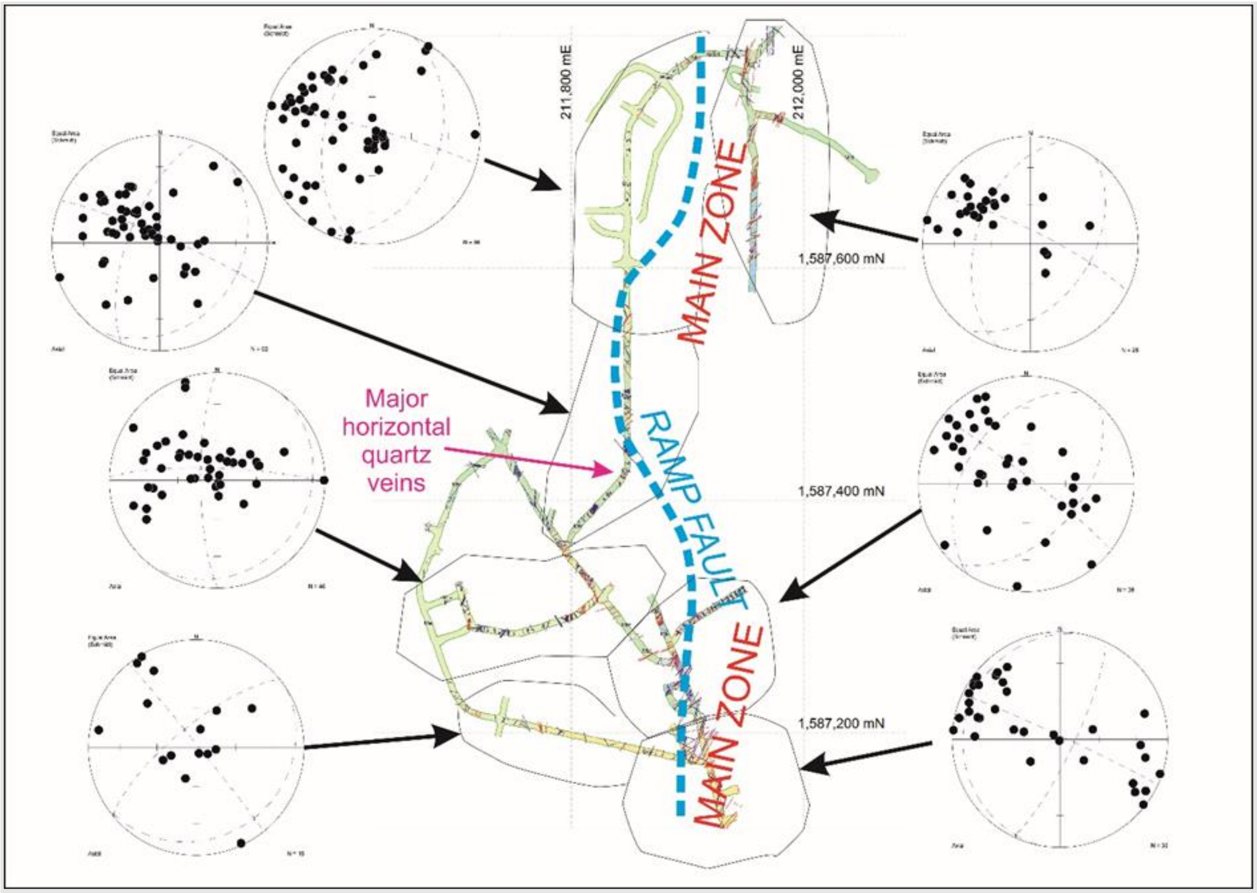
**Figure 7-15: Horizontal Slices at Different Elevations through Cerro Blanco**




Note: North is up. Red – veins; blue – faults. Source: Bluestone (2020).

There are some secondary (conjugate) vein directions, but stereograms for sub-areas (Figure 7-16) show consistent patterns: steeper veins are mostly in the east and shallow veins in the west. In particular, a swarm of thick, sub horizontal veins occur in the immediate footwall of the Ramp Fault. The cumulative thickness of the veins exceeds 3 m. The flat veins clearly imply reverse (compressive) movement on the Ramp Fault. Clearly, the major faults played an important role in partitioning vein development.

**Figure 7-16: Stereograms for More Detailed Sub-Areas in Underground Mapping**




Source: Pratt and Gordon (2019).

The stress regime during vein formation can also be calculated from conjugate veins. The stereogram for all quartz veins measured underground shows the intersection between the two principal vein directions is sub-horizontal). The dominant extension direction seems to have been vertical, which is highly unusual; epithermal veins generally develop during horizontal extension. The predominance of horizontal veins in the west supports the idea of vertical extension.

Field observations, 3D modelling, and stereograms therefore imply that the veins developed during compression, rather than extension, at least in the initial stages of mineralization. This fits with the overall compressional geometry of the west-verging folds and reverse faults, later reactivated as normal, extensional faults. recently discovered steeply dipping/vertical veins in the hanging wall of the south zone possibly record this change from a more compressional to extensional regime during the latter part of the mineralizing event. As some steep veins cut the Salinas Group and the sinters are contemporaneous with hydrothermal activity, this suggests that the hydrothermal / geothermal activity spanned the change from compressional to extensional tectonics.

## 7.4 Deposit Geology

The Cerro Blanco deposit is a classic hot springs-related, low-sulphidation quartz-adularia-calcite vein system. It is localized along a complex fault intersection created during late Miocene-Pliocene tectonic extension within the active Central American volcanic arc. Local igneous activities that drove the Cerro Blanco hydrothermal system include a vesicular andesite dike swarm and mineralization stage rhyolite / dacite flow dome eruption and cryptodome intrusion.

The Cerro Blanco vein systems are best developed (widest and most continuous) between the 300 masl to 500 masl elevation ranges. Principal host rocks include a lithic tuff – calcareous shallow marine-volcaniclastic sequence and, to a lesser extent, the overlying volcaniclastic-hydrothermal breccia sequence of probable Pliocene age. Vein zones often appear to transition to barren calcite beneath the ±300 m elevation in the northern half of the deposit. To the south, high-grade quartz-adularia-calcite vein zones continue at least another 100 m down to 200 m elevation. Some veins remain open at depth.

Massive chalcedonic silicification, referred to as a “silica cap”, dominates the conglomerates of the Salinas unit. Silica-flooded volcaniclastics and phreatic breccia are interbedded with chalcedonic silica sinter from the present surface to depths of ±100 m. Silicification also occurs in the underlying Mita as irregular envelopes, up to several meters wide, around the main veins as well as in the upper part of the limestone horizon as jasperoid. The red-bed siltstone is partially bleached and altered to a grey green, illite and smectite-bearing rock. Chlorite, in addition to illite and smectite, is a prominent alteration mineral in the ignimbrite where it is concentrated in the fiamme.

Wall rock alteration to a large extent determines geotechnical rock hardness and presents contrasting resistivity and electrical chargeability characteristics that could be exploited across the district in the search for new gold occurrences beneath thin colluvial or basalt cover.

## 7.5 Mineralization

The Cerro Blanco gold deposit occurs within a large hydrothermal alteration zone covering an area of about 5 km long and 1 km wide. This zone exhibits the effects of strong, pervasive hot spring type hydrothermal alteration.

Gold mineralization is hosted within a broadly north-south striking sequence of westerly-dipping siltstones, sandstones, and limestones (Mita Group) that are capped by silicified conglomerates and argillaceous sediments with contemporaneous dacite / rhyolite flow domes or cryptodomes (Salinas Unit). The Salinas rocks are syn-mineral and believed to have accumulated progressively in a low-relief graben characterized by a shallow groundwater table. The Salinas conglomerate was presumably derived by erosion of the flanking horst blocks as relief was created during active faulting. The topographic inversion required to explain the current prominent position of the graben fill is ascribed to the silicic character of the Salinas unit and its consequent resistance to erosion.

The west and east sides of the Cerro Blanco ridge consist of flat agricultural plains characterized by Quaternary basalts, interbedded with boulder beds and sands. These rocks also appear down-faulted to lower elevations, implying major post-mineral extensional movements on such faults; and they may be neotectonic (active).

The current gold resource occurs under a small hill and is confined within an area about 400 m x 800 m. Gold and silver occur almost exclusively in quartz-dominated veins of low-sulphidation epithermal origin and in low-grade disseminated mineralization within the Salinas conglomerates and rhyolites. Highest grades are hosted by high to low angle banded chalcedony veins, locally with calcite replacement textures.

Gold-bearing structures in the Cerro Blanco Gold Project area extend 3 km to the northwest of the gold deposit and occur largely confined within the hydrothermal alteration zone. Exposures are poor and locally covered by alluvium and post-mineral rocks. Gold-bearing structures extend at least 1 km south and southwest of the deposit under valley fill and post-mineral rocks.

Geothermal well MG7, located about 0.5 km east of the deposit, encountered a 27 m zone averaging 6.3 g Au/t and 22 g Ag/t at a depth of 634 m. The upper 6 m of this zone averages 23.9 g Au/t and 79 g Ag/t. Although the geometry is uncertain and the sampling methodology of the drill cuttings cannot be determined, possibly this vein material was caught up in a fault crush zone / splay within the East Fault (much like the other exotic lithologies seen within the fault zone), or conversely, represents a separate mineralized system distinct from the main deposit.

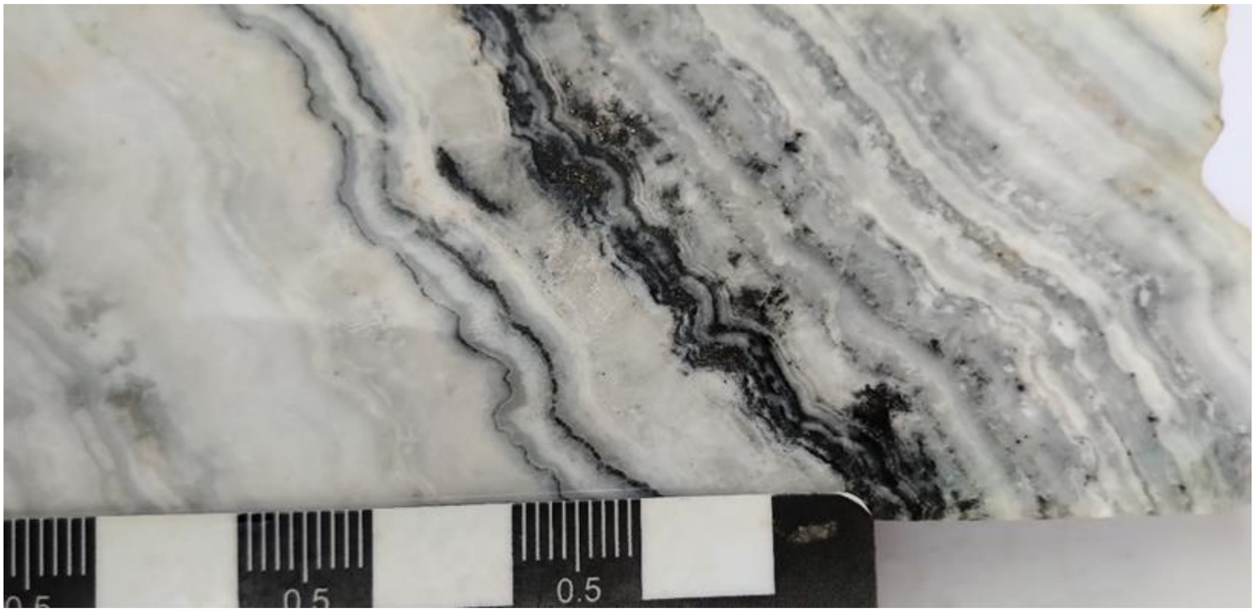
### 7.5.1 Vein Zones

Petrographic descriptions of four vein zones, by Economic Geology Consulting (Thompson, et al., 2006), concluded that the veins consist of crustiform banded chalcedony, quartz, adularia, calcite, sulphides, and visible gold. The samples represent a range of almost 300 m in elevation. Bladed calcite or pseudomorphs after bladed calcite (lattice blade texture) were observed in all four samples. Bladed calcite is a rapid depositional texture, common when calcite precipitates from boiling fluids. A wide variety of recrystallization textures in quartz and chalcedony may also indicate changing fluid conditions and periodic boiling. Figure 7-17 shows a high-grade intercept in drill hole CB-20-430 with banded chalcedony-adularia-acanthite and visible gold that assayed 144 g Au/t and 282 g Ag/t.

Observations suggest that mineralization occurred as one principal multi-stage event as banded vein material, dominated by cryptocrystalline and originally amorphous silica phases (jigsaw quartz and chalcedony) characteristic of both the north and south zone vein swarms. Colloform banding with gel-like precursor textures are common, and observations from drill core suggest that banding is characteristic of high-grade zones, with coarser crustiform and crystalline bands more associated with lower-grade veins. Higher grades are associated with fine-grained (<100 µm) electrum, kustelite and acanthite concentrated in bands of fine- to very fine-grained jigsaw quartz (crystallized amorphous silica, Albinson, 2019). Gold-silver minerals are accompanied by rare presence of tetrahedrite and chalcopyrite.

Repetitive "crack and seal" pulses and associated boiling/flashing events very close to the paleosurface are suggested as the main mechanism for precious metal deposition. The higher-grade, often bonanza grade core intersections with coarser and more abundant sulphides, electrum and free gold appear to represent an earlier series of events. Multistage banding can be very finely repetitive down to 5 to 10 µm widths for individual bands. Soft sediment-type deformation is visible commonly in the bands with mamillary colloform bands deformed into flame-like textures due to deformation of the bands by turbulent fluid flow. Sulphides and electrum are present mainly in the fine- or very fine-grained jigsaw quartz bands. Adularia-rich bands are not easily visible with the hand lens and are very fine grained.

**Figure 7-17: High-grade Drill Hole Intercept Hole CB20-430 – 144 g/t Au, 282 g/t Ag (227.3 to 228.9 m)**




Source: Bluestone (2020).

Lack of inter-stage hydrothermal brecciation and coarse-grained primary quartz textures suggest that the mineralizing event was a fairly short-lived event that occurred very close to the paleosurface. The lack of post-mineral structural displacement of veins and distribution of high grades over a +300 m vertical profile, attest to the pristine nature of the veins.

Underground observations include the following:

- Vein zones are best developed throughout the model between elevations of 300 and 500 m. This elevation range roughly coincides with the Mcv contact beneath and the Salinas contact above. Thus, the principal host rocks are the Mita Group sandstones, calcareous sediments and overlying tuffs.
- The quartz veins at Cerro Blanco occur mainly within Mita sediments and Mcv tuffaceous rocks. These moderate to steep veins are associated with a subsidiary conjugate set of low-angle veins. The majority of veins appear to stop at the Salinas contact, with the exception of sub-vertical veins in the southeast part of the south zone that cut the Salinas and continue to surface.

- Vein zones occur as two upward-flared arrays that appear to converge downwards and merge with basal master veins around the contact with the Mcv. The south zone vein array is the better formed
- High gold grades locally persist at least down to the 200 m elevation, notably in the southern third of the model where at least one vein merges with the main footwall feeder structure.
- In several locations north of 1,587,400N, drill holes pass beneath high-grade quartz veins but encounter only massive barren calcite. This is an indication that the bottoms of productive veins have been found at those locations. Within vein zone envelopes, individual veins do not form a random stockwork, but tend to run parallel or sub-parallel to the main structural trends.

The definition of economic mineralization depends on the vein thickness, grade, and spacing. The structural control on the veins is discussed above. Most individual veins exposed in the underground workings do not exceed 1 to 2 m; much thicker veins, up to 7 m width, do appear in the vicinity of the north zone ramp (Figure 7-18) and in deeper levels of the south zone. Closely spaced veins or zones of convergence form wide zones of high-grade mineralization (Figure 7-18).

**Figure 7-18: View of Veins VN-05, 06, 07 in the North Ramp Underground Workings**

Note: Section assayed 20.4 m grading 18.9 g Au/t and 33.2 g Au/t. Source: Bluestone (2020).

Figure 7-19 shows vein textures associated with gold mineralization; they include bladed calcite, a classic indicator of boiling fluids, subsequently replaced by quartz or leached to give a skeletal framework. Other classic textures include crustiform banding, bands of cream-pinkish euhedral adularia, and quartz with minor dark grey silver sulphides / sulphosalts.

Inspection of vein textures suggest that gold and silver were introduced as one major event of multistage finely banded veining (originally amorphous silica) with subordinate bands of platy calcite that is mostly pseudomorphed to cryptocrystalline silica phases.

**Figure 7-19: Examples of Vein Textures from Cerro Blanco**


Source: Bluestone (2020).

Many veins and siliceous rocks (rhyolite / dacite) at Cerro Blanco display siliceous mudstone / sandstone dykes. There are also common geopetal structures, late cavities filled by horizontally banded siliceous sediments of hydrothermal origin mixed with vein gangue (Figure 7-20). These “fossil spirit levels” indicate proximity to the paleosurface and are confirmed by the presence of sinter immediately above.

It is unusual to see epithermal veins developed immediately beneath sinter, although other examples do exist (e.g., McLoughlin, California), implying the topography at the time of mineralization was low and the water table was very high. This is supported by the presence of accretionary lapilli in the Salinas Group and Mbt siltstones; they are typical of wet phreatic-dominated eruptions and pyroclastic surges. Diatremes and rhyolite flow domes are also typical in this environment.

In summary, the principal control on gold mineralization at Cerro Blanco was probably the boiling level in a hydrothermal system. The best grades are associated with boiling textures. At many low-sulphidation epithermal deposits, the vertical interval of economic grade is restricted to the former boiling level. This can be less than 100 m. These boiling levels form flat ore shoots. There are occurrences of high gold grade down to 640 m (downhole depth) in a geothermal hole (MG-07).

**Figure 7-20: Example of Geopetal Structure**

Source: Pratt and Gordon (2019).

### 7.5.2 Disseminated Mineralization

The Salinas unit shows widespread and low-grade disseminated gold mineralization associated with weak to strongly silicified polymictic conglomerates and altered rhyolite breccias and flows. Mineralization grading 0.2 to 2 g/t Au is pervasive and present in variably silicified bedded conglomerates and appears to be driven by intrusive rhyolite dykes and breccias (Figure 7-21). Locally, parts of the base of the Salinas are marked by an aphanitic rhyolite body, probably a cryptodome given it is underlain by narrow rhyolite dykes. The thicker Sinter horizons do not contain significant gold values, nor do strongly argillic-altered lithologies and fault gouge zones.

**Figure 7-21: Salinas Unit – Examples of Disseminated Mineralization Rock Types, Salinas Unit**

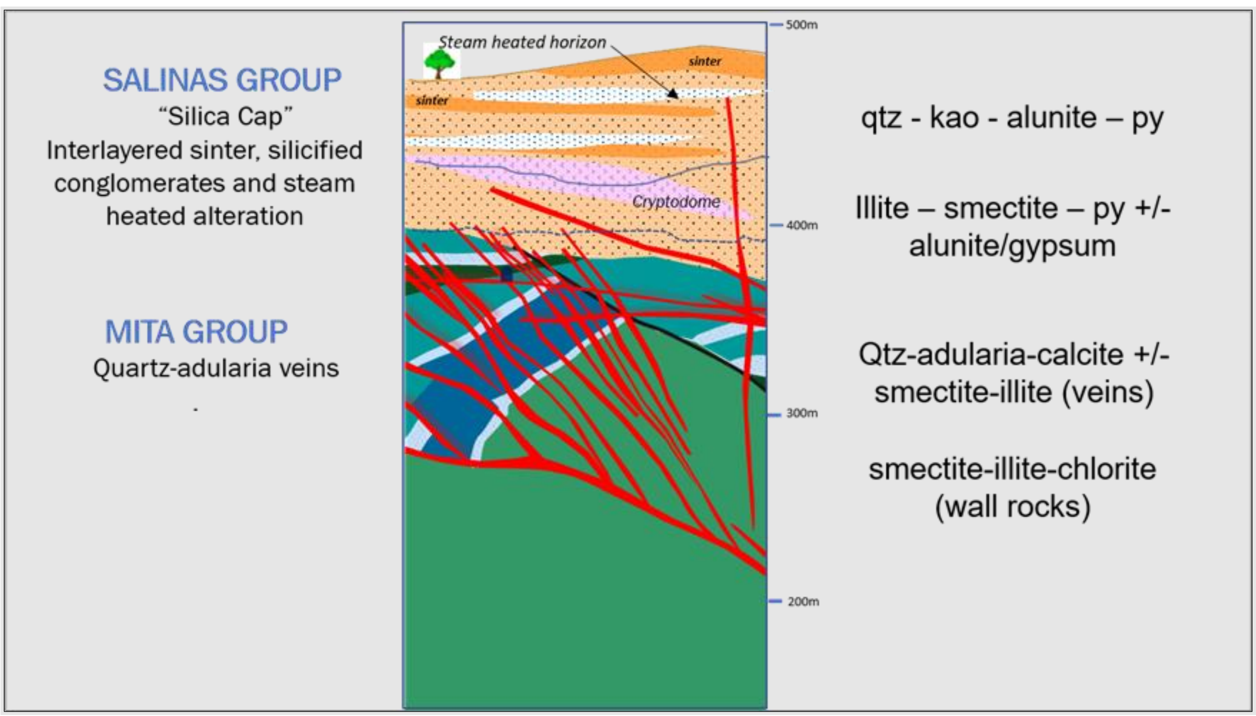

Source: Bluestone (2020).

### 7.5.3 Hydrothermal Alteration

Many low-sulphidation epithermal vein deposits have significant, mechanically weak, halos of illite / smectite + pyrite + sphene / leucoxene; however, the wall rocks at Cerro Blanco are generally only weakly clay altered and have a very low sulphide content (Figure 7-22). Most clay alteration is concentrated along some late faults, for example the East and Cross faults, and within some of the hydrothermal breccias, particularly the phreatic breccias in the Salinas Group.

A study using drill core hyperspectral imaging spectroscopy in the 500 nm to 2,500 nm wavelength range and detailed petrographic, SEM and EDS studies revealed two paragenetic stages of vein formation (Savinova, 2020). The main auriferous veins consist of multi-stage crustiform and colloform bands that are characterized by paragenetic Stage 1 equilibrium assemblage of quartz (chalcedony)-adularia-calcite-ankerite. Sulphides are located mostly in ginguro bands that consist of fine-grained pyrite, chalcopyrite, tetrahedrite and acanthite. Stage 2 of the paragenesis is characterized by intense overprinting of the quartz-adularia veins by montmorillonite and interstratified illite. Locally, bladed calcite is replaced by quartz. Hydrothermal alteration in the proximal zone of the sedimentary and volcanoclastic wall rocks is characterized by quartz-adularia-illite-montmorillonite. Wall rock hosted illite suggests a temperature of formation >230°C. The distal alteration zone is marked by illite-chlorite-calcite.

**Figure 7-22: Vertical Alteration Profile through Cerro Blanco**



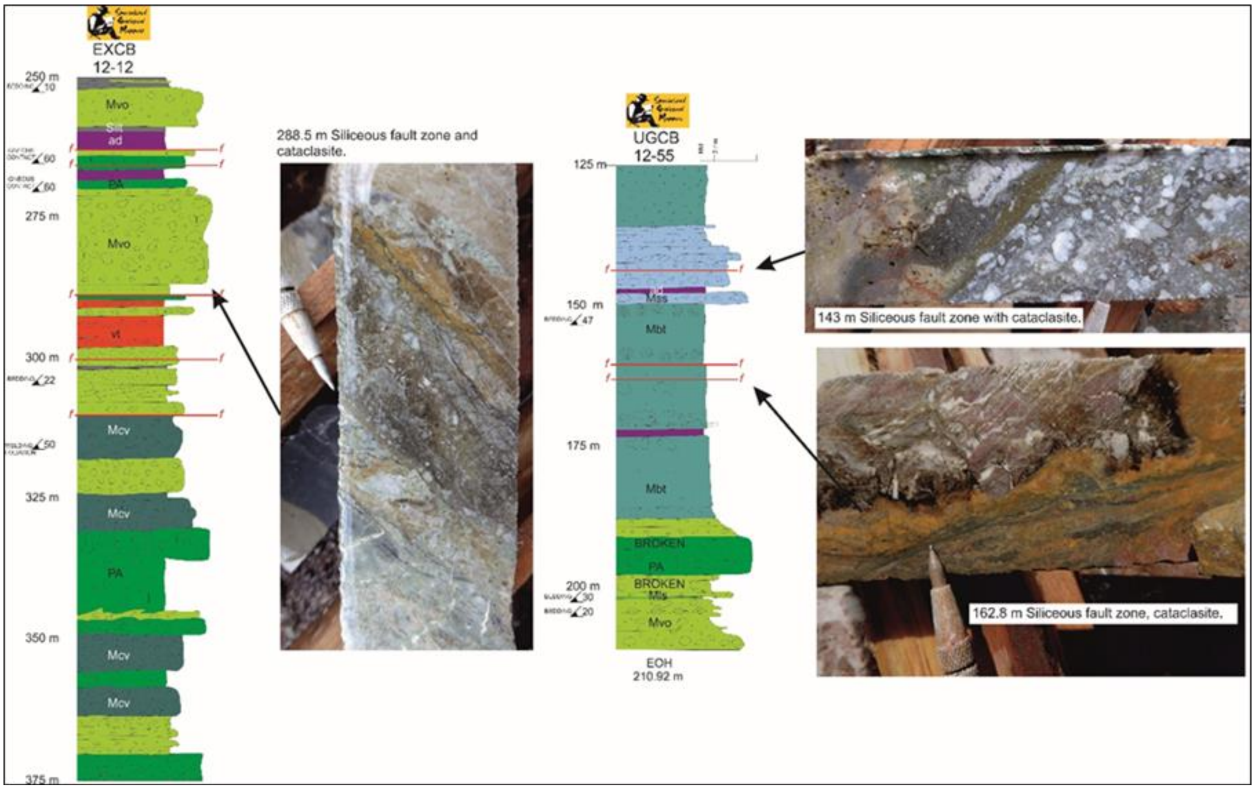

Source: Savinova, (2020).

Silicification is widely developed within the Salinas, and more selectively in the underlying Mita Group where it occurs as irregular envelopes, up to several meters wide, around the main veins as well as in the upper part of the limestone horizon as jasperoid. The most impressive alteration feature at Cerro Blanco is the large "silica cap" hosted in the Salinas sediments, typically beginning at or below 400 m elevation and continuing upward to the surface. Most silica is directly related to hot spring activity; the sinters and pisolithic beds contain abundant silica (although it is possible that some had carbonate precursors). However, there are also numerous beds of sandstone, conglomerate and mass flow deposits in the Salinas Group that are highly siliceous and locally flooded by chalcedony and fine-grained pyrite. These rocks are black when fresh, white and limonite stained when oxidized. Exposures around the Cerro Blanco ridge show that this silicification can be very capricious and replaced abruptly and laterally by smectite-rich clay alteration.

Where the paleo-groundwater table was several meters below the paleosurface, conglomerate in its immediate vicinity was silicified and the vadose zone above it was subjected to steam-heated alteration. The steam-heated alteration, containing cristobalite, kaolinite and alunite (an advanced argillic assemblage), was the product of acidic solutions formed by the condensation of ascendant $H_2S$-bearing steam into downward-percolating groundwater. The overall result is an interlayered sequence of sinter, silicified conglomerate and steam-heated alteration.

Many faults at Cerro Blanco are sealed by silica and are pre-mineral. Examples are shown in Figure 7-23.

**Figure 7-23: Examples of Sealed, Silicified Fault Zones**




Source: Pratt and Gordon (2019).

The boiling hydrothermal fluids that formed the Cerro Blanco vein system produced an even larger volume of intensely altered wall rock. Alteration types and zoning are typical of low-sulphidation epithermal systems. The remnant sinter above the deposit suggests that the Cerro Blanco system remains largely intact.

Silicification continues locally down to 300 m elevation along fault zones and in favourable rock types. Overall, the Cerro Blanco silica cap averages 400 m wide and is up to 150 m deep for at least a kilometer in strike. Within 50 m to 100 m of the surface, silicification is manifested by opaline silica flooding in the fragmental Svc and Rp units. At depth, very fine-grained quartz replacement of Mita Group calcareous sediments (locally forming jasperoid) and tuffs dominate. The Mcv crystal lithic tuff is generally only silicified near contacts with overlying sediments and along fault zones.

Silicification typically yields outward to moderate to strong sericitic alteration above 400 or 450 m elevation. At deeper levels, silicified zones grade outward and downward into large volumes of clay-sericite-pyrite±calcite alteration in Mita Group sediments and tuffs. Pyrite contents are commonly in the range of 1 -3%, locally reaching 5%.

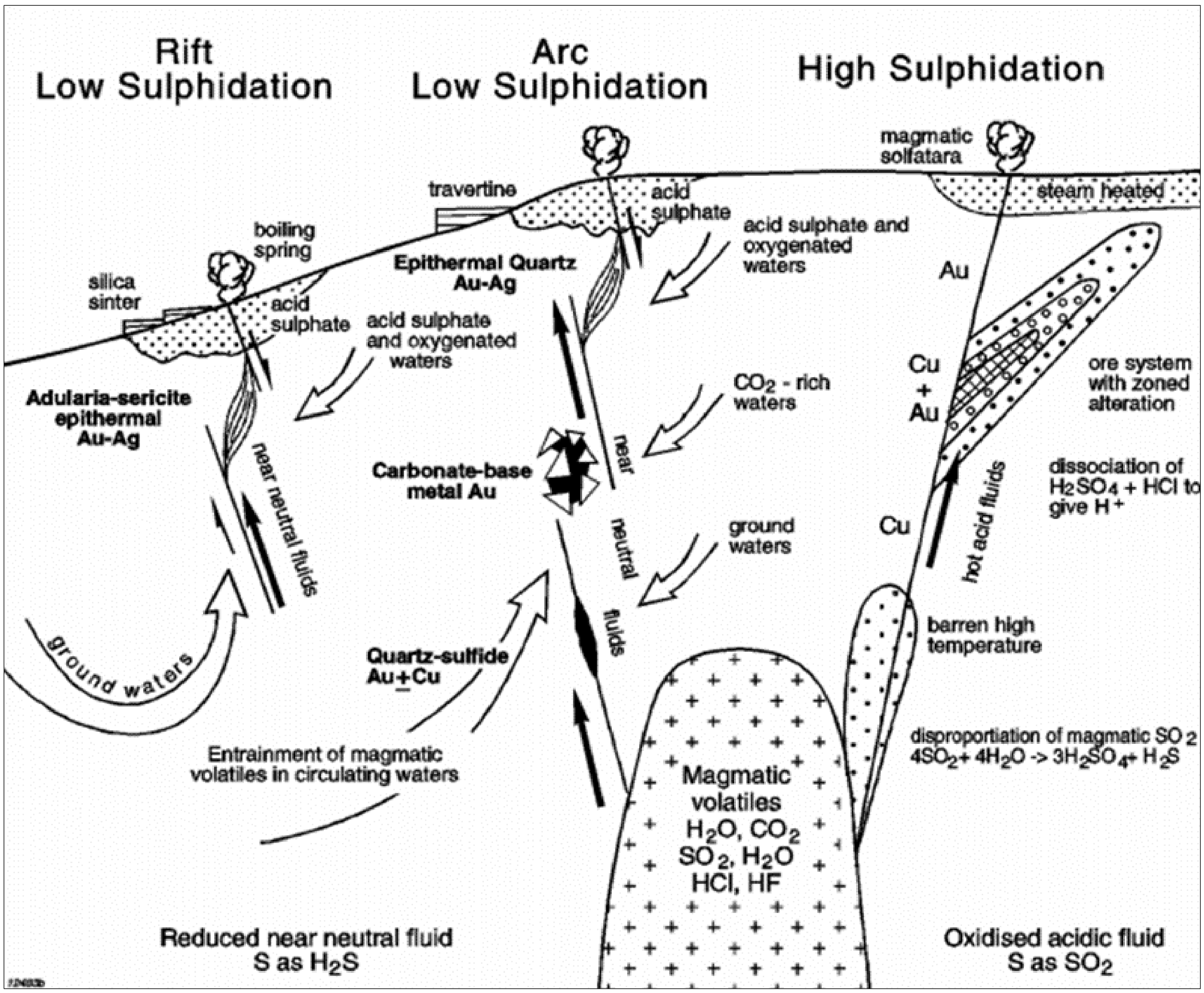
The Mcv is pervasively sericite-chlorite-pyrite±calcite altered virtually everywhere it has been drilled. Sericite dominates closer to mineralized faults and higher. Chlorite-calcite dominates outward and at depth. Pyrite is ubiquitous, but generally less than 0.5%.

# 8. DEPOSIT TYPES

The low sulphide content and near absence of base metals in the Cerro Blanco veins confirm it as a classic hot springs-related, low-sulphidation epithermal deposit. In common with most low-sulphidation deposits, it appears to be linked to compositionally bimodal, basalt-rhyolite volcanism, the hallmark of intra- and back-arc rift settings worldwide. The hydrothermal system seems likely to have been initiated during rhyolite dyke and cryptodome emplacement, at the base of the Salinas unit, with the rhyolitic magma and magmatic input to the mineralizing fluid both being derived from the same deep parental magma chamber.

Arc-related low-sulphidation gold deposits occur at highest crustal levels most removed from inferred intrusion source rocks. Figure 8-1 shows the generalized deposit model.

**Figure 8-1: Generalized Deposit Model Schematic**




Source: Corbett and Leach (1998).

Adularia-sericite epithermal gold-silver deposits characteristically occur as banded fissure veins and local vein / breccias which comprise predominantly colloform banded quartz, adularia, quartz pseudomorphing carbonate, and dark sulphidic material termed ginguro bands. Examples of adularia-sericite epithermal gold-silver deposits include Waihi and Golden Cross, Pajingo, Vera Nancy, Cracow, Hishikari, Sado, Konamai, Tolukuma, Toka Tindung, Lampung, Chatree, Cerro Vanguardia, Esquel, El Peñon.

At near surficial levels, many are capped by eruption breccias and sinter deposits. Eruption (phreatic) breccias, which form by the rapid expansion of depressurized geothermal fluids, are characterized by intensely silicified matrix and generally angular fragments including sinter, host rock and local surficial plant material. Although sinter deposits formed distal to fluid upflows commonly associated with eruption breccias, sinters tend to be barren with respect to gold but may be anomalous in other elements such as boron, arsenic and antimony.

Although cooling and traditional boiling models still hold for the deposition of gangue minerals (adularia, quartz pseudomorphing platy calcite and chalcedony) and some gold, mixing of rising pregnant fluids with oxygenated or collapsing acid sulphate (low pH) groundwater is also favoured as a mechanism for the development of characteristic bonanza gold-silver grades. Adularia-sericite vein systems are silver-rich with gold-to-silver ratios greater than 1:10 being common.

Wall rock alteration formed as halos to veins occurs as sericite (illite) grading to peripheral smectite clays with associated pyrite and chlorite, and this alteration grades to more marginal chlorite-carbonate (propylitic) alteration. Low temperature acid waters developed by the condensation of volatiles in the vadose zone contribute towards the formation of surficial acid sulphate alteration comprising silica (chalcedony, opal), kaolin, and local alunite and these acid sulphate waters are interpreted to collapse to deeper levels and so aid in mineral deposition.

Structure and host rock competency are important mineralization controls in adularia-sericite vein systems. High-grade mineralized shoots often develop in dilational jogs or flexures in through going veins where veins of greater thickness and higher gold grade develop and the intersections of fault splays. Bonanza grade material may also develop at preferred sites of fluid quenching at rock competency changes. Recent studies (e.g., Rhys, et al., 2020) attest that fault systems in very shallow epithermal systems characterized by sinter, lacustrine sediments and hydrothermal breccias, similar to Cerro Blanco, may represent syn-volcanic low-displacement growth faults that manifest as larger displacement pre-mineral faults at depth.

The connection between modern hot spring deposits and ancient hydrothermal systems, some with gold mineralization, has long been recognized (Lindgren, 1933). Epithermal mineral deposits are defined as those that develop close to the Earth's surface (within 1,000 m). They developed from fluids like those in modern geothermal systems. Sillitoe and Hedenquist (2003) defined the three types of epithermal deposit: high, intermediate, and low sulphidation. The low-sulphidation variant commonly occurs in rift settings, with bimodal volcanism in young, often Tertiary, volcanic arcs (e.g., Henley and Ellis, 1983). It is commonly associated with maar volcanoes, diatremes, and felsic flow domes.

Cerro Blanco shows all the characteristics of a completely preserved, non-eroded epithermal deposit. The occurrence of hot springs (sinters, silicified reeds, pisoliths) directly above the presumed feeder veins at Cerro Blanco implies a high water table and swampy conditions (cf. McLaughlin, California). In areas of high topographic relief, outflow springs (sinter) are usually found several kilometers from the upflow zones. The widespread occurrence of lacustrine and fluvial clastic sediments in the Salinas Group and accretionary lapilli, typical of water-rich pyroclastic surges, supports this interpretation. Sedimentation probably kept up with subsidence. Mudstone dykes and geopetal structures—open fractures filled by horizontally bedded chalcedonic and sulphide-rich sediment—reinforce the interpretation.

# 9. EXPLORATION

Exploration is planned during 2022 along the mineralized trend directly north of the Cerro Blanco deposit to find additional mineral resources that could potentially increase mine life. Work planned consists of re-logging historic drillholes, geological mapping, interpretation and generation of targets for drilling.

# 10. DRILLING

## 10.1 Drilling Summary

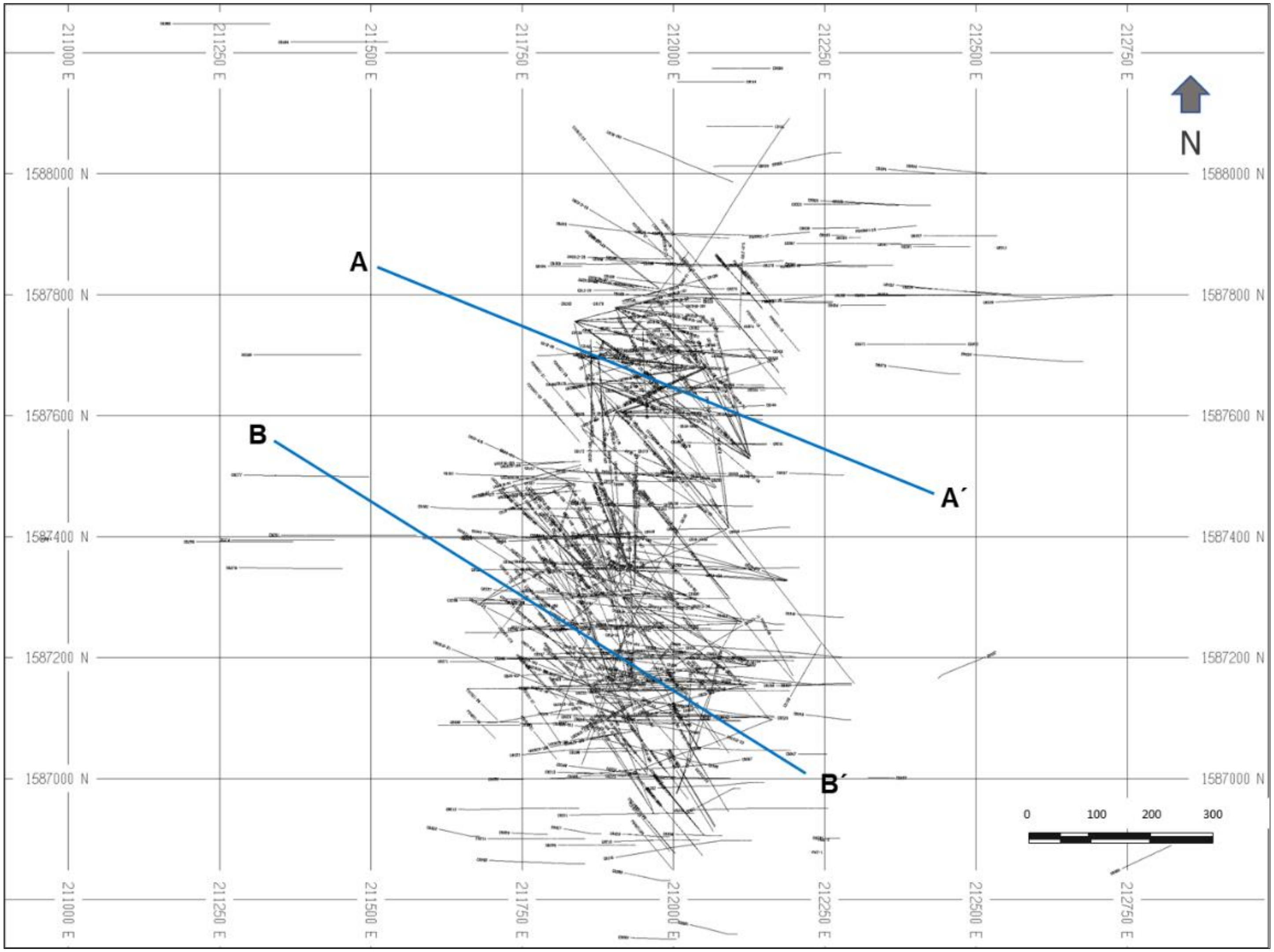
As of the end of 2021, Bluestone had drilled approximately 228 holes for a total of 36,462 m on the Cerro Blanco property since acquiring it from Goldcorp. Table 10.1 summarizes historical drilling on the property.

**Table 10.1: Summary of Drilling**

| Year | Company | Holes Drilled | Meters |
|---|---|---|---|
| 1998 | Mar-West | 9 | 1,340 |
| 1999 | Glamis | 48 | 7,074 |
| 2000 | Glamis | 18 | 3,525 |
| 2002 | Glamis | 23 | 6,525 |
| 2004 | Glamis | 42 | 9,370 |
| 2005 | Glamis | 120 | 29,065 |
| 2006 | Glamis | 67 | 15,129 |
| 2007 | Goldcorp | 47 | 12,373 |
| 2008 | Goldcorp | 2 | 586 |
| 2009 | Goldcorp | 1 | 140 |
| 2010 | Goldcorp | 10 | 2,277 |
| 2011 | Goldcorp | 28 | 5,898 |
| 2012 | Goldcorp | 96 | 21,370 |
| 2017 | Bluestone | 8 | 2,324 |
| 2018 | Bluestone | 74 | 13,993 |
| 2019 | Bluestone | 61 | 8,403 |
| 2020 | Bluestone | 74 | 15,172 |
| 2021 | Bluestone | 50 | 5,833 |
| Total | | 778 | 160,397 |

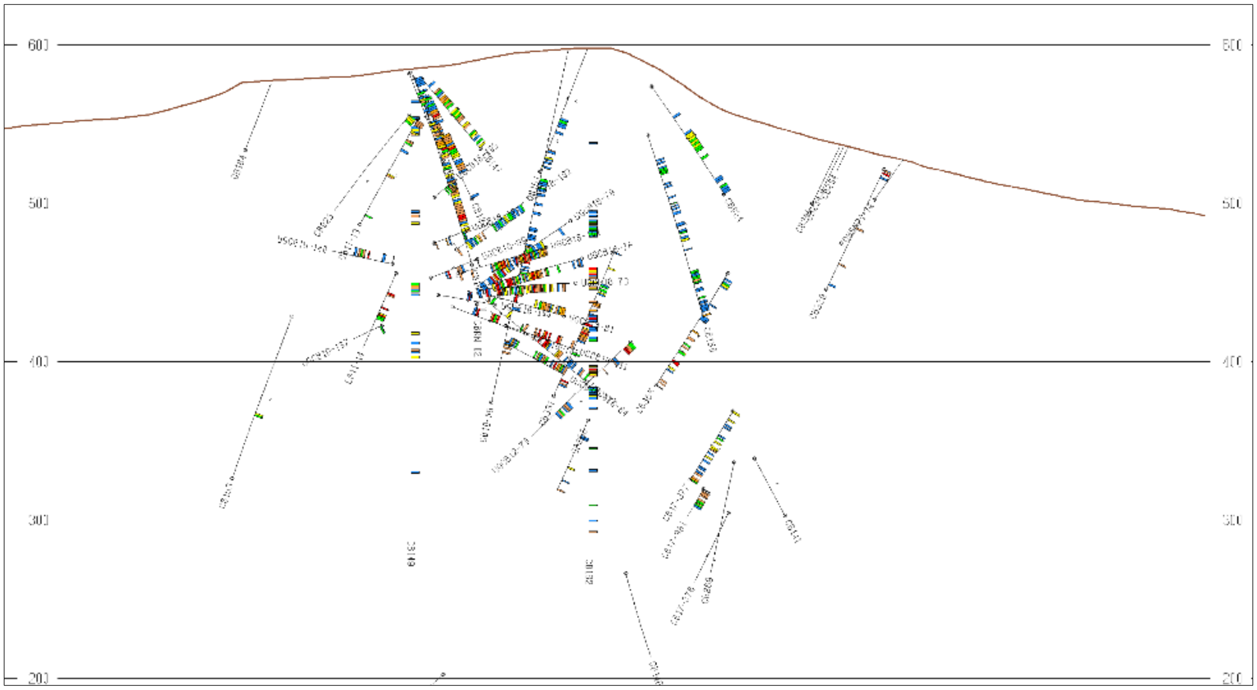
Figure 10-1 shows a plan view of drill hole locations. Figure 10-2 and Figure 10-3 show representative section views of the drilling along with gold assay data and topography.

**Figure 10-1: Plan View of Drill Hole Locations**



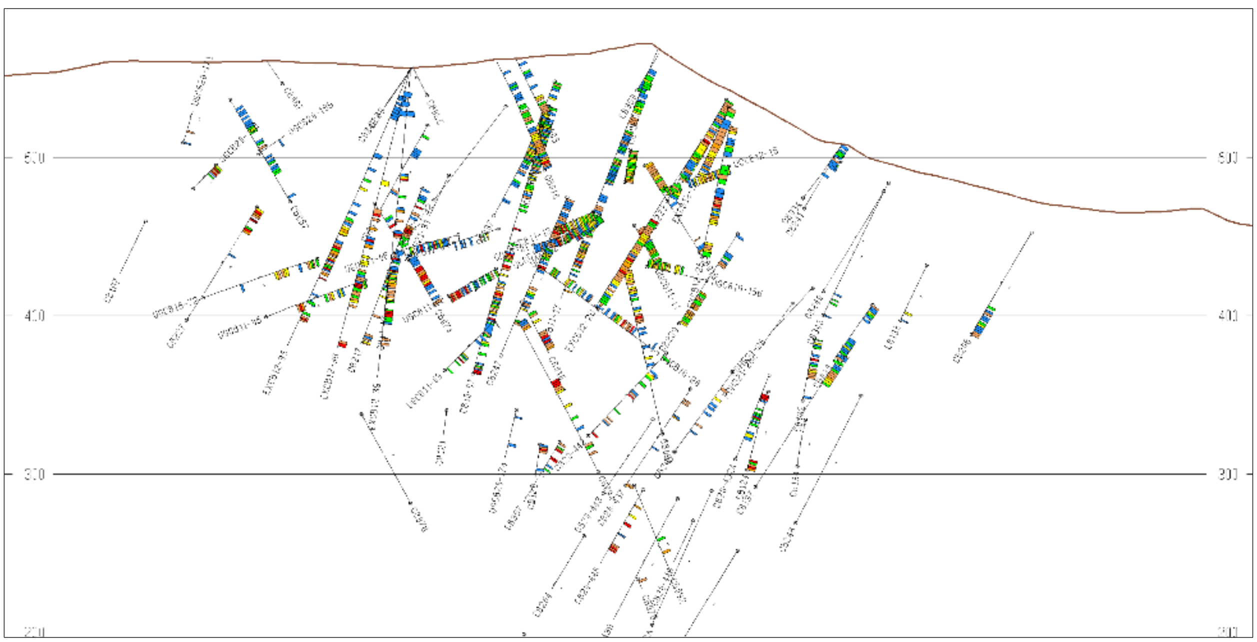

Source: Kirkham (2021).

**Figure 10-2: Section View A-A' (Azimuth 110°)**



Source: Kirkham (2021).

**Figure 10-3: Section View B-B' (Azimuth 110°)**



Source: Kirkham (2021).

## 10.2 Goldcorp & Glamis Drilling (Pre-2017)

Prior to Bluestone's ownership, reverse-circulation (RC) and diamond drilling (DD) was carried out. Many early holes were collared using RC size core before switching to NQ size core. Collar data from these historical programs was surveyed with a differential global positioning system (GPS), and down-hole survey measurements were taken with either a single-shot Sperry-Sun camera system or a multi-shot Flexit instrument.

Many of the earlier drill holes by previous operators were not drilled perpendicular to the strike and dip of the veining and therefore drilled widths of many veins were not representative. The most common vein intersections occur from between 0° and 60° to the core access. These intervals are thought to belong to steep to moderately dipping vein sets. These core intervals would be longer than the true thickness of the actual veining. Intersections ranging from 60° to 90° to the core axis are less common and are believed to belong to flat to near-flat vein structures. These vein intervals would be closer to the true thickness of the veining, but still longer than the true thickness. Only vein intervals drilled perpendicular to the strike and dip of the veining would represent the true thickness of the vein. Based on previous reports from Glamis Gold, the ratio to the true thickness of the vein on average is about 1.73 (i.e., every 1.7 m represents 1 m of true vein thickness).

## 10.3 Data Validation

Historical core logging, sampling, and quality assurance / quality control (QA/QC) procedures were first reviewed and documented by Golder in 2014. Ten core samples were collected from one-quarter sawn NQ core, and selected drill hole collars were surveyed using a GPS. Assayed gold and silver grades were found to be consistent with those reported by Goldcorp. Golder was satisfied that the drill hole data was collected in a manner consistent with industry best practice standards.

As part of the core logging data verification, Golder compared a selection of core logs against half-core stored at the project site. Five half-core drill holes were reviewed from the North and South deposits. The Microsoft Excel files were reviewed first, and drill holes were selected that represented the typical mineralization style for each deposit. In addition, 10 verification samples were taken from these drill holes. Each verification sample was a half-core sample sawed into quarters, with one quarter sample sent for analysis and the other returned to the core racks. Table 10.2 on the following page summarizes the samples selected for core logging review and verification sampling.

Samples were sawed and bagged under Golder supervision and were transported off site via helicopter and plane to Canada, and then by ground transportation to ALS Chemex Laboratories in Sudbury for sample preparation and analysis. Comparison of the Excel files against the drill core indicated an excellent match between the core logs and the retained core. Table 10.3 provides a list of the drill hole collar surveys completed by Golder.

Eight drill sites were visited, with multiple drill holes located at some sites. Casings had been removed for most drill holes. The data collected was a mixture of pre-Goldcorp drill holes (2006 or earlier) and drilling completed by Goldcorp during 2010 and 2011. All drill holes from surface were grouted to prevent water flow into the underground workings.

**Table 10.2: Verification Samples**

| Drill Hole ID | Duplicate Sample No. | Original Sample No | From (m) | To (m) | Deposit | Metal Analysed | Rock Type |
|---|---|---|---|---|---|---|---|
| CB-152 | 205873 | 82225 | 128 | 129 | North | Au, Ag | Lapilli Tuff |
| CB-152 | 205874 | 82226 | 129 | 130 | North | Au, Ag | Lapilli Tuff |
| CB-200 | 205884 | 407101 | 156 | 157 | South | Au, Ag | Quartz Tuff |
| CB-200 | 205885 | 407102 | 157 | 158 | South | Au, Ag | Quartz Tuff |
| CB-241 | 205891 | 404849 | 111.4 | 112.6 | South | Au, Ag | Conglomerate |
| CB-241 | 205892 | 404850 | 112.6 | 113.5 | South | Au, Ag | Fault |
| CB-254 | 205895 | 414397 | 100.5 | 102 | South | Au, Ag | Volcaniclastic Sediments |
| CB-254 | 205896 | 414398 | 102 | 103.5 | South | Au, Ag | Volcaniclastic Sediments |
| CB-10-15 | 205871 | 435941 | 135 | 136.23 | North | Au, Ag | Lapilli Tuff |
| CB-10-15 | 205872 | 435943 | 136.23 | 137.46 | North | Au, Ag | Lapilli Tuff |

Source: Goldcorp (2014).

**Table 10.3: Drill Hole Collar Survey (NAD 27 Zone 16N)**

| Drill Hole ID | Golder | | Cerro Blanco | |
|---|---|---|---|---|
| | Easting | Northing | Easting | Northing |
| C 10 08 | 212015.1 | 1587867 | 212009 | 1587748 |
| C 11 12 | 211906.8 | 1587714 | 211904 | 1587605 |
| C 11 15 | 211969.7 | 1587769 | 211966 | 1587655 |
| C 11 18 | 211866.4 | 1587405 | 211873.2 | 1587297 |
| C 11 21 | 211901.6 | 1587414 | 211898.9 | 1587307 |
| C 151 | 212025.1 | 1587821 | 212020.8 | 1587707 |
| C 247 | 211985.5 | 1587315 | 211978.8 | 1587202 |

Source: Goldcorp (2014).

Approximately 5% of the drill holes (20 holes) were subjected to data verification checks by Golder. The 20 selected holes, summarized in Table 10.4, included a variety of historical data as well as some of the more recent holes. The data verification checks consisted of the following:

- comparison of final assays to the original laboratory certificates
- analysis of external laboratory duplicate assays by generating XY scatterplots
- review of down hole survey measurements to identify anomalous changes to hole orientation.

For the 20 holes reviewed, the comparison of final assays to the original assay certificates did not identify any material differences in assay values.

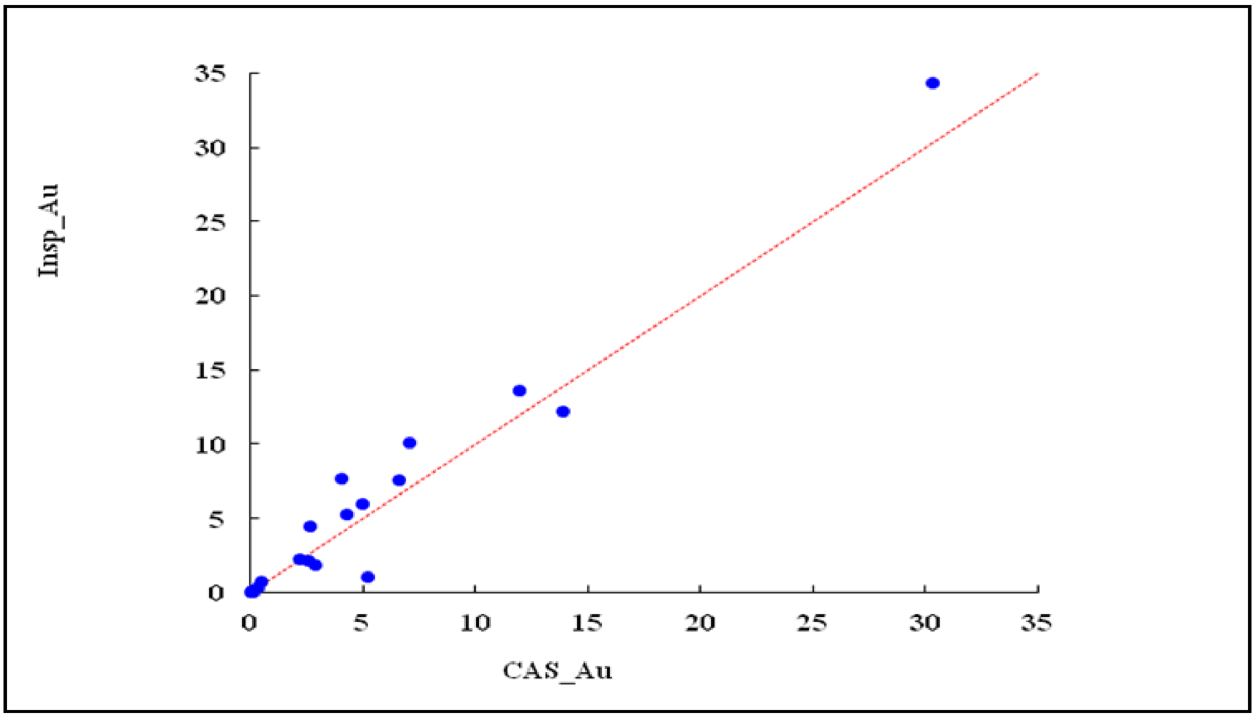
External laboratory duplicate assays were reviewed to assess the reliability of the primary assay laboratory. XY scatterplots were generated for each of the 20 holes. With the exception of a few outliers, the majority of the data compared well. Figure 10-4 illustrates an example of the XY scatterplots used to compare assay results.

**Table 10.4: Drill Holes Selected for Data Verification**

| Drill Hole IDs | |
|---|---|
| CB-012 | CB-200 |
| CB-016 | CB-227 |
| CB-063 | CB-244 |
| CB-078 | CB-247 |
| CB-095 | CB-305 |
| CB-10-02 | CB-309 |
| CB-120 | CB-314 |
| CB-142 | CB-345 |
| CB-146 | CB-357 |
| CB-151 | CB-362 |

Source: Goldcorp (2014).

**Figure 10-4: Example of XY Scatterplot for Hole CB34**




Source: Goldcorp (2014).

## 10.4 Bluestone Drilling (2017-2021)

Drilling completed by Bluestone between 2017 and 2020 was a combination of surface and underground diamond core drilling. Underground channel sampling was also performed and included in the resource estimation.

Drills were operated by Continental Drilling of Guatemala. The surface drilling was performed using two Hydracore 1000 portable drill rigs, one of which was replaced later in the program by a Boart Longyear

LM-75 belonging to Bluestone, which was later converted for underground drilling. During the height of the drill program, five LM-75s were operative. Drill holes were developed by drilling larger diameter (HQ) core at the early stage of the hole, and then decreasing to NQ and/or BQ size if the drilling conditions became difficult.

Core recoveries were high, and by utilizing several drill core sizes, Bluestone was able to ensure drill hole target completion. To date, 89 holes have been drilled from surface, and 128 holes from underground.

Drill hole collars were surveyed using a total station (coordinate system UTM NAD 27 Zone 16N). In-hole drill surveying for azimuth and dip was completed using the Reflex EZ-Shot system approximately every 25 m down-hole. Orientation of drill core was performed throughout Bluestone's drill program using Reflex ACT III downhole survey equipment.

## 10.5 Significant Assay Results

Table 10.5 provides a selection of significant drill hole intervals from the Cerro Blanco drill hole database. Drill hole intervals are reported as actual core lengths and many may not represent the true thickness.

**Table 10.5: Gold & Silver Samples from the Drill Hole Database**

| Hole | Company | From | To | Length (m) | Au (g/t) | Ag (g/t) |
|---|---|---|---|---|---|---|
| CB-012 | Mar-West/Glamis | 99.50 | 108.50 | 9.00 | 13.7 | 46.5 |
| CB-012 | Mar-West/Glamis | 141.50 | 147.50 | 6.00 | 12.9 | 75.8 |
| CB-012 | Mar-West/Glamis | 195.50 | 198.50 | 3.00 | 3.0 | 8.0 |
| CB-012 | Mar-West/Glamis | 236.00 | 237.50 | 1.50 | 13.0 | 6.0 |
| CB-016 | Mar-West/Glamis | 192.55 | 195.35 | 2.80 | 3.3 | 0 |
| CB-063 | Glamis | 88.50 | 99.00 | 10.50 | 4.4 | 25.7 |
| CB-063 | Glamis | 114.00 | 126.00 | 12.00 | 3.2 | 21.0 |
| CB-063 | Glamis | 183.00 | 186.00 | 3.00 | 7.7 | 20.0 |
| CB-063 | Glamis | 196.50 | 199.50 | 3.00 | 4.2 | 25.0 |
| CB-063 | Glamis | 207.00 | 210.00 | 3.00 | 18.7 | 20.0 |
| CB-063 | Glamis | 225.00 | 228.00 | 3.00 | 37.3 | 75.0 |
| CB-063 | Glamis | 241.50 | 244.50 | 3.00 | 5.1 | 3.5 |
| CB-078 | Glamis | 158.20 | 161.40 | 3.20 | 3.4 | 4.1 |
| CB-078 | Glamis | 242.10 | 245.10 | 3.00 | 3.5 | 4.7 |
| CB-078 | Glamis | 248.10 | 273.75 | 25.65 | 66.1 | 42.2 |
| CB-078 | Glamis | 299.25 | 303.75 | 4.50 | 4.7 | 17.7 |
| CB-078 | Glamis | 338.25 | 345.75 | 7.50 | 10.8 | 17.4 |
| CB-095 | Glamis | 155.00 | 158.00 | 3.00 | 3.7 | 204.9 |
| CB-095 | Glamis | 179.00 | 182.00 | 3.00 | 17.8 | 7.4 |
| CB-095 | Glamis | 233.00 | 236.00 | 3.00 | 88.0 | 98.6 |
| CB-10-02 | Goldcorp | 117.50 | 120.30 | 2.80 | 14.7 | 79.5 |
| CB-10-02 | Goldcorp | 135.75 | 139.50 | 3.75 | 12.9 | 91.8 |
| CB-10-02 | Goldcorp | 146.00 | 149.00 | 3.00 | 9.5 | 79.6 |
| CB-10-02 | Goldcorp | 168.86 | 173.00 | 4.14 | 26.2 | 144.8 |
| CB-10-02 | Goldcorp | 197.00 | 200.00 | 3.00 | 20.3 | 19.9 |
| CB-120 | Glamis | 219.00 | 238.50 | 19.50 | 17.5 | 20.3 |

| Hole | Company | From | To | Length (m) | Au (g/t) | Ag (g/t) |
|---|---|---|---|---|---|---|
| CB-120 | Glamis | 246.00 | 249.00 | 3.00 | 8.8 | 20.6 |
| CB-142 | Glamis | 163.50 | 171.50 | 8.00 | 16.0 | 72.2 |
| CB-142 | Glamis | 196.20 | 204.50 | 8.30 | 19.2 | 11.7 |
| CB-142 | Glamis | 302.75 | 306.00 | 3.25 | 19.3 | 14.3 |
| CB-146 | Glamis | 80.30 | 86.00 | 5.70 | 14.0 | 196.8 |
| CB-146 | Glamis | 109.00 | 112.40 | 3.40 | 10.3 | 78.9 |
| CB-146 | Glamis | 118.90 | 130.00 | 11.10 | 70.4 | 226.3 |
| CB-146 | Glamis | 139.00 | 143.00 | 4.00 | 12.4 | 35.4 |
| CB-146 | Glamis | 149.00 | 152.00 | 3.00 | 3.7 | 8.0 |
| CB-146 | Glamis | 156.00 | 159.00 | 3.00 | 21.1 | 30.6 |
| CB-146 | Glamis | 182.00 | 185.00 | 3.00 | 4.2 | 2.5 |
| CB-151 | Glamis | 162.40 | 165.50 | 3.10 | 25.6 | 152.8 |
| CB-151 | Glamis | 172.90 | 179.30 | 6.40 | 13.6 | 24.7 |
| CB-151 | Glamis | 327.50 | 330.50 | 3.00 | 5.0 | 5.5 |
| CB-200 | Glamis | 117.00 | 120.00 | 3.00 | 5.7 | 26.0 |
| CB-200 | Glamis | 144.00 | 147.00 | 3.00 | 5.0 | 13.0 |
| CB-200 | Glamis | 152.00 | 161.00 | 9.00 | 7.5 | 13.6 |
| CB-200 | Glamis | 165.00 | 168.50 | 3.50 | 16.7 | 212.9 |
| CB-227 | Glamis | 117.34 | 124.96 | 7.62 | 15.4 | 20.6 |
| CB-227 | Glamis | 131.00 | 134.00 | 3.00 | 5.6 | 22.0 |
| CB-244 | Glamis | 90.00 | 99.00 | 9.00 | 10.3 | 57.0 |
| CB-244 | Glamis | 139.50 | 142.50 | 3.00 | 4.2 | 4.0 |
| CB-244 | Glamis | 234.00 | 237.00 | 3.00 | 22.5 | 21.0 |
| CB-247 | Glamis | 135.00 | 138.00 | 3.00 | 3.5 | 25.5 |
| CB-247 | Glamis | 159.00 | 162.00 | 3.00 | 4.0 | 4.5 |
| CB-247 | Glamis | 231.00 | 234.00 | 3.00 | 6.8 | 15.7 |
| CB-247 | Glamis | 240.00 | 243.00 | 3.00 | 28.6 | 98.5 |
| CB-305 | Glamis | 86.00 | 90.00 | 4.00 | 5.0 | 9.5 |
| CB-305 | Glamis | 138.00 | 141.50 | 3.50 | 5.5 | 21.3 |
| CB-309 | Glamis | 128.50 | 132.00 | 3.50 | 3.5 | 8.6 |
| CB-309 | Glamis | 183.00 | 186.70 | 3.70 | 130.1 | 304.6 |
| CB-309 | Glamis | 193.50 | 196.50 | 3.00 | 40.3 | 17.0 |
| CB-314 | Glamis | 99.50 | 102.50 | 3.00 | 5.3 | 11.0 |
| CB-314 | Glamis | 111.50 | 119.50 | 8.00 | 8.3 | 19.9 |
| CB-314 | Glamis | 124.50 | 127.50 | 3.00 | 24.2 | 113.6 |
| CB-314 | Glamis | 131.50 | 134.50 | 3.00 | 13.6 | 30.7 |
| CB-314 | Glamis | 140.50 | 143.50 | 3.00 | 11.8 | 45.0 |
| CB-314 | Glamis | 151.50 | 154.50 | 3.00 | 3.7 | 15.0 |
| CB-314 | Glamis | 175.50 | 178.50 | 3.00 | 85.6 | 386.9 |
| CB-314 | Glamis | 186.00 | 189.00 | 3.00 | 4.2 | 12.5 |
| CB-345 | Glamis | 231.70 | 234.70 | 3.00 | 13.1 | 20.8 |
| CB-345 | Glamis | 315.50 | 318.50 | 3.00 | 5.8 | 6.7 |
| CB-357 | Glamis | 63.00 | 66.00 | 3.00 | 5.5 | 33.3 |
| CB-357 | Glamis | 140.00 | 143.00 | 3.00 | 3.4 | 2.7 |
| CB-357 | Glamis | 159.00 | 162.50 | 3.50 | 4.0 | 2.7 |
| CB-357 | Glamis | 184.00 | 187.00 | 3.00 | 3.6 | 22.0 |
| CB-357 | Glamis | 192.50 | 195.50 | 3.00 | 46.4 | 126.3 |

| Hole | Company | From | To | Length (m) | Au (g/t) | Ag (g/t) |
|---|---|---|---|---|---|---|
| CB-357 | Glamis | 200.00 | 206.20 | 6.20 | 12.6 | 6.3 |
| CB-357 | Glamis | 217.50 | 220.80 | 3.30 | 4.3 | 5.0 |
| CB-362 | Glamis | 128.50 | 131.50 | 3.00 | 4.2 | 6.0 |
| CB-362 | Glamis | 219.00 | 222.20 | 3.20 | 4.5 | 6.0 |
| CB17-376 | Bluestone | 221.90 | 224.40 | 2.50 | 17.1 | 33.0 |
| CB18-386 | Bluestone | 243.80 | 246.47 | 2.63 | 5.1 | 5.6 |
| CB18-388 | Bluestone | 37.70 | 41.00 | 3.30 | 8.6 | 3.5 |
| CB18-389 | Bluestone | 104.70 | 110.00 | 5.30 | 7.9 | 35.1 |
| CB18-390 | Bluestone | 164.27 | 169.57 | 5.30 | 16.0 | 29.1 |
| CB18-393 | Bluestone | 253.60 | 261.50 | 7.90 | 16.5 | 18.4 |
| CB18-394 | Bluestone | 110.60 | 128.00 | 17.40 | 7.0 | 65.2 |
| CB18-395 | Bluestone | 46.30 | 51.00 | 4.70 | 5.8 | 4.2 |
| CB18-396 | Bluestone | 103.08 | 108.15 | 5.07 | 7.1 | 24.7 |
| CB18-396 | Bluestone | 167.14 | 181.41 | 14.27 | 16.2 | 20.6 |
| UGCB18-71 | Bluestone | 0.00 | 27.69 | 27.69 | 5.5 | 17.1 |
| UGCB18-71 | Bluestone | 0.00 | 27.69 | 27.69 | 5.5 | 17.1 |
| UGCB18-72 | Bluestone | 88.10 | 90.00 | 1.87 | 7.6 | 23.5 |
| UGCB18-73 | Bluestone | 6.00 | 23.00 | 17.00 | 5.1 | 17.2 |
| UGCB18-73 | Bluestone | 37.19 | 43.13 | 5.94 | 5.2 | 10.3 |
| UGCB18-73 | Bluestone | 13.20 | 16.85 | 3.65 | 19.3 | 59.4 |
| UGCB18-74 | Bluestone | 37.62 | 41.23 | 3.61 | 9.0 | 28.5 |
| UGCB18-74 | Bluestone | 54.40 | 56.39 | 1.99 | 21.3 | 63.4 |
| UGCB18-75 | Bluestone | 45.72 | 51.22 | 5.50 | 7.3 | 60.9 |
| UGCB18-76 | Bluestone | 12.61 | 47.10 | 34.49 | 5.8 | 18.6 |
| UGCB18-76 | Bluestone | 12.61 | 16.53 | 3.92 | 26.8 | 84.4 |
| UGCB18-79 | Bluestone | 11.31 | 20.82 | 9.51 | 5.6 | 33.9 |
| UGCB18-80 | Bluestone | 47.77 | 53.25 | 5.48 | 9.3 | 105.3 |
| UGCB18-80 | Bluestone | 85.95 | 88.47 | 2.52 | 13.9 | 85.2 |
| UGCB18-81 | Bluestone | 100.50 | 105.07 | 4.57 | 20.8 | 46.9 |
| UGCB18-81 | Bluestone | 122.18 | 125.20 | 3.02 | 11.2 | 13.1 |
| UGCB18-82 | Bluestone | 71.16 | 81.18 | 10.02 | 15.0 | 32.5 |
| UGCB18-84 | Bluestone | 53.33 | 56.08 | 2.75 | 44.7 | 39.9 |
| UGCB18-85 | Bluestone | 52.34 | 59.12 | 6.78 | 24.6 | 92.8 |
| UGCB18-85 | Bluestone | 70.05 | 71.13 | 1.08 | 21.2 | 60.9 |
| UGCB18-86 | Bluestone | 23.50 | 30.50 | 7.00 | 17.2 | 94.9 |
| UGCB18-86 | Bluestone | 33.35 | 37.19 | 3.84 | 9.1 | 28.9 |
| UGCB18-86 | Bluestone | 43.55 | 51.81 | 8.26 | 32.7 | 79.6 |
| UGCB18-87 | Bluestone | 97.74 | 98.81 | 1.07 | 16.0 | 26.8 |
| UGCB18-88 | Bluestone | 43.00 | 52.20 | 9.22 | 9.8 | 29.9 |
| UGCB18-88 | Bluestone | 62.20 | 64.20 | 2.00 | 9.8 | 35.7 |
| UGCB18-89 | Bluestone | 50.72 | 65.72 | 15.00 | 16.7 | 105.4 |
| UGCB18-89 | Bluestone | 92.01 | 101.37 | 9.36 | 14.3 | 68.5 |
| UGCB18-91 | Bluestone | 12.90 | 15.85 | 2.95 | 17.9 | 27.6 |
| UGCB18-92 | Bluestone | 36.80 | 58.20 | 21.40 | 9.6 | 34.9 |
| UGCB18-92 | Bluestone | 112.30 | 117.60 | 5.40 | 12.8 | 10.8 |
| UGCB18-93 | Bluestone | 10.30 | 11.30 | 1.00 | 24.5 | 32.2 |
| UGCB18-94 | Bluestone | 98.10 | 100.30 | 2.20 | 7.2 | 15.7 |

| Hole | Company | From | To | Length (m) | Au (g/t) | Ag (g/t) |
|---|---|---|---|---|---|---|
| UGCB18-95 | Bluestone | 6.40 | 7.60 | 1.20 | 8.9 | 49.2 |
| UGCB18-95 | Bluestone | 14.10 | 15.60 | 1.50 | 12.2 | 27.3 |
| UGCB18-96 | Bluestone | 39.40 | 52.40 | 13.00 | 11.5 | 48.6 |
| UGCB18-96 | Bluestone | 56.40 | 61.40 | 5.00 | 7.1 | 30.5 |
| UGCB18-98 | Bluestone | 108.20 | 110.60 | 2.30 | 9.9 | 8.7 |
| UGCB18-98 | Bluestone | 115.20 | 116.20 | 1.00 | 28.6 | 112.0 |
| UGCB19-126 | Bluestone | 32.20 | 43.00 | 10.20 | 13.1 | 25.0 |
| UGCB19-143 | Bluestone | 57.00 | 66.00 | 9.00 | 8.4 | 53.2 |
| UGCB19-144 | Bluestone | 98.80 | 106.70 | 7.50 | 19.0 | 44.3 |
| UGCB19-147 | Bluestone | 62.80 | 76.50 | 13.70 | 11.2 | 78.0 |
| UGCB19-152 | Bluestone | 39.60 | 41.90 | 2.30 | 49.2 | 42.0 |
| UGCB19-155 | Bluestone | 75.30 | 82.30 | 7.00 | 11.9 | 18.0 |
| UGCB19-157 | Bluestone | 132.30 | 139.30 | 7.00 | 10.7 | 131.5 |
| CB19-410 | Bluestone | 222.40 | 233.90 | 11.50 | 8.5 | 7.1 |
| CB19-411 | Bluestone | 215.90 | 225.40 | 9.50 | 7.2 | 16.0 |
| UGCB20-174 | Bluestone | 120.83 | 128.20 | 7.40 | 14.9 | 54.9 |
| UGCB20-176 | Bluestone | 128.30 | 142.40 | 14.10 | 24.9 | 38.6 |
| UGCB20-179 | Bluestone | 61.30 | 73.10 | 11.90 | 86.3 | 364.9 |
| UGCB20-179 | Bluestone | 68.60 | 73.10 | 4.20 | 194.0 | 810.4 |
| CB20-180 | Bluestone | 170.60 | 175.93 | 5.40 | 334.7 | 538.8 |
| CB20-181 | Bluestone | 210.60 | 215.70 | 5.10 | 75.7 | 32.8 |
| CB20-188 | Bluestone | 177.70 | 186.74 | 9.00 | 26.0 | 26.8 |
| CB20-191 | Bluestone | 24.80 | 126.20 | 101.40 | 2.4 | 9.6 |
| CB20-420 | Bluestone | 179.50 | 195.00 | 15.50 | 21.6 | 51.7 |
| CB20-427 | Bluestone | 215.80 | 218.90 | 3.00 | 19.1 | 15.0 |
| CB20-429 | Bluestone | 22.90 | 212.14 | 189.30 | 0.8 | 2.5 |
| CB20-430 | Bluestone | 227.30 | 236.47 | 9.30 | 34.6 | 66.9 |
| CB20-433 | Bluestone | 75.60 | 293.20 | 217.60 | 1.4 | 5.6 |
| CB20-433 | Bluestone | 293.10 | 314.30 | 21.20 | 11.2 | 11.7 |
| CB20-442 | Bluestone | 263.50 | 292.10 | 28.60 | 11.6 | 12.3 |
| CB20-442 | Bluestone | 282.60 | 28.88 | 6.30 | 29.0 | 30.1 |
| CB20-444 | Bluestone | 54.60 | 166.30 | 111.80 | 2.1 | 12.5 |
| CB20-444 | Bluestone | 136.50 | 143.56 | 9.50 | 7.6 | 55.6 |
| CB20-449 | Bluestone | 43.30 | 158.20 | 114.90 | 2.5 | 13.4 |
| CB21-460 | Bluestone | 114.60 | 172.21 | 57.60 | 3.1 | 9.9 |
| CB21-469 | Bluestone | 1.52 | 141.73 | 140.20 | 1.1 | 8.2 |
| CB21-487 | Bluestone | 85.30 | 92.90 | 7.60 | 30.2 | 85.5 |

Source: Goldcorp (2014), Bluestone (2021).

# 11. SAMPLE PREPARATION, ANALYSES & SECURITY

## 11.1 Sampling Method & Approach

### 11.1.1 Sampling Preparation, Analyses & Security (prior to November 2006)

Prior to Goldcorp taking ownership of the Cerro Blanco Gold Project in November 2006, all previous drilling, sampling, and assaying was under the control of Glamis.

All sample data used in the Cerro Blanco mineral resource calculations was produced by either diamond drilling (DD) or reverse-circulation (RC) drilling. Drilling contractors were hired to supply the drilling equipment and perform the work under the direct supervision of owner field personnel.

The Glamis drill hole program used a variable combination of sample collection, as follows:

- double-tube HQ core in the upper reaches of the hole switching to double-tube NQ core deeper in the hole
- RC drilling in the upper reaches of the hole above the water table and/or the anticipated mineralization zone, switching to double-tube NQ core deeper in the hole
- RC drilling for the entire hole.

Rotary samples collected from the 4¾ inch, face-sampling, hammer-drilled RC holes were initially collected in a five-gallon bucket. The weight was then recorded, and the sample placed into the hopper of a Gilson splitter. The process was repeated until the entire 1.5 m sample was collected. The total weight was recorded on the sample sheet along with the sample identification and the time of day collected. Weights were only recorded for the dry portion of the drill hole. The Gilson splitter was set to split the sample into two halves, with one half retained and the other wasted. The remaining 50% was placed into the hopper again and another 50% split was made. The two samples were placed into pre-labelled plastic sample bags, one for assay and the other for storage. An air hose and nozzle were provided for cleaning the Gilson splitter, pan, and buckets. A geologist was assigned to the rotary rig to supervise sample collection and log geology. A chip tray was created as a permanent record of each hole.

Core was collected and placed in wooden core boxes. The core was washed to obtain a clean surface for geological and geotechnical logging and placed in a covered logging facility. All core was photographed on print film. Core was sawn longitudinally with a diamond saw and half the core, on a nominal 1.5 m interval broken at lithologic boundaries, and was placed in pre-labelled plastic bags.

The other half was retained for inspection or additional tests as warranted. Splits from the core holes were shipped to a facility operated by CAS Laboratories (CAS Honduras) in Tegucigalpa, Honduras. Unused core was retained for inspection on site.

Samples were transported from Cerro Blanco to the laboratory in Tegucigalpa, Honduras by CAS personnel and all sample preparation and analyses were conducted at CAS Honduras. Reject samples and pulps were stored at the CAS Honduras facility. Samples were analysed for gold using a 30 g pulp with fire assay, atomic absorption (AA) finish. Samples that ran over 1.0 g/t Au from this method were re-analysed for both gold and silver using a 30 g pulp, fire assay with gravimetric finish.

Glamis had established a limited QA/QC program focused on coarse reject and pulp reject checks. A frequency of 1 in 20 pulps was systematically submitted to the Chemex Laboratories in Nevada for gold and silver analysis in addition to coarse rejects.

The drill samples were initially quick-logged to locate and mark significant changes in volcanic stratigraphy. Each volcanic unit was then described, and the location of structure and their orientations, the percentage of quartz veining, and the type of alteration was recorded.

Standard logging conventions were used to capture information from the drill sample. Detailed, daily logging was transcribed onto log sheets and independently entered into Excel spreadsheets. The geologist checked data entry before data was merged with the main database.

Detailed core logging was done by capturing data in four tables: lithology, alteration, sulphide type, and geotechnical information. Lithology was captured using standardized abbreviations. Alteration was captured as a numeric value corresponding to alteration type. The visible sulphide types were captured as a total modal percentage and as relative ratios. Structural data was captured in the “comments / structures” table in the database, as type and angles taken related to core axis are displayed in an area as a graphical representation. The geotechnical data recorded rock quality designation (RQD) data for the core portion of the hole.

All independent laboratories used in the Cerro Blanco Gold Project employed quality control procedures and protocols that included duplicates, standard reference materials, and blanks. These were available to Glamis, but were not included in assay reports.

### 11.1.2 Sample Preparation, Analyses & Security (Goldcorp 2010 through 2012)

Drilling completed by Goldcorp (2010 to 2012) was a combination of surface and underground diamond core drilling. Drills were operated by both contract and Goldcorp personnel. The Goldcorp underground drill rig (Boart Longyear LM-75) was used on the surface and converted for underground drilling. Drill holes were developed by drilling larger diameter (HQ) core at the early stage of the hole, decreasing to NQ and/or BQ if the drilling conditions became difficult.

Drill recovery was high and, by utilizing several drill core sizes, Goldcorp was able ensure drill hole target completion. Drill hole collar surveys were completed using a GPS Trimble system (UTM NAD 27 Zone 16N). In-hole drill surveying for azimuth and dip was completed using the Reflex EZ-Shot system approximately every 50 m along the drill hole.

Drill cores (surface and underground) were stored in wooden labelled boxes from the drill and transported to the surface core logging facility at the Cerro Blanco surface core facility.

Technicians first prepared the core boxes by reviewing drill hole depth tags and re-assembling broken sections (from zones of poor recovery).

Core logging to identify lithology, alteration, RQD, and sampling selection for core sawing was completed by geologists or by technicians under the direction of the geologist. Sampling was also completed by Goldcorp personnel that included technicians and geologists. The typical sample lengths were 1.0 to 1.5 m with maximum lengths of 2.0 and 3.0 m; sample lengths were based on the lithology and alteration. Logs and the sample database indicated that low-grade and high-grade gold and silver samples were of the same lengths and were not broken out separately or collected in a way to cause sample bias. Samples were collected along the footwall, mineralized zones, and hanging walls without breaks in sampling. Blanks were inserted by Goldcorp personnel when a core sample was submitted. All data was initially collected on paper logs and later transferred to Excel files. This data was then entered in MapInfo™ and MineSight™ software for geological modelling.

The core selected for analysis was transported to Inspectorate Laboratories in Guatemala City for sample preparation. Samples were prepared at Inspectorate (Guatemala) by crushing and pulverizing the drill core to 100 g pulp samples.

One pulp sample was sent to Goldcorp's Marlin Mine for gold assaying (fire assay with AA or gravimetric finish) and silver assaying (AA or AA with gravimetric finish). The second pulp sample was sent to Inspectorate Laboratory in Reno, Nevada for gold assaying (fire assay with AA or gravimetric finish) and silver assaying (AA or AA with gravimetric finish). The Marlin Mine assays were completed quickly, which assisted the geologists in developing the drilling program. The Inspectorate assays were used for the purposes of mineral resource modelling and estimation.

The QA/QC program employed at the Cerro Blanco Gold Project was under the direction of Goldcorp. Blank samples were inserted by Goldcorp geologists prior to shipping to Inspectorate at a frequency of 1 in 25 sample submission. No duplicates of coarse rejects or standards were included in the QA/QC program at Cerro Blanco; however, it was recommended that duplicates of the coarse rejects be analyzed and compared and that standards be inserted into the QA/QC sample stream for future drilling campaigns. All analytical results were provided to Goldcorp staff and stored first in Excel and later in MapInfo™ and MineSight™ software. All half-core samples collected by both Goldcorp and Glamis are stored adjacent to the core logging facility on the Cerro Blanco Gold Project site. The Cerro Blanco site is fully controlled by perimeter fencing and security. All samples removed from site were under the control of Inspectorate Laboratories.

### 11.1.3 Sampling Preparation, Analyses & Security (Bluestone 2017 to 2021)

Drill core from surface and underground was stored in labelled wooden boxes (Figure 11-1) at the drill site and transported to the surface core logging facility. Before core splitting and logging commences, drill core was systematically photographed in high resolution using a tripod-mounted camera and digitally archived for reference as part of the drill database.

**Figure 11-1: Example of Core Box Photography**


Source: Bluestone (2019).

Logging and sampling were undertaken on site at Cerro Blanco by company personnel under a QA/QC protocol developed by Bluestone. Technicians first prepared the core boxes by reviewing drill hole depth tags, re-assembling broken sections, and photographing core. Core logging to identify lithology, alteration, RQD, and sampling selection for core sawing was completed by technicians under the direction of the geologist. Sampling was also completed by Bluestone technicians. The typical sample lengths are 1.0 to 1.5 m with a minimum sample width of 1 m and maximum lengths of 2.0 m; sample lengths were based on the lithology and alteration. Samples are collected along the footwall, mineralized zones, and hanging walls without breaks in sampling. All data was initially captured on paper logs and later transferred to Microsoft Excel. The data was then entered into MapInfo™ and MineSight™ software for geological modelling.

Specific gravity readings of all representative lithologies and vein material were taken during the various drill campaigns using the displaced water method. Samples were sealed with paraffin wax to account for natural voids/vugs.

A total of 591 channel samples were taken along representative veins exposed in the side walls of the Cerro Blanco underground tunnels using a portable rock saw. The sampling was undertaken across and perpendicular to the mineralized structures wherever possible and carefully surveyed with XYZ coordinates for use in 3D modelling. The samples were subject to the same QA/QC protocols as drill core and were deemed suitable for use in calculating resources. Figure 11-2 shows a saw-cut channel sample across a mineralized vein in the South Ramp of the Cerro Blanco underground workings.

**Figure 11-2: Example of Underground Channel Sample**

Source: Bluestone (2019).

Samples were transported in security-sealed bags to Inspectorate Laboratories in Guatemala City for sample preparation until March 2020, and thereafter to Inspectorate Laboratories in Managua due to closure of the Guatemalan facility. Samples were prepared at Inspectorate by crushing and pulverizing the drill core down to 85% passing -75 µm. Pulps were weighed and individually packaged into 100 g envelopes and shipped for analysis. Both coarse rejects and pulp were stored for future use and utilized in Bluestone's QA/QC program. All half-core and coarse rejects are stored adjacent to the core logging facility on the Cerro Blanco Gold Project site. The Cerro Blanco site is fully controlled by perimeter fencing and security.

Pulps are shipped for regular and QA/QC analysis to Inspectorate Laboratories (a division of Bureau Veritas) in Reno, Nevada, USA and ALS Chemex in Vancouver, BC, Canada, respectively. Both are ISO 17025 accredited laboratories. Gold and silver were analysed by a 30 g charge with atomic absorption with gravimetric finish for values exceeding 5 g Au/t and 100 g Ag/t.

All analytical results were provided to Bluestone by respective laboratory secure servers in Excel, .csv and .pdf formats (certificates). Bluestone database files are stored and managed in Access and Excel formats before being transferred to MapInfo™ and MineSight™ software.

During Q3 and Q4 2020, the Cerro Blanco database was transitioned to the AcQuire/GMSuite platform, providing an enhanced, secure, and high standard of data management.

## 11.2 Quality Assurance & Quality Control

### 11.2.1 QA/QC Performance & Discussion for Samples prior to 2017

Field blanks of non-mineralized material were inserted into the sample series every 25 samples (4%) to test for any potential carry-over contamination that might occur in the crushing phase of sample preparation due to poor cleaning practices. A total of 1,390 blanks were analysed with 558 performed at Inspectorate Laboratories, 302 at CAS Honduras, and 530 at the Marlin Mine laboratory. An analysis of the Inspectorate blanks resulted in five fails or 0.01%, with one re-failing on resample. This appears to be the result of sample misclassification as both the original and resample are relatively high grade. The CAS Honduras results showed eight fails or 0.03%, with four of those failing on resample. There may have been some cleaning issues at CAS Honduras, although it was not widespread or significant. The blanks from the Marlin Mine laboratory resulted in 14 fails or 0.03%, which is not significant. Considering that the Marlin Mine assaying was utilized for fast turnaround to guide the program and not for resource estimation purposes, this fail rate does not pose an issue.

Core duplicate samples were used to evaluate analytical precision and to determine if any biases exist between laboratories that may affect the overall assay database. The core duplicate samples were quarter-spilt cores sampled on site and sent to Inspectorate Laboratories and CAS Honduras. A total of 1,060 samples with gold values >2 g/t were selected in the drill hole database through hole CB-222. Of those, a total of 797 samples were submitted for check analyses with 618 samples being submitted to Inspectorate for checks of original CAS Honduras analyses, while 179 samples were submitted to CAS Honduras for checks of original Inspectorate analyses. The 618 Inspectorate duplicate check samples show the CAS Honduras original samples to be 3% higher in gold and 16% higher in silver on an individual basis and 3% and 2.8% higher in gold and silver, respectively, on an overall basis.

The 179 CAS Honduras duplicate check samples show the Inspectorate original samples to be 1.5% lower in gold and 27% lower in silver on an individual basis, and 6.8% and 11.4% lower in gold and silver, respectively, on an overall basis.

Duplicate analyses from both labs show high variation in individual gold values, potentially attributable to nugget effect particularly for higher grade samples. However, on average, the samples show a better correlation which has greater implications on a global or resource scale. The CAS Honduras check samples appeared to show a relatively small grade bias.

Standards are used to test the accuracy of the assays and to monitor the consistency of the laboratory over time. Neither Glamis nor Goldcorp employed the use of standards. It was recommended that a QA/QC program be implemented during all future drill programs that includes the insertion and analysis of standards, blanks, and duplicates, as well as umpire assays.

### 11.2.2 QA/QC Performance & Discussion of Results (Bluestone 2017 to 2021)

Since 2017, Bluestone has implemented a comprehensive QA/QC program employing industry standards and best practices for all its drill core and channel sampling. This includes the insertion of blind certified reference materials (blanks and standards) into the sample stream, in addition to field blanks. Furthermore, duplicate analysis of pulps and coarse rejects was performed at a second laboratory to independently assess analytical precision and accuracy of each samples batch as they are received from the laboratory. Additionally, pulp and coarse rejects were systematically submitted to ALS Chemex Laboratories in Vancouver for check analysis and additional quality control.

A total of 7,652 control samples (Table 11.1) were assigned for QA/QC purposes, accounting for approximately 20% of the total samples taken during the program.

**Table 11.1: Quantity of Control Samples by Type (Bluestone 2017 to 2021)**

| Control Type | Number |
|---|---|
| Standards | 1,602 |
| Field Blanks | 685 |
| Pulp Blanks | 859 |
| Pulp and Coarse Reject Duplicates | 4,506 |
| Total | 7,652 |

Source: Bluestone (2021).

Standards are used to test the accuracy of the assays and to monitor the consistency of the laboratory over time. A variety of certified standards of various gold grades were purchased from CDN Laboratories (Table 11.2) and inserted by the logging geologists.

**Table 11.2: Summary of Standards (Bluestone 2017 to 2021)**

| Control Sample | Au PPM | Standard Deviation | Analysis |
|---|---|---|---|
| CDN-GS-16 | 16.48 | 0.315 | Fire Assay Gravimetric |
| CDN-GS-11B | 11.04 | 0.44 | Fire Assay Gravimetric |
| CDN-GS-6F | 6.79 | 0.15 | Fire Assay Gravimetric |
| CDN-GS-6E | 6.06 | 0.16 | Fire Assay Gravimetric |
| CDN-GS-5T | 4.76 | 0.105 | Fire Assay AA Finish |
| CDN-GS-1W | 1.063 | 0.038 | Fire Assay AA Finish |
| CDN-GS-1T | 1.08 | 0.05 | Fire Assay AA Finish |
| CDN-GS-1X | 1.299 | 0.06 | Fire Assay AA Finish |
| CDN-BL-10 | <0.01 | - | Fire Assay AA Finish |
| FIELD BLANKS | <0.01 | - | Fire Assay AA Finish |

Source: Bluestone (2021).

Field blanks are non-mineralized material sourced locally that are inserted into the sample series every 20 samples (5%). Field blanks are inserted to test for any potential carry-over contamination that might occur in the crushing phase of sample preparation due to poor laboratory cleaning practices.

Duplicate analysis of pulps and quarter-core are used to evaluate the analytical precision and to determine if any biases exist between laboratories. Duplicate analysis of coarse rejects is used to analyse preparation error. Table 11.3 shows the QA/QC sample insertion rate.

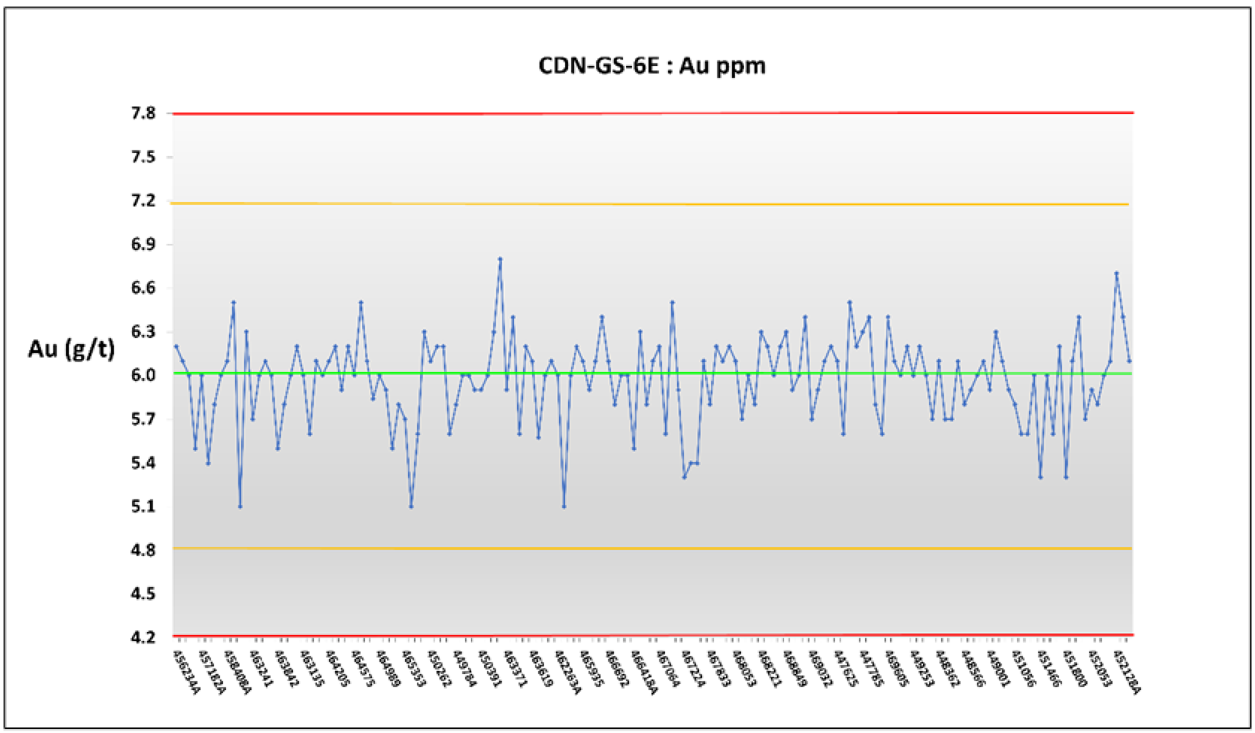
QA/QC assay results were checked by a Bluestone database QA/QC manager on a batch-by-batch basis for analytical or batch errors. No evidence of obvious analytical bias was noted. Figure 11-3 shows a control plot for standard CDN-GS-6E.

**Table 11.3: Bluestone QA/QC Sample Insertion Rates**

| Batch Size – 45 Samples | Minimum Insertion Rates | Notes |
|---|---|---|
| Standards | 1 every 20 | Inserted according to estimated grade of mineralization before, within or immediately after a mineralized interval. Insertion at regular intervals avoided. |
| Field Blanks | 1 every 20 | Usually inserted at the end of mineralized runs to measure carry-over |
| Pulp Blanks | 1 every 20 | Usually inserted at the end of mineralized runs to measure carry-over |
| Pulp Duplicates | 1 every 20 | Undertaken at second laboratory with same analytical technique. High- and low-grade mineralized samples are usually chosen |
| Coarse Duplicates | 1 every 20 | Normally choose mineralized samples, used to measure laboratory sample preparation |

Source: Bluestone (2020).

**Figure 11-3: Batch Plot of Standard CDN-GS-6E**

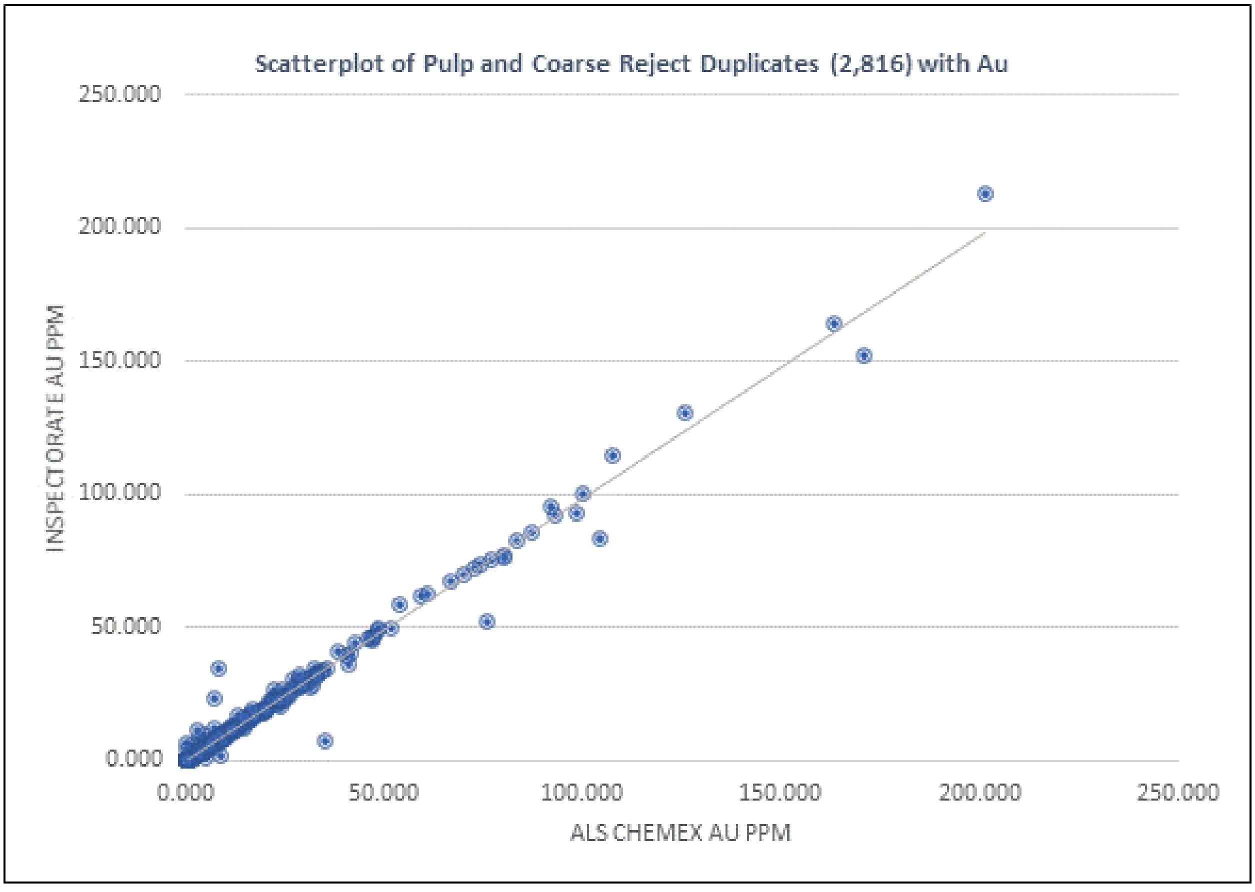



Source: Bluestone (2020).

Except for one standard, the performance of the control samples was very good, reflecting the overall high quality of the analysis. Standard CDN-GS5T (4.76 g Au/t) utilized early in the Bluestone drill program plotted consistently along the highest acceptable threshold for fire assay with instrumental finish. Check analysis at both Inspectorate and ALS Chemex laboratories gave similar results. As lower-grade CRM / blanks and the laboratories' internal QA/QC procedures ruled out any calibration issues, use of this particular standard was discontinued.

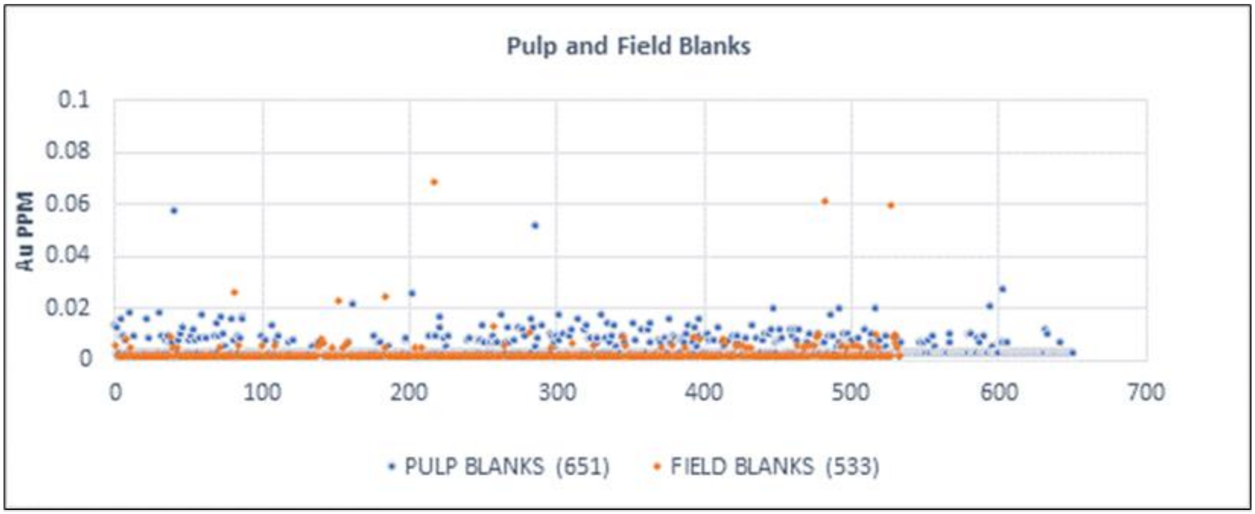
Duplicates of pulp and coarse rejects were sent to ALS Chemex in Vancouver for check gold analysis with the analysis at the principal laboratory, Inspectorate Laboratories in Reno. As shown in Figure 11-4, the results indicate very good correlation at both low and high gold levels and excellent reproducibility between the two laboratories, with a correlation coefficient of 0.993. The results can be interpreted as a reflection of the micron-sized nature of the gold and lack of coarse nuggety gold in the Cerro Blanco deposit. Analyses of both pulp and field blanks (Figure 11-5) consistently yielded gold values near or below the detection limit of the primary laboratory. No sample contamination was detected.

**Figure 11-4: Plot of Pulp & Coarse Reject Duplicates (Bluestone 2017-2021)**




Source: Bluestone (2017 to 2021).

**Figure 11-5: Pulp & Field Blanks (Bluestone 2017 to 2021)**




Source: Bluestone (2021).

It is the opinion of the QP, Garth Kirkham, P. Geo., that the sampling preparation, security, analytical procedures and quality control protocols used by Bluestone are consistent with generally accepted industry best practices and are therefore reliable for the purpose of resource estimation.

# 12. DATA VERIFICATION

Multiple site visits were conducted by several of the QPs, as detailed in Section 2.2. The purpose of these visits was to fulfil the requirements specified under N.I. 43-101 guidelines and become familiar with the property. These site visits consisted of underground tours of mineralized and non-mineralized headings, as well as an inspection of the surface core logging, sampling, storage areas, and existing infrastructure.

## 12.1 General Project & Mining

Mathieu Gignac, P. Eng., visited the property from November 13 to 26, 2020, February 15 to 25, 2021 and February 10 to 12, 2022. The site visits included an inspection of the property, offices, underground vein exposures, core storage facilities, water treatment plant, and stockpiles; a tour of major centers and surrounding villages most likely to be affected by any potential mining operation; and a detailed review of the existing Environmental and Social Impact Assessment (ESIA) and permits. All other data related to permitting, water management and environment were reviewed in detail to ensure authenticity.

Project data was verified during multiple site visits and a thorough review of previous studies completed for the site. Studies were updated and revised as required to align with the FS mine design and mine plan. Furthermore, all mine optimization, planning and design data including operating cost estimates were verified during thorough reviews of previous studies completed for the site. Studies were updated and revised as required to align with the FS mine design and mine plan. It is Mr. Gignac's opinion that the available project and Mining data is adequate and reliable for an FS-level technical report as required by N.I. 43-101 guidelines.

## 12.2 Geology, Drilling & Assaying

Garth Kirkham, P. Geo., has been involved with the property since its acquisition in early 2017 when he performed the initial due diligence and authored the updated resource estimate for Bluestone. Mr. Kirkham first visited the property on May 8, 2017 to satisfy the site visit requirements related to the 2017 N.I. 43-101 Technical Report. The site visit included an inspection of the property, offices, underground vein exposures, core storage facilities, water treatment plant, and stockpiles, and a tour of major centers and the surrounding villages most likely to be affected by any potential mining operation.

In September 21 to 22, 2017, Mr. Kirkham inspected the progress with the recommended historic drill core rehabilitation program and initiated structural studies.

In, April 24 to 28, 2018, Mr. Kirkham's site visit focused on advancing the planning of sampling and drilling along with supporting lithological and structural modelling.

In February 16 to 22, 2020, Mr. Kirkham provided guidance on the planning and development of advanced drilling and sampling, as well as grade vein modelling.

In January 10 to 15, 2021, Mr. Kirkham assisted with validating drill and sample data, refining high-grade models, reviewing low-grade models, and providing guidance for the finalization of the open pit bulk tonnage resource scenario.

Continued data validation and verification processes have not identified any material issues with the Cerro Blanco sample and assay data. Mr. Kirkham is satisfied that the assay data is of suitable quality to be used as the basis for this resource estimate.

During Q3 and Q4 2020, the Cerro Blanco drill and assay database was switched over to the AcQuire - GMSuite platform hosted by CSA Global, providing an enhanced and more secure standard of data management.

Mr. Kirkham is confident that the data and results are valid and can be relied upon. Mr. Kirkham is also confident that the methods and procedures used are reliable. It is the opinion of Mr. Kirkham that all work, procedures, and results have adhered to best practices and industry standards as required by N.I. 43-101.

## 12.3 Metallurgy

Historical metallurgical testing was performed on Cerro Blanco samples by Kappes, Cassiday & Associates (KCA) between 1999 and 2012, with auxiliary testing carried out by SGS Lakefield Research Ltd., Carson GeoMIn Inc., Pocock Industrial Inc., Phillips Enterprises Inc. and CyPlus GmbH. Most recent metallurgical testwork programs were completed in 2018, 2020, 2021 and 2022 by Base Metallurgical Laboratories Ltd. (BaseMet) in Kamloops, British Columbia, Canada, and in 2020 by Paterson & Cooke (P&C) in Sudbury, Ontario, Canada.

Drill core used for metallurgical testwork completed by BaseMet were plotted against the planned area to be mined and were found to be spatially representative and provide sufficient variability in head grade for both gold and silver. For the FS-level metallurgical testwork program, lower grade Salinas ore and two composites representing the initial mining years and used together with high-grade samples to derive recovery curves. This equation was used to estimate recovery for the first six years of mining. A separate recovery equation was developed for Salinas zones 1 and 2 which is used together with the main equation for subsequent mining years.

Neil Lincoln, P. Eng., is the QP and author of Section 13 and Section 17. It is Mr. Lincoln’s opinion that there is sufficient data and testwork to estimate metallurgical recoveries and define the flowsheet at an FS level as required by N.I. 43-101 policy and CIM Best Practice Guidelines for Mineral Processing. Mining.

## 12.4 Geotechnical

### 12.4.1 Pit Slopes & Stability

Norberto Contador, P.E. visited site on September 22 to 24, 2021, Mr. Contador reviewed the current site conditions and gathered additional data for the pit stability study that E Mining Technology designed. Based on this study and his review, he proposed the design parameters for the open pit. E Mining approved the final mine design for the Cerro Blanco Gold Project used in this FS.

Mr. Contador is the QP for Section 16.2.2 and the design parameters used in the mine plan. It is Mr. Contador's opinion that the design parameters are adequate and reliable for an FS-level technical report as required by N.I. 43-101 guidelines.

## 12.5 Infrastructure

### 12.5.1 Roads, Buildings & Services

Joël Lacelle, P. Eng., visited the site March 30, 2021 and confirmed the quality and location of the different roads, buildings and services (e.g., electrical and water distribution). Mr. Lacelle also observed available equipment and its status at site.

Project infrastructure was developed based on available data at this stage of development. Capital costs were estimated from FS-level engineering designs as well as updated equipment, material and labor pricing from national and international sources. .

It is the opinion of Mr. Lacelle that the data is adequate and reliable for an FS-level technical report as required by N.I. 43101 guidelines.

### 12.5.2 Dry Stack Tailings Facility

Carl J. Burkhalter, P.E., visited the site on October 13 and 14, 2020, Mr. Burkhalter reviewed the current site conditions and gathered additional data as needed. Based on his review, he proposed and supervised an FS-level geotechnical field and laboratory investigations, including confirmation of the DSTF and surface infrastructure design. A site-specific seismic hazard report was developed for use in future work.

Mr. Burkhalter is the QP for Section 18.13 and the information presented on the proposed DSTF. It is Mr. Burkhalter's opinion that the available geotechnical data is adequate and reliable for an FS-level technical report as required by N.I. 43-101 guidelines.

### 12.5.3 Mine Water Treatment Plant Design

H.C. Liang, P. Chem., is the QP and co-author of the mine water treatment plant design sections. It is Mr. Liang's opinion that there is sufficient data and testwork relevant to water treatment design for an FS-level report as required by N.I. 43-101 guidelines.

### 12.5.4 Geochemical Characterization

Jim B. Finley, P. Geo, is the QP and co-author of the report sections regarding geochemical characterization and prediction of waste rock and tailings. It is Mr. Finley's opinion that there is sufficient data and testwork relevant to geochemical characterization for an FS-level report as required by N.I. 43-101 guidelines.

### 12.5.5 Environmental Studies

Rolf Schmitt, P. Eng, is the QP and the author of the environmental studies, permitting and social or community section. It is Mr. Schmitt's opinion that there is sufficient data and testwork relevant to the environmental studies for an FS-level report as required by N.I. 43-101.

# 13. MINERAL PROCESSING & METALLURGICAL TESTING

## 13.1 Introduction

This section discusses and summarizes the relevant results used as the basis for the process plant design and recovery methods presented in Section 17.

Metallurgical testwork was previously completed at Base Metallurgical Laboratories Ltd. (BaseMet) in Kamloops, Canada on ore samples at depth when the Project was considering an underground mine operation. Recent metallurgical testwork at BaseMet focused on testing ore samples near surface in the Salinas Zone to support the FS and current proposed open pit mine operation as well as composite samples representing the initial years of mining. Gold deportment studies were also completed by Surface Science Western (SSW), a department at the University of Western Ontario, and vendor tailings dewatering tests were completed by Metso:Outotec using their laboratory equipment at SGS Lakefield.

Previous and recent metallurgical testwork results and modelling reports are listed below for reference:

- Kappes, Cassiday & Associates (1999). “Cerro Blanco Project, Results of Cyanide Leach Tests” (Issued: April 8, 1999).
- Kappes, Cassiday & Associates (2000a). “Cerro Blanco Project, Results of Cyanide Bottle Roll Tests” (Issued: January 12, 2000).
- Kappes, Cassiday & Associates (2000b). “Cerro Blanco Project, Bottle Roll Tests” (Issued: August 24, 2000).
- Kappes, Cassiday & Associates (2002). “Cerro Blanco Project, Results of Leaching Tests and Gravity Concentration Tests” (Issued: October 8, 2002).
- SGS Lakefield Research Ltd. (2005). “Cerro Blanco North Zone Samples for Metallurgical Testing at SGS-Lakefield” (Issued: August 2005).
- Kappes, Cassiday & Associates (2005). “Cerro Blanco Project” (Issued: December 15, 2005).
- Carson GeoMIn Inc. (2005). “Mineralogy of Ore Composites and Related Cyanide Tailings from the Cerro Blanco Gold Project” (Issued: December 29, 2005).
- Kappes, Cassiday & Associates (2006a). “Cerro Blanco Project” (Issued: January 18, 2006).
- Kappes, Cassiday & Associates (2006b). “Cerro Blanco Project” (Issued: April 21, 2006).
- Phillips Enterprises LLC (2011). “Comminution Tests, Cerro Blanco” (Issued: June 11, 2011).
- Pocock Industrial Inc. (2011). “Sample Characterization and PSA, Flocculant Screening, Gravity Sedimentation, Pulp Rheology, Vacuum Filtration and Pressure Filtration Studies Conducted for Kappes, Cassiday & Associates Cerro Blanco Project” (Issued: October 2011).
- Kappes, Cassiday & Associates (2012). “Cerro Blanco Project, Report of Metallurgical Testwork, January 2012” (Issued: January 25, 2012).

- Base Metallurgical Laboratories Ltd. (2018) "BL0246: Process Optimization and Tailings Generation – Cerro Blanco Project" (Issued: September 26, 2018).
- Base Metallurgical Laboratories Ltd. (2020a) "BL0569: Metallurgical Testing of Samples from the Cerro Blanco Project" (Issued: May 28, 2020).
- Base Metallurgical Laboratories Ltd. (2020b) "BL0593: Metallurgical Variability Assessment and Optimization of Cerro Blanco" (Issued: October 29, 2020).
- Base Metallurgical Laboratories Ltd. (2021) "BL0727: Preliminary Metallurgical Assessment of the Salinas Zone" (Issued: September 24, 2021)
- Base Metallurgical Laboratories Ltd. (2022) "BL0723: Metallurgical Variability Assessment of the Salina Open Pit Zone, Cerro Blanco" (Issued: February 25, 2022).
- Paterson & Cooke Canada Inc. (2020) "32-0399-00-TW-REP-0001 Rev. A Testwork Report" (Issued: December 18, 2020).
- FLSmidth (2020) "Gravity Modelling Report" (Issued: May 28, 2020).
- Metso:Outotec (2022a) "Filtration Test Report" (Issued: January 18, 2022)
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- Orway Mineral Consultants (2020a) "7251.40-RPT-001 Rev. 0 - Cerro Blanco Stage 1 Comminution Circuit Design" (Issued: May 1, 2020).
- Orway Mineral Consultants (2020b) "7251.40-RPT-002 Rev. 0 - Cerro Blanco Stage 2 Comminution Circuit Design" (Issued: November 11, 2020).
- Orway Mineral Consultants (2020c) "7266.40-RPT-001 Rev. 0 - Cerro Blanco SAB Comminution Circuit Modelling" (Issued: November 11, 2020).
- Orway Mineral Consultants (2021) "7269.40-RPT-001 Rev. 0 – Stage 1 Mill Sizing Evaluation for 15 kt/d" (Issued: January 18, 2021).
- Orway Mineral Consultants (2022) "7269.40-RPT-002 Rev. 1 – Stage 2 Mill Sizing Evaluation for 4 Mtpa" (Issued: February 9, 2022).
- Surface Science Western (2021) "SSW Analysis Report: 33221SD.BRI – Gold Deportment Report" (Issued November 25, 2021).


## 13.2 BaseMet (2018) Metallurgical Testwork Program

The primary objective of the BaseMet (2018) metallurgical testwork program was to generate tailings from a bulk sample for downstream geotechnical and environmental studies. The bulk sample was separated into north and south areas of the deposit at depth and prepared to create two bulk composites. Sub-samples from the north and south were tested using the optimized flowsheet to confirm gold and silver extractions. Limited process optimization testwork was also conducted to further the understanding and optimization of the processing characteristics to support preliminary designs for the previous FS.

The testwork program included sample preparation, interval assaying, gravity concentration, cyanide leach optimization, and bulk cyanide leaching to produce material for continuous cyanide destruction testwork. A single global composite was created from drill core intervals to carry out the gravity concentration, cyanide leach and cyanide destruction testing.

### 13.2.1 Sample Selection

Samples were received on April 6, 2018 at BaseMet in two forms. Approximately 90 kg arrived as cut drill core (one-quarter and one-half core) and about 590 kg arrived as bulk rock samples. In total, 180 individual interval samples were received.

The global composite was created using the individual drill core. The drill core was initially inspected and weighed. Each interval was then individually stage crushed to a nominal 3.36 mm (6 mesh). The crushed material was blended, and a 250 g sample was riffle split and pulverized for subsequent assaying.

A representative sub-sample of the global composite was removed during sample preparation and pulverized. The head assay results are shown in Table 13.1.

**Table 13.1: Head Assays**

| Composite | Au (g/t) | Ag (g/t) | Cu (%) |
|---|---|---|---|
| Global Composite Head 1 | 4.21 | 23 | 0.007 |
| Global Composite Head 2 | 5.65 | 21 | 0.007 |
| Average | 4.93 | 22 | 0.007 |

Source: BaseMet (2018).

### 13.2.2 Gravity Concentration Results

One-kilogram test charges were ground in a laboratory rod mill to three target grind sizes of $P_{80}$ of 50 µm, 75 µm and 100 µm before being passed through a laboratory Knelson MD-3 centrifugal gravity concentrator. Knelson concentrates were then panned to reject entrained gangue, targeting a 0.1% to 0.5% mass recovery. The gravity concentration results, which are presented in Table 13.2, indicate moderate gravity recoverable gold. Gravity results do not show a definitive relationship between gold recovery and grind size.

### 13.2.3 Bottle Roll Leach Results

Leaching testwork was carried out on whole ore and gravity tailings samples. All tests were completed in closed bottles on rolls, allowing constant agitation of the pulp as the sample leached for 72 hours. Cyanide levels, dissolved oxygen, and pH were monitored and controlled throughout each test. Kinetic sampling was done at 2, 6, 24, 48 and 72 hours.

The optimization testwork focused on the effect of leach time, pre-oxidation, lead nitrate addition, and primary grind size on gold recovery and leach kinetics. The results are summarized in Table 13.3.

**Table 13.2: Gravity Concentration Results**

| Test No. | Test Type | Grind Size (µm) | Mass Recovery (%) | Au Recovery (%) |
|---|---|---|---|---|
| 4 | Gravity / Leach | 50 | 0.317 | 22.5 |
| 10 | Gravity / Leach | 50 | 0.186 | 21.1 |
| 11 | Gravity / Leach | 50 | 0.230 | 15.1 |
| 17 | Gravity / Leach | 53 | 0.319 | 20.8 |
| 18 | Gravity / Leach | 53 | 0.301 | 17.9 |
| 19 | Gravity / CIL | 53 | 0.274 | 16.7 |
| 20 | Gravity / CIL | 53 | 0.326 | 21.7 |
| 21 | Gravity / CIL | 53 | 0.185 | 16.3 |
| 25 | Gravity / CIL | 53 | 0.173 | 33.2 |
| 26 | Gravity / CIL | 53 | 0.692 | 28.6 |
| 27 | Gravity / CIL | 53 | 0.309 | 25.0 |
| 2 | Gravity / Leach | 75 | 0.239 | 29.8 |
| 6 | Gravity / Leach | 75 | 0.270 | 14.7 |
| 7 | Gravity / Leach | 75 | 0.314 | 20.7 |
| 8 | Gravity / Leach | 75 | 0.398 | 20.0 |
| 3 | Gravity / Leach | 75 | 0.480 | 17.7 |
| 12 | Gravity / Leach | 75 | 0.291 | 15.9 |
| 5 | Gravity / Leach | 100 | 0.534 | 15.6 |

Source: BaseMet (2018).

**Table 13.3: Bottle Roll Leach Results**

| Test No. | Grind Size (µm) | Consumption | | Gravity Au Recovery (%) | Cumulative Gold Extraction (%) | | | | Final 72 h Recovery | |
|---|---|---|---|---|---|---|---|---|---|---|
| | | NaCN (kg/t) | Lime (kg/t) | | 2 h | 6 h | 24 h | 48 h | Au (%) | Ag (%) |
| 4 | 50 | 0.84 | 0.87 | 22.5 | 75.3 | 92.4 | 95.9 | 95.7 | 96.1 | 92.4 |
| 10 | 50 | 0.36 | 1.17 | 21.1 | 78.9 | 92.4 | 94.4 | 95.7 | 97.5 | 69.6 |
| 11 | 50 | 0.52 | 1.02 | 15.1 | 81.5 | 92.7 | 94.0 | 96.5 | 97.3 | 78.6 |
| 17 | 53 | 0.52 | 1.22 | 20.8 | 91.9 | 93.2 | 94.2 | 95.2 | 95.9 | 88.4 |
| 18 | 53 | 0.50 | 1.12 | 17.9 | 82.2 | 91.6 | 96.6 | 95.1 | 96.1 | 92.3 |
| 19 | 53 | 0.86 | 1.33 | 16.7 | 87.7 | 94.9 | 98.1 | 97.4 | 94.5 | 70.8 |
| 20 | 53 | 0.60 | 1.50 | 21.7 | 80.7 | 90.9 | 94.1 | 92.8 | 96.3 | 69.7 |
| 21 | 53 | 0.28 | 0.96 | 16.3 | 80.9 | 90.1 | 91.5 | 91.6 | 94.7 | 67.2 |
| 2 | 75 | 0.82 | 0.86 | 29.8 | 61.2 | 78.5 | 91.9 | 94.1 | 94.7 | 86.9 |
| 6 | 75 | 0.82 | 0.84 | 14.7 | 82.9 | 90.4 | 92.4 | 93.1 | 94.2 | 82.9 |
| 7 | 75 | 1.00 | 0.71 | 20.7 | 77.8 | 90.8 | 92.9 | 93.7 | 94.4 | 84.2 |
| 8 | 75 | 0.46 | 0.82 | 20.0 | 75.0 | 88.3 | 92.7 | 93.2 | 93.6 | 83.1 |
| 3 | 75 | 0.76 | 0.89 | 17.7 | 68.6 | 87.2 | 92.5 | 93.0 | 94.0 | 93.2 |
| 12 | 75 | 0.20 | 1.00 | 15.9 | 80.6 | 89.3 | 91.7 | 93.8 | 95.6 | 65.2 |
| 5 | 100 | 0.58 | 0.71 | 15.6 | 66.3 | 82.4 | 91.2 | 91.7 | 91.9 | 82.7 |
| 1 | 75 | 2.98 | 0.50 | no gravity | 3.2 | 10.1 | 88.4 | 92.2 | 93.1 | 84.7 |
| 9 | 75 | 0.90 | 0.77 | no gravity | 67.4 | 86.5 | 92.6 | 92.1 | 94.4 | 86.3 |

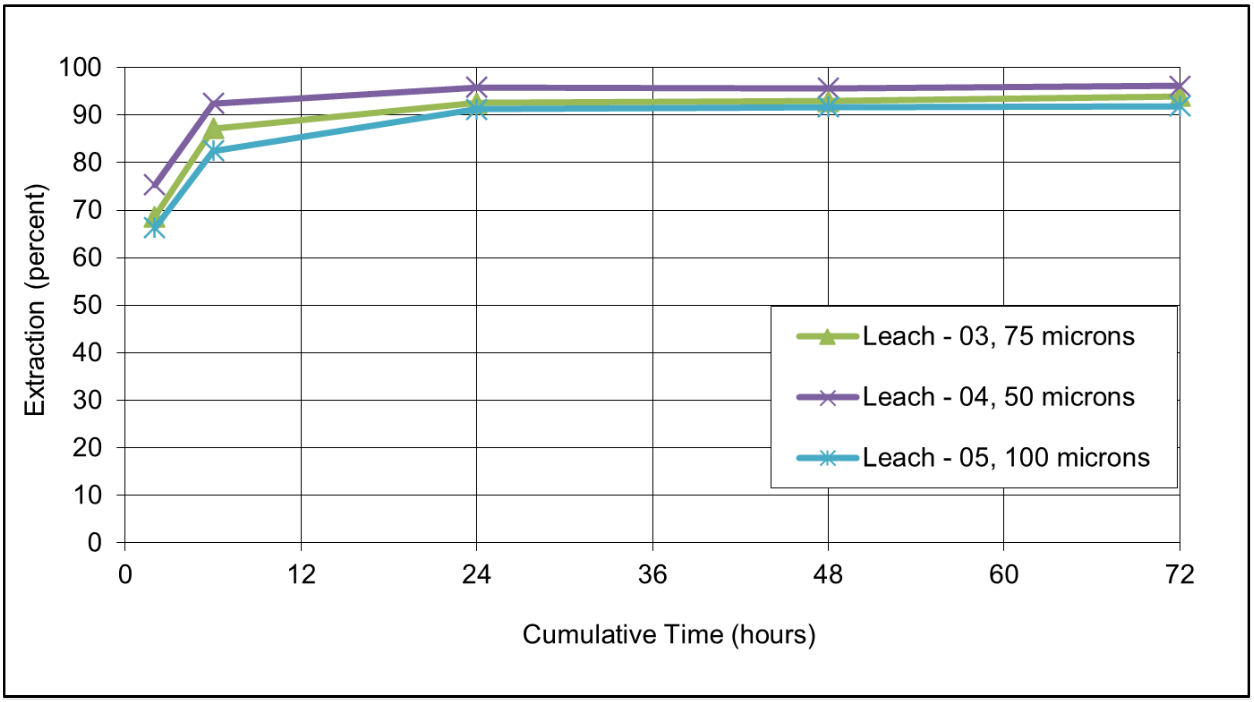
Source: BaseMet (2018).

A decrease in grind size was found to improve precious metal recovery. As grind size decreased from a $P_{80}$ of 100 µm to a $P_{80}$ of 50 µm, gold and silver recovery improved by 4.2% and 9.7%, respectively (72-hour leach time). Figure 13-1 shows gold extraction versus time at the different grind sizes.

Tests completed at a $P_{80}$ of 50 µm showed an increase in silver recovery from 78.6% (Test 11) to 88.4% (Test 17) with the addition of lead nitrate. However, in contrast tests completed at a $P_{80}$ of 75 µm showed a decrease in silver recovery from 86.9% (Test 2) to 82.9% (Test 6) with the addition of lead nitrate.

Pre-oxidation of the slurry with oxygen resulted in improved leach kinetics. Test 1 did not include pre-oxidation and resulted in a measured dissolved oxygen of below 1 ppm until after six hours of leaching. This had a significant impact on initial leach rates in the first 24 hours. When pre-oxidation was incorporated under similar conditions in Test 12, leach kinetics improved considerably.

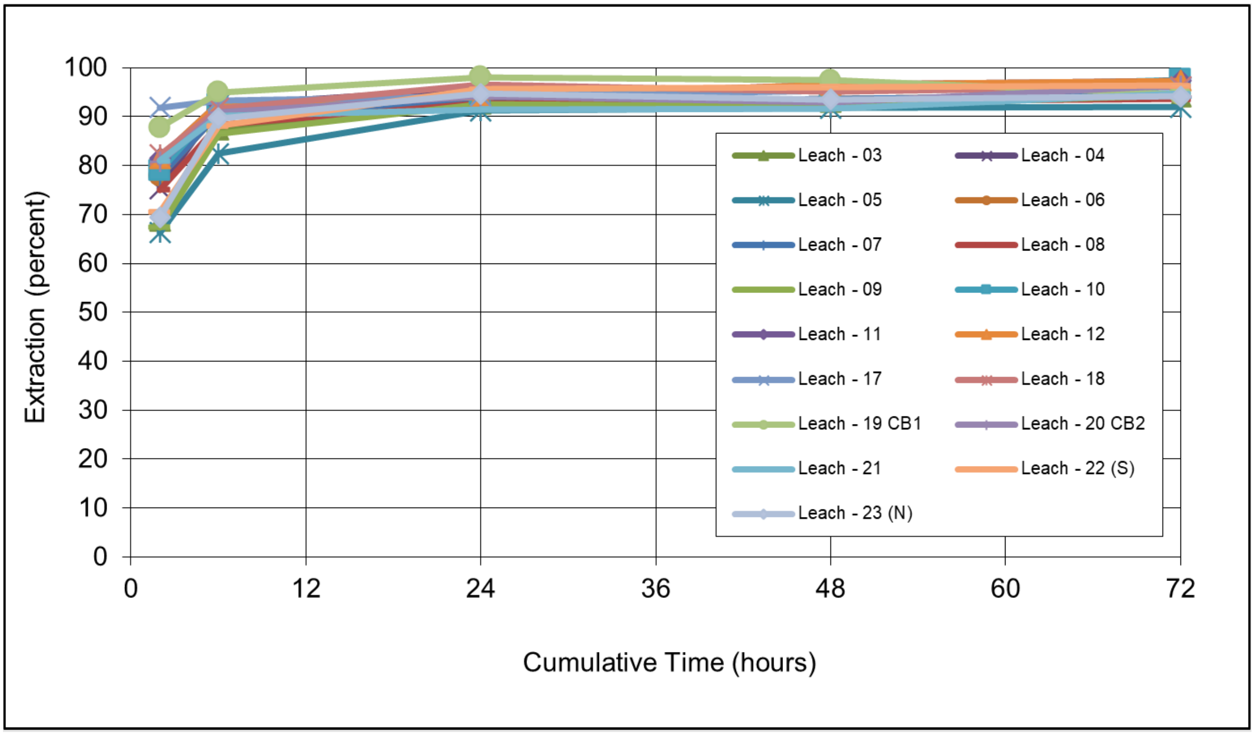
**Figure 13-1: Effect of Grind Size on Gold Extraction**




Source: BaseMet (2018).

Tests were conducted using the optimized flowsheet and testwork parameters to investigate gold and silver extraction at 40°C, with site treated water (CB-1), untreated site water (CB-2), and bulk rock sample composites from the north and south deposits. Gold extractions results ranged from 94% to 96% and silver ranged between 77% and 92%. The general trend for all tests showed that there is minimal advantage to leaching after 36 hours, as shown in Figure 13-2.

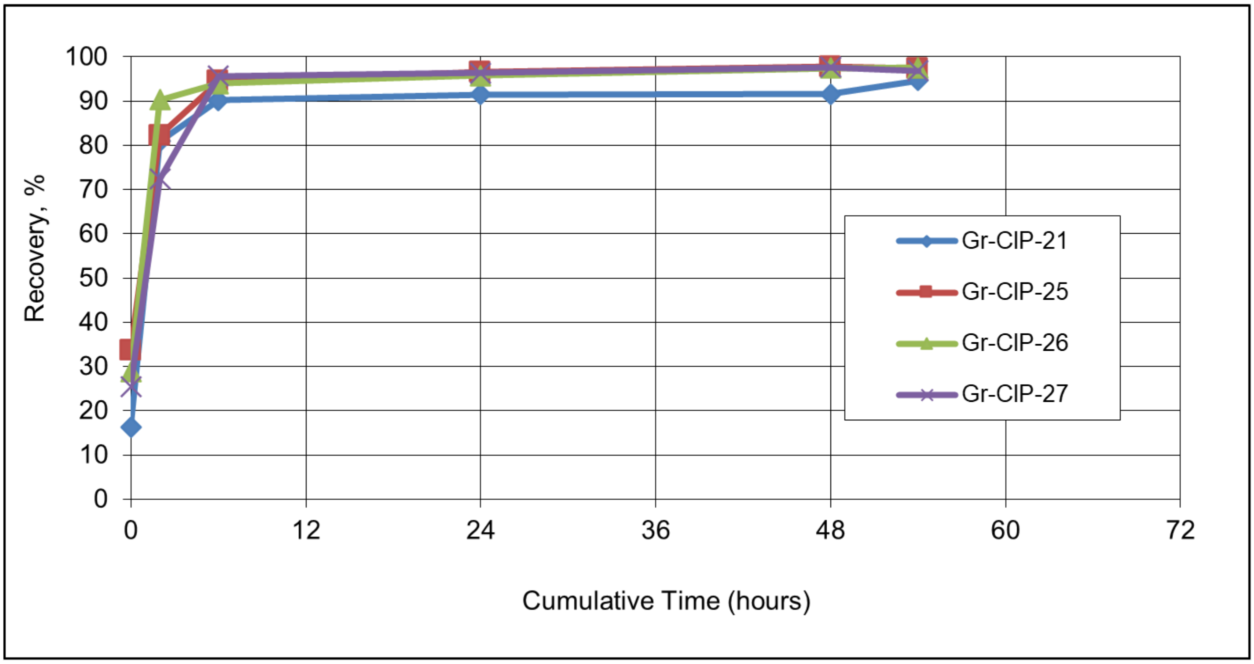
**Figure 13-2: Gold Extraction vs. Time**




Source: BaseMet (2018).

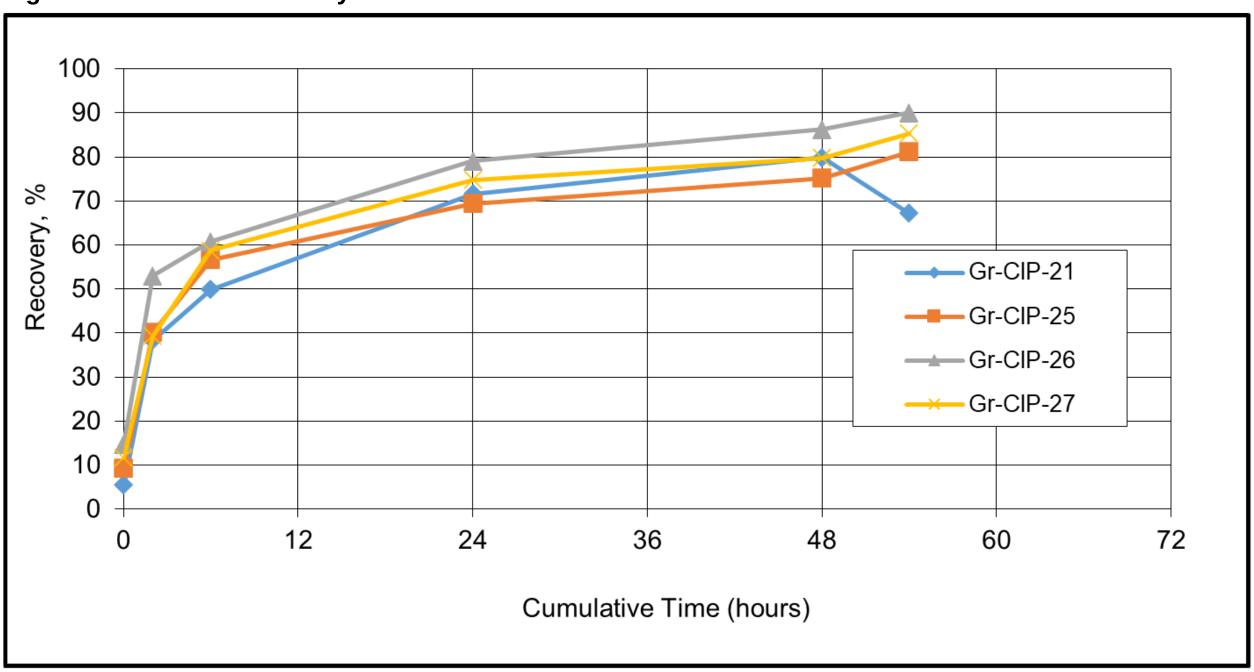
A global composite was further tested to determine the adsorption of gold and silver on carbon. Test CIP-21 was carried out at a carbon concentration was 25 g/L for six hours following the 48-hour leach. The overall recovery was 94.7% for gold and 67.2% for silver. Based on the results an additional three tests, CIP-25, CIP-26 and CIP-27, were completed at 50 g/L carbon. Tests CIP-26 and CIP-27 included the addition of 250 g/t lead nitrate. The three additional tests produced higher recoveries for both gold and silver. The addition of lead nitrate appears to improve silver leach kinetics and final recovery. The recovery curves for gold and silver are illustrated in Figure 13-3 and Figure 13-4, respectively.

**Figure 13-3: Gold Recovery vs. Time**




Source: BaseMet (2018).

**Figure 13-4: Silver Recovery vs. Time**




Source: BaseMet (2018).

### 13.2.4 Cyanide Destruction Test Results

Feed for the cyanide destruction testwork was created from bulk leach tests. To determine cyanide species, a representative pulp sample was taken and filtered. The filtrate was then submitted for analysis. The cyanide solution from the produced pulp contained 283 mg/L total cyanide ($CN_T$), 270 mg/L weak acid dissociable cyanide ($CN_{WAD}$), 10.2 mg/L Fe, and 0.2 mg/L Zn.

Continuous cyanide destruction testwork was completed to produce a treated product using the $SO_2$/air process, targeting less than 5 mg/L $CN_{WAD}$. A batch test (CND-B1) was conducted on the leached pulp to produce a starting pulp with a low residual $CN_{WAD}$. A series of continuous cyanide destruction tests were then completed to establish the cyanide destruction circuit design criteria and understand the effect of reagent dosage on the oxidation of cyanide.

The cyanide pulp produced during the test program responded well to the $SO_2$/air cyanide destruction process, producing a treated pulp with < 1 mg/L $CN_{WAD}$ and < 4 mg/L $CN_T$. The results are shown in Table 13.4.

**Table 13.4: Cyanide Destruction Results**

| Test No. | Retention Time (mins) | pH | Final Solution Composition | | | | Reagent Addition (g/g $CN_{WAD}$) | | Cu (mg/L of solution) |
|---|---|---|---|---|---|---|---|---|---|
| | | | $CN_T$ (mg/L) | $CN_{WAD}$ (mg/L) | Cu (mg/L) | Fe (mg/L) | $SO_2$ Equiv. | Lime | |
| CND-C1 | 90 | 8.5 | 4.59 | 1.8 | 1.02 | < 0.1 | 7 | 4.6 | 100 |
| CND-C2 | 90 | 8.5 | 2.96 | 0.17 | 0.25 | < 0.1 | 5.5 | 2 | 100 |
| CND-C3 | 90 | 8.3 | 0.49 | 0.22 | 0.39 | 0.1 | 4 | 2.2 | 100 |
| CND-C4 | 90 | 8.4 | 2.94 | 0.14 | 0.47 | < 0.1 | 4 | 1.6 | 50 |
| CND-C5 | 90 | 8.5 | 3.02 | 0.24 | 1.2 | < 0.1 | 4 | 1.4 | 25 |
| CND-C6 | 90 | 9.0 | 18.3 | 4.18 | 16.5 | 5 | 4 | - | 0 |
| CND-C7 | 60 | 8.5 | 3.56 | 0.48 | 3.93 | 1.1 | 4 | 0.8 | 25 |

Source: BaseMet (2018).

## 13.3 BaseMet (2020a) Metallurgical Testwork Program

The primary objective of the BaseMet (2020a) metallurgical testwork program was to assess the gold metallurgical response and comminution properties of four samples that represented key major lithologies consisting of a blend of waste and vein material.

The flowsheet developed during the BaseMet 2018 metallurgical testwork program was used as the basis for this testwork program.

The testwork program included sample preparation, Bond ball work index (BWi) testing, CIP gravity leach tests, carbon-in-leach (CIL) and gravity recoverable gold (GRG) tests.

### 13.3.1 Sample Selection

Eight individual samples were received on February 19 and March 3, 2020 at BaseMet as bulk rock. The samples received consisted of either dilution waste or vein material. The samples were crushed to a nominal 3.36 mm (6 mesh) and blended, and approximately 100 to 200 g were riffle split and pulverized for head assays. Upon assay results, blend composites were constructed in preparation for testing. The head assay results are shown in Table 13.5. Detailed ICP analysis is provided in BaseMet's BL0569 report.

**Table 13.5: Head Assays**

| Sample | Au (g/t) | Ag (g/t) | Cu (%) |
|---|---|---|---|
| MET20-01D | 2.64 | 10 | 0.008 |
| MET20-02V | 23.2 | 26 | 0.006 |
| MET20-02B | 16.9 | 20 | 0.006 |
| MET20-03V | 3.10 | 15 | 0.005 |
| MET20-04D | 0.37 | 2 | 0.004 |
| MET20-05D | 0.79 | 10 | 0.004 |
| MET20-06V | 66.4 | 367 | 0.009 |
| MET20-06B | 24.9 | 148 | 0.007 |
| MET20-07D | 0.30 | 8 | 0.003 |
| MET20-08V | 59.6 | 317 | 0.004 |
| MET20-08B | 14.5 | 82 | 0.004 |

Note: D – dilution material, V – vein material, and B – blended. Source: BaseMet (2020).

### 13.3.2 Bond Ball Mill Work Index Results

To assess the hardness of the vein and dilution material, a BWi determination was performed. The tests were completed at a closed screen size of 75 µm which resulted in a $P_{80}$ of approximately 50 µm (see Table 13.6 for a summary of the results). There was very little difference in hardness between the vein and dilution samples.

**Table 13.6: Bond Ball Mill Work Indices**

| Sample | BWi (kWh/t) |
|---|---|
| MET20-01D | 20.5 |
| MET20-02V | 23.2 |
| MET20-02B | 22.5 |
| MET20-03V | 25.6 |
| MET20-06B | 23.9 |
| MET20-07D | 22.7 |
| MET20-08V | 23.0 |

Note: D – dilution material; B – blend; and V – vein material. Source: BaseMet (2020).

### 13.3.3 Metallurgical Response to the Flowsheet

Test conditions previously developed during the BaseMet (2019) metallurgical testwork program were used as a basis for leaching tests for this program.

Gravity concentration using a Knelson concentrator, followed by hand panning, was carried out on 1 kg samples of the MET20-02B, MET20-03, MET20-06B, and MET20-08B composites, ground to a nominal $P_{80}$ of 53 µm. The gravity tailings were then subjected to a CIP cyanide leach test for 54 hours. Tests were conducted at a pulp density of 50% with a two-hour pre-oxidation and with pulp pH monitored and maintained at 10.5 with lime. Cyanide concentration of 500 ppm was maintained, and lead nitrate was added at 250 g/t. Carbon at 50 g/L was added at 48 hours for six hours and the test was sparged with oxygen. Samples were taken at 2, 6, 24, 36, 48 and 54 hours.

The gravity leach results are summarized in Table 13.7.

**Table 13.7: Gravity Leach Results**

| Composite | NaCN (kg/t) | Lime (kg/t) | Gravity Rec. (Au %) | Combined Extraction (%) | | |
|---|---|---|---|---|---|---|
| | | | | Au | Ag | Cu |
| MET20-02B | 0.48 | 1.25 | 4.8 | 97.2 | 79.7 | 41.3 |
| MET20-03 | 0.62 | 3.48 | 4.2 | 91.7 | 88.1 | 50.6 |
| MET20-06B | 0.57 | 1.74 | 14.7 | 97.6 | 93.5 | 52.7 |
| MET20-08B | 0.66 | 2.14 | 3.5 | 98.0 | 84.5 | 43.8 |

Source: BaseMet (2020a).

Additional testwork was completed on the Global South composite at BaseMet (2018); the results are summarized in Table 13.8. The baseline test was Test 7. Overall, the gold and silver extractions were similar for each test after 54 hours. Test 8 used a coarser primary grind size for the leach and resulted in a slightly lower gold recovery of 94.8%. Comparing similar tests with and without gravity concentration indicated no net benefit of using gravity on the final gold and silver recovery of a 54-hour CIP circuit. The test without gravity had slower initial leach kinetics and slightly elevated cyanide consumption.

The use of oxygen over air as a pre-treatment gas and sparge gas was investigated further. The use of air indicated poor initial leach kinetics, which is understandable, as air-sparged systems can only obtain about 7 to 8 ppm dissolved oxygen in the slurry. The use of oxygen increases the dissolved oxygen to over 20 ppm, speeding the reaction of gold dissolution. The cyanide consumption was also increased for the tests that were air sparged.

A CIL test was performed as an alternative to the CIP flowsheet. The metallurgical performance of the circuit was very good, as 96.1% gold and 92.5% silver were recovered. The test did indicate much higher cyanide consumption.

A test performed without lead nitrate resulted in poorer leach kinetics for gold and silver.

**Table 13.8: Summary of Conditions & Overall Metallurgical Response (Global South Composite)**

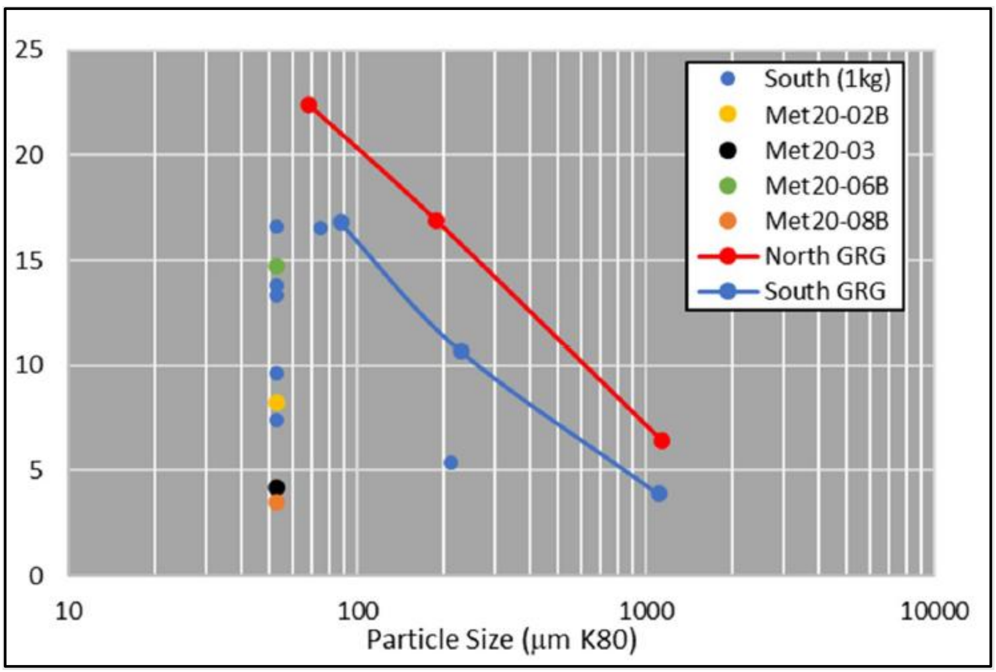
| Test | Grind Size $K_{80}$ | | Pre-treatment | Circuit | Reagents (kg/t) | | | Final Recovery (%) | |
|---|---|---|---|---|---|---|---|---|---|
| | Gravity | Leach | | | Lead Nitrate | NaCN | Lime | Au | Ag |
| T06 | 212 | 53 | $O_2$-2 h | CIP-$O_2$ | 0.25 | 0.59 | 1.32 | 96.3 | 88.7 |
| T07 | 53 | 53 | $O_2$-2 h | CIP- $O_2$ | 0.25 | 0.67 | 1.31 | 95.3 | 90.1 |
| T08 | 75 | 75 | $O_2$-2 h | CIP- $O_2$ | 0.25 | 0.61 | 1.15 | 94.8 | 88.3 |
| T09 | 53 | 53 | $O_2$-2 h | CIP-Air | 0.25 | 0.96 | 1.36 | 95.2 | 86.3 |
| T10 | 53 | 53 | air-24 h | CIP-Air | 0.25 | 0.72 | 1.39 | 95.2 | 86.5 |
| T11 | 53 | 53 | $O_2$-2 h | CIL- $O_2$ | 0.25 | 1.35 | 1.05 | 96.1 | 92.5 |
| T12 | none | 53 | $O_2$-2 h | CIP- $O_2$ | 0.25 | 0.81 | 1.20 | 95.8 | 88.5 |
| T14 | 53 | 53 | $O_2$-2 h | CIP- $O_2$ | none | 0.67 | 1.19 | 95.7 | 88.2 |

Source: BaseMet (2020a).

### 13.3.4 Gravity Recoverable Gold (GRG)

Two gravity recoverable gold (GRG) tests were performed on Global South and Global North samples by BaseMet using the standard test procedure required by FLSmidth to model Knelson gravity concentrators. The GRG results shown in Figure 13-5 include gravity results from batch testwork from various composites.

**Figure 13-5: Gravity Gold Results (Recovery vs. Particle Size)**




Source: BaseMet (2020a).

Since the GRG tests were completed at different grind sizes compared to the target $P_{80}$ of 53 µm, FLSmidth recalibrated the BaseMet results to 18.3% (South) and 24.2% (North).

FLSmidth completed GRG modelling; key pertinent conclusions are noted below:

- Three scenarios were modelled for each of the two samples (i.e., Knelson concentrator treating 33% of the cyclone underflow, 45% of the cyclone underflow, and 60% of the ball mill discharge). The overall gold recovery ranged from 7.8% to 15.6%.
- The mineralized ore is considered low in GRG, with the presence of electrum, and only moderate in coarseness.
- At a grind size of $P_{80}$ of 53 µm, the small quantity of GRG available will be locked in the grinding circuit for an extended period, providing a higher probability of recovery via gravity while using only a conservatively sized gravity circuit. This is also expected to occur at $P_{80}$ of 75 µm.

## 13.4 BaseMet (2020b) Metallurgical Testwork Program

The metallurgical testwork program commenced in June 2020 and was completed by mid-October 2020. A total of 678 kg of sample was available for testing in this program as one-half and one-quarter drill core. BaseMet completed variability and optimization testwork on 29 variability composites prepared from blends of vein and dilution material from the north and south zones, including the main lithologies of Mat, Mbt, Mls, Mss, Mvo and Svc (refer to Section 7.3.1 for descriptions).

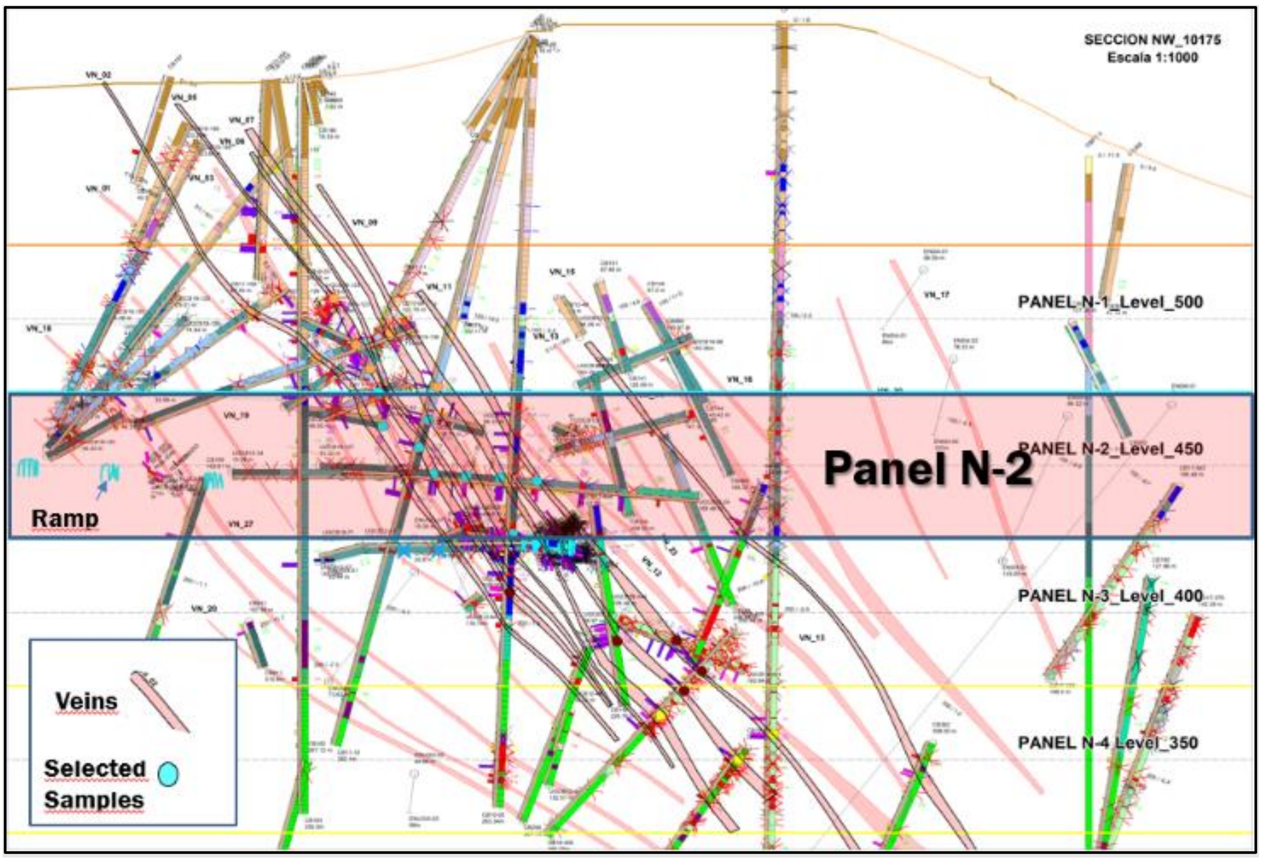
The test program was organized and completed in three phases as follows:

- Phase 1 – Variability Assessment
  - chemical analysis and mineral content determination
  - comminution testing including rod mill work index (RWI), Bond ball mill work index (BWI), and SAG mill comminution (SMC) tests
  - hardness index tester (HIT) A x b values and abrasion index tests
  - gravity / leach versus whole ore leach testing
  - settling tests
- Phase 2 – Optimization Testing
  - whole-of-ore tests testing the sensitivity of primary grind size and cyanide concentration
  - oxygen uptake rate measurement
  - leach kinetic and carbon adsorption tests
  - cyanide detoxification testing
- Phase 3 – Tailings Generation
  - tailings characterization and testing at P&C

### 13.4.1 Sample Selection & Representativity

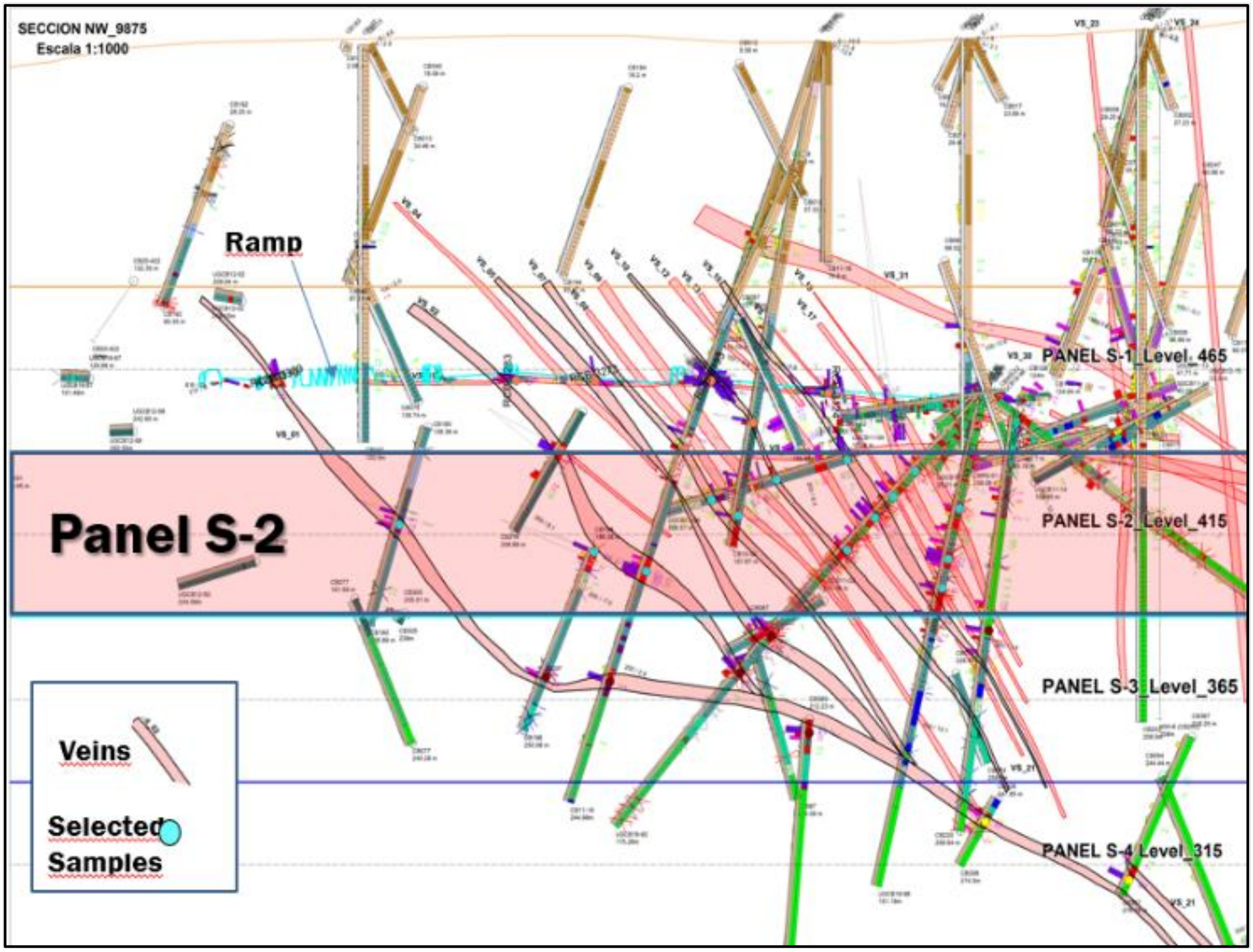
Samples for the BaseMet (2020b) metallurgical testwork program were based on 29 variability composites from the north and south zones. Six sections through the north and south zones were used to provide a representative selection of vein intervals laterally through the deposit. Both north and south were then divided into four elevation panels at 50 m intervals to provide a representative selection of vein intervals vertically through the deposit. The composites were comprised of drill core intervals of veins and adjacent wall rocks (sampled separately). The samples are represented in Figure 13-6 and Figure 13-7.

**Figure 13-6: North Zone Section Showing Drill Holes & Sample Intervals**




Source: Bluestone (2020).

**Figure 13-7: South Zone Section 9,875N Showing Drill Holes & Sample Intervals**




Source: Bluestone (2020).

The main rock types in the north and south zones are listed in Table 13.9. From each panel, drill core intervals of principal veins were chosen with the same lithology that is spatially within the same area to make up the composites. Nine master lithology composites were created on the main lithologies.

**Table 13.9: Major Lithologies**

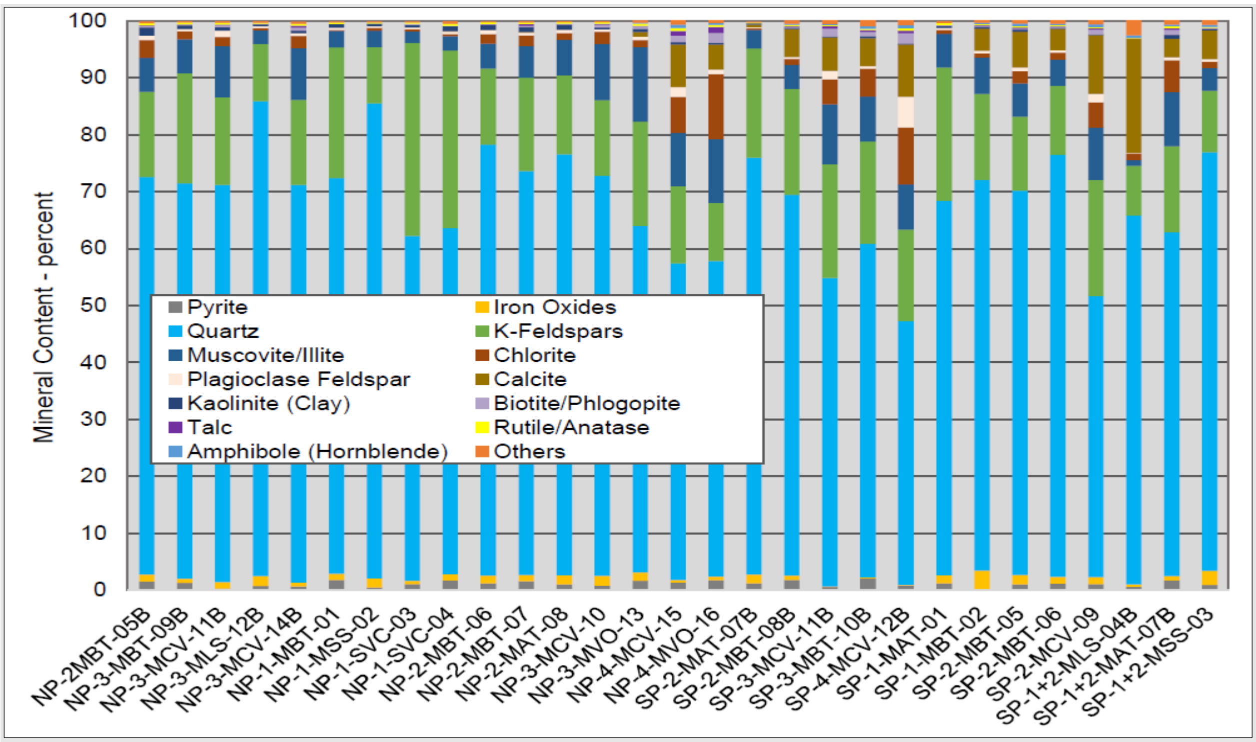
| Lithology | North | South | Description |
|---|---|---|---|
| Mbt | 45% | 37% | Massive, fine grained tuffaceous siltstone with local calcareous concretions and accretionary lapilli |
| Mat | 9% | 14% | Andesitic crystal-rich tuff |
| Mcv | 3% | 25% | Lapilli tuff, andesite-dacite with fiamme and weak welding |
| Mss | 0% | 5% | Coase sandstone, conglomerate, metamorphic quartz pebbles |
| Mls | 6% | 14% | Micritic limestone, fossiliferous, nodular |
| Svc | 12% | 5% | Volcaniclastic sandstones, conglomerates, with rhyolite breccias and sinter horizons |
| Mvo | 25% | 0% | Volcaniclastics, mudstones, siltstones with intercalated andesite flows |

### 13.4.2 Chemical Composition & Mineral Content

Detailed head assays, including detailed multi-element inductively coupled plasma (ICP) scans, were performed in duplicate for each of the variability composites. Gold content was highly variable and ranged between 3.8 g/t and 32.5 g/t, averaging 9.9 g/t. The highest gold was measured in the Mat lithology. Silver content in the samples ranged between 7 g/t and 192 g/t, averaging 42 g/t. The higher silver grade composites were in the Mat and Svc lithologies.

The variability composites were submitted for bulk mineral analysis using a "quantitatively evaluation of minerals by scanning electron microscopy" (QEMSCAN). This assessment provides an unsized mineral composition along with information regarding sulphur distribution in the samples. A summary of the results is shown in Figure 13-8.

**Figure 13-8: Mineral Content of Variability Composites**




Source: BaseMet (2020b).

The dominant non-sulphide minerals across the range of samples in order of abundance were quartz, K-feldspars, muscovite/illite and chlorite. A number of samples also contained a higher concentration of calcite. The dominant sulphide mineral across the range of samples tested was pyrite. On average, 96% of the sulphur in the samples was contained in pyrite, which accounted for only 1%, on average, of the sample mass.

### 13.4.3 Comminution Testing

Comminution tests were completed on 49 dilution, vein, and blended variability composites to determine sample abrasiveness and energy requirements for comminution circuit design. The BWi tests were conducted on select samples with a closing screen size of 75 µm. The average BWi was determined to be 20.8 kWh/t and the minimum and maximum values recorded were 16.4 and 25.6 kWh/t, respectively. There was no distinct variation in the BWi between the various lithological zones tested. There were 23 RWi determinations performed for these samples sets.

The RWi tests were performed at ALS Metallurgy. These values indicated that the ore was moderately hard, averaging 18 kWh/t, for rod mill grinding. The SMC A x b values were determined when sufficient sample mass was available. This parameter provides a measure of resistance to impact breakage in a SAG mill. A smaller number means that the sample had a higher level of resistance to impact breakage. The average A x b value, as derived from the SMC test, was 39.1 indicating these samples are classified as hard by the JKTech report. When sample mass was not sufficient for an SMC test, an HIT was conducted on the remaining samples. The HIT test is an abbreviated test using a small sample size, providing an estimate of the A x b value. The HIT A x b value of these samples averaged at 37.9, similar to the SMC derived A x b value.

The Bond abrasion indices for a number of samples was measured, ranging between 0.104 and 1.019, averaging at 0.413. These samples would be considered moderate to highly abrasive. The comminution results determined there was no specific trend developed relating a physical property or geological lithology of the sample to hardness. Furthermore, there was little variance in the comminution properties of the variability samples tested.

The comminution test results, together with historical comminution test results, were analyzed and modelled by Orway Mineral Consultants to determine comminution circuit options and to size the grinding mills. Orway's ore interpretation consists of statistical evaluation of bench-scale comminution testing ore parameters and is summarized in Table 13.10 by zone.

The coefficient of variation (standard deviation / average %) is an indication of testwork variability. For an FS, a typical 15% to 20% coefficient of variation is recommended and the impact breakage A x b, RWi, BWi and relative density ore characterization meet FS-level requirements based on the coefficient of variation.

In general, vein material is slightly more competent (lower A x b and higher BWi) compared to dilution material. South zone A x b is marginally lower (i.e., more competent than north zone), but in general they are similar to one another. Similar BWi hardness is observed for the north and south zones, particularly at the 85$^{th}$ percentile. A 15$^{th}$ percentile A x b and 85$^{th}$ percentile BWi was used for mill sizing.

**Table 13.10: Summary of Comminution Testwork Parameters Statistics by Zone**

| HIT Axb | North | South | Other | Overall |
|---|---|---|---|---|
| Count | 16 | 14 | - | 30 |
| Min | 51.3 | 44.0 | - | 51.3 |
| Max | 34.1 | 29.2 | - | 29.2 |
| **15th Percentile** | **35.2** | **32.0** | - | **33.4** |
| Average | 39.3 | 36.1 | - | 37.9 |
| Standard Deviation | 4.9 | 4.7 | - | 5.0 |
| Coef of Variation | 13% | 13% | - | 13% |

| SMC Axb | North | South | Other | Overall |
|---|---|---|---|---|
| Count | 7 | 9 | - | 16 |
| Min | 45.7 | 46.4 | - | 46.4 |
| Max | 32.8 | 30.0 | - | 30.0 |
| **15th Percentile** | **34.3** | **34.6** | - | **34.5** |
| Ave | 39.4 | 38.9 | - | 39.1 |
| Standard Deviation | 5.0 | 5.1 | - | 4.9 |
| Coef of Variation | 13% | 13% | - | 13% |

| Axb (HIT + SMC) | North | South | Other | Overall |
|---|---|---|---|---|
| Count | 19 | 15 | - | 34 |
| Min | 47.9 | 46.4 | - | 47.9 |
| Max | 32.8 | 29.2 | - | 29.2 |
| **15th Percentile** | **34.6** | **32.1** | - | **34.1** |
| Average | 38.6 | 37.5 | - | 38.1 |
| Standard Deviation | 4.2 | 5.3 | - | 4.7 |
| Coef of Variation | 11% | 14% | - | 12% |

| BWI | North | South | Other | Overall |
|---|---|---|---|---|
| Count | 23 | 19 | 6 | 48 |
| Min | 18.2 | 18.4 | 16.4 | 16.4 |
| Max | 25.3 | 25.6 | 22.2 | 25.6 |
| **85th Percentile** | **23.0** | **22.9** | **21.7** | **22.9** |
| Average | 20.7 | 21.3 | 19.4 | 20.8 |
| Standard Deviation | 1.9 | 1.7 | 2.3 | 2.0 |
| Coef of Variation | 9% | 8% | 12% | 9% |

| Relative Density | North | South | Other | Overall |
|---|---|---|---|---|
| Count | 8 | 10 | 1 | 19 |
| Min | 2.48 | 2.55 | - | 2.48 |
| Max | 2.60 | 2.62 | - | 2.65 |
| 85th Percentile | 2.60 | 2.61 | - | 2.60 |
| **Average** | **2.54** | **2.58** | **2.65** | **2.57** |
| Standard Deviation | 0.05 | 0.03 | - | 0.04 |
| Coef of Variation | 2% | 1% | - | 2% |

| AI | North | South | Other | Overall |
|---|---|---|---|---|
| Count | 4 | 5 | 6 | 15 |
| Min | 0.311 | 0.133 | 0.104 | 0.104 |
| Max | 1.019 | 0.835 | 0.546 | 1.019 |
| 85th Percentile | 0.770 | 0.671 | 0.383 | 0.560 |
| **Average** | **0.545** | **0.453** | **0.291** | **0.413** |
| Standard Deviation | 0.322 | 0.268 | 0.151 | 0.251 |
| Coef of Variation | 59% | 59% | 52% | 61% |

Source: Orway Mineral Consultants (2020b).

### 13.4.4 Gravity & Leaching Testwork

The metallurgical testwork program consisted of evaluating whole-of-ore leaching versus initial gravity separation for coarse gold recovery, followed by cyanide leaching of gravity tails. Two grind calibrations were conducted on each of the variability composites, targeting a primary grind size of $P_{80}$ of 53 µm. Each variability composite was tested using the basic flowsheet developed in the previous testwork programs, with and without gravity separation ahead of leaching.

Whole-of-ore leach tests were conducted at pH 10.5 and modified using lime, a two-hour pre-oxidation, 250 g/t of lead nitrate, and at a sodium cyanide concentration of 500 ppm. Samples were sparged with oxygen. Initially for these tests, a pulp density of 50% solids was targeted. Throughout testing, due to poor settling properties, pulp density was reduced to 40% to allow for sufficient settling and sampling. Sample measurements were taken at 2, 6, 24, 36 and 48 hours. Results at 36 and 48 hours are summarized in Table 13.11.

**Table 13.11: Average Metallurgical Response by Lithology (Whole-of-ore Leach)**

| Test | Gold Extraction % | | Silver Extraction % | | Consumption, kg/t | |
|---|---|---|---|---|---|---|
| | 36 h | 48 h | 36 h | 48 h | NaCN | Lime |
| Mat | 96 | 97 | 70 | 77 | 0.78 | 1.57 |
| Mbt | 95 | 97 | 78 | 83 | 0.68 | 1.55 |
| Mvc | 93 | 95 | 79 | 82 | 0.52 | 1.34 |
| Mls | 92 | 94 | 79 | 85 | 0.35 | 1.24 |
| Mss | 95 | 95 | 82 | 85 | 0.44 | 1.27 |
| Mco | 88 | 88 | 81 | 82 | 0.72 | 1.16 |
| Svc | 93 | 95 | 68 | 74 | 1.21 | 2.80 |

Source: BaseMet (2020b).

The variability composites responded exceptionally well to cyanide leaching, following a primary grind sizing of $P_{80}$ of 53 µm. Gold from the feed was, on average, 94.0% recovered after 36 hours of leaching. Only one sample had a much lower gold extraction of 78%; NP-4-MVO-16. The cause of the lower performance of this sample was not determined. Silver recovery for the samples ranged between 54% and 93% after 36 hours of leaching, averaging at 77%. Cyanide consumption for the samples averaged at 0.63 kg/t, while lime consumption, on average, was measured at 1.5 kg/t. No discernible correlation between the variability feed characteristics and gold extraction was noted, and no variation in gold performance between the various lithologies was observed.

The samples also responded well to gravity concentration followed by direct leaching of the gravity tailings. Results indicate the presence of moderate gravity recoverable gold. On average, 28.0% of the feed gold reported to the gravity concentrate. Overall total gold recovery was similar to cyanide leach alone, with an average final gold recovery of 95.0%. However, with the recovery of the heavier gold into the gravity concentrate, gold leach kinetics were superior with a gravity circuit implemented. These results indicate some degree of nugget gold present in the samples, which could benefit from gravity concentrate recovery.

Silver recovery on average was 10.3% recovered into the gravity concentrates, and on average, 68.5% was extracted from the leach process for an overall average silver recovery of 78.8%. Average cyanide consumption was 0.58 kg/t and average lime consumption was 1.3 kg/t. These values were only marginally lower than the reagent consumption measured for the whole-of-ore cyanide leach tests. Similar to the whole-of-ore leach testing, no discernible trend between gravity recovery and gold feed parameters was noted. Results at 36 and 48 hours are summarized in Table 13.12.

**Table 13.12: Average Metallurgical Response by Lithology (Gravity-Leach)**

| Test | Gold Extraction % | | | Silver Extraction % | | | Consumption, kg/t | |
|---|---|---|---|---|---|---|---|---|
| | Pan. Con. | 36 h | 48 h | Pan. Con. | 36 h | 48 h | NaCN | Lime |
| Mat | 26 | 95 | 97 | 8 | 74 | 82 | 0.90 | 1.60 |
| Mbt | 23 | 96 | 96 | 8 | 81 | 85 | 0.54 | 1.30 |
| Mvc | 33 | 96 | 95 | 16 | 83 | 86 | 0.47 | 1.10 |
| Mls | 22 | 93 | 94 | 7 | 76 | 79 | 0.31 | 1.27 |
| Mss | 37 | 96 | 96 | 3 | 62 | 64 | 0.45 | 1.25 |
| Mco | 36 | 91 | 91 | 14 | 80 | 83 | 0.62 | 1.17 |
| Svc | 26 | 95 | 96 | 15 | 79 | 83 | 0.87 | 2.25 |

Note: Pan. Con. – panned concentrates. Source: BaseMet (2020b).

#### 13.4.5 Settling Tests

Following cyanide leach testing, it was observed that the sample tailings had very poor settling properties. Approximately 10 g/t of the flocculant MagnaFloc 351 was added to the tailings to improve settling. Using flocculant improved settling velocity drastically for majority of the samples. It is apparent that certain lithologies possess considerably slower settling rates. The Mcv and Mvo samples had the slowest settling rates, even with the use of flocculant. The settling velocity appears to be the only variable that is distinctly different between the lithologies.

#### 13.4.6 Optimization Testing

Four main composites were constructed based on spatial lithology, metallurgical performance, and settling properties. Table 13.13 summarizes the chemical composition of the four composites.

**Table 13.13: Head Assays for Composites**

| Composite | Au (g/t) | Ag (g/t) | Cu (g/t) | S(t) (%) | C(t) (%) | Fe (%) |
|---|---|---|---|---|---|---|
| Composite 1 | 8.86 | 44 | 37.0 | 0.73 | 0.33 | 1.7 |
| Composite 2 | 10.20 | 27 | 36.0 | 0.58 | 0.82 | 2.1 |
| Composite 3 | 8.03 | 70 | 57.7 | 0.70 | 0.07 | 1.1 |
| Composite 4 | 7.97 | 32 | 44.1 | 0.40 | 3.43 | 0.9 |

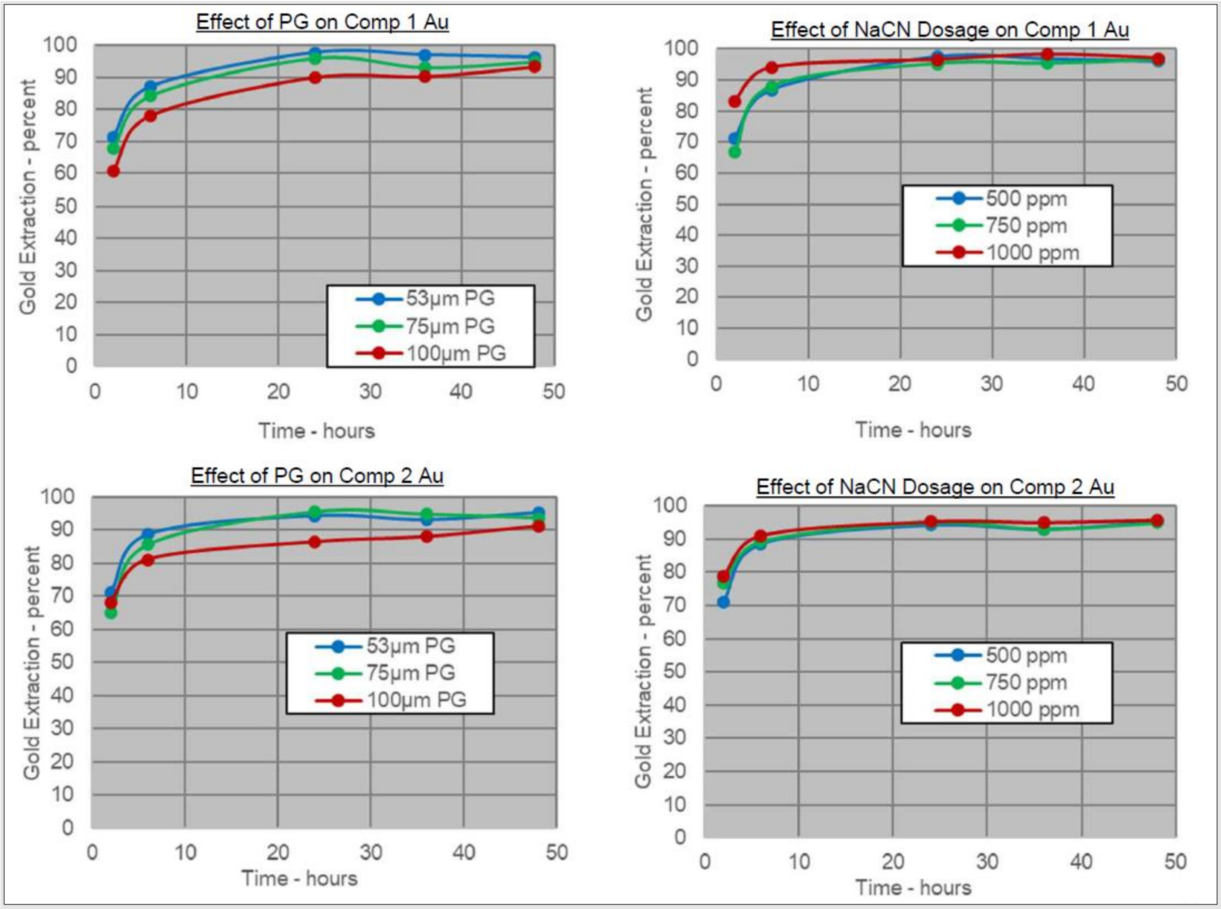
Source: BaseMet (2020b).

Optimization testing was conducted on Composite 1 and Composite 2, evaluating the effect of primary grind sizing and sodium cyanide concentration on gold and silver extraction. Results are summarized in Figure 13-9. Primary grind size was evaluated at $P_{80}$ of 53, 75 and 100 µm. For Composite 1, coarser primary grind sizing resulted in slower leach kinetics for gold. After 48 hours of leaching, final gold extraction was only slightly superior at the finest primary grind sizing. Gold extraction was between 93% and 96%.

Silver leach kinetics and overall silver extraction were inferior at the coarser primary grind size. Silver extraction was between 77% and 85%. Cyanide concentration testing demonstrated that higher initial gold leach kinetics were observed at 1,000 ppm cyanide; however, increasing cyanide concentration had no impact on overall gold extraction. Higher cyanide concentrations were beneficial for silver leach performance and overall recovery but resulted in higher cyanide consumptions. At a cyanide concentration of 500 ppm, cyanide consumption was measured at 0.6 kg/t. When cyanide dosage was increased to 750 and 1,000 ppm, cyanide consumption increased to 1.0 and 0.9 kg/t, respectively. Often the higher cyanide consumption levels will increase the cost of downstream cyanide destruction.

For Composite 2, gold leach kinetics declined at $P_{80}$ of 100 µm, but were not impacted at 75 µm. Overall gold extracted was between 91% and 95% after 48 hours of leaching, silver extracted was between 74% and 84%. Similar to Composite 1, higher cyanide concentration did not impact gold leach kinetics or overall gold extraction for Composite 2. However, silver leach rates and overall recovery were superior at higher cyanide consumptions. At a cyanide concentration of 500 ppm, cyanide consumption was measured at 0.6 kg/t. When cyanide dosage was increased to 750 and 1,000 ppm, cyanide consumption increased to 0.8 and 1.0 kg/t, respectively.

**Figure 13-9: Effect of Primary Grind & NaCN Dosage on Gold Extraction (Composites 1 & 2)**




Source: BaseMet (2020b).

A cyanide leach test was conducted on Composites 3 and 4, at a primary grind size of $P_{80}$ of 53 µm and cyanide dosage of 500 ppm. A comparison of all four composites at similar conditions demonstrated slower initial leach kinetics for Composite 3, but overall gold extraction for all four samples was similar ranging from 93.1% to 95.5%. In contrast, silver performance varied slightly for each composite ranging between 72.9% and 81.8%.

### 13.4.7 Oxygen Uptake Rate Determination

One kilogram each of Composite 1 and Composite 2 was utilized for oxygen uptake rate determinations. The results are summarized in Table 13.14.

**Table 13.14: Oxygen Uptake Rate Determination**

| Sample | Oxygen Consumption – mg/L/h @ Time (hours) | | | | | | | |
|---|---|---|---|---|---|---|---|---|
| | 0 | 1 | 2 | 3 | 4 | 5 | 6 | 24 |
| Test 74-Composite 1 | 0.00 | 0.18 | 0.07 | 0.04 | 0.03 | 0.03 | 0.02 | 0.19 |
| Test 75-Composite 2 | 0.00 | 0.09 | 0.04 | 0.03 | 0.02 | 0.03 | 0.02 | 0.04 |

Source: BaseMet (2020b).

#### 13.4.8 Carbon Adsorption Tests

Fleming kinetic constants were determined from sequential carbon triple contact testing, in which a known mass of fresh activated carbon is contacted with a known pulp volume, kinetic solution samples are removed over a two-hour period at which point the carbon is transferred to fresh leached pulp and contacted for four hours. A third and final carbon transfer to fresh pulp was contacted for an additional 16 hours (24 hours total) before the carbon was assayed. The Fleming constant 'k' was determined to be 45 and 43 $h^{-1}$ for Composites 1 and 2, respectively, while the constant "n" was measured at 0.68 for both.

#### 13.4.9 Cyanide Destruction Tests

The optimized leach process was performed on the four main composites with the cyanidation tailing slurry going directly to several small-scale continuous cyanide detoxification via the $SO_2$/air method. The conditions included 500 ppm cyanide in solution, a pH of 10.5 and a primary grind target of $P_{80}$ of 53 µm. Lead nitrate was added at 250 g/t. A two-hour pre-oxidation was performed with oxygen in advance of leaching.

For Composite 1, the target $CN_{WAD}$ value was obtained at an $SO_2$/g $CN_{WAD}$ ratio of 4 and a retention time of 61 minutes. For this composite, no copper addition was necessary during the final detoxification conditions; however, copper addition was required to start the process. For Composite 2, the target $CN_{WAD}$ value was obtained at an $SO_2$/g $CN_{WAD}$ ratio of 4 and concentration of 25 mg/L of copper. The test conducted without copper addition during the detoxification process was not effective. For Composite 3, the target $CN_{WAD}$ value was obtained at an $SO_2$/g $CN_{WAD}$ ratio of 5 and concentration of 50 mg/L of copper. For Composite 4, the target $CN_{WAD}$ value of 0.9 ppm was obtained. Further testing is required to lower than value to less than 0.5 ppm.

For all the variability composites tested, the target $CN_{WAD}$ value was obtained at an $SO_2$/g $CN_{WAD}$ ratio of 4 and concentration of 25 to 100 mg/L of copper.

### 13.5 BaseMet (2021) Scoping Metallurgical Testwork Program

The primary objective of the BaseMet (2021) scoping metallurgical testwork program was to assess the metallurgical response of samples from the Salinas zone of the proposed open pit. Three non-representative samples were provided to BaseMet for testing and the head assay results are shown in Table 13.15. SVC material was mineralized and was further tested.

**Table 13.15: Head Assays**

| Samples (Lithologies) | Cu (%) | Fe (%) | Ag (g/t) | Au (g/t) | S (%) | C (%) |
|---|---|---|---|---|---|---|
| TBX | 0.001 | 1.86 | 0.6 | 0.02 | 0.37 | 0.69 |
| SS | 0.000 | 0.25 | 0.3 | 0.05 | 0.01 | 0.05 |
| SVC | 0.000 | 0.73 | 2.1 | 0.45 | 0.18 | 0.05 |

Source: BaseMet (2021).

Cyanidation leach tests (bottle rolls) were completed on the SVC material using the flowsheet established during the BaseMet (2020b) metallurgical testwork program. The results are shown in Table 13.16 and summarized below:

- primary grind of $P_{80}$ of 53 and 75 µm
- two-hour pre-oxidation with oxygen
- 250 g/t lead nitrate addition
- 250 to 750 ppm NaCN concentration
- oxygen used in leaching (20 ppm dissolved oxygen level).

**Table 13.16: Metallurgical Response by Test Condition**

| Test | Grind Size $P_{80}$ µm | Calculated Head Assays (g/t) | | NaCN Concentration (ppm) | Carbon Contact | Gold Extraction % | | Silver Extraction % | | Consumption kg/t | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| | | Au | Ag | | | 24 h | 48 h | 24 h | 48 h | NaCN | Lime |
| CN-01 | 53 | 0.55 | 3.45 | 500 | - | 76.9 | 77.4 | 91.1 | 91.3 | 0.06 | 2.2 |
| CN-02 | 75 | 0.55 | 2.84 | 500 | - | 80.4 | 78.2 | 92.4 | 93.0 | 0.21 | 1.8 |
| CN-03 | 53 | 0.57 | 3.53 | 500 | 225g | 86.8 | 81.6 | 85.6 | 89.5 | 0.30 | 1.4 |
| CN-04 | 53 | 0.58 | 3.58 | 250 | - | 75.5 | 78.6 | 82.1 | 86.0 | 0.20 | 1.8 |
| CN-05 | 53 | 0.59 | 3.59 | 750 | - | 76.8 | 77.3 | 81.8 | 86.1 | 0.53 | 1.8 |

Source: BaseMet (2021).

Primary grind size was evaluated at $P_{80}$ of 53 and 75 µm. The coarser primary grind size resulted in slightly faster leach kinetics for gold and similar kinetics for silver. After 24 hours of leaching, gold and silver extraction was higher for the coarser primary grind size.

NaCN concentration ranges were evaluated. At $P_{80}$ of 53 µm similar gold extractions were achieved (~78% after 48 hours leaching) and the best silver extractions were achieved at 500 ppm (~91% after 48 hours leaching).

At a cyanide concentration of 500 ppm, cyanide consumption was measured at 0.3 kg/t. When cyanide dosage was increased to 750 ppm, cyanide consumption increased to 0.53 kg/t.

## 13.6 BaseMet (2021-2022) Feasibility Metallurgical Testwork Program

The primary objectives of the feasibility metallurgical testwork program were as follows:

- assess the ore properties (chemical and mineralogical) of a number of discrete variability samples, primarily from the Salinas zone
- assess the metallurgical performance of several discrete variability composites utilizing cyanide leaching to recover gold and silver
- test composites representing the initial mining years

- conduct specific tests to generate engineering data such as tailings settling, carbon loading, carbon adsorption and cyanide detoxification data
- generate tailings for environmental, geotechnical and vendor dewatering studies.

The test program commenced in May 2021 and was completed by December 2021. The test program was organized and completed over two main phases. An additional test program was initiated during Phase 1 (called Phase 1B) to investigate the poor metallurgical performance of some of the Phase 1 samples. The focus of the phases is as follows:

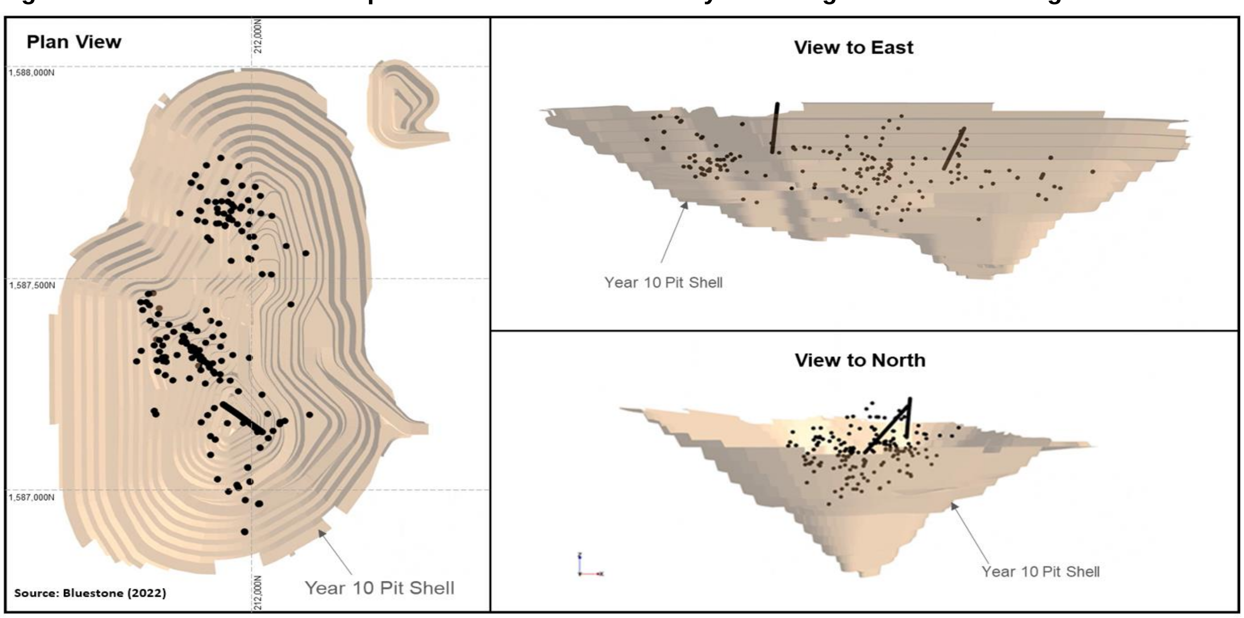
- Phase 1 – Variability Assessment
- Phase 1B – Investigation of Poor Metallurgical Performance of Phase 1 Samples
- Phase 2 – Mine Plan Composites and Engineering Data

The drill core and assay laboratory coarse rejects were first used to make variability composites. Using the already developed flowsheet the samples were tested and evaluated for metallurgical response. The variability data was compared to the feed sample characteristics to explain any anomalous metallurgical response. Once the analysis was complete, variability samples and other samples were then used to assemble larger composites for detailed testing to generate advanced engineering data.

### 13.6.1 Comminution Circuit Trade-off Study

A total of 994 kg of samples was collected for testing in this program. The samples were received as assay laboratory rejects, one-quarter and one-half drill core. The samples are represented in Figure 13-10.

**Figure 13-10: Location of Samples for 2021-2022 Feasibility Metallurgical Testwork Program**




Source: Bluestone (2022).

### 13.6.2 Phase 1 – Variability Assessment

#### 13.6.2.1 Chemical Composition

Sixty-one variability samples were used for Phase 1: 37 coarse rejects and 24 drill core intervals. Detailed head assays were performed for each of the variability composites. Table 13.17 shows the average element specific assays for these samples. A multi-element ICP scan was also conducted on these samples and is available in BaseMet (2022) report. Sample identification starting with REJ indicates samples were constructed from coarse rejects from south zone (SZ) and north zone (NZ). All other samples were constructed from drill core. As shown in Table 13.17, the average gold grade was 0.9 g/t Au with a minimum of 0.08 g/t Au to a maximum of 6.2 g/t. Silver content in the samples averaged 4.6 g/t. The copper and sulphur contents were, on average, 22 g/t and 0.55%, respectively.

**Table 13.17: Gold & Silver Assays of Phase 1 Samples**

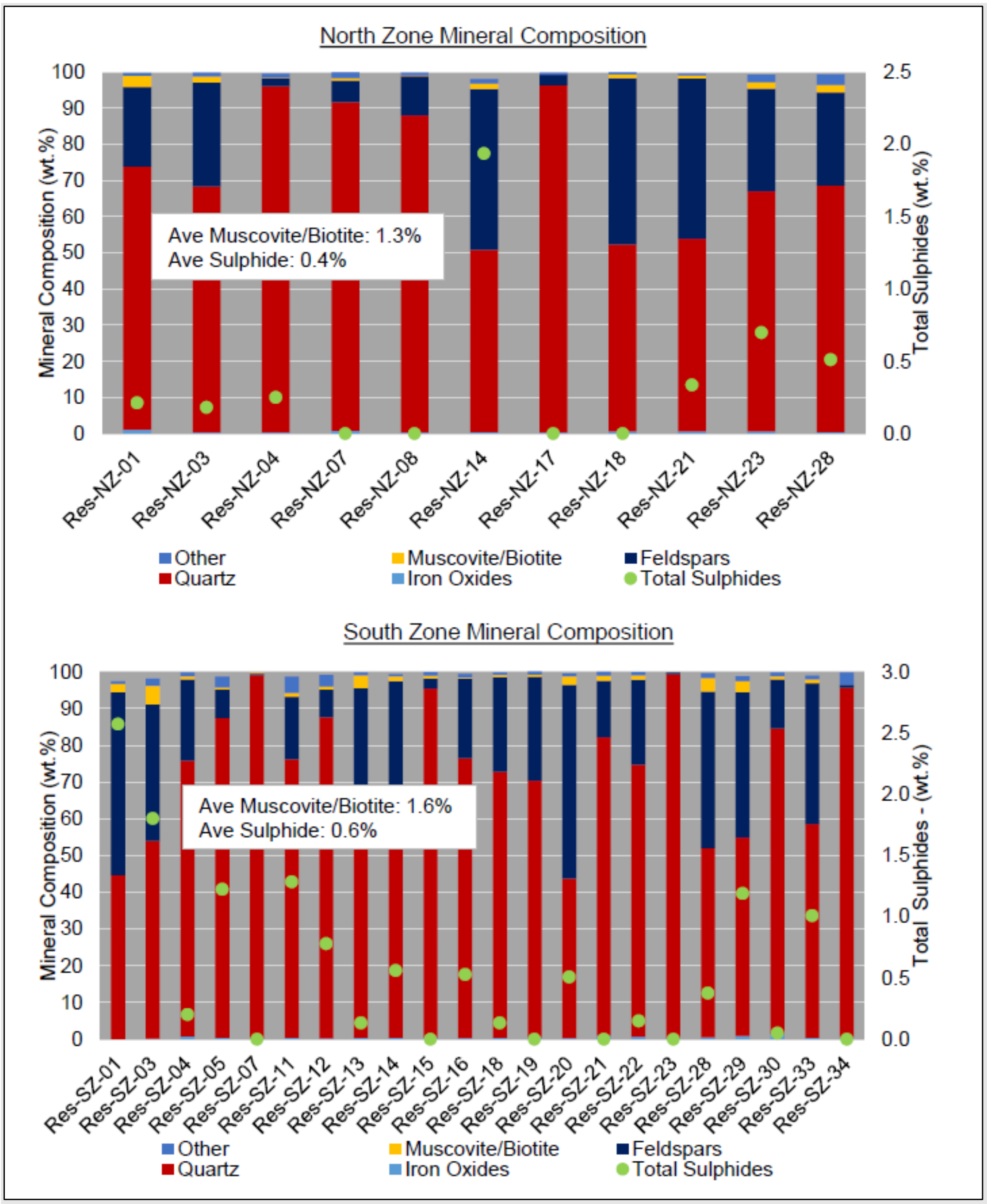
| Sample ID | Au (g/t) | Ag (g/t) | Cu (%) | S (%) | Sample ID | Au (g/t) | Ag (g/t) | Cu (%) | S (%) |
|---|---|---|---|---|---|---|---|---|---|
| REJ-SZ-1 | 0.55 | 1.9 | 7 | 1.4 | REJ-NZ-21 | 0.37 | 1.6 | 7 | 0.18 |
| REJ-SZ-3 | 0.56 | 1.4 | 20 | 0.91 | REJ-NZ-22 | 0.09 | 1.7 | 17 | 1.5 |
| REJ-SZ-4 | 0.73 | 1.5 | 10 | 0.23 | REJ-NZ-23 | 0.52 | 11 | 9 | 0.39 |
| REJ-SZ-5 | 0.65 | 2.4 | 12 | 0.70 | REJ-NZ-25 | 0.08 | 1.2 | 20 | 2.2 |
| REJ-SZ-7 | 0.62 | 0.6 | 5 | 0.09 | REJ-NZ-28 | 0.54 | 10 | 18 | 0.36 |
| REJ-SZ-11 | 0.79 | 2.2 | 15 | 1.08 | REJ-NZ-29 | 0.22 | 1.8 | 6 | 0.01 |
| REJ-SZ-12 | 0.68 | 5.8 | 14 | 0.76 | DD-SZ-38 | 1.1 | 2.4 | 40 | 0.13 |
| REJ-SZ-13 | 0.91 | 4.0 | 12 | 0.08 | DD-SZ-39 | 1.2 | 5.3 | 16 | 0.03 |
| REJ-SZ-14 | 0.28 | 1.3 | 12 | 0.50 | DD-SZ-40 | 1.3 | 6.1 | 12 | 0.04 |
| REJ-SZ-15 | 0.39 | 0.9 | 5 | 0.03 | DD-SZ-41 | 1.03 | 5.2 | 12 | 0.04 |
| REJ-SZ-16 | 0.30 | 0.8 | 7 | 0.53 | DD-SZ-42 | 1.02 | 7.9 | 4 | 0.28 |
| REJ-SZ-18 | 0.84 | 1.1 | 4 | 0.19 | DD-SZ-43 | 0.53 | 6.5 | 8 | 0.07 |
| REJ-SZ-19 | 0.53 | 0.2 | 3 | 0.003 | DD-SZ-44 | 1.8 | 9.0 | 28 | 0.09 |
| REJ-SZ-20 | 0.89 | 6.9 | 19 | 0.29 | DD-SZ-45 | 1.4 | 10 | 16 | 0.38 |
| REJ-SZ-21 | 0.47 | 2.0 | 5 | <0.01 | DD-SZ-46 | 0.55 | 5.3 | 20 | 0.25 |
| REJ-SZ-22 | 0.29 | 2.8 | 9 | 0.38 | DD-SZ-47 | 1.38 | 25 | 4 | 0.08 |
| REJ-SZ-23 | 0.54 | 0.7 | 4 | <0.01 | DD-SZ-48 | 0.64 | 6.1 | <4 | 0.07 |
| REJ-SZ-28 | 0.66 | 2.2 | 14 | 0.22 | 481260 | 1.0 | 1.4 | 34 | 0.71 |
| REJ-SZ-29 | 0.76 | 2.8 | 16 | 0.63 | 481261 | 1.1 | 3.0 | 55 | 1.2 |
| REJ-SZ-30 | 0.61 | 3.1 | 10 | 0.01 | 481266 | 1.5 | 2.8 | 141 | 1.7 |
| REJ-SZ-33 | 0.50 | 1.2 | 15 | 0.91 | 483213 | 1.6 | 1.9 | 56 | 0.03 |
| REJ-SZ-34 | 0.52 | 0.4 | 3 | 0.49 | 481132 | 1.7 | 2.2 | 39 | 1.1 |
| REJ-SZ-35 | 0.15 | 1.0 | 17 | 1.9 | 481262 | 2.2 | 3.2 | 34 | 1.6 |
| REJ-NZ-01 | 0.48 | 1.9 | 4 | 0.14 | 480816 | 1.9 | 1.8 | 29 | 0.65 |
| REJ-NZ-03 | 0.49 | 3.0 | 7 | 0.10 | 468854 | 0.86 | 8.4 | 35 | 0.95 |
| REJ-NZ-04 | 1.0 | 14 | 10 | 0.13 | 483214 | 3.1 | 38 | 54 | 0.16 |
| REJ-NZ-07 | 0.34 | 0.7 | 3 | <0.01 | 480815 | 1.0 | 2.4 | 30 | 0.14 |
| REJ-NZ-08 | 0.44 | 3.6 | 3 | <0.01 | 481131 | 1.9 | 2.5 | 31 | 1.6 |
| REJ-NZ-14 | 0.38 | 3.0 | 23 | 1.2 | 468851 | 0.34 | 4.7 | 27 | 0.89 |
| REJ-NZ-17 | 1.3 | 8.2 | 4 | <0.01 | 481267 | 6.2 | 7.5 | 220 | 1.2 |
| REJ-NZ-18 | 0.46 | 1.6 | 3 | 0.03 | - | - | - | - | - |

Source: BaseMet (2022).

#### 13.6.2.2 Mineralogy

Only the coarse rejects were submitted for bulk mineral analysis using QEMSCAN. The dominant non-sulphide minerals across the range of samples, in order of abundance, were quartz, K-feldspars, muscovite/illite, and chlorite. The dominant sulphide mineral across the range of samples tested was pyrite. On average, 96% of the sulphur in the samples was contained in pyrite, which accounted for only 1%, on average, of the sample mass. There was no apparent difference in mineral content between the samples in this program and previously studied samples. A summary of the results is shown in Figure 13-11.

**Figure 13-11: Mineral Composition of North & South Zone Phase 1 Samples**



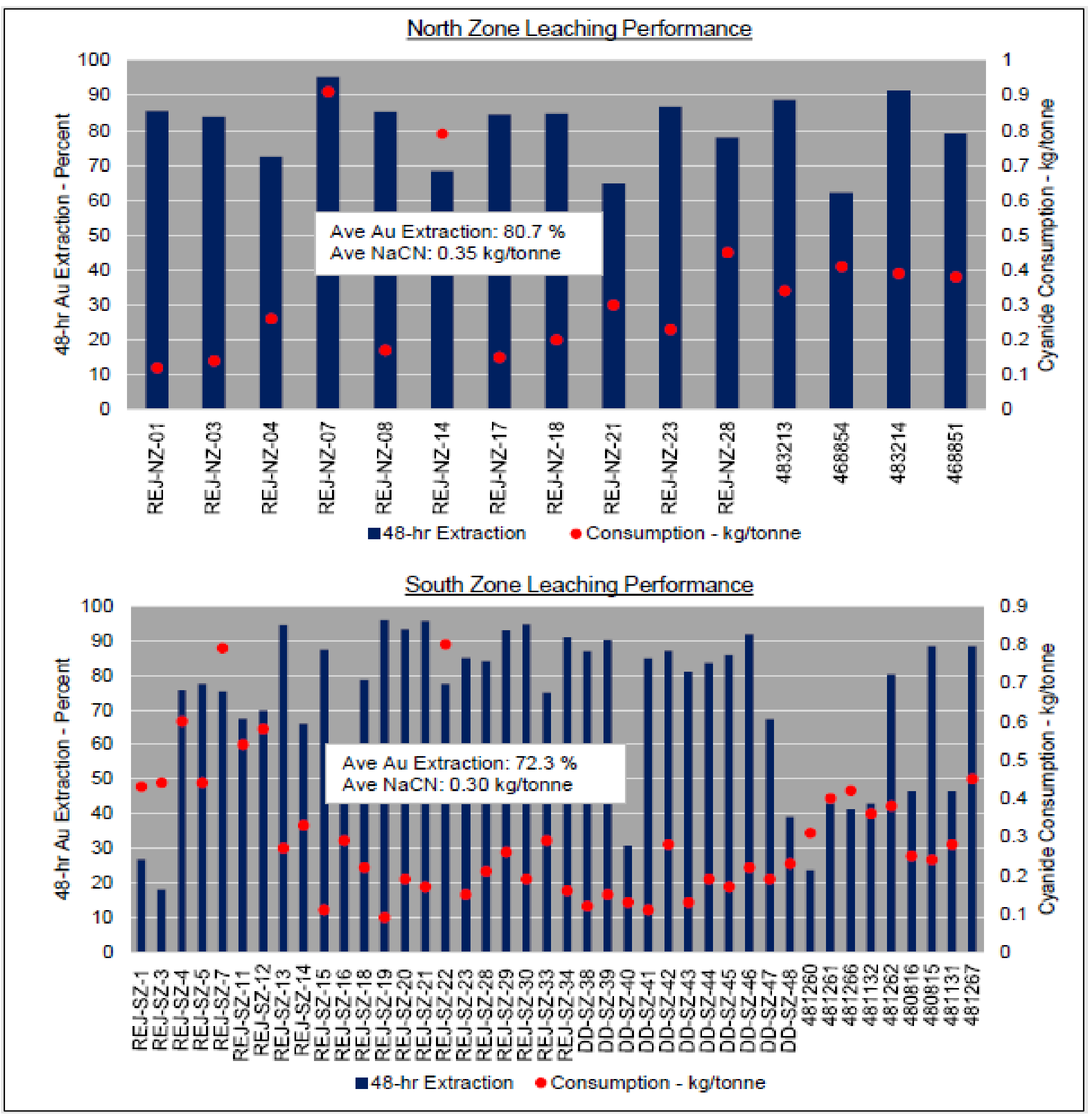

Source: BaseMet (2022).

#### 13.6.2.3 Metallurgical Testing

The metallurgical testwork program evaluated direct cyanide leaching. The previously developed flowsheet was used to evaluate the samples. Samples were ground to a primary grind size of $P_{80}$ of 53 µm. Prior to leaching, the ground sample was subjected to a two-hour pre-oxidation step with oxygen. The leach test was a bottle roll test with 2, 6, 24 and 48-hour sampling periods. Lime and cyanide levels were measured at each sampling period and were adjusted to maintain test parameters. The leach solution had 500 ppm NaCN and the pulp pH was maintained at 10.5 with lime. Lead nitrate was added at a dosage of 250 g/t.

A summary of the test results is provided in Figure 13-12. The top chart shows the samples from the north zone and the bottom chart are samples from the south zone. The blue bars show gold extraction after 48 hours of leaching. The red dots indicate cyanide consumption.

**Figure 13-12: Overall Variability Test Summary (North & South Zone Samples)**




Source: BaseMet (2022).

Unlike previous samples tested, there was considerable variability in metallurgical performance. Some samples achieved over 93% extraction which was consistent in previous metallurgical testwork programs. Other samples had significantly poorer extractions.

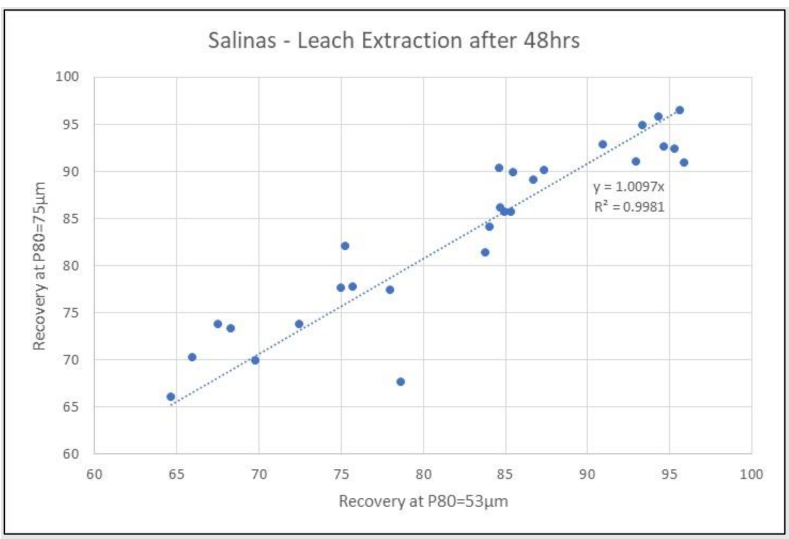
The cyanide consumption averaged 0.31 kg/t and was lower than the average consumption previously observed. This would be an expected result as the gold grades were considerably higher in previous testwork programs.

Lime consumption averaged 2.0 kg/t for this phase of the program, which represents an increase from previous testwork programs. However, it is noted that most of the samples used in Phase 1 were coarse rejects. These samples may need higher levels of lime because coarse reject samples are normally dried in an oven and finely crushed, a method that can increase sulphide oxidation. The coarse rejects samples had an average lime consumption of 2.3 kg/t compared to an average of 1.5 kg/t for the drill core samples.

#### 13.6.2.4 Effect of Primary Grind

A second set of leach tests were run on the coarse reject samples at $P_{80}$ of 75 µm. The drill core samples were not tested at a coarser grind size due to limited sample mass. The same bottle roll procedure was used for each sample. Results at 48 hours are compared for both grind sizes in Figure 13-13. The data showed no statistically significant difference in metallurgical performance between the 53 and 75 µm grind sizes. The coarser primary grind tests also lowered cyanide and lime consumption.

**Figure 13-13: Effect of Primary Grind**




Source: GMS (2022).

### 13.6.3 Phase 1B – Investigation of Poor Metallurgical Performance of Phase 1 Samples

After the results of the Phase 1 variability program were progressively reviewed, it was clear that the performance of some of the samples was much poorer than expected using the developed process. Numerous investigative tests were undertaken as described below.

#### 13.6.3.1 Process Parameter Changes

Three repeat tests were performed to determine if a simple adjustment of key process parameters could correct the performance. Three problematic samples were selected (DD-SZ-42, REJ-NZ-04 and REJ-SZ-01), and pre-oxidation time and lead nitrate were increased. Gold extractions were either similar or slightly worse than the baseline conditions (refer to Table 13.18 and Figure 13-14).

#### 13.6.3.2 Repeat Tests

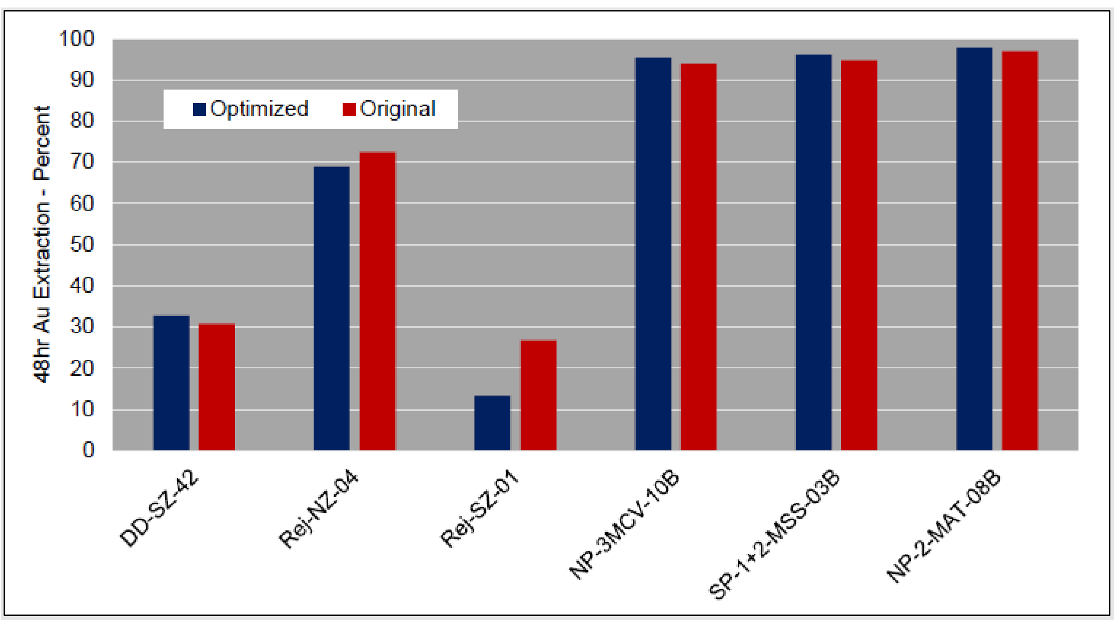
Three samples (NP-3-MCV-10B, SP-1+2-MSS-03B and NP-2-MAT-08B) from the previous program were selected for the baseline test, which was performed as a process verification check. The samples were tested using identical process conditions as the baseline. The metallurgical performance was reproduced on the samples as shown in Table 13.18 and Figure 13-14, indicating the change in performance could be attributed to the new samples.

**Table 13.18: Gold Extraction Summary for Repeat Tests**

| Test | Sample ID | Pre-Ox (h) | Lead Nitrate (g/t) | Consumption (kg/t) | | 48-h Extraction (%) | |
|---|---|---|---|---|---|---|---|
| | | | | Lime | NaCN | Au | Ag |
| BL0723-79 | DD-SZ-42 | 4 | 500 | 1.7 | 0.5 | 32.7 | 68.3 |
| BL0723-80 | REJ-NZ-04 | 4 | 500 | 1.7 | 0.5 | 69.0 | 82.8 |
| BL0723-81 | REJ-SZ-01 | 4 | 500 | 3.3 | 0.5 | 13.4 | 47.7 |
| BL0723-95 | NP-3-MCV-10B | 2 | 250 | 1.7 | 0.3 | 95.4 | 89.0 |
| BL0723-96 | SP-1+2-MSS-03B | 2 | 250 | 1.1 | 0.3 | 96.1 | 86.8 |
| BL0723-97 | NP-2-MAT-08B | 2 | 250 | 1.7 | 0.7 | 97.9 | 88.8 |

Source: BaseMet (2022).

**Figure 13-14: Summary of Repeat Tests**




Source: BaseMet (2022).

#### 13.6.3.3 Diagnostic Leach Tests

Some samples were subjected to diagnostic leach tests to help determine the cause of poor performance. The results are summarized in Table 13.19.

**Table 13.19: Diagnostic Leach Summary**

| Test | Sample ID | Au (g/t) | Cu (g/t) | S (%) | Au Extraction (%) | | |
|---|---|---|---|---|---|---|---|
| | | | | | Free Gold | Sulphide-Carbonates | Silicate Refractory |
| BL0723-02 | REJ-SZ-03 | 0.56 | 20.3 | 0.91 | 18.1 | 76.0 | 5.9 |
| BL0723-69 | DD-SZ-40 | 1.3 | 12 | 0.04 | 86.9 | 11.4 | 1.7 |
| BL0723-71 | DD-SZ-42 | 1.02 | 4 | 0.28 | 30.8 | 53.6 | 15.6 |
| BL0723-74 | DD-SZ-45 | 1.35 | 16 | 0.38 | 80.9 | 16.3 | 2.8 |

Source: BaseMet (2022).

These tests indicated that an aqua regia digestion on the cyanide leach tailings had high gold extractions. This digestion would extract refractory gold from carbonates and sulphide minerals. The mineral data on the variability samples indicated that samples contained sulphides (pyrite) but practically no carbonates. The data may indicate that a portion of gold in the samples is refractory with sulphides; however, no viable trend between gold extraction and sulphide content was observed in the samples.

#### 13.6.3.4 Phase 1B Composites & Other Metallurgical Tests

As a result of the variability sample performance, a more significant review of the process parameters was warranted. A limited number of poor performing variability samples were selected and were combined into composites. These samples, which were identified as Phase 1B Composites 1 and 2, contained 1.22 g/t Au and 0.55 g/t Au, respectively. Similarly, the sulphur grade was 1.35% and 0.66% for Composites 1 and 2, respectively.

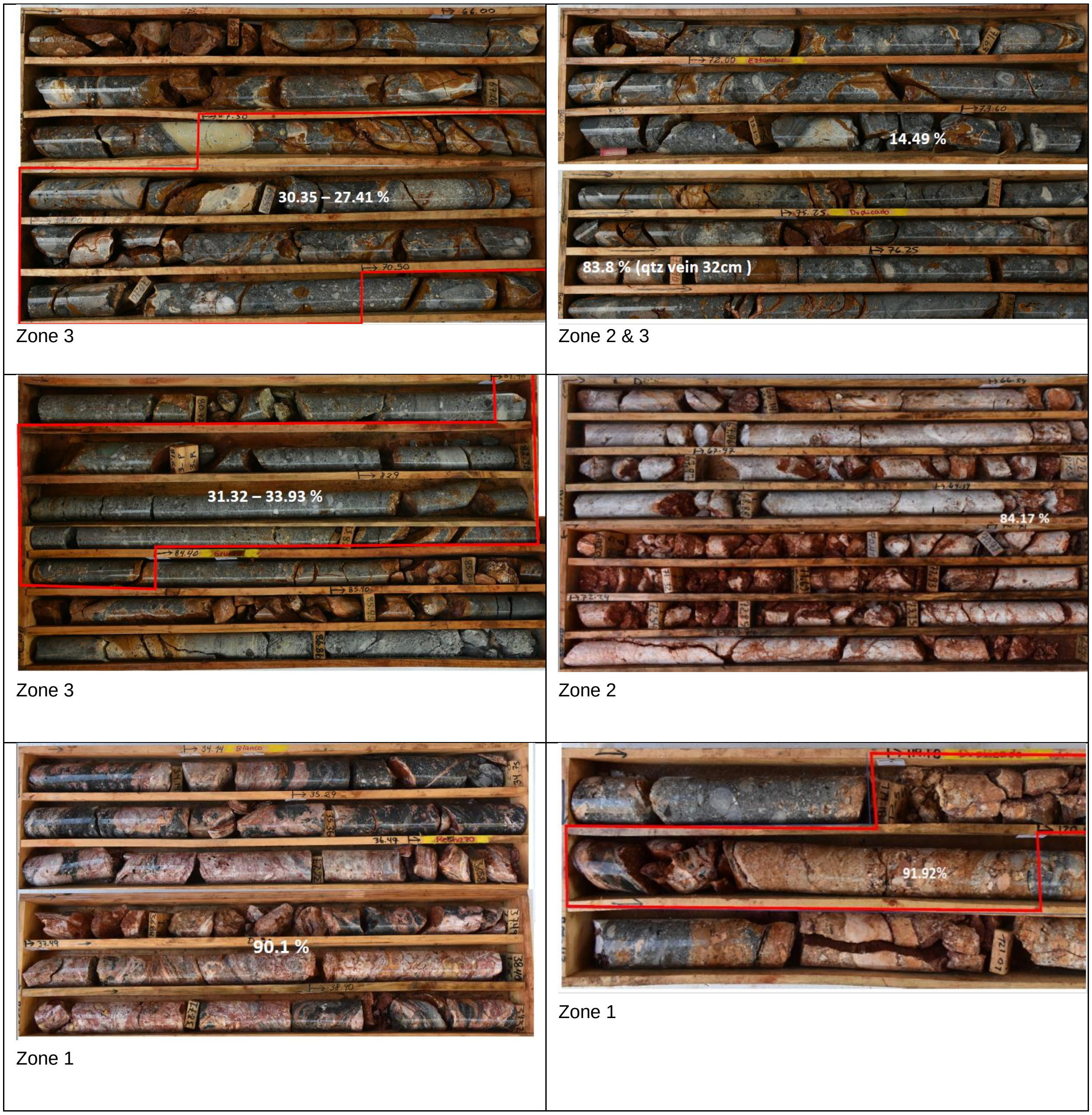
The parameters investigated included primary grind size, cyanide dosage, and pH. Alternative processes, including carbon in leach, gravity/leach, and flotation/leach hybrid circuits, were also examined to improve performance.

The Phase 1B Composite 1 achieved only 37% gold extraction using the baseline conditions. There was some evidence that grinding finer improved the performance, but the response was not proportional to the grind size. Gold extractions were still less than 50%. None of the other changes to parameters dramatically improved the performance over the baseline. The cause of poor gold extraction was not readily apparent from the tests performed. The performance of Phase 1B Composite 2 was better; gold extractions were 80.7% using the baseline conditions. Again, primary grind size data indicated a weak relationship between finer grinding and improved extractions. The other parameters had little effect on gold extraction rates. The best test was the hybrid flotation with regrind and leaching of the concentrate and tailings. This approach may have preferentially improved the performance by achieving much finer grind sizes of sulphides that may be containing finely disseminated gold.

#### 13.6.3.5 Salinas Zones

From the metallurgical investigation of the poor performing samples, it was evident that specific ore domains existed within the Salinas. As a result, three metallurgical domains were identified: Zone 1 (+85% recoveries), Zone 2 (60% to 85% recoveries) and Zone 3 (less than 60% recoveries). The three zones are visually different as shown in some core photos in Figure 13-15. Core with low recoveries is generally dark grey, strongly silicified and un-oxidized with fine grained disseminated pyrite (Zone 3). Zone 3 material represents approximately 1 Mt of predominately Scgl conglomerates at their deepest level in the South Zone and were consequently designated as waste in the block model.

**Figure 13-15: Select Core Photos of Salinas Zones**




Source: Bluestone (2021).

### 13.6.3.6 Gold Deportment Studies

A gold deportment study was completed at SSW on a sub sample from Phase 1B Composite 1 (feed sample that previously achieved 37% gold extraction) and a combined tailings sample from BL0723-98 (Test 98) and BL0723-110 (Test 110) which both originated from Phase 1B Composite 1.

The scope of the study was to establish the gold deportments for the two samples (feed and tailings samples) and involved a comprehensive full gold deportment study procedure which included characterization of all forms and carriers of gold in these samples. The procedure involved characterization of two major forms of gold in the samples: visible and sub-microscopic (refractory) gold. The evaluation techniques involved assays, QEMSCAN, high spatial resolution scanning electron microscopy with energy dispersive X-ray spectroscopy (SEM/EDX) scans for visible gold phases, identification and quantitative analysis of carriers of sub-microscopic gold among the relevant mineral phases present in the samples by Dynamic SIMS (D-SIMS).

The main observations from the deportment study on the feed sample were:

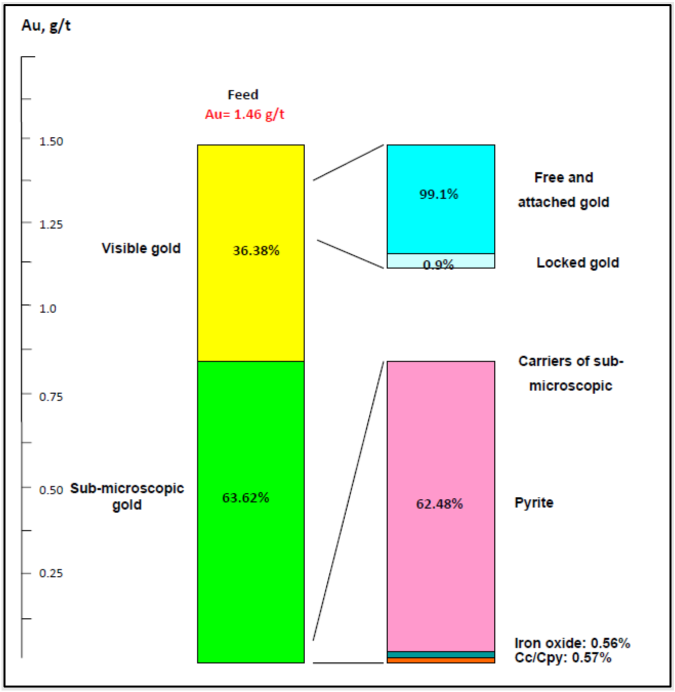
- Visible gold accounted for 36.38% of the assayed total gold in the sample.
  - Size distribution: Of the identified gold these grains, 54.8% were identified as being below 5 µm in length and 87% of the grains below 20 µm. No gold grains larger than 42 µm were found.
  - Liberation: 61.3% of the total identified gold grain population were free and fully liberated, and 22.6% of the population were partially liberated. For grains above 5 µm, 92.9% were fully liberated, with one locked grain in the 5 to 10 µm size class. Only 35% of the grains below 5 µm were free and fully liberated; the rest were partially liberated (or locked indicating a bias) of fully liberated grains towards the larger size fractions.
  - Surface area: Based on the surface area data, most of the visible gold in the feed sample was free and fully liberated, making up 98.6% of the total exposed area. Comparatively, 0.5% of the exposed surface area was partially liberated and 0.9% was fully locked.
  - Association: The locked and partially liberated grains were mainly associated with pyrite (75%). Iron oxide and sphalerite made up the remainder 25% mineral host association. No gold grains were found attached or locked in gangue mineral phases.
- The sub-microscopic (refractory) gold was the dominant form of gold in the sample and accounted for 63.62% of the assayed gold in the sample. Pyrite was the dominant carrier of sub-microscopic gold. The pyrite mineral phase in this sample carried 62.48% of the total assayed gold. The average gold concentration of sub-microscopic gold in pyrite showed strong correlation with the various morphological types and varied in the range of 6.2 ppm (for coarse pyrite) to 49/87 ppm (for microcrystalline/disseminated pyrite).

The main observations from the deportment study on the tailings sample were:

- Visible gold accounted for 1.2% of the assayed total gold in the sample.
  - Size distribution: Three gold grains were identified within the leach tails sample. Of these grains, 100% were identified as being below 5 µm in length.
  - Liberation: 100% of the total identified gold grain population were locked.
  - Surface area: The total surface area of the locked visible gold grains was very small (<10 µm2), which indicates small potential losses of residual, unleached visible gold in the leach tails sample.
  - Association: All identified locked gold grains were associated with the pyrite host mineral phase.
- The sub-microscopic (refractory) gold was the dominant form of gold in the sample and accounted for 98.81% of the assayed gold in the sample. Pyrite was the dominant carrier of sub-microscopic gold. The pyrite mineral phase in this sample carried 97.05% of the total assayed gold.

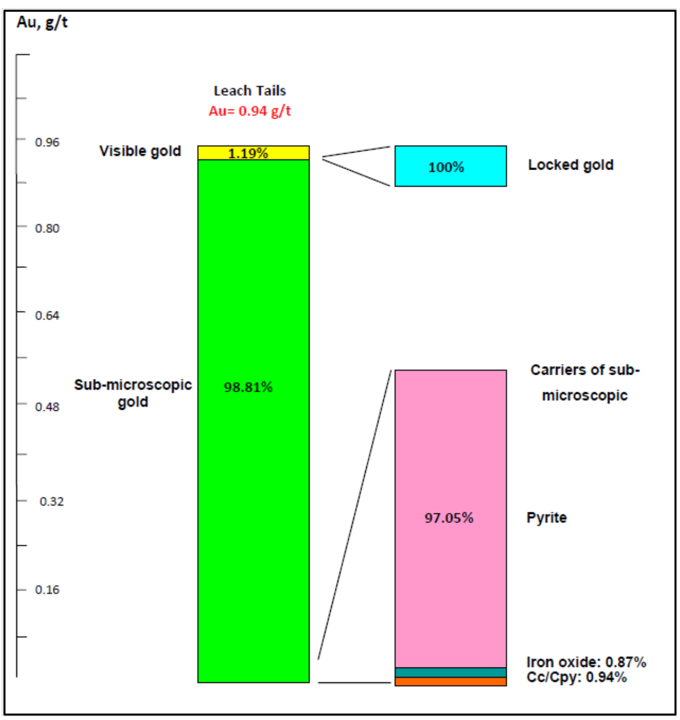
The gold deportment balances are summarized in Figure 13-16 and Figure 13-17.

**Figure 13-16: Gold Deportment Balance – Feed Sample**




Source: SSW (2021).

**Figure 13-17: Gold Deportment Balance – Tails Sample**




Source: SSW (2021).

### 13.6.4 Phase 2 – Mine Plan Composites & Engineering Data

#### 13.6.4.1 Composites

Two composite samples were created to represent two mining periods: Phase 2 Composite 1 represents ore from mining years 1 to 3, and Phase 2 Composite 2 represents ore from mining years 4 to 6. Samples included select Salinas samples and other lithology samples that were tested in previous metallurgical testwork plans. For the initial six years of mining, Salinas material will predominately originate from Salinas Zone 1. A summary of the two composites is provided in Table 13.20.

**Table 13.20: Phase 2 Composites**

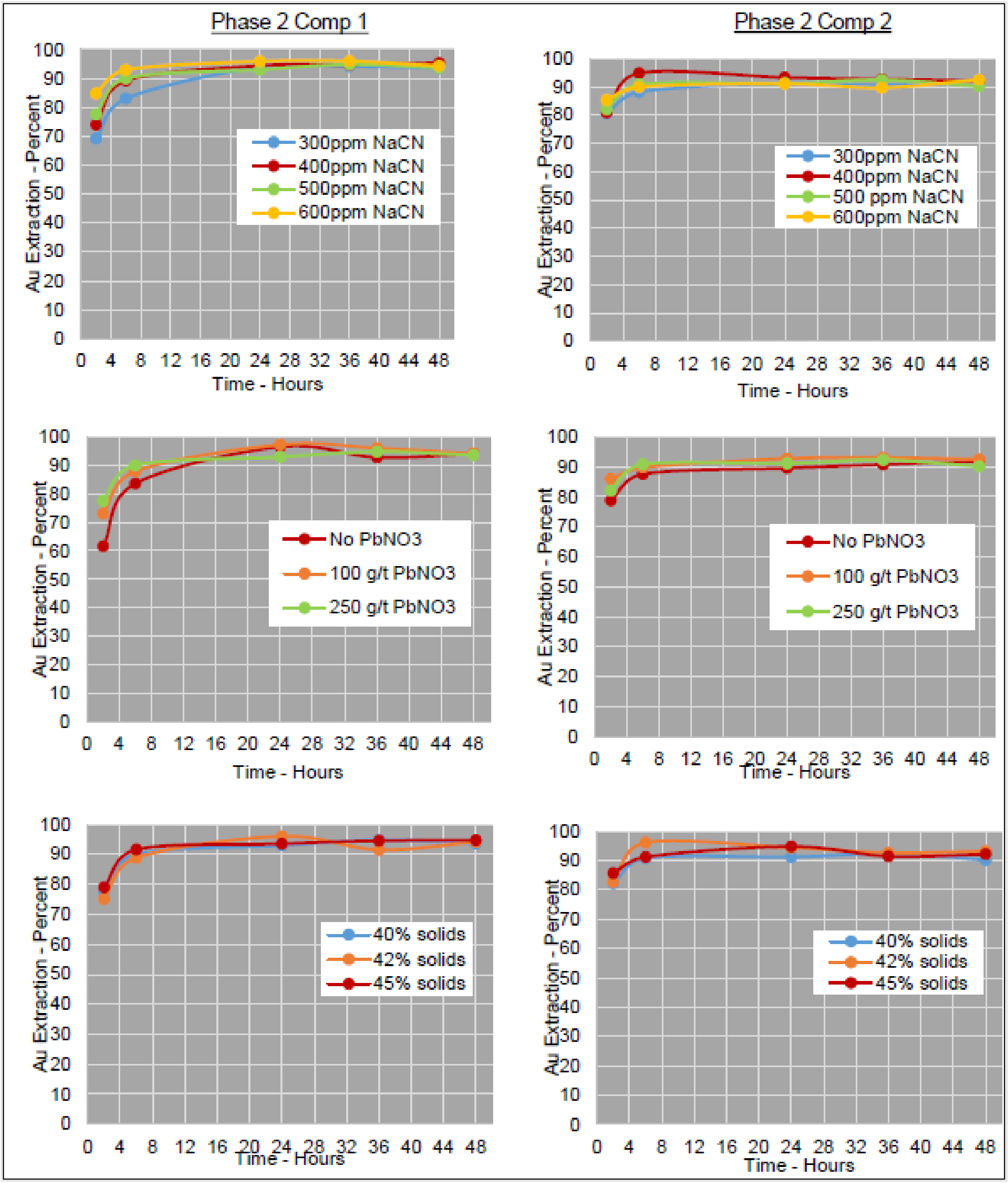
| Description | Mine Plan (Years 1 to 3) | | | Actual Composite | | |
|---|---|---|---|---|---|---|
| | Proportion | Au (g/t) | Ag (g/t) | Proportion | Au (g/t) | Ag (g/t) |
| Svc | 33% | 1.59 | 10.20 | 39% | 0.98 | 8.32 |
| Sinter | 4% | 1.03 | 6.02 | 8% | 0.54 | 1.10 |
| Scgl | 6% | 2.33 | 14.51 | 8% | 1.05 | 12.05 |
| Mbt | 31% | 3.00 | 16.10 | 29% | 2.98 | 15.01 |
| Mcv | 12% | 4.01 | 19.71 | 6% | 12.90 | 52.00 |
| Mvo | - | - | - | - | - | - |
| Mat | 8% | 2.45 | 14.00 | 7% | 5.06 | 35.00 |
| Mss | 2% | 2.01 | 10.80 | 0% | 0.00 | 0.00 |
| Mls | 5% | 4.00 | 20.02 | 3% | 8.34 | 36.00 |
| **Total/Average** | **100%** | **2.52** | **13.97** | **100%** | **2.77** | **15.37** |
| | **Mine Plan (Years 4 to 6)** | | | **Actual Composite** | | |
| | **Proportion** | **Au (g/t)** | **Ag (g/t)** | **Proportion** | **Au (g/t)** | **Ag (g/t)** |
| Svc | 40% | 1.23 | 7.68 | 38% | 1.28 | 9.62 |
| Sinter | 2% | 0.51 | 3.14 | 3% | 1.52 | 8.03 |
| Scgl | 14% | 1.21 | 6.46 | 15% | 1.72 | 7.97 |
| Mbt | 28% | 1.67 | 8.16 | 27% | 2.58 | 14.77 |
| Mcv | 4% | 1.47 | 4.91 | 6% | 1.87 | 3.30 |
| Mvo | 5% | 1.38 | 3.88 | 5% | 2.90 | 3.38 |
| Mat | 4% | 2.37 | 12.28 | 5% | 3.80 | 4.25 |
| Mss | 2% | 1.18 | 7.75 | 0%[1] | 0.00 | 0.00 |
| Mls | 1% | 1.43 | 6.69 | 2% | 2.63 | 1.40 |
| **Total/Average** | **100%** | **1.40** | **7.44** | **100%** | **1.96** | **9.66** |

Note: [1]Not enough MSS sample was available for Composite 2. Source: GMS (2021).

#### 13.6.4.2 Leach Extraction Tests

Optimization testing was conducted on Composites 1 and 2, evaluating the effect of primary grind sizing and sodium cyanide concentration on gold and silver extraction (Figure 13-18). The primary grind size was evaluated at $P_{80}$ of 53 µm and 75 µm. As noted with the variability testing, there was no statistically significant difference in gold leach performance between the grind sizes for both composites. Gold extraction was 94% and 92% for Composites 1 and 2, respectively.

**Figure 13-18: Phase 2 Testwork Summary**




Source: BaseMet (2022).

For Composite 1, cyanide concentration testing demonstrated increased initial gold leach kinetics with increase cyanide dosage. Composite 2, however, did not show a relationship between cyanide dosage and leach kinetic rate. It should be noted that the kinetic rates were exceptionally fast for both samples. Gold extractions were completed within 24 hours. Using lower dosages of cyanide in the plant can significantly decrease the cyanide detoxification operating costs. This may be a result of the lower gold content of these samples compared to previous testing.

The addition of lead nitrate ($PbNO_3$) showed a small improvement in initial leach kinetics as the dosage was increased from 0 g/t to 250 g/t.

Three different leach densities were tested and there was no perceivable difference in gold extraction performance.

The results from the Phase 2 test results reconfirmed the selected leach-CIP flowsheet and optimized the design criteria for the process plant design.

#### 13.6.4.3 Oxygen Uptake Rate Determination

One kilogram each of Composites 1 and 2 were utilized for oxygen uptake rate determinations in Tests 130 and 131. The test procedure described by G.M. Fraser was utilized. The samples were ground to a target grind size of $P_{80}$ of 53 µm and the freshly milled slurry sample was utilized to determine the oxygen demand for circuit design purposes.

A 1 kg charge from each sample was ground and transferred to an agitated vat, where pH of the sample was adjusted to 10.5 and 500 ppm of NaCN was added. Each test lasted 24 hours, with dissolved oxygen measurements being taken every minute for 15 minutes at time intervals of 0, 1, 2, 3, 4, 5, 6 and 24 hours.

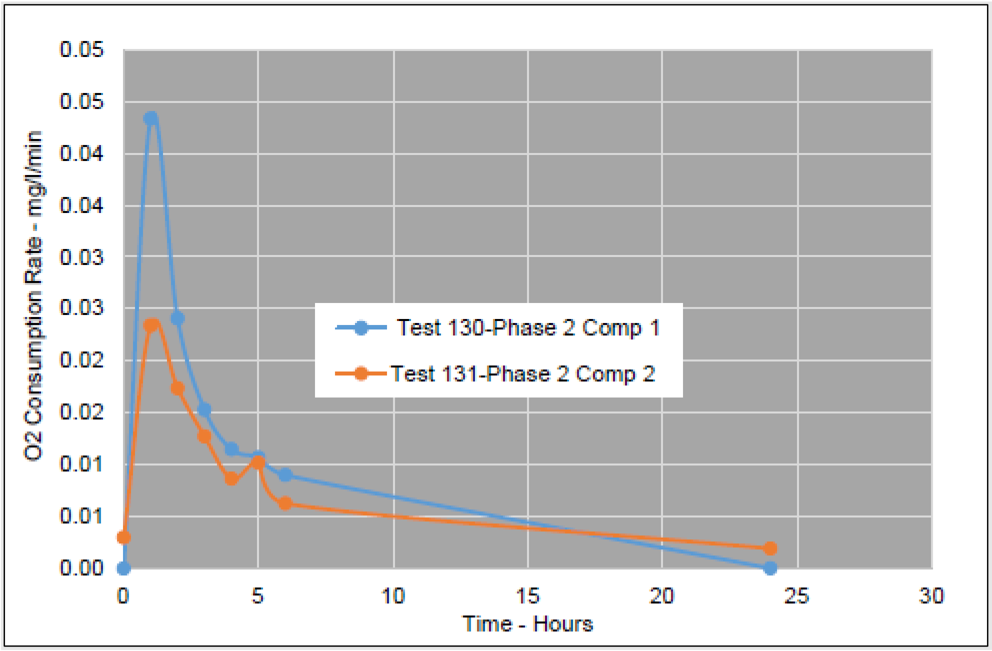
Oxygen consumption results are provided in Table 13.21 and Figure 13-19. The composite samples consumed low levels of oxygen; most of the oxygen demand would be met within the first two hours. It is noted that the oxygen consumption rates for this phase were lower than the previous phase (BaseMet 2020b).

**Table 13.21: Oxygen Uptake Rate Determination**

| Sample | Oxygen Consumption – mg/L/h @ Time (hours) | | | | | | | |
|---|---|---|---|---|---|---|---|---|
| | 0 | 1 | 2 | 3 | 4 | 5 | 6 | 24 |
| Test 130-Phase 2 Composite 1 | 0.00 | 0.04 | 0.02 | 0.02 | 0.01 | 0.01 | 0.01 | 0.00 |
| Test 131-Phase 2 Composite 2 | 0.00 | 0.02 | 0.02 | 0.01 | 0.01 | 0.01 | 0.01 | 0.00 |

Source: BaseMet (2022).

**Figure 13-19: Phase 2 – Oxygen Uptake Chart**




Source: BaseMet (2022).

#### 13.6.4.4 Carbon Adsorption Tests

Equilibrium carbon loading and sequential triple contact carbon in pulp tests were performed on each composite at two grind sizes. The carbon used for all tests was pre-attritioned carbon, manufactured by Calgon. Leach tests 125 to 128 were conducted to generate pregnant leach slurry and liquor for the carbon loading analysis.

The Fleming kinetic constants are determined from sequential carbon triple contact testing in which a known mass of fresh activated carbon is contacted with a known pulp volume, and kinetic solution samples are removed over a two-hour period at which point the carbon is transferred to fresh leached pulp and contacted for four hours. A third and final carbon is transferred to fresh pulp which is contacted for an additional 16 hours (24 hours total) before the carbon is assayed.

The Fleming constant ‘k’ and ‘n’ for the samples are summarized in Table 13.22.

**Table 13.22: Sequential Triple Contact Results**

| Test | Primary Grind ($P_{80}$ of µm) | Feed Solution (Au-ppm) | Fleming Kinetic Constants | |
|---|---|---|---|---|
| | | | k ($h^{-1}$) | n |
| Phase 2 Composite 1 | 53 | 1.8 | 29.2 | 1.5 |
| | 75 | 1.7 | 40.7 | 1.3 |
| Phase 2 Composite 2 | 53 | 1.2 | 31.3 | 1.1 |
| | 75 | 1.1 | 13.7 | 1.3 |

Source: BaseMet (2022).

#### 13.6.4.5 Cyanide Detoxification Testing

The optimized leach process was performed on the two main composites, with the cyanidation tailing slurry going directly to several small-scale cyanide detoxification via the $SO_2$/air method. The conditions included 500 ppm cyanide in solution, pH 10.5, and 100 g/t lead nitrate. Two primary grind sizes were investigated: $P_{80}$ of 53 and 75 µm. A two-hour pre-oxidation was performed with oxygen in advance of leaching.

Initially, cyanide detoxification testing was conducted on Phase 2 Composites 1 and 2. A wide range of parameters were evaluated for these detoxification tests. The $CN_{WAD}$ values to the target for this program was less than 10 ppm. The batch testing indicated that copper additions of between 50 and 75 mg/L were required to achieve the target $CN_{WAD}$ of less than 10. The continuous tests were run with 50 to 75 mg/L of Cu. The $SO_2$g/g $CN_{MP}$ ratio was used to achieve the target $CN_{WAD}$. The results of the continuous tests are summarized in Table 13.23.

**Table 13.23: Detoxification Continuous Test Summary**

| Sample | Test | Test Parameters | | | | | | Feed/Detox Solution Assays | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | pH | Retention Time (min) | Reagents Used | | Test Length | | $CN_{MP}$ (ppm) | Cu (ppm) | Fe (ppm) | Ni (ppm) | Zn (ppm) |
| | | | | $SO_2$ g/g | Cu mg/L | Min | No. of Disp | | | | | |
| Phase 2 Composite 1 | Feed | 9.8 | - | - | - | - | - | 81.7 | 12.1 | 120 | 0.27 | 0.76 |
| | 125-C7 | 8.6 | 60 | 3 | 50 | 240 | 4 | 0.5 | 0.01 | 5.3 | 0.05 | <0.01 |
| Phase 2 Composite 2 | Feed | 9.9 | - | - | - | - | - | 87.6 | 11 | 9.34 | 0.41 | 1.24 |
| | 127-C9 | 8.4 | 60 | 3 | 75 | 240 | 4 | 2.5 | 0.05 | 0.58 | <0.01 | 0.01 |

Source: BaseMet (2022).

## 13.7 Tailings Dewatering Vendor Testwork

Bulk tailings samples from Phase 2 Composites 1 and 2 at $P_{80}$ of 53 µm and 75 µm (four samples in total) from the BaseMet 2022 metallurgical testwork program were further tested by Metso:Outotec at SGS Lakefield. The testwork involved completing tailings thickening and filtration tests on the four samples.

The main conclusions from the thickener tests were as follows:

- All tailings samples were successfully thickened to high densities and achieved minimum thickener underflow densities of 50%. Most tests achieved densities in the upper 50s, which is desirable feed for filtration.
- The tailings thickener diameter of 32 m sized for the FS is adequate based on a 570 t/h design (496 t/h nominal) solid feed rate and target underflow density of 50%.
- The 53 µm sample showed marginally above +200 ppm thickener overflow clarity with effective rise rate level, considering <200 ppm as a preferred and acceptable range.
- Flocculant consumption and coagulant consumption rates at 50 g/t were higher than expected.

Key results of the filtration testwork results are summarized in Table 13.24.

**Table 13.24: Vendor Filtration Testwork Results**

| Sample Type | Primary Grind ($P_{80}$ of µm) | Filter Cake Thickness | Filter Cake Moisture |
|---|---|---|---|
| Phase 2 Composite 1 | 53 | 38-45mm | 16.5-16.7% |
| | 75 | 37-45mm | 15.6-16.1% |
| Phase 2 Composite 2 | 53 | 38-45mm | 16.5-18.4% |
| | 75 | 40-45mm | 17.7-18.5% |

Source: GMS (2022).

The main conclusions from the filtration tests are as follows:

- All tailings samples were successfully filtered.
- Composite 1 samples achieved the target 16% moisture.
- Composite 2 samples will require longer drying times to achieve 16% moisture.
- There was no major difference between filtering 53 µm and 75 µm material; however, there was a difference between Composite 1 and Composite 2, which may be related to mineralogy. This has not been quantified but may be related to an increase of Mvo lithology material in the mill feed.

## 13.8 Trade-off Studies

Several trade-off studies were completed to determine key process design criteria and select optimal unit operations. The main conclusions of this studies are summarized in the following subsections.

### 13.8.1 Comminution Circuit Trade-off Study

Four comminution circuits were evaluated, as follows:

1. primary crushing, SAG mill, ball mill and pebble crusher (SABC)
2. primary crushing, SAG mill and ball mill (SAB)
3. three-stage crushing and ball mill (3CBM)
4. three-stage crushing and two ball mills in series (3CBM2).

Based on comminution testwork results, each circuit was sized and the main mechanical equipment was determined. Differential economic analysis and a risk ranking was completed for all options using factored capital cost estimates and indicative operating cost estimates. Option #2 (SAB circuit) was selected.

### 13.8.2 Leach Circuit Trade-off Study

Three leach/carbon adsorption circuits were evaluated, as follows: 1. CIL; 2. leach / CIP; 3. leach / CIP carousel. Due to the high silver content in the ore, using a CIL or leach / CIP circuit configuration would result in higher capital costs due the extra mechanical equipment required to handle the higher silver carbon loading. For this Project, the leach / CIP carousel circuit has the lowest capital cost. Leach / CIP and leach

/ CIP carousel circuit configurations have lower operating costs in comparison to CIL circuits. When taking these benefits into account and considering the precious metals locked up inventory, option #3 (a leach / CIP carousel circuit) was selected, as it provided the best value for the Project.

### 13.8.3 Gravity vs. Whole-of-Ore Leach Trade-off Study

Using the gravity and whole-of-ore leach metallurgical testwork results, a trade-off study was completed to assess the benefit of a gravity circuit. The results of the leach tests showed there was no clear benefit to including a gravity circuit in the overall process design. The average gravity / leach test gold extraction was 94.9% compared to 94.4% gold extraction for the whole-of-ore leach tests. The test results showed a consistent variation in the gold extraction of 94% to 96% and cyanide consumption of 0.6 kg/t to 0.8 kg/t regardless of gravity / leach or whole-of-ore leach. An installed single-train gravity circuit will only recover a small fraction of the average 29% GRG recovered in the lab-scale gravity / leach tests. The operating cost of the gravity circuit is higher, despite the higher 20% cyanide consumption associated with whole-of-ore leaching. As such, a gravity circuit was not included in the flowsheet.

### 13.8.4 Flotation Circuit Trade-off Study

To evaluate improving the overall gold recovery of Salinas Zone 1 and 2 ore samples, the inclusion of a flotation circuit prior to leaching was considered and a trade-off study was completed. The flotation flowsheet considered included a rougher flotation circuit and concentrate regrind circuit to liberate suspected locked-up gold. The rougher concentrate and flotation tailings would then be combined and leached in the existing leach circuit.

Two rougher bench scale tests were performed on two subcomposites using Phase 1B samples (i.e., Composite 1 with samples that achieved very poor leach extraction results of ~35%, and Composite 2 with samples that achieved moderate leach extraction results of ~80%). The resulting concentrates per test were reground to $P_{80}$ of 15 to 20 µm and then leached in cyanide. The flotation tailings were separately leached. The results from the composite 2 tests were used to evaluate the inclusion of a flotation circuit and are summarized in Table 13.25.

**Table 13.25: Flotation Results (Phase 1B Composite 2)**

| Test | Feed g/t Au | Tails g/t Au | Conc. g/t Au | Mass Recovery % | % Recovery |
|---|---|---|---|---|---|
| CN-111 (Whole-of-Ore Leach) | 0.80 | 0.16 | - | - | 80.0% |
| R-115 (Flotation) | 1.92 | 0.45 | 12.28 | 12.5 | 79.7% |
| CN-115B (Flotation Tails Leach) | 0.45 | 0.12 | - | - | 74.3% |
| CN-115C (Flotation Concentrate Leach) | 12.28 | 0.55 | - | - | 95.5% |
| Overall Flotation/Leach Recovery | - | - | - | - | 91.3% (~10% increase compared to CN-111) |

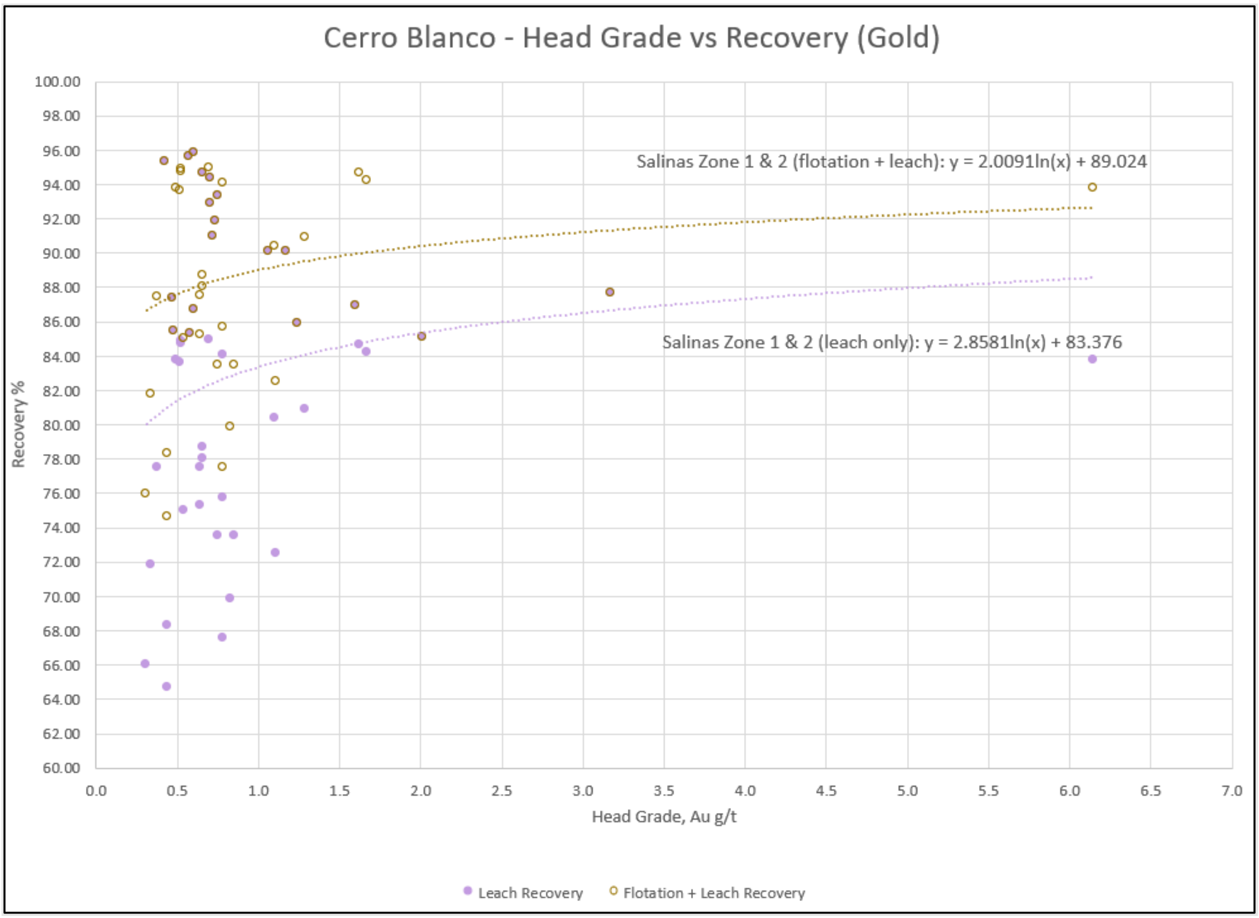
Source: BaseMet & G-Mining (2022).

To develop a grade-recovery curve for the flotation circuit, 10% was added to previous bottle roll tests that resulted in 60% to 85% recoveries (Zone 2) and curve fit equations established. The grade-recovery curves used for the trade-off study are provided in Figure 13-20.

An order-of-magnitude capital cost estimate of $23.8 million was developed based on factoring the expected mechanical supply costs of the flotation-regrind circuit. A preliminary operating cost of $1.32/t was determined based on the expected reagent consumption, regrind media and consumables consumption, power and maintenance requirements. It was assumed no additional operators were required.

A preliminary cash flow model was developed using the Salinas component of mine plan and then applying separately the gold recovery for leaching whole of ore and leaching flotation concentrate and tailings. With an approximate 10% increase in gold recovery for the Salinas ore and the increased capital and operating costs for a flotation circuit, the financial analysis showed that it was not feasible to install a flotation circuit.

**Figure 13-20: Recovery Curves Used for the Flotation Trade-off Study**




Source: GMS (2021).

### 13.8.5 Mercury Retort Trade-off Study

During the variability testwork program at BaseMet (2020b), mercury concentrations were measured on 29 composite head samples and four loaded carbon samples. Typically, the mercury concentration was less than 1 ppm in both the head and loaded carbon samples and is the minimum detection level.

Based on these results, a mercury retort is not required in the gold room, but a layout provision was made to include the retort in the future, if necessary. As a safety precaution, a portable Jerome meter will be used by operators in the gold room to detect any measurable mercury emissions.

### 13.8.6 Primary Grind Size Review

Following the completion of the FS metallurgical testwork program (BaseMet 2022), the primary grind size was reviewed against gold extraction results and tailings dewatering vendor tests.

The leach testwork results showed little to no loss in gold extraction loss at a coarser grind size of $P_{80}$ of 75 µm. Dewatering vendor tests on bulk tailings samples showed no significant difference between dewatering 53 µm and 75 µm ore, but filter cake moisture may be ~2% higher in later years of the mine life when the blend changes. The filter cake will require longer drying times to achieve target moisture levels.

The design criteria of $P_{80}$ of 53 µm for SAG and ball mill sizing was maintained to allow flexibility for operations, but the process plant can be operated to produce $P_{80}$ of 75 µm.

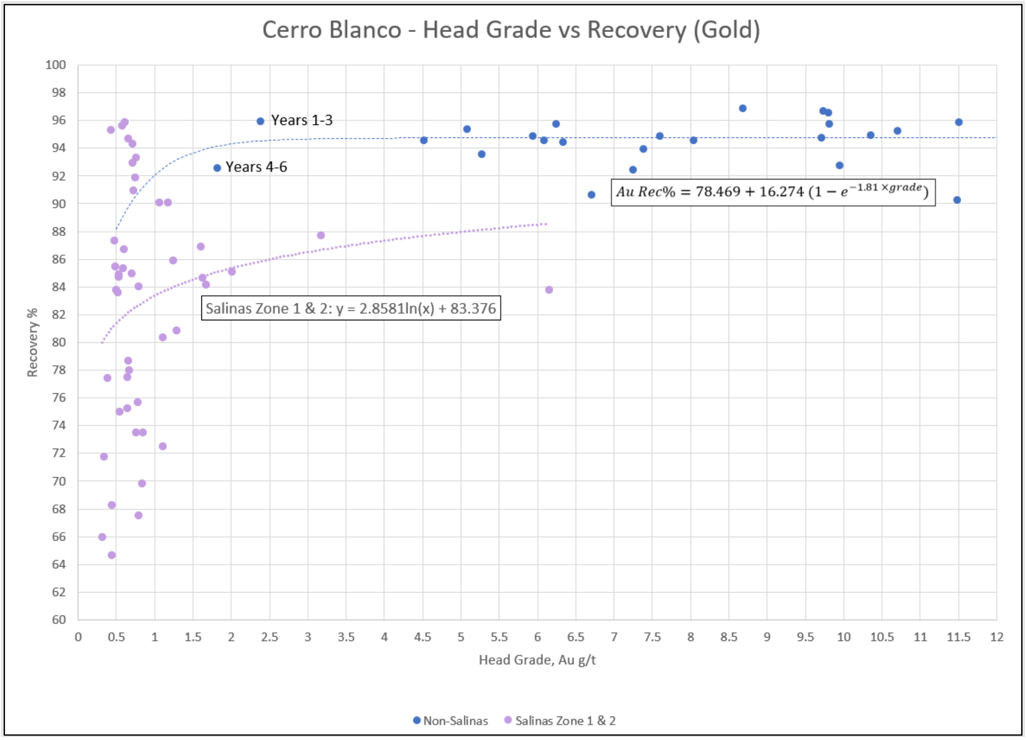
## 13.9 Gold & Silver Recoveries

Gold and silver recoveries for the Project were evaluated based on the whole-of-ore leach test results from the BaseMet test programs (2020b, 2021, 2022). In general, the test results showed no difference in gold recoveries between the north and south zones. Gold recovery is more of a function of lithology and grades.

The bulk of the metallurgical tests were completed on samples with a gold grade greater than 4.5 g/t Au and a silver grade greater than 4.4 g/t Ag. For the FS metallurgical testwork program, lower-grade Salinas ore and two composites representing the initial mining years were used together with the high-grade samples to derive recovery curves. Recognizing recoveries are lower for lower-grade ore, a first order reaction formula was used to develop the equation using all high-grade results, results from phase 2 (BaseMet 2022), Salinas samples less than 1 g/t Au, and Salinas samples that produced greater than 80% recovery. This equation was used to estimate recovery for the first six years of mining. A separate recovery equation developed for Salinas Zones 1 and 2 was used together with the main equation for subsequent mining years. The gold grade-recovery curve is provided in Figure 13-21.

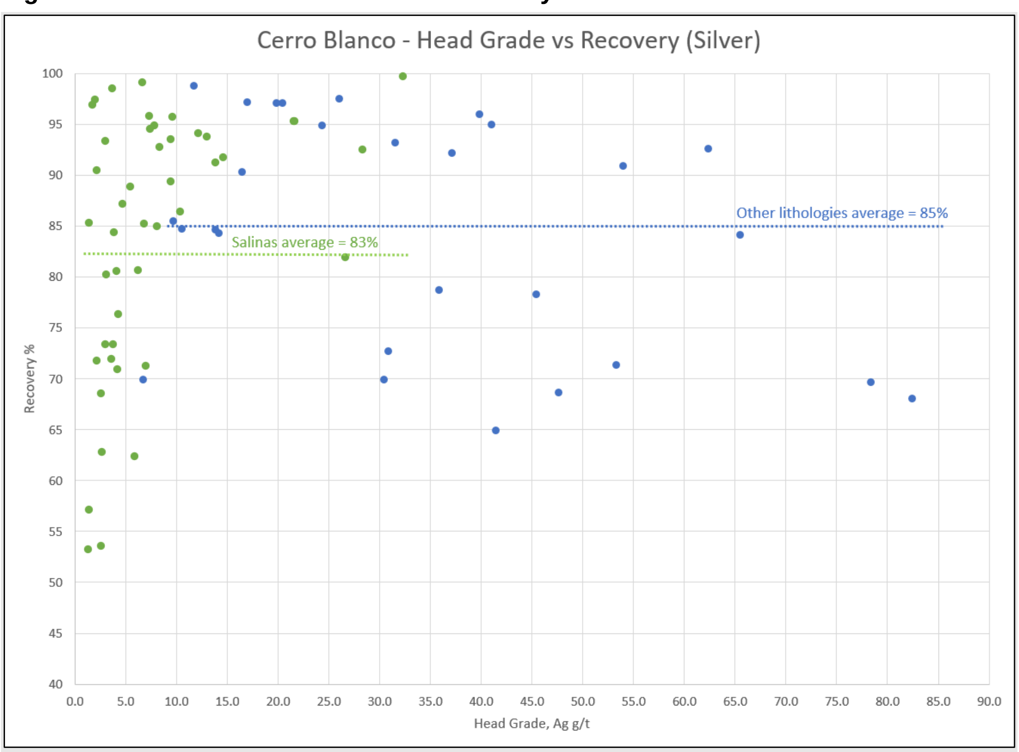
There is no recovery relationship with head grade and lithologies for silver. An average of the test results was used for silver recovery (Figure 13-22).

**Figure 13-21: Head Grade vs. Gold Recovery**




Source: GMS (2022).

**Figure 13-22: Head Grade vs. Silver Recovery**




Source: GMS (2022).

## 13.10 Process Design Criteria

Key process design criteria based on metallurgical testwork are summarized in Table 13.26.

**Table 13.26: Key Process Design Criteria**

| Condition | Unit | Value |
|---|---|---|
| Primary $P_{80}$ Grind Size | µm | 53 (design), 75 (operating) |
| BWi (85th percentile) – North Zone | kWh/t | 23.0 |
| BWi (85th percentile) – South Zone | kWh/t | 22.9 |
| SMC A x b (15th percentile) – North Zone | - | 34.3 |
| SMC A x b (15th percentile) – South Zone | - | 34.6 |
| Abrasion Index – North Zone | - | 0.55 |
| Abrasion Index – South Zone | - | 0.45 |
| Gravity Concentration Included | Y/N | No |
| Operating pH | - | 10.5 |
| Slurry Density | % solids w/w | 42 |
| Lead Nitrate Addition | g/t | 250 |
| Sodium Cyanide Concentration | ppm | 500 |
| Pre-Oxidation Time | hours | 2 |
| Leach Time | hours | 36 |
| Sodium Cyanide Consumption | kg/t | 0.3 |
| Lime Consumption | kg/t | 1.5 |

Source: BaseMet (2022) & GMS (2022).

# 14. MINERAL RESOURCE ESTIMATE

## 14.1 Introduction

This section describes the work undertaken by Kirkham Geosystems Ltd (KGL), including key assumptions and parameters used to prepare the mineral resource models for the Cerro Blanco deposit, together with appropriate commentary regarding the merits and possible limitations of such assumptions.

Cerro Blanco is a classic hot-springs-related, low-sulphidation epithermal gold-silver deposit comprising both high-grade vein and low-grade disseminated mineralization. Most of the high-grade mineralization is hosted in the Mita unit as two upward-flaring vein swarms (north and south zones) that converge downwards and merge into basal feeder veins where drilling has demonstrated widths of high-grade mineralization (e.g., 15.5 m 21.4 g Au/t and 52 g Ag/t). Bonanza gold grades are associated with ginguru banding and carbonate replacement textures. Sulphide contents are low, typically less than 3% by volume.

The Mita rocks are overlain by the Salinas unit, a sub-horizontal sequence of volcanogenic sediments and sinter horizons approximately 100 m thick that form the low-lying hill at the Project. Low-grade disseminated and veinlet mineralization within and as halos around the high-grade vein swarms is well documented in drilling since discovery of the deposit, with grades typically ranging from 0.3 to 1.5 g Au/t. The overlying Salinas cap rocks are also host to low-grade mineralization associated with silicified conglomerates and rhyolite intrusion breccias.

In profile, the inverted wedge-shape of the high-grade veins (upward flaring arrays) and their low-grade halos overlain by the mineralized Salinas cap rocks to surface, render the deposit amenable to exploitation by surface methods with a low strip ratio.

The mineral resource has a footprint of 800 x 400 m between elevations of 525 and 200 m above sea level (masl). The mineral resource estimate is the result of 141,969 m of drilling by Bluestone and previous operators (1,256 drill holes and channel samples by Bluestone). The 3.4 km of underground infrastructure allowed for underground mapping, sampling, and over 30,000 m of underground drilling that enhanced the current understanding and validation of the Cerro Blanco geological model. The mineral resource estimate is based on a scenario that considers open pit mining methods and therefore requires improved and refined geological models of the lithologic units. These broad mineralized lithologies are host to the high-grade veins that have been the focus of the potential underground mining scenario. The resulting domain models and estimation strategy were designed to accurately represent the grade distribution.

There is more than 3.09 Moz of gold and 13.4 Moz silver contained in the measured and indicated mineral resources, along with of 0.031Moz of gold and 0.112 Moz silver contained in the inferred mineral resources. The Cerro Blanco Mineral Resource Estimate is reported at a base case above a 0.4 g Au/t cut-off, as tabulated in Table 14.1.

**Table 14.1: Mineral Resource Statement**

| Resource Category | Tonnes (kt) | Au Grade (g/t) | Ag Grade (g/t) | Contained Gold (koz) | Contained Silver (koz) |
|---|---|---|---|---|---|
| Measured | 40,947 | 1.8 | 7.9 | 2,382 | 10,387 |
| Indicated | 22,595 | 1.0 | 4.2 | 706 | 3,058 |
| Measured & indicated | **63,542** | **1.5** | **6.6** | **3,089** | **13,445** |
| Inferred | 1,672 | 0.6 | 2.1 | 31 | 112 |

Notes: The mineral resource statement is subject to the following: (1) Mineral resource statement prepared by Garth Kirkham of KGL in accordance with N.I. 43-101. (2) Effective date: June 20, 2021. All mineral resources have been estimated in accordance with Canadian Institute of Mining and Metallurgy and Petroleum (CIM) definitions, as required under N.I. 43-101. (3) Mineral resources reported demonstrate reasonable prospect of eventual economic extraction, as required under N.I. 43-101. Mineral resources are not mineral reserves and do not have demonstrated economic viability. The mineral resources may be materially affected by environmental, permitting, legal, marketing, and other relevant issues. (4) Cut-off grades are based on a price of US$1,600/oz gold, US$20/oz silver and a number of operating cost and recovery assumptions, including a reasonable contingency factor. (5) An inferred mineral resource has a lower level of confidence than that applying to an indicated mineral resource and must not be converted to a mineral reserve. It is reasonably expected that most of the inferred mineral resources could be upgraded to indicated mineral resources with continued exploration. 6) Mineral resources are inclusive of mineral reserves. Source: Kirkham (2021).

Several resource estimates have been published by Bluestone since 2017 in four technical reports, as follows:

- Preliminary Economic Assessment (March 20, 2017)
- Preliminary Economic Assessment Update (June 2, 2017)
- Feasibility Study (January 29, 2019)
- Preliminary Economic Assessment Update (February 28, 2021).

The first three reports and resource estimates considered an underground mining scenario. The last resource estimate (Preliminary Economic Assessment dated February 28, 2021) was for the open pit scenario. All estimates were authored by Garth Kirkham, P. Geo; QP.

All four technical reports were filed on the System for Electronic Document Analysis and Retrieval (SEDAR).

## 14.2 Data

The drill hole database was supplied in electronic format (i.e., Microsoft Excel and Access) by Bluestone. This included collars, down hole surveys, lithology data and assay data (i.e., grams per tonne of gold and silver, and down hole “from” and “to” intervals in metric units). Lithology group and description information was provided, along with abbreviated alpha-numeric and numeric codes (Table 14.2).

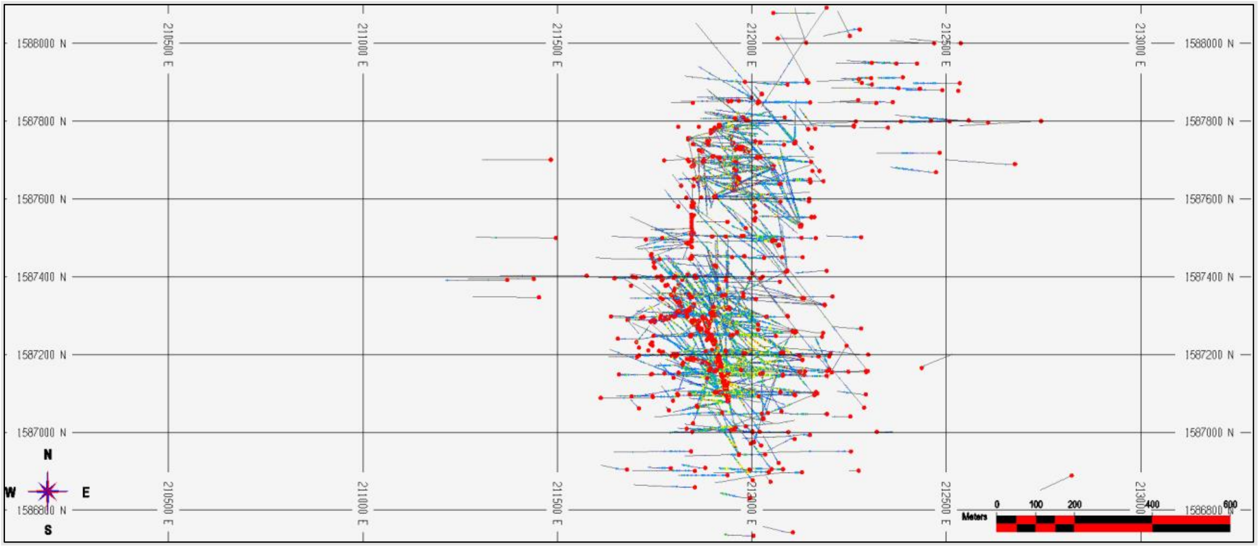
Figure 14-1 shows the plan view of drill holes with collars. A total of 107,452 assay values and 55,285 lithology values were supplied for the Project. Validation and verification checks were performed during import to confirm there were no overlapping intervals, typographic errors, or anomalous entries.

**Table 14.2: Lithology Units & Codes**

| Lithology | Code | Code B | Lithology Group | Lithology Description |
|---|---|---|---|---|
| Qc | 10 | 1 | Post-Mineral Cover Rock - Quaternary | Colluvium |
| Qb | 11 | 1.1 | | Basalt Flows |
| Bi | 20 | 2 | Cross-Cutting Rock Types | Basaltic Intrusive Dikes |
| Cbx | 30 | 3 | | Collapse Breccia |
| Dp | 180 | 18 | | Dacite |
| Gr | 40 | 4 | | Granite |
| Ad | 50 | 5 | | Andesite Dike |
| Rp | 60 | 6 | | Quartz Eye Rhyolite |
| Vt | 70 | 7 | | Vein |
| Stock | 71 | 7.1 | | Stockwork |
| Hbx | 72 | 7.2 | | Hydrothermal Breccia |
| RF | 80 | 8 | | Rhyolite Flow |
| SZ | 81 | 8.1 | | Shear Zone |
| Ss | 90 | 9 | Salinas Group | Sinter |
| Svc | 91 | 9.1 | | Volcanic Sediments |
| Srt | 92 | 9.2 | | Quartz Eye Rhyolite |
| Sfx | 93 | 9.3 | | Phreatic Breccia |
| Slt | 94 | 9.4 | | Siltstone |
| Sct | 95 | 9.5 | | Ash Tuff |
| Scgl | 96 | 9.6 | | Conglomerate |
| Mss | 100 | 10 | Mita Group | Sandstone |
| Mat | 101 | 10.1 | | Andesite Tuff |
| Mlt | 102 | 10.2 | | Crystal Tuff |
| Mbt | 103 | 10.3 | | Lapilli Tuff |
| Msc | 104 | 10.4 | | Calcareous Limestone |
| Mls | 105 | 10.5 | | Limestone |
| Mcv | 106 | 10.6 | | Quartz Latite Crystal Lithic Tuff |
| Mvo | 107 | 10.7 | | Conglomerate |
| Mlm | 190 | 19 | | Upper Limestone |
| Silt | 108 | 10.8 | | Siltstone - mudstone |
| PA | 130 | 13 | | Porphyritic andesite |
| Tcb | 110 | 11 | Tempisque Volcanic Complex | Basalt-dominated |
| Tca | 111 | 11.1 | | Andesite-dominated |

Source: Kirkham (2021).

**Figure 14-1: Plan View of Drill Holes**




Source: Kirkham (2021).

## 14.3 Data Analysis

Table 14.3 shows statistics of gold and silver assays for each of the lithologic units listed in Table 14.2. It should be noted that the total number of values from section to section vary depending on the parameter being analyzed and the value for reporting these varied data sub-sets is to detect and investigate issues or anomalies. Included for the statistical analysis, there are 130,307 gold assays (153,078 m) total, which average 0.68 g/t, and there are 130,238 (153,003 m) silver assays by lithology logged, which average 3.75 g/t. The maximum gold assay is 1,380 g/t, while the maximum silver assay is 8,656.7 g/t. It is important to note is that 73 gold assays are greater than 100 g/t and 54 silver assays are greater than 500 g/t which may be a reflection of the non-nuggety nature of the mineralization present at Cerro Blanco. Note that the method utilized for the treatment of outliers is by cutting the composites as opposed to cutting assay values, and the following table is for illustrative purposes.

**Table 14.3: Statistics for Weighted Gold & Silver Assays**

| Code | Metal | Valid | Length (m) | Maximum (g/t) | Mean (g/t) | CV |
|---|---|---|---|---|---|---|
| Total Cut | AU | 130,307 | 153,077.8 | 1,380.0 | 0.68 | 9.9 |
| | AG | 130,238 | 153,003.0 | 8,656.7 | 3.75 | 11.1 |
| Total Uncut | AU | 131,215 | 154,481.6 | 1,380.0 | 0.69 | 9.8 |
| | AG | 131,146 | 154,406.9 | 8,656.7 | 3.78 | 11.0 |

Source: Kirkham (2021).

Table 14.4 shows intervals that intersect the high grade are primarily encountered within the Vt unit, as would be expected. The Vt unit which represents the majority of the very high-grade populations, has 7,554 gold (3,716.8 m) and 7,553 (3,716.7 m) silver assay intersections, resulting in an average grade of

9.94 g Au/t and 38.92 g Ag/t. The coefficient of variation is relatively high with 3.3 for gold and 4.0 for silver. These are reviewed once compositing and cutting is applied which will reduce the CV to reasonable values. Also, of particular interest within the Cross-cutting group are the Stock, which shows 2,899 values (3,714 m) with 1.64 g/t Au and 8.11 g/t Ag and HBX shows 1592 values (1,067 m) with 1.08 g/t Au and 6.94 g/t Ag, respectively. The grades within the Stock and Hbx intervals display very high variability due to a small number of very high-grade outliers. These values are fairly widely distributed within the Salinas and Mita units which may positively skew the grades within the low-grade envelopes. However, as they are disseminated and to be treated within the domains, they will be cut appropriately to ensure that they reasonably represent the estimated grades.

**Table 14.4: Statistics for Weighted Gold & Silver Assays for Quaternary & Cross-Cutting Rock Types**

| Code | Lithology | Code | Metal | Valid | Length (m) | Maximum (g/t) | Mean (g/t) | CV |
|---|---|---|---|---|---|---|---|---|
| 10 | Qc | 10 | Au | 787 | 1,271.00 | 5.10 | 0.05 | 2.9 |
| | | | Ag | 786 | 1,270.60 | 35.00 | 0.97 | 2.1 |
| 11 | Qb | 11 | Au | 144 | 214.70 | 0.06 | 0.01 | 0.4 |
| | | | Ag | 144 | 214.70 | 1.00 | 0.83 | 0.4 |
| 30 | Cbx | 30 | Au | 4,016 | 4,466.60 | 1,380.00 | 0.78 | 14.3 |
| | | | Ag | 4,016 | 4,466.60 | 2,194.00 | 3.86 | 5.4 |
| 40 | Gr | 40 | Au | 419 | 685.10 | 0.25 | 0.01 | 1.5 |
| | | | Ag | 419 | 685.10 | 2.30 | 0.81 | 0.5 |
| 50 | Ad | 50 | Au | 1,780 | 2,268.60 | 313.97 | 0.47 | 13.3 |
| | | | Ag | 1,780 | 2,268.60 | 801.20 | 2.73 | 7.8 |
| 60 | Rp | 60 | Au | 2,899 | 3,714.10 | 46.30 | 0.22 | 2.9 |
| | | | Ag | 2,899 | 3,714.10 | 241.00 | 2.12 | 2.9 |
| 70 | Vt | 70 | Au | 7,554 | 3,716.80 | 1,380.00 | 9.94 | 3.3 |
| | | | Ag | 7,553 | 3,716.70 | 4,677.80 | 38.9 | 4.0 |
| 71 | Stock | 71 | Au | 2,383 | 2,214.90 | 148.75 | 1.64 | 3.7 |
| | | | Ag | 2,383 | 2,214.90 | 409.00 | 8.11 | 2.6 |
| 72 | Hbx | 72 | Au | 1,592 | 1,067.40 | 266.09 | 1.08 | 7.9 |
| | | | Ag | 1,591 | 1,067.30 | 969.00 | 6.94 | 4.7 |
| 80 | RF | 80 | Au | 5,494 | 6,923.00 | 150.70 | 0.28 | 9.2 |
| | | | Ag | 5,489 | 6,919.00 | 8,656.70 | 5.11 | 26.6 |
| 81 | SZ | 81 | Au | 36 | 31.50 | 8.40 | 0.27 | 3.0 |
| | | | Ag | 36 | 31.50 | 55.70 | 2.49 | 2.2 |

Source: Kirkham (2021).

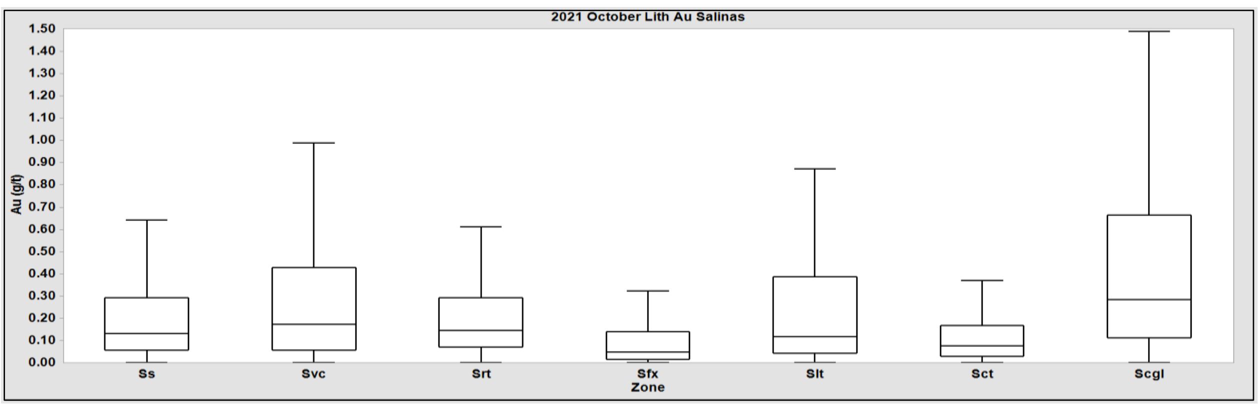
**Table 14.5** lists the statistics for the Salinas Group rocks units with the predominant unit being the Volcanic Sediments (Svc) showing mean gold and silver grades of 0.48 g/t and 3.41 g/t, respectively, with relatively high variability (CV) of 4.0 and 3.9. It is apparent from logging and modelling of the Salinas that the Sinter (Ss) and the Basal Conglomerate (Scgl) illustrate consistency and continuity. In addition, the Sinter has relatively lower grades with a mean of 0.27 g/t gold, while the Basal Conglomerate results show higher grades with a mean of 0.71 g/t gold, as illustrated in Figure 14-2. Therefore, observations and statistical analysis supports the resultant domaining for the Salinas of the Sinter, Basal Conglomerate and the remaining sedimentary units with the Volcanic Sediments (Svc) the predominant rock type.

**Table 14.5: Statistics for Weighted Gold & Silver Assays for the Salinas Group Rocks**

| Code | Lithology | Metal | Valid | Length (m) | Maximum (g/t) | Mean (g/t) | CV |
|---|---|---|---|---|---|---|---|
| 90 | Ss | Au | 4,200 | 6,269.7 | 15.67 | 0.27 | 2.1 |
| | | Ag | 4,198 | 6,269.2 | 187.80 | 1.52 | 2.7 |
| 91 | Svc | Au | 19,081 | 24,245.9 | 131.60 | 0.48 | 4 |
| | | Ag | 19,032 | 24,189.7 | 1,346.90 | 3.41 | 3.9 |
| 92 | Srt | Au | 1,215 | 1,522.1 | 16.47 | 0.27 | 2.3 |
| | | Ag | 1,215 | 1,522.1 | 88.00 | 2.38 | 2.1 |
| 93 | Sfx | Au | 1,495 | 2,334.3 | 194.70 | 0.34 | 10.8 |
| | | Ag | 1,495 | 2,334.3 | 267.40 | 2.59 | 4.4 |
| 94 | Slt | Au | 273 | 399.3 | 9.06 | 0.4 | 2.2 |
| | | Ag | 273 | 399.3 | 74.00 | 1.47 | 4 |
| 95 | Sct | Au | 242 | 347.7 | 3.57 | 0.19 | 1.9 |
| | | Ag | 242 | 347.7 | 32.00 | 1.42 | 1.6 |
| 96 | Scgl | Au | 3,189 | 3,481.8 | 157.43 | 0.71 | 3.8 |
| | | Ag | 3,189 | 3,481.8 | 1,552.00 | 4.15 | 5.7 |
| Total | | Au | 29,695 | 38,600.8 | 194.70 | 0.45 | 4.4 |
| | | Ag | 29,644 | 38,544.1 | 1,552.00 | 3.04 | 4.3 |

Source: Kirkham (2021).

**Figure 14-2: Box Plot Gold Assays for the Salinas Group Rocks**




Source: Kirkham (2021).

Table 14.6 lists the statistics for the Mita Group rocks units with the predominant unit being the Sandstone (Mss), Crystal Lithic Tuff (Mcv) and Lapilli Tuff (Mbt) units showing mean gold grades of 0.33 g/t, 0.61 g/t, 0.32 g/t and silver grades of 2.37 g/t, 1.49 g/t, 3.71 g/t, respectively. It is noted that the variability is very high with CVs ranging from 4.5 to 12.0. It is again clear from logging and modelling of the Mita that the Mbt and the Mcv represent the main stratigraphic units which are distinct and significant showing consistency and continuity throughout Cerro Blanco.

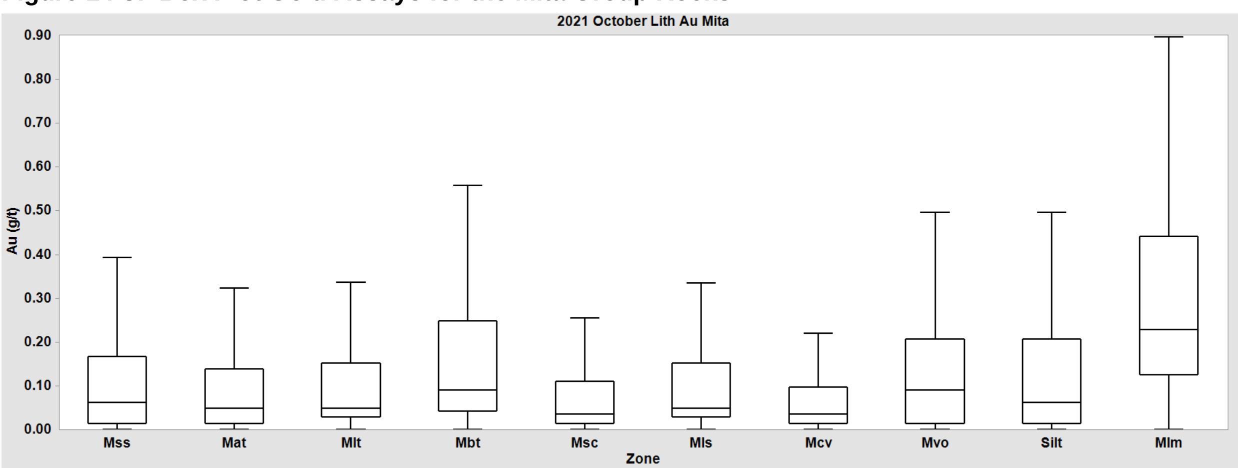
Figure 14-3 shows that the Lapilli Tuff (Mbt), Conglomerate (Mvo) and Siltstone (Silt) are statistically similar, and the Upper Limestone (Mlm) is statistically different from all of the other Mita rock units. All other rock units are statistically similar as shown in Figure 14-3. Further analysis and modelling for the purpose of grouping and domaining takes these observations and conclusion into account.

**Table 14.6: Statistics for Weighted Gold & Silver Assays for the Mita Group Rocks**

| Code | Lithology | Metal | Valid | Length (m) | Maximum (g/t) | Mean (g/t) | CV |
|---|---|---|---|---|---|---|---|
| 100 | Mss | Au | 10,292 | 12,214.2 | 368.33 | 0.33 | 12.0 |
| | | Ag | 10,292 | 12,214.2 | 2,405.90 | 2.37 | 10.8 |
| 101 | Mat | Au | 5,303 | 6,472.7 | 105.65 | 0.45 | 7.1 |
| | | Ag | 5,303 | 6,472.7 | 1,257.00 | 2.7 | 5.9 |
| 102 | Mlt | Au | 3,387 | 4,703.9 | 62.06 | 0.36 | 5.7 |
| | | Ag | 3,387 | 4,703.9 | 419.00 | 2.26 | 4.7 |
| 103 | Mbt | Au | 22,353 | 24,157.8 | 1,380.00 | 0.61 | 10.0 |
| | | Ag | 22,353 | 24,157.8 | 2,863.00 | 3.71 | 7.0 |
| 104 | Msc | Au | 3,183 | 3,988.7 | 180.73 | 0.34 | 8.4 |
| | | Ag | 3,183 | 3,988.7 | 624.60 | 2.2 | 5.5 |
| 105 | Mls | Au | 2,750 | 2,981.8 | 163.30 | 0.59 | 6.7 |
| | | Ag | 2,750 | 2,981.8 | 1,202.00 | 3.73 | 6.8 |
| 106 | Mcv | Au | 21,432 | 28,724.3 | 287.13 | 0.32 | 9.9 |
| | | Ag | 21,422 | 28,710.9 | 997.70 | 1.49 | 4.5 |
| 107 | Mvo | Au | 2,488 | 2,192.1 | 210.30 | 0.53 | 7.5 |
| | | Ag | 2,488 | 2,192.1 | 271.00 | 1.94 | 2.9 |
| 108 | Mlm | Au | 988 | 852.2 | 45.00 | 0.37 | 3.4 |
| | | Ag | 988 | 852.2 | 50.60 | 2.08 | 1.7 |
| 120 | Silt | Au | 2 | 6.1 | 0.00 | 0.00 | n/a |
| | | Ag | 2 | 6.1 | 0.00 | 0.00 | n/a |
| 130 | PA | Au | 497 | 388.5 | 132.90 | 0.29 | 7.0 |
| | | Ag | 497 | 388.5 | 125.00 | 1.56 | 2.4 |
| 190 | Mlm | Au | 98 | 73.0 | 14.90 | 0.86 | 2.6 |
| | | Ag | 98 | 73.0 | 101.00 | 6.08 | 1.7 |
| Total | | Au | 72,773 | 86,755.4 | 1,380.00 | 0.43 | 9.9 |
| | | Ag | 72,763 | 86,742.0 | 2,863.00 | 2.49 | 7.5 |

Source: Kirkham (2021).

**Figure 14-3: Box Plot Gold Assays for the Mita Group Rocks**




Source: Kirkham (2021.)

Table 14.7 below shows intervals that intersect Tempisque Volcanic Complex are primarily treated as waste.

**Table 14.7: Statistics for Weighted Gold & Silver Assays**

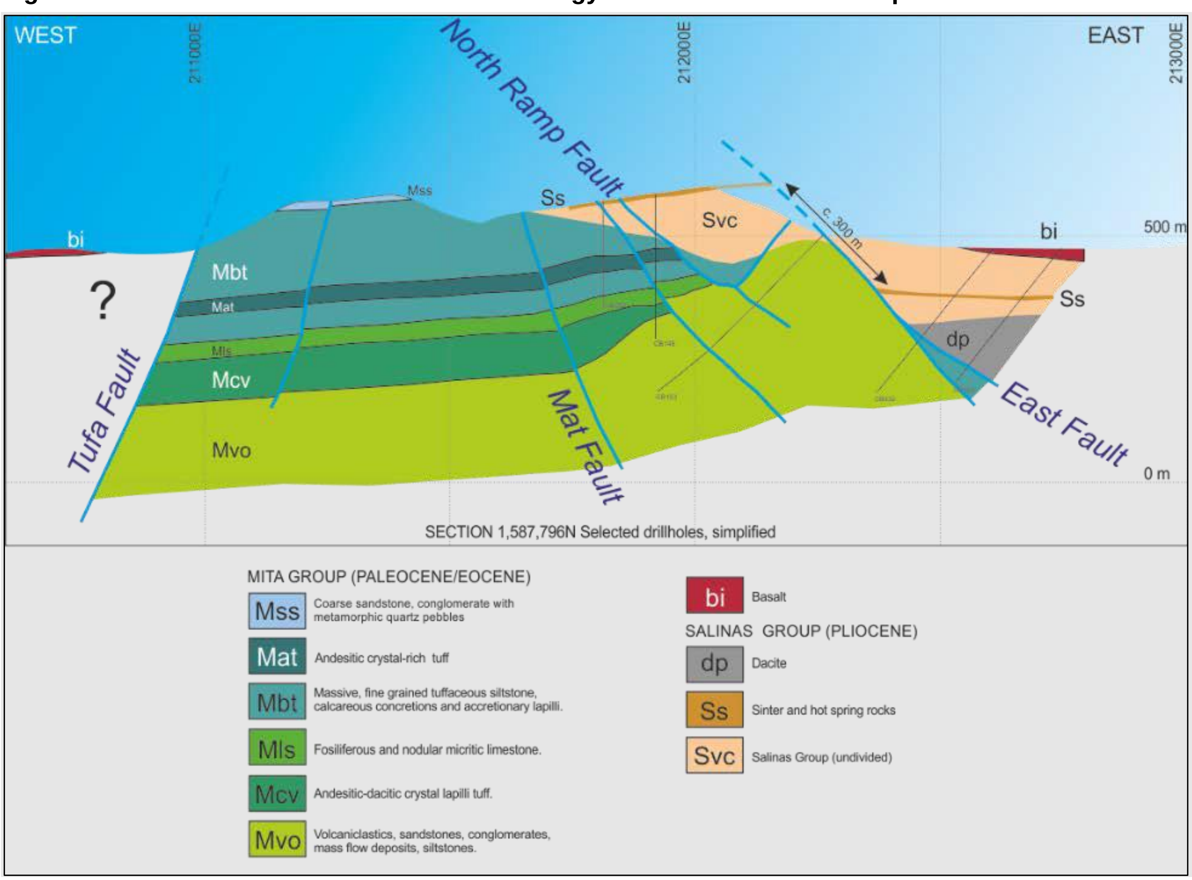
| Code | Lithology | Metal | Valid | Length (m) | Maximum (g/t) | Mean (g/t) | CV |
|---|---|---|---|---|---|---|---|
| 110 | Tcb | Au | 37 | 49.5 | 0.05 | 0.01 | 1.3 |
| | | Ag | 37 | 49.5 | 1.00 | 0.39 | 1.0 |
| 111 | Tca | Au | 697 | 1096.8 | 1.33 | 0.03 | 3.1 |
| | | Ag | 697 | 1096.8 | 13.00 | 0.77 | 1.2 |

Source: Kirkham (2021).

## 14.4 Geology & Domain Model

A three-phased modelling approach was taken to creating geology and estimation domains that included a lithostratigraphic model, detailed vein modelling, and domain modelling to estimate low-grade host rock solids within the Salinas and the Mita lithology units. The lithology models were completed using the lithology codes within the database (Section 14.3) as shown in Figure 14-4.

**Figure 14-4: Section View Schematic of Lithology for the Cerro Blanco Deposit**




Source: Kirkham (2021).

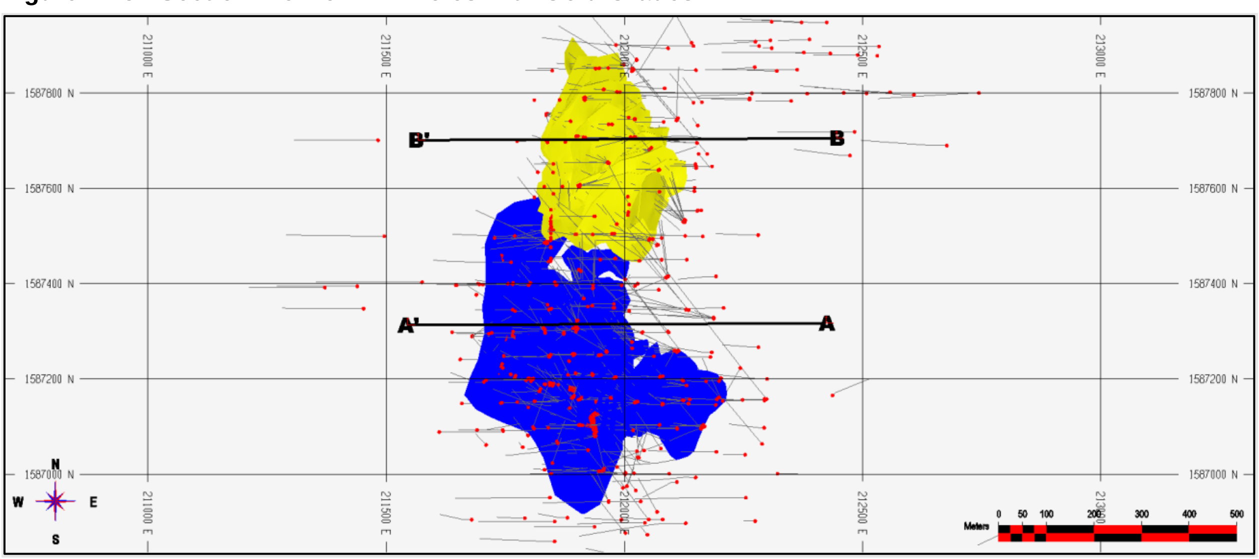
The models were created from first principals within LeapFrog™ and refined in MineSight™ for statistical analysis and to be used for the estimation process. Figure 14-3 illustrates the sectional interpretation of the main significant lithology units, namely the Salinas and Mita Group rock units discussed in Section 14.3. In addition, logging showed that within the Salinas there appeared to be zones of gouge potentially related to fault zones termed “TBX” that were determined to require modelling so that they could be masked out of the domain models.

In addition, solid models of each of the individual veins were created. These are displayed in plan view in Figure 14-5 with the north veins in yellow and the south veins in blue. In preparation for the creation of the vein models, a comprehensive structural model was developed that incorporated the current drilling, underground sampling, mapping, and extensive re-logging of drill core. The models were also created from first principals using the lithostratigraphic models and the structural modelling as guides by Bluestone staff within LeapFrog™ under the supervision of the independent QP. This was done utilizing the current and re-logged data, and from sectional interpretations that were subsequently wireframed based on a combination of lithology and gold grades.

Once completed, intersections were inspected, and all of the solids were then manually adjusted to match the drill intercepts. Once the solid models were edited and complete, they were used to code the drill hole assays and composites for subsequent statistical and geostatistical analysis. The solid zones were utilized to constrain the block model, by matching assays to those within the zones.

The orientation and ranges (distances) utilized for the search ellipsoids used in the estimation process were omni-directional and guided the strike and dip of the lithologic solids for the low-grade domains and by the highly constrained vein solids for the high-grade domains shown in Figure 14-5. The vein models were employed to estimate the high-grade structures on a partial block basis that are to be combined with the low-grade component to derive the whole block diluted grade for each block.

**Figure 14-5: Section View of Drill Holes with Gold Grades**

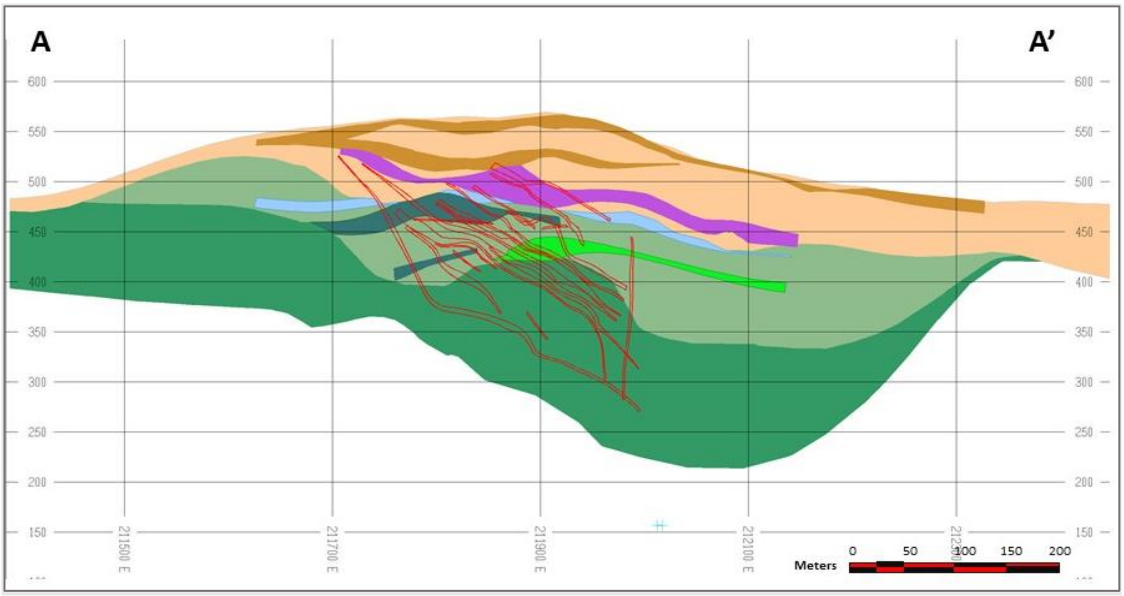



Notes: Yellow – north veins, blue – south veins. Source: Kirkham (2021)

The low-grade estimation domains were created using lithology as described in Section 14.3. The methodology was to determine which lithology units could be segregated or grouped based on grade profiles and it was determined that the Salinas be modelled as Salinas, Sinter, Basal Conglomerate. Within the Mita Group, the moderately mineralized volume that envelops that North and South vein clusters are predominantly the Mbt and Mcv units.

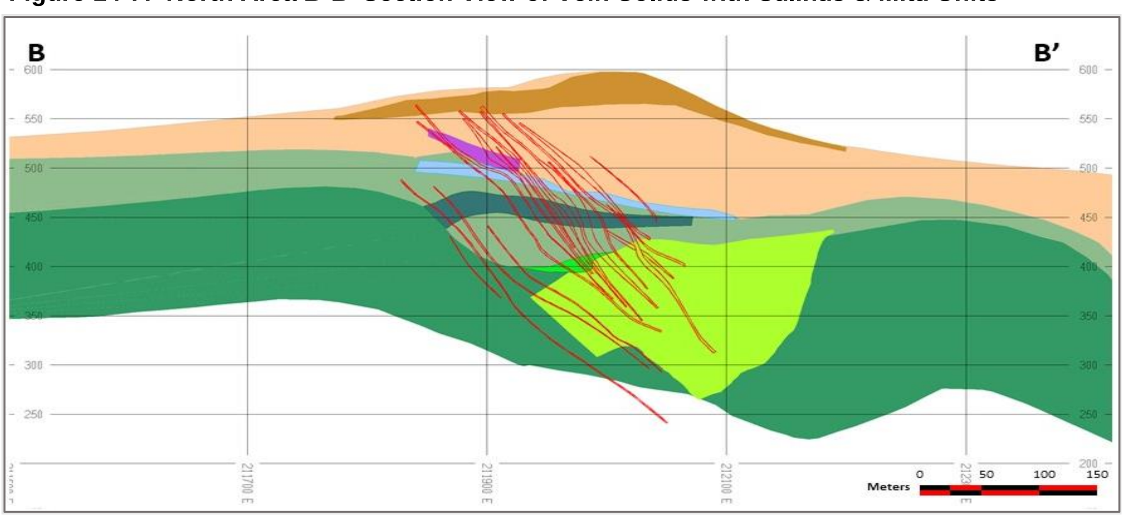
Figure 14-6 and Figure 14-7 illustrate the estimation domains in the north and south, respectively, which include the veins, Salinas and Mita units.

**Figure 14-6: South Area Section A-A′ View of Drill Holes, Vein Solids with Salinas & Mita Units**




Notes: Vein Solids – red; Sinter - brown polygons; Salinas - beige; Scgl conglomerate - purple; Mss - pale blue; Mat – sapphire blue, Mbt - pale green, Mls – bright green; Mcv – dark green. Source: Kirkham (2021).

**Figure 14-7: North Area B-B′ Section View of Vein Solids with Salinas & Mita Units**




Notes: Vein Solids – red; Sinter - brown polygons; Salinas - beige; Scgl conglomerate - purple; Mss - pale blue; Mat – sapphire blue, Mbt - pale green, Mls – bright green; Mcv – dark green. Source: Kirkham (2021).

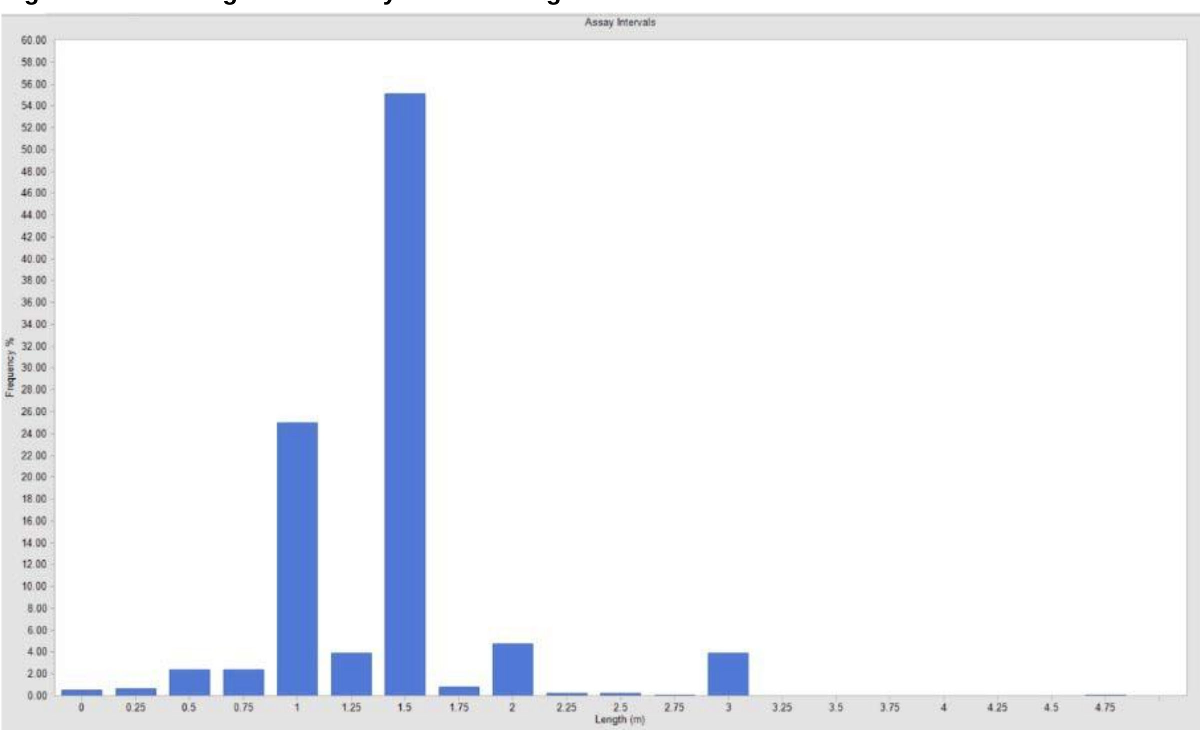
The solids were coded into the composite database in separate fields so as accurately account for the low- and high-grade components of each block along with the waste.

## 14.5 Composites

It was determined that the 1.5 m composite lengths offered the best balance between supplying common support for samples and minimizing the smoothing of grades. Figure 14-8 shows a histogram illustrating the distribution of the assay interval lengths for the complete database with 90% of the data having interval lengths greater than 1.5 m. Figure 14-9 shows the histogram of for the assay intervals limited to within the high-grade veins where 97.5% are less than or equal to 1.5 m; 16% are less than or equal to 1.0 m; and 2% are less than or equal to 0.5 m. To determine whether there may be selective sampling, an analysis of high-grade gold samples versus assay interval lengths was performed. The scatterplot of Figure 14-10 for samples within the high-grade veins shows that the assay intervals and corresponding gold grade have the same distribution and illustrate that there is not a high-grade bias within the small intervals and sample selectivity is not occurring.

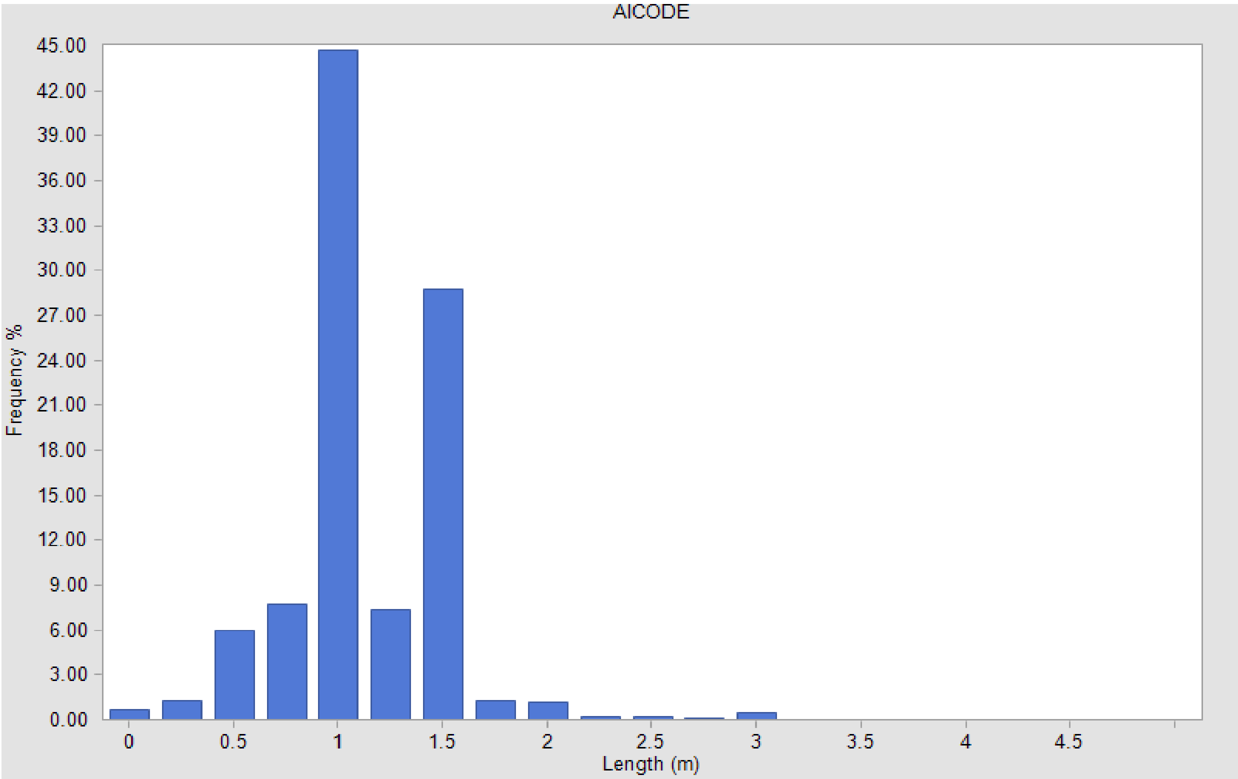
The 1.5 m sample length also was consistent with the distribution of sample lengths. It should be noted that although 1.5 m is the composite length, any residual composites of greater than 0.75 m in length and less than 1.5 m remained to represent a composite, while any composites residuals less than 0.75 m were combined with the composite above.

**Figure 14-8: Histogram of Assay Interval Lengths in Meters**



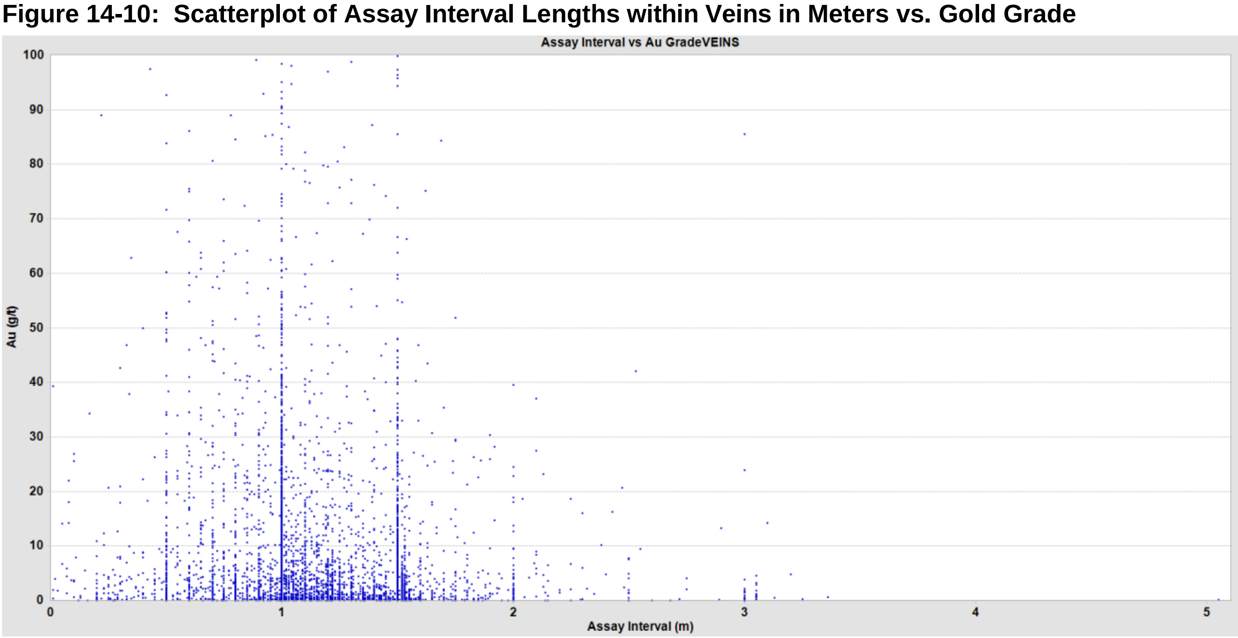

Source: Kirkham (2021).

**Figure 14-9: Histogram of Assay Interval Lengths within Veins in Meters**




Source: Kirkham (2021)

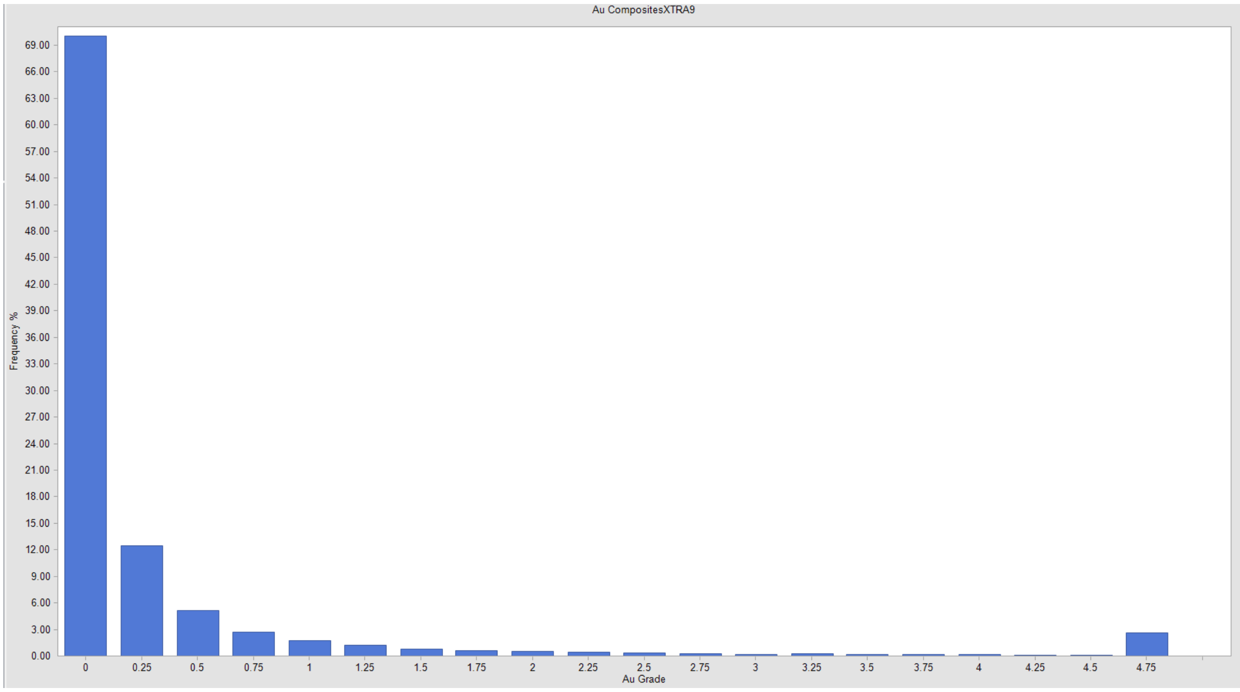
**Figure 14-10: Scatterplot of Assay Interval Lengths within Veins in Meters vs. Gold Grade**




Source: Kirkham (2021)

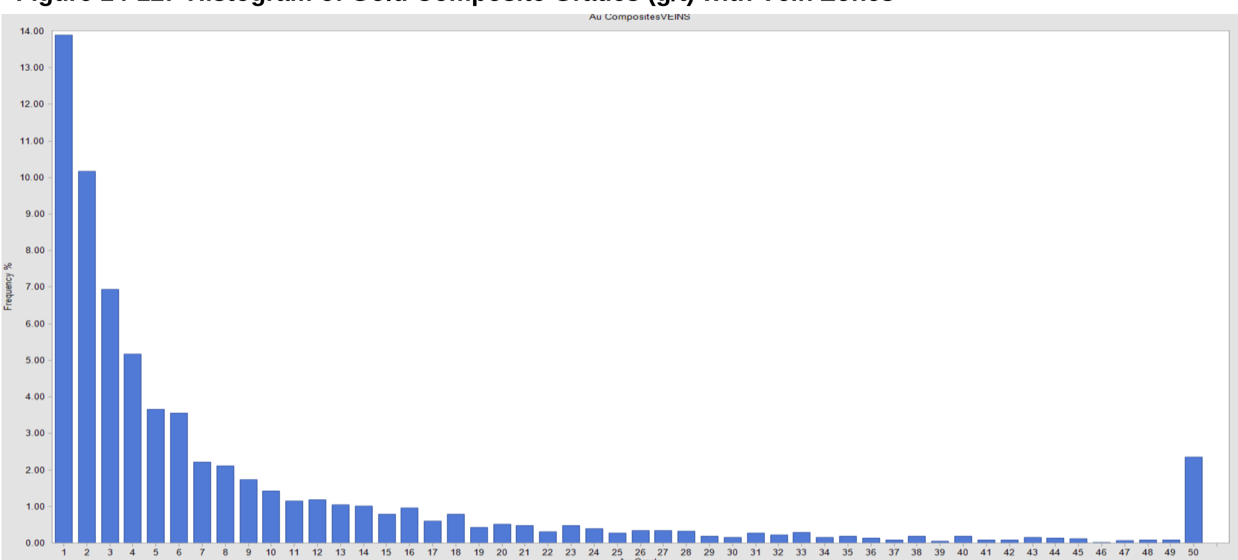
Figure 14-11 and Figure 14-12 show histograms of the gold composite values for all composites and for those that are assigned to the high-grade veins.

**Figure 14-11: Histogram of Gold Composite Grades (g/t)**




Source: Kirkham (2021)

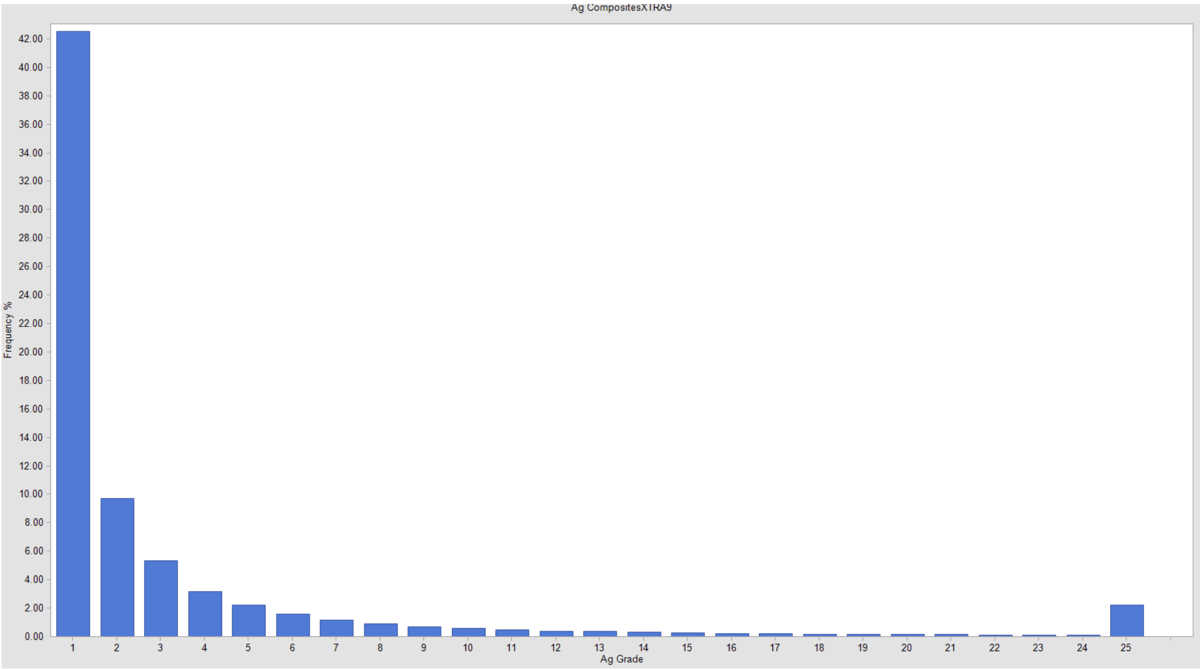
**Figure 14-12: Histogram of Gold Composite Grades (g/t) with Vein Zones**




Source: Kirkham (2021).

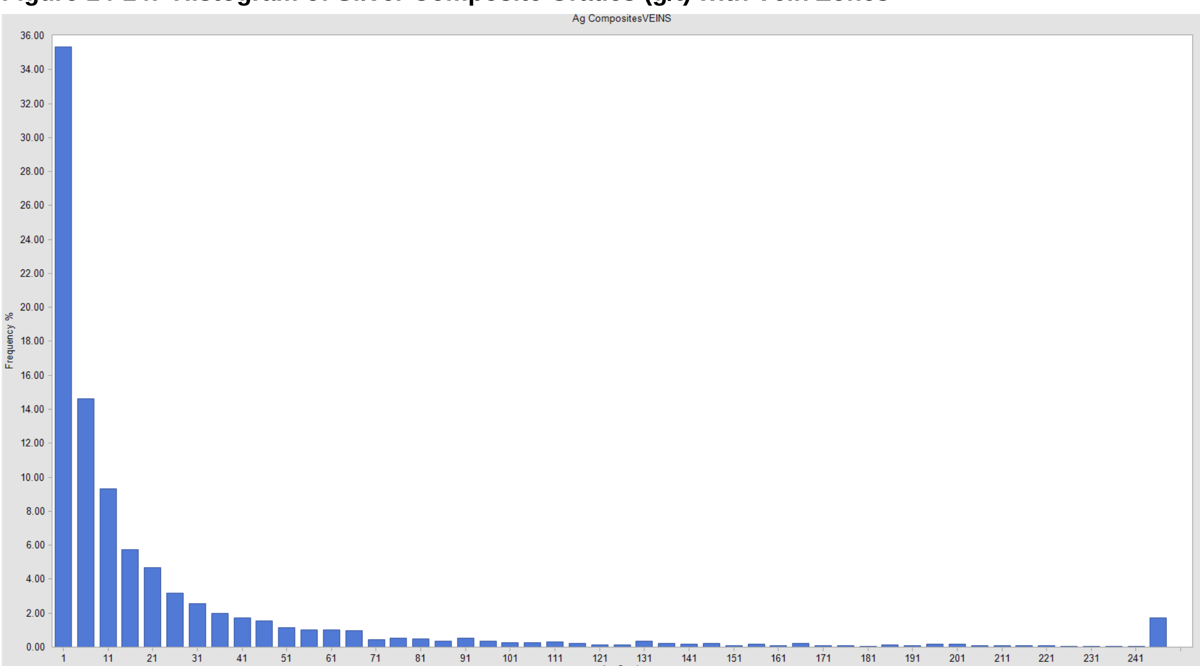
Figure 14-13 and Figure 14-14 show histograms silver composite values for all composites and for those that are assigned to the high-grade veins, respectively. The composite data demonstrates log-normal distributions in both cases.

**Figure 14-13: Histogram of Silver Composite Grades (g/t)**




Source: Kirkham (2021).

**Figure 14-14: Histogram of Silver Composite Grades (g/t) with Vein Zones**




Source: Kirkham (2021).

### 14.5.1 High-Grade Composite Analysis

The high-grade veins for north and south were grouped for statistical, geostatistical and estimation purposes by location and orientation in addition to relative grade profile. The results of these groupings are shown in Table 14.8 where there are two vein groups in the north and six groups in the south.

**Table 14.8: Vein Groupings for Derived for Statistical, Geostatistical & Estimation**

| Vein Domains Group | Vein Ranges |
|---|---|
| VN Group 1 | VN1-VN16, VN21-VN23, VN25 |
| VN Group 2 | VN17, VN18-VN20, VN24, VN26-VN30 |
| VS Group 11 | VS101 - VS103, NS121 |
| VS Group 12 | VS105-VS118 |
| VS Group 13 | VS119-VS120 |
| VS Group 14 | VS122-VS128 |
| VS Group 15 | VS132-VS138 |
| VS Group 16 | VS130-VS131, VS139 |

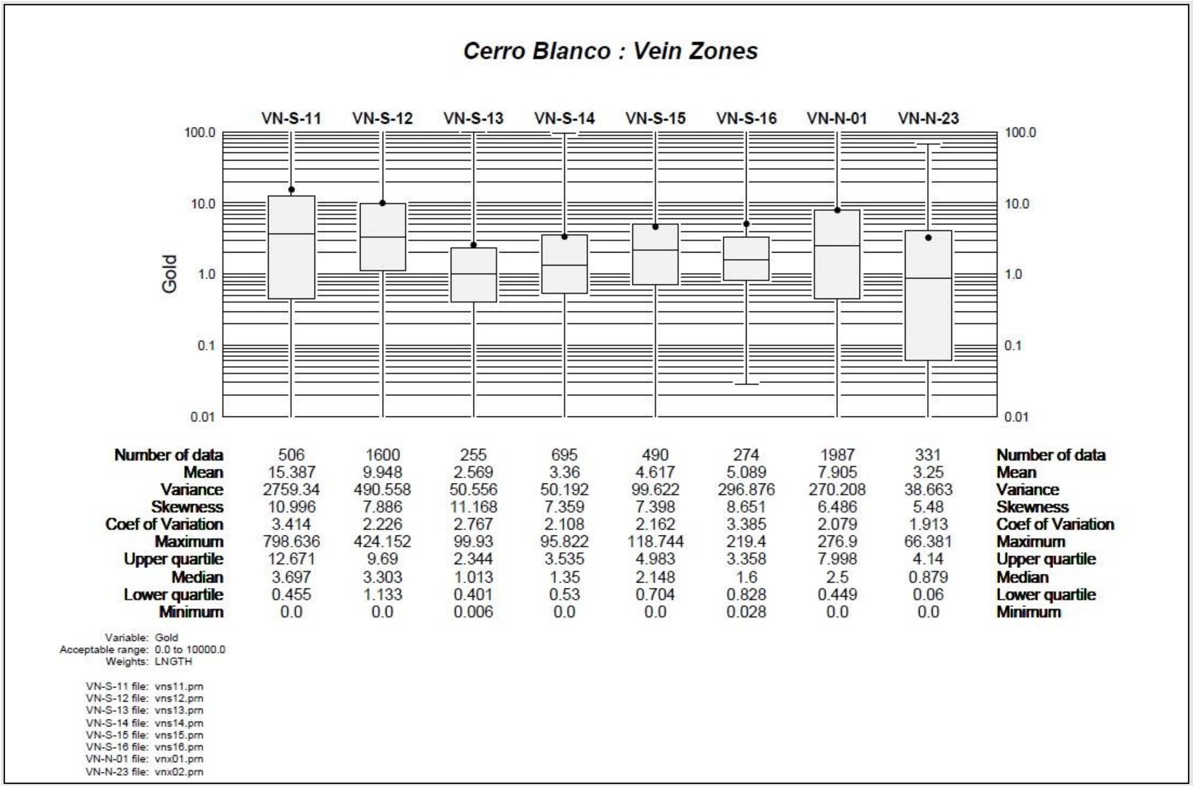
Source: Kirkham (2021)

Statistical analyses show the box plots and basic statistics for the grouped gold and silver composites for the high-grade vein domains (Figure 14-15 and Figure 14-16). Table 14.9 and Table 14.10 how the basic statistics for the 1.5 m gold and silver composite grades within the mineralized domains, respectively. There is a total of 6,107 composites or specifically 3,791 in the north zone and 2,316 in the south zone composites with 30 veins in the north and 36 veins in the south.

The weighted average gold grades for the north zone are 7.97 g/t and 7.28 g/t in the south zone with CVs being 3.2 and 2.1, respectively. Silver grades range from 31.6 g/t in the north and 26.8 g/t in the south with CVs being 3.4 to 3.4, respectively. CVs or variability is typically high for precious metal deposits primarily due to the nuggety nature of gold within epithermal veins; grade cutting will further reduce the CVs.

The box plots and statistics show that the mean gold grade very consistent between the north and the south zones. However, the spread (i.e., SD or standard deviation) and therefore the variability (i.e., CV) are higher in the south zone. This may be due to significant outlier grades in the south which has a maximum composite value of 792.3 g/t Au which is in the very high-grade volume in VS-101 versus 276.9 g/t in VN-6 in the north. Similarly, the mean silver grades are higher in the south versus the north at 31.57 g/t and 26.77 g/t, respectfully. In addition, the silver grades have similar distribution characteristics, not only north and south but also within the individual vein groupings, with there being approximately a 4:1 silver-to-gold ratio. Furthermore, variability is significantly greater in the south which is partially due to significant outlier grades where the maximum composite value is 3,540 g/t Ag in the south within VS-106 versus 1,257 g/t in the north within VN-5.

**Figure 14-15: Box Plot of Gold Composites for Veins**



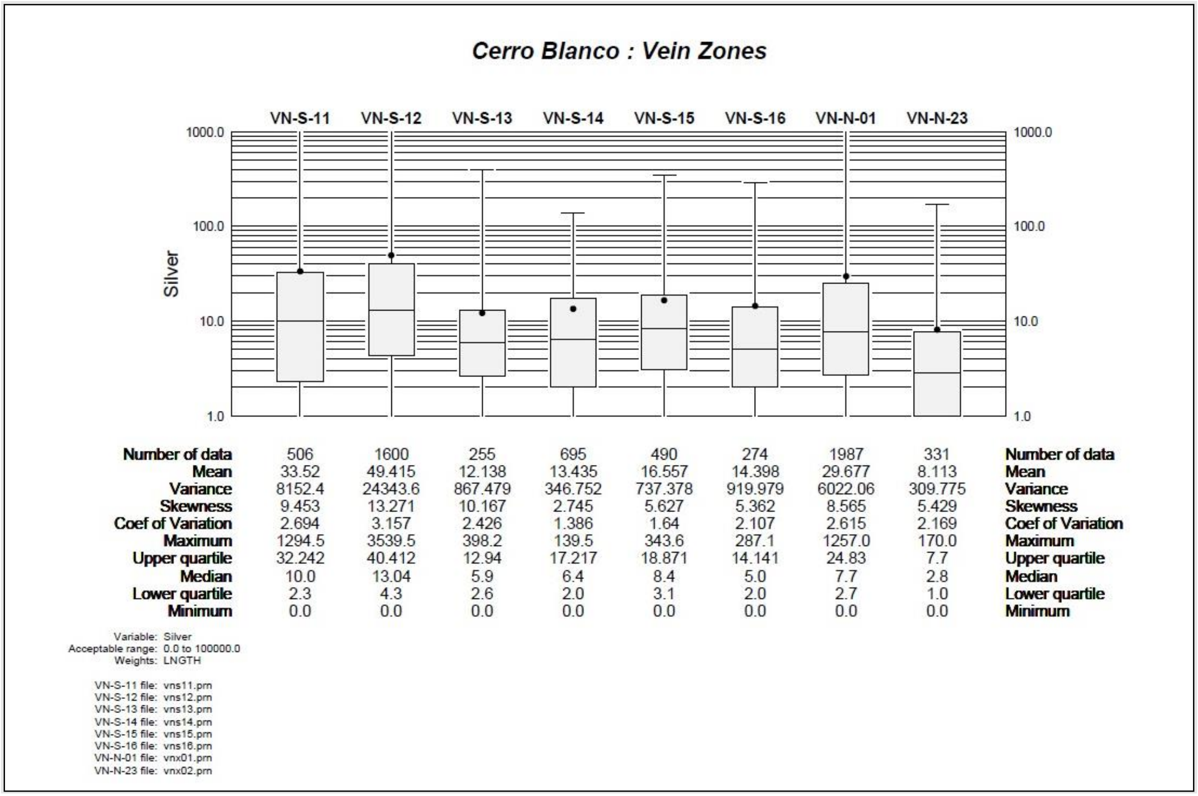

Source: Kirkham (2021).

**Table 14.9: Au Composite Statistics Weighted by Length for Veins**

| Gold (g/t) Composites | South | North |
|---|---|---|
| Valid | 3,791 | 2,316 |
| Length | 5,536 | 3,249.2 |
| Minimum | 0 | 0 |
| Maximum | 798.64 | 276.90 |
| Mean | 7.97 | 7.28 |
| First Quartile | 0.70 | 0.35 |
| Median | 2.30 | 2.33 |
| Third Quartile | 6.48 | 7.20 |
| Standard Deviation | 25.32 | 15.54 |
| Variance | 641.32 | 241.43 |
| Coefficient of Variation | 3.2 | 2.1 |

Source: Kirkham (2021)

**Figure 14-16: Box Plot of Silver Composites for Veins**




Source: Kirkham (2021)

**Table 14.10: Silver Composite Statistics Weighted by Length for Veins**

| Silver (g/t) Composites | South | North |
|---|---|---|
| Valid | 3,791.00 | 2,316.00 |
| Length | 5,536.00 | 3,249.20 |
| Minimum | 0.00 | 0.00 |
| Maximum | 3,539.50 | 1,257.00 |
| Mean | 31.57 | 26.77 |
| First Quartile | 3.03 | 2.34 |
| Median | 8.96 | 6.71 |
| Third Quartile | 25.36 | 21.99 |
| Standard Deviation | 108.70 | 72.83 |
| Variance | 11,814.74 | 5,303.76 |
| Coefficient of Variation | 3.40 | 2.70 |

Source: Kirkham (2021)

### 14.5.2 Low-Grade Composite Analysis

Figure 14-17 and Figure 14-18 show the box plots and basic statistics for the grouped (Table 14.11) gold and silver composites for the low-grade estimation domains. Table 14.12 and Table 14.13 show the basic statistics for the 1.5 m gold and silver composite grades within the low-grade domains, respectively.

**Table 14.11: Numeric Codes for Lithologies**

| Code | Lithological Unit |
|---|---|
| 60 | Salinas (Svc) |
| 61 | Sinter (Ss) |
| 62 | Mat |
| 70 | Mbt |
| 71 | Mcv |
| 72 | Mvo |
| 73 | Mat |
| 74 | Mss |
| 75 | Mls |
| 99 | Outside |

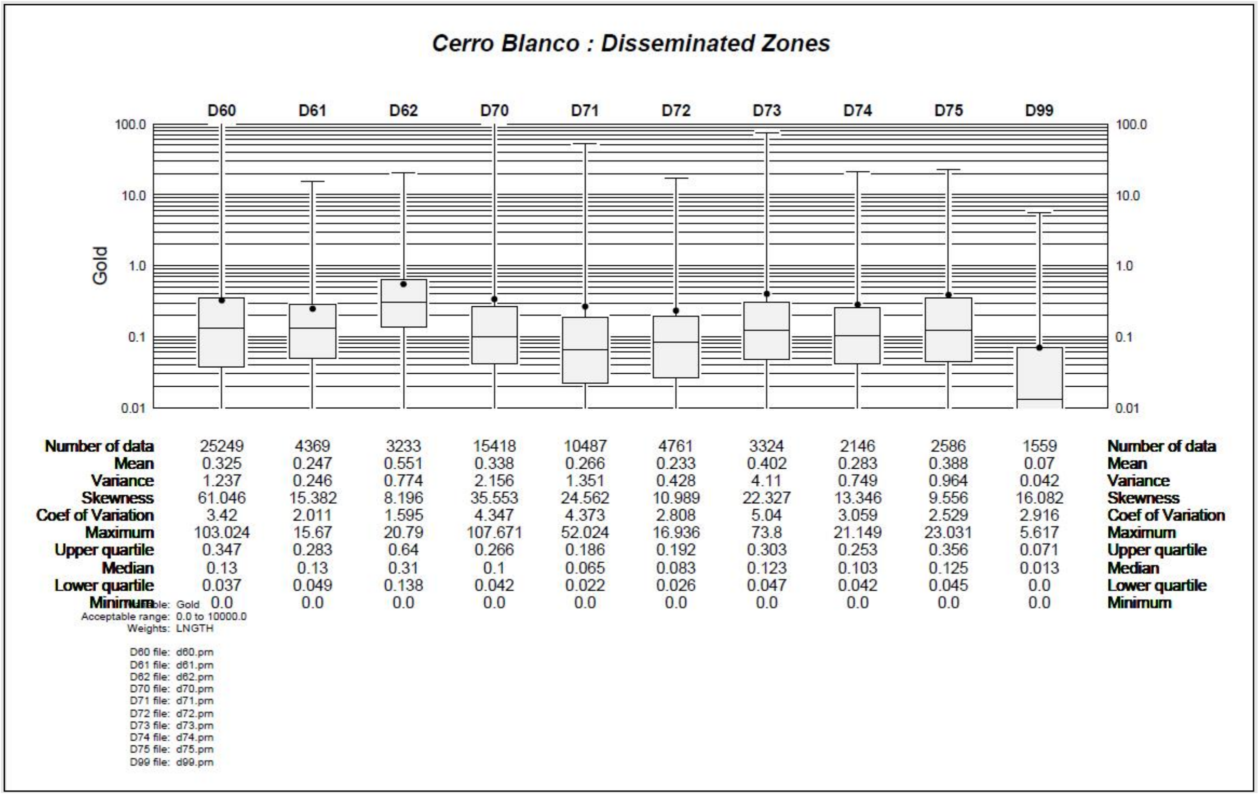
Source: Kirkham (2021)

The low-grade envelopes show weighted average gold grades of between 0.23 and 0.55 g/t, whilst CVs between 1.6 and 5.0 show moderate to very high variability which are addressed by a conservative grade limiting and cutting strategy. It is interesting to note that the Salinas (Svc) are markedly higher grade than those analyzed previously which have increased from 0.19 g/t to 0.32 g/t. This may be primarily attributable to updated and revised modelling of the Salinas and Sinter units that was guided by the 2021 drilling program and which focused on delineating and defining the surface resources. In addition, the Salinas Group Basal Conglomerate (Scgl) is a significantly higher-grade unit that has a mean gold grade (0.55 g/t) defined by the updated modelling.

The mean silver grades range from 1.7 to 3.4 g/t which is lower than the 3.6 to 6.9 g/t ranges for the low-grade envelopes previously, with the CVs ranging from low (1.2) to extreme (maximum of 39.0). As with the gold, grade limiting or cutting will further reduce the CVs. Again, it is clear that the low-grade domain composites require aggressive cutting.

In addition, the silver and gold grades have similar distribution characteristics, with there being an approximate silver-to-gold ratio of 7:1.

**Figure 14-17: Box Plot of Gold Composites for Low-Grade Domains**



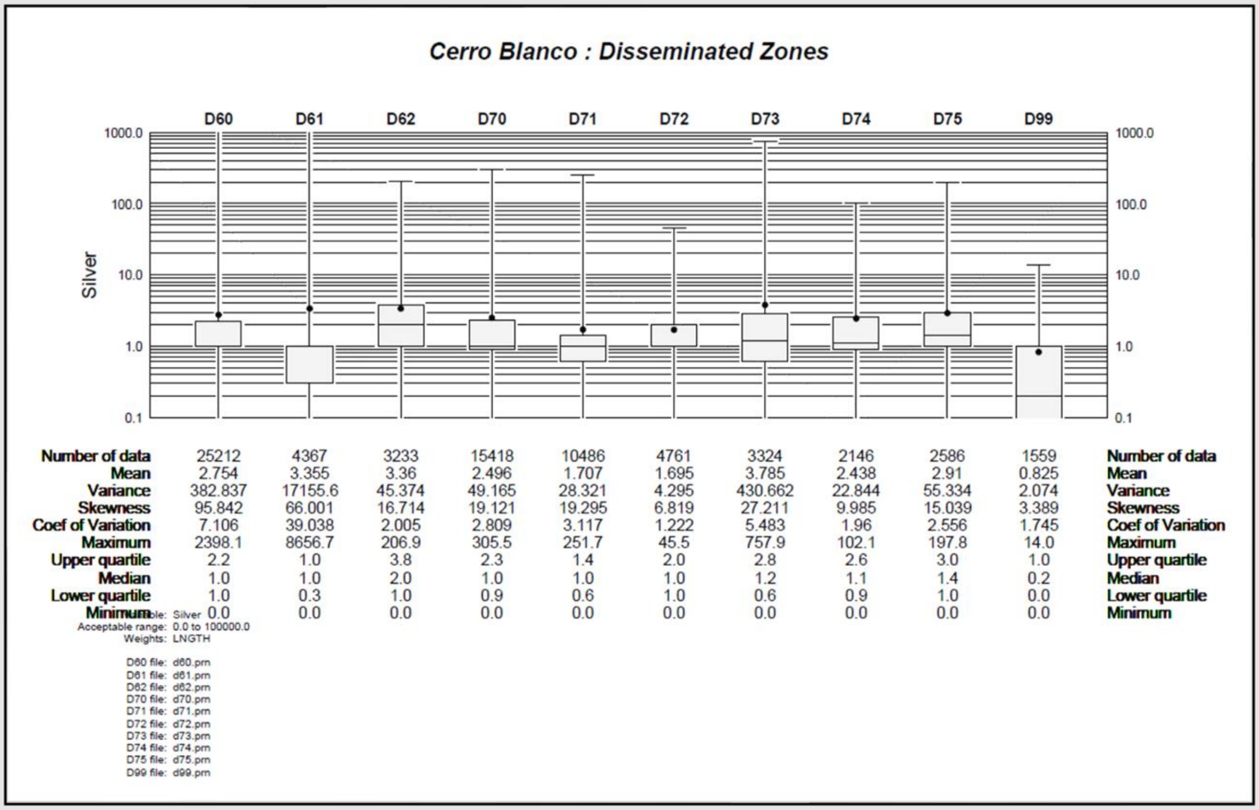

Source: Kirkham (2021).

**Table 14.12: Gold Composite Statistics Weighted by Length for Low-Grade Domains**

| Domain Code | Domain Name | No. | Length (m) | Maximum (g/t) | Mean (g/t) | CV |
|---|---|---|---|---|---|---|
| 60 | Svc | 25,248 | 37,832.51 | 103.02 | 0.32 | 3.4 |
| 61 | Ss | 4,369 | 6,556.73 | 15.67 | 0.25 | 2.0 |
| 62 | Scgl | 3,233 | 4,848.43 | 20.79 | 0.55 | 1.6 |
| 70 | Mbt | 15,418 | 23,098.32 | 107.67 | 0.34 | 4.3 |
| 71 | Mcv | 10,487 | 15,718.36 | 52.02 | 0.27 | 4.4 |
| 72 | Mvo | 4,761 | 7,125.94 | 16.94 | 0.23 | 2.8 |
| 73 | Mat | 3,324 | 4,934.78 | 73.80 | 0.40 | 5.0 |
| 74 | Mss | 2,146 | 3,217.06 | 21.15 | 0.28 | 3.1 |
| 75 | Mls | 2,586 | 3,871.43 | 23.03 | 0.39 | 2.5 |
| 99 | Outside | 1,559 | 2,336.16 | 5.62 | 0.07 | 2.9 |

Source: Kirkham (2021).

**Figure 14-18: Box Plot of Silver Composites for Low-Grade Domains**




Source: Kirkham (2021)

**Table 14.13: Silver Composite Statistics Weighted by Length for Low-Grade Domains**

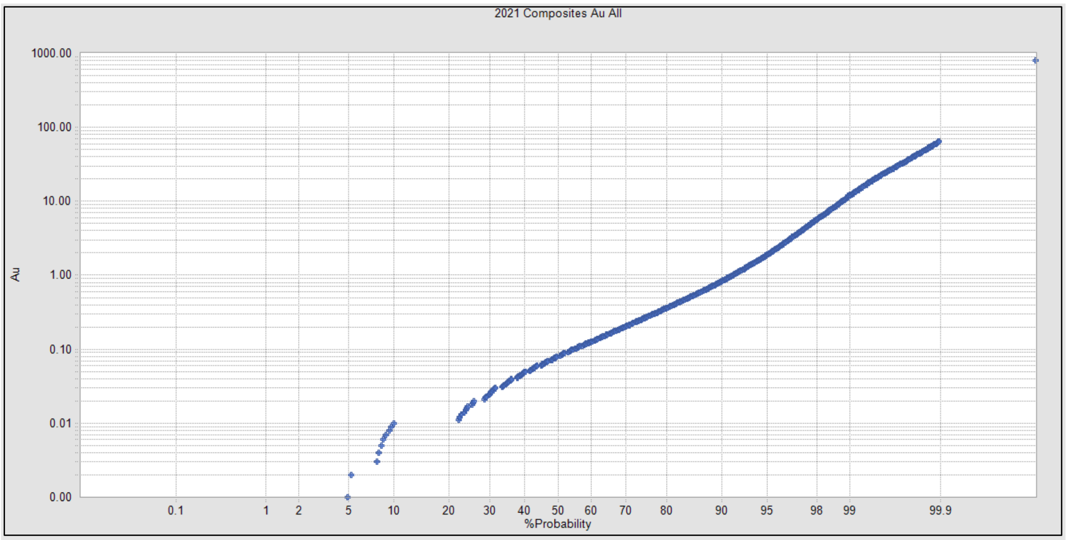
| Domain Code | Domain Name | No. | Length (m) | Maximum (g/t) | Mean (g/t) | CV |
|---|---|---|---|---|---|---|
| 60 | Svc | 25,211 | 37,777.01 | 2,398.1 | 2.75 | 7.1 |
| 61 | Ss | 4,367 | 6,553.73 | 8,656.7 | 3.36 | 39.0 |
| 62 | Scgl | 3,233 | 4,848.43 | 206.9 | 3.36 | 2.0 |
| 70 | Mbt | 15,418 | 23,098.32 | 305.5 | 2.50 | 2.8 |
| 71 | Mcv | 10,486 | 15,717.11 | 251.7 | 1.71 | 3.1 |
| 72 | Mvo | 4,761 | 7,125.94 | 45.5 | 1.70 | 1.2 |
| 73 | Mat | 3,324 | 4,934.78 | 757.9 | 3.78 | 5.5 |
| 74 | Mss | 2,146 | 3,217.06 | 102.1 | 2.44 | 2.0 |
| 75 | Mls | 2,586 | 3,871.43 | 197.8 | 2.91 | 2.6 |
| 99 | Outside | 1,559 | 2,336.16 | 14.0 | 0.83 | 1.8 |

Source: Kirkham (2021).

## 14.6 Evaluation of Outlier Assay Values

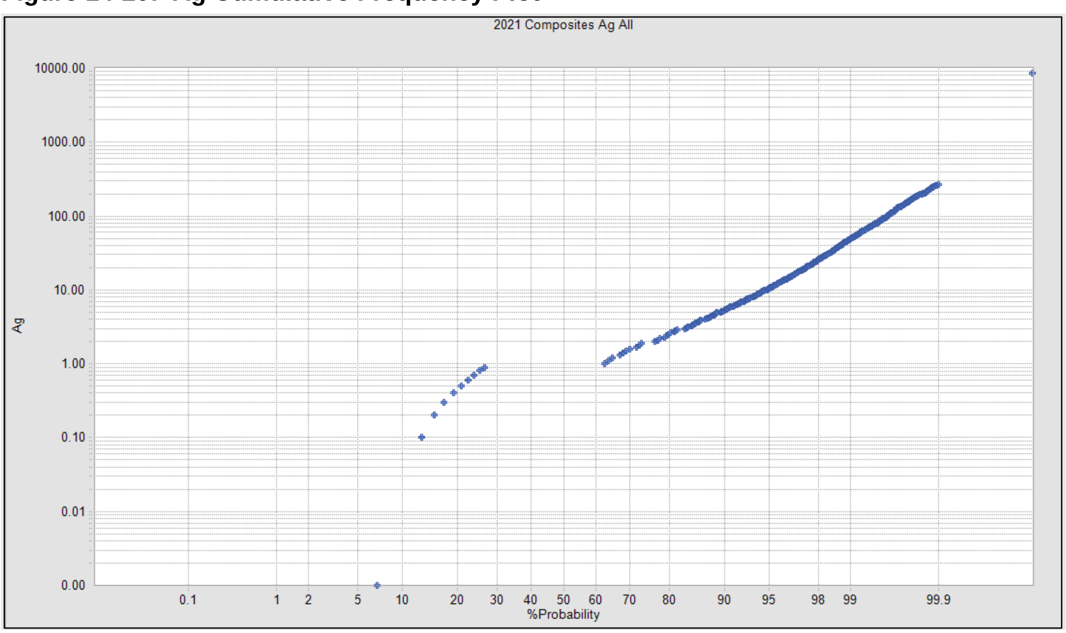
During the estimation process, the influence of outlier composites is controlled to limit their influence and to ensure against over-estimation of metal content. The high-grade outlier thresholds were chosen by domain and are based on an analysis of the breaks in the cumulative frequency plots for each of the vein groupings and the individual low-grade domains. Figure 14-19 and Figure 14-20 show examples of the gold and silver cumulative frequency plots for all composites, respectively.

**Figure 14-19: Au Cumulative Frequency Plot**




Source: Kirkham (2021).

**Figure 14-20: Ag Cumulative Frequency Plot**




Source: Kirkham (2021).

In the case of the gold composites within the high-grade vein domains, values as high as 110 g/t were cut; for the silver composites, values as high as 500 g/t were cut. Table 14.14 shows the various cut thresholds for the vein groupings, while Table 14.15 shows the cut grades for the low-grade domains.

**Table 14.14: Cut Grades for Au & Ag within Vein Domains**

| Vein Domains Group | Domains | Au Cut Threshold (g/t) | Ag Cut Threshold (g/t) |
|---|---|---|---|
| VN Group 1 | VN1-VN16, VN21-VN23, VN25 | 80 | 280 |
| VN Group 2 | VN17, VN18-VN20, VN24, VN26-VN30 | 15 | 40 |
| VS Group 11 | VS101 - VS103, VS121 | 110 | 180 |
| VS Group 12 | VS105-VS118 | 110 | 500 |
| VS Group 13 | VS119-VS120 | 10 | 110 |
| VS Group 14 | VS122-VS128 | 22 | 90 |
| VS Group 15 | VS132-VS138 | 20 | 95 |
| VS Group 16 | VS130-VS131, VS139 | 50 | 110 |

Source: Kirkham (2021).

**Table 14.15: Cut Grades for Au & Ag within Low-Grade Domains**

| Low-Grade Domain Name | Domain Code | Au Cut Threshold (g/t) | Ag Cut Threshold (g/t) |
|---|---|---|---|
| Salinas | 60 | 11 | 110 |
| Sinter | 61 | 6 | 50 |
| SCGL | 62 | 10 | 50 |
| MBT | 70 | 15 | 50 |
| MCV | 71 | 4.5 | 15 |
| MVO | 72 | 4.5 | 45.5 |
| MAT | 73 | 4 | 50 |
| MSS | 74 | 7 | 35 |
| MLS | 75 | 5 | 50 |
| Outside | 99 | 0.6 | 10 |

Source: Kirkham (2021).

Tables 14.16 and 14.17 show the effects of cutting the outlier grades within the high-grade vein domain groupings and the low-grade Salinas and Mita units, respectively. The conclusion is that the cutting strategy is highly successful in addressing the outlier grade populations, both within the high-grade veins and the lower-grade Salinas and Mita units.

**Table 14.16: Cut vs. Uncut Comparisons for Gold & Silver Composites – High-Grade Vein Domain Groupings**

| Gold | Maximum (g/t) | Mean (g/t) | CV | Cut Threshold (g/t) | Mean (g/t) | CV | Mean % | CV (%) |
|---|---|---|---|---|---|---|---|---|
| **1** | 276.9 | 7.9 | 2.1 | 80 | 7.53 | 1.7 | -5 | -16% |
| **2** | 66.38 | 3.27 | 1.9 | 15 | 2.87 | 1.4 | -12 | -2 |
| **11** | 798.64 | 15.39 | 3.4 | 110 | 11.91 | 1.8 | -23 | -48 |
| **12** | 424.15 | 9.95 | 2.2 | 110 | 9.38 | 1.8 | -6 | -19 |
| **13** | 99.93 | 2.57 | 2.8 | 10 | 2.03 | 1.3 | -21 | -54 |
| **14** | 95.82 | 3.36 | 2.1 | 22 | 2.99 | 1.4 | -11 | -31 |
| **15** | 118.74 | 4.65 | 2.2 | 20 | 3.8 | 1.2 | -18 | -45 |
| **16** | 219.4 | 5.09 | 3.4 | 50 | 3.89 | 1.9 | -24 | -43 |
| **Total** | 798.64 | 7.7 | 2.9 | 110 | 6.93 | 2 | -10 | -32 |
| **Silver** | **Maximum (g/t)** | **Mean (g/t)** | **CV** | **Cut Threshold (g/t)** | **Mean (g/t)** | **CV** | **Mean (%)** | **CV (%)** |
| **1** | 1,257.00 | 29.68 | 2.6 | 280 | 26.56 | 1.9 | -11 | -28 |
| **2** | 170 | 8.2 | 2.2 | 40 | 6.65 | 1.4 | -19 | -33 |
| **11** | 1,294.50 | 33.52 | 2.7 | 180 | 26.97 | 1.5 | -20 | -44 |
| **12** | 3,539.50 | 49.42 | 3.2 | 500 | 42.88 | 1.9 | -13 | -40 |
| **13** | 398.2 | 12.14 | 2.4 | 110 | 10.82 | 1.5 | -11 | -40 |
| **14** | 139.5 | 13.44 | 1.4 | 90 | 13.18 | 1.3 | -2 | -5 |
| **15** | 343.6 | 16.74 | 1.6 | 95 | 15.55 | 1.3 | -7 | -22 |
| **16** | 287.1 | 14.4 | 2.1 | 110 | 12.91 | 1.6 | -10 | -22 |
| **Total** | 3,539.50 | 29.75 | 3.3 | 500 | 26.16 | 2.1 | -12 | -37 |

Source: Kirkham (2021).

**Table 14.17: Cut vs. Uncut Comparisons for Gold & Silver Composites – Salinas & Mita Domains**

| Gold | Maximum (g/t) | Mean (g/t) | CV | Cut Threshold (g/t) | Mean (g/t) | CV | Mean (%) | CV (%) |
|---|---|---|---|---|---|---|---|---|
| 60 | 103.02 | 0.324 | 3.4 | 11 | 0.316 | 2 | -2 | -41 |
| 61 | 15.67 | 0.247 | 2 | 6 | 0.242 | 1.6 | -2 | -21 |
| 62 | 20.79 | 0.551 | 1.6 | 10 | 0.546 | 1.5 | -1 | - |
| 70 | 107.67 | 0.361 | 4 | 15 | 0.344 | 2.7 | -5 | -33 |
| 71 | 52.02 | 0.26 | 4.5 | 4.5 | 0.223 | 2.4 | -14 | -46 |
| 72 | 16.94 | 0.233 | 2.8 | 4.5 | 0.221 | 2.2 | -5 | -20 |
| 73 | 73.8 | 0.403 | 5 | 4 | 0.306 | 1.8 | -24 | -64 |
| 74 | 21.15 | 0.284 | 3.1 | 7 | 0.268 | 2.3 | -6 | -24 |
| 75 | 23.03 | 0.388 | 2.5 | 5 | 0.36 | 1.9 | -7 | -24 |
| Total | 107.67 | 0.327 | 3.7 | 15 | 0.308 | 2.3 | -6 | -38 |
| **Silver** | **Maximum (g/t)** | **Mean (g/t)** | **CV** | **Cut Threshold (g/t)** | **Mean (g/t)** | **CV** | **Mean (%)** | **CV (%)** |
| 60 | 2,398.10 | 2.75 | 7.1 | 110 | 2.55 | 2.1 | -7 | -70 |
| 61 | 8,656.70 | 3.36 | 39 | 50 | 1.35 | 1.9 | -60 | -95 |
| 62 | 206.9 | 3.36 | 2 | 50 | 3.24 | 1.4 | -4 | -32 |
| 70 | 305.5 | 2.61 | 2.8 | 50 | 2.46 | 1.8 | -6 | -35 |
| 71 | 251.7 | 1.69 | 3.1 | 15 | 1.43 | 1.4 | -15 | -54 |
| 72 | 45.5 | 1.7 | 1.2 | 45.5 | 1.7 | 1.2 | 0 | 0 |
| 73 | 757.9 | 3.79 | 5.5 | 50 | 2.97 | 2 | -22 | -63 |
| 74 | 102.1 | 2.44 | 2 | 35 | 2.35 | 1.5 | -4 | -21 |
| 75 | 197.8 | 2.91 | 2.6 | 50 | 2.73 | 1.7 | -6 | -35 |
| Total | 8,656.70 | 2.6 | 13.3 | 110 | 2.28 | 1.9 | -12 | -85 |

Source: Kirkham (2021)

## 14.7 Specific Gravity Estimation

Table 14.18 shows the specific gravity (SG) assignment by zone using 1,308 individual measurements and standard water displacement methods. The SG assigned for the veins is determined at 2.52 from 534 measurements. It is recommended that future work programs should continue to include SG measurements to expand the density distributions, particularly within the upper lithology units.

**Table 14.18: SG Zone Assignments**

| Lithology Group | Domain / Rock | No. | Density ($gm/mm^3$) | Average Density ($gm/mm^3$) |
|---|---|---|---|---|
| Salinas Group | Ss | 27 | 2.49 | |
| | Scgl | 35 | 2.46 | |
| | Svc | 115 | 2.46 | |
| | Rp | 6 | 2.48 | |
| | **Total** | **183** | | **2.47** |
| Mita Group | Mat | 48 | 2.54 | |
| | Mbt | 272 | 2.58 | |
| | Mss | 88 | 2.56 | |
| | Mls | 36 | 2.62 | |
| | Mcv | 102 | 2.59 | |
| | Mvo | 38 | 2.52 | |
| | Silt | 7 | 2.56 | |
| | **Total** | **591** | | **2.57** |
| Vein | Vt | 534 | 2.52 | |
| | **Total** | **1,308** | | **2.54** |

Source: Kirkham (2021).

## 14.8 Variography

Experimental variograms and variogram models in the form of correlograms were generated for gold and silver grades. The definition of nugget value was derived from the downhole variograms. The correlograms for gold and silver within veins in the south and north zones are shown in Figure 14-21 and 14-22 for gold and silver, respectively. These variogram models were used to estimate gold and silver grades using ordinary kriging as the interpolator to estimate the high-grade veins.

**Figure 14-21: Gold Correlogram Models**

### Cerro Blanco North Vein Zone 1 Correlograms

Medsystem and Vulcan Rotation Conventions

Nugget ==> 0.450
C1 ==> 0.415
C2 ==> 0.135

First Structure -- Spherical

LH Rotation about the Z axis ==> -0
RH Rotation about the X' axis ==> 52
LH Rotation about the Y' axis ==> 26
Range along the Z' axis ==> 9.6 Azimuth ==> 212 Dip ==> 33
Range along the Y' axis ==> 11.8 Azimuth ==> 360 Dip ==> 52
Range along the X' axis ==> 2.5 Azimuth ==> 111 Dip ==> 16

Second Structure -- Spherical

LH Rotation about the Z axis ==> 22
RH Rotation about the X' axis ==> 30
LH Rotation about the Y' axis ==> 27
Range along the Z axis ==> 88.1 Azimuth ==> 247 Dip ==> 50
Range along the X' axis ==> 15.2 Azimuth ==> 126 Dip ==> 23
Range along the Y' axis ==> 100.1 Azimuth ==> 22 Dip ==> 30

Modeling Criteria

Minimum number pairs req'd ==> 50
Sample variogram points weighted by # pairs

### Cerro Blanco North Vein Zone 2-3 Correlograms

Medsystem and Vulcan Rotation Conventions

Nugget ==> 0.101
C1 ==> 0.334
C2 ==> 0.565

First Structure -- Spherical

LH Rotation about the Z axis ==> 48
RH Rotation about the X' axis ==> -14
LH Rotation about the Y' axis ==> -29
Range along the Z' axis ==> 5.8 Azimuth ==> 114 Dip ==> 58
Range along the Y' axis ==> 29.2 Azimuth ==> 48 Dip ==> -14
Range along the X' axis ==> 31.1 Azimuth ==> 145 Dip ==> -28

Second Structure -- Spherical

LH Rotation about the Z axis ==> -13
RH Rotation about the X' axis ==> 19
LH Rotation about the Y' axis ==> 5
Range along the Z axis ==> 90.8 Azimuth ==> 182 Dip ==> 70
Range along the X' axis ==> 14.4 Azimuth ==> 79 Dip ==> 5
Range along the Y' axis ==> 15.6 Azimuth ==> 347 Dip ==> 19

Modeling Criteria

Minimum number pairs req'd ==> 50
Sample variogram points weighted by # pairs

### Cerro Blanco South Vein Zone 11 Correlograms

Medsystem and Vulcan Rotation Conventions

Nugget ==> 0.500
C1 ==> 0.276
C2 ==> 0.223

First Structure -- Spherical

LH Rotation about the Z axis ==> -31
RH Rotation about the X' axis ==> 26
LH Rotation about the Y' axis ==> 53
Range along the Z' axis ==> 1.4 Azimuth ==> 220 Dip ==> 33
Range along the Y' axis ==> 6.1 Azimuth ==> 329 Dip ==> 26
Range along the X' axis ==> 31.8 Azimuth ==> 89 Dip ==> 46

Second Structure -- Spherical

LH Rotation about the Z axis ==> -44
RH Rotation about the X' axis ==> 49
LH Rotation about the Y' axis ==> 58
Range along the Z axis ==> 110.1 Azimuth ==> 201 Dip ==> 21
Range along the X' axis ==> 9.7 Azimuth ==> 96 Dip ==> 34
Range along the Y' axis ==> 77.8 Azimuth ==> 316 Dip ==> 49

Modeling Criteria

Minimum number pairs req'd ==> 50
Sample variogram points weighted by # pairs

### Cerro Blanco South Vein Zone 12 Correlograms

Medsystem and Vulcan Rotation Conventions

Nugget ==> 0.450
C1 ==> 0.347
C2 ==> 0.203

First Structure -- Spherical

LH Rotation about the Z axis ==> -7
RH Rotation about the X' axis ==> -30
LH Rotation about the Y' axis ==> 26
Range along the Z' axis ==> 11.2 Azimuth ==> 309 Dip ==> 51
Range along the Y' axis ==> 2.3 Azimuth ==> 353 Dip ==> -30
Range along the X' axis ==> 2.8 Azimuth ==> 69 Dip ==> 22

Second Structure -- Spherical

LH Rotation about the Z axis ==> -24
RH Rotation about the X' axis ==> 38
LH Rotation about the Y' axis ==> -10
Range along the Z axis ==> 13.7 Azimuth ==> 140 Dip ==> 51
Range along the X' axis ==> 78.1 Azimuth ==> 60 Dip ==> -8
Range along the Y' axis ==> 39.3 Azimuth ==> 336 Dip ==> 38

Modeling Criteria

Minimum number pairs req'd ==> 50
Sample variogram points weighted by # pairs

## Cerro Blanco South Vein Zone 13 Correlograms

Medsystem and Vulcan Rotation Conventions

Nugget ==> 0.462
C1 ==> 0.282
C2 ==> 0.256

First Structure -- Spherical

LH Rotation about the Z axis ==> 48
RH Rotation about the X' axis ==> -1
LH Rotation about the Y' axis ==> -3
Range along the Z' axis ==> 13.0 Azimuth ==> 118 Dip ==> 86
Range along the Y' axis ==> 5.4 Azimuth ==> 48 Dip ==> -1
Range along the X' axis ==> 20.8 Azimuth ==> 138 Dip ==> -3

Second Structure -- Spherical

LH Rotation about the Z axis ==> 40
RH Rotation about the X' axis ==> -25
LH Rotation about the Y' axis ==> -52
Range along the Z axis ==> 41.0 Azimuth ==> 111 Dip ==> 34
Range along the X' axis ==> 14.7 Azimuth ==> 159 Dip ==> -45
Range along the Y' axis ==> 6.2 Azimuth ==> 40 Dip ==> -25

Modeling Criteria

Minimum number pairs req'd ==> 50
Sample variogram points weighted by # pairs

## Cerro Blanco South Vein Zone 14 Correlograms

Medsystem and Vulcan Rotation Conventions

Nugget ==> 0.048
C1 ==> 0.799
C2 ==> 0.153

First Structure -- Spherical

LH Rotation about the Z axis ==> 74
RH Rotation about the X' axis ==> 7
LH Rotation about the Y' axis ==> 7
Range along the Z' axis ==> 55.7 Azimuth ==> 298 Dip ==> 80
Range along the Y' axis ==> 3.0 Azimuth ==> 74 Dip ==> 7
Range along the X' axis ==> 5.2 Azimuth ==> 164 Dip ==> 7

Second Structure -- Spherical

LH Rotation about the Z axis ==> -11
RH Rotation about the X' axis ==> -43
LH Rotation about the Y' axis ==> 35
Range along the Z axis ==> 24.6 Azimuth ==> 304 Dip ==> 37
Range along the X' axis ==> 3.3 Azimuth ==> 54 Dip ==> 25
Range along the Y' axis ==> 5.2 Azimuth ==> 349 Dip ==> -43

Modeling Criteria

Minimum number pairs req'd ==> 50
Sample variogram points weighted by # pairs

## Cerro Blanco South Vein Zone 15 Correlograms

Medsystem and Vulcan Rotation Conventions

Nugget ==> 0.248
C1 ==> 0.505
C2 ==> 0.247

First Structure -- Spherical

LH Rotation about the Z axis ==> 6
RH Rotation about the X' axis ==> 75
LH Rotation about the Y' axis ==> -54
Range along the Z' axis ==> 3.3 Azimuth ==> 131 Dip ==> 9
Range along the Y' axis ==> 16.2 Azimuth ==> 6 Dip ==> 75
Range along the X' axis ==> 7.5 Azimuth ==> 43 Dip ==> -12

Second Structure -- Spherical

LH Rotation about the Z axis ==> 30
RH Rotation about the X' axis ==> 80
LH Rotation about the Y' axis ==> -41
Range along the Z axis ==> 20.8 Azimuth ==> 169 Dip ==> 8
Range along the X' axis ==> 2.5 Azimuth ==> 80 Dip ==> -7
Range along the Y' axis ==> 21.1 Azimuth ==> 30 Dip ==> 80

Modeling Criteria

Minimum number pairs req'd ==> 50
Sample variogram points weighted by # pairs

## Cerro Blanco South Vein Zone 16 Correlograms

Medsystem and Vulcan Rotation Conventions

Nugget ==> 0.020
C1 ==> 0.770
C2 ==> 0.210

First Structure -- Spherical

LH Rotation about the Z axis ==> -2
RH Rotation about the X' axis ==> 4
LH Rotation about the Y' axis ==> -19
Range along the Z' axis ==> 37.4 Azimuth ==> 100 Dip ==> 71
Range along the Y' axis ==> 16.8 Azimuth ==> 358 Dip ==> 4
Range along the X' axis ==> 5.0 Azimuth ==> 86 Dip ==> -19

Second Structure -- Spherical

LH Rotation about the Z axis ==> -29
RH Rotation about the X' axis ==> 17
LH Rotation about the Y' axis ==> -8
Range along the Z axis ==> 27.1 Azimuth ==> 124 Dip ==> 71
Range along the X' axis ==> 49.3 Azimuth ==> 58 Dip ==> -8
Range along the Y' axis ==> 80.1 Azimuth ==> 331 Dip ==> 17

Modeling Criteria

Minimum number pairs req'd ==> 50
Sample variogram points weighted by # pairs

**Figure 14-22: Silver Correlogram Models**

## Cerro Blanco North Vein Zone 1 Ag Correlograms

Medsystem and Vulcan Rotation Conventions

Nugget ==> 0.296
C1 ==> 0.596
C2 ==> 0.108

First Structure -- Spherical

LH Rotation about the Z axis ==> 20
RH Rotation about the X' axis ==> 15
LH Rotation about the Y' axis ==> 35
Range along the Z' axis ==> 33.7 Azimuth ==> 270 Dip ==> 52
Range along the Y' axis ==> 4.2 Azimuth ==> 20 Dip ==> 15
Range along the X' axis ==> 3.1 Azimuth ==> 121 Dip ==> 34

Second Structure -- Spherical

LH Rotation about the Z axis ==> 28
RH Rotation about the X' axis ==> 47
LH Rotation about the Y' axis ==> 54
Range along the Z axis ==> 376.0 Azimuth ==> 270 Dip ==> 24
Range along the X' axis ==> 39.8 Azimuth ==> 163 Dip ==> 34
Range along the Y' axis ==> 212.4 Azimuth ==> 28 Dip ==> 47

Modeling Criteria

Minimum number pairs req'd ==> 50
Sample variogram points weighted by # pairs

## Cerro Blanco North Vein Zone 2-3 Ag Correlograms

Medsystem and Vulcan Rotation Conventions

Nugget ==> 0.271
C1 ==> 0.536
C2 ==> 0.193

First Structure -- Spherical

LH Rotation about the Z axis ==> 33
RH Rotation about the X' axis ==> 87
LH Rotation about the Y' axis ==> -36
Range along the Z' axis ==> 14.2 Azimuth ==> 177 Dip ==> 2
Range along the Y' axis ==> 5.2 Azimuth ==> 33 Dip ==> 87
Range along the X' axis ==> 20.4 Azimuth ==> 87 Dip ==> -2

Second Structure -- Spherical

LH Rotation about the Z axis ==> -20
RH Rotation about the X' axis ==> -43
LH Rotation about the Y' axis ==> -2
Range along the Z axis ==> 242.0 Azimuth ==> 343 Dip ==> 47
Range along the X' axis ==> 46.2 Azimuth ==> 71 Dip ==> -1
Range along the Y' axis ==> 6.3 Azimuth ==> 340 Dip ==> -43

Modeling Criteria

Minimum number pairs req'd ==> 50
Sample variogram points weighted by # pairs

## Cerro Blanco South Vein Zone 11 Ag Correlograms

Medsystem and Vulcan Rotation Conventions

Nugget ==> 0.203
C1 ==> 0.679
C2 ==> 0.118

First Structure -- Spherical

LH Rotation about the Z axis ==> -2
RH Rotation about the X' axis ==> -9
LH Rotation about the Y' axis ==> 6
Range along the Z' axis ==> 4.0 Azimuth ==> 324 Dip ==> 79
Range along the Y' axis ==> 8.3 Azimuth ==> 358 Dip ==> -9
Range along the X' axis ==> 17.0 Azimuth ==> 87 Dip ==> 6

Second Structure -- Spherical

LH Rotation about the Z axis ==> 13
RH Rotation about the X' axis ==> 1
LH Rotation about the Y' axis ==> 40
Range along the Z axis ==> 315.0 Azimuth ==> 283 Dip ==> 50
Range along the X' axis ==> 10.1 Azimuth ==> 104 Dip ==> 40
Range along the Y' axis ==> 80.0 Azimuth ==> 13 Dip ==> 1

Modeling Criteria

Minimum number pairs req'd ==> 50
Sample variogram points weighted by # pairs

## Cerro Blanco South Vein Zone 12 Ag Correlograms

Medsystem and Vulcan Rotation Conventions

Nugget ==> 0.450
C1 ==> 0.248
C2 ==> 0.302

First Structure -- Spherical

LH Rotation about the Z axis ==> -51
RH Rotation about the X' axis ==> 56
LH Rotation about the Y' axis ==> 31
Range along the Z' axis ==> 3.4 Azimuth ==> 165 Dip ==> 29
Range along the Y' axis ==> 5.0 Azimuth ==> 309 Dip ==> 56
Range along the X' axis ==> 1.0 Azimuth ==> 66 Dip ==> 17

Second Structure -- Spherical

LH Rotation about the Z axis ==> 10
RH Rotation about the X' axis ==> 34
LH Rotation about the Y' axis ==> 60
Range along the Z axis ==> 82.8 Azimuth ==> 262 Dip ==> 24
Range along the X' axis ==> 9.7 Azimuth ==> 144 Dip ==> 46
Range along the Y' axis ==> 28.9 Azimuth ==> 10 Dip ==> 34

Modeling Criteria

Minimum number pairs req'd ==> 50
Sample variogram points weighted by # pairs

**Cerro Blanco South Vein Zone 13 Ag Correlograms**

Medsystem and Vulcan Rotation Conventions

Nugget ==> 0.113
C1 ==> 0.751
C2 ==> 0.136

First Structure -- Spherical

LH Rotation about the Z axis ==> 44
RH Rotation about the X' axis ==> -16
LH Rotation about the Y' axis ==> 52
Range along the Z' axis ==> 18.3 Azimuth ==> 326 Dip ==> 37
Range along the Y' axis ==> 5.6 Azimuth ==> 44 Dip ==> -16
Range along the X' axis ==> 21.9 Azimuth ==> 115 Dip ==> 49

Second Structure -- Spherical

LH Rotation about the Z axis ==> -52
RH Rotation about the X' axis ==> -2
LH Rotation about the Y' axis ==> 6
Range along the Z axis ==> 45.9 Azimuth ==> 242 Dip ==> 84
Range along the X' axis ==> 14.5 Azimuth ==> 38 Dip ==> 6
Range along the Y' axis ==> 4.9 Azimuth ==> 308 Dip ==> -2

Modeling Criteria

Minimum number pairs req'd ==> 50
Sample variogram points weighted by # pairs

In addition, experimental variograms and variogram models in the form of correlograms were alsogenerated for gold and silver grades within the low-grade domains namely, Salinas and Mita units. As above, the definition of nugget value was derivedfrom the downhole variograms. The correlograms models for gold and silver are shown in Table 14.19 and 14.20, respectively. These variogram models were used to estimate gold and silver grades using ordinary kriging as the interpolator.

**Table 14.19: Geostatistical Model Parameters for Gold by Lithology Unit**

| Code | 60 | 61 | 62 | 70 | 71 | 72 | 73 | 74 | 75 |
|---|---|---|---|---|---|---|---|---|---|
| Domain Name | Salinas | Sinter | ScglMat | Mbt | Mcv | Mvo | Mat | Mss | Mls |
| Nugget (C0) | 0.45 | 0.1 | 0.384 | 0.475 | 0.50 | 0.597 | 0.184 | 0.588 | 0.60 |
| First Sill (C1) | 0.439 | 0.512 | 0.406 | 0.466 | 0.456 | 0.343 | 0.56 | 0.236 | 0.333 |
| Second Sill (C2) | 0.111 | 0.388 | 0.21 | 0.059 | 0.044 | 0.059 | 0.256 | 0.176 | 0.067 |
| 1st Structure | | | | | | | | | |
| Range along the Z' | 18.1 | 3.6 | 9.7 | 7.2 | 7.8 | 7.9 | 26.9 | 8.9 | 2.0 |
| Range along the X' | 10.8 | 26.9 | 9.4 | 4.9 | 4.9 | 22.3 | 2.9 | 33.9 | 5.8 |
| Range along the Y' | 25.7 | 2.3 | 4.5 | 5.2 | 5.5 | 3.6 | 31.8 | 1.9 | 44.2 |
| R1 about the Z | -151 | -91 | -7 | 4 | -21 | 15 | 91 | 1 | 37 |
| R2 about the X' | 35 | -52 | 8 | -37 | -50 | 57 | -47 | 41 | -2 |
| R3 about the Y' | -4 | 2 | -11 | 56 | 57 | 81 | 73 | -42 | -4 |
| 2nd Structure | | | | | | | | | |
| Range along the Z' | 136.6 | 152.6 | 204.4 | 196.5 | 100.8 | 275 | 76.2 | 12 | 302.3 |
| Range along the X' | 103 | 56.1 | 94.3 | 63.6 | 55 | 67.5 | 13.6 | 82 | 126.8 |

| | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|
| Range along the Y' | 402.9 | 105.6 | 49.8 | 134.6 | 289 | 332 | 26.5 | 246 | 1405.4 |
| R1 about the Z | 2 | 24 | 45 | 2 | -73 | 34 | 32 | 19 | -15 |
| R2 about the X' | -10 | 56 | 1 | 24 | 58 | 171 | 14 | 41 | 37 |
| R3 about the Y' | -4 | -23 | -14 | 30 | 54 | -28 | 33 | 54 | 41 |

Source: Kirkham (2021).

**Table 14.20: Geostatistical Model Parameters for Silver by Lithology Unit**

| Code | 60 | 61 | 62 | 70 | 71 | 72 | 73 | 74 | 75 |
|---|---|---|---|---|---|---|---|---|---|
| Domain Name | Salinas | Sinter | Scgl | Mbt | Mcv | Mvo | Mat | Mss | Mls |
| Nugget (C0) | 0.40 | 0.231 | 0.3 | 0.425 | 0.167 | 0.462 | 0.35 | 0.279 | 0.274 |
| First Sill (C1) | 0.415 | 0.528 | 0.465 | 0.494 | 0.542 | 0.377 | 0.533 | 0.599 | 0.44 |
| Second Sill (C2) | 0.185 | 0.241 | 0.235 | 0.081 | 0.291 | 0.161 | 0.117 | 0.122 | 0.285 |
| 1st Structure | | | | | | | | | |
| Range along the Z' | 20.2 | 3.8 | 8.2 | 6.2 | 17.3 | 6.8 | 4.9 | 5.1 | 20.1 |
| Range along the X' | 4.0 | 32 | 3.4 | 9.3 | 8.3 | 17.9 | 30.6 | 37.4 | 7.9 |
| Range along the Y' | 8.8 | 2.7 | 33.5 | 4.2 | 3.8 | 43.8 | 19.8 | 2.7 | 1.8 |
| R1 about the Z | 1.0 | 7.0 | -67 | -34 | 4 | 23 | -14 | -54 | -18 |
| R2 about the X' | -44 | -13 | 87 | 23 | -10 | 9 | -31 | -15 | -1 |
| R3 about the Y' | 41 | -24 | 20 | 52 | -15 | -22 | 33 | -53 | -20 |
| 2nd Structure | | | | | | | | | |
| Range along the Z' | 278.7 | 133.2 | 265.1 | 153 | 157.8 | 132.8 | 77.6 | 70.3 | 108.3 |
| Range along the X' | 45.5 | 10 | 86.3 | 67.6 | 16.8 | 278.3 | 19 | 115.7 | 13.4 |
| Range along the Y' | 70.8 | 89.5 | 73.4 | 208.2 | 27.9 | 71 | 117.9 | 67.3 | 36.7 |
| R1 about the Z | -16 | 8 | 49 | 42 | 15 | 7 | 61 | -27 | 79 |
| R2 about the X' | 21 | 32 | 43 | 182 | -30 | 35 | 10 | 90 | 15 |
| R3 about the Y' | 71 | -8 | 21 | -39 | 36 | -44 | -39 | -5 | -20 |

Source: Kirkham (2021).

## 14.9 Block Model Definition

The block model used for estimating the resources was defined according to the origin and orientation shown in Figure 14-23 and the limits specified in Figure 14-24.

**Figure 14-23: Block Model Origin & Orientation**

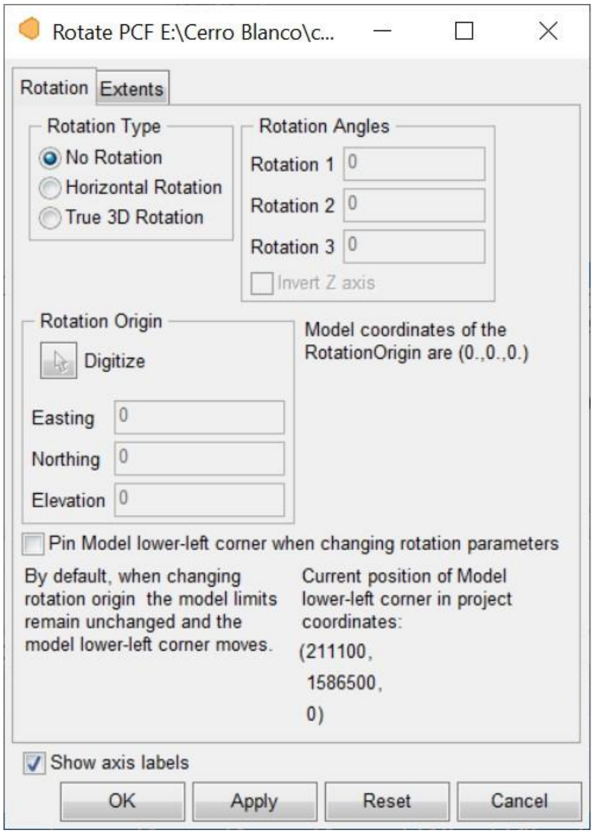
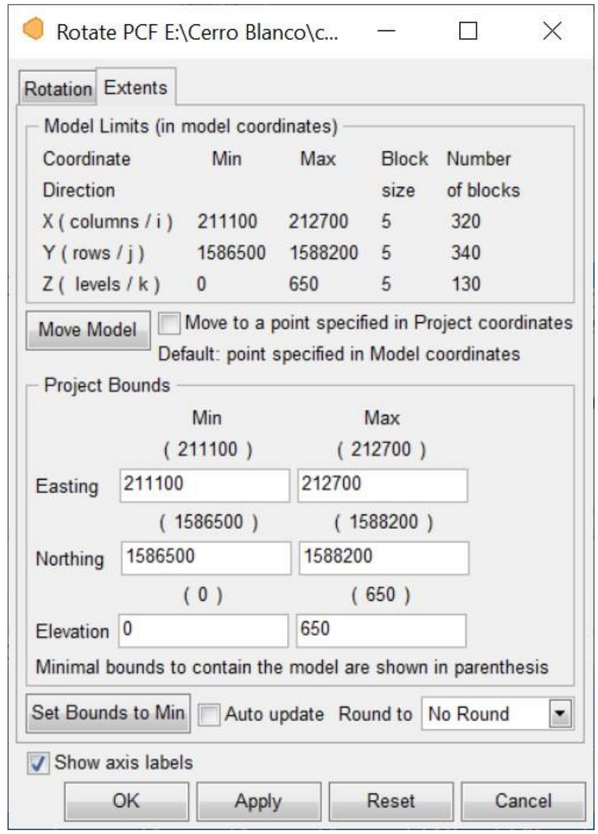



**Figure 14-24: Block Model Extents & Dimensions**




Source: Kirkham (2021).

The block model employs whole blocking for ease of mine planning and is orthogonal and non-rotated, roughly reflecting the orientation of the north and the south vein sets within the deposit. The block size chosen was 5 m x 5 m x 5 m. Note that MineSight™ uses the centroid of the blocks as the origin.

## 14.10 Resource Estimation Methodology

The estimation strategy was a two-step process that entailed estimating the high-grade vein component of each block and the low-grade mineralized host rock component. Once completed, the final whole block grades were created and determined by way of a weighted average calculation.

The estimation plan for the high-grade vein component was as follows:

- vein code of modelled mineralization stored in each block along with partial percentage
- specific gravity estimation for the vein
- block gold and silver grade estimation by ordinary kriging
- one pass estimation for each individual vein using hard boundaries.

A minimum of three composites and maximum of nine composites, and a maximum of three composites per hole, were used to estimate block grades. Following Herco analysis, it was determined there is an appropriate amount of smoothing.

For the vein domains that make up the Cerro Blanco deposit, the search ellipsoids are omni-directional to a maximum of 100 m; however, hard boundaries were used so that the domains are tightly constrained and grade is not smeared between veins.

The estimation plan for the low-grade mineralized host rock component included the following:

- domain code of modelled mineralization stored in each block
- specific gravity estimation based on rock type code
- block gold and silver grade estimation by ordinary kriging
- one pass estimation for each domain using hard boundaries.

A minimum of three composites and maximum of twelve composites, and a maximum of three composites per hole, were informed to estimate block grades. Following Herco analysis, it was determined there is an appropriate amount of smoothing for the low-grade domains.

For the vein domain domains that make up the Cerro Blanco deposit, the search ellipsoids are omni-directional to a maximum of 100 m, and hard boundaries were used so that grade is not smeared between the units.

The final whole block gold (AU) and silver (AG) grades in addition to density (SG) are calculated using a weighted average formula that is comprised of the vein component (HG), the low-grade host rock component (LG). In addition, the blocks are also weighted by topography (TOPO) and voids (UG) particularly the underground ramp and workings. The formula is as follows:

*AU Total = (AUHG* %HG + AULG * %LG + %TOPO + %UG) / 100*

*AG Total = (AGHG* %HG + AGLG * %LG + %TOPO + %UG) / 100*

*SG Total = (SGHG* %HG + SGLG * %LG + %TOPO + %UG) / 100*

## 14.11 Mineral Resource Classification

Mineral resources were estimated in conformity with generally accepted CIM "Estimation of Mineral Resource and Mineral Reserve Best Practices" Guidelines (2019). Mineral resources are not mineral reserves and do not have demonstrated economic viability. Mineral resources for the Cerro Blanco deposit were classified according to the CIM Definition Standards for Mineral Resources and Mineral Reserves (2014) by Garth Kirkham, P. Geo., of Kirkham Geosystems Ltd. (KGL), an Independent Qualified Person as defined by N.I. 43-101.

The mineral resources may be impacted by further infill and exploration drilling that may result in an increase or decrease in future resource evaluations. The mineral resources may also be affected by subsequent assessment of mining, environmental, processing, permitting, taxation, socio-economic and other factors.

Mineral resource categories can be based on an estimate of uncertainty within a theoretical measure of confidence. The thresholds for the uncertainty and confidence are based on rules of thumb; however, they can vary from project to project depending upon the risk tolerance that the Project and the company is willing to bear. Indicated resources may be estimated so the uncertainty of yearly production is approximately ±15% with 90% confidence and measured resources may be estimated so the uncertainty of quarterly production is no greater than ±15% with 90% confidence. The results presented above indicate the reliability is around ±15% for the assumed production rate at roughly 50 m spacing.

It should also be noted that the confidence limits only consider the variability of grade within the deposit. There are other aspects of deposit geology and geometry such as geological contacts or the presence of faults or offsetting structures that may impact the drill spacing.

The spacing distances are intended to define contiguous volumes and they should allow for some irregularities due to actual drill hole placement. The final classification volume results typically must be adjusted manually to come to a coherent classification scheme. The thresholds should be used as a guide and boundaries interpreted and defined to ensure continuity.

Drill hole spacing is sufficient for preliminary geostatistical analysis and evaluating spatial grade variability. The classification of resources was based primarily upon distance to the nearest composite; however, the multiple quantitative measures, as listed below, were inspected and taken into consideration.

The estimated blocks were classified according to the following:

- confidence in interpretation of the mineralized zones
- number of composites used to estimate a block
- number of composites allowed per drill hole
- distance to nearest composite used to estimate a block
- average distance to the composites used to estimate a block.

Therefore, the following lists the spacing for each resource category to classify the resources assuming the current rate of metal production:

- Measured: Note that based on the CIM definitions, continuity must be demonstrated in the designation of measured (and indicated) resources. Therefore, measured resources were delineated from at least three drill holes spaced on a nominal 25 m pattern.
- Indicated: Resources in this category would be delineated from at least three drill holes spaced on a nominal 50 m pattern.
- Inferred: Any material not falling in the categories above and within a maximum 100 m of one hole.

To ensure continuity, the boundary between the indicated and inferred categories was contoured and smoothed, eliminating outliers and orphan blocks. The spacing distances are intended to define contiguous volumes and they should allow for some irregularities due to actual drill hole placement. The final classification volume results typically must be adjusted manually to come to a coherent classification scheme.

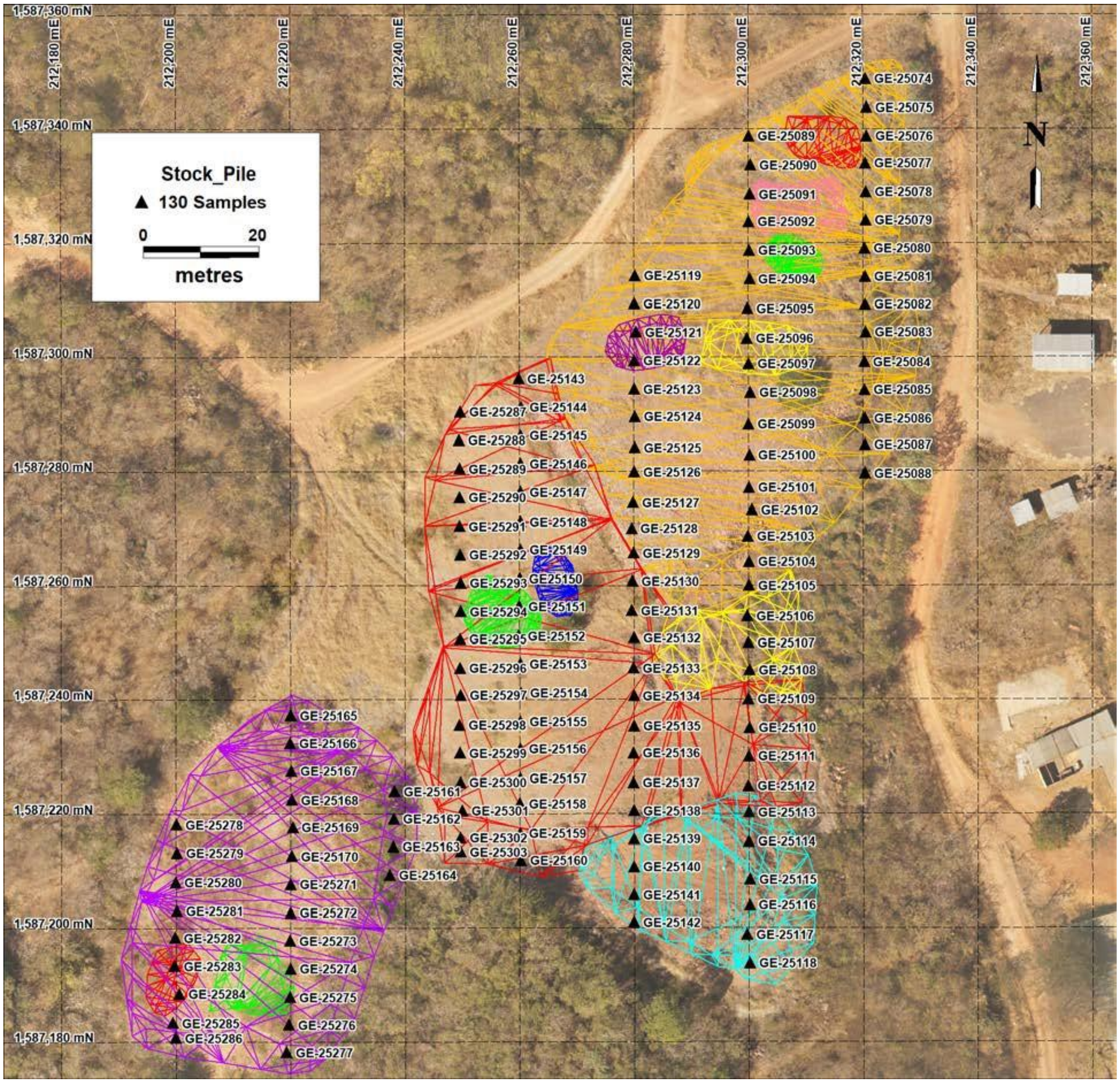
Mineral resources are classified under the categories of “measured”, “indicated” and “inferred” according to CIM guidelines. Mineral resource classification for gold was based primarily on drill hole spacing and on continuity of mineralization. Measured resources were defined as blocks within a distance to nearest composite of 25 m. Indicated resources were defined as those within a distance to three drill holes of less than ~50 m. Inferred resources were defined as those with an average drill hole spacing of less than ~100 m and meeting additional requirements. All resources are constrained in the following manner: primarily, by the continuous vein solids, secondarily, the low-grade envelope, and thirdly, by the Salinas group tertiary member. Blocks outside the aforementioned were estimated in a last pass to determine waste grade and volumes. Final resource classification shells were manually constructed on plan and sections.

The suggested classification parameters are roughly consistent with the past classification scheme. Classification in future models may differ, but principal differences should be due to changes in the amount of drilling.

## 14.12 Stockpile Resources

Mineralized material from mining activities undertaken to date at Cerro Blanco, including ramp development and access, has been stockpiled on site and segregated for future processing. This material may be considered for inclusion within the initial years of mine production or within the ramp-up phase. However, this requires an accurate representation of the volumes and grades so a comprehensive sampling program was designed and implemented. The stockpile surfaces were surveyed to accurately determine volumes and the sampling program entailed excavating trenches on 20 m grid lines and 2 m sample intervals as shown in Figure 14-25.

**Figure 14-25: Plan View of Stockpile, Sample Locations & Domain Solids**




Source: Kirkham (2019).

Correlograms for gold and silver were created and employed to estimate the stockpile resources using ordinary kriging. The estimate was validated using nearest neighbour and inverse distance methods, illustrating good agreement of results.

Table 14.21 shows the volume and tonnage based on an unconsolidated specific gravity of 2.0 gm/mm$^3$ along with gold and silver grades and metal content. These resources are classified as measured.

**Table 14.21: Stockpile Resource Estimate (Measured Resource)**

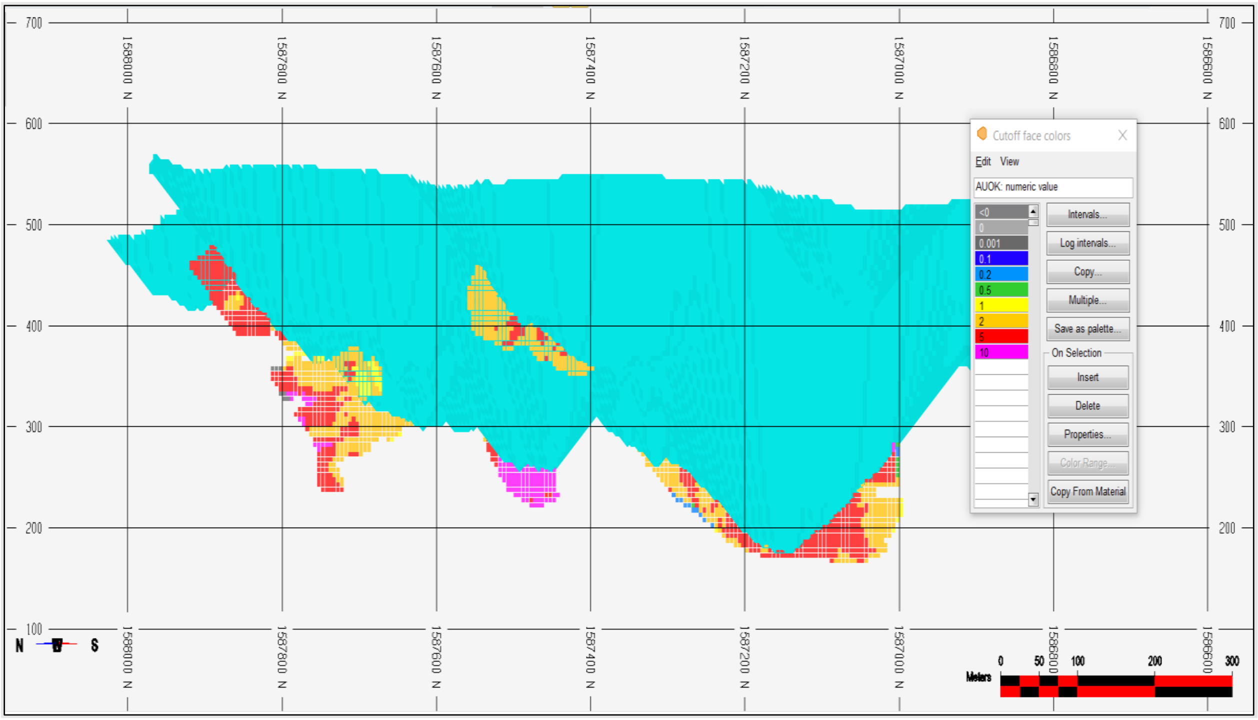
| Volume(BCM) | Mine(t) | Au (g/t) | Ag (g/t) | Au (oz) | Ag (oz) |
|---|---|---|---|---|---|
| 14,863 | 29,726 | 5.35 | 22.59 | 5,108 | 21,590 |

Source: Kirkham (2021).

## 14.13 Underground Resources

Mineralized material that is remaining within the high-grade veins and below the reasonable prospects pit remain as potential for limited underground mining methods. Figure 14-26 illustrates the location and the continuity of these resources which were delineated by approximate mining shapes above a reasonable cut-off grade and leaving a sufficient crown pillar, if necessary.

**Figure 14-26: Section View of Potentially Underground Mineable Resources**




Source: Bluestone (2021).

Table 14.22 shows the indicated underground mineable resources at Cerro Blanco at a cut-off grade of 3.5 g/t Au.

**Table 14.22: Underground Mineable Resources**

| Resource Category | Tonnes (kt) | Au Grade (g/t) | Ag Grade (g/t) | Contained Gold (koz) | Contained Silver (koz) |
|---|---|---|---|---|---|
| Indicated | 189 | 5.7 | 13.4 | 35 | 82 |

Notes: The mineral resource statement is subject to the following: (1) All mineral resources have been estimated in accordance with Canadian Institute of Mining and Metallurgy and Petroleum (CIM) definitions, as required under National Instrument 43-101 (N.I. 43-101), with an effective date of December 31, 2020. (2) Mineral resources reported demonstrate reasonable prospect of eventual economic extraction, as required under N.I. 43-101; mineral resources are not mineral reserves and do not have demonstrated economic viability. (3) Underground mineral resources are reported at a cut-off grade of 3.5 g Au/t. Cut-off grades are based on a price of US$1,600/oz gold, US$20/oz silver and a number of operating cost and recovery assumptions, plus a contingency. (4) Numbers are rounded. (5) The mineral resources may be affected by subsequent assessment of mining, environmental, processing, permitting, taxation, socio-economic and other factors. (6) An inferred mineral resource has a lower level of confidence than that applying to an indicated mineral resource and must not be converted to a mineral reserve. It is reasonably expected that the majority of inferred mineral resources could be upgraded to indicated mineral resources with continued exploration. Source: Kirkham (2021).

## 14.14 Mineral Resource Estimate

This estimate is based upon the reasonable prospect of eventual economic extraction based on continuity an optimized pit, using estimates of operating costs and price assumptions. The "reasonable prospects for eventual economic extraction" were tested using floating cone pit shells based on reasonable prospects of eventual economic assumptions, as shown in Table 14.23. Figure 14-27 illustrates the gold block model along with the "reasonable prospects of eventual economic extraction" pit. The pit optimization results are used solely for testing the "reasonable prospects for eventual economic extraction" and do not represent an attempt to estimate mineral reserves.

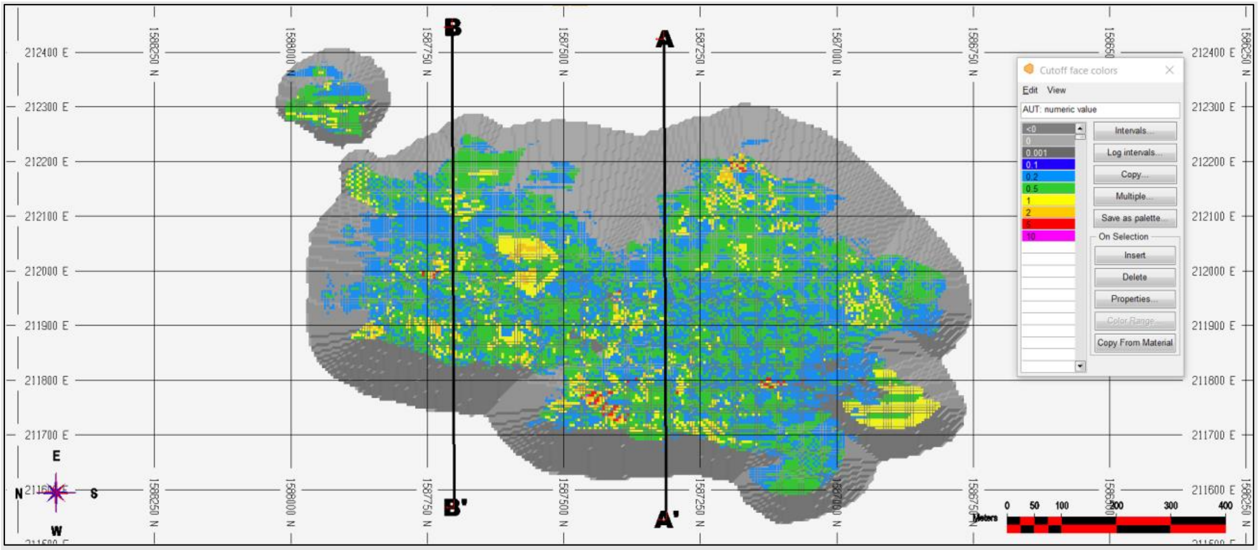
Table 14.24 shows tonnage and grade in the Cerro Blanco Deposit and includes all domains at a 0.4 g Au/t cut-off grade.

**Table 14.23: Parameters Used for Pit Optimization**

| Parameter | Unit | Resource |
|---|---|---|
| **Revenue, Smelting & Refining** | | |
| Gold Price | US$/oz Au | 1,600 |
| Exchange Rate | USD:CAD | 0.77 |
| Payable Metal | % | 100.00 |
| TC/RC/Transport | US$/oz Au | 6.50 |
| Royalty | US$/oz Au | 0.00 |
| Net Gold Value per Gram | US$/g | 49.51 |
| **Operating Cost Estimates** | | |
| Open Pit Waste Mining Cost | US$/t waste mined | 2.00 |
| Open Pit Ore Mining Cost | US$/t ore mined | 2.00 |
| Strip Ratio (estimated) | Waste:Ore | 1.8 |
| Open Pit Mining Cost | US$/t processed | 7.36 |
| Process Cost | US$/t processed | 10.50 |
| G&A | US$/t processed | 1.00 |
| **Total Operating Cost (excluding mining)** | US$/t processed | 14.10 |
| **Total Operating Cost (including mining)** | US$/t processed | 21.70 |
| **Recovery & Dilution** | | |
| External Mining Dilution | % | 0 |
| Mining Recovery | % | 100 |
| **Gold Recovery** | | |
| Gold Recovery | % | 80.00 |
| **Cut-off Grade Calculations** | | |
| **External/Mine Cut-off (incl. mining)** | | |
| Gold Cut-off Grade | g Au/t | 0.3 |
| **Internal/Mill Cut-off (excl. mining)** | | |
| **Gold Cut-off Grade** | **g Au/t** | **0.25** |
| Overall Pit Slope Angles | degrees | 45 |
| Discount Rate | % | 5 |
| Process Production Rate | t/d | 20,000 |
| Process Production Rate | t/a | 7,300,000 |

Source: Kirkham (2021).

**Figure 14-27: Plan View of Gold Block Model with Reasonable Prospects Optimized Pit**




Source: Kirkham (2021).

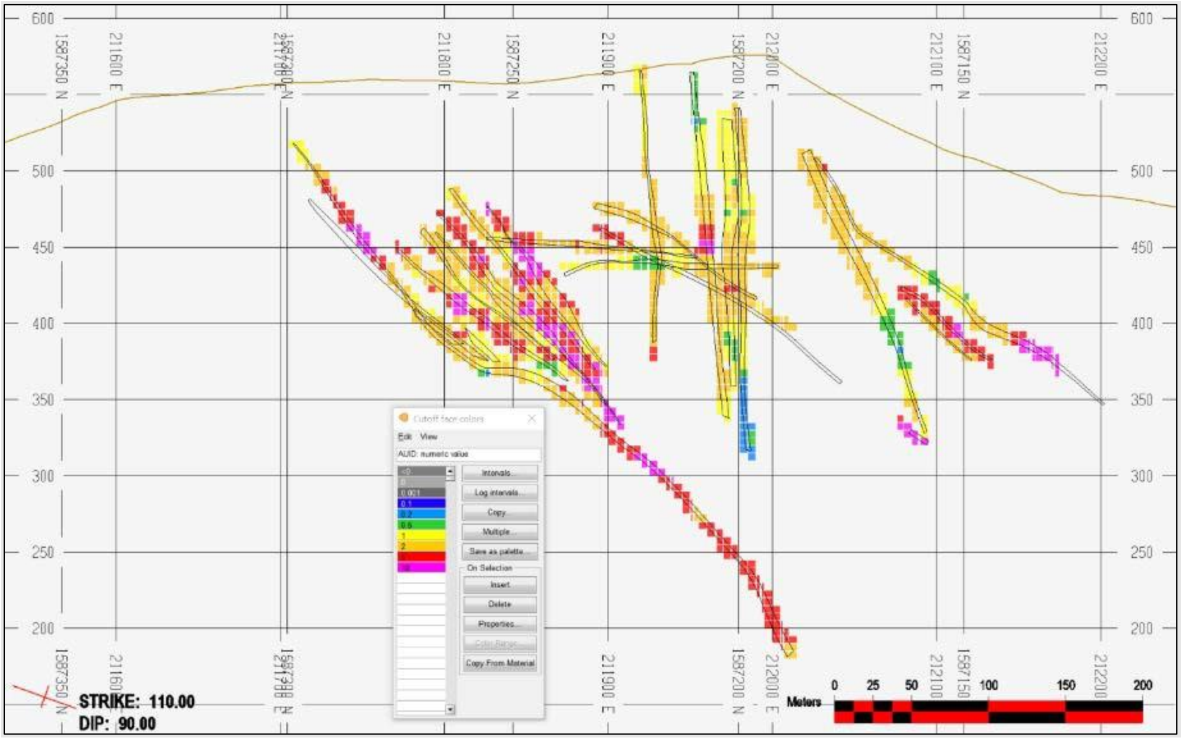
**Table 14.24: Resource Estimate using 0.4 g Au/t Cut-off**

| Resource Category | Tonnes (kt) | Au Grade (g/t) | Ag Grade (g/t) | Contained Gold (koz) | Contained Silver (koz) |
|---|---|---|---|---|---|
| Measured | 40,947 | 1.8 | 7.9 | 2,382 | 10,387 |
| Indicated | 22,595 | 1.0 | 4.2 | 706 | 3,058 |
| **Measured & Indicated** | **63,542** | **1.5** | **6.6** | **3,089** | **13,445** |
| Inferred | 1,672 | 0.6 | 2.1 | 31 | 112 |
| | | | | | |
| **Below Pit (Indicated) *** | **189** | **5.7** | **13.4** | **35** | **82** |
| **Stockpile (Measured)** | **30** | **5.4** | **22.6** | **5** | **22** |

Notes: The mineral resource statement is subject to the following: (1) All mineral resources have been estimated in accordance with Canadian Institute of Mining and Metallurgy and Petroleum (CIM) definitions, as required under National Instrument 43-101 (N.I. 43-101), with an effective date of December 31, 2020. (2) Mineral resources reported demonstrate reasonable prospect of eventual economic extraction, as required under N.I. 43-101; mineral resources are not mineral reserves and do not have demonstrated economic viability. (3) *Underground mineral resources are reported at a cut-off grade of 3.5 g Au/t. Cut-off grades are based on a price of US$1,600/oz gold, US$20/oz silver and a number of operating cost and recovery assumptions, plus a contingency. (4) Numbers are rounded. (5) The mineral resources may be affected by subsequent assessment of mining, environmental, processing, permitting, taxation, socio-economic and other factors. (6) An inferred mineral resource has a lower level of confidence than that applying to an indicated mineral resource and must not be converted to a mineral reserve. It is reasonably expected that the majority of inferred mineral resources could be upgraded to indicated mineral resources with continued exploration. (7) Mineral Resources are inclusive of mineral reserves. Source: Kirkham (2021).

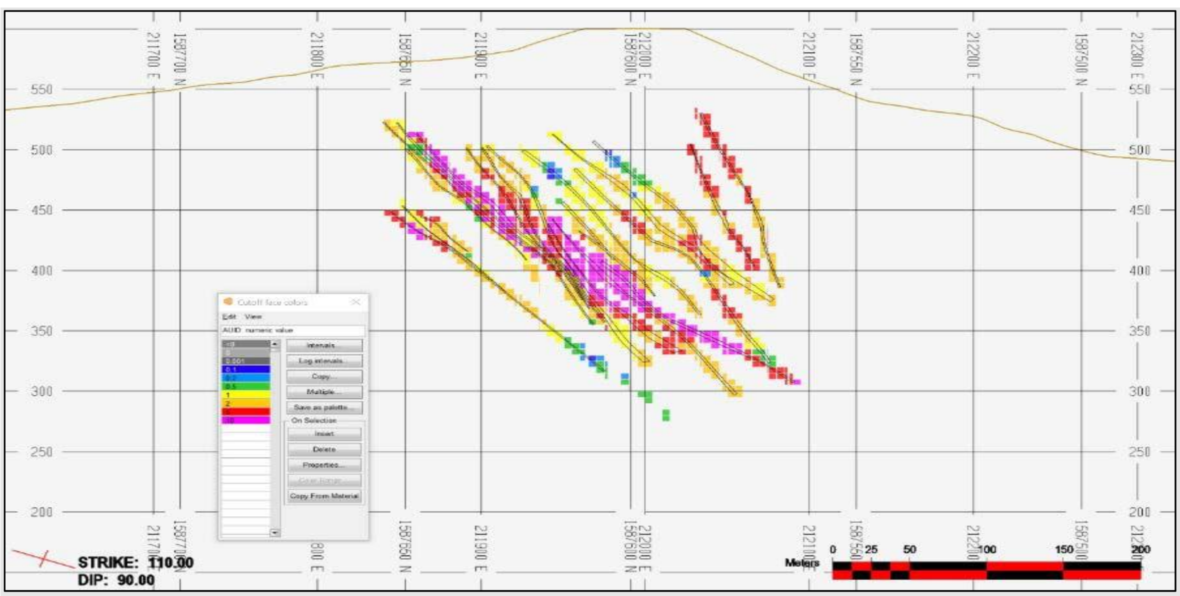
Figure 14-28 through Figure 14-31 show sectional views of the high-grade veins for gold and silver in the north and south, respectively. Figure 14-32 through Figure 14-35 show sectional views of the total block model with the high-grade vein and low-grade host rock components resulting in the whole block grade for gold and silver in the north and south, respectively.

**Figure 14-28: A-A' Section View of Au South Zone Veins**



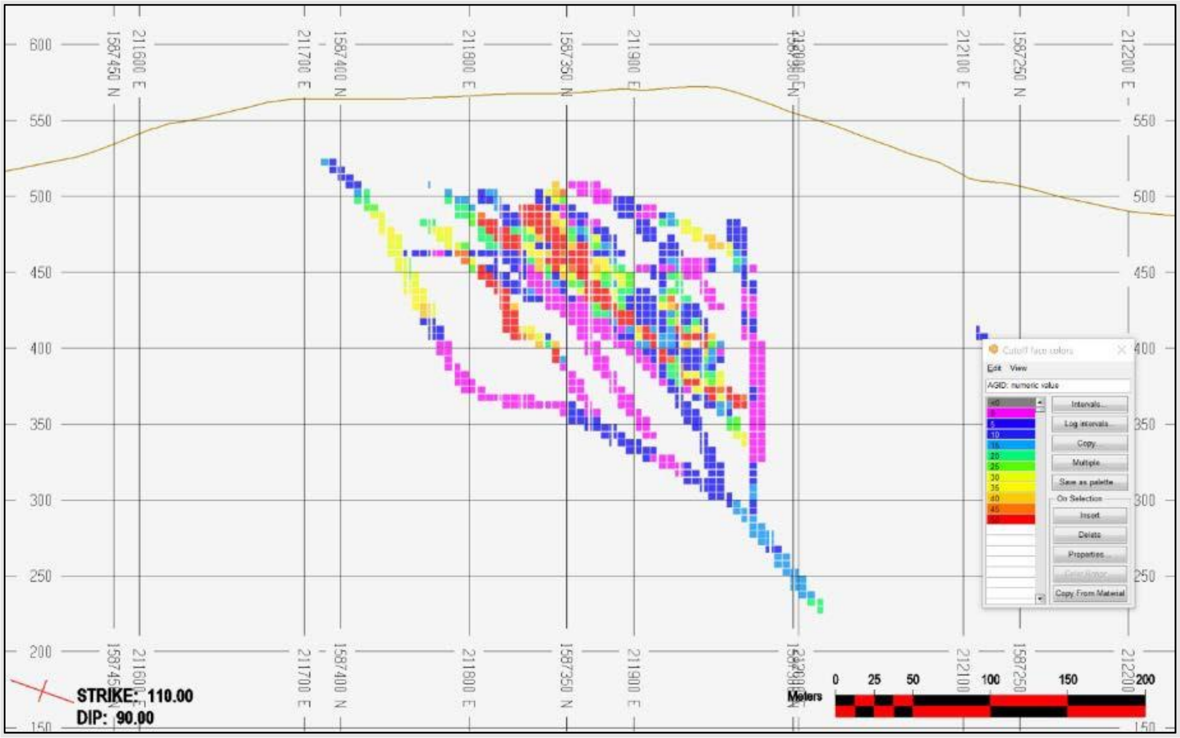

Source: Kirkham (2021).

**Figure 14-29: B-B' Section View of Au Block Model North Zone Veins**



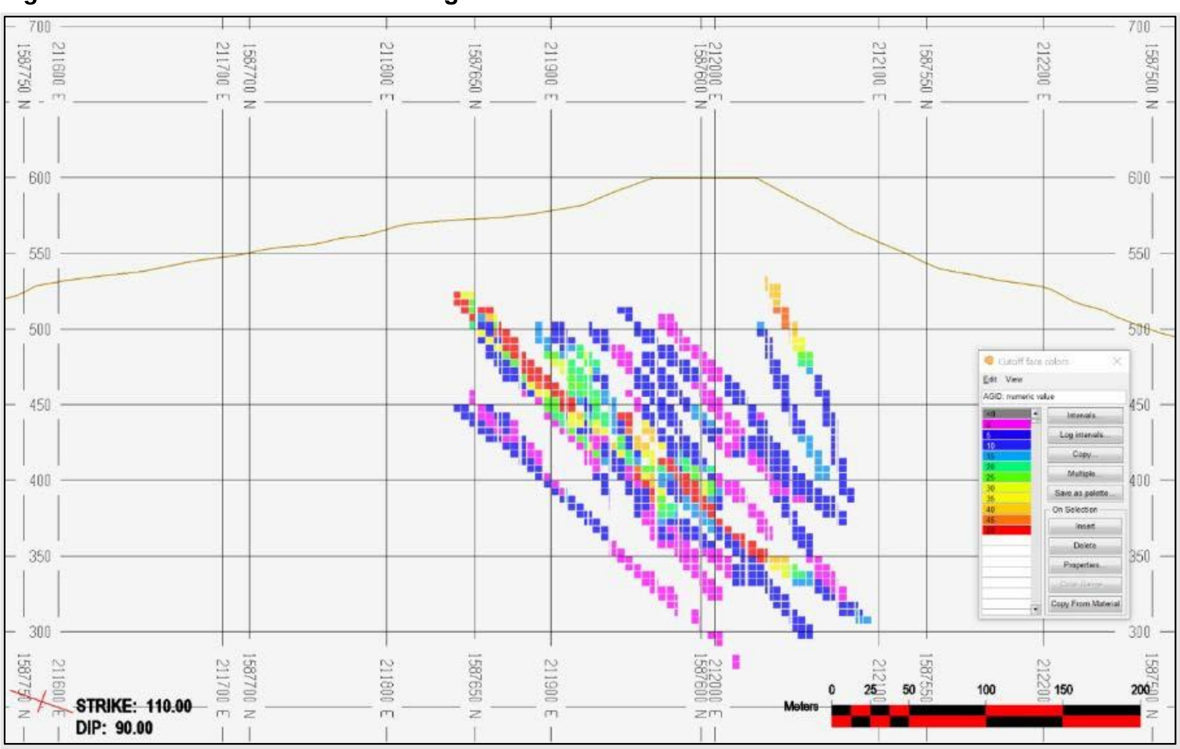

Source: Kirkham (2021).

**Figure 14-30: A-A' Section View of Ag Block Model South Zone Veins**



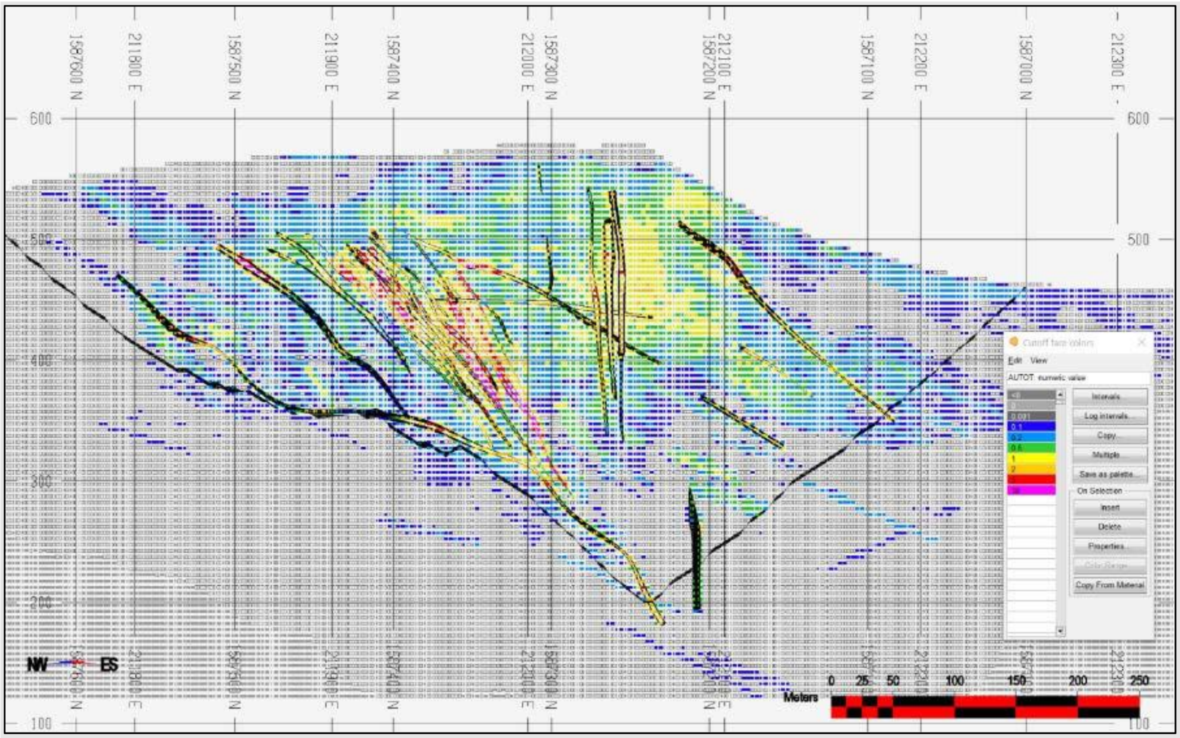

Source: Kirkham (2021).

**Figure 14-31: B-B' Section View of Ag Block Model North Zone Veins**



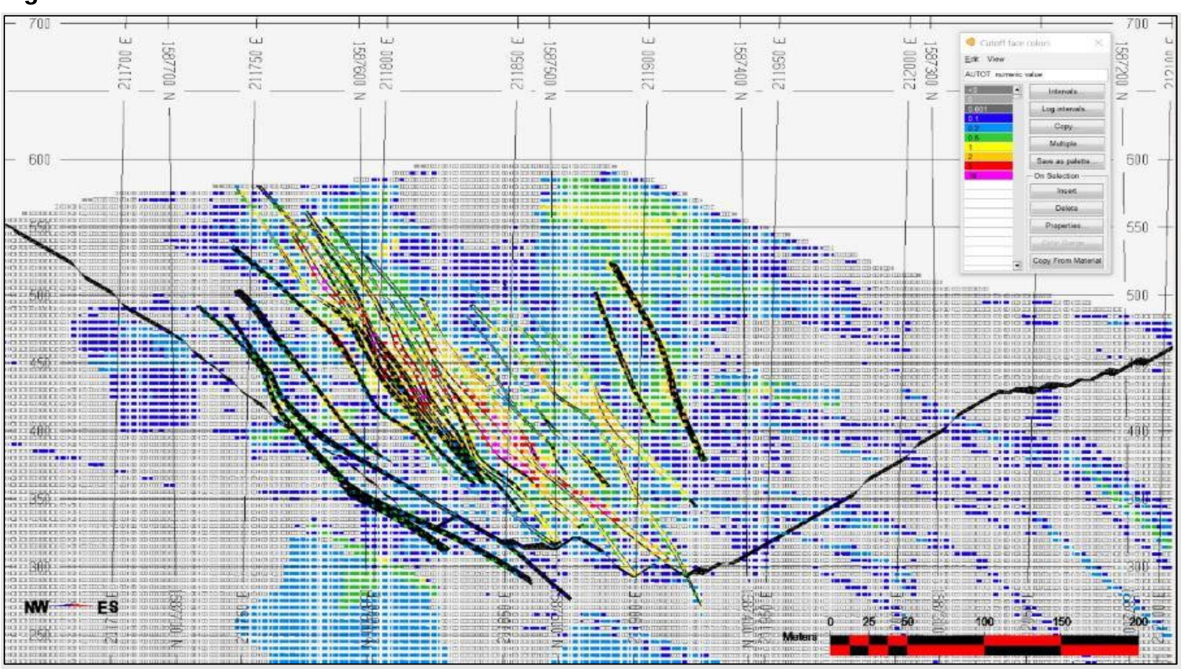

Source: Kirkham (2021).

**Figure 14-32: A-A' Section View of Au Block Model South**



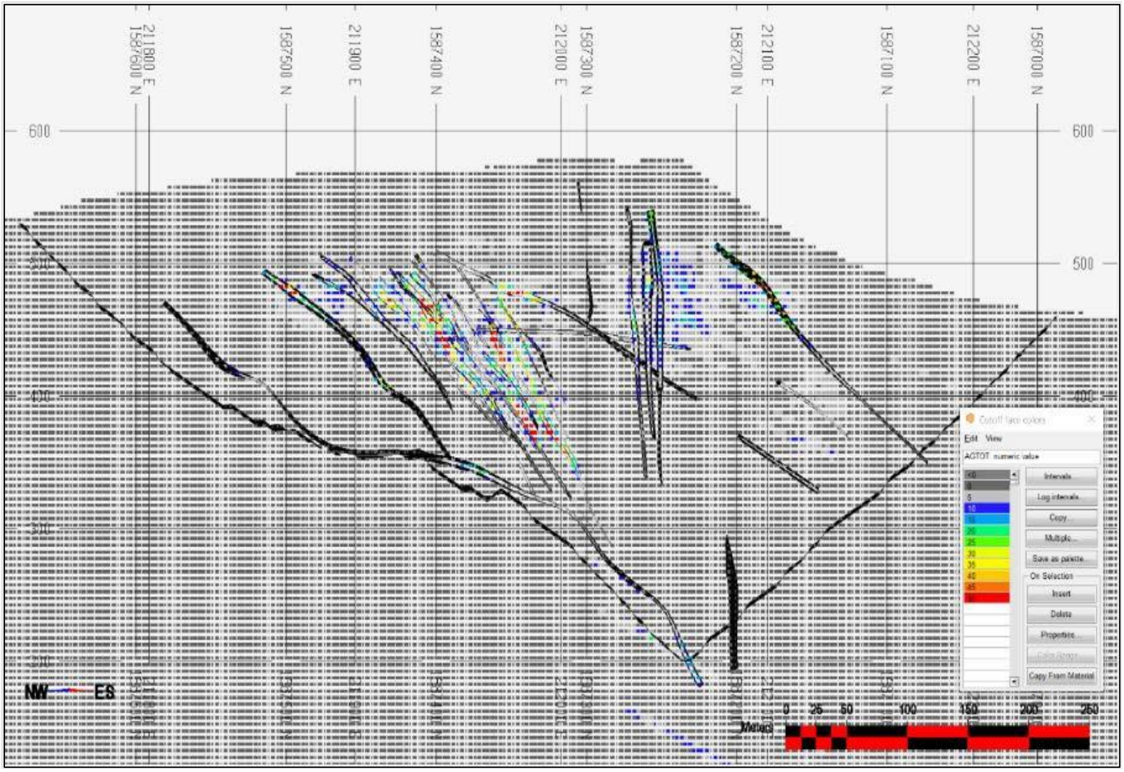
Source: Kirkham (2021).

**Figure 14-33: B-B' Section View of Au Block Model North**



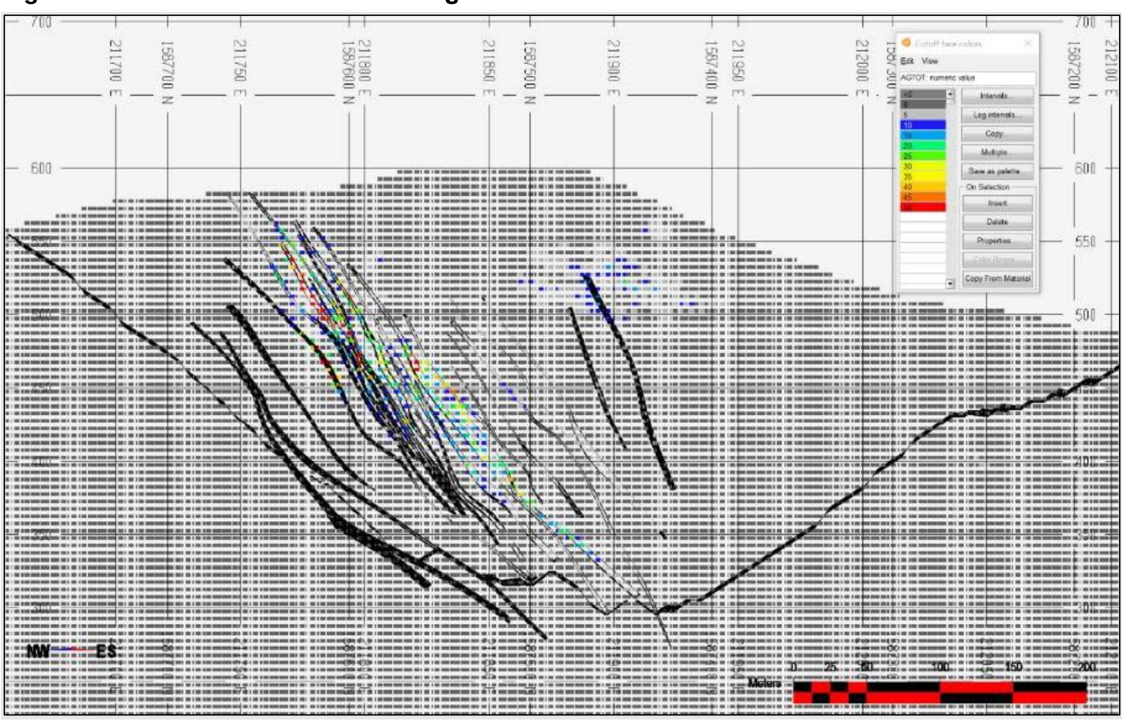
Source: Kirkham (2021).

**Figure 14-34: A-A' Section View of Ag Block Model North**



Source: Kirkham (2021).

**Figure 14-35: B-B' Section View of Ag Block Model South**



Source: Kirkham (2021).

## 14.15 Sensitivity of the Block Model to Selection Cut-off Grade

The mineral resources are sensitive to the selection of cut-off grade. Table 14.25 shows tonnage and grade in the Cerro Blanco deposit at different gold cut-off grades.

The reader is cautioned that these values should not be misconstrued as a mineral reserve. The reported quantities and grades are only presented as a sensitivity of the resource model to the selection of cut-off grade.

**Table 14.25: Sensitivity Analyses of Tonnage along with Au & Ag Grades at Various Au Cut-off Grades**

| Resource Category | Cut-off Grade | Tonnes (kt) | Au Grade (g/t) | Ag Grade (g/t) | Contained Gold (koz) | Contained Silver (koz) |
|---|---|---|---|---|---|---|
| Measured | 0.25 | 54,479 | 1.44 | 6.5 | 2,520 | 11,350 |
| | 0.3 | 49,165 | 1.57 | 7 | 2,474 | 11,002 |
| | 0.35 | 44,704 | 1.69 | 7.4 | 2,427 | 10,679 |
| | **0.4** | **40,947** | **1.81** | **7.9** | **2,382** | **10,387** |
| | 0.5 | 34,981 | 2.04 | 8.8 | 2,296 | 9,841 |
| | 0.6 | 30,379 | 2.27 | 9.6 | 2,215 | 9,347 |
| Indicated | 0.25 | 41,171 | 0.68 | 3.4 | 894 | 4,448 |
| | 0.3 | 33,046 | 0.77 | 3.7 | 822 | 3,889 |
| | 0.35 | 27,230 | 0.87 | 3.9 | 762 | 3,441 |
| | **0.4** | **22,595** | **0.97** | **4.2** | **706** | **3,058** |
| | 0.5 | 16,230 | 1.18 | 4.7 | 615 | 2,452 |
| | 0.6 | 12,089 | 1.4 | 5.2 | 542 | 2,013 |
| Inferred | 0.25 | 2,816 | 0.48 | 2.1 | 43 | 187 |
| | 0.3 | 2,449 | 0.51 | 2 | 40 | 154 |
| | 0.35 | 2,068 | 0.54 | 2 | 36 | 132 |
| | **0.4** | **1,672** | **0.58** | **2.1** | **31** | **112** |
| | 0.5 | 1,087 | 0.65 | 2.1 | 23 | 73 |
| | 0.6 | 359 | 0.85 | 2.5 | 10 | 29 |

Notes: The mineral resource statement is subject to the following: (1) All mineral resources have been estimated in accordance with Canadian Institute of Mining and Metallurgy and Petroleum (CIM) definitions, as required under National Instrument 43-101 (N.I. 43-101), with an effective date of December 31, 2020. (2) Mineral resources reported demonstrate reasonable prospect of eventual economic extraction, as required under N.I. 43-101; mineral resources are not mineral reserves and do not have demonstrated economic viability. (3) Cut-off grades are based on a price of US$1,600/oz gold, US$20/oz silver and a number of operating cost and recovery assumptions, plus a contingency. (4) Numbers arerounded. (5) The mineral resources may be affected by subsequent assessment of mining, environmental, processing, permitting, taxation, socio-economic and other factors. (6) An inferred mineral resource has a lower level of confidence than that applying to an indicated mineral resource and must not be converted to a mineral reserve. It is reasonably expected that the majority of inferred mineral resources could be upgraded to indicated mineral resources with continued exploration. 7. Mineral Resources are inclusive of mineral reserves. Source: Kirkham (2021).

## 14.16 Resource Validation

A graphical validation was done on the block model. The purpose of this graphical validation is to:

- check the reasonableness of the estimated grades, based on the estimation plan and the nearby composites.
- check the general drift and the local grade trends, compared to the drift and local grade trends of the composites.
- ensure that all blocks in the core of the deposit have been estimated
- check that topography has been properly accounted for
- check against partial model to determine reasonableness
- check against manual approximate estimates of tonnage to determine reasonableness
- inspect and explain potentially high-grade block estimates in the neighbourhood of extremely high assays.

A full set of cross-sections, long sections and plans were used to check the block model on the computer screen, showing the block grades and the composites. No evidence of any block being wrongly estimated was found; it appears that every block grade could be explained as a function of the surrounding composites and the estimation plan applied.

These validation techniques included the following:

- visual inspections on a section-by-section and plan-by-plan basis
- the use of grade-tonnage curves
- swath plots comparing kriged estimated block grades with inverse distance and nearest neighbour estimates
- an inspection of histograms of distance of the first composite to the nearest block, and the average distance to blocks for all composites used, which gives a quantitative measure of confidence that blocks are adequately informed in addition to assisting in the classification of resources
- validation of the block models by estimating the resources within the vein domains using partial block where the vein solids were coded as a percentage within the blocks.

## 14.17 Discussion with Respect to Potential Material Risks to the Resources

There are no known environmental, permitting, legal, taxation, title, socio-economic, political or other relevant factors that materially affect the mineral resources.

# 15. MINERAL RESERVE ESTIMATES

A mineral reserve is the economically mineable part of a measured and/or indicated mineral resource demonstrated by at least a Preliminary Feasibility Study (CIM, 2010). This FS contains adequate information on mining, processing, metallurgical, economic, and other relevant factors that demonstrate, at the time of reporting, that economic extraction is justified.

Mineral reserves are those parts of mineral resources, which, after the application of all mining factors, result in an estimated tonnage and grade that form the basis of an economically viable project. Mineral Reserves are inclusive of diluting material that will be mined in conjunction with the economic mineralized rock and delivered to the process plant or equivalent facility. The term “mineral reserve” need not necessarily signify that extraction facilities are in place, or operative, or that all governmental approvals have been received. It does signify that there are reasonable expectations of such approvals.

Mineral reserves are subdivided in order of increasing confidence into probable mineral reserves and proven mineral reserves; a probable mineral reserve has a lower level of confidence than a proven mineral reserve. The reserve classifications used in this report conform to the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) classification of NI 43-101 resource and reserve definitions and Companion Policy 43-101CP. These are listed below:

A “proven mineral reserve” is the economically mineable part of a measured mineral resource demonstrated by at least a PFS. This FS must include adequate information on mining, processing, metallurgical, economic, and other relevant factors that demonstrate, at the time of reporting, that economic extraction is justified. Application of the proven mineral reserve category implies that the QP has the highest degree of confidence in the estimate with the consequent expectation in the minds of the readers of the report. The term should be restricted to that part of the deposit where production planning is taking place and for which any variation in the estimate would not significantly affect potential economic viability.

A “probable mineral reserve” is the economically mineable part of an indicated mineral resource, and in some circumstances a measured mineral resource, demonstrated by at least a PFS. The study must include adequate information on mining, processing, metallurgical, economic, and other relevant factors that demonstrate, at the time of reporting, that economic extraction can be justified.

## 15.1 Summary

Mineral reserves for the Cerro Blanco Gold Project are estimated at 53.9 Mt at an average grade of 1.64 g/t of gold for 2,846,000 ozs, and 7.27 g/t of silver for 12,602,000 ozs, as summarized in Table 15.1. The Mineral Reserve Estimate (MRE) was prepared by G Mining Services Inc. (GMS).

**Table 15.1: Cerro Blanco Gold Project Open Pit Mineral Reserve Estimate**

| Reserve Category | Tonnage (kt) | Gold (g/t) | Gold (koz) | Silver (g/t) | Silver (koz) |
|---|---|---|---|---|---|
| Proven | 37,618 | 1.89 | 2,286 | 8.33 | 10,084 |
| Probable | 16,279 | 1.07 | 560 | 4.81 | 2,518 |
| **Proven & Probable** | **53,896** | **1.64** | **2,846** | **7.27** | **12,602** |

Notes: (1) CIM definitions were followed for mineral reserves. (2) Effective date of the estimate is Nov 1, 2021. (3) Mineral reserves are estimated at a cut-off grade of 0.50 g/t Au Eq. (4) Mineral Reserves are estimated using the following long-term metal prices (Au = US$1,550/oz and Ag = US$20/oz). (5) Bulk density of ore is variable but averages 2.70 t/m$^3$. (6) The average strip ratio is 2.7:1. (7) The average mining dilution factor is 6.7%. (8) Other costs and factors used for gold cut-off grade determination were process, G&A, and other costs of $21.17/tonne, a royalty of $31.60 /oz Au and gold and silver metallurgical recoveries of 91% and 85% respectively. (9) Tonnages are rounded to the nearest 1,000 tonnes; metal grades are rounded to two decimal places. 7) Tonnage and grade measurements are in metric units; contained gold and silver are reported as thousands of troy ounces. (10) The mineral reserves may be affected by subsequent assessment of mining, environmental, processing, permitting, taxation and socio-economic factors. (11) Mineral resources are inclusive of mineral reserves.

The mine design and MRE were completed to a level appropriate for feasibility studies. The MRE stated herein is consistent with CIM definitions and is suitable for public reporting. As such, the mineral reserves are based on measured and indicated mineral resources, and do not include any inferred mineral resources. The inferred mineral resources contained within the mine design are classified as not having economic value.

## 15.2 Resource Block Model

The resource block model was completed by Kirkham Geosystems Ltd comprising the estimation of a high-grade vein component and low-grade mineralized host rock component of each block, with final block grades determined by a weighted average calculation. For mine planning purposes, GMS used the percent block model with the standard SMU block size of 5 m x 5 m x 5 m.

## 15.3 Pit Optimization

The open pit optimization was conducted in GEOVIA Whittle™ to determine the optimal economic shape of the open pit to guide the pit design process. This task was undertaken using the Whittle software which utilizes the Lerchs-Grossmann algorithm. The method works on the resource block model of the ore body and progressively constructs lists of related blocks that are designed as ore and waste. The method uses the values of the blocks to define a pit outline that has the highest possible total economic value, subject to the required pit slopes defined as structure arcs in the software. This section describes the parameters used to calculate block values in Whittle™.

### 15.3.1 Pit Slope Geotechnical Assessment

E-Mining Technology produced a pit slope design study to support the FS mine designs. The conclusions of this study have been used as an input to the pit optimization and design process.

The pit area was divided into sectors based on the data collected from orientated and geotechnical drill hole core. Generally, the pit area is controlled by geological domains and bench geometry. The Salinas and the Mita domains have different parameters with variable bench geometry. Twelve geotechnical zones were identified.

The open pit will be developed as two individual domains of different rock mass quality. The Salinas domain has lower geotechnical than the Mita domain, the latter presenting a rock mass of generally good a regular geotechnical quality. The rock mass characteristics for each domain are as follows:

- Salinas: GSI range 30 - 54, UCS range 65 – 265 MPa and average Young's modulus of 57 GPa.
- Mita: GSI range 47 - 66, UCS range 49 - 93 and Young's modulus range 22 – 51 GPa.

E-Mining Technology identified six design sectors based on the distribution of geomechanical domains. Slope analyses were undertaken on each sector to establish achievable slope configurations; the results are shown in Figure 15-1.

Eight dewatering wells will be required to control the quantity of groundwater pit inflows, which will also contribute to the in-pit dewatering that will operate during the life of the mine. Phreatic surface water developing behind pit walls will be monitored and depressurized as required.

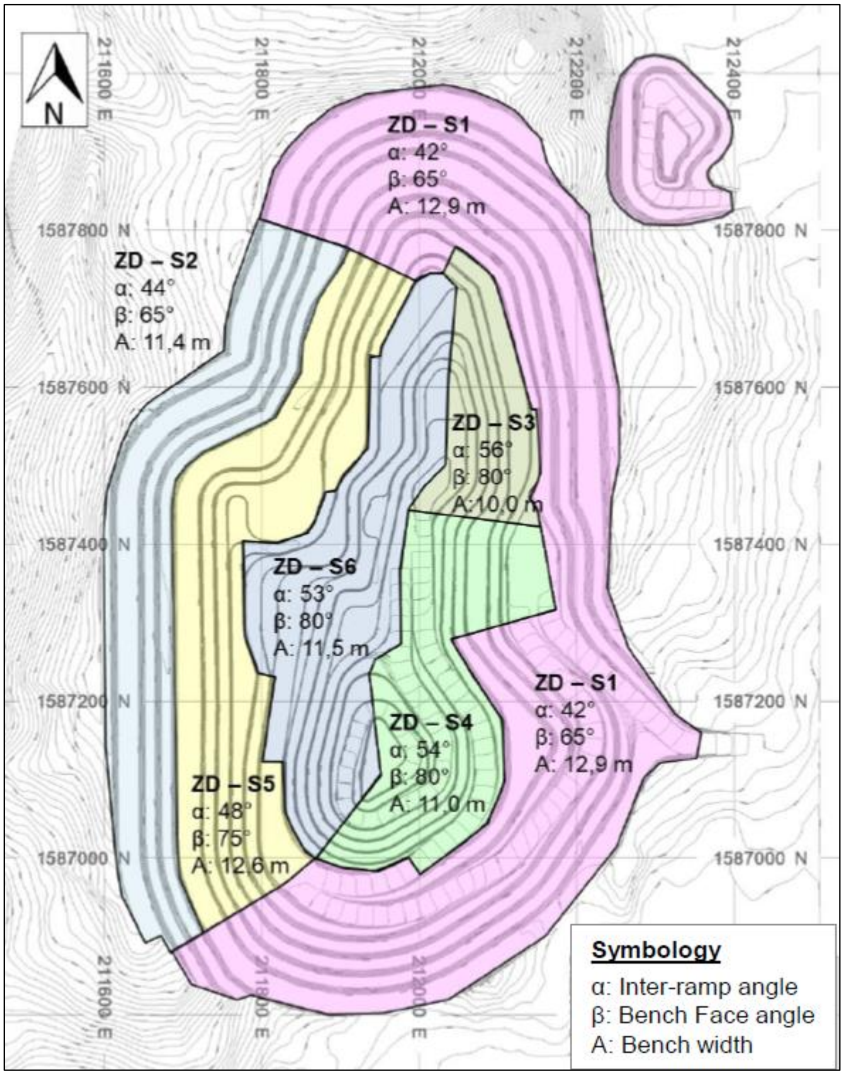
A slope monitoring program is recommended for all stages of pit development. The program should include geotechnical and tension crack mapping, and a surface displacement monitoring program using surface prisms.

Slope configuration options are presented in Table 15.2. Double benching will be required with pre-split, no sub-grade drilling and well-controlled blasting practices.

The final pit was designed using a double benching configuration to a final height of 20 m. The pit slope profile is based on recommendations by E-Mining Technology as presented in Table 15.2.

E-Mining Technology did not consider the overburden in the domain definition process and analysis because it is expected to form only a minor part of the proposed pit slopes. It is suggested during future site investigations to characterize the overburden along the crest of the proposed pit to evaluate the stability of the upper part of the pit slopes.

**Figure 15-1: E-Mining Technology Geotechnical Recommendations**




Source: E-mining Technology (2021).

**Table 15.2: Cerro Blanco Final Wall Geotechnical Recommendations**

| Slope Parameter | Values |
|---|---|
| Final Bench Height (m) | 20.0 |
| Bench Face Angle (º) | 65 to 80 |
| Design Catch Bench Width (m) | 10.0 to 12.9 |
| Inter-ramp Angle (º) | 42 to 56 |
| Overall Slope Angle (º) | 41 to 53 |
| Geotechnical berms (m) | 14 |

Source: GMS (2021).

### 15.3.2 Mining Dilution & Ore Loss

A mining dilution assessment was made by evaluating the number of contacts for blocks above an economic cut-off grade. The block contacts were then used to estimate a dilution skin around ore blocks to estimate the amount of dilution anticipated. The dilution skin consists of 0.5 m of material in a north-south direction (across strike) and 0.5 m in an east-west direction (along strike). The dilution is therefore specific to the geometry of the orebody and the number of contacts between ore and waste. The orebody consists of both vein and disseminated mineralization.

For each mineralized block, diluted grades and densities were calculated by considering in-situ grades and in-situ densities of the surrounding blocks.

### 15.3.3 Initial Pit Optimization Parameters

Unit reference mining costs are used for a “reference mining block” usually located near the pit crest or surface and are incremented with depth corresponding to additional cycle time and therefore hauling costs. The reference mining cost is estimated at US$2.20/t with an incremental depth factor of US$0.05/t per 10 m bench. The total ore-based cost is estimated at US$23.35/t which includes processing, general and administration costs, sustaining capital, and a closure cost provision as summarized in Table 15.3 below.

The overall slope angles utilized in the Whittle analysis are based on inter-ramp angles recommended by E-Mining Technology with provisions for ramps and geotechnical berms. The overall slope angle in competent rock is 41 to 53 degrees based on a designed inter-ramp angle of 42 to 56 degrees.

**Table 15.3: Ore-Based Cost Assumption**

| Ore-Based Cost Assumptions | US$/t |
|---|---|
| Processing (including Power) | 14.40 |
| General & Administration | 2.85 |
| Site Services & Infrastructures | 3.98 |
| Rehabilitation & Closure | 0.67 |
| Sustaining Capital | 1.45 |
| **Total Ore-Based Cost** | **23.35** |

Source: GMS (2021).

A summary of the initial pit optimization parameters is presented in Table 15.4 for a nominal milling rate of 4 Mt per year based on long-term metal price assumptions. It should be noted that these initial parameters were based on the PEA (dated February 19, 2021.) For the definition of reserves, some parameters were later modified to reflect more realistic costs.

**Table 15.4: Cerro Blanco Gold Project Optimization Parameters**

| Optimization Parameters | Unit | Cerro Blanco |
|---|---|---|
| **Economic Parameters** | | |
| Discount Rate | % | 5.0% |
| Gold Price | US$/oz | 1,550 |
| Silver Price | US$/oz | 20 |
| Transport & Refining Cost | US$/Au oz | 9.36 |
| Royalty Rate | % | 2.05% |
| Royalty Cost - Gold | US$/Au oz | 31.60 |
| Net Gold Value | US$/Au oz | 1,509 |
| **Recovery & Dilution Factors** | | |
| Average Gold Recovery (Equations) | % | 89.7% |
| Average Silver Recovery (Equations) | % | 85.0% |
| Mining Dilution (included in BM) | % | 6.0% |
| Mining Loss (included in BM) | % | 1.0% |
| **Other Ore-Based Costs** | | |
| Total Processing Cost including Power | US$/t milled | 14.40 |
| General & Administration Costs | US$/t milled | 2.85 |
| Site Services & Infrastructure | US$/t milled | 3.98 |
| Rehabilitation & Closure | US$/t milled | 0.67 |
| Sustaining Capital | US$/t milled | 1.45 |
| **Total Ore-Based Cost** | | |
| Total Ore-Based Cost | US$/t milled | **23.35** |
| **Mining** | | |
| Total Mining Reference Cost | US$/t mined | 2.20 |
| Mine Sustaining Capital | US$/t mined | 0.35 |
| Total Mining Reference Cost including Sustaining Capital | US$/t mined | 2.55 |
| Incremental Bench Cost | US$/10 m bench | 0.050 |

Source: GMS (2021).

Some parameters used in the Whittle shell analysis were later modified to reflect updated inputs for the determination of Mineral Reserves. Those changes included increased gold recovery in line with metallurgical testwork and updated FS-Level costs.

### 15.3.4 Open Pit Optimization Results

The Whittle nested shell results are presented in Table 15.5 using measured and indicated mineral resource blocks. The nested shells are generated by using revenue factors to scale up and down from the base case selling price.

The shell selection is presented in Table 15.6. Pit shell 39 was selected as the optimum final pit shell which corresponds to a discounted cash flow @ 5% of US$1,750 million pit shell (revenue factor 0.79). This shell has a total tonnage of 161.4 Mt including 46.7 Mt of ore. This is the smallest shell that achieves close to maximum value using a practical phasing approach. A pit-by-pit graph of the measured and indicated resource is shown in Figure 15-2.

**Table 15.5: Measured & Indicated Mineral Resource Whittle Shell Results for Combined Diluted Model**

| Pit Shell | Best Case Disc. @ 5% (M$) | Specified Disc. @ 5% (M$) | Worst Case Disc. @ 5% (M$) | Total Tonnage (kt) | Waste Tonnage (kt) | Ore Tonnage (kt) | Strip Ratio (W:O) | Au Grade (g/t) | Ag Grade (g/t) |
|---|---|---|---|---|---|---|---|---|---|
| 12 | 1,591 | 1,549 | 1,437 | 90,113 | 56,464 | 33,649 | 1.68 | 1.88 | 8.88 |
| 13 | 1,601 | 1,558 | 1,444 | 91,552 | 57,505 | 34,046 | 1.69 | 1.88 | 8.85 |
| 14 | 1,670 | 1,618 | 1,496 | 103,515 | 66,987 | 36,528 | 1.83 | 1.87 | 8.60 |
| 15 | 1,697 | 1,642 | 1,515 | 108,649 | 71,196 | 37,453 | 1.90 | 1.87 | 8.52 |
| 16 | 1,701 | 1,646 | 1,518 | 109,372 | 71,729 | 37,643 | 1.91 | 1.87 | 8.51 |
| 17 | 1,715 | 1,658 | 1,525 | 112,218 | 74,046 | 38,171 | 1.94 | 1.87 | 8.48 |
| 18 | 1,739 | 1,678 | 1,541 | 117,890 | 78,718 | 39,172 | 2.01 | 1.86 | 8.39 |
| 19 | 1,744 | 1,682 | 1,545 | 119,007 | 79,562 | 39,445 | 2.02 | 1.86 | 8.36 |
| 20 | 1,750 | 1,687 | 1,547 | 120,268 | 80,486 | 39,782 | 2.02 | 1.86 | 8.33 |
| 21 | 1,751 | 1,687 | 1,548 | 120,417 | 80,572 | 39,845 | 2.02 | 1.86 | 8.32 |
| 22 | 1,777 | 1,708 | 1,558 | 127,837 | 86,481 | 41,355 | 2.09 | 1.84 | 8.19 |
| 23 | 1,793 | 1,720 | 1,568 | 132,956 | 90,705 | 42,251 | 2.15 | 1.83 | 8.12 |
| 24 | 1,798 | 1,724 | 1,570 | 135,036 | 92,502 | 42,534 | 2.17 | 1.83 | 8.09 |
| 25 | 1,810 | 1,732 | 1,576 | 139,366 | 96,119 | 43,248 | 2.22 | 1.83 | 8.02 |
| 26 | 1,814 | 1,735 | 1,578 | 141,446 | 97,960 | 43,485 | 2.25 | 1.83 | 8.00 |
| 27 | 1,820 | 1,739 | 1,581 | 143,906 | 99,962 | 43,944 | 2.27 | 1.82 | 7.95 |
| 28 | 1,823 | 1,741 | 1,580 | 145,596 | 101,404 | 44,192 | 2.3 | 1.82 | 7.93 |
| 29 | 1,826 | 1,743 | 1,581 | 146,954 | 102,597 | 44,357 | 2.3 | 1.82 | 7.91 |
| 30 | 1,829 | 1,745 | 1,580 | 148,374 | 103,781 | 44,593 | 2.3 | 1.82 | 7.89 |
| 31 | 1,829 | 1,745 | 1,580 | 148,622 | 103,944 | 44,678 | 2.3 | 1.82 | 7.88 |
| 32 | 1,830 | 1,745 | 1,580 | 148,806 | 104,083 | 44,723 | 2.3 | 1.81 | 7.88 |
| 33 | 1,833 | 1,747 | 1,581 | 150,715 | 105,692 | 45,023 | 2.3 | 1.81 | 7.86 |
| 34 | 1,836 | 1,749 | 1,581 | 153,480 | 108,095 | 45,385 | 2.4 | 1.81 | 7.82 |
| 35 | 1,837 | 1,750 | 1,581 | 154,014 | 108,484 | 45,530 | 2.4 | 1.81 | 7.81 |
| 36 | 1,840 | 1,750 | 1,579 | 155,921 | 110,000 | 45,921 | 2.4 | 1.80 | 7.79 |
| 37 | 1,843 | 1,750 | 1,577 | 158,335 | 112,021 | 46,314 | 2.4 | 1.79 | 7.76 |
| 38 | 1,844 | 1,750 | 1,573 | 160,378 | 113,812 | 46,565 | 2.4 | 1.79 | 7.74 |
| **39** | **1,845** | **1,750** | **1,573** | **161,398** | **114,717** | **46,682** | **2.5** | **1.79** | **7.73** |
| 40 | 1,845 | 1,750 | 1,572 | 162,196 | 115,416 | 46,781 | 2.5 | 1.79 | 7.72 |
| 41 | 1,846 | 1,750 | 1,572 | 162,461 | 115,635 | 46,826 | 2.5 | 1.79 | 7.72 |
| 42 | 1,846 | 1,750 | 1,572 | 163,007 | 116,088 | 46,919 | 2.5 | 1.79 | 7.71 |
| 43 | 1,847 | 1,749 | 1,571 | 164,725 | 117,619 | 47,106 | 2.5 | 1.78 | 7.70 |
| 44 | 1,847 | 1,749 | 1,571 | 165,108 | 117,946 | 47,162 | 2.5 | 1.78 | 7.69 |
| 45 | 1,848 | 1,749 | 1,570 | 166,713 | 119,313 | 47,400 | 2.5 | 1.78 | 7.67 |
| 46 | 1,849 | 1,748 | 1,569 | 168,490 | 120,853 | 47,637 | 2.5 | 1.78 | 7.65 |
| 47 | 1,850 | 1,746 | 1,564 | 172,556 | 124,487 | 48,069 | 2.6 | 1.77 | 7.62 |
| 48 | 1,850 | 1,746 | 1,563 | 174,811 | 126,575 | 48,236 | 2.6 | 1.77 | 7.60 |
| 49 | 1,850 | 1,746 | 1,561 | 175,609 | 127,290 | 48,319 | 2.6 | 1.77 | 7.59 |
| 50 | 1,850 | 1,745 | 1,561 | 176,056 | 127,666 | 48,390 | 2.6 | 1.77 | 7.59 |
| 51 | 1,850 | 1,745 | 1,560 | 176,504 | 128,003 | 48,501 | 2.6 | 1.77 | 7.58 |
| 52 | 1,850 | 1,745 | 1,560 | 176,793 | 128,260 | 48,532 | 2.6 | 1.77 | 7.58 |
| 53 | 1,850 | 1,744 | 1,559 | 177,153 | 128,555 | 48,598 | 2.6 | 1.76 | 7.57 |
| 54 | 1,850 | 1,743 | 1,557 | 179,208 | 130,469 | 48,739 | 2.7 | 1.76 | 7.56 |
| 55 | 1,850 | 1,742 | 1,555 | 180,047 | 131,175 | 48,873 | 2.7 | 1.76 | 7.55 |
| 56 | 1,850 | 1,742 | 1,554 | 180,629 | 131,713 | 48,916 | 2.7 | 1.76 | 7.54 |

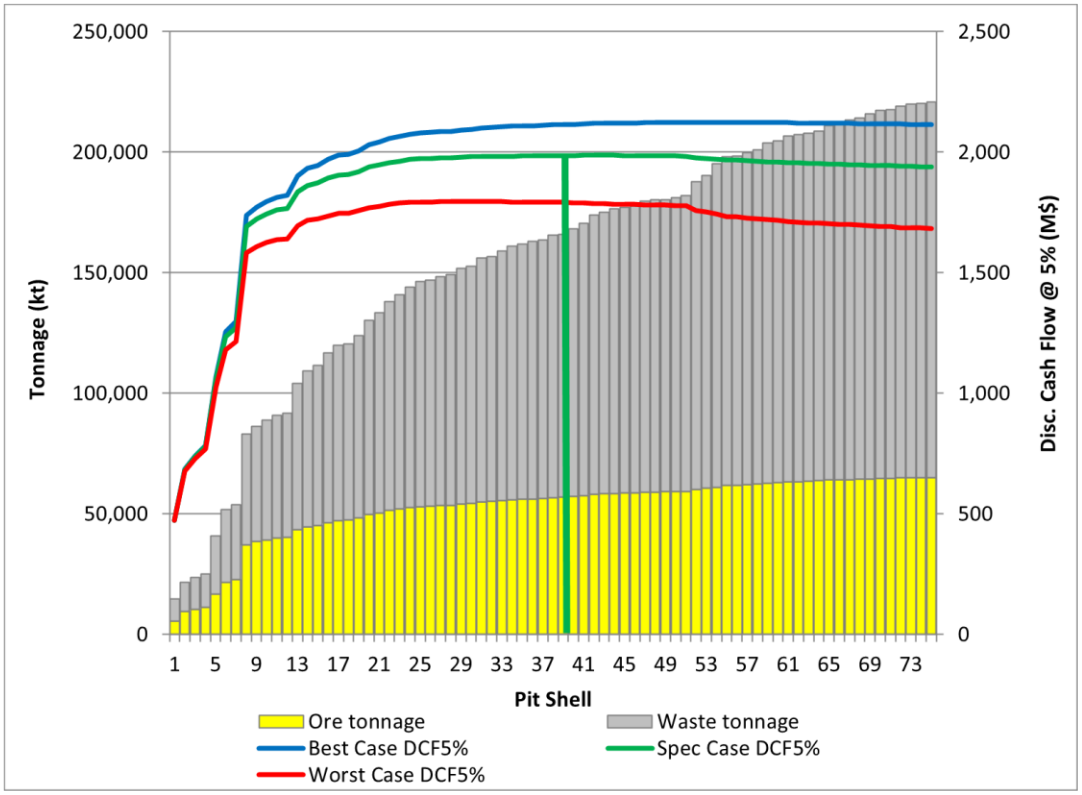
Source: GMS (2021).

**Table 15.6: Pit Shell Selection**

| Shell Selection | Best | Specified | Worst | Selection |
|---|---|---|---|---|
| Shell Number | 52 | 39 | 11 | 39 |
| Shell Revenue Factor | 1.00 | 0.79 | 0.34 | 0.79 |
| Shell Price | 1550 | 1225 | 525 | 1225 |
| Total Tonnage (kt) | 176,793 | 161,398 | 88,692 | 161,398 |
| Waste Tonnage (kt) | 128,260 | 114,717 | 55,471 | 114,717 |
| Strip Ratio (W:O) | 2.64 | 2.46 | 1.67 | 2.46 |
| Ore Tonnage (kt) | 48,532 | 46,682 | 33,222 | 46,682 |
| Au Grade (g/t) | 1.77 | 1.79 | 1.89 | 1.79 |
| Ag Grade (g/t) | 7.58 | 7.73 | 8.91 | 7.73 |
| In-situ Gold (Moz) | 2.76 | 2.69 | 2.02 | 2.69 |
| In-situ Silver (Moz) | 11.82 | 11.60 | 9.51 | 11.60 |
| Recovered Gold (Moz) | 2.47 | 2.41 | 1.81 | 2.41 |
| Recovered Silver (Moz) | 10.05 | 9.86 | 8.09 | 9.86 |
| Discounted Cash Flow @ 5 % ($M) | 1,850 | 1,750 | 1,430 | 1,750 |
| Life of Mine (Years) | 11.1 | 10.7 | 7.6 | 10.7 |

Source: GMS (2021).

**Figure 15-2: Pit-by-Pit Graph M&I Resource**




Source: GMS (2021).

## 15.4 Mine Design

### 15.4.1 Ramp Design Criteria

The ramps and haul roads are designed for the largest equipment, which is a 90-tonne class haul truck with a canopy width of 5.98 m. For double-lane traffic, industry best practice is to design a travelling surface of at least three times the width of the largest vehicle. Ramp gradients are established at 10%.

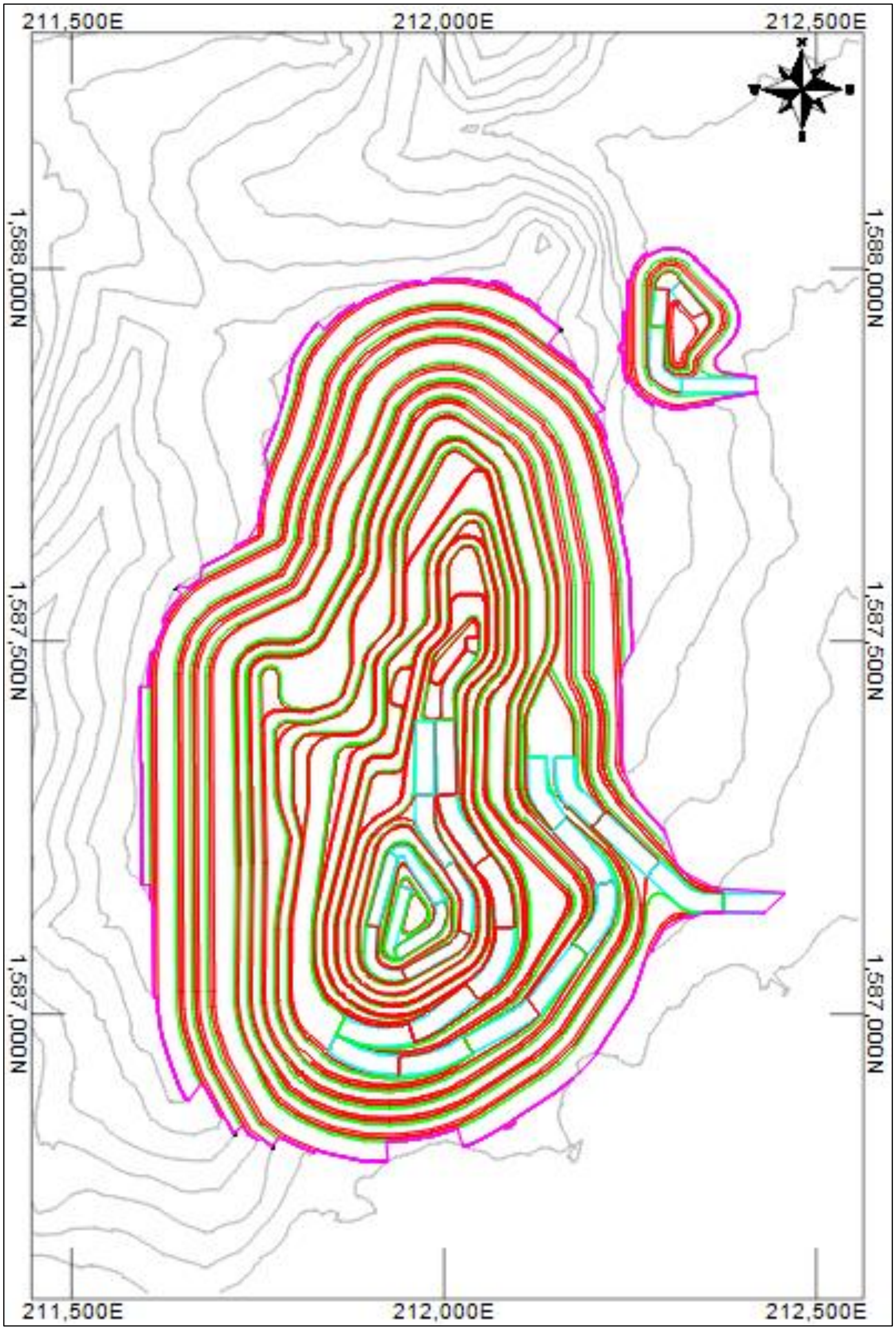
A shoulder barrier or safety berm on the outside edge of haul roads will be constructed of crushed rock to a height equal to the rolling radius of the largest tire using the ramp. The rolling radius of the truck tire is 1.35 m. These shoulder barriers are required wherever a drop-off greater than 3 m exists and will be designed at 1.1H:1V. A ditch planned on the highwall will capture run-off from the pit wall and ensure proper drainage of the running surface. The ditch will be 1.0 m wide. To facilitate drainage of the roadway a 2% cross-slope on the ramp is planned.

The double-lane ramp is 26.0 m wide, and the single-lane ramp is 16.0 m wide. Single-lane ramps are introduced in the pit bottom when the benches start narrowing and when mining rates will be significantly reduced.

### 15.4.2 Open Pit Mine Design Results

The Cerro Blanco deposit will be mined from one main pit and one satellite pit as presented in Figure 15-3. The dimensions of the main pit at the end of mine life will be 1,200 m long and 660 m wide, with a final depth of 360 m. The satellite pit will be 220 m long and 160 m wide, reaching a final depth of 80 m. The main and satellite final pit designs have one exit to the east to facilitate access towards the crusher and the primary waste dump. The ramp system introduces several switchbacks in several instances to reduce the overall slope angle.

**Figure 15-3: Final Pit Designs**




Source: GMS (2021).

## 15.5 Mineral Reserve Statement

The mineral reserve and stripping estimates are based on the final pit design presented in the previous section.

The proven and probable mineral reserves are inclusive of mining dilution and ore loss. The total ore tonnage before dilution and ore loss is estimated at 50.5 Mt at an average grade of 1.74 g Au/t and 7.60 g Ag/t.

The dilution envelope around the remaining ore blocks results in a dilution tonnage of 3.37 Mt. The dilution tonnage represents 6.7% of the ore tonnage before dilution and the dilution grade is estimated from the block model and corresponds to the average grade of the dilution skin. Table 15.7 presents a resource-to-reserve reconciliation.

**Table 15.7: Resource-to-Reserve Reconciliation**

| Description | Tonnage (kt) | Au Grade (g/t) | Ag Grade (g/t) |
|---|---|---|---|
| Ore Before Dilution | 50,520 | 1.74 | 7.60 |
| With Mining Dilution | 3,377 | 0.22 | 2.33 |
| Proven & Probable Mineral Reserve | 53,896 | 1.64 | 7.27 |

Source: GMS (2021).

Mineral reserves are stated in Figure 15.1. Proven mineral reserves total 37.6 Mt at an average grade of 1.89 g/t Au and 8.33 g/t Ag. Probable mineral reserves total 16.3 Mt at an average grade of 1.07 g/t Au and 4.81 g/t Ag. Total proven and probable reserves are 53.9 Mt an average grade of 1.64 g/t of gold for 2,846 koz, and 7.27 g/t of silver for 12,602 koz. The total tonnage to be mined is estimated at 199.2 Mt for an average strip ratio of 2.7, which includes overburden.

# 16. MINING METHODS

## 16.1 Introduction

The Cerro Blanco Gold Project will be mined by conventional open pit mining techniques using trucks and diesel-powered hydraulic excavators supported by production wheel loaders. Trucks will haul the ore to the primary crusher and to a run-of-mine (ROM) pad where it will be rehandled and process through the concentrator.

## 16.2 Open Pit Mining

Mining of the Cerro Blanco Gold Project is planned with eight phases split between the main pit and a smaller satellite pit. The mining phases and pits are summarized in Table 16.1 and depicted in Figure 16-1. The objective of pit phasing is to improve the economics of the project by feeding the mill with higher-grade material during the earlier years and/or delaying waste stripping until later years. Internal phases are designed to have a lower stripping ratio than the subsequent phases. The main pit is split into seven phases; three are in the north end, three are in the south end, and one is the final pit that includes both the north and south ends. The satellite pit has one phase.

Over the mine life, the Project will produce 53.9 Mt of ore and 145.4 Mt of waste at an overall ore-to-waste stripping ratio of 1 to 2.7.

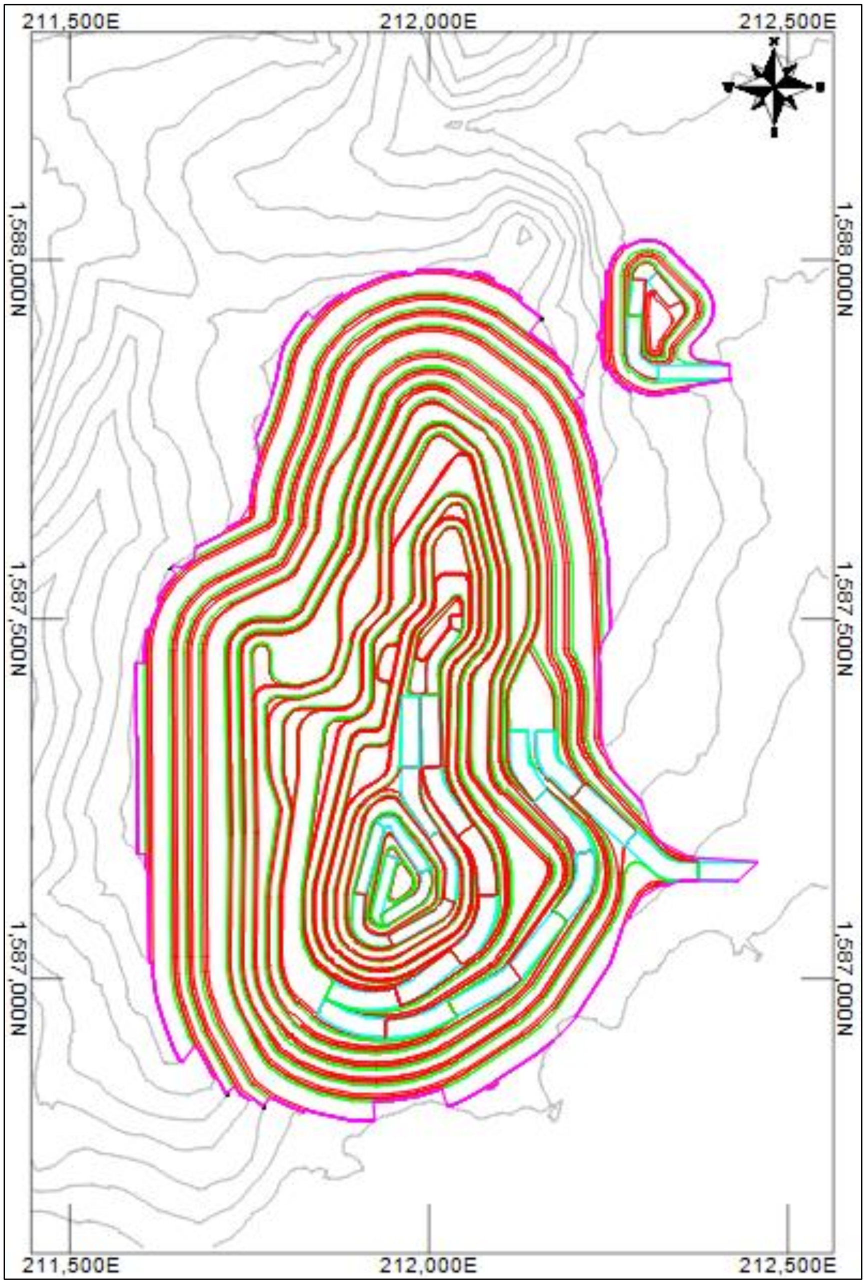
The pit designs are based on the optimized whittle shells described in Section 15 and created with the parameters outlined in Section 16.2.1.

**Table 16.1: Pit Phase Design Summary**

| Description | Unit | Total | Main Pit | | | | | | | | | Satellite Pit |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | | South | | | | North | | | | Global | |
| | | | Phase 1 | Phase 2 | Phase 3 | Total | Phase 1 | Phase 2 | Phase 3 | Total | Phase 4 | Phase 1 |
| Total[1] | kt | 199,283 | 21,692 | 28,003 | 57,125 | 106,820 | 19,897 | 12,409 | 2,864 | 35,169 | 55,814 | 1,480 |
| Waste | kt | 145,387 | 14,674 | 17,289 | 40,556 | 72,519 | 15,044 | 8,829 | 2,782 | 26,655 | 45,097 | 1,115 |
| Ore | kt | 53,896 | 7,018 | 10,714 | 16,569 | 34,301 | 4,853 | 3,580 | 82 | 8,514 | 10,717 | 364 |
| Stripping Ratio | W:O | 2.70 | 2.09 | 1.61 | 2.45 | 2.11 | 3.10 | 2.47 | 33.95 | 3.13 | 4.21 | 3.06 |
| Au | g/t | 1.64 | 2.02 | 1.67 | 1.50 | 1.66 | 1.50 | 1.52 | 0.80 | 1.50 | 1.72 | 0.84 |
| Au Recovery | g/t | 1.53 | 1.89 | 1.56 | 1.40 | 1.55 | 1.40 | 1.42 | 0.73 | 1.40 | 1.60 | 0.70 |
| Contained Au | koz | 2,845.7 | 455.3 | 576.4 | 800.2 | 1,832.0 | 234.1 | 175.0 | 2.1 | 411.2 | 592.7 | 9.9 |
| Recovered Au | koz | 2,645 | 421.3 | 538.4 | 744.2 | 1,709.1 | 218.0 | 163.0 | 1.9 | 382.9 | 550.2 | 8.2 |
| Ag | g/t | 7.27 | 10.22 | 8.64 | 5.84 | 7.61 | 9.88 | 6.50 | 4.27 | 8.40 | 5.27 | 7.83 |
| Ag Recovery | g/t | 6.13 | 8.63 | 7.29 | 4.91 | 6.41 | 8.28 | 5.48 | 3.63 | 7.06 | 4.46 | 6.50 |
| Contained Ag | koz | 12,601.6 | 2,306.8 | 2,976.2 | 3,109.6 | 8,392.6 | 1,541.1 | 748.3 | 11.3 | 2300.7 | 1,816.6 | 91.7 |
| Recovered Ag | koz | 10,617.3 | 1,947.6 | 2,510.2 | 2,613.5 | 7,071.3 | 1,291.1 | 631.1 | 9.6 | 1,931.8 | 1,538.1 | 76.1 |

Note: [1]Total material moved. Source: GMS (2021).

**Figure 16-1: End of Life of Mine Pit Layout**




Source: GMS (2021).

### 16.2.1 Open Pit Optimization

All phases use specific Whittle shells to optimize shape and layout. This ensures that mining achieves maximum individual net present value. Table 16.2 depicts the nomenclature of each of the pits and the Whittle shell that guided their design. For more details on the individual pit shells, refer to Section 15.

**Table 16.2: Pit Shell Hierarchy**

| Phasing | Whittle Shell Number |
|---|---|
| South Pit Phase 3 | Shell 13 |
| → South Pit Phase 2 | Shell 7 |
| → South Pit Phase 1 | Shell 1 |
| North Pit Phase 3 | Shell 13 |
| → North Pit Phase 2 | Shell 13 |
| → North Pit Phase 1 | Shell 5 |
| Final Pit | Shell 39 |
| Satellite Pit | Shell 39 |

Source: GMS (2021).

Whittle shells are made without ramps, consideration of minimum mining width, or ramp access. These details are included in subsequent pit designs. Whittle shells are designed with a lower overall slope angle (OSA) to account for these changes. Even with these accommodations, it is typical for the inventory of the pits to fluctuate from the Whittle designs and the actual design. Table 16.3 depicts the comparison between the inventory calculated by Whittle and the inventory in the designed pits. There is a 23% increase in total tonnage of the pits and a 15% increase in ore tonnage with a significant increase of the stripping ratio. In the Whittle designs, it is assumed that the impact of ramping is spread over the entirety of the walls, and it is not possible to target ore rich areas to place ramping.

**Table 16.3: Shell & Design Comparison**

| Description | Units | Shell | Design |
|---|---|---|---|
| Total | kt | 161,398 | 199,283 |
| Waste | kt | 114,717 | 145,387 |
| Ore | kt | 46,682 | 53,896 |
| Stripping Ratio | W:O | 2.46 | 2.70 |
| Au | g/t | 1.79 | 1.64 |
| Ag | g/t | 7.73 | 7.27 |

Source: GMS (2021).

### 16.2.2 Open Pit Design Criteria

The open pit designs are established with the design criteria described in the following subsections.

#### 16.2.2.1 Geotechnical Parameters

Table 16.4 summarizes the geotechnical parameters used in the design of the pit walls. Figure 16-2 outlines the geotechnical zones with its respective parameters overlaying the mine design. The justification for these parameters is in a pit stability study report prepared by E-Mining Technology.

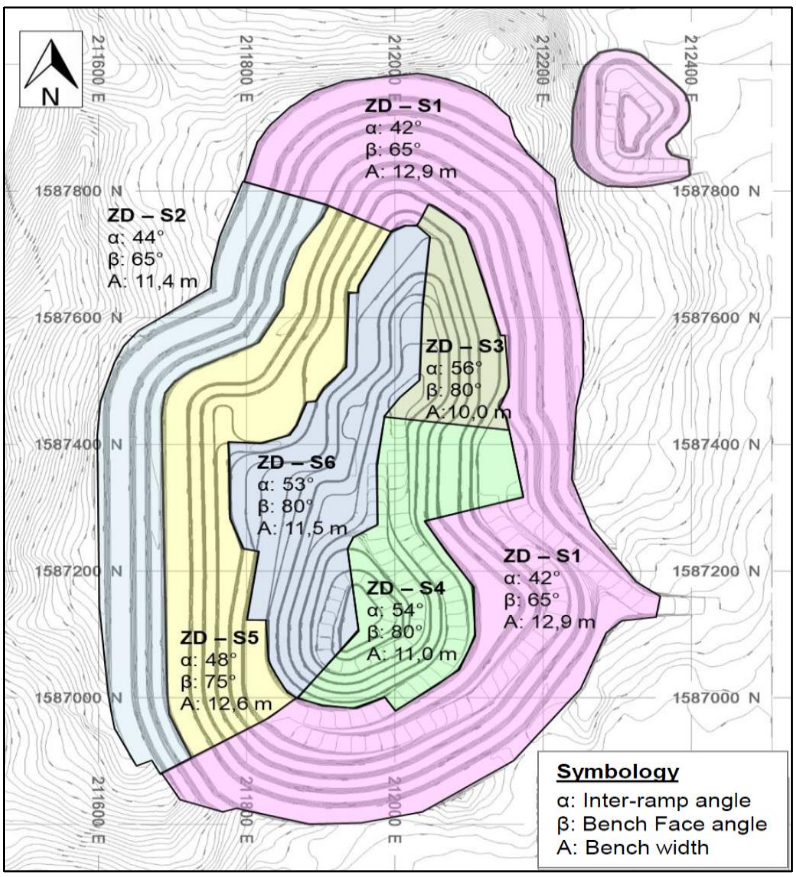
Geotechnical berms were used in the pits and are separated by lithology group. In the Salinas group, geotechnical berms are present every 60 m of elevation. In the Mita group, geotechnical berms are present every 100 m of elevation. Temporary walls between phases are assumed to have the same parameters as final walls as pre-split is planned for all mining.

**Table 16.4: Geotechnical Design Parameters Summary**

| Design Area | Bench Height = 20 m | | |
|---|---|---|---|
| | Inter-Ramp Angle, α (°) | Bench Face Angle, β (°) | Bench Width (m) |
| ZD-S1 | 42° | 65° | 12,9 |
| ZD-S2 | 44° | 65° | 11,4 |
| ZD-S3 | 56° | 80° | 10,0 |
| ZD-S4 | 54° | 80° | 11,0 |
| ZD-S5 | 48° | 75° | 12,6 |
| ZD-S6 | 53° | 80° | 11,5 |

Source: GMS (2021).

**Figure 16-2: Application of Geotech Zones**

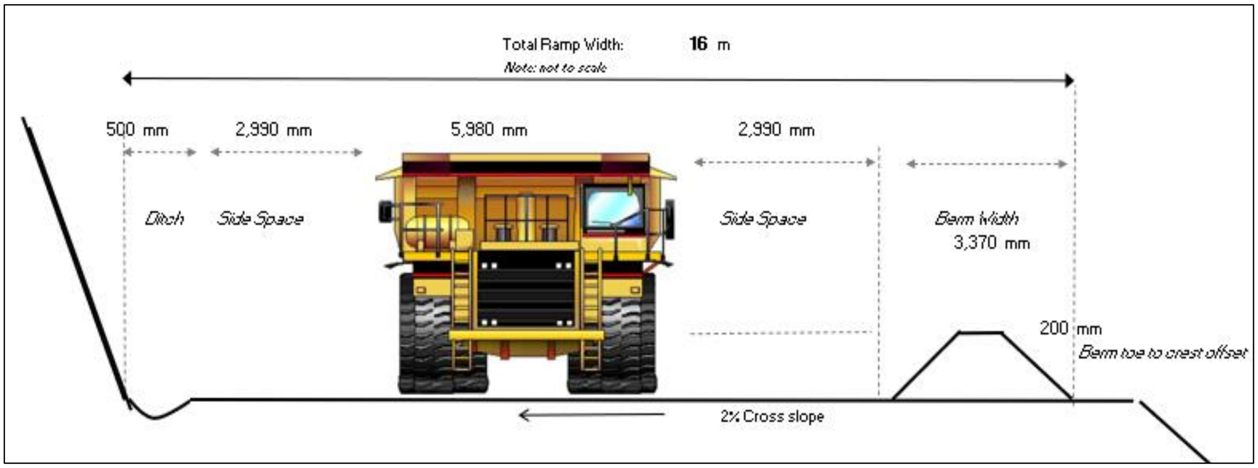



Source: GMS (2021).

#### 16.2.2.2 Ramp & Road Design

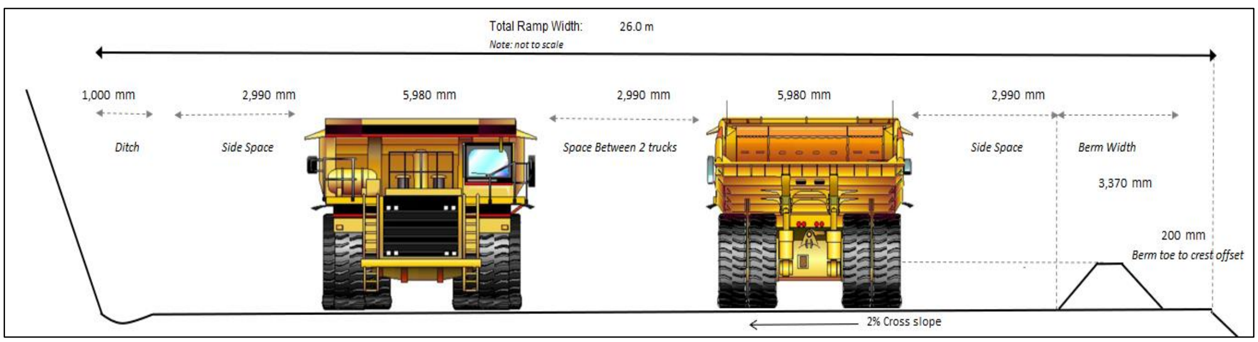
Ramp designs are shown in Figure 16-3 and Figure 16-4 for the single lane and double lane ramps. The ramps are designed specifically for the primary hauling equipment, a 90-tonne truck, in accordance with SME Standard, the ramp with needs to be 2.0 x (for single lane) and 3.5 x (for double lane) of the vehicle operating width. The operating width of the 90-tonne truck is 5.98 m. The ramp includes adequate distance for the vehicles to operate, a safety berm on the pit side, and a drainage ditch on the wall side. The safety berm is designed to be at least half the height of the tallest tire to be used on site, in this case the tires of the 90-tonne truck.

**Figure 16-3: Single-Lane Ramp Haul Road Profile**




Source: GMS (2021).

**Figure 16-4: Double-Lane Haul Road Profile**




Source: GMS (2021).

#### 16.2.2.3 Mine Design Parameters

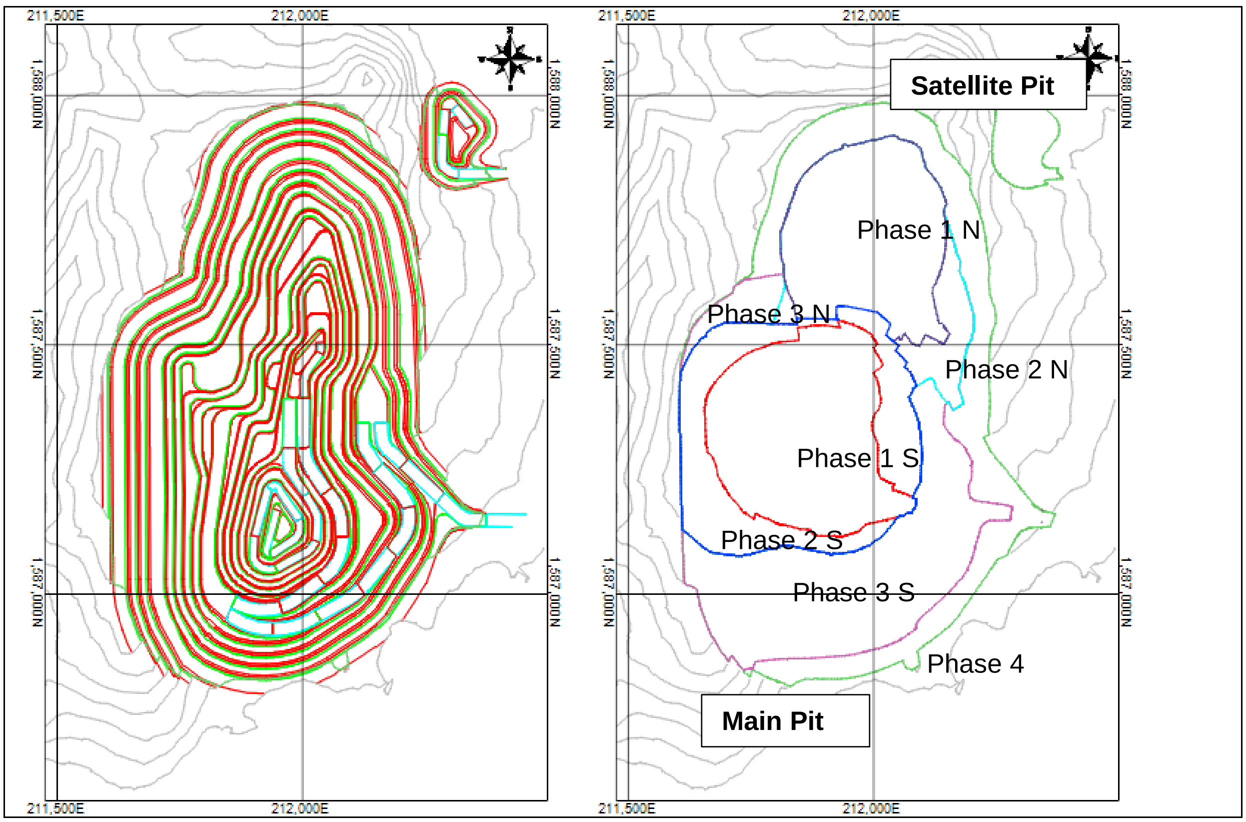
A minimum mining width of 30 m was used to safely and optimally mine between phases or at the bottom of a pit. This value is determined by the operating width of the primary excavator, the width required for a double lane road with berm, and the area required for the 90-tonne truck to safely complete a three-point turn.

Single lane ramps were used in the bottom 40 to 60 m of the final pits to reduce stripping and capture more ore. Single lane ramps can cause bottlenecks and reduce the fleet productivity. This method is only used sparingly at the bottom of the pits. Reductions in productivity stemming from single lane ramping is captured in reduced production or compensated by mining in other pits or phases. To attain additional ore at the bottom of the pit, 10 m box cuts are used.

### 16.2.3 Open Pit Designs

Final pit designs and phase limits are depicted in Figure 16-5 . Ramps for the pits are designed to exit east of the pits to better access the primary waste dump and the ore crusher.

**Figure 16-5: End of LOM Pit Layout & Phase Limits**




Note: End of mine life (left), phase limits (right). Source: GMS (2021).

The main pit consists of seven phases. For the north and south end, each phase is deeper than the previous one. All the phases exit to the east side of the pit, providing access to the crusher and the primary dump. Using the pushback method, a new ramp will be needed for each phase.

The south portion of the main pit consists of three phases in the main orebody. Parts of the west wall are shared among phases 2, 3, and 4. The fourth and final phase includes both the south and north phases.

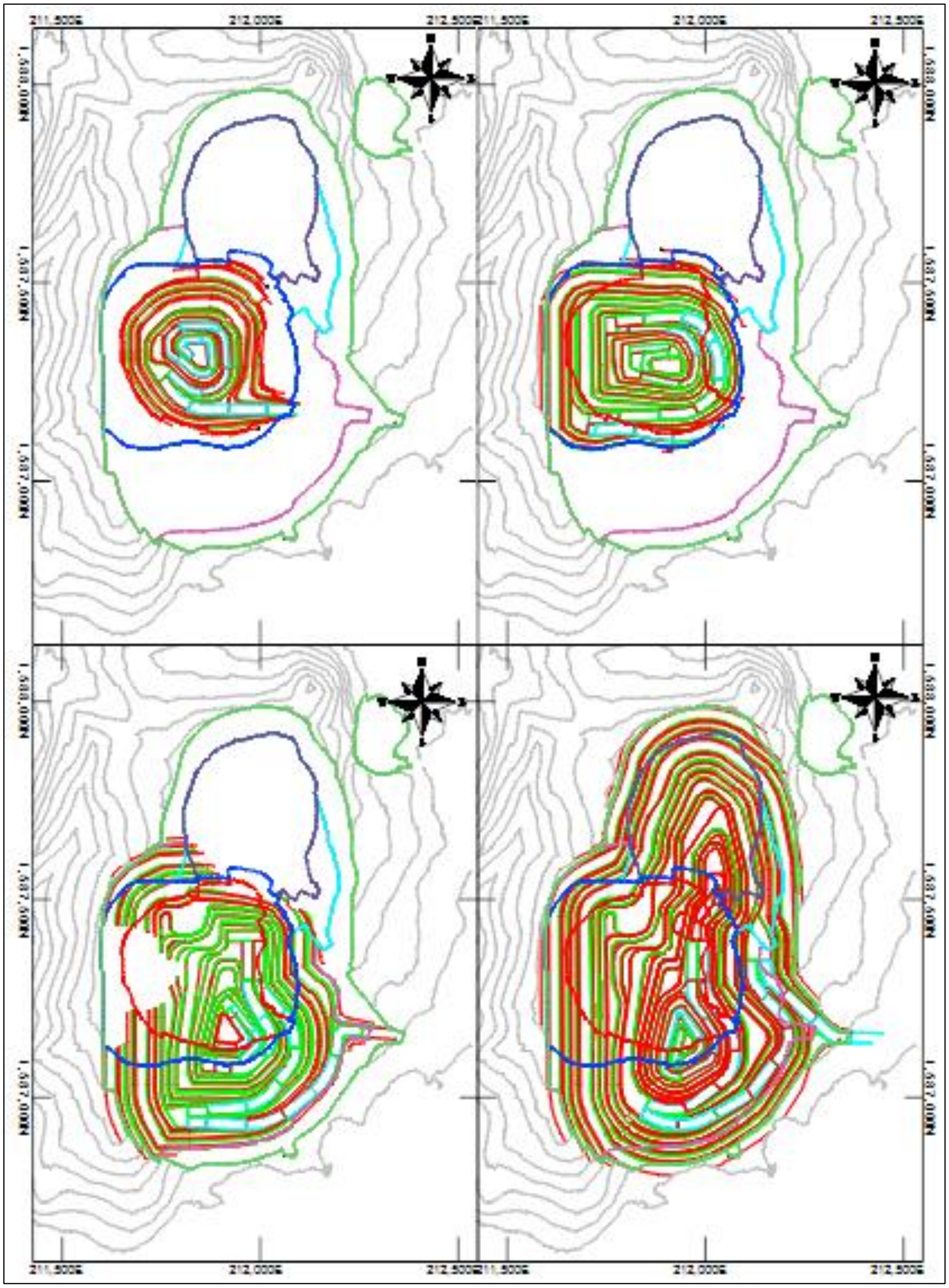
Phase 1 defines part of the west wall of the main pit. A geotechnical berm is planned on the north part of this phase. There are 60 m of single-lane ramping at the bottom and a 10 m box cut.

Phase 2 reaches the final wall on parts of the west wall. There is a 10 m box cut at the bottom and 60 vertical meters of single-lane ramping.

Phase 3 expands on the final wall of the west wall of phase 2. It also contains the phase 3 north, which will be mined first. There is a 5 m box cut at the bottom and contains 50 m of single-lane ramping.

Phase 1 south has a depth of 200 m and is approximately 440 m long and 360 m wide. Phase 2 South has a depth of 250 m and is approximately 470 m long and 510 m wide. Phase 3 South has a depth of 305 m and is approximately 780 m long at its longest and is 580 m wide. Phase 4 is the final phase of the pit and includes both south and north phases. It has a depth of 360 m and is approximately 1.2 km long and 660 m wide. Refer to Figure 16-6 for details.

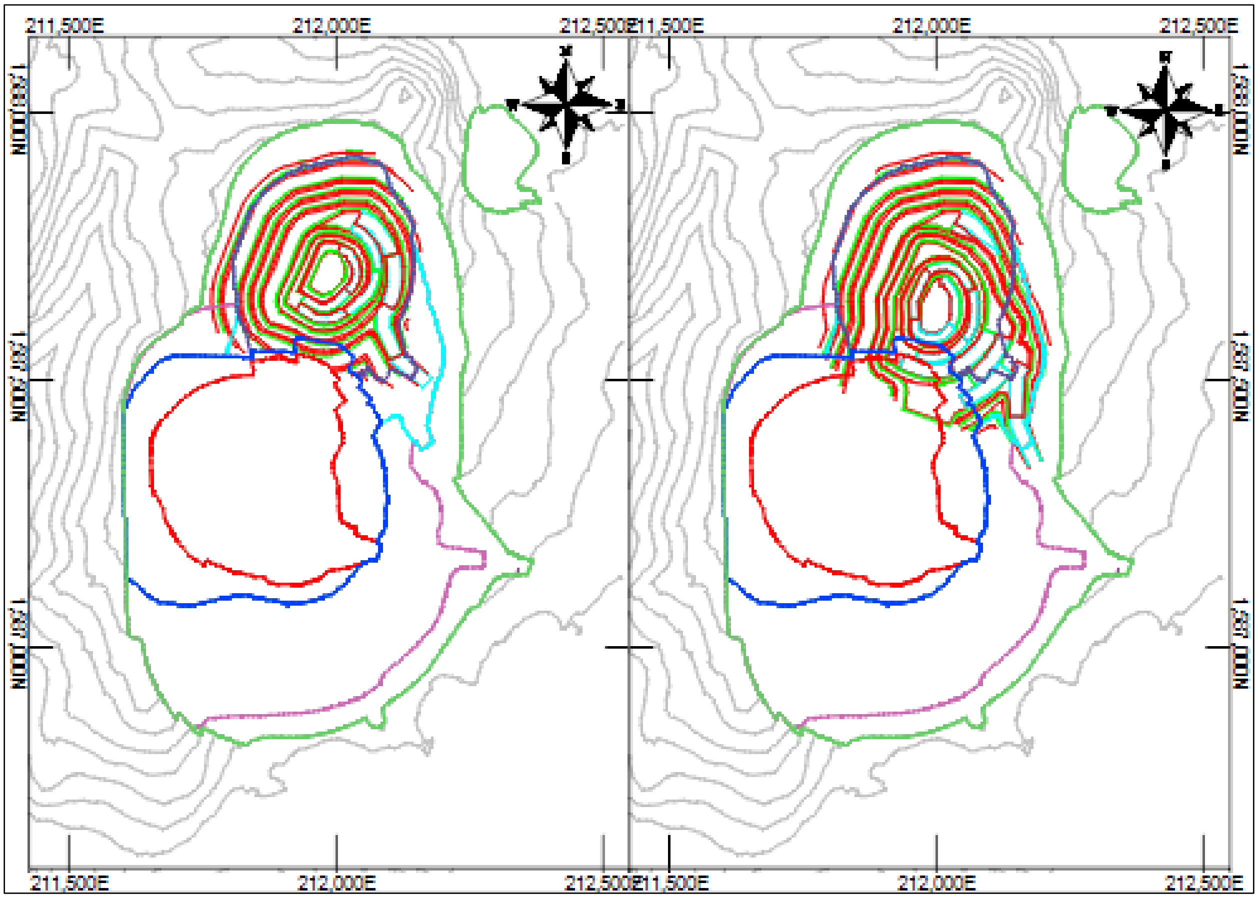
**Figure 16-6: South Phases 1, 2, 3 & 4**



Note: Phase 1 South (top left), Phase 2 South (top right), Phase 3 South (bottom left), Phase 4 (bottom right). Source: GMS (2021).

The north phases are not the located where the main ore body is and are therefore second in importance. Phase 3 north is also only a small portion to be mined at the end of Phase 2 north and the beginning of Phase 3 south. It is not represented here as it is part of Phase 3 south. Phase 1 north has a depth of 150 m and is approximately 400 m long and 340 m wide. Phase 2 north has a depth of 200 m and is approximately 510 m long and 370 m wide. Phase 3 north is a small part on the north-west end of the pit (not represented in Figure 16-7).

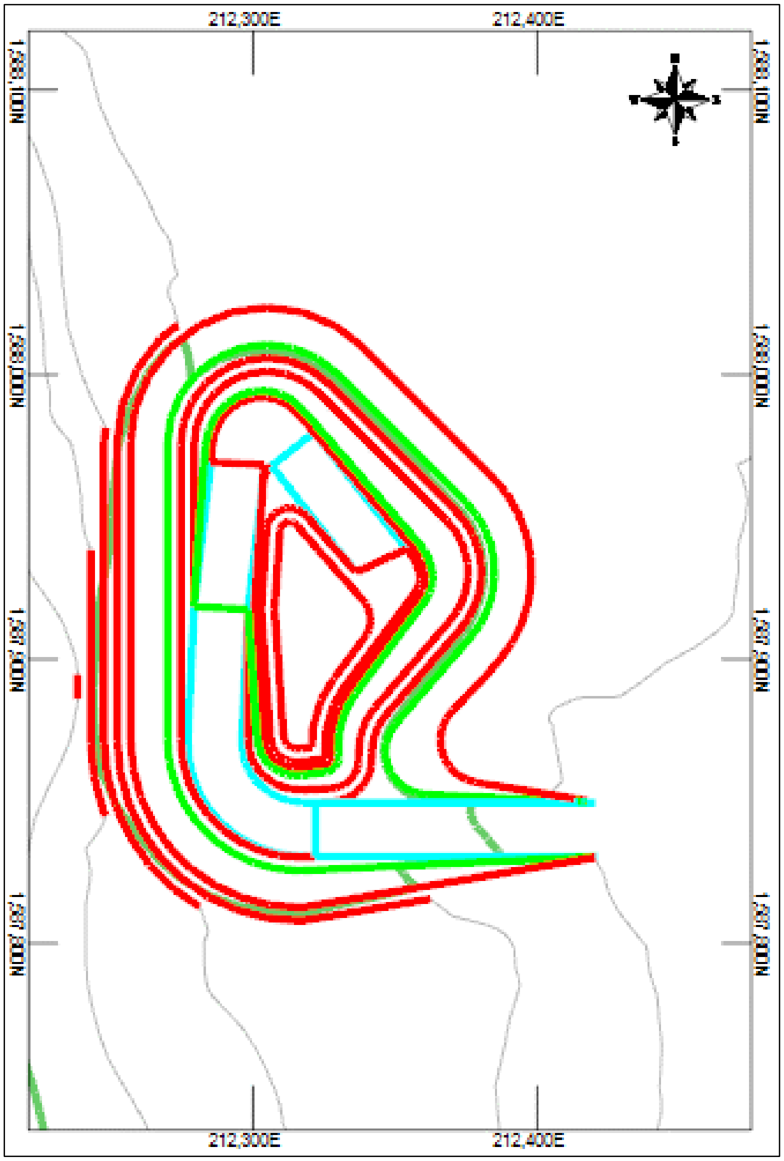
**Figure 16-7: North Phases 1 & 2**




Note: North Pit: Phase 1 (left), Phase 2 (right). Source: GMS (2021).

The satellite pit consists of one phase. As the ore is a lower grade than the main pit, it will be mined last. The ramp is located on the east side to take advantage of the natural topography. The satellite pit is 80 m deep with a length of 220 m and width of 160 m. Refer to Figure 16-8 for details.

**Figure 16-8: Satellite Pit Phase 1**




Source: GMS (2021)

### 16.2.4 Overburden & Waste Rock Storage

A total of 145.4 Mt of waste rock is produced over the mine life. The overburden tonnage is included in the waste tonnage total. The waste is split into three categories of potentially acid-producing (PAG or “Type 2”) rock; non-potentially acid-producing (NPAG or “Type 1”) rock; and overburden. They will all be stacked at the same dump location. The storage of tailings will follow a dry stacking method and will be located in the north portion of the East Dump for an estimated 55.4 Mt of material. The topsoil will be stockpiled for use during mine closure. Table 16.5 depicts the design parameters of each of the dumps.

**Table 16.5: Overburden & Dump Design Parameters**

| Waste Dump | Avg. Catch Bench Width (m) | Pile Face Angle (deg) | Overall Slope Angle (H:V) | Maximum Elevation (m) | Approximate Height (m) |
|---|---|---|---|---|---|
| East Waste Dump | 22 | 33.5 | 2.5:1 | 600 | 150 |
| West Waste Dump | 0 | 26.6 | 2.0:1 | 571 | 100 |

Source: GMS (2021).

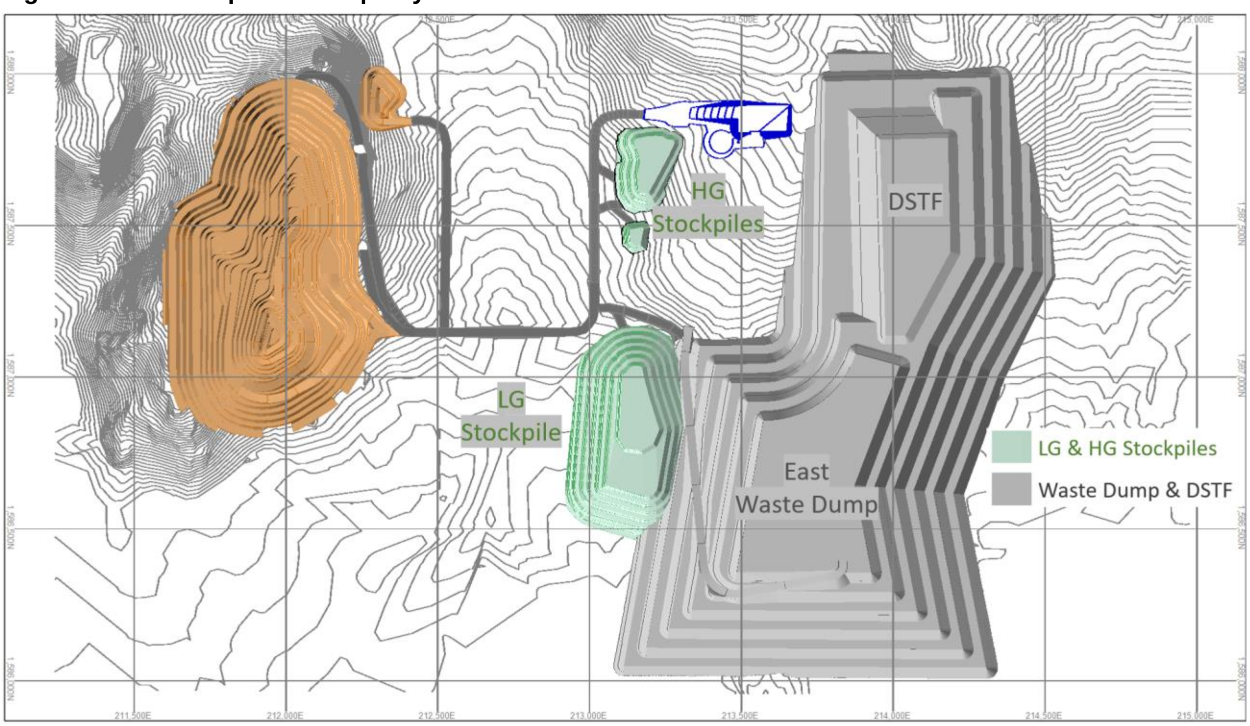
Table 16.6 depicts the various dumps, their capacities and expected fill rates. Figure 16-9 depicts a plan view of the dumps shown at maximum capacity. See Section 16.3.3, Surface Schedule, for dumps at specific periods.

**Table 16.6: Surface Storage Capacities**

| Waste Dump | Capacity (Mt) | Capacity ($Mm^3$) | Surface Area (ha) | % Filled |
|---|---|---|---|---|
| East Waste Dump | 168.6 | 97.6 | 193.3 | 86.0 |
| West Waste Dump | 50.0 | 29.0 | 77.0 | 0.0 |
| Dry Stack Tailings Facility | 54.7 | 38.4 | 70.1 | 99.5 |

Source: GMS (2021).

**Figure 16-9: Stockpile & Dump Layout**




Source: GMS (2021).

The East Dump is planned to be the only waste dump in the project. A contingency dump located on the northwest end of the pit was also designed. The dry stack tailings facility (DSTF) will be located inside the East Dump, where the berms will be built from waste and the tailings will be dry stacked at a humidity between 15% and 20%. The dump is located 900 m east of the open pit and contains three accesses on its west and north side. A system of pools and ditches located around the dump will contain water originating from both the dump and the DSTF. In-pit dumping was not considered an option due to the mining sequence, except for the satellite pit that will be filled with waste rock during the last year of mine life.

### 16.2.5 Ore Stockpile

The ore stockpiles in Figure 16-9 represent the maximum capacity required for the life of mine. The high-grade stockpile will be used to store higher-grade material to blend and achieve a stable crusher feed. The low-grade stockpile will be used to store marginally lower-grade material and to allow for the preferential plant feed of higher margin ore. The ore stockpile tonnage will fluctuate over the mine life. All ore placed on the stockpile is included in the life-of-mine plan and will be fed and milled over the mine life. The low-grade stockpile will be processed in the final years once mining is complete and all high-grade material has been milled. Table 16.7 depicts the cut-off grades in gold equivalent (Au Eq.) of the three stockpiles as well as the mineralized waste grade. Most of the material stockpiled will be low-grade ore. The mineralized waste will be sent to the waste dump. Different ore groupings will be stored separately within the stockpile to manage in-dump dilution. Table 16.8 depicts the design parameters for the stockpile.

**Table 16.7: Ore Grade Stockpile Definitions**

| Grade Stockpile Definition | Cut-off Grade (Au g/t Eq.) |
|---|---|
| Mineralized Waste | 0.4 to 0.5 |
| Low-Grade Stockpile | 0.5 |
| High-Grade Stockpile | 0.8 |

Source: GMS (2021).

**Table 16.8: Stockpile Design Parameters & Capacities**

| Ore Stockpiles | Catch Bench Width (m) | Overall Slope Angle (H:V) | Maximum Elevation (masl) | Approximate Height (m) | Maximum Capacity (Mt) |
|---|---|---|---|---|---|
| Low-Grade Stockpile | 10.0 | 1:1.5 | 535 | 85 | 16.3 |
| High-Grade Stockpile | 12.5 | 1:1.5 | 515 | 40 | 1.7 |

Source: GMS (2021).

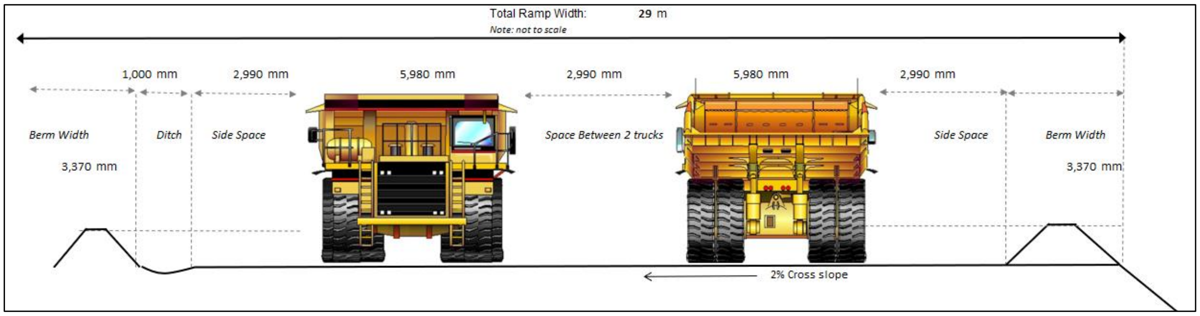
The stockpile is accessible from the access road east of the pit. Material to be rehandled will require a loading unit, typically a front-end loader, and haul trucks to bring the material from the LG stockpile and from the HG stockpile and up to the crusher via the crusher ramp. These routes have a cycle time of 14.9 minutes for the LG stockpile and 8.8 minutes for the HG stockpile. The cycle times include loading and dumping. Over the life of mine, the peak inventory of stockpiled material is 15.4 Mt in Year 8 of mining and a total of 21.3 Mt of ore will be stored and reclaimed from the stockpile.

### 16.2.6 Surface Mine Haul Roads & Access

This section refers only to the haul and access roads accessible by haul trucks and heavy mining equipment.

Haul roads for the 90-tonne trucks are designed with berms on both sides of the road and include a drainage ditch on one side. All surface roads have double lanes. The total width of a surface road is 29 m (Figure 16-10).

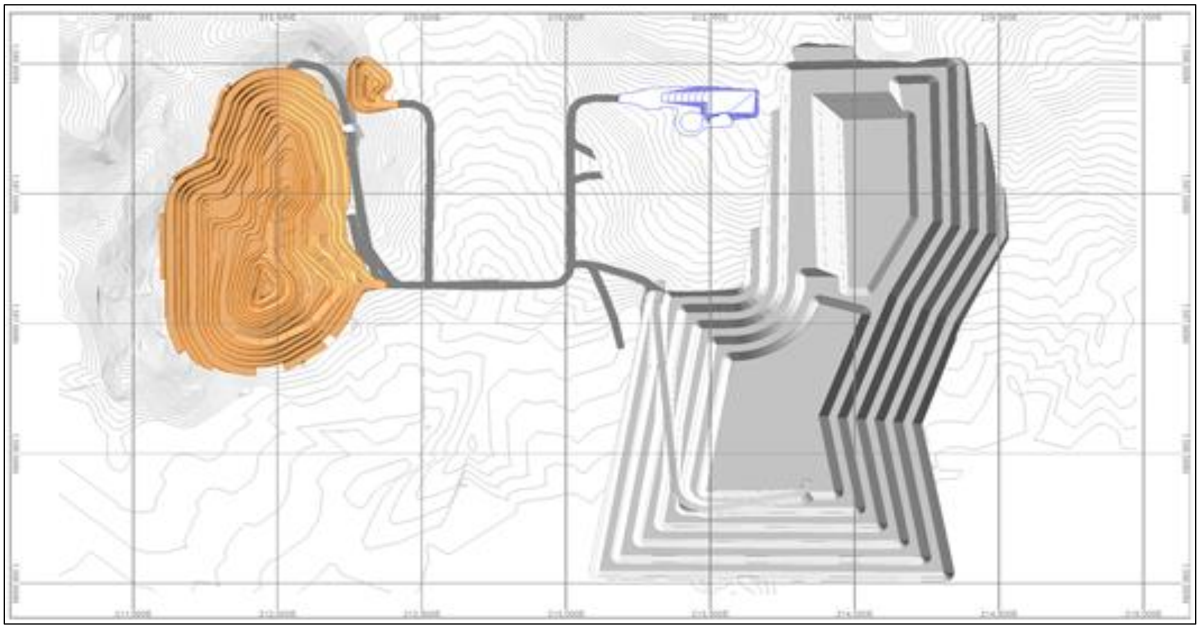
**Figure 16-10: Double-Lane Surface Haul Road Profile**




Source: GMS (2021).

Figure 16-11 depicts the final haulage roads. Table 16.9 presents the cut and fill requirements for the road sections by period as well as the crusher pad and ramp. The road sections include the mine access roads and pit access roads. The crusher ramp consists of a ramp up to the crusher located east of the main pit. The crusher ramp is connected to the mine access road.

**Figure 16-11: Site Mine Road Layout**



Source: GMS (2021).

**Table 16.9: Ramp Cut & Fill Design Parameters & Requirements**

| Road | Period | Fill (000xm$^3$) | Cut (000xm$^3$) | Net (000xm$^3$) |
|---|---|---|---|---|
| Roads | Start | 290.4 | 7.0 | 283.4 |
| Crusher Ramp | Start | 300.0 | 0 | 300 |
| Roads | Year -1 | 307.5 | 120.3 | 187.2 |
| Roads | Year 2 | 9.3 | 49.5 | -40.2 |
| Roads | Year 4 | 63.6 | 4.8 | 58.8 |
| Roads | Year 6 | 0 | 0 | 0 |
| Roads | Year 8 | 2.3 | 6.8 | -4.5 |
| **Total** | | **673.1** | **488.4** | **184.7** |

Source: GMS (2021).

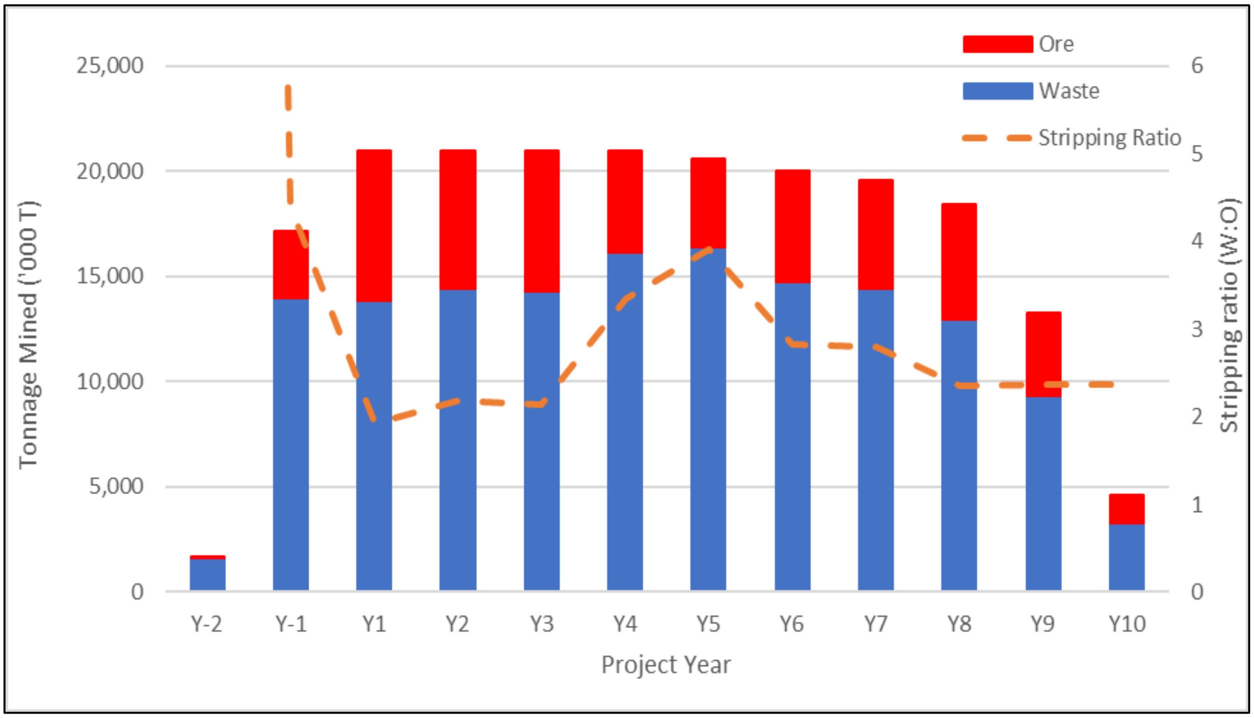
## 16.3 Mine Production Schedule

The mining and milling schedule were optimized by Alicanto Scheduler to maximize the project's net present value (NPV). The optimization includes mine sequencing and mining rate, stockpile usage and rehandling, and fleet usage. The results from Alicanto Scheduler were further detailed and a Deswik schedule was used to track material movements, stockpile inventory, mill blending, block mining, waste movements, and equipment usage/movements. Mine production details are provided in Table 16.10 on the following page.

### 16.3.1 Mining Schedule

Mining activities for the project take place over ~12 years. This includes a ramp-up period of two years; eight years of mining at peak capacity; and a two-year post-peak production ramp down. The peak mining rate is 21 Mt/a (57,500 t/d) at an average stripping ratio of 1:2.7. Figure 16-12 below outlines the production schedule by material type and the stripping ratio. Ore feed is consistent through the mine life with no periods of significant stripping required to meet mill requirements.

**Figure 16-12: Mine Production Schedule**




Source: GMS (2021)

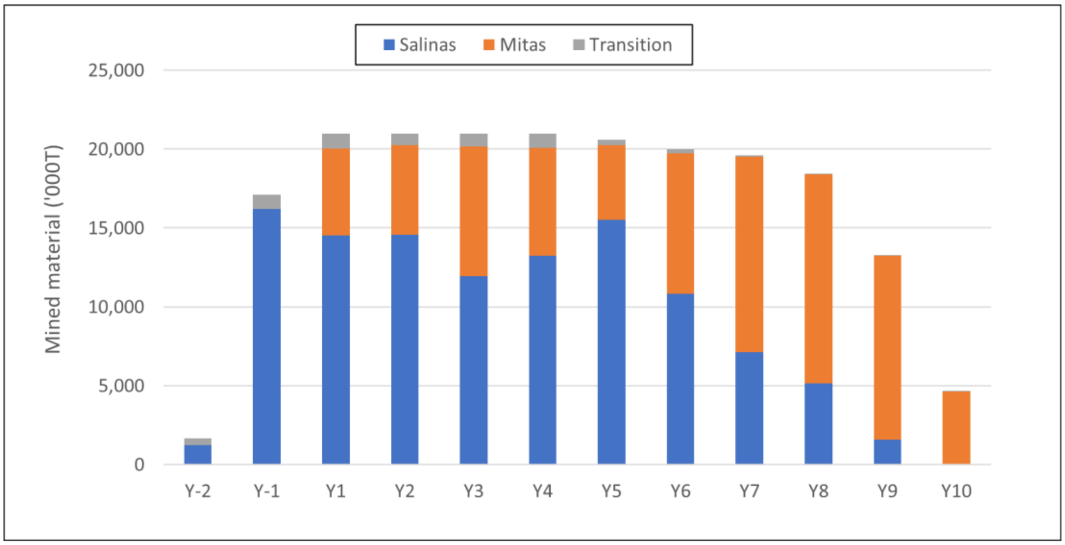
**Table 16.10: Detailed Mine Production**

| Description | Unit | Total | Y-2 | Y-1 | Y1 | Y2 | Y3 | Y4 | Y5 | Y6 | Y7 | Y8 | Y9 | Y10 | Y11 | Y12 | Y13 | Y14 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Total Tonnage | Mt | 199.3 | 1.7 | 17.1 | 21.0 | 21.0 | 21.0 | 21.0 | 20.6 | 20.0 | 19.6 | 18.4 | 13.3 | 4.60 | 0.00 | 0.00 | 0.00 | 0.00 |
| Total Waste | Mt | 145.4 | 1.6 | 14.0 | 13.8 | 14.4 | 14.3 | 16.2 | 16.4 | 14.8 | 14.4 | 12.9 | 9.3 | 3.20 | 0.00 | 0.00 | 0.00 | 0.00 |
| Strip Ratio | W:O | 2.7 | 50.7 | 4.4 | 1.93 | 2.19 | 2.14 | 3.34 | 3.91 | 2.82 | 2.8 | 2.35 | 2.36 | 2.37 | 0.00 | 0.00 | 0.00 | 0.00 |
| Ore Tonnage | Mt | 53.9 | 0.0 | 3.2 | 7.2 | 6.6 | 6.7 | 4.8 | 4.2 | 5.2 | 5.2 | 5.5 | 4.0 | 1.40 | 0.00 | 0.00 | 0.00 | 0.00 |
| Gold Grade | g/t | 1.64 | 0.00 | 1.04 | 2.87 | 1.95 | 2.83 | 2.16 | 1.14 | 1.52 | 1.75 | 2.07 | 2.04 | 1.73 | 0.56 | 0.56 | 0.56 | 0.56 |
| Silver Grade | g/t | 7.27 | 0.00 | 4.51 | 14.55 | 11.75 | 13.88 | 9.81 | 6.29 | 7.36 | 6.08 | 5.32 | 5.02 | 4.96 | 3.66 | 3.66 | 3.66 | 3.66 |

Source: GMS (2021).

Mining is split between the Mita and the Salinas domains. Figure 16-13 depicts the material mined by each domain through the years.

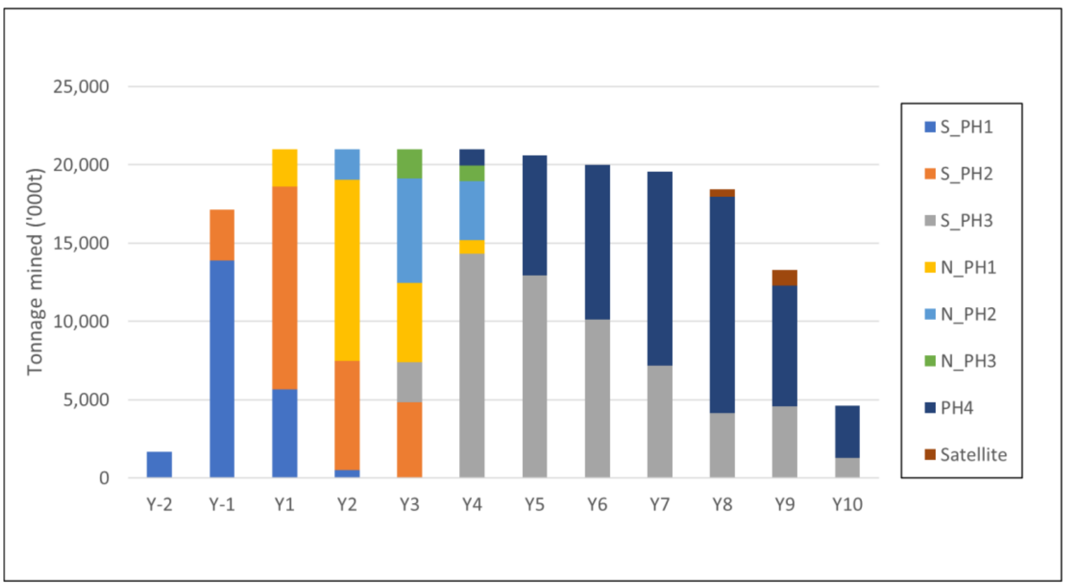
**Figure 16-13: Material Mined by Domain Groupings**




Source: GMS (2021).

Figure 16-14 depicts the mining in each of the phases in the project. The main pit will be mined by Year 10, while the satellite pit will be mined during Years 8 and 9.

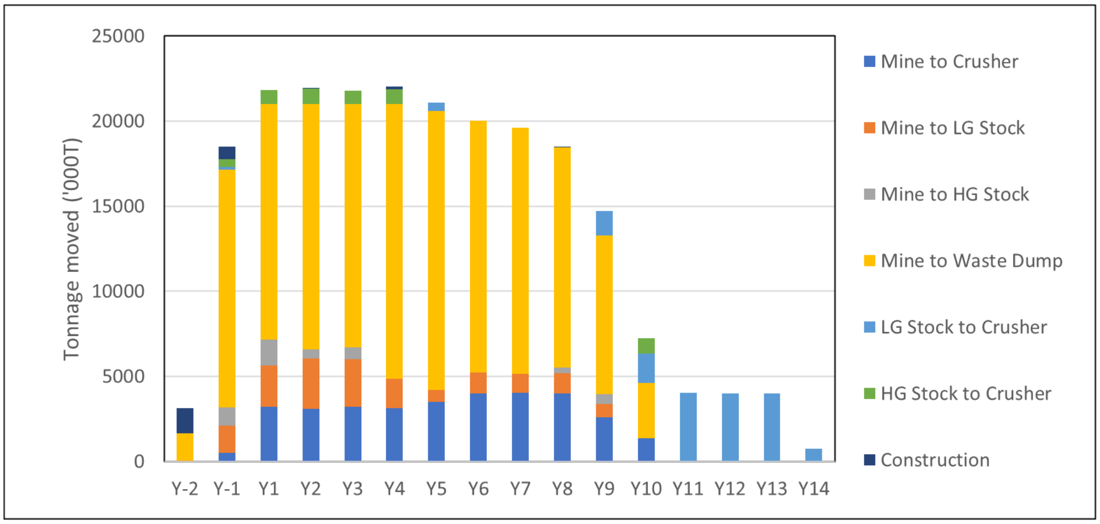
**Figure 16-14: Material Movements from Pit**




Source: GMS (2021).

Figure 16-15 shows the movements of materials from the pit and stockpiles and the associated destinations.

**Figure 16-15: Material Movements from the Pit & Stockpiles**

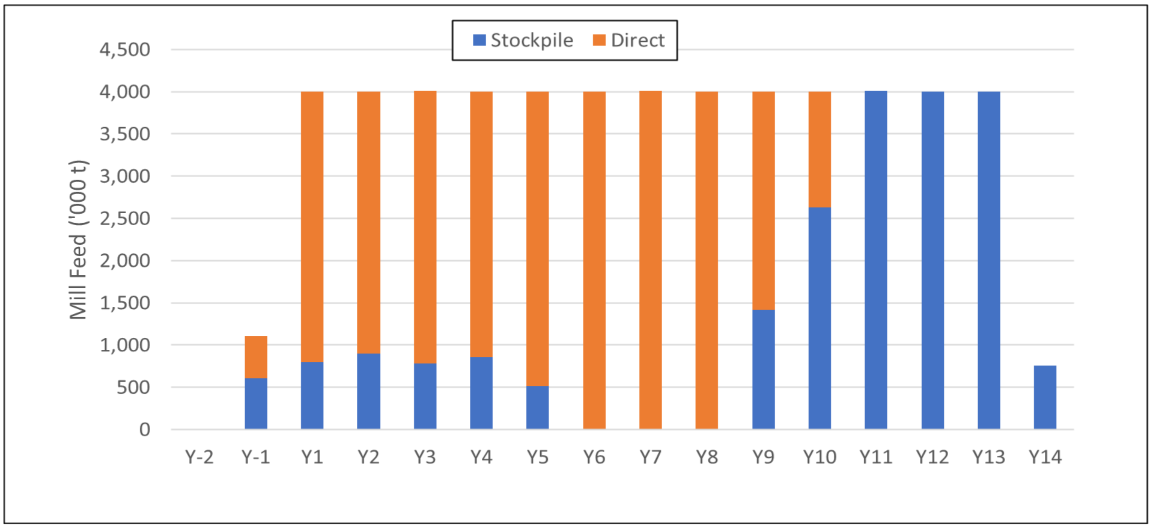



Source: GMS (2021).

### 16.3.2 Milling Schedule

The mill life for the project is 13.7 years. The peak milling capacity is 4.0 Mt per year with a ramp-up of 1.1 Mt in the first year before reaching peak milling rate. The mill will be fed with the highest-grade ore available, therefore the mill feed rate is maintained with ore coming directly from the pit and with rehandled ore from the stockpile. During the final years of the project when the pit is depleted, the mill will be fed only with rehandled ore from the stockpile. Milling is optimized to maximize the project NPV and to minimize rehandling. Medium- and high-grade ore are prioritized to maximize revenue. The last four years of milling consists entirely of low-grade material that was previously stockpiled. Figure 16-16 outlines the mill feed by source. A detailed milling schedule is provided in Table 16.11.

**Figure 16-16: Mill Production**




Source: GMS (2021).

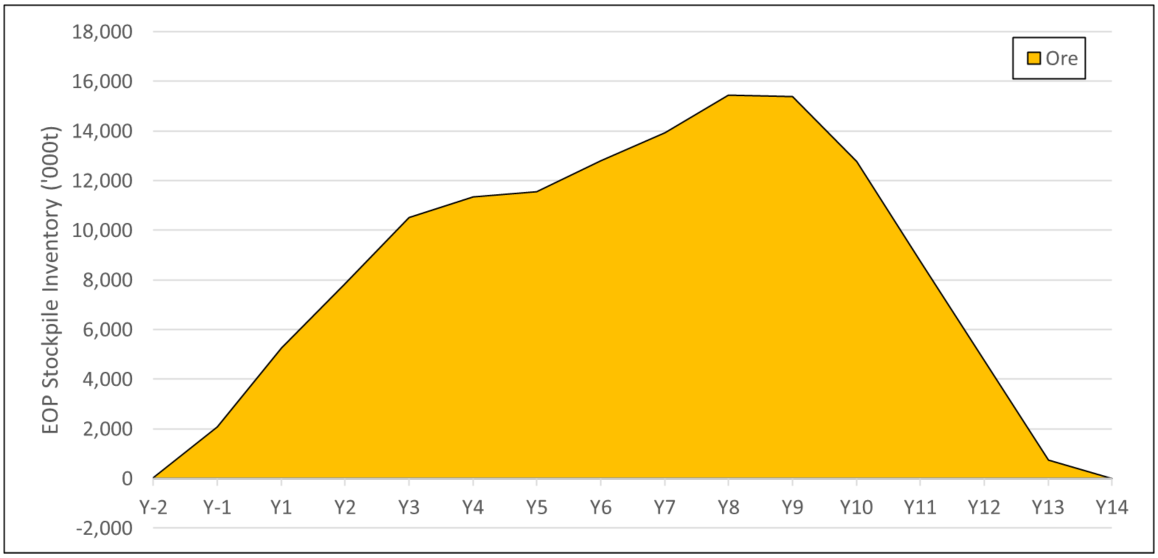
**Table 16.11: Detailed Milling Schedule**

| Description | Unit | Total | Y-2 | Y-1 | Y1 | Y2 | Y3 | Y4 | Y5 | Y6 | Y7 | Y8 | Y9 | Y10 | Y11 | Y12 | Y13 | Y14 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Ore Milled | Mt | **54** | - | 1 | 4 | 4 | 4 | 4 | 4 | 4 | 4 | 4 | 4 | 4 | 4 | 4 | 4 | 1 |
| Au Grade | g/t | **1.64** | 0 | 1.04 | 2.87 | 1.95 | 2.83 | 2.16 | 1.14 | 1.52 | 1.75 | 2.07 | 2.04 | 1.73 | 0.56 | 0.56 | 0.56 | 0.56 |
| Ag Grade | g/t | **7.27** | 0 | 4.51 | 14.55 | 11.75 | 13.88 | 9.81 | 6.29 | 7.36 | 6.08 | 5.32 | 5.02 | 4.96 | 3.66 | 3.66 | 3.66 | 3.66 |
| Au Recovered Oz | koz | **2,645** | 0 | 29 | 347 | 235 | 344 | 261 | 135 | 183 | 209 | 248 | 245 | 207 | 63 | 63 | 63 | 12 |
| Ag Recovered Oz | koz | **10,617** | 0 | 133 | 1,580 | 1,271 | 1,512 | 1,064 | 674 | 798 | 665 | 579 | 547 | 538 | 395 | 394 | 394 | 74 |
| Au Gross Rev. | US$M | **4,232** | 0.0 | 46.2 | 555.9 | 375.8 | 551.1 | 417.2 | 216.4 | 292.3 | 334.6 | 396.4 | 392.8 | 330.8 | 101.5 | 101.2 | 101.2 | 19.1 |
| Ag Gross Rev. | US$M | **212** | 0 | 2.7 | 31.6 | 25.4 | 30.2 | 21.3 | 13.5 | 16.0 | 13.3 | 11.6 | 10.9 | 10.8 | 7.9 | 7.9 | 7.9 | 1.5 |

Source: GMS (2021).

Figure 16-17 depicts the stockpile inventories by period. The peak stockpile capacity is approximately 15.4 Mt. All material is milled by the end of project life.

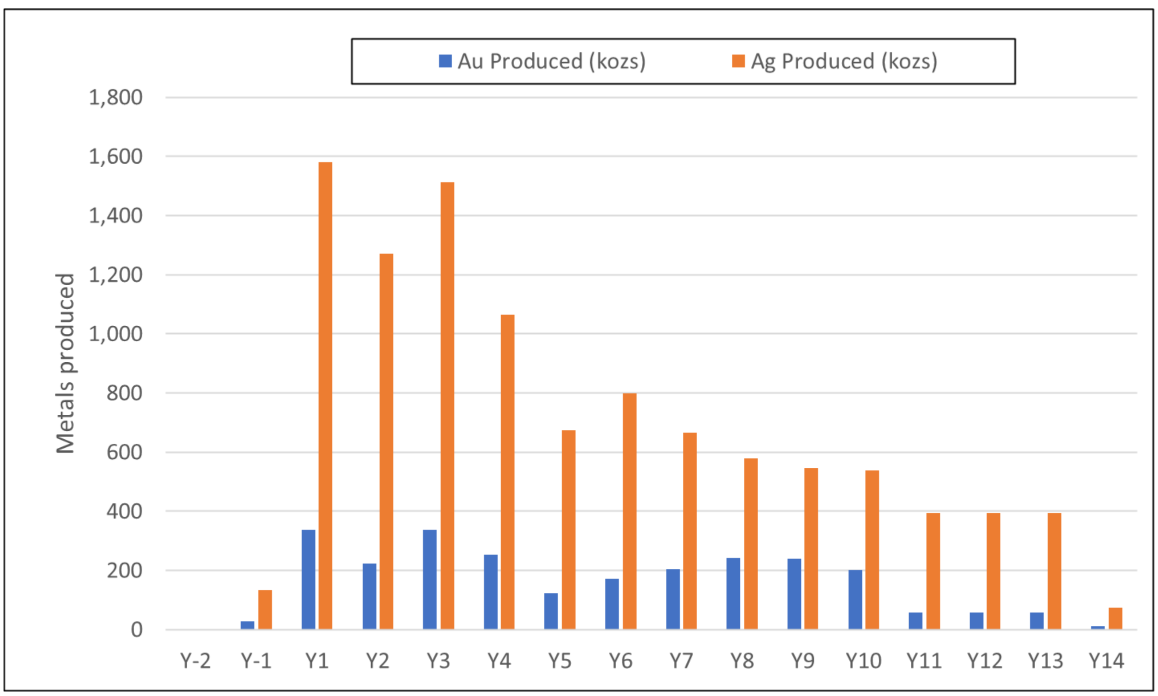
**Figure 16-17: Stockpile Inventory**




Source: GMS (2021).

Figure 16-18 depicts the metals produced by the mill each year. Over the mine life, a total of 2,645 koz of gold and 10,617 koz of silver will be recovered.

**Figure 16-18: Metals Produced**

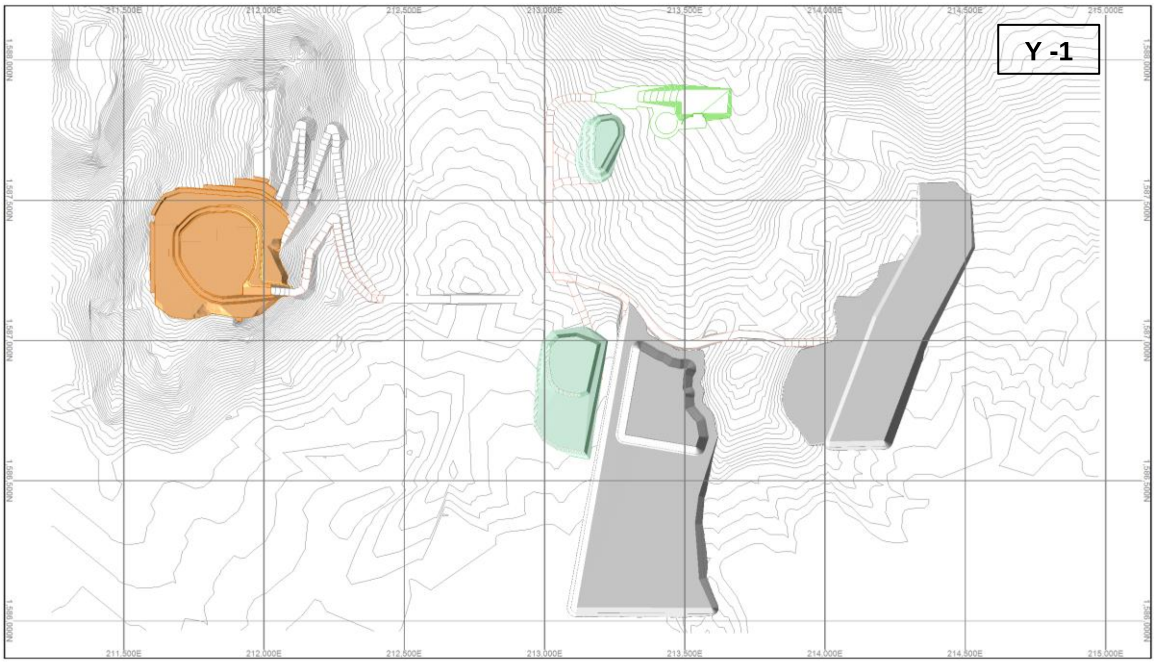


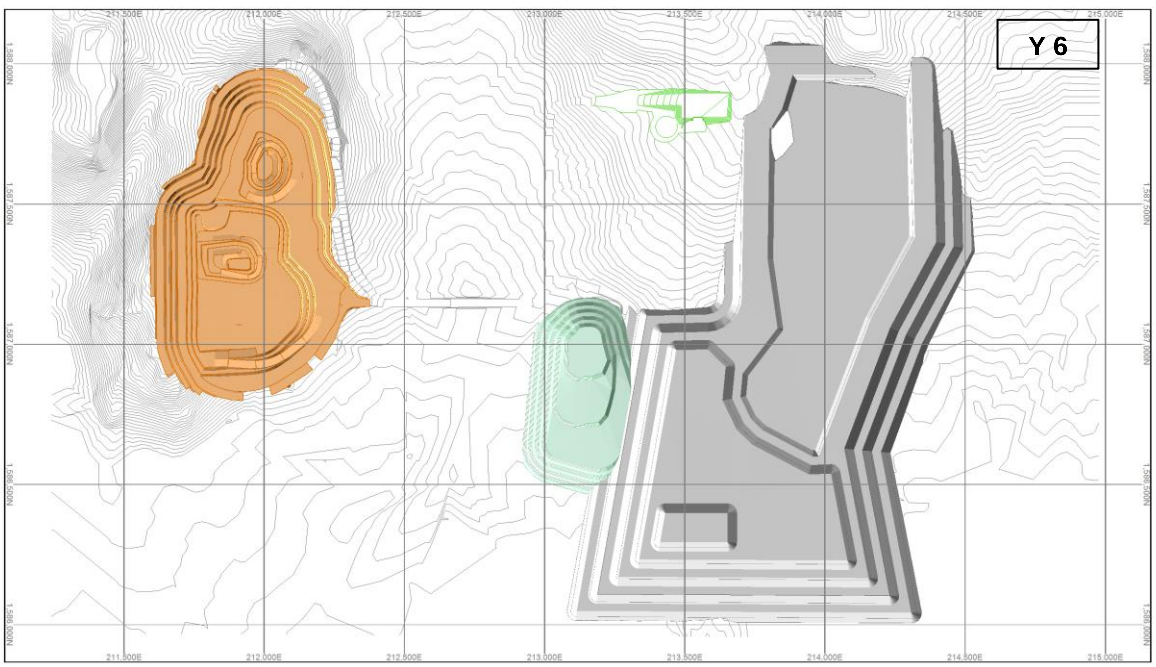

Source: GMS (2021).

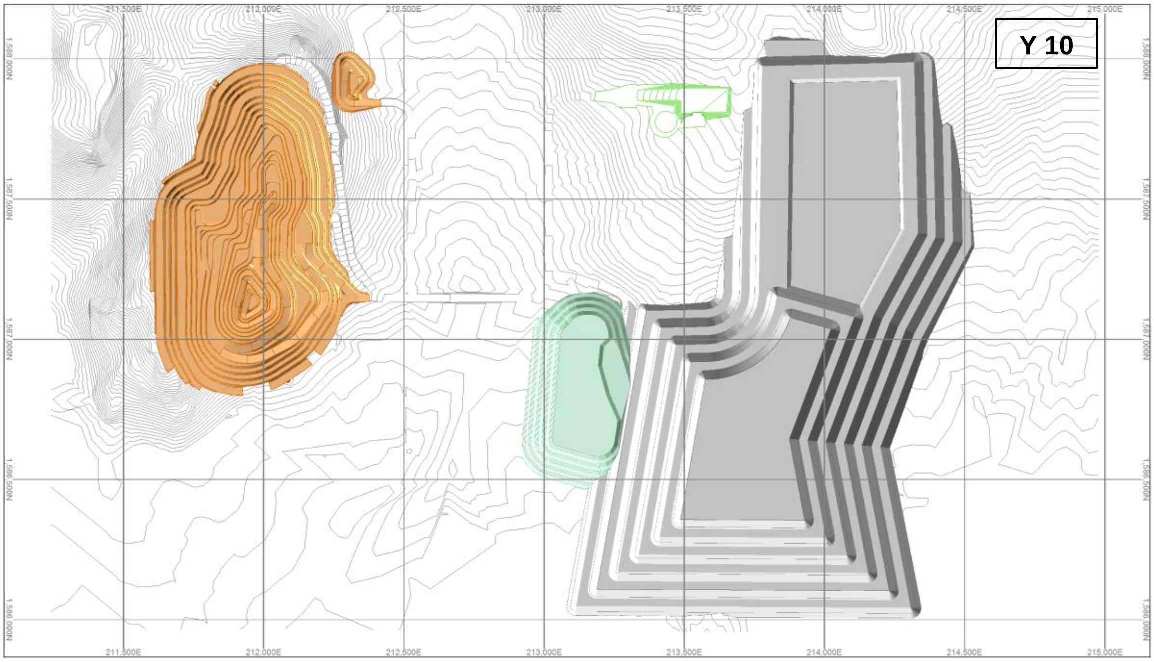
### 16.3.3 Surface Schedule

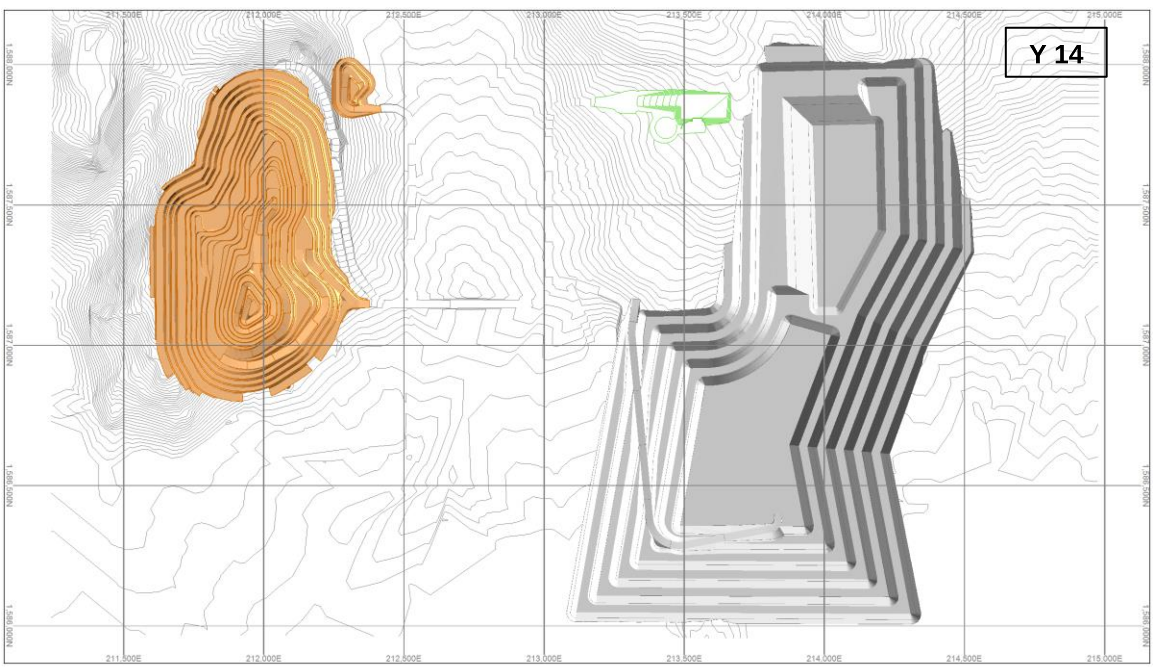
Figure 16-19 depicts the working surfaces of the mine at the specified years. This includes the mined material by period and the dump inventories. Year 14 depicts the end of mine life for the project prior to closure. These images do not include surface works or construction.

**Figure 16-19: End of Project Surface (Years -1, 6, 10, 14)**






Source: GMS (2021).

## 16.4 Mine Operations & Equipment Selection

### 16.4.1 Mine Operations Approach

Mining will be carried out using conventional open pit techniques with hydraulic shovels, wheel loaders, and mining trucks in a bulk mining approach with 10 m benches. An owner-mining open pit operation is planned with the outsourcing of certain support activities such as explosives manufacturing and blasting.

### 16.4.2 Production Drilling & Blasting

Drilling and blasting parameters are established to effectively drill a 10 m bench in a single pass. A 197 mm blast hole size is proposed with a 5.5 m x 6.3 m staggered pattern in ore and a 6.0 m x 6.9 m staggered pattern in waste, both patterns with 1.5 m of sub-drill. These drilling and blasting parameters combined with a high-energy bulk emulsion with a density of 1.15 kg/m$^3$ result in a powder factor of 0.29 kg/t in ore and 0.25 kg/t in waste.

Fifty percent of the blast holes are initiated with NONEL detonators and 50% are initiated with electronic detonators. Both types of initiation are primed with 450 g boosters.

The bulk emulsion product is a gas sensitized pumped emulsion blend specifically designed for use in wet blasting applications. When high temperatures are present in the blast hole (>65°C), high temperature explosives will be used.

A variety of rock types will be mined in the pit, including rhyolite, limestone, tuffs, sandstones and volcanic conglomerate. The average rock hardness varies between 49 and 187 MPa.

The average drill productivity for the production rigs, using down-the-hole drill string, is estimated at 37 m/h instantaneous with an overall penetration rate of 24 m/h on average. The overall drilling factor represents time lost in the cycle when the rig is not drilling, including move time between holes, moves between drill patterns, and drill bit changes. The average drilling productivity is estimated at 1,850 t/h in ore and 2,200 t/h in waste.

Table 16.12 summarizes the drill and blast parameters for ore and waste. The drill rig selected for production drilling will have a hole diameter range of 152 mm to 251 mm with a maximum single pass drill depth of 16.1 m with a 23.3 m tower configuration. This rig will have both rotary and percussion drilling capabilities. It is expected that the rotary drilling mode will be the most efficient.

Although not considered in the FS design, the selection of the drill rig included the possibility of an automatic drill rod loading system. The use of a drill automation system will be evaluated during the next phase.

**Table 16.12: Drill & Blast Parameters**

| Description | Unit | Ore | Waste |
|---|---|---|---|
| **Drill Pattern** | | | |
| KS: Spacing/Burden | | 1.15 | 1.15 |
| KB: Burden/Diameter | | 27.94 | 30.48 |
| KJ: Subdrill/Burden | | 0.27 | 0.25 |
| KT: Stemming/Burden | | 0.75 | 0.68 |
| KH: Height/Burden | | 1.82 | 1.67 |
| Explosive Density | $g/cm^3$ | 1.15 | 1.15 |
| Hole Diameter | in | 7.75 | 7.75 |
| Diameter (D) | m | 0.197 | 0.197 |
| Burden (B) | m | 5.5 | 6.0 |
| Spacing (S) | m | 6.3 | 6.9 |
| Subdrill (J) | m | 1.5 | 1.5 |
| Stemming (T) | m | 4.1 | 4.1 |
| Bench Height (H) | m | 10.0 | 10.0 |
| Blasthole Length (L) | m | 11.5 | 11.5 |
| **Pattern Yield** | | | |
| Rock Density | t/bcm | 2.54 | 2.54 |
| BCM/Hole | bcm/hole | 348 | 414 |
| Yield per Hole | t/hole | 884 | 1052 |
| Yield per Meter Drilled | t/m drilled | 77 | 91 |
| Explosive Column (LE) | m | 7.4 | 7.4 |
| Volume of Explosives/Hole | m3 | 0.23 | 0.23 |
| Weight of Explosives/Hole | kg | 259 | 259 |
| Powder Factor | kg/t | 0.29 | 0.25 |
| Loading Factor | kg/bcm | 0.74 | 0.63 |
| **Drill Productivity** | | | |
| Re-drills | % | 5.0% | 5.0% |
| Pure Penetration Rate | m/h | 37.0 | 37.0 |
| Overall Drilling Factor (%) | % | 0.65 | 0.65 |
| Overall Penetration Rate | m/h | 24.1 | 24.1 |
| Drilling Efficiency | t/h | 1,848 | 2,199 |
| Drilling Efficiency | holes/h | 2.09 | 2.09 |

Source: GMS (2021).

Blasting activities will be outsourced to a provider for supply and delivery of explosives through a service contract. The mine engineering department will be responsible for designing blast patterns and relaying hole information to the drills via a wireless network.

### 16.4.3 Grade Control

The ore control program will consist of establishing dig limits for ore and waste in the field to guide loading unit operators. A high-precision system combined with a stick-and-boom geometry system will allow shovels to target small dig blocks and perform selective mining. The system will give operators a real-time view of dig blocks, ore boundaries, and other positioning information.

For optimal ore-waste boundary identification, blasthole sampling will target 100% of all ore material and capture 100% of the total waste in the pit. An estimated 20% of the pit will be sampled through a reverse-circulation drilling program.

The ore control boundaries will be established by the technical services department based on grade control information obtained by blast hole sampling. Although the FS does not anticipate using a blast movement measurements system, it should be evaluated during the pre-production period. This could allow for better grade control using post-blast boundaries adjusted for blast movement measurements made using a BMM® system.

Samples collected will be sent to the site laboratory for sample preparation and assaying. Samples will be collected on the bench and tagged by grade control samplers on each shift.

### 16.4.4 Pre-Split & Depressurization holes

Pre-split drill and blast is planned to maximize stable bench faces and to maximize inter-ramp angles along pit walls, as prescribed by the geotechnical pit slope study by E-Mining Technology. The pre-split consists of a row of closely spaced holes along the design excavation limit of interim and final walls. The holes shall be loaded with a light charge and detonated simultaneously or in groups separated by short delays. Firing the pre-split row creates a crack that forms the excavation limit and helps to prevent wall rock damage by venting explosive gases and reflecting shock waves. As a best practice, it is recommended that operations restrict production blasts to within 50 m of a non-blasted pre-shear line. Once the pre-split is shot, production blasts will be taken to within 10 m of the pre-shear followed by a trim shot used to clean the face. Pre-split holes spaced 2 m apart will be 20 m in length and drilled with a smaller diameter of 165 mm (6.5 in).

As presented in Table 16.13, blasting of the pre-split holes will use a special packaged pre-split explosive internally traced with 5 g/m detonating cord that ensures fast and complete detonation of the decoupled charge. For this specific application, a string of 410 mm long cartridges will be used, which corresponds to a complete case of 25 kg. This load factor of 1.47 kg/m allows for a targeted charge weight of 0.60 kg/m$^2$ of face.

The drill selected for this application is more flexible type of rig capable of drilling angled holes for probe drilling and pit wall drain holes. The hole size range of this rig is between 110 mm and 229 mm with a maximum hole depth of 56 m.

Given that the deposit is in a geothermal area, high-pressure pockets in some areas of the mine are expected. To reduce risk, some 12,600 m of depressurization holes will be drilled per year before accessing the high-pressure zones for mining.

**Table 16.13: Pre-Split Parameters**

| Description | Units | Pre-Split Holes |
|---|---|---|
| **Drill Pattern** | | |
| Hole Diameter | in | 6.5 |
| Diameter (D) | m | 0.1651 |
| Spacing (S) | m | 2.0 |
| Stemming (T) | m | 3.7 |
| Bench Height (H) | m | 20 |
| Blasthole Length (L) | m | 20.7 |
| Face Area | $m^2$ | 41.4 |
| Explosive Charge | kg | 25 |
| Charge Factor | $kg/m^2$ face | 0.60 |
| **Cartridge Charge** | | |
| Nb Cartridges | Qty | 41 |
| Cartridge Length | m | 0.41 |
| Cartridge Loading Factor | kg/m | 1.47 |
| Decoupled Charge Length | m | 17.00 |
| Decoupled Charge | kg | 25.0 |
| **Toe Charge** | | |
| Explosive Column (LE) | m | 0 |
| Volume of Explosives/Hole | $m^3$ | 0 |
| Weight of Explosives/Hole | kg | 0 |
| **Blasting** | | |
| Packaged Pre-Split Explosive | kg | 25.0 |
| Surface Delay NONEL | unit | 1.00 |
| Emulsion Toe Charge | kg | 0 |
| Booster 400 g. | unit | 0 |
| Detonating Cord | m | 5 |
| Explosives Product Cost | $/hole | 196.33 |
| **Drill Productivity** | | |
| Pure Penetration Rate | m/h | 21.0 |
| Overall Drilling Factor (%) | % | 0.35 |
| Overall Penetration Rate | m/h | 7.4 |
| Drilling Efficiency | holes/h | 0.4 |
| Meters of Drilling per m Crest | m/m of crest | 10.35 |

Source: GMS (2021).

### 16.4.5 Loading

The loading productivity assumptions for both types of loading equipment are presented in Table 16.14.

**Table 16.14: Loading Specifications**

| | | Rock | Rock |
|---|---|---|---|
| **Loading Unit** | | **Diesel Hydraulic Excavator (15 m³) Komatsu PC3000** | **Wheel Loader (11 m³) Komatsu WA900** |
| **Haulage Unit** | | **Mining Haul Truck (90 t) Komatsu HD785** | **Mining Haul Truck (90 t) Komatsu HD785** |
| Rated Truck Payload | t | 92 | 92 |
| Heaped Tray Volume | $m^3$ | 60 | 60 |
| Bucket Capacity | $m^3$ | 14.5 | 11.5 |
| Bucket Fill Factor | % | 90% | 85% |
| In-Situ Dry Density | t/bcm | **2.60** | **2.60** |
| Moisture | % | 3% | 3% |
| Swell | % | 35% | 35% |
| Wet Loose Density | t/lcm | 1.98 | 1.98 |
| Actual Load Per Bucket | t | 25.89 | 19.39 |
| Passes (Decimal) | # | 3.54 | 4.73 |
| Passes (Whole) | # | 3.50 | 4.50 |
| Actual Truck Wet Payload | t | 91 | 87 |
| Actual Truck Dry Payload | t | 88 | 85 |
| Actual Heaped Volume | $m^3$ | 46 | 44 |
| Payload Capacity | % | 99% | 95% |
| Heaped Capacity | % | 76% | 73% |
| **Cycle Time** | | | |
| Hauler Exchange | min | 0.60 | 0.70 |
| First Bucket Dump | min | 0.10 | 0.10 |
| Average Cycle Time | min | 0.67 | 0.77 |
| Load Time | min | 2.37 | 3.51 |
| Cycle Efficiency with Wait Time | % | 65% | 65% |
| No. of Trucks Loaded per Hour | # | 16.5 | 11.12 |
| **Production / Productivity** | | | |
| Productivity Dry Tonnes / Operating Hour | t/h | 1,450 | 942 |
| Effective Hours per Year | hours/y | 5,447 | 5,304 |
| Dry Annual Production Capacity | kt/y | 7,896,605 | 4,997,056 |
| Number of Units | # | 2 | 2 |
| Tonnes | t/y | 15,793,211 | 9,994,112 |
| | | | |
| Productivity Dry Tonnes / Operating Hour | t/h | 1,450 | 942 |
| Hour per Shift | h/shift | 10.00 | 10.00 |
| Number of Shift per Day | shift/day | 2.00 | 2.00 |
| Number of Day in Period | days | 30.00 | 30.00 |
| Hour per Period | h/period | 600 | 600 |
| Number of Units | # | 1.00 | 1.00 |
| Number of Units | t/y | 869,755 | 565,315 |

Source: GMS (2021).

Loading in the pit will be performed by two 15 $m^3$ face shovels and two 11 $m^3$ wheel loaders. The loading units will be matched with a fleet of 90-tonne payload capacity mine haul trucks.

The hydraulic shovels will be loading 100% of the ore. The rest of their loading availability will be used to load waste. The wheel loaders will primarily be engaged with stockpile rehandling activities while assisting the shovels in waste. The two 15 $m^3$ face shovels are expected to achieve a productivity of 1,450 t/h based on a 3.5-pass match with the mine trucks and an average load time of 2.37 minutes.

The wheel loaders are expected to achieve a productivity of 942 t/h based on a 4.5-pass match and an average load time of 3.51 minutes in ore and waste.

### 16.4.6 Hauling

Haulage of ore and waste will be performed with a fleet of 90-tonne class haul trucks. Cycle times have been estimated for each period and all possible destinations given there are several storage area destinations.

#### 16.4.6.1 Haulage Site Inputs & Assumptions

The typical equipment usage model assumptions are established by equipment groupings as presented in Table 16.15. The annual net operating hours varies approximately between 5,000 and 6,000 hours per year. The overall equipment effectiveness is the gold standard for measuring mobile equipment productivity. It defines the percentage of equipment operating time that is truly productive.

**Table 16.15: Equipment Usage Model Assumptions**

| Parameter | Unit | Excavators | Loaders | Trucks | Drills | Ancillary | Support |
|---|---|---|---|---|---|---|---|
| Days in Period | days | 365 | 365 | 365 | 365 | 365 | 365 |
| Weather, Schedule Outages | days | 5 | 5 | 5 | 5 | 5 | 5 |
| Shifts per Day | per | 2 | 2 | 2 | 2 | 2 | 2 |
| Hours per shift | per | 12 | 12 | 12 | 12 | 12 | 12 |
| Availability | % | 85 | 85 | 89 | 85 | 80 | 85 |
| Use of Availability | % | 80 | 88 | 85 | 73 | 50 | 80 |
| Utilization | % | 68 | 75 | 76 | 62 | 40 | 68 |
| Effectiveness | % | 80 | 87 | 90 | 65 | 80 | 80 |
| Overall Equipment Effectiveness | % | 54 | 65 | 68 | 40 | 32 | 54 |
| Total Hours | h | 8,760 | 8,760 | 8,760 | 8,760 | 8,760 | 8,760 |
| Scheduled Hours | h | 8,640 | 8,640 | 8,640 | 8,640 | 8,640 | 8,640 |
| Down Hours | h | 1,314 | 1,314 | 966 | 1,318 | 876 | 1,318 |
| Delay Hours | h | 1,191 | 854 | 665 | 1,913 | 350 | 1,195 |
| Standby Hours | h | 1,489 | 896 | 1,173 | 2,000 | 1,752 | 1,493 |
| Operating Hours | h | 5,957 | 6,570 | 6,645 | 5,467 | 1,752 | 5,973 |
| Ready Hours | h | 4,765 | 5,716 | 5,981 | 3,553 | 1,402 | 4,778 |

Source: GMS (2021).

#### 16.4.6.2 Haulage Site Inputs & Assumptions

The assumptions and input factors for the simulations can be found in Tables 16.16 to 16.18. Two speed limits were applied in the simulation. For all downhill ramps with an incline greater than 5%, the speed was limited to 30 km/h; otherwise, the maximum truck speed was set at 50 km/h.

**Table 16.16: Site Speed Limits**

| Description | Loaded Speed (km/hr) | Empty Speed (km/hr) |
|---|---|---|
| In Pit | 25 | 27 |
| Out of Pit | 35 | 40 |
| Going Up | 12 | 20 |
| Going Down | 22 | 30 |

Source: GMS (2021).

**Table 16.17: Site Rolling Resistance Assumptions**

| Description | Rolling Resistance (%) |
|---|---|
| Main Road | 2.0 |
| Pit Ramp | 2.0 |
| Dump | 2.0 |
| Pit Floor | 2.0 |

Source: GMS (2021).

**Table 16.18: Fixed Cycle Time Components**

| Description | Time (mins) |
|---|---|
| **Hydraulic Shovels** | |
| Queue Time | 0.50 |
| Loading and Spotting Time | 2.37 |
| Dumping and Spotting time | 2.00 |
| Total Loading | **4.87** |
| **Loaders** | |
| Queue Time | 0.50 |
| Loading and Spotting Time | 3.51 |
| Dumping and Spotting time | 2.00 |
| Total Loading | **6.01** |

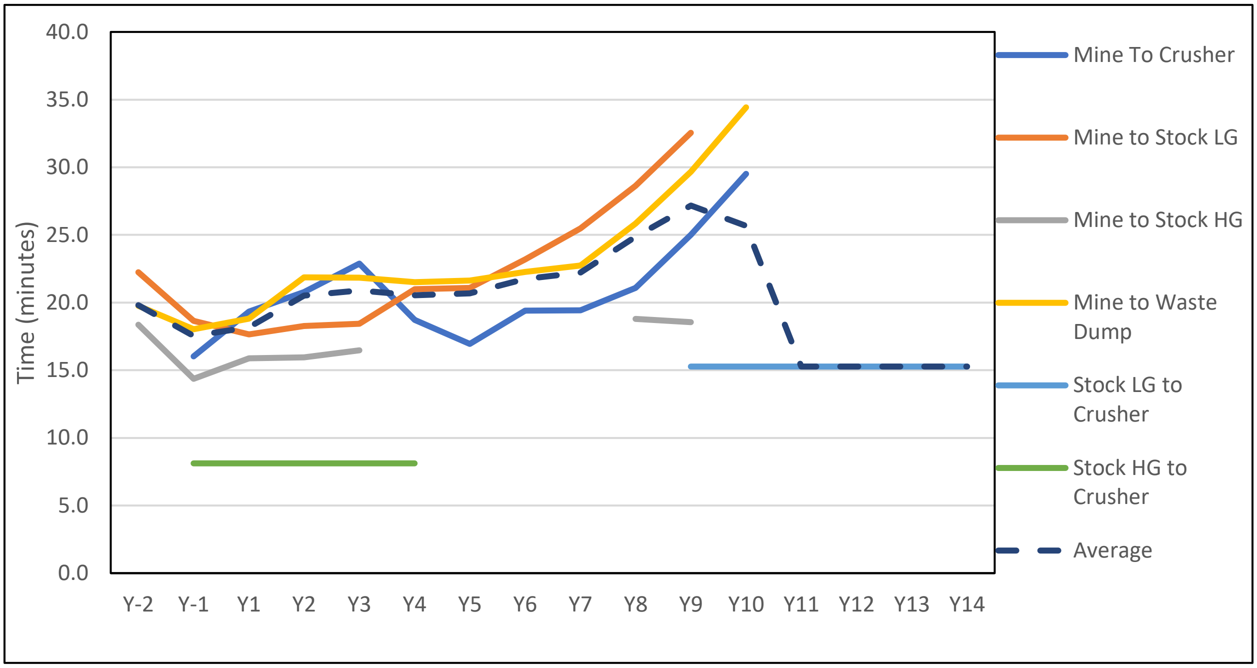
Source: GMS (2021).

#### 16.4.6.3 Haulage Simulation

A multiple waste dumps strategy was used to help level the truck requirements for the project. The closest waste dump is filled first before starting to fill the second waste dump since the cycle time is longer.

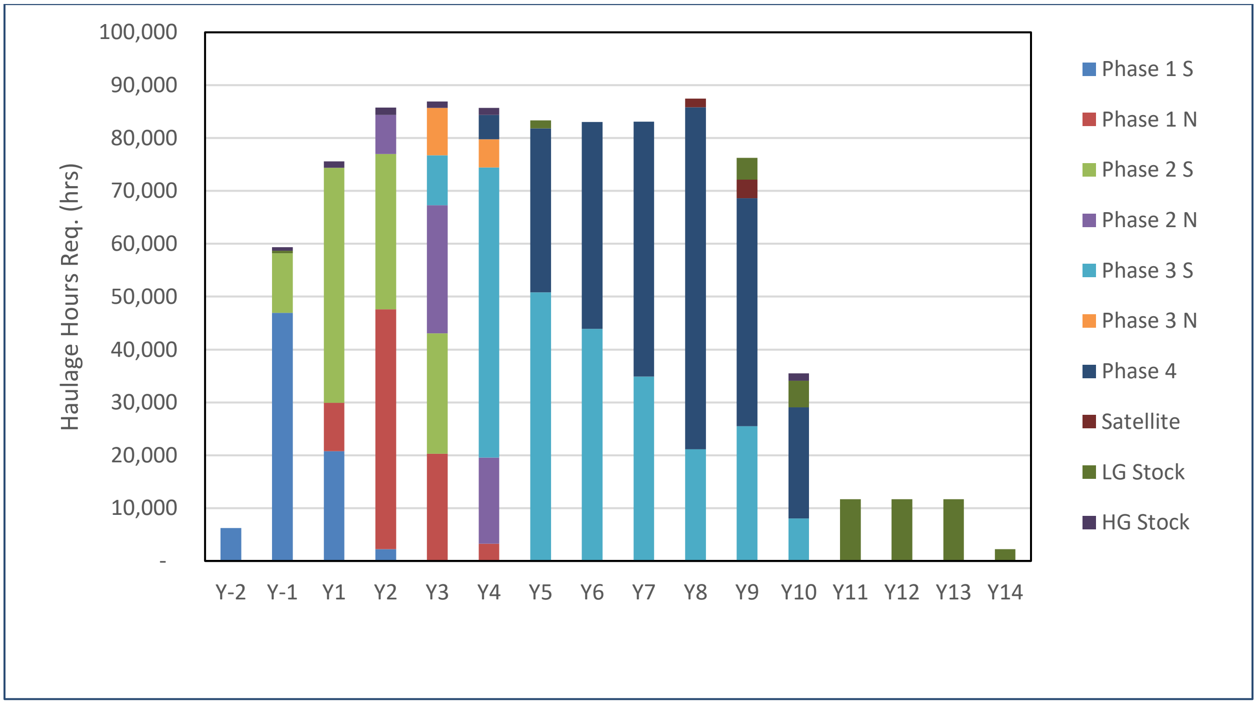
Figure 16-20 shows the trucks cycle times by source and destination, while Figure 16-21 summarizes the haulage hours by source.

**Figure 16-20: Truck Cycle Times by Source & Destination**




Source: GMS (2021)

**Figure 16-21: Haulage Hours by Phase**

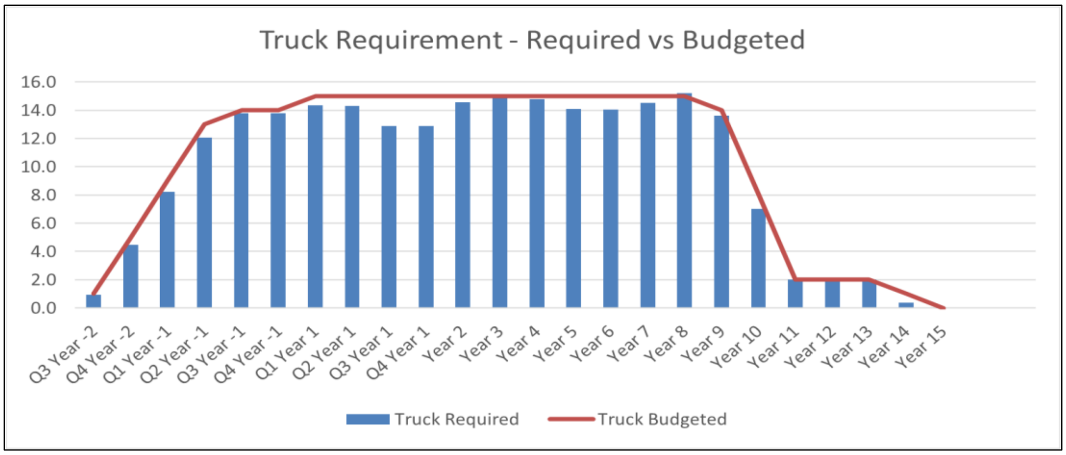



Source: GMS (2021).

Cycle time typically increases with depth of the pit over the mine life. Cycle time is also dependant on the dumping schedule and the distance each dump is from the pit. The dump schedule was planned so that cycle times have a tendency to plateau at a maximum limit to allow for a consistent fleet over most of the mine life. Large variation in cycle time between years from the same source represents material being mined at a deeper depth with a corresponding new cycle time.

The total haul hours required by period coupled with the truck mechanical availability were used to determine the number of trucks required throughout the life of mine. The truck fleet reaches a maximum of 15 units in Year 2 and remains at this level until Year 8 when truck requirements decrease with a reduced mining rate. Figure 16-22 summarizes the truck requirements.

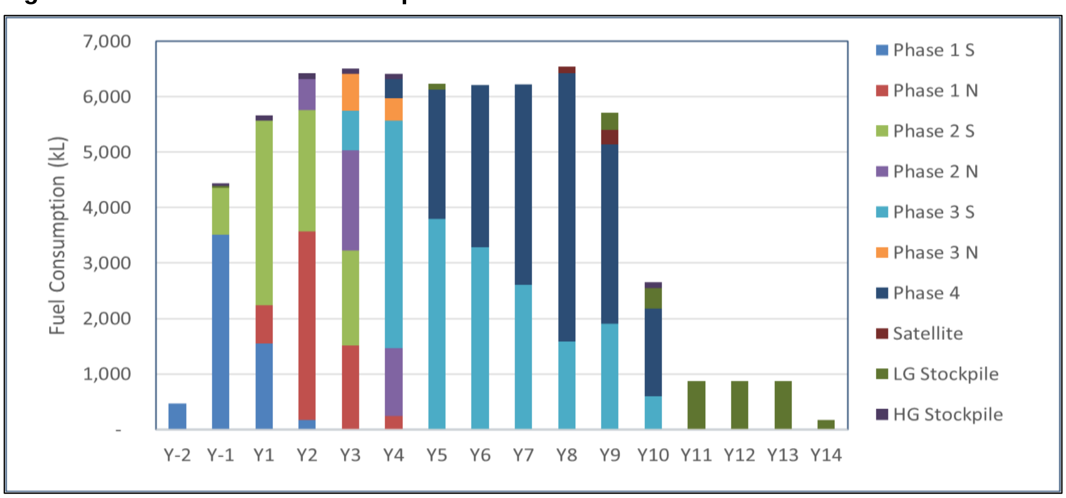
**Figure 16-22: Truck Requirements**




Source: GMS (2021).

Figure 16-23 depicts the fuel usage by year. Over the life of mine, a total of 67 ML of fuel will be consumed by the haulage fleet.

**Figure 16-23: Estimated Fuel Requirements**




Source: GMS (2021).

### 16.4.7 Mine Dewatering

Eight dewatering wells will be drilled during the life of mine to lower the water table during mining operations. Rainfall water and infiltration groundwater not pumped by the dewatering wells will be handled by an in-pit dewatering system that consists of two levels of fixed electric submersible pumps and four mobile diesel pumps located at the bottom of the pit. Each submersible pump will be connected to the electric grid and will have its own 70 $m^3$ water tank. The total pumping capacity of the system will be 350 $m^3/h$. The water will be pumped directly to the water treatment plant located near to the processing plant.

### 16.4.8 Road & Dump Maintenance

Waste and mill feed storage areas will be maintained by a fleet of two 354 hp wheel dozers and three 460 hp track dozers.

Mine roads will be maintained by two 14 ft blade motor graders. Four water trucks will be used to spray roads to suppress dust during dry months. Wheel dozers will also be used for road maintenance activities.

### 16.4.9 Support Equipment

All construction-related work, such as berm construction and water ditch cleaning, will be done by two 49-tonne excavators, of which one will be equipped with a hydraulic hammer.

Two pit buses will transport workers to their assigned workplace and 15 pick-up trucks will be purchased for the mining and maintenance department.  Several other equipment purchases are planned to support mining activities. The list of equipment includes a wheel backhoe loader and a boom-truck (28 tonnes).

### 16.4.10 Mine Maintenance

The project does not intend to enter a maintenance-and-repair contract (MARC) for its mobile equipment fleet. Consequently, the maintenance department has been structured to be self-sufficient by performing maintenance planning and training employees. However, reliance on dealer and manufacturer support will be key for the initial years of the project, and major components rebuilds will be supported by the original equipment manufacturers' (OEMs) dealer throughout the life of mine.

Tire monitoring, rotation, and/or replacement will be done in-house, and a tire handler truck has been planned as part of the maintenance equipment fleet. Some large equipment, such as two fuel and lube trucks, a dedicated lube truck, and a forklift, will be purchased for maintenance activities and to support the operation. The purchase of some small equipment, such as tower lights, welding machines, and/or portable air compressors, will also be required.

A computerized maintenance management system will be used to plan, schedule and manage maintenance and repair operations. This system will keep up-to-date service history, maintenance needs, the status of each equipment and provide data for key performance indicators (KPIs) and cost-tracking purposes.

### 16.4.11 Roster Schedules

At this stage of the study, a rotating schedule of 14 days on and 7 days off has been planned based on 12-hour shifts by hourly employees (direct labour). Three crews are required to operate on a continuous basis 24 hours per day, 365 days per year.

For expatriate positions, a rotating schedule of 30 days on and 20 days off, working 12 hours per day, has been planned. Some local staffing positions (e.g., engineers and geologists) are scheduled to work a rotation of 7 days on and 7 days off, working 12 hours per day.

### 16.4.12 Fleet Management

A fleet management system will be implemented from Year -1 to manage the operation, monitor machine health, and track KPIs. The system will be managed by a dispatcher on each crew who will control the system by sending operators onscreen instructions to work at peak efficiency. A dispatch system coordinator will be required to assure proper functioning of system hardware and software with ongoing annual vendor support.

### 16.4.13 Mine Equipment Requirements

The main factors that influenced the selection of the major mine equipment include the annual production requirements and the equipment capacity size. An analysis was performed to determine the optimal fleet size and equipment type. Table 16.19 presents the equipment requirement schedule, Table 16.20 presents the equipment purchase schedule, while the fleet size for major mining units is presented in Figure 16-24.

**Table 16.19: Equipment Requirement Schedule**

| Equipment Purchase Schedule | Max | Y-3 | Y-2 | Y-1 | Y1 | Y2 | Y3 | Y4 | Y5 | Y6 | Y7 | Y8 | Y9 | Y10 | Y11 | Y12 | Y13 | Y14 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| **Major Equipment** | | | | | | | | | | | | | | | | | | |
| Mining Haul Truck (90 t) | **15** | - | 5 | 14 | 15 | 15 | 15 | 15 | 15 | 15 | 15 | 15 | 14 | 8 | 2 | 2 | 2 | 1 |
| Diesel Hydraulic Excavator (15 m³) | **2** | - | 1 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 1 | - | - | - | - |
| Wheel Loader (11 m³) | **2** | - | - | 1 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 1 | 1 | 1 | 1 | 1 |
| Production Drill (6-10") | **2** | - | 1 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 1 | 1 | - | - | - | - |
| Grade Control Drill (4-6") | **1** | - | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | - | - | - | - |
| Auxiliary Pre-split Drill (4.5-8") | **2** | - | - | - | 1 | 1 | 1 | 1 | 1 | 1 | 2 | 2 | 2 | 2 | - | - | - | - |
| Track Dozer (460 hp) | **3** | - | 1 | 3 | 3 | 3 | 3 | 3 | 3 | 3 | 3 | 3 | 2 | 1 | 1 | 1 | 1 | 1 |
| Wheel Dozer (354 hp) | **2** | - | 1 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 1 | - | - | - | - |
| Motor Grader (14 ft) | **2** | - | 1 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 1 | 1 | 1 | 1 |
| Water Truck (15 kL tank) | **4** | - | 2 | 4 | 4 | 4 | 4 | 4 | 4 | 4 | 4 | 4 | 3 | 2 | 1 | 1 | 1 | 1 |
| **Total Maximum Major Equipment** | **35** | **-** | **13** | **31** | **34** | **34** | **34** | **34** | **34** | **34** | **35** | **35** | **31** | **20** | **6** | **6** | **6** | **5** |
| Support Equipment | | | | | | | | | | | | | | | | | | |
| Excavator (49 t) | **2** | - | 1 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 1 | 1 | 1 | 1 | 1 |
| Tire Handler Truck | **1** | - | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 |
| Lube Truck | **1** | - | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 |
| Fuel & Lube Truck 10 Wheel | **2** | - | 1 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 1 | 1 | 1 | 1 | 1 |
| Mechanic Service Truck | **2** | - | 1 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 1 | 1 | 1 | 1 | 1 |
| Boom Truck (28 t) | **1** | - | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 |
| Forklift (2.5 t) | **1** | - | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 |
| Telehandler (4.0 t) | **1** | - | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 |
| Mobile Air Compressor (185 CFM) | **1** | - | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 |
| Light Plant | **10** | - | 3 | 10 | 10 | 10 | 10 | 10 | 10 | 9 | 9 | 8 | 7 | 4 | 2 | 2 | 2 | 1 |
| Genset (6 kW) | **3** | - | 1 | 3 | 3 | 3 | 3 | 3 | 3 | 3 | 3 | 3 | 2 | 1 | 1 | 1 | 1 | 1 |
| Welding Machine Electric | **2** | - | 1 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 1 | 1 | 1 | 1 | 1 |
| Welding Machine Diesel (400 A) | **2** | - | 1 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 1 | 1 | 1 | 1 | 1 |
| Welding Machine Diesel (500 A) AIR | **1** | - | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 |
| Water Pump (6"x3") - Gasoline | **8** | - | 2 | 4 | 6 | 6 | 6 | 8 | 8 | 8 | 8 | 8 | 8 | 8 | - | - | - | - |
| Water Pump (8") - Electric | **4** | - | - | - | 2 | 2 | 2 | 2 | 2 | 2 | 4 | 4 | 4 | 4 | - | - | - | - |
| Pit Bus | **5** | - | 1 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 1 | 1 | 1 | 1 |
| Pick-up Truck | **15** | 2 | 8 | 14 | 15 | 15 | 15 | 15 | 14 | 13 | 13 | 12 | 10 | 5 | 3 | 3 | 3 | 1 |
| Emulsion Truck | **1** | - | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 |
| Stemming Loader | **1** | - | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 |
| Spare Bucket for Excavator | **-** | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - |
| Spare Bucket for Loaders | **-** | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - |
| Spare Box for Haul Trucks | **-** | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - |
| Spare Bucket for Small Excavator | **-** | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - |
| Dispatch System | **1** | - | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 |
| FMS Hardware in Mobile Equipment | **1** | - | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 |
| Communication Radio Package | **1** | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 |
| Survey Tools | **2** | - | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | - | - | - | - |
| Slope Monitoring System | **-** | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - |
| **Total (Excluding Pipe)** | **66** | **3** | **34** | **57** | **62** | **62** | **62** | **64** | **63** | **61** | **63** | **61** | **57** | **42** | **22** | **22** | **22** | **19** |

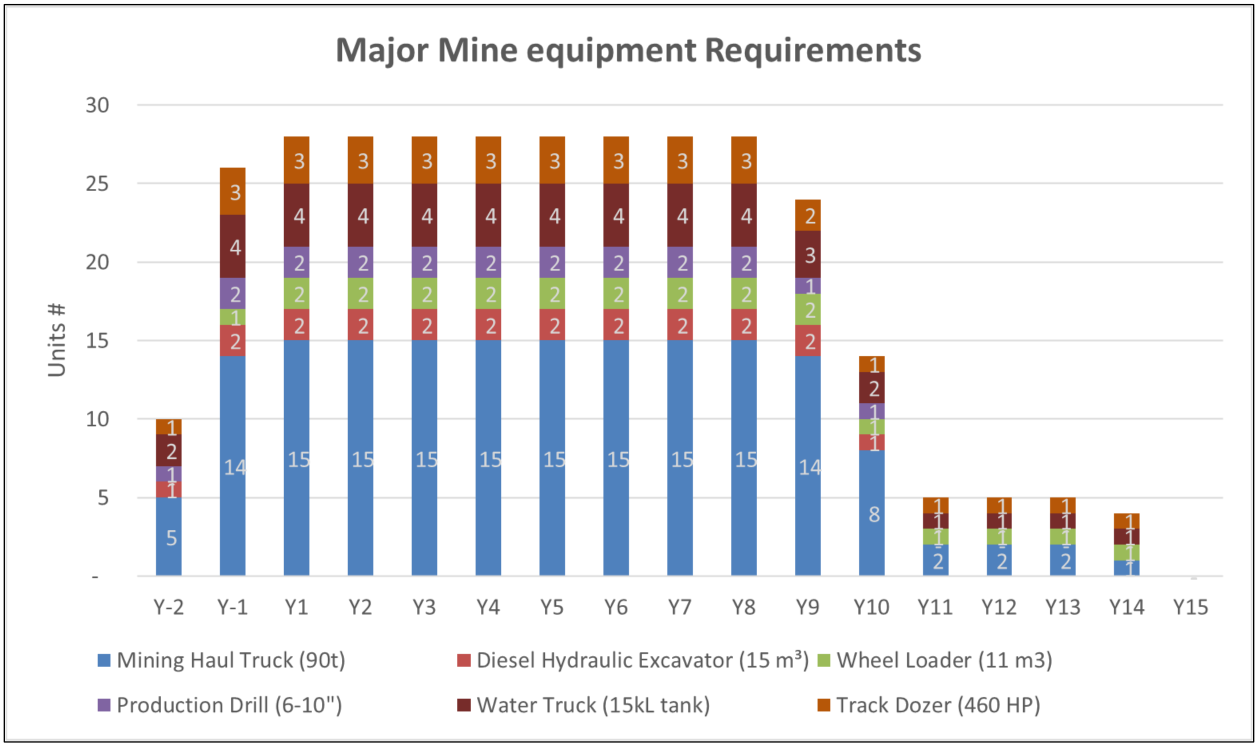
Source: GMS (2021).

**Table 16.20: Equipment Purchase Schedule**

| Equipment Purchase Schedule | Total | Y-3 | Y-2 | Y-1 | Y1 | Y2 | Y3 | Y4 | Y5 | Y6 | Y7 | Y8 | Y9 | Y10 | Y11 | Y12 | Y13 | Y14 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| **Major Equipment** | | | | | | | | | | | | | | | | | | |
| Mining Haul Truck (90 t) | **17** | - | 7 | 7 | 1 | - | - | - | - | - | - | - | - | 1 | 1 | - | - | - |
| Diesel Hydraulic Excavator (15 m³) | **2** | - | 2 | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - |
| Wheel Loader (11 m³) | **2** | - | - | 1 | 1 | - | - | - | - | - | - | - | - | - | - | - | - | - |
| Production Drill (6-10") | **2** | - | 1 | 1 | - | - | - | - | - | - | - | - | - | - | - | - | - | - |
| Grade Control Drill (4-6") | **2** | - | 1 | - | - | - | - | - | 1 | - | - | - | - | - | - | - | - | - |
| Auxiliary Pre-split Drill (4.5-8") | **2** | - | - | - | 1 | - | - | - | - | 1 | - | - | - | - | - | - | - | - |
| Track Dozer (460 hp) | **7** | - | 2 | 1 | - | - | 1 | 2 | - | - | - | - | - | 1 | - | - | - | - |
| Wheel Dozer (354 hp) | **4** | - | 2 | - | - | - | 1 | 1 | - | - | - | - | - | - | - | - | - | - |
| Motor Grader (14 ft) | **6** | - | 1 | 1 | - | - | 1 | 1 | - | - | - | 1 | 1 | - | - | - | - | - |
| Water Truck (15 kL tank) | **11** | - | 2 | 2 | - | - | 3 | 1 | - | - | 3 | - | - | - | - | - | - | - |
| **Total Major Equipment** | **55** | **-** | **18** | **13** | **3** | **-** | **6** | **5** | **1** | **1** | **3** | **1** | **1** | **2** | **1** | **-** | **-** | **-** |
| Support Equipment | | | | | | | | | | | | | | | | | | |
| Excavator (49 t) | **5** | - | 1 | 1 | - | - | 1 | 1 | - | - | - | 1 | - | - | - | - | - | - |
| Tire Handler Truck | **2** | - | 1 | - | - | - | - | - | - | 1 | - | - | - | - | - | - | - | - |
| Lube Truck | **2** | - | 1 | - | - | - | - | - | 1 | - | - | - | - | - | - | - | - | - |
| Fuel & Lube Truck 10 Wheel | **5** | - | 1 | 1 | - | - | - | - | 2 | - | - | - | - | - | 1 | - | - | - |
| Mechanic Service Truck | **4** | - | 1 | 1 | - | - | - | - | 1 | 1 | - | - | - | - | - | - | - | - |
| Boom Truck (28 t) | **1** | - | 1 | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - |
| Forklift (2.5 t) | **2** | - | 1 | - | - | - | - | - | 1 | - | - | - | - | - | - | - | - | - |
| Telehandler (4.0 t) | **3** | - | 1 | - | - | - | - | 1 | - | - | - | - | 1 | - | - | - | - | - |
| Mobile Air Compressor (185 CFM) | **7** | - | 1 | - | 1 | - | 1 | - | 1 | - | 1 | - | 1 | - | 1 | - | - | - |
| Light Plant | **35** | - | 5 | 5 | - | 3 | 7 | - | 2 | 7 | - | - | 4 | - | - | 2 | - | - |
| Genset (6 kW) | **18** | - | 2 | 1 | 3 | - | 3 | - | 3 | - | 3 | - | 1 | - | 1 | - | 1 | - |
| Welding Machine Electric | **7** | - | 1 | 1 | - | - | 2 | - | - | 2 | - | - | 1 | - | - | - | - | - |
| Welding Machine Diesel (400 A) | **9** | - | 1 | 1 | - | 2 | - | - | 2 | - | - | 2 | - | - | - | 1 | - | - |
| Welding Machine Diesel (500 A) AIR | **5** | - | 1 | - | - | 1 | - | - | 1 | - | - | 1 | - | - | - | 1 | - | - |
| Water Pump (6"x3") - Gasoline | **22** | - | 2 | 4 | - | 4 | 4 | - | - | 4 | 4 | - | - | - | - | - | - | - |
| Water Pump (8") - Electric | **6** | - | - | 2 | - | - | - | - | 2 | 2 | - | - | - | - | - | - | - | - |
| Pit Bus | **5** | - | 1 | 1 | - | - | - | 2 | - | - | - | - | 1 | - | - | - | - | - |
| Pick-up Truck | **25** | 2 | 6 | 7 | - | - | - | - | - | 2 | 7 | 1 | - | - | - | - | - | - |
| Emulsion Truck | **1** | - | 1 | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - |
| Stemming Loader | **1** | - | 1 | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - |
| Spare Bucket for Excavator | **1** | 1 | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - |
| Spare Bucket for Loaders | **1** | 1 | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - |
| Spare Box for Haul Trucks | **2** | 2 | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - |
| Spare Bucket for Small Excavator | **1** | 1 | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - |
| Dispatch System | **1** | - | 1 | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - |
| FMS Hardware in Mobile Equipment | **1** | - | 1 | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - |
| Communication Radio Package | **1** | 1 | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - |
| Survey Tools | **2** | - | 2 | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - |
| Slope Monitoring System | **1** | - | - | 1 | - | - | - | - | - | - | - | - | - | - | - | - | - | - |
| **Total (Excluding Pipe)** | **176** | **8** | **34** | **26** | **4** | **10** | **18** | **4** | **16** | **19** | **15** | **5** | **9** | **-** | **3** | **4** | **1** | **-** |

Source: GMS (2021).

**Figure 16-24: Equipment Requirements**




Source: GMS (2021).

### 16.4.14 Mine Workforce Requirements

Table 16.21 presents the mine workforce requirements over the life of mine. The mine workforce peaks at 263 individuals in Years 1 and 2.

**Table 16.21: Workforce Requirements**

| Department | Max | Y-3 | Y-2 | Y-1 | Y1 | Y2 | Y3 | Y4 | Y5 | Y6 | Y7 | Y8 | Y9 | Y10 | Y11 | Y12 | Y13 | Y14 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Mine Operations | 132 | 0 | 68 | 123 | 132 | 132 | 126 | 120 | 120 | 120 | 123 | 123 | 112 | 76 | 20 | 20 | 20 | 17 |
| Maintenance | 94 | 12 | 38 | 85 | 94 | 94 | 93 | 93 | 93 | 93 | 93 | 93 | 87 | 69 | 41 | 41 | 41 | 29 |
| Geology | 15 | 12 | 15 | 15 | 15 | 15 | 15 | 15 | 15 | 15 | 15 | 15 | 14 | 7 | 3 | 3 | 3 | 3 |
| Mine Engineering | 22 | 5 | 18 | 22 | 22 | 22 | 22 | 22 | 22 | 22 | 22 | 20 | 19 | 19 | 0 | 0 | 0 | 0 |
| **Total** | **263** | **29** | **139** | **245** | **263** | **263** | **256** | **250** | **250** | **250** | **253** | **251** | **232** | **171** | **64** | **64** | **64** | **49** |

Source: GMS (2021).

# 17. RECOVERY METHODS

## 17.1 Overall Process Design

The process plant design for the project is based on a robust metallurgical flowsheet to treat gold- and silver-bearing ore to produce doré. The flowsheet is based on metallurgical testwork, industry standards and conventional unit operations.

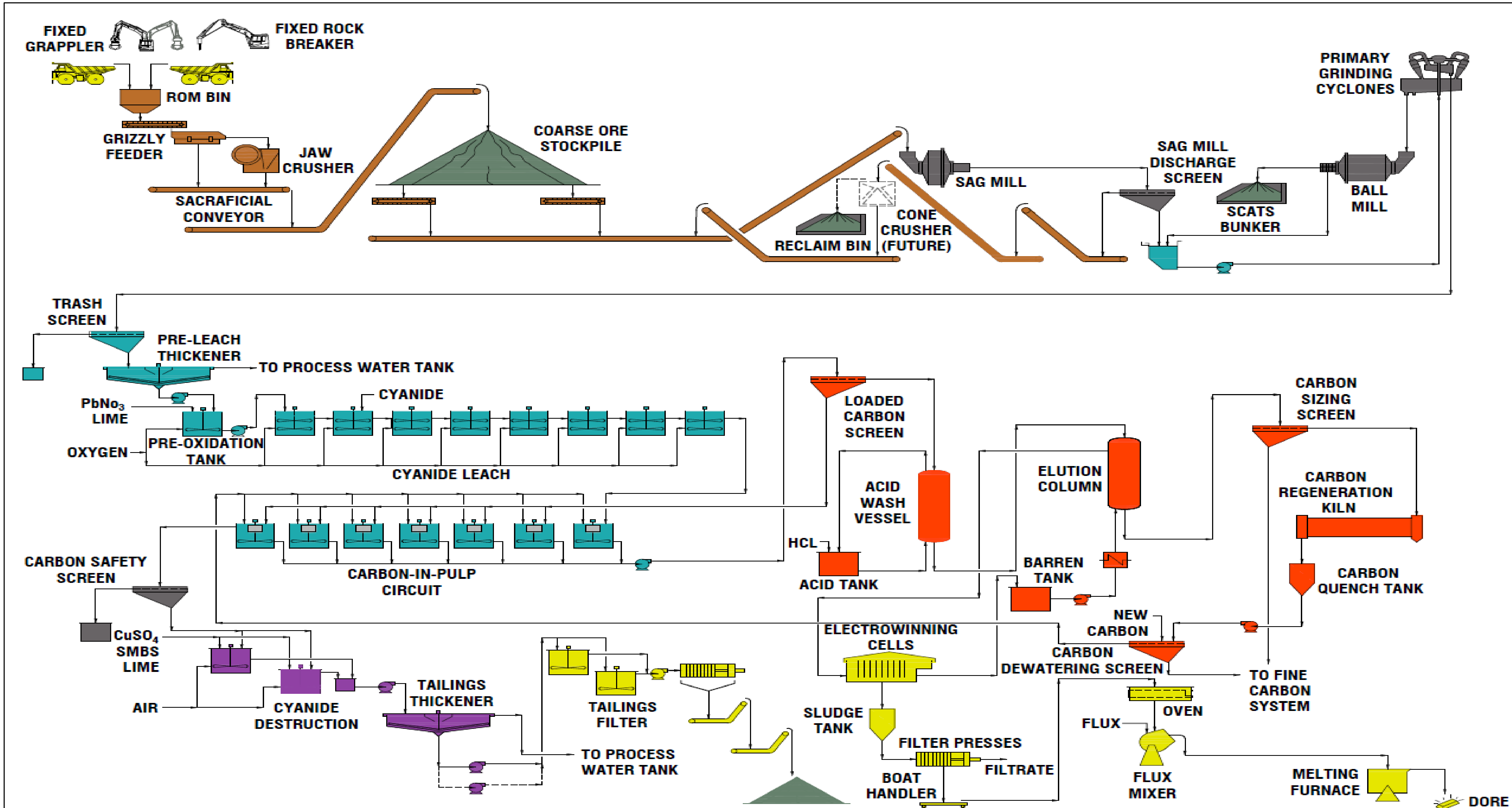
The process plant will treat 4.0 million tonnes per year (Mt/a) of ore and will consist of comminution, cyanide leach, carbon adsorption via carbon-in-pulp (CIP), carbon elution and gold recovery circuits. CIP tailings will be treated in a cyanide destruction circuit and dewatered to produce a filtered tailings product.

The key project design criteria for the process plant are listed below:

- nominal throughput of 4.0 Mt/a of ore
- crushing plant availability of 75%
- grinding, leach, gold recovery availability of 92% and tailings filtering circuit availability of 85% through the use of standby equipment in critical areas, inline crushed ore stockpile and reliable power supply
- comminution circuit to produce a particle size of 80% passing ($P_{80}$) 53 µm
- leach residence time of 36 hours to achieve optimal gold and silver extraction
- cyanide destruction circuit to produce weak acid dissociable (WAD) cyanide levels of less than 10 ppm
- sufficient process plant control to minimize the need for continuous operator interface and to allow for manual override and control if and when required
- equipment selection based on suitability for the required duty, reliability, and ease of maintenance
- plant layout that provides ease of access to all equipment for operating and maintainability, while facilitating concurrent construction activities in multiple areas of the plant.

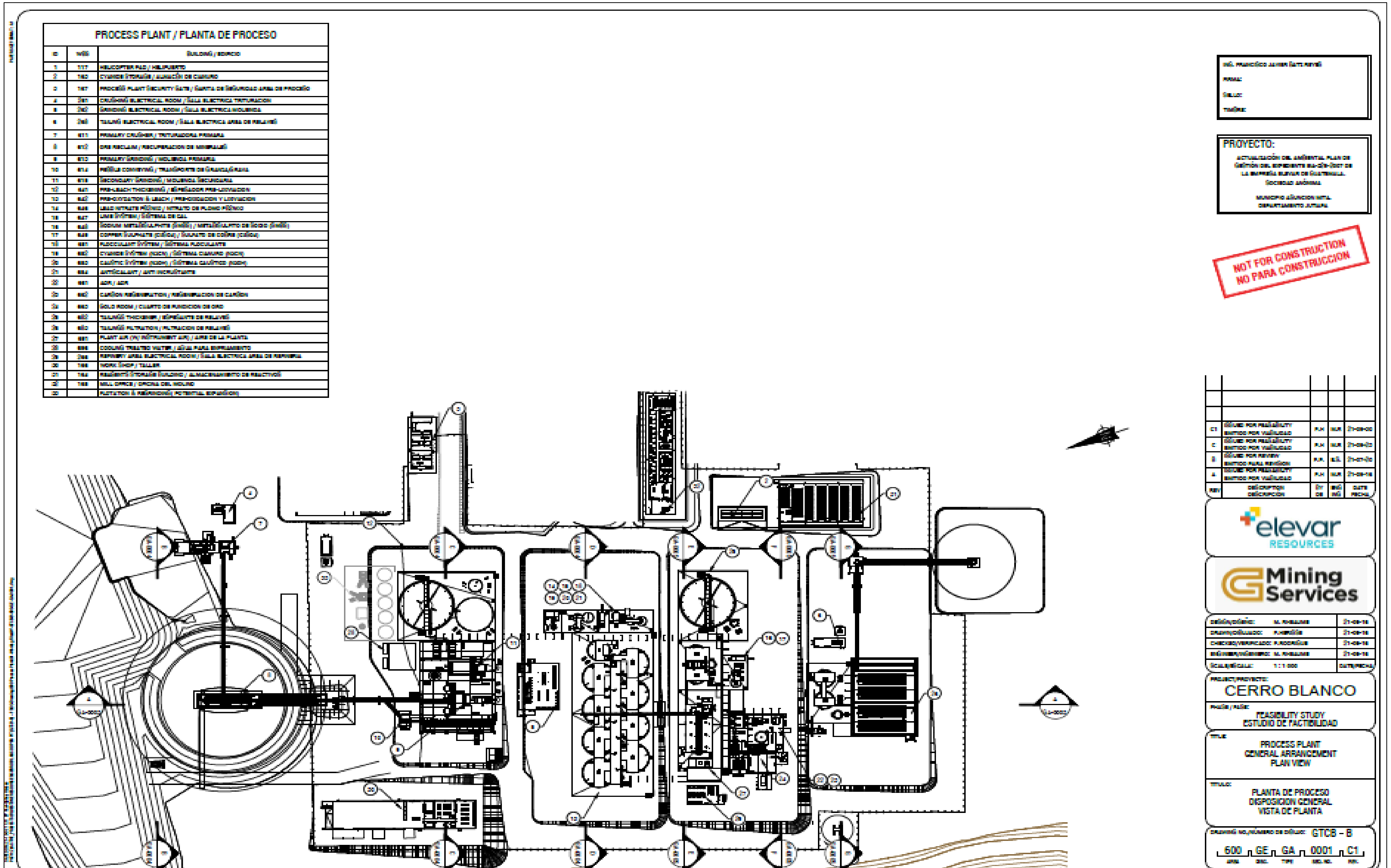
An overall process flow diagram depicting the various unit operations is presented in Figure 17-1. A layout of the proposed plant is provided in Figure 17-2.

Figure 17-1: Overall Process Flow Diagram




Source: GMS (2021).

**Figure 17-2: Process Plant Layout**




Source: GMS (2021).

## 17.2 Process Plant Design Criteria

The proposed process design is comprised of the following circuits:

- primary crushing of run-of-mine (ROM) material
- crushed ore stockpile
- semi-autogenous (SAG) mill with a discharge screen; ball mill with hydrocyclone classification
- pre-oxidation
- pre-leach thickening
- cyanide leaching and carbon adsorption using CIP
- pressure Zadra elution
- carbon handling and regeneration
- electrowinning and smelting to produce doré
- cyanide destruction of tailings using $SO_2$/air process
- tailings thickening and filtering to produce filtered tailings for storage at surface
- reagents storage and make-up systems
- air and oxygen circuits
- water systems (potable water, treated water, gland seal water and process water).

Key process design criteria are summarized in Table 17.1.

**Table 17.1: Key Process Design Criteria**

| Criteria | Unit | Nominal Value | Source |
|---|---|---|---|
| General | | | |
| Annual Throughput | Mt/a | 4.0 | 2021 Mine Plan v8 |
| Plant Availability | % | 92 | GMS |
| Plant Throughput | dry t/h | 496 | GMS |
| LOM Average Gold Head Grade | g/t | 1.64 | 2021 Mine Plan v8 |
| LOM Average Silver Head Grade | g/t | 7.27 | 2021 Mine Plan v8 |
| Overall Gold Recovery | % | 93% | GMS |
| Overall Silver Recovery | % | 84.3% | GMS |
| Crushing | | | |
| Crusher Type | - | C160 Jaw Crusher | GMS |
| Availability/Utilization | % | 75 | GMS |
| Crushing Plant Throughput | dry t/h | 609 | GMS |
| Number of Crushing Stages | - | 1 | GMS |
| Crushing Final Product Size ($P_{80}$) | mm | 140 | GMS |
| Crushed Material Stockpile | | | |
| Stockpile Capacity (live) | h | 12 | GMS |
| Stockpile Capacity (live) | t | 5,956 | GMS |
| Grinding | | | |
| Bond Ball Mill Work Index (85$^{th}$ percentile) | kWh/t | 23 | Testwork |
| SMC Axb (15$^{th}$ percentile) | - | 34.5 | Testwork |
| Bond Abrasion Index | - | 0.50 | Testwork |
| Primary Grinding Mill Type | - | SAG mill | GMS |
| SAG Mill Diameter | m | 9.75 | GMS |
| SAG Mill Length | m | 5.18 EGL | GMS |
| Installed Power | kW | 9,500 | GMS |
| Circuit Configuration | - | Open | GMS |
| Primary Grinding Transfer Size ($T_{80}$) | µm | 419 | GMS |
| Secondary Grinding Mill Type | - | ball mill | GMS |
| Ball Mill Diameter | m | 7.32 | GMS |
| Ball Mill Length | m | 12.95 EGL | GMS |
| Installed Power | kW | 14,000 | GMS |
| Circuit Configuration | - | Closed | GMS |
| Size Classification Type | - | hydrocyclones | GMS |
| Circulating Load | % | 300-375 | GMS |
| Final Target Product Size ($P_{80}$) | µm | 53 (design) 75 (operating) | GMS/ Testwork |
| Pre-Leach Thickening | | | |
| Thickener Loading Rate | t/h/m2 | 0.4 | Testwork |
| Thickener Underflow Density | % w/w | 42 | GMS |
| Leaching | | | |
| Pre-Oxidation | Y / N | yes | Testwork |
| Pre-Oxidation Retention Time | h | 2 | Testwork |
| Dissolved Oxygen Target | ppm | >20 | Testwork |
| Leach Retention Time | h | 36 | GMS |
| Number of Leach Tanks | # | 6 | GMS |
| Sodium Cyanide Consumption | kg/t | 0.3 | Testwork |
| Lime Consumption | kg/t | 1.50 | Testwork |
| CIP | | | |
| CIP Retention Time | h | 2 | Vendor Modelling |
| Number of CIP Tanks | # | 7 | Vendor Modelling |
| Carbon Concentration | ppm | 50 | Vendor Modelling |
| Gold Carbon Loading | g/t | 1,355 | Vendor Modelling |
| Silver Carbon Loading | g/t | 5,968 | Vendor Modelling |
| Total Gold + Silver Carbon Loading | g/t | 7,323 | Vendor Modelling |
| Carbon Processing | | | |
| Carbon Handling Capacity | t/d | 12.5 | GMS |
| Acid Wash Type | # | hydrochloric acid | GMS |
| Elution Operating Temperature | ºC | 150 | GMS |
| Elution Operating Pressure | kPa | 500 | GMS |
| Smelting Furnace Type | - | induction furnace | GMS |
| Carbon Consumption Rate | g/t | 30 | GMS |
| Cyanide Destruction | | | |
| Feed Solution, $CN_{WAD}$ | ppm | 149 | Testwork |
| Discharge Solution, $CN_{WAD}$ | ppm | < 10.0 | Testwork |
| Design Retention Time | h | 1.0 | Testwork |
| Number of Tanks | # | 2 | GMS |
| $SO_2$ Consumption | g / g $CN_{WAD}$ | 4.0 | Testwork |
| Lime Consumption | g / g $CN_{WAD}$ | 0.8 | Testwork |
| $CuSO_4$-5$H_2O$ Concentration | ppm | 25 | Testwork |
| Tailings Management | | | |
| Disposal Type | - | Filtered tailings storage | GMS |
| Final Moisture Content | % | 16-18 | GMS |

Source: GMS (2021).

## 17.3 Process Plant Description

The overall process flowsheet includes a primary jaw crusher and a SAG and ball (SAB) grinding circuit with the ball mill in closed circuit with hydrocyclones to achieve the final product size. The hydrocyclone overflow stream will flow by gravity to two vibrating trash screens, one in duty and one standby, ahead of a pre-leach thickener. The thickened slurry is pumped to a pre-oxidation circuit and then to a leach circuit where sodium cyanide, lime slurry, lead nitrate, and oxygen are added for gold and silver leaching. A CIP carousel circuit will adsorb dissolved gold and silver onto activated carbon. A Zadra elution circuit will be used to recover gold and silver from loaded carbon to produce doré. A cyanide destruction circuit using $SO_2$ and air will reduce the WAD cyanide level in the tailings stream to less than 10 ppm. The tailings stream will be thickened and filtered to produce a filter cake and transported to a dry stack tailings storage facility.

### 17.3.1 Primary Crushing

Ore from the open pit, at a maximum lump size of 900 mm, will be transported to the plant by 90-tonne capacity end dump haul trucks. The trucks will tip directly into the ROM bin; however, if they are not permitted to directly tip into the ROM bin, then the load will be dumped onto the ROM pad. The ROM pad will be primarily utilized for storage and blending if required. ROM material will be reclaimed to the ROM bin by a front-end loader.

Ore will be withdrawn from the ROM bin by a variable speed apron feeder and fed directly to a vibrating grizzly (100 mm aperture size). Oversize from the grizzly will report directly to the jaw crusher, which will operate in open circuit. A rock breaker and grapper will be installed to assist in breaking down oversize material retained above the jaw crusher. Crushed ore from the crusher discharges, together with undersize from the grizzly and will be withdrawn by a sacrificial conveyor. A belt magnet at the sacrificial conveyor discharge will recover the trash metal. The sacrificial conveyor will feed crushed material to the stockpile feed conveyor and to the crushed stockpile. The stockpile feed conveyor will be fitted with a weightometer to monitor the primary crusher throughput and to control the apron feeder variable speed drive.

The crushing circuit will be serviced by a single dust collection system consisting of multiple extraction hoods, ducting, and a baghouse. Dust collected from this system will be discharged onto the stockpile feed conveyor.

### 17.3.2 Crushed Stockpile

The crushed material will be conveyed to the crushed material stockpile. The stockpile will have a live capacity of approximately 6,000 tonnes (equivalent to 12 hours of mill feed). Two reclaim apron feeders located underneath the stockpile will be installed with variable speed drives (VSDs) to control the reclaim rate feeding the grinding circuit. A single apron feeder will be capable of providing the total plant throughput of 496 t/h and will feed the ore to the SAG mill feed conveyor.

### 17.3.3 Grinding

Reclaimed material from the crushed stockpile will feed a 9.75 m diameter x 5.18 m EGL SAG mill via the SAG mill feed conveyor. The SAG mill will be installed with 9,500 kW synchronous dual motors and a VSD to control the speed of the SAG mill. A belt-scale on the SAG feed conveyor will monitor the feed rate. Process water will be added to the SAG mill to maintain the slurry discharge of the SAG mill at a constant density of 70%. SAG mill discharge will pass over a screen to remove grinding media scats and a small number of pebbles. The SAG mill discharge screen undersize will report to the cyclone feed pump box, combining with ball mill discharge. The SAG mill discharge will have an average transfer size ($T_{80}$) of 419 µm. SAG discharge screen oversize will be conveyed to the SAG mill feed conveyor.

Slurry from the cyclone feed pump box will be transferred to a cluster of 24 (19 operating / 4 standby) 375 mm hydrocyclones for size classification. The cyclone overflow, at a final target product $P_{80}$ of 53 µm (design), 75 µm (operating) will be transferred to the pre-leach thickener. The hydrocyclones have been designed for a 300% circulating load.

Cyclone underflow will feed a 7.32 m diameter x 12.95 m EGL overflow ball mill with VSD-controlled 14,000 kW synchronous dual motors. Slurry will overflow from the ball mill into a trommel screen attached to the discharge end of the ball mill. The trommel screen oversize, consisting mainly of grinding media scats, will discharge into a trash bin for removal from the grinding circuit, while the undersize will flow into the cyclone feed pump box.

### 17.3.4 Pre-Leach Thickening

Hydrocyclone overflow, at the target size of $P_{80}$ of 53 µm, will flow onto two vibrating trash screens in parallel, one in duty and one standby, to remove trash material. Oversize material will discharge into a trash bin, while screen undersize will flow by gravity to a 32.0 m diameter high-rate pre-leach thickener. Flocculant solution will be added to the pre-leach thickener feed to promote the settling of fine solids. The thickener will dewater the slurry to 42% solids. The pre-leach thickener underflow will be pumped to the pre-oxidation tank, while the pre-leach thickener overflow will report to the process water tank.

### 17.3.5 Leaching

Pre-leach thickener underflow will be pumped to a 14.0 m diameter x 14.5 m high pre-oxidation tank prior to leaching. Oxygen will be injected into the slurry with an external high-shear mixer. The agitated tank and slurry will be conditioned for two hours to oxidize sulphide minerals.

The oxidized slurry will flow to the first of six 18.0 m diameter x 22.0 m high mechanically agitated leach tanks. The leach circuit is designed to provide 36 hours of retention time. Lime slurry will be stage-added at a rate of up to 1.5 kg/t to maintain circuit alkalinity at a pH of 10.5, preventing the creation of hydrogen cyanide gas. Sodium cyanide solution will be added to the circuit at a rate of up to 0.3 kg/t, while oxygen will be injected into the slurry with an external high-shear mixer for the first two tanks and sparged into the sides of other leach tanks via slamjets to maintain dissolved oxygen above 20 ppm. As the slurry progresses through the circuit, gold and silver dissolution will occur. Slurry from the leach circuit will then flow by gravity to the CIP circuit for carbon adsorption.

### 17.3.6 Carbon-in-Pulp

Leached slurry will flow to a CIP pump cell plant consisting of seven 6.5 m diameter x 10.25 m high CIP tanks operating in a carousel arrangement. Each tank will be installed with a pumping inter-stage screen mechanism for retaining activated carbon. In the CIP carousel circuit, the feed and discharge points of each tank rotate to simulate the counter-current movement of carbon against the slurry. Carbon management is simplified, as each cell contains a discrete batch of carbon. As the slurry flows through the CIP circuit, gold and silver cyanide complexes will be adsorbed onto the activated carbon until precious metal values in the solution progressively decrease. The average carbon concentration in the CIP circuit is expected to be approximately 50 g/L to maximize adsorption.

Once per day, loaded carbon from a CIP tank will be pumped to the loaded carbon screen where the slurry will be separated and the loaded carbon transferred to the acid wash circuit. The slurry will then be transferred back to the CIP circuit. Fresh activated carbon from the regeneration circuit will be pumped into the CIP circuit.

Tailings from the CIP circuit will be pumped to a carbon safety screen to capture any residual fine carbon particles. Captured carbon will be collected in bags and processed to recover any residual gold and silver. The carbon safety screen undersize will flow via gravity to the cyanide destruction circuit.

### 17.3.7 Elution Circuit

The elution circuit is a 12.5-tonne pressure Zadra circuit. Loaded carbon from the CIP circuit will flow by gravity to one 12.5-tonne acid wash vessels. The loaded carbon will be treated with a circulating 3% hydrochloric acid (HCl) solution to remove calcium deposits, magnesium, sodium salts, silica, and fine iron particles. Organic foulants, such as oils and fats, are unaffected by the acid and will be removed after the elution step in the regeneration circuit using a horizontal kiln.

During the acid wash cycle, the carbon will be rinsed with fresh water. HCl solution will be pumped from the dilute acid tank upward through the acid wash vessel, overflowing back into the dilute acid tank. The loaded carbon will then be neutralized with a weak sodium hydroxide solution followed by a rinse with fresh water.

A recessed impeller pump will transfer the acid-washed loaded carbon from the acid wash vessel to one of two 8-tonne elution vessels using fresh water. The loaded carbon slurry will discharge directly into the top of the elution vessel.

The carbon stripping (elution) process will utilize barren strip solution to strip the loaded carbon, creating a pregnant gold and silver solution that will be pumped through the electrowinning cells for precious metal recovery. The barren solution exiting the electrowinning cells will be circulated back to the barren solution tank for re-use.

The elution vessel will be a carbon steel tank with a capacity to hold approximately 12.5 tonnes of loaded carbon. During the strip cycle, solution containing approximately 2% sodium hydroxide and 0.2% sodium cyanide, at a temperature of 150°C will be pumped up through the elution vessel at a pressure of 500 kPa. Solution exiting the top of the elution vessel will be cooled below its boiling point by a heat recovery heat exchanger. Heat from the outgoing solution will be transferred to the incoming barren solution prior to passing through the solution heater. An electric in-line solution heater will be used as the primary heating source. The strip will be complete in approximately eight hours, allowing additional strips to accommodate higher feed grade material.

#### 17.3.8 Carbon Regeneration

The carbon regeneration circuit will thermally regenerate the stripped carbon, re-activating the pores and removing any organic foulants, such as oils and fats. Fresh activated carbon will be added to account for any carbon lost during the adsorption and desorption processes.

A recessed impeller pump will transfer the stripped carbon from the elution vessel to the carbon sizing dewatering screen. A Sweco shaker screen or equivalent will be provided to dewater and screen the carbon. Oversize carbon from the screen will discharge by gravity into the regeneration kiln feed hopper. Screen undersize carbon, containing carbon fines and water, will drain by gravity into the carbon fines system. Periodically, the carbon fines will be filtered and collected in bags and sent to a refinery to recover gold and silver.

A horizontal kiln will be utilized to treat 600 kg/h of carbon per day, equivalent to 100% regeneration of stripped carbon. The regenerated carbon from the kiln will flow by gravity into the carbon quench tank and be pumped back to the CIP circuit.

To compensate for carbon losses from attrition and impact, fresh carbon will be added to the carbon attrition tank and mixed with fresh water to activate the carbon pores. The fresh carbon will then drain into the carbon quench tank and combine with the regenerated carbon discharging from the kiln.

#### 17.3.9 Electrowinning & Refining

Pregnant solution from the elution circuit will be pumped to the refinery for electrowinning, producing a gold and silver sludge. The sludge will then be filtered, dried, and smelted in an induction furnace producing doré.

Pregnant solution will be pumped through two dual electrowinning cells operating in parallel. Gold and silver will plate on a pair of 33 stainless steel cathodes, while the barren solution will flow into the barren return tank and be pumped back to the barren solution tank for re-use. To prevent a build-up of impurities, a 15% daily bleed of barren solution will be pumped to the leach circuit.

The gold- and silver-rich sludge will periodically be washed off the stainless-steel cathodes into the electrowinning sludge tank using a high-pressure water spray. Once the tank is filled, the sludge will be drained, filtered, dried, mixed with fluxes, and smelted in a 150 kW induction furnace producing doré. This process will take place within a secure and supervised area and the precious metal product will be stored in a vault until shipped off site.

#### 17.3.10 Cyanide Destruction

The cyanide destruction circuit will consist of two 11.0 m diameter x 12.0 m high mechanically agitated tanks, each with a capacity to handle the full slurry flow for the required residence time of one hour. Residual cyanide will be destroyed using the $SO_2$/air process. Treated slurry from the circuit will then be pumped to the tailings thickener. The cyanide destruction circuit will treat CIP tailings slurry, process spills from various contained areas, and process bleed streams.

Low-pressure air will be injected into the cyanide destruction tanks. Lime slurry will be added to maintain the optimum pH of 8.0 to 8.5, and copper sulphate will be added as a catalyst to maintain a 25 ppm concentration in solution. A sodium metabisulphite solution at a rate of 1.7 kg/t ore will be dosed into the system as the source of $SO_2$. This system has been designed to reduce the WAD cyanide concentration to a target of less than 10 ppm.

#### 17.3.11 Tailings Handling

Treated slurry from the cyanide destruction circuit will be pumped to a 32.0 m diameter high-rate tailings thickener to produce a tailings density of approximately 50% solids. Flocculant solution will be added to the tailings thickener feed to promote the settling of fine solids. The tailings thickener underflow will then be pumped to two tanks with a total capacity of six hours. Horizontal plate-and-frame pressure filters will be used to further dewater the tailings. Four filters will be provided, three in operation and one on standby.

The filters will produce a tailings filter cake of 16% to 18% moisture. The filtrate from the filters will be pumped back to the tailings thickener to recover any residual fines. The tailings thickener overflow will report to the process water tank.

The filter cake will be discharged onto conveyors below the filters which will convey the tailings to a nearby stockpile that will be reclaimed by front-end loader and loaded into haul trucks that will transport the filtered tailings to the dry stack tailings facility.

### 17.4 Reagent Handling & Storage

Reagents consumed within the process plant will be prepared on site and distributed via various reagent handling and make-up systems. These reagents include sodium cyanide, quick lime, lead nitrate, hydrochloric acid, sodium hydroxide, copper sulphate, sodium metabisulphite, antiscalant, flocculant, and activated carbon.

For the management of unexpected reagent spills, the reagent preparation and storage facilities will be located within containment areas designed to accommodate 50% more than the content of the largest tank. Where required, each reagent system will be located within its own containment area to facilitate its return to its respective storage vessel and to avoid the mixing of incompatible reagents. Storage tanks will be equipped with level indicators, instrumentation, and alarms to ensure spills do not occur during normal operation. Appropriate ventilation, fire and safety protection, eye wash stations and showers, and material safety data sheet (MSDS) stations will be located throughout the facilities. Sumps and sump pumps will be provided for spillage control.

The reagents will be mixed, stored, and delivered to the thickeners, leach, CIP, acid wash, elution, and cyanide destruction circuits. Dosages will be controlled by flow meters and control valves. The capacity of the storage tanks will be sized to typically handle one day of production. The reagents will be delivered in dry form, except for hydrochloric acid and antiscalant, which will be delivered as solutions.

### 17.4.1 Sodium Cyanide

Sodium cyanide in briquettes will be delivered in 20 tonne ISO containers and stored in the reagent shed. Treated water will be added to ISO containers to achieve 20% w/v concentration. Sodium hydroxide will be added to provide protective alkalinity to avoid generation of hydrogen cyanide gas. After mixing for a pre-set period, the sodium cyanide solution will be completely transferred to a 110 $m^3$ sodium cyanide holding tank using the sodium cyanide transfer pump. From this holding tank, the sodium cyanide solution will be pumped to a 50 $m^3$ distribution tank. The sodium cyanide solution will be pumped to the leach circuit and barren solution tank via metering pumps. The sodium cyanide mixing area will be ventilated using the cyanide system exhaust fan.

### 17.4.2 Quick Lime

Lime will be delivered in solid form as quick lime (92% CaO purity) in bulk and stored in a 200-tonne silo. Treated water will be added to a 150 kg/h slaker system. The slaker will feed a 120 $m^3$ agitated lime mixing tank. Lime will be dissolved in water to achieve 22.5% solids concentration. Lime slurry will be pumped to the pre-oxidation, leach, and cyanide destruction circuits via a lime ring main. The lime mixing area will be ventilated using the lime dust collector system.

### 17.4.3 Lead Nitrate

Lead nitrate will be delivered in solid granular form in 1-tonne bulk bags and stored in the reagent shed. Treated water will be added to the 15.0 $m^3$ agitated lead nitrate mixing tank. The lead nitrate solution will be pumped by transfer pump to a 22.5 $m^3$ distribution tank. Bags will be lifted into the bag breaker on top of the mixing tank by the area hoist. Lead nitrate will fall into the tank and be dissolved in water to achieve the required 20% w/v concentration. Lead nitrate solution will be pumped to the pre-oxidation tank. The lead nitrate mixing area will be ventilated with a small stack.

### 17.4.4 Copper Sulphate

Copper sulphate will be delivered in solid granular form in 1-tonne bags and stored in the reagent shed. Treated water will be added to the 15 $m^3$ agitated copper sulphate mixing tank. The copper sulphate solution will be pumped by transfer pump to a 22.5 $m^3$ distribution tank. Bags will be lifted into the copper sulphate bag breaker on top of the mixing tank by the area hoist. The reagent will fall into the tank and be dissolved in water to achieve the required 20% w/v concentration. Copper sulphate will be pumped to the cyanide destruction circuit. The copper sulphate mixing area will be ventilated using the copper sulphate system exhaust fan.

### 17.4.5 Sodium Metabisulphite

Sodium metabisulphite (SMBS) will be delivered in solid granular form in 1-tonne bags and stored in the reagent shed. Treated water will be added to the 100 $m^3$ agitated SMBS mixing tank. The SMBS solution will be pumped by transfer pump to a 200 $m^3$ distribution tank. Bags will be lifted into the SMBS bag breaker on top of the mixing tank by the area hoist. The reagent will fall into the tank and be dissolved in water to achieve the required 20% w/v concentration. SMBS will be pumped to the cyanide destruction circuit. The SMBS mixing area will be ventilated using the SMBS dust collector system.

### 17.4.6 Sodium Hydroxide

Sodium hydroxide will be delivered in solid pearls or beads in 1-tonne bags and stored in the reagent shed. Treated water will be added to the 13 $m^3$ agitated sodium hydroxide mixing tank. The sodium hydroxide solution will be pumped by transfer pump to an 18 $m^3$ distribution tank. Bags will be lifted into the sodium hydroxide bag breaker on top of the mixing tank by the area hoist. The reagent will fall into the tank and be dissolved in water to achieve the required 20% w/v concentration. Sodium hydroxide will be pumped to the elution circuit and sodium cyanide mix tank. The sodium hydroxide mixing area will be ventilated using the sodium hydroxide system exhaust fan.

### 17.4.7 Hydrochloric Acid

Hydrochloric acid will be delivered in intermediate bulk containers (IBCs) as a 32% w/w solution and stored in a dedicated section of the reagent shed until required. Hydrochloric acid will be mixed with treated water to achieve the required 3% concentration. Hydrochloric acid will be delivered to the acid wash circuit using the hydrochloric acid metering pump.

### 17.4.8 Flocculant

Powdered flocculant will be delivered to site in 1-tonne bags and stored in the reagent shed. A mixing and dosing system will be installed that includes a flocculant storage hopper, flocculant blower, flocculant wetting head, flocculant mixing tank, and flocculant transfer pump. Powder flocculant will be loaded into the flocculant storage hopper using the area hoist. Dry flocculant will be pneumatically transferred into the wetting head, where it will be contacted with treated water. Flocculant solution, at 0.50% w/v, will be agitated in the flocculant mixing tank for a pre-set period. The flocculant will then be transferred to the flocculant storage tank using the flocculant transfer pump. Prior to pumping to the pre-leach and tailings thickeners, the flocculant solution will be further diluted in the flocculant make-up area.

### 17.4.9 Activated Carbon

Activated carbon will be delivered as solid granular form in bulk bags. The carbon will be introduced into the carbon attrition tank where it is slurried and agitated to remove the friable edges of the carbon particles. The carbon slurry will be pumped over the sizing screen where the carbon fines discharge to the fine carbon hopper and the coarse carbon particles are pumped to the CIP circuit.

### 17.4.10 Antiscalant

Antiscalant will be delivered as solution form in 210-L drums and stored in the reagent shed. Antiscalant will be dosed neat using metering pumps to the barren solution tank.

### 17.4.11 Gold Room Smelting Fluxes

Silica sand, sodium nitrate, and soda ash will be delivered as solid crystals/pellets in bags or plastic containers. These items will be stored in the reagent shed until needed in the gold room.

## 17.5 Air & Oxygen Supply

High-pressure air at 700 kPa (gauge) will be required in the plant and produced by compressors. The entire high-pressure air supply will be dried and used to satisfy both plant air and instrument air demands. Dried air will be distributed via the air receivers located throughout the plant. The filtered tailings area will have its own dedicated high-pressure air system.

Low-pressure air at 310 kPa will be required in the cyanide destruction $SO_2$/air process and produced by dedicated compressors.

Oxygen will be used in the pre-oxidation and leach areas and will be supplied by an on-site 4 to 5 t/d oxygen generation system.

## 17.6 Water Services

### 17.6.1 Process Water

Process water will consist of overflow water from the pre-leach and tailings thickeners and will be stored in an 18 m diameter x 12 m high process water tank providing 1.3 hours of live residence time. The process water will have a low precious metals concentration; the water will be used predominantly in the grinding circuit to dilute slurry to the required densities, and as spray water on the vibrating screens and ball mill trommel.

The process plant will generate approximately 18 $m^3$/h more process water than is used, and excess process water will be pumped to the water treatment circuit. Treated water will be added to the process water tank as clean make-up water when high amounts of process water are removed as a bleed stream from the circuit.

### 17.6.2 Treated Water

Treated water will be supplied from the water treatment plant to an 8.5 m diameter x 9.0 m high treated water tank providing approximately five hours of live residence time. Treated water will be used as gland seal water, reagent make-up, metallurgical samplers, cooling water, and process make-up water.

### 17.6.3 Domestic Water

Domestic water will be supplied from the domestic water treatment plant for safety showers and eye wash stations and personnel use.

## 17.7 Metallurgical Accounting

A weightometer on the stockpile feed conveyor will measure primary crushed ore tonnage, and a weightometer on the SAG mill feed conveyor will determine mill feed tonnage.

A manual belt cut sampling point on the SAG mill feed conveyor will allow for the collection of a mill feed head grade sample for cross-checking the calculated head grade. This sample will also be utilized to establish the moisture content of the mill feed.

Regular sampling of the leach feed stream and the final CIP tailings via full stream cross-cut metallurgical samplers will provide representative composite shift samples for leach head grade and tails solution and residue grades.

Solution samplers and flowmeters will be utilized to measure the daily pregnant solution grade and mass of pregnant solution produced. Solution samplers within the electrowinning circuit will provide representative composite shift samples for solution grade analysis.

Regular surveys of the gold and silver in circuit will allow a reconciliation of precious metals in the feed compared to doré production.

A density and flowmeter on the leach feed and tailings line will allow the dry tonnage of solids pumped to the leach and filters to be determined as a cross-check on the mill feed tonnage determined from the mill feed weightometer. In conjunction with the leach feed and tails samples, the mass flow measurements will allow the gold recovered in the CIP to be calculated.

Water supplied and used in the various areas will be continuously monitored.

Reconciliation of the reagents used over relatively long periods will be achieved by delivery receipts and stock takes. On an instantaneous basis, reagent usage rates of cyanide and elution reagent to unit operations will be measured and accumulated using flowmeters.

## 17.8 Plant Control System

The general control philosophy for the plant will be one with a moderate level of automation and remote-control facilities to allow critical process functions to be carried out with minimal operator intervention. Instrumentation will be provided within the plant to measure and control key process parameters.

The main control room, located in the process plant, will house PC-based operator-interface terminals (OIT) and a single server. These workstations will act as the control system supervisory control and data acquisition (SCADA) terminals. The control room is intended to provide a central area from where the plant is operated and monitored and from which the regulatory control loops can be monitored and adjusted. All key process and maintenance parameters will be available for trending and alarming on the process control system (PCS).

Additional OITs will be provided for data logging and engineering / programming functions.

A field touch panel will be installed in the feed preparation area to allow local operator control of the crushing plant to facilitate ease of operation for rock breaking and stockpiling if required. A second field touch panel will be installed in the elution area to allow local operator control of the elution sequence. A third field touch panel will be supplied for the grinding circuit area.

The process control system that will be used for the plant will be a programmable logic controller (PLC) and SCADA-based system. The PCS will control the process interlocks and PID control loops for non-packaged equipment. Control loop set-point changes for non-packaged equipment will be made at the OIT.

In general, the plant process drives will report their ready, run, and start push-button status to the PCS and will be displayed on the OIT. Local control stations will be located in the field in proximity to the relevant drives. These will, as a minimum, contain start and latch-off-stop (LOS) push-buttons that will be hard-wired to the drive starter. Plant drives will predominantly be started by the control room operator after the equipment has been inspected by an operator in the field.

The OITs will allow drives to be selected to *Auto*, *Local*, *Remote*, *Maintenance* or *Out-of-Service* modes via the drive control popup. Statutory interlocks, such as emergency stops and thermal protection, will be hardwired and will apply in all modes of operation. All PLC-generated process interlocks will apply in *Auto*, *Local* and *Remote* modes. Process interlocks will be disabled or bypassed in *Maintenance* mode with the exception of critical interlocks, such as lubrication systems on the mill.

Local selection will allow each drive to be operated by the operator in the field via the local start push-button, which is connected to a PLC input. Remote selection will allow the equipment to be started from the control room via the drive control popup. Maintenance selection will allow each drive to be operated by maintenance personnel in the field via the local start push-button, which is connected to a PLC input. A PLC output will be wired to each drive starter circuit for starting and stopping drives. Status indication of process interlocks as well as the selected mode of operation will be displayed on the OIT.

Vendor-supplied packages will use vendor standard control systems as required throughout the project. Vendor packages will generally be operated locally with limited control or set-point changes from the PCS system. General equipment fault alarms from each vendor package will be monitored by the PCS system and displayed on the OIT. Fault diagnostics and troubleshooting of vendor packages will be performed locally.

The use of actuated isolation or control valves will be implemented around the plant for automatic control loops or sequencing as part of the plant control or the elution sequence. All actuated valves and control valves will be operated from the OITs with remote position indication available. Automatic control valves will be controlled by PID loops within the PCS.

The PCS will perform all digital and analogue control functions, including PID control, for all non-packaged plant. Faceplates on the PCS displays will facilitate the entry of set-points, readout of process variables (PVs) and controlled variables (CVs), and entry of the three PID parameters (proportional, integral and derivative).

The majority of equipment interlocks will be software configurable. However, selected drives will be hard-wired to provide the required level of personal safety protection (e.g., the emergency stop buttons associated with every motor and the pull wire switches associated with conveyors).

All alarm and trip circuits from field or local panel-mounted contacts will be based on fail-safe activation. Alarm and trip contacts will open on abnormal or fault condition. If equipment shutdown occurs due to loss of main power supply, the equipment will return to a de-energized state and will not automatically restart upon restoration of power.

Sequential group starts and sequential group stops will not be incorporated for non-packaged plant equipment, except for the elution circuit. However, in any process, critical safety and equipment protection interlocks will cause a cascade stop in the event of interlocked downstream equipment stopping (e.g., trip of SAG mill feed conveyor will result in stop of the upstream apron feeder). Standard vendor packages may include automatic sequence start / stop controls within the vendor package only.

## 17.9 Plant Consumption

### 17.9.1 Water

Approximately 104 $m^3/h$ of treated water is required for users.

### 17.9.2 Energy

The power demand for the plant, along with the rest of the project, will be provided by grid power. The average power demand for the process plant is summarized in Table 17.2. The average power demand does not reflect the instantaneous power demand for equipment start-up and power plant capacity sizing.

**Table 17.2: Power Requirements**

| Area | Average Power (kW) | Annual Power Consumption (MWh) |
|---|---|---|
| Comminution | 29,253 | 256,259 |
| Leach/CIP/Cyanide Destruction | 3,245 | 28,427 |
| Reagents | 95 | 832 |
| Elution/Gold Room | 2,472 | 21,653 |
| Tailings Handling | 2,838 | 24,862 |
| Water/Air/Oxygen Services | 2,262 | 19,811 |
| Total | 40,165 | 351,844 |

Source: GMS (2021).

### 17.9.3 Reagents & Consumables

Reagent storage, mixing and pumping facilities will be provided for all reagents for the process plant. Reagents and consumables usage are summarized in Table 17.3 and Table 17.4.

**Table 17.3: Reagents Consumption**

| Description | Delivered Form | Daily Usage |
|---|---|---|
| Sodium Cyanide | ISO Containers (dry) | 3.3 t/d |
| Quick Lime (@92% CaO) | bulk (dry) | 19 t/d |
| Lead Nitrate | 1 tonne bags (dry) | 3 t/d |
| Hydrochloric Acid (32% strength) | 1 $m^3$ totes | 1.5 $m^3$/d |
| Sodium Hydroxide | 1 tonne bags (dry) | 1 t/d |
| Copper Sulphate | 1 tonne bags (dry) | 2.3 t/d |
| SMBS | 1 tonne bags (dry) | 8.6 t/d |
| Flocculant | 1 tonne bags (dry) | 1 t/d |
| Activated Carbon | 1 tonne bags (dry) | 0.3 t/d |

Source: GMS (2021).

**Table 17.4: Key Consumables Consumption**

| Description | Delivered Form | Usage |
|---|---|---|
| Jaw Crusher – Fixed Jaw | lot | 6.8 sets/year |
| Jaw Crusher – Movable Jaw | lot | 5.2 sets/year |
| Jaw Crusher – Upper Cheek Plate | lot | 1.6 sets/year |
| Jaw Crusher – Lower Cheek Plate | lot | 3.6 sets/year |
| SAG Mill Liners | lot | 1.0 set/year |
| Ball Mill Liners | lot | 1.0 set/year |
| Primary Ball Mill Grinding Media (125 mm Chrome Steel) | bulk | 0.57 kg/t |
| Secondary Ball Mill Grinding Media (65 mm Chrome Steel) | bulk | 1.29 kg/t |
| Carbon Screens | lot | 2.0 sets/ year |
| Filter Cloths and Membranes | lot | 9.0 sets/ year |

Source: GMS (2021).

## 17.10 Process Plant Personnel

The personnel for the process plant will consist of management, technical support, shift supervision, laboratory, operators, and maintenance staff. The management and technical support staff will work five 8-hour days per week with weekends off. Shift supervision, laboratory staff, shift operators, and maintenance staff will work 12-hour days and night shifts on a rotation cycle, using three rotating crews.

Annual process plant personnel requirements are provided in Table 17.5.

**Table 17.5: Process Plant Personnel**

| Department | Position | Compliment |
|---|---|---|
| Management | Senior Process Manager | 1 |
| | Administrative Assistant/Clerk | 1 |
| Process Operations | Operations Manager | 1 |
| | Senior Plant Supervisor | 1 |
| | Shift Supervisors | 3 |
| | Control Room Operators | 3 |
| | Plant Equipment Operator | 3 |
| | Crushing Operators | 6 |
| | Grinding Operators | 3 |
| | Leaching/CIP/ADR/Detoxification | 9 |
| | Gold Room Operator | 6 |
| | Tailings Process Operator | 3 |
| | Tailings Filtration Operator | 6 |
| | Reagents Operator | 6 |
| | Helper/Labourer | 9 |
| Water Management | Water Treatment Supervisor | 1 |
| | Dewatering Worker | 3 |
| | Water Plant Operator | 9 |
| | Surface Collection Water | 3 |
| Process Maintenance | Maintenance Manager | 1 |
| | Reliability Engineer | 1 |
| | Maintenance Planner | 2 |
| | Maintenance Supervisors | 2 |
| | Maintenance Technician | 18 |
| | Electrical/Instrumentation | 11 |
| Technical Support | Senior Metallurgist | 1 |
| | Metallurgist | 2 |
| | Metallurgical Technician | 3 |
| Laboratory | Senior Chief Assayer | 1 |
| | Assay Technician | 21 |
| Total | | 140 |

Source: Bluestone (2021).

# 18. PROJECT INFRASTRUCTURE

## 18.1 General

The Cerro Blanco Gold Project infrastructure is designed to support a nominal 4 Mt/a processing plant with an annual mining rate of 21.5 Mt/a that is operating 24 hours per day, 7 days per week. is the project infrastructure is also designed for local conditions and topography.

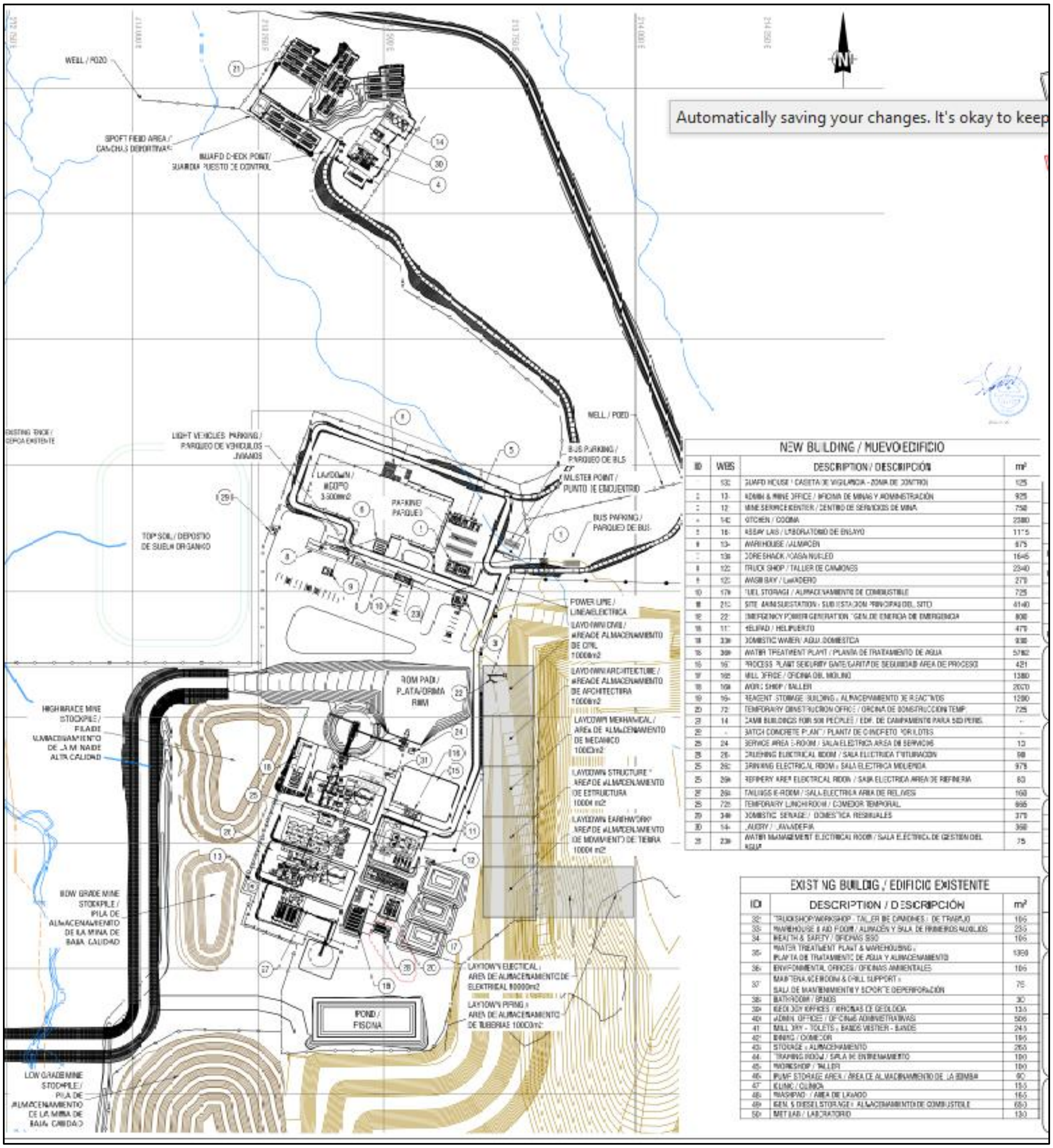
The main infrastructure items include the following:

- 5 km new site access road (including a 110 m long bridge)
- new 138 kV powerline
- on-site 138 kV to 13.8 kV substation
- 500 permanent man camp with kitchen and laundry
- water treatment and water management facilities including diversion channels, ditches, and collection ponds
- process plant site pad and buildings
- administration and mine office building
- warehouse
- assay and metallurgical laboratory
- primary crusher pad
- emergency power generator(s)
- one dewatering well (seven future wells deferred to sustaining capital)
- reagent warehouse and cyanide storage pad
- mine service center
- mine maintenance facility and wash bay
- fresh and fire water tanks
- process water tank
- fuel and gasoline storage and distribution bay
- helipad
- contact water treatment plant
- sewage treatment plant
- domestic water treatment plant

- solid waste disposal facilities (including topsoil deposit, temporary waste storage pad and overburden deposit)
- dry stack tailings facility (DSTF)
- waste rock storage facility
- mine haul roads
- on-site access roads for plant and facilities
- additional security facilities including site access control station.

The plant site and arrangement of the main infrastructure are presented in Figure 18-1.

**Figure 18-1: Plant Site General Arrangement**




Source: GMS (2021).

## 18.2 Site Layout

The proposed site layout has been designed to minimize environmental and community impacts, provide security-controlled site access, minimize construction costs, and optimize operational efficiency. Primary buildings have been located to allow easy access for construction and to utilize existing topography to minimize bulk earthwork volumes.

The open pit is considered the reference point of the project with DSTF positioned on its east side.

The process plant is located on the northwest side of the DSTF. The primary crusher is located on the northeast side of the proposed process plant location.

The service mining facilities (mine service center, maintenance facility, wash, and fuel bay) are north of the process plant and allow easy access for heavy equipment. A Safety Traffic Management Plan governs light-vehicle circulation in this sector and facilitates entry to the administration area, change room and office building. The laboratory and warehouse are also located in this area.

Most of the existing site infrastructure is within the blast radius of the mining operation and will be decommissioned. Existing electrical infrastructures will initially be used for temporary power but will be dismantled to accommodate the new site layout.

## 18.3 Site Geotechnical

A geotechnical investigation, including field work, laboratory analyses and reporting, was completed by NewFields in 2021 for the proposed location of the process plant and infrastructure, DSTF and waste rock facility (WRF). The field work included test pits, borehole drilling, and geophysical surveys in the plant, DSTF and WRF areas. Laboratory testing was completed on soil and rock samples at in-country laboratories, while specialized testing was completed at NewFields' Elko, Nevada, laboratory.

Foundation recommendations were established for the following:

- primary crushing plant, including crushed ore stockpile
- grinding circuit
- leach and CIP circuit
- thickeners
- tailings filter plant
- elution plant
- DSTF and waste rock facility (WRF).

Laboratory testing of different composite tailings samples produced by Bluestone was conducted at NewFields' Elko laboratory in Nevada.

Based on the results of the 2021 work, further investigations for detailed design have been proposed to be completed in 2022 and include the following:

- additional drilling and down-hole geophysical surveys, additional test pitting to supplement the project-wide field campaign conducted to date in the contingency waste rock area
- cone penetration tests (CPTs) in the WRF area
- additional laboratory evaluations of the various strata.

If needed, additional work in areas with liquefaction potential will be undertaken to better understand whether the material would be susceptible to liquefaction or display a cyclic softening behaviour.

A site-specific Seismic Hazard Assessment Report was completed in 2020 by NewFields and will be used in the detailed design and geotechnical analyses of the DTSF and WRF.

## 18.4 Fresh Water Supply

To minimize additional water withdrawals, the mine water will be the primary source of fresh water. Open pit and/or cooled dewatering well water will be the source for dust control.

Mine dewatering well water will undergo primary treatment for cooling and metals removal in the water treatment plant. The treated effluent will then supply the domestic water treatment plant where water will be further treated and disinfected for distribution to on-site camp, office facilities, mine service center, and staff / contractor dining facilities for washing, laundry, and bathing. Drinking water and cooking water will be provided by purchasing potable bottled water from a local vendor.

## 18.5 Surface Water Management

Potential contaminant sources are areas where runoff water is in contact with mineralized material or tailings and operation reagents or consumables. This includes the process plant, DSTF, waste rock facility, open pit, and the balance of the plant footprint. Water runoff from these areas is referred to as “contact” water. “Non-contact” water does not fall onto or run off from these facilities. The process plant area where filtered tailings will be produced, placed, and loaded into trucks prior to being offloaded at the DSTF will produce contact water, along with the surface runoff and seepage water from the DSTF and WRF. All contact water will be sent to the water treatment plant (WTP). Non-contact water will be diverted via channels around the developed areas and released at specified points.

A total of five stormwater conveyance channel sections will divert stormwater around the facilities and discharge non-contact water directly into existing drainages. The stormwater channels consist of excavated trapezoidal channels lined with geotextile and riprap.

The contact water management configuration contains four ponds, as follows:

1. DSTF contact water pond
2. waste rock contact water pond #1

3. waste rock contact water pond #2
4. plant site contact water pond.

Process water within the milling, leaching, and filtration circuit will be contained within the circuit and any excess stored in lined ponds within the plant site.

Contact water will be stored within the contact water ponds as it is generated from precipitation runoff and seepage through the DSTF. It will then be pumped to the WTP feed pond.

During storm events, the portion of the storm water that does not infiltrate into the filtered tailings will either be collected within prepared sumps constructed on the surface of the DSTF with compacted tailings or will flow as runoff over the surface of the waste rock shell and be collected by a lined channel at the toe of the DSTF. This surface water will be treated as contact water and will be collected and pumped back to the plant for reuse on the process or treated prior to discharge.

The contact water ponds are sized to retain the 100-year, 24-hour storm. Their respective pumping capacity to convey the contact water to the WTP has been based on the outputs of the site-wide water balance, utilizing historic climate record from 1970 to 2017.

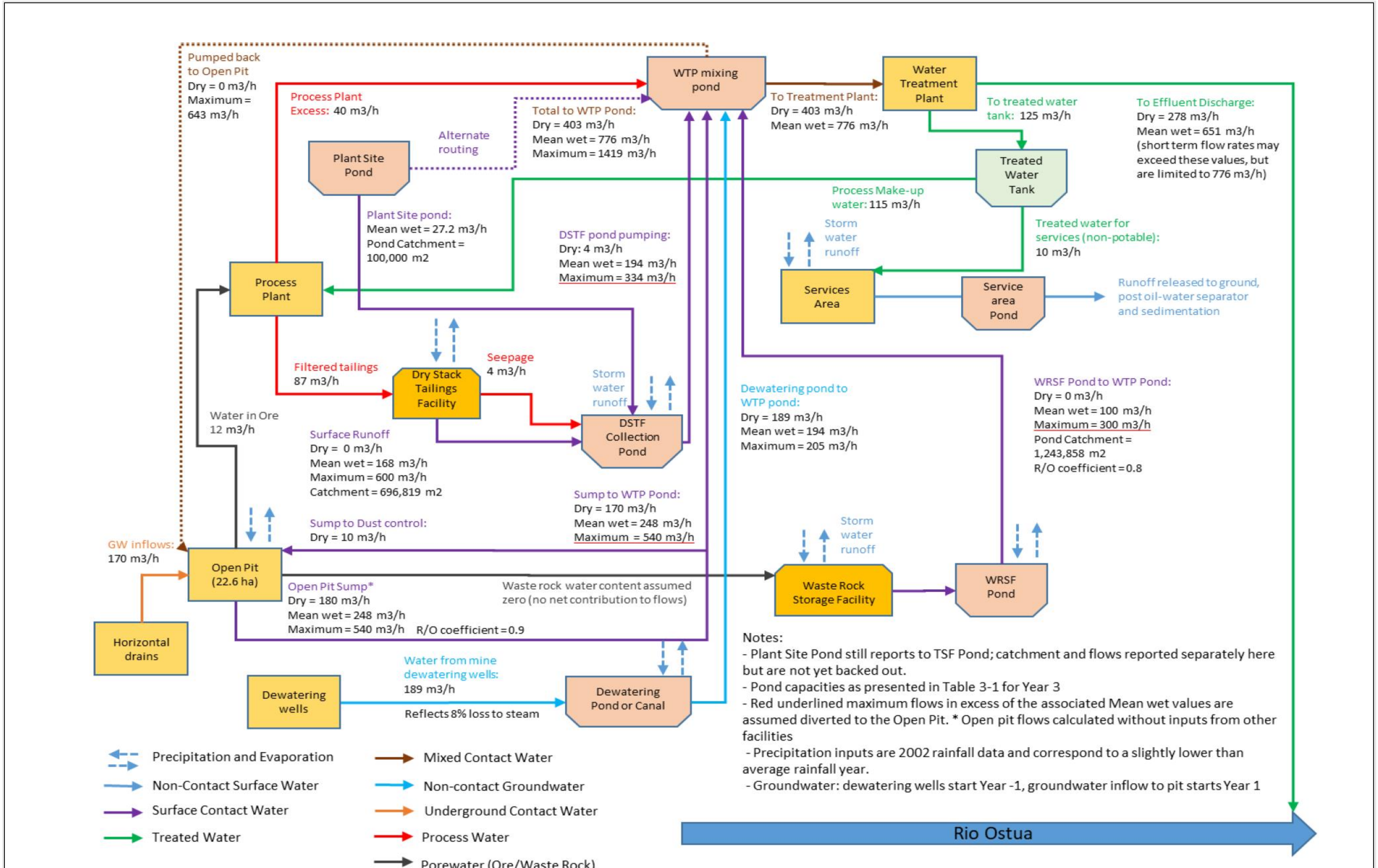
Given the proximity of the site to two water courses - *Quebrada Tempisque* and *Rio Tancushapa* - a floodplain analysis was performed for the 100-year flood and probable maximum flood (PMF) events, calculated on the probable maximum precipitation (PMP) depth of 450 mm. Two flood protection berms were considered to divert the water in case of a flood event. Optimized culvert sizing will be performed in the detailed engineering phase.

## 18.6 Site Water Balance

The site water balance was developed to inform the management of water and the sizing of associated infrastructure (ponds, piping, pumps, WTP). Other potential uses requiring water, such as dust suppression or water requirements for operations, were not simulated. This water may be taken from the WTP, from non-contact water ponds, or other local ponds developed during construction. The water balance was designed utilizing the management strategy of separating contact and non-contact water, while integrating planned mine dewatering and mine processing water requirements. The Cerro Blanco Gold Project is a water-positive mine, meaning there is more water than needed for operations. Excess water will be sent to the WTP and discharged to the Rio Ostua.

The water balance modelling analysis was performed in GoldSim (Version 12.1). The primary modelling inputs included runoff areas reporting to the contact water ponds, precipitation and pan evaporation data collected from two weather stations within 5 km of the project site, the proposed dewatering schedule, water inflows and outflows to the process plant and filter plant, and inflows and outflows from the WTP. The conceptual water balance model for Year 3 is shown on Figure 18-2.

**Figure 18-2: Site Water Balance Year 3 Footprint**




Source: Bluestone (2021).

The water balance model was used to:

- assess scenarios for the process bleed requirements
- confirm the treatment requirements (flows and contaminants concentration)
- estimate the required pumping capacity from the contact water ponds to WTP under typical and extreme operating conditions
- determine pumping rates for mine dewatering and contact water pond level management
- evaluate the capacity to use collected contact water as make-up water in the process plant.

## 18.7 Dewatering Infrastructure

Eight dewatering wells and the supporting infrastructure required for their operation will be installed in the periphery of the pit to ensure the water table is below the bottom of the pit to reduce inflow in the pit and pressure on pit walls.

The depth, location and dewatering rate were established based on the updated numerical hydrogeological model (Stantec, 2021). The water will be conveyed to the WTP for treatment.

## 18.8 Water Treatment Infrastructure

### 18.8.1 Mine Water Treatment Plant

The contact WTP will be built in two stages: the first stage will provide 600 $m^3$/h capacity for up to Year 3 of the mine life and the second stage will increase capacity to 1,200 $m^3$/h to meet the treatment requirement to end of the mine life. The capacity and design have been established based on the latest water balance model, which includes a mass balance module for influent water quality predictions (ERM, 2021).

Water from the dewatering wells will be treated in a separate stream at a rate of up to 1,065 $m^3$/h. Water from this stream will undergo cooling using cooling towers followed by inline coagulation and flocculation using ferric sulphate and flocculant to precipitate arsenic and other dissolved metals and a final step of multimedia filtration to remove metal precipitates.

The main components of the contact WTP are as follows:

- cooling of mine water to ambient temperature using ponds and cooling towers
- trace cyanide destruction of process plant and DSTF streams
- influent mixing into feed pond
- high-density sludge process for metal precipitation
- first stage multimedia filtration to remove metal precipitates
- biological nitrification / denitrification for nitrogen species removal
- second stage multimedia filtration to remove suspended sludges.

The WTP has been designed to produce a treated effluent that meets Guatemala discharge regulation and IFC EHS mining effluent limits, whichever is more stringent. In addition, performance objectives have been established beyond the discharge criteria for key parameters to further protect the receiving environment.

### 18.8.2 Potable Water Treatment

Potable water will be provided by purchasing bottled water from a local vendor, eliminating the need for a potable water treatment facility.

### 18.8.3 Domestic Water Treatment

Fresh water from a treated deep water well will be further treated at the domestic water treatment plant using filtration, reverse osmosis, ion exchange for boron removal and disinfection. Treated domestic water will be used for showers, toilets, laboratory, and laundry. A consumption of 90 $m^3$/d has been used for plant design.

### 18.8.4 Sewage Treatment

Sanitary sewage water from the process plant and main infrastructure site will drain by gravity to the sewage treatment plant. The sewage treatment plant consists of a two-stage biological treatment (pre anoxic and aerobic reactors), membrane filtration, and disinfection.

Once treated, the sewage water will be discharged. Sludge from the sewage treatment plant will be disposed of off-site.

### 18.8.5 Reagent Handling & Storage

Reagents consumed within the WTP, domestic and sewage treatment plants will be prepared on site and distributed via various reagent handling and make-up systems. These reagents include hydrated lime, hydrochloric acid, sodium hydroxide, copper sulphate, sodium metabisulphite, antiscalant, flocculant, ferric sulphate, acetic acid, phosphoric acid, sodium hydrosulphide and sodium hypochlorite.

For the management of unexpected reagent spills, the reagent preparation and storage facilities will be located within containment areas designed to accommodate 50% more than the content of the largest tank. Where required, each reagent system will be located within its own containment area to facilitate its return to its respective storage vessel and to avoid the mixing of incompatible reagents.

Storage tanks will be equipped with level indicators, instrumentation, and alarms to ensure spills do not occur during normal operation. Appropriate ventilation, fire and safety protection, eye wash stations and showers, and material safety data sheet (MSDS) stations will be located throughout the facilities. Sumps and sump pumps will be provided for spillage control.

The reagents will be mixed, stored, and delivered to the thickeners, leach, CIP, acid wash, elution, and cyanide destruction circuits. Dosages will be controlled by flow meters and control valves. The capacity of the storage tanks will be sized to typically handle three days of production. The reagents will be delivered in dry form or solution.

## 18.9 Support Infrastructure

### 18.9.1 Fuel Storage & Distribution

A new fuel storage facility will service the mining and site surface fleet with an 80,000 US-gallon (~302,830 L) capacity for approximately seven days of mobile equipment operations including a 1,500 US gallon (5,680 L) tank of gasoline for light vehicles. This infrastructure is located in the mine service area.

An existing diesel fuel storage facility is installed at site for the power generators. It consists of two 10,000 US gallon (~37,500 L) tanks within a concrete containment area and includes fuel dispensing equipment. These tanks will be removed and relocated near the new emergency generator area when ready.

In the event of a grid power outage, the fuel storage capacity for the backup generators will be for four days running all critical support loads.

### 18.9.2 Explosives Storage

The existing explosives storage and magazine will be decommissioned and relocated northwest of the pit, outside of the pit blasting radius (Figure 18-1). The warehouse will be able to store 340 tonnes of explosives and 3,840 detonators, which is approximately three weeks of support for mining operations.

Bi-monthly deliveries will be organized based on operational requirements. The storage building is of concrete and reinforced concrete block construction surrounded by earthen berms and/or rock berms.

### 18.9.3 Solid & Hazardous Waste Management

#### 18.9.3.1 Non-Hazardous Solid Waste

Inert solid waste will be disposed of in an off-site landfill area. A temporary facility is currently located northwest of the site infrastructure. This facility stores the WTP sludge waste until it can be disposed of underground. Waste from this temporary facility will be transported into the DSTF for storage.

To accommodate the new site footprint, an area along the access to the explosives magazine facility will be reserved to temporarily store the solid waste.

#### 18.9.3.2 Hazardous Waste Disposal

Anticipated hazardous waste consists primarily of waste oils, process reagents and laboratory chemicals. Waste oils will be incinerated or recycled by the supplier. Most reagents and chemicals that require disposal will be disposed of within the process; the remainder will be recycled with the supplier.

All cyanide containers or packaging and other reagent containers will be washed using fresh water. Washing will be done in contained areas at the reagent warehouse and storage area. Washing of cyanide containers will comply with the International Cyanide Code standards for disposal of cyanide products. Neutralized products and containers will be disposed of or recycled with the supplier in accordance with local regulations.

Laboratory fire assay wastes may contain small amounts of lead. These wastes, along with any lead contaminated dust from the baghouses, will be disposed of in accordance with local regulations.

Clean-up of hazardous spills on site will be given the highest operating priority and will generally consist of excavating the contaminated soils, neutralizing the affected site, and disposing of and/or neutralizing the affected soils on site or at a licensed facility off site. Site surface equipment will be made available for use in such circumstances.

### 18.9.4 Site Buildings

Buildings will be equipped with smoke, carbon monoxide and heat detectors, and appropriate chemical fire extinguishers looped together and reporting to one main fire alarm panel.

Several of the existing facilities will continue to be used during construction. These are described in the subsections below.

#### 18.9.4.1 Administration Office

Because the existing administration building is located within the blast radius of the mining operation, a new building will be required. The new two-storey building will be used by site management and the administrative team. The existing administration building will continue to be used until pit excavation begins.

#### 18.9.4.2 Security Office & Entrance Gate

As the main entrance location will change, a new security office will be built on the northeast side of the property. The existing security office will continue to be used until the main entrance has been established. The new facility will be built from 40 ft containers modified as a field office with a control room. The existing entrance gate will be closed off once the new security guardhouse and site entrance are ready for use.

#### 18.9.4.3 Personnel Dry

The site currently has a 60-person mine dry, with personnel change areas, showers, bathrooms, lockers, and PPE storage room. Because this facility is located within the blasting radius, a new change room will be built from containers in the future mining service sector. Once completed, the actual dry will be decommissioned.

#### 18.9.4.4 Geology Office

The on-site geology office consists of three modified 40 ft containers in a U–shaped configuration with a roof to provide an open working area to view and log core. This office will be relocated to align with the new site layout.

#### 18.9.4.5 Assay Laboratory Facility

No facility is in place to perform sample preparation and assay testing. A package of partial laboratory equipment was purchased from the recently closed Marlin Mine in Guatemala and transported to site.

A complete sample preparation and assay laboratory facility will be built in the administration area. The facility will include a complete equipment package to comply with international standards and best practices. It will serve as an on-site testing and analytical facility for the geology department.

This facility will be a 14 m x 60 m structural steel building with offices for staff. It will be equipped with an air compressor and storage space for various laboratory consumables.

#### 18.9.4.6 Clinic & Emergency Response Training Room

The existing first aid clinic is within the blasting radius of the mining operation. This facility will remain operative until its relocation within the administration building on the first floor.

#### 18.9.4.7 Safety Office & Training / Meeting Room

An existing office building will house the site health and safety coordinators and surface support technical staff. Adjacent to this building is a single-room structure that will serve as the training facility for site orientation and staff training.

The future canteen building will also be available for larger group meetings.

#### 18.9.4.8 Warehouse & Storage Facilities

An existing small parts warehouse that services ongoing drilling programs will continue to be used until the construction of the new facility.

A new warehouse will be built with containers under a fabric roof in the administration area. Ten lockable shipping containers will be used, including one for a cold storage room and four for clerical office space. The estimated building (24 m x 28 m) will include an open space surface floor on concrete slab for shelving. The surrounding area will be cleared and fenced as a secured laydown area for large parts, and the building will be equipped with pallet racking.

#### 18.9.4.9 Site Workshop & Storage

An existing maintenance facility, truck shop, workshop, and storage area with a wash pad is located north of the WTP. This facility will continue to be used until the new truck shop is constructed and fully functional, after which it will be decommissioned. The new mine maintenance facility and the wash bay are described in more detail in Section 18.9.13.

The process plant maintenance facility consists of a 19 m x 90 m structural steel building with offices for the maintenance team and a secured tool crib area equipped with tools and an air compressor.

An additional locked and secured storage area houses spare dewatering pumps and equipment. Dedicated areas for mechanical, electrical, and instrumentation workshops and lubricant storage are located within the shop and are serviced by a 10 tonne crane.

#### 18.9.4.10 Drill Maintenance Shop

A second storage and maintenance shop is currently used by the contract drillers and exploration team. The shop consists of a roofed working area, 19 m wide x 8 m long, between double-stacked 20 ft shipping containers and includes a back wall. This will be converted in a temporary truck shop during construction.

#### 18.9.4.11 Environmental Offices & Facility

There is an existing office facility for the environmental department consisting of double-stacked 20 ft containers covered by a roof. This serves as the office area, sample preparation and records area.

### 18.9.5 Site Access & Security

An access gate and guard facility with a barrier and fenced gate will be installed at the new site entrance. Preliminary screening of all traffic entering the property will be conducted at this location. A parking lot will be located on the property a short distance from the access gate.

The control building will house the security access control office and provide control of all personnel entering and leaving site. Only approved and security-cleared vehicles will be allowed to proceed beyond this point. Employees and visitors (non-approved vehicles) will park their vehicles at this location before passing through the security control point.

The project property is enclosed by chain-link fencing with barbed wire or razor wire crowns where accessible to the public, and/or four-strand barbed wire fencing to prevent livestock from wandering onto the property. Additional security fencing (chain-link fence with barbed wire or razor wire crowns) will be installed around new installations, including warehouse area, parts of the refinery, and electrical substations.

### 18.9.6 Camps & Accommodation

The permanent camp facilities will be located on the north side of the site, approximately 1.3 km from the process plant. Camp services will include a kitchen, recreation and sports areas, fire protection system, sewage system, water treatment plant, bus shelter, parking, laundry, restrooms, and a welcome center with an administration office.

The camp will have capacity of approximately 500 people and will include services facilities and food preparation facilities. The camp will mainly be for expatriate and national specialized workers from Guatemala City. The camp facilities will be erected early during project construction to accommodate construction personnel.

There will be three types of camps. Type A camps will have individual bedrooms with private bathrooms for management personnel, type B camps will have individual bedrooms with shared bathrooms for second line management personnel and type C camps will have shared bedrooms and bathrooms for non-management personnel.

### 18.9.7 Mine Offices

Technical staff for open pit mining operations will initially use the existing available office space and will be transferred to the administration office building when it is complete. There will not be a separate building for mine offices.

### 18.9.8 Process Plant Buildings

The primary crusher will not require a covered shelter over the gyratory jaw crusher. A 15 m tall mechanically stabilized earth (MSE) wall will be constructed to allow access to the crusher dump pocket. The proposed crusher location will be designed to allow maintenance and mobile crane access to critical equipment.

### 18.9.9 Refinery & Gold Room

The refinery system will be a stick-built building on a concrete slab. A complete security system will be installed with cameras, digital recording, monitors, and the ability to connect this system to the main plant control room.

The refinery and gold room will be separated from other work areas by security fencing and will have a single security-controlled access point.

### 18.9.10 MCC Buildings & Operator Control Rooms

The electrical equipment buildings, which house the motor control centers (MCCs) and operator control rooms for the crusher, process plant, tailings filtering, emergency power and surface infrastructure areas, will be pre-engineered buildings meeting all local electrical and fire codes and regulations. Small, remote electrical rooms in prefabricated shipping containers with pre-installed MCCs and control equipment will be used to reduce the amount of field work. These buildings will be accessed only by authorized personnel.

### 18.9.11 Mill Office & Personnel Facilities

A mill office and personnel facility including metallurgical laboratory will be located next to the process plant. It will be a stick-built building with a concrete slab floor, approximately 14 m x 73 m. The facility will be divided to provide an office and working area, as well as a personnel lunch and washroom facility.

### 18.9.12 Reagent Warehouse & Cyanide Storage

All process plant chemical and reagents, except cyanide, will be stored in a warehouse on a concrete slab with approximately 1200 $m^2$ of covered storage space (approximately 49 m x 24m). The reagents will be segregated by concrete curbs where required to avoid potential cross-contamination. Liquid spillage in this area will be contained within the building.

A separate concrete slab will house the sodium cyanide storage area. Sodium cyanide will be shipped to site and stored within the same iso-containers. The delivery trucks will return with an empty iso-container for exchange. These areas will be within the fenced and gated area of the process plant.

### 18.9.13 Mine Maintenance Facility

The mine truck shop and maintenance facility with a dimension of 23.5 m x 97 m will be in the mine service area north of the DSTF.

The truck shop will have four heavy vehicle service bays, one light vehicle service bay, a shop and weld bay, and an oil change / lubrication bay. An outdoor wash bay will be located nearby. An office area within the truck shop will house the mine maintenance supervisor and planner. A mobile equipment parts and spares warehouse will also be within this facility.

A separate area and container will be dedicated to storing special lubricants and types of grease. A controlled area will be provided around flammable products such as solvents and paints.

The building will be a steel structure with metal cladding and concrete slab-on-grade. A 20-ton service crane will be provided for the service bays. A controlled receiving and storage area will be provided adjacent to the warehouse.

### 18.9.14 Mine Change Room

A new mine change room will be installed in the mine service area for the open pit personnel. A shipping container arrangement covered with a steel-supported fabric roof will provide locker areas between the shower and change room modules. It will serve approximately 150 mining workers with separate areas for men and women.

### 18.9.15 On-Site Water Tanks

Two new fire water tanks will be erected and dedicated for fire protection. They will have a minimum capacity of 500,000 L allowing approximately two hours of firefighting capability. One will be located near the explosive magazine and the second one will be located near the future water treatment plant.

A new treated water tank with 500,000 L capacity will be erected to service the process plant. Adjacent to the treated water tank, a new 50,000 L domestic water tank will provide water for safety showers.

A new 2,500,000 L process water tank will be erected to receive the overflow of the thickeners and service the process plant.

### 18.9.16 Soil Stockpile Areas

Excess topsoil removed during site preparation will be stockpiled on site. The stockpile will remain until the end of operations, when it will be reclaimed and used for site closure.

### 18.9.17 Site Access Road

The project site is currently accessed by gravel road from the eastern edge of Asunción Mita, crossing the Rio Ostua using a bridge (El Tule). The route is not considered suitable for year-round delivery of heavy equipment and materials during construction and operations.

A base case design including a road and river crossing over the Rio Ostua has been prepared to support the cost estimate. The new access road will be able to support the heavy equipment loads anticipated during operations.

The road is designed for a maximum speed of 50 km/h to accommodate two-way traffic. The road design includes a new bridge to be constructed over the Rio Ostua. The new Warren truss bridge construction started on March 23, 2022 and is being built to accommodate HL-93 designs trucks or approximately 48 tonnes.

### 18.9.18 Site Roads

The main site road - including the access road from the access control entrance to the plant and infrastructure site, and various hauling roads to the waste dump, crusher ROM pad, and DSTF - will be developed. Additional ancillary roads will be constructed to access the explosives magazine. Various roads have already been developed to access wells and drilling locations. These will continue to be used wherever practical.

The access road will stay within the property boundary and security fencing from the access point to the plant site.

Mine production will be transported on the mine haul road from the pit ramp to the crusher. Waste rock will be transported from the open pit main ramp to the waste dump via the waste haul road. The DSTF haul road will be used to haul the waste rock required to build the DSTF retaining wall.

Various temporary construction access roads will be made or modified from existing roads for the temporary construction laydown facilities, the staged DSTF construction, and for construction access, where required.

## 18.10 Power Supply & Distribution

### 18.10.1 Power Supply

Electrical power to the project will be supplied from the Jutiapa main substation approximately 35 km west of the project by a single-circuit 138 kV overhead transmission line. The main substation is owned by RECSA.

The total estimated operating power demand for the site, considering the full dewatering and injection pump operation, is provided in Table 18.1.

**Table 18.1: Electrical Load Summary**

| Item | Value | Unit |
|---|---|---|
| Total Operating Load | 37.7 | MW |
| Total Connected Load | 54.3 | MW |

Source: GMS (2021).

### 18.10.2 Surface Electrical Power Distribution

The proposed electrical distribution systems are described in the subsections below.

#### 18.10.2.1 Primary Power Distribution

The main on-site 138 kV switchyard contains a line disconnect device, SF6 dead tank circuit breakers, motor-operated disconnect switches, an overhead transmission structure, and step-down power transformers. Redundant transformer trains are installed, including a main circuit breaker each, a trench for secondary power cables, and power transformers with associated metering and protection devices. Control, protection, and metering devices will be housed in a prefabricated electrical room and located close to the main switchyard equipment.

The main transformers are connected to the main 13.8 kV switchgear at the process plant through underground power cables. Secondary voltage at 13.8 kV is defined as the on-site main power distribution voltage for major loads. Distribution is from the main switchgear to the key electrical rooms/substations at the process plant and in the crusher area through a cable tray or underground cables network.

Power correction equipment, such as harmonic filters, will be distributed and installed close to the major loads.

#### 18.10.2.2 Secondary Power Distribution

The selected secondary distribution voltage levels for the plant are 4.16 kV, 3-phase, 60 Hz (medium voltage) for medium loads and 480 V, 3-phase, 60 Hz (low voltage) for smaller loads. The secondary distribution system will originate at the main electrical room of the process plant (e.g., grinding, crusher, and refinery electrical rooms) and distribute power to each individual end user.

The existing on-site overhead power distribution lines at 4.16 kV will be used for temporary power during construction, but will be dismantled when they are no longer required. A new aerial network will be built to feed secondary loads (e.g., mine, explosive magazine, dewatering and injection wells, surface buildings remote pumphouse, etc.).

The area substation step-down transformers have been based on the area electrical load calculation, with at least 25% growth factor applied and rounded up to the next closest rating level of standard transformers.

### 18.10.3 Emergency Power

The existing on-site power plant has been altered to become the emergency power plant with diesel generators that were received from the Marlin site. Some generators are at the end of their useful life and will need higher maintenance until the new power plant is in place.

A new generator power plant will be installed close to the main switchyard using four generators of 1.8 MW already purchased and sitting at site. These 480 V generators are not operating and will be engineered with step-up transformers to reach output voltages at 13.8 kV for a total capacity of 7.2 MW in run mode. One diesel tank of 10,000 US gallons (approximately 37,500 L) will serve the generators in this area.

The purpose of the standby power system is to provide an alternate source of power to critical process and non-process equipment in the event of failure of the grid power supply. Emergency generators will allow an orderly shutdown of the process plant and maintain the operation of critical loads. The output from the standby power generators will be connected to a common standby synchronized power switchgear assembly.

Upon loss of connection to utility power, the diesel generator units will start to provide power to all area priority substations through the site power distribution network.

The total estimated emergency power demand for the plant is around 4.5 MVA.

### 18.10.4 Construction Power

The existing synchronized and standalone generators will supply power required during the construction phase. Temporary construction generators will be used where required to provide power to remote locations or where distribution from the existing generators is not practical.

## 18.11 Communications / IT

A fibre-optic communication connection installed to site in 2021 will provide sufficient capacity for the construction and operation phases.

The site communications deployment will be by fibre-optic cable distributed among the site facilities with the power distribution infrastructure.

## 18.12 Mobile Equipment

A list of surface equipment has been prepared for the site surface activities. Bluestone has purchased some equipment from the Marlin Mine, and this has been considered for the new equipment required. The existing equipment that will be used during operations is listed in Table 18.2. The new equipment required for the site is listed in Table 18.3.

**Table 18.2: Existing Mobile Surface Equipment**

| Equipment Required | Operating Description | Quantity |
|---|---|---|
| 3 $m^3$ Wheel Loader | Loading material at filter presses | 1 |
| 5 t Telehandler | Building / site equipment maintenance | 1 |
| 15 t Water Truck | Dust control, water supply where required for civil works | 1 |
| Tool Carrier Wheel Loader | Large yard forklift, front-end loader with forks | 1 |
| 20 t Flat Deck Truck with Rigid Boom Crane | Materials management and maintenance support | 1 |
| 5 t Yard Forklift | Medium-size yard material handler with forks | 1 |
| 100 t Rough Terrain Crane | Heavy equipment maintenance/construction support | 1 |
| 60 t Rough Terrain Crane | Light equipment maintenance/construction support | 1 |
| 15 t Carry Deck Crane | Light equipment maintenance/construction support | 1 |
| 60 ft Aerial Work Platform | Construction support and mill/surface maintenance | 1 |
| Pick-up Trucks | Supervisor/crew on-site transport | 21 |
| Ambulance | Ambulance van | 1 |

Source: Bluestone (2021).

**Table 18.3: New Mobile Surface Equipment**

| Equipment Required | Operating Description | Quantity |
|---|---|---|
| Pick-up Trucks | Supervisor/crew on-site transport | 53 |
| Pick-up Trailers | For construction support and to move material and tools | 3 |
| Tractor & Lowboy Trailer | For construction support and to move material and equipment | 1 |
| Ambulance | Ambulance van | 1 |
| Fire Truck (Pumper / ladder truck) | Emergency response | 1 |
| Crane Carry Deck (15 t) | Construction support/process plant maintenance | 1 |
| Crane Rough Terrain (55 t) | Construction support/process plant maintenance | 2 |
| Crane Rough Terrain (130 t) | Construction support/process plant maintenance | 2 |
| Container Handler (Telescopic) | Materials management | 1 |
| Flat Deck Truck with 28 t Rigid Boom Crane | Materials management and maintenance support | 1 |
| 4 t Telehandler | Construction support/material management and mill/surface maintenance | 4 |
| 5 t Telehandler | Construction support/material management and mill/surface maintenance | 2 |
| 2 t Electric Warehouse Forklift | Warehouse forklift for internal/pavement use | 1 |
| 2.5 t Off-Road forklift | Construction support/material management and mill/surface maintenance | 2 |
| 5 t Off-Road forklift | Construction support/material management and mill/surface maintenance | 1 |
| 45 ft Aerial Work Platform (Lift) | Construction support and mill/surface maintenance | 2 |
| 60 ft Aerial Work Platform (Lift) | Construction support and mill/surface maintenance | 2 |
| 80 ft Aerial Work Platform (Lift) | Construction support and mill/surface maintenance | 2 |
| 19 ft Electric Scissorlift | Construction support and mill/surface maintenance | 4 |
| 26 ft Electric Scissorlift | Construction support and mill/surface maintenance | 2 |
| 5.5 ft Flatbed Truck | Construction support/tools and material transport | 5 |
| Multipurpose Truck (Roll-off) | For waste management and 20 ft containers | 1 |
| Waste Containers for Multipurpose Truck | For waste management (waste segregation at the source) | 15 |
| Fuel & Lube Truck | Construction support and mill/surface maintenance | 1 |
| Multipurpose Truck - Vacuum Tank | Site support | 1 |
| Mobile Diesel Generator Light Plants | Safety lights for night work and maintenance | 20 |
| Mobile Diesel Air Compressor (185 CFM) | Construction support and mill/DSTF/surface mobile maintenance | 2 |
| Mobile Diesel Welding Machine Diesel (500A) | Construction support and mill/DSTF/surface mobile maintenance | 15 |
| Electric Welding Machines | Construction support and mill/DSTF/surface mobile maintenance | 10 |
| Portable Generator (2.2 kW - 5 kW - 10 kW) | Construction support and mill/DSTF/surface mobile maintenance | 10-20-2 |
| 2 inch diameter - Electric Submersible pump | Construction support and mill/DSTF/surface mobile maintenance | 20 |
| 3 inch diameter - Gasoline Dewatering Pump | Construction support and mill/DSTF/surface mobile maintenance | 4 |
| 6 inch diameter - Diesel Dewatering Pump | Construction support and mill/DSTF/surface mobile maintenance | 4 |
| Vibrating Plate Compactor (600 mm x 900 mm) | Construction support and surface support | 6 |
| Skid Steer Wheeled loader | Construction support and mill operation | 1 |
| Wheel Backhoe loader | Construction support and surface support | 1 |
| Excavator (5 t) | Construction support and surface support | 1 |
| Excavator (20 t) | Construction support and surface support | 1 |

Source: GMS (2021).

## 18.13 Dry Stack Tailings Facility

The DSTF design consists of a starter facility constructed from earth fill / rockfill and a downstream waste rock buttress. The main components of the initial capital construction stage include the following:

- a portion of the starter waste rock buttress facility
- haul road for tailings delivery (via dump trucks)
- contact water ponds for DSTF and WRF
- DSTF basin regrading
- DSTF underdrainage system and outlet to contact water pond
- foundation drains under the tailings
- contact water channels
- non-contact surface stormwater channels
- reclaim water pipelines from the contact water ponds and DSTF sumps.

Also included are pumps for the contact water ponds and DSTF rainy season sumps as well as associated mechanical and electrical components. A sump will be located within the DSTF at selected locations to collect and remove contact water from the surface. The general layout of the DSTF is provided Figure 18-3.

### 18.13.1 Design Objectives & Criteria

The design objectives and criteria for the DSTF were selected to meet the objective of storing tailings materials in a stable and environmentally responsible manner. Specific design objectives are as follows:

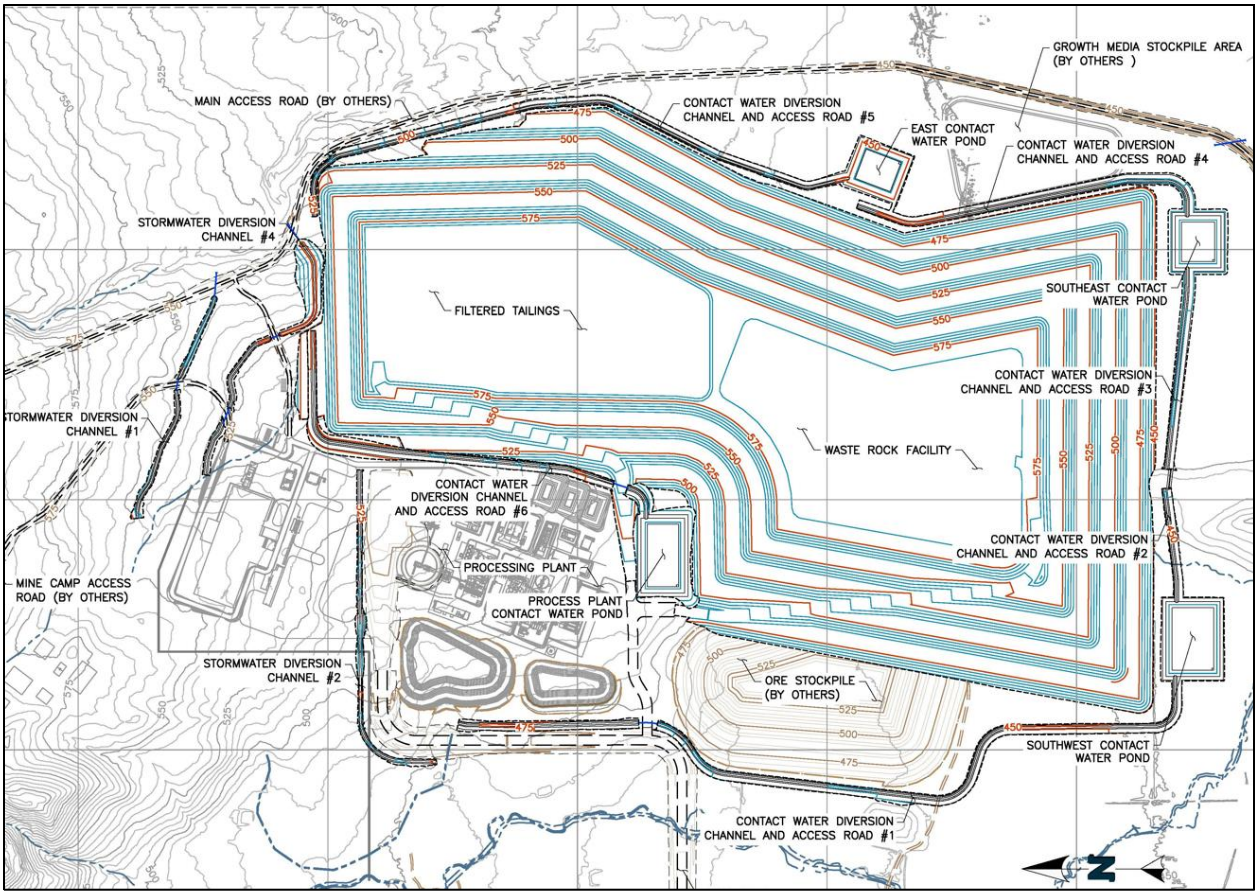
- satisfy internationally accepted stability criteria for tailings facility construction in areas of high seismicity
- follow current engineering practices and technology for all phases of the facility
- satisfy Guatemalan regulatory requirements associated with the construction of the DSTF and WRF.

The goals of this FS are as follows:

- evaluate and select the most appropriate initial and on-going construction methodology
- evaluate the best potential site and preliminary infrastructure footprint
- evaluate the best location of the DSTF embankment, impoundment and associated structures
- develop a geotechnical engineering plan for the field investigation required to complete the detailed engineering
- estimate the required starter facility height and timing for subsequent raises
- perform water balance calculations to estimate the required make-up water (presented in a separate document)

- evaluate the hydrologic and hydraulic conditions and propose stormwater management measures.

**Figure 18-3: DSTF General Layout**




Source: NewFields (2021).

The design basis and operational requirements / assumptions and design criteria adopted for design of the DSTF are presented in Table 18.4 and Table 18.5.

**Table 18.4: Design Basis & Operational Requirements / Assumptions**

| Description | Design Basis |
|---|---|
| **Site Characteristics – Climatic Considerations** | |
| Climate and Vegetation Type | Tropical Dry Forest Environment |
| Average Daily Temperature | 26°C |
| Maximum Temperature | 41°C |
| Minimum Temperature | 10°C |
| Dry Season | November - April |
| Wet Season | May - October |
| Elevation | approximately 500 masl |
| Annual Average Rainfall | 1,342 mm |
| Annual Average Pan Evaporation | 2,533 mm |
| Annual Average Humidity | 62% |
| **DSTF Operational Requirements / Assumptions** | |
| Ultimate Capacity | 54 Mt |
| Life of Mine | 16 years (Y-2 to Y14) |
| Tailings Production (per year) | 4M tonnes |
| Tailings Deposition Methodology | Filtered, hauled with trucks, spread in thin lifts, disced and roller compacted |
| Tailings Geochemistry | Non-acid-generating (NAG) |
| Water Management Basis | Separated non-contact and contact water management systems |
| **Starter Facility** | |
| Overall Tailings Downstream Slope | ~1.3H:1V (angle of repose of waste rock) |
| Overall Tailings Upstream Slope | ~1.3H:1V (angle of repose of waste rock) |
| Minimum Crest Width | 100 m (minimum) |
| Crest Elevation | 180 m (minimum) |
| **Ultimate Facility** | **Variable, 20 m (maximum)** |
| Overall Tailings Downstream Slope | 3H :1V |
| Overall Tailings Upstream Slope | 2.5H:1V |
| Minimum Crest Width | 50 m to 100 m |
| Crest Elevation | 585 masl |

Source: GMS (2021).

**Table 18.5: DSTF Design Criteria**

| Regulations | Measure | Description |
|---|---|---|
| Flood Handling Requirement | Probable maximum flood | Decant to stormwater pond |
| Seismicity / Earthquake Load | 4,750- to 10,000-year return period | Canadian Dam Association, Dam Safety Guidelines, 2019 |
| Slope Stability Minimum Factors of Safety (Limit Equilibrium Method) | Static - 1.5<br>Pseudostatic - 1.0<br>Post-earthquake - 1.2 | Canadian Dam Association, Dam Safety Guidelines, 2019 |
| Maximum Deformation | 1.5 m | For pseudostatic cases with factors of safety below 1, to minimize release of fluid from the impoundment |
| Seepage Collection System | Corrugated polyethylene pipes (CPeP) covered with drain gravel | Network of underdrain pipes at the base of the DSTF to collect seepage and conduct as contact water to the DSTF pond |
| Assumed Power Outage Duration | 12 hours | Seepage during this period to be contained within sump |
| Flood Storage Requirement | 1-in-100-year,<br>24-hour storm | Contain runoff from design event within pond, and assume event coincides with power outage |
| Flood Handling Requirement | Probable maximum flood | Decant to stormwater pond |
| Flood Handling Requirement | Probable maximum flood | Decant to stormwater pond |
| Seismicity / Earthquake Load | 4,750- to 10,000-year return period | Canadian Dam Association, Dam Safety Guidelines, 2019 |

Source: NewFields (2022).

### 18.13.2 Geotechnical Investigations

An FS-level geotechnical and hydrogeological subsurface investigation program was performed in 2021, as discussed in Section 18.3, to characterize the site conditions and develop design criteria for the proposed DSTF. In addition to the 2021 work, several previous site geotechnical investigations have been carried out, as follows:

- A Phase I geotechnical investigation program and updated FS was performed by Golder in 2012 and 2013, respectively.
- A third FS-level geotechnical program was performed by Stantec between May and July 2018 to assess and characterize site conditions for locating the DSTF.

Although the 2012, 2013 and 2018 investigations were carried out for a DSTF site located outside of the project boundaries, the subsurface and laboratory data collected from these programs was used to the extent possible for the engineering analysis for the current DSTF.

### 18.13.3 Tailings Geochemistry

#### 18.13.3.1 Geochemical Test Results from the 2020 to 2021 Testing Program - Tailings

The 2020-2021 geochemical characterization program consisted of two phases of sample selection. The first was based on an underground mine plan, and the second included additional sample selections based on the open pit. Most of the samples collected for the underground mine plan were in the zone of saturated groundwater; conversely, the samples selected for the open pit mine plan are predominately from the unsaturated zone. Thus, geochemical properties are expected to be different for samples from the unsaturated zone due to a geological history of what is essentially supergene enrichment from natural chemical weathering processes associated with oxidation of sulphide minerals (to the extent present), chemical weathering of mineral phases, transport by percolating meteoric water, and re-precipitation at the oxidation-reduction (redox) boundary of the saturated zone. The process is enhanced where the sulphide mineral content is high but would occur regardless of the sulphide content. For this reason, the adoption of the open pit mine plan required an update to the original geochemical characterization sampling plan as is described herein.

Sampling for geochemical characterization occurred under three phases that correspond to the evolution of the mine plan: Phase I is the underground, Phase II is the transition to open pit (few samples available), and Phase III is the mature open pit mine plan.

#### 18.13.3.2 Phase I Sampling Plan – Underground

The initial geochemical characterization program by Stantec was to support the underground mine plan for the project. The underground mine included the North and South Zones, primarily targeting Mita Group lithologies, but also the Svc of the Salinas Group. The mine plan specified that tailings would be processed to generate cemented paste for mine backfill and dry tailings would be placed in the above-ground DSTF. Prior to the bench-scale generation of cemented rock fill (CRF) and cemented paste, the mine plan was modified to an open pit and therefore the backfill-related material was not generated or subject to geochemical tests.

Dry stack tailings are assumed to be relatively homogenous due to processing, which includes blending the ore feed for optimization of gold recovery, milling, gold extraction, and tailings dewatering. Because of the handling of ore and associated homogenization, processed materials were not divided by the major lithological units.

In total, 13 Phase I tailing samples were selected and analysed geochemically; 28 metallurgical variability samples had been generated from individual lithologies within eight vertical panels in the north and south underground zones (i.e., four panels per zone). For geochemical characterization, and due to limited sample mass, these 28 samples were composited into nine samples, with samples grouped by zone and lithology. Four additional master composites were generated to represent both zones and similar lithology groups. Prior to geochemical testing, the 13 tailings samples were subjected to cyanide detoxification.

#### 18.13.3.3 Phase II Sampling Plan – Open Pit

In early 2021, Bluestone submitted a single tailing sample for geochemical analyses: sample BL 727-03, which was a composite of Svc, Tbx and Ss material intended to increase representativeness of open pit lithologies in the sample database, while waiting for completion of the bench-scale generation of tailings from the open pit mine plan. Later in 2021, an additional tailings sample was included for geochemical characterization and to include the effect of the carbon-in-pulp process on metals availability. As of the time of this report, the availability of additional samples is pending, and no Phase II geochemical characterization samples had been submitted aside from BL 727-03 from Phase IIa and BL723-112+113 from Phase III.

For Phases I, IIa, and III, 15 samples (as of the date of this report) have been analysed for acid base accounting (ABA), net acid generation (NAG) pH, and shake flask extraction (SFE). Based on ABA testing, which determines the neutralization potential (NP) and acid potential (AP), five of the samples classify as potentially acid generating (PAG), four are uncertain, and six are non-PAG (Table 18.6). This summary is focused on the analyses for samples associated with the former underground mine plan with only two samples collected specifically for the open pit design. Therefore, the geochemical characterization of tailings for the open pit mine plan is on-going pending further metallurgical processing.

**Table 18.6: Acid Base Accounting Results for Cerro Blanco Tailings**

| Bluestone ID | LabID | Lithology | Zone[1] | NP (T/kt) | AP (T/kt) | NP/AP[2] |
|---|---|---|---|---|---|---|
| NP-MBT-MSS | BL593-80 | Mbt-Mss | North UG | 9.1 | 19.4 | 0.47 |
| NP-SVC | BL593-81 | Svc | North UG | 9.7 | 17.8 | 0.54 |
| NP-MAT | BL593-82 | Mat | North UG | 13.2 | 13.8 | 0.96 |
| NP-MCV-MVO | BL593-83 | Mcv-Mvo | North UG | 39.9 | 22.8 | 1.75 |
| NP-MLS | BL593-84 | Mls | North UG | 8.5 | 8.1 | 1.05 |
| SP-MAT | BL593-85 | Mat | South UG | 30.9 | 17.5 | 1.77 |
| SP-MLS | BL593-86 | Mls | South UG | 330.0 | 7.2 | 45.91 |
| SP-MBT-MSS | BL593-87 | Mbt-Mss | South UG | 85.6 | 15.6 | 5.48 |
| SP-MCV | BL593-88 | Mcv | South UG | 156.9 | 11.3 | 13.95 |
| - | BL593 Comp 1 | Mbt-Mat-Mss | UG | 32.8 | 17.5 | 1.87 |
| - | BL593 Comp 2 | Mcv-Mvo | UG | 74.5 | 13.4 | 5.54 |
| - | BL593 Comp 3 | Svc | UG | 5.8 | 16.6 | 0.35 |
| - | BL593 Comp 4 | Mls | UG | 308.7 | 8.1 | 37.99 |
| - | BL 727-03 | Svc-Tbx-Ss | Pit | 6.8 | 1.6 | 4.35 |
| | BL723-112+113 | Svc-Tbx-Ss | Pit | 8.0 | 25.6 | 0.31 |

Note:[1]UG = Underground. [2]NP/AP ratios below 1 (PAG) are highlighted in yellow; NP/AP ratios between 1 and 2 (Uncertain) are highlighted in grey. Source: Stantec, 2022.

Shake flask extractions were performed on the 15 tailing samples, which comprise composite material representative of single or multiple lithologies. All tailings SFE solutions were near-neutral in pH. Mercury was the only constituent exceeding International Finance Corporation (IFC) and/or U.S. Environmental Protection Agency (EPA) standards, with a maximum concentration of 17.9 µg/L. Mercury concentrations from SFE were compared with whole rock mercury content of the same samples. The higher mobility of mercury in tailings is hypothesized to be the result of contact with cyanide during gold extraction process. However, Stantec notes that the tested tailings samples were not fully processed through the carbon-in-pulp (CIP) circuit, which would likely remove both gold and mercury from the tailings material. Additional testing of tailings that has gone through the complete ore process flowsheet was completed to confirm that the CIP process removes mercury. The subsequent concentration of the test sample was below detection limit.

Humidity cell testing (HCT) included five tailings samples. Four tailings HCT samples classified as PAG or uncertain by ABA testing and one sample was non-PAG. At the time of this report, results for all tailings HCTs were available to week 41 with the exclusion of one sample which was terminated at week 25. All five samples have produced near-neutral leachate with minimum pH values of 7.0 to 7.8. Furthermore, pH values have been relatively constant with subsequent HCT cycles. A sixth HCT was initiated on a PAG Salinas composite sample and results to week 13 were available at time of this report. This sample has also generated circumneutral pH. Circumneutral leachate from PAG material suggests that NP is readily available and can neutralize acid generated by sulphide oxidation.

Mercury was the only constituent concentration exceeding IFC or US EPA water quality guidelines. As stated previously, the tailings samples available were not fully processed by CIP.

The results of Phases I, IIa, and III testwork confirm the results from previous work where tailings were classified as non-acid-generating and non-metal-leaching.

Based on the information presented herein, the tailings are considered to be non-PAG and non-metal leaching, and no liner is included in the DSTF design.

### 18.13.4 Dry Stack Tailings Facility Design

Tailings from the milling process will be further dewatered through filtration before being transported to the designated filtered tailings facility. The DSTF is designed to accommodate 54.7 Mt of filtered tailings.

#### 18.13.4.1 Material Balance

A material balance model was developed at an FS-level design for the DSTF. The purpose of the material balance was to estimate the size and elevation of the waste rock and filtered tailings on a bi-annual basis throughout the operation of the DSTF and WRF. The model was also used to establish the regrading required within the tailings foundation area, the overall footprint of waste rock and tailings, a plan for placement of waste rock, tailings, and structural tailings in a bi-annual basis, and to estimate the ultimate capacity and geometry of both facilities at the end of the life of the mine.

#### 18.13.4.2 Layout & Staging

The initial stage of the DSTF, constructed partially during the first year of the mine’s expected 25-month pre-production period, will provide storage capacity for at least the first wet season of tailings production.

The initial stage comprises regrading, partial construction of the DSTF toe perimeter berm, seepage collection piping system, DSTF pond, and starter buttress constructed of mine waste material.

The facility will be continuously raised during operation, maintaining a high-strength perimeter shell comprised of structural tailings (moisture conditioned and compacted) and waste rock. The outer structural zone of the filtered tailings will be constructed of compacted filtered tailings to keep a 100 m minimum width of structural tailings all around the tailings area, up to elevation 565 (Year 12), which is reduced to a minimum of 50 m on the final lifts. The compacted structural tailings will be placed during the dry season, and during the wet season as necessary. Placement will be done with the use of trucks, bull dozers, motorgrader and conventional compaction equipment. Sufficient drying of the tailings in the compacted zone will be required to meet moisture and density specifications. The successful compaction of this structural zone is vitally important. Annual construction of the dry season structural zone is designed to provide capacity of the subsequent wet season, where non-structural tailings will be placed in the center of the facility.

The estimated total capacity of the ultimate facility, after 14 years of tailings production, will be 54.7 Mt. Considering an average dry density of 1.50 t/m$^3$, this equates to approximately 36.4 Mm$^3$ of required capacity.

To help reduce the likelihood of increased pore pressures within the tailings stack over time, a network of seepage underdrains will be installed at the foundation level of the facility. Water collected within these pipe networks will report to the contact water pond located southeast of the DSTF area. A sump has been designed within the pond, and pumps will be installed to return collected water to the WTP mixing pond for treatment prior to use back in the process or final discharge. The foundation underdrain systems will be stage-constructed ahead of tailings deposition as much as practicably possible.

#### 18.13.4.3 Filtered Tailings Placement Considerations

The filtered tailings will be transported from the filter plant stockpile to the DSTF via a fleet of trucks. Following the construction of the starter facility, filtered tailings will be deposited as structural or non-structural tailings, as described above.

### 18.13.5 DSTF Conceptual Closure Plan

Because the DSTF is continuously buttressed with waste rock over its operational life, and the waste rock shell is constructed as a regular waste rock dump, concurrent partial reclamation is possible as the lower waste rock lifts are abandoned. At the end of operation, the slopes of the WRF and DSTF will be regraded to smooth out the construction benches, to a general grade of 2.5H:1V. The foundation underdrains will continue to report to the contact water pond and pumped to the water treatment plant. Eventually, when flow rates diminish, the pond will be converted into passive treatment system.

The final top surface of the DSTF will be regraded to a convex shape so it sheds precipitation, rather than impound it. A layer of crushed or selected waste rock and alluvium fill will be placed over the final grades of waste rock and tailings. Topsoil that was stockpiled from the DSTF footprint will be spread over the top and regraded slope surfaces of the DSTF and WRF. A native grass seed mixture will be planted to reduce erosion.

## 18.14 Waste Rock Facility

The current mine plan has all waste rock reporting to the DTSF area as shown on Figure 18.3. The WRF at the DSTF area has been designed with a capacity to store up to 170 Mt of waste rock assuming an in-place dry density of 1.6 t/m$^3$, which is equivalent to approximately 106 Mm$^3$, providing an additional contingency in excess of 15% over the 145 Mt anticipated per the current mine plan. Should additional waste rock be developed, a contingency waste rock dump (WRF) is located west of the open pit, adjacent to the Cerro Blanco hill. The facility has been designed to accommodate in excess of 47 Mm$^3$ of waste rock.

### 18.14.1 Previous Testwork

Testwork performed on waste rock and ore in 2006, 2012 and 2015 reported mixed conclusions ranging from the ore being mostly non-PAG (48%) to the waste rock being mostly PAG (45%) and to PAG being identified, but not quantified, in waste rock.

### 18.14.2 Geochemical Test Results from the 2020 - 2021 Testing Program

The waste rock geochemical characterization program consists of three (3) phases of sample selection: (1) the first was based on an underground mine plan; (2) the second included additional sample selections based on the open pit; and (3) third included additional waste sample for improved spatial and lithological distribution and to align Mine Environmental Neutral Drainage (MEND) Canadian guidelines on sample density. Most of the samples collected for the underground mine plan were from within the saturated groundwater system; conversely, the samples selected for the open pit mine plan are predominately from the unsaturated zone. Thus, geochemical properties are expected to be different for samples from the unsaturated zone due to a geologic history of what is essentially supergene enrichment from natural chemical weathering processes associated with oxidation of sulphide minerals (to the extent present), chemical weathering of mineral phases, transport by percolating meteoric water, and re-precipitation at the redox boundary of the saturated zone. The process is enhanced where the sulphide mineral content is high but would occur regardless of the sulphide content. For this reason, the adoption of the open pit mine plan required an update to the original sampling plan as is described herein.

Sampling for geochemical characterization occurred under three phases that correspond to the evolution of the mine plan: Phase I is the underground, Phase II is the transition to open pit (few samples available), and Phase III is the mature open pit mine plan.

#### 18.14.2.1 Phase I Sampling Plan – Underground

The initial geochemical characterization program developed by Stantec was to support the underground mine plan for the Project. The underground mine included the North and South Zones, primarily targeting

Mita Group lithologies but also the Svc of the Salinas Group. A total of 106 samples were submitted for static testwork.

#### 18.14.2.2 Phase II Sampling Plan – Open Pit

Additional geochemical characterization of waste rock from the open pit was initiated in 2021. This work expanded the Phase I program that focused on the previously proposed underground mine plan (106 samples). In early 2021, three (3) samples were selected to begin characterizing waste rock lithologies that would be encountered in an open pit scenario. These included the Svc, Ss and Tbx lithology units, all of which overlie the Mita Group, which was the main geologic unit of the underground mine plan and target initial phase of geochemical characterization. On further review, the determination was made that the waste rock from the open pit, primarily from the Salinas Group, was under-represented and an additional 81 samples were selected to characterize Salinas Group materials from the open pit.

#### 18.14.2.3 Phase III Sampling Plan – Open Pit Density & Spatial Distribution

Following the selection of 187 samples for geochemical characterization of waste rock that encompasses both the underground and open pit mine plans in Phases I and II, the open pit mine plan was further refined, increasing the estimated waste rock tonnage from 123 metric tons (Mt) to 145 Mt and increasing estimates of waste rock tonnage by lithology. The geochemical characterization program was again evaluated for data gaps based on this new information. An estimated 125 additional waste rock samples were identified to characterize the four major waste rock lithologies of the open pit: Svc (48% of waste rock tonnage), Mbt (21%), Ss (10%), and Mcv (8 %).

The 125 additional samples were selected from core material within the planned open pit volume, with a focus on portions of the open pit that were previously under-sampled. Spatial variability throughout the pit volume is important to characterization as waste rock peripheral to the ore body may be geochemically different than waste rock in mineralized zones in the immediate vicinity of the ore body. Furthermore, characterization in the vicinity of the pit wall is especially important as the pit walls can have strong influence on post-closure pit lake chemistry.

#### 18.14.2.4 Results of Phase I & II

The two main lithologies within the total waste rock tonnage are Svc at 47.6% and Mbt at 21.5%. Both classify as uncertain based on a median ABA NP/AP ratio of 1.26 each (Table 18.7). For these main lithologies, 42% and 37% of Svc and Mbt samples analysed classify as PAG by ABA, respectively. Scgl and Mat units, both classify as PAG based on median values of ABA and represent 3.6% of the total waste rock tonnage jointly. The Sinter and Mcv are the 3$^{rd}$ and 4$^{th}$ most significant lithologies at abundances of 10.2 and 8% respectively and both classify as non-PAG based on median NP/AP ratio. Mls has the highest NP/AP but only represents 1.5% of the total tonnage. In summary, 36% of the total waste rock tonnage from the open pit is projected to be PAG, 15% as uncertain and 49% as non-PAG according to static testing (Table 18.8).

**Table 18.7: Median Values of Acid Base Accounting for Cerro Blanco Waste Rock from Open Pit**

| Lithology | Percent of Waste Rock | Sample Count | Paste pH | Acid Base Accounting | |
|---|---|---|---|---|---|
| | | | | Sobek NP/AP1 | Percent PAG |
| Svc | 47.6% | 72 | 5.39 | 1.26 | 42% |
| Rp | | 5 | 6.15 | 38 | 0% |
| Tbx | | 22 | 2.57 | 0.24 | 77% |
| Ss | 10.2% | 47 | 4.72 | 22.7 | 2% |
| Scgl | 2.8% | 16 | 3.68 | 0.23 | 88% |
| Mbt | 21.5% | 67 | 3.43 | 1.26 | 37% |
| Mcv | 8.0% | 38 | 6.35 | 2.81 | 21% |
| Mvo | 2.7% | 11 | 6.43 | 1.76 | 27% |
| Mat | 0.8% | 16 | 5.20 | 0.65 | 63% |
| Mls | 1.5% | 15 | 6.07 | 69.7 | 27% |
| Mss | 0.9% | 6 | 4.89 | 1.71 | 33% |
| Other | 4.0% | - | - | - | - |

Note 1: NP/AP ratios below 1 (PAG) are highlighted in yellow; NP/AP ratios between 1 and 2 (Uncertain) are highlighted in grey.
Note 3: Predominantly Svc with minor fractions of the Rp and Tbx units

**Table 18.8: Acid Base Accounting Results Categorized by Sobek NP/AP Ratio & Weighted to Waste Rock Tonnage**

| Parameter | Non-PAG | Uncertain | PAG | All |
|---|---|---|---|---|
| Percent of Waste Rock | 48.8 | 15.5 | 35.8 | 100.0 |
| Paste pH | 6.55 | 6.57 | 5.54 | 6.19 |
| Sulphide Sulphur (%) | 0.09 | 0.27 | 0.79 | 0.37 |
| Total Sulphur (%) | 0.06 | 0.22 | 0.70 | 0.32 |
| TIC NP (T/kt) | 12.75 | 2.68 | 0.40 | 6.78 |
| Sobek NP (T/kt) | 19.1 | 9.69 | 6.43 | 13.1 |
| AP (T/kt) | 1.98 | 6.99 | 21.9 | 9.88 |
| TIC NP/AP[2] | 6.45 | 0.38 | 0.02 | 0.69 |
| Sobek NP/AP[2] | 9.68 | 1.39 | 0.29 | 1.33 |

Shake flask extraction concentrations for waste rock are summarized herein from the *Geochemical Characterization and Mine Water Source Terms Cerro Blanco Mine*, Guatemala (Stantec, 2022) For reference, water quality was compared to the IFC and US EPA (40 CRF § 440.102a) mining discharge water quality standards.

The Svc rock represents the greatest tonnage of material in the waste rock category. SFE concentrations from the Svc that exceed IFC and/or US EPA standards include pH, arsenic, copper, iron, mercury, nickel, and zinc. Water quality standard exceedances in other rock units include those listed for Svc in addition to cyanide in a single Tbx sample. While the SFE data are indicative of potential for a constituent to rinse from a given rock material, the results might represent the history of the sample analysed rather than how the rock would respond to field conditions in a waste rock facility. Humidity cell testing, a kinetic test method, followed by water quality prediction modelling that is on-going will provide a more robust measure of the potential for constituents to be generated from waste rock.

Waste rock humidity cell testing includes 31 waste rock samples to date. By ABA, 14 of these samples were PAG, five were uncertain, and 12 were non-PAG.

At the time of this report, 21 of the waste rock HCT have results through week 41 of testing. In total, 11 HCTs have produced acidic leachate with minimum pH values of 2.3 to 5.0. Values of pH have been relatively constant with subsequent HCT cycles. An exception is a cell of Mls that classified as PAG, which had a pH above 7.0 at week 0 and then decreased to below 3.0 at week 15 of testing. All samples classified as PAG via ABA became acidic with minimum pH values between 2.1 and 3.8. Aside from pH, constituent concentrations exceeding IFC or US EPA water quality guidelines include arsenic, copper, iron, mercury, nickel, and zinc. Exceedances have only been measured in acidic samples; there is no evidence of neutral metal leaching. Generally, metal concentrations have been elevated at week 0, followed by one of two trends: (1) decreasing and stabilizing concentrations or (2) decreasing, then increasing, and stabilizing concentrations. Fluctuations in metal concentrations are generally inversely correlated to changes in pH (i.e., decreasing pH is accompanied by increasing metal concentrations).

Sulphate concentrations have generally been highest at week 0, followed by exponential declines and stabilizing low concentrations. Seven of the HCTs had a late onset of increased sulphate concentrations. Stable sulphate concentrations indicate constant rates of sulphate production via sulphide oxidation. Stabilized sulphate concentrations are generally less than 20 mg/L per week in circum-neutral samples and approximately 50 to 400 mg/L per week in acidic samples.

Samples designated as Uncertain had a wide range of minimum pH values ranging between 2.7 and 6.9; acidic samples had NP/AP ratios of 1.19 or lower. However, sample HC 16 had an NP/AP ratio of 1.17 and remained circumneutral. Therefore, more than the NP/AP ratio determines whether these materials become acidic. Factors such as kinetic availability of NP and AP likely affected the pH of the uncertain samples. Uncertain samples that became acidic also had metal concentrations exceeding IFC Mining Effluent guidelines, including copper, iron, and zinc. The slightly acidic HC 10 and circumneutral HC 16 did not have any metal exceedances.

Samples classified as non-PAG via ABA had minimum HCT pH values between 4.7 and 6.9. None of these samples had metal concentrations exceeding IFC Mining Effluent guidelines. Slightly acidic to acidic pH values in these samples were likely driven by low-level sulphide oxidation coupled with NP that was sufficient to buffer the pH at somewhat acidic pH values.

A second phase of 10 HCT samples were initiated in September 2021 and results were available up to week 13 at the time of this report. The initial results are generally consistent with observations made previously.

Based on the results of chemical characterization of waste rock for the open pit, there is reasonable likelihood that a portion of the waste rock will produce low pH, metal-bearing seepage that could affect receiving water quality. Potential mitigation measures will be evaluated and include addition of limestone to increase the neutralizing capacity of waste rock, as well as special handling of PAG rock to the extent feasible to limit oxidation of sulphide minerals.

# 19. MARKET STUDIES & CONTRACTS

## 19.1 Market Studies

No market study was completed on the potential sale of doré from the Cerro Blanco Gold project. The terms used in the economic analysis, however, are considered to be in line with current market conditions and were sourced by contacting various refiners and transportation companies. The indicative terms were reviewed and found to be reasonable.

The study recommends that as the project advances towards development, a detailed marketing report and logistics study should be undertaken to ensure the accuracy of the terms. Table 19.1 outlines the terms used in the economic analysis.

**Table 19.1: Net Smelter Return (NSR) Assumptions**

| Off-Site Costs & Payables | Item | Unit | Value |
|---|---|---|---|
| Payables for Doré | Gold | % | 99.92 |
| | Silver | % | 99.50 |
| Doré Refining Costs | Gold | US$/payable oz | 1.00 |
| | Silver | US$/payable oz | 0.50 |
| Transportation Costs | | US$/oz doré | 1.21 |
| Guatemalan Government Royalty | | % NSR | 1.00 |
| Third Party Royalty | | % NSR | 1.05 |

Source: Bluestone (2021).

## 19.2 Contracts

At this time, no contractual arrangements transportation or refining exist, nor are there any contractual arrangements for the doré.

## 19.3 Royalties

The Cerro Blanco Gold project is subject to two royalties, both of which have been included in the economic analysis and cash flow model. Table 19.2 outlines the assumed royalty terms. Royalties for the project total US$90 million over the life of mine.

**Table 19.2: Royalty Assumptions**

| Parameter | Unit | Value |
|---|---|---|
| Guatemalan Government Royalty | % NSR | 1.00 |
| Third Party Royalty | % NSR | 1.05* |

Note*: 1.05% royalty has been grossed up to account for country withholding tax.
Source: Bluestone (2021).

## 19.4 Metal Prices & Exchange Rates

The precious metals markets are highly liquid and benefit from terminal markets around the world (London, New York, Tokyo, and Hong Kong).

The gold and silver prices selected are in line with recently released comparable technical reports. The USD:CAD foreign exchange rate selected was based on an average exchange rate. A sensitivity analysis on metal prices was completed as part of the overall economic analysis. The results of this are discussed in Section 23. Table 19.3 outlines the metal prices and exchange rates used in the economic analysis.

It must be noted that metal prices are highly variable and are driven by complex market forces that are difficult to predict.

**Table 19.3: Metal Prices & Exchange Rates**

| Assumptions | Unit | Value |
|---|---|---|
| Gold (Au) Price | US$/oz | 1,600 |
| Silver (Ag) Price | US$/oz | 20.00 |
| FX Rate | GTQ:USD | 7.69 |
| | CAD:USD | 0.76 |

Source: Bluestone (2021).

# 20. ENVIRONMENTAL STUDIES, PERMITTING & SOCIAL OR COMMUNITY IMPACT

Cerro Blanco Gold Project maintains a comprehensive permit registry and environmental/social management system. Two dedicated site-based teams provide ongoing management of these Project aspects: an environmental team and a community relations team. The teams continue to provide extensive knowledge, understanding, and feedback regarding the local environmental and socio-economic contexts to Bluestone management.

In addition, Bluestone has reviewed, and as required, updated the environmental, social and permitting aspects based on current international good practices which meet or exceed local regulations. This has been done with the support of experienced international and Guatemalan consultants.

Bluestone is committed to developing the Project with well-defined sustainability objectives that protect the environment and are aligned with the local communities, for both near-term and future needs. In 2018, Bluestone deployed significant efforts to develop and implement new corporate social responsibility (CSR), human resources, and health, safety and environment (HSE) policies and procedures, and is committed to continued improvement in these areas as the Project advances.

## 20.1 Environment

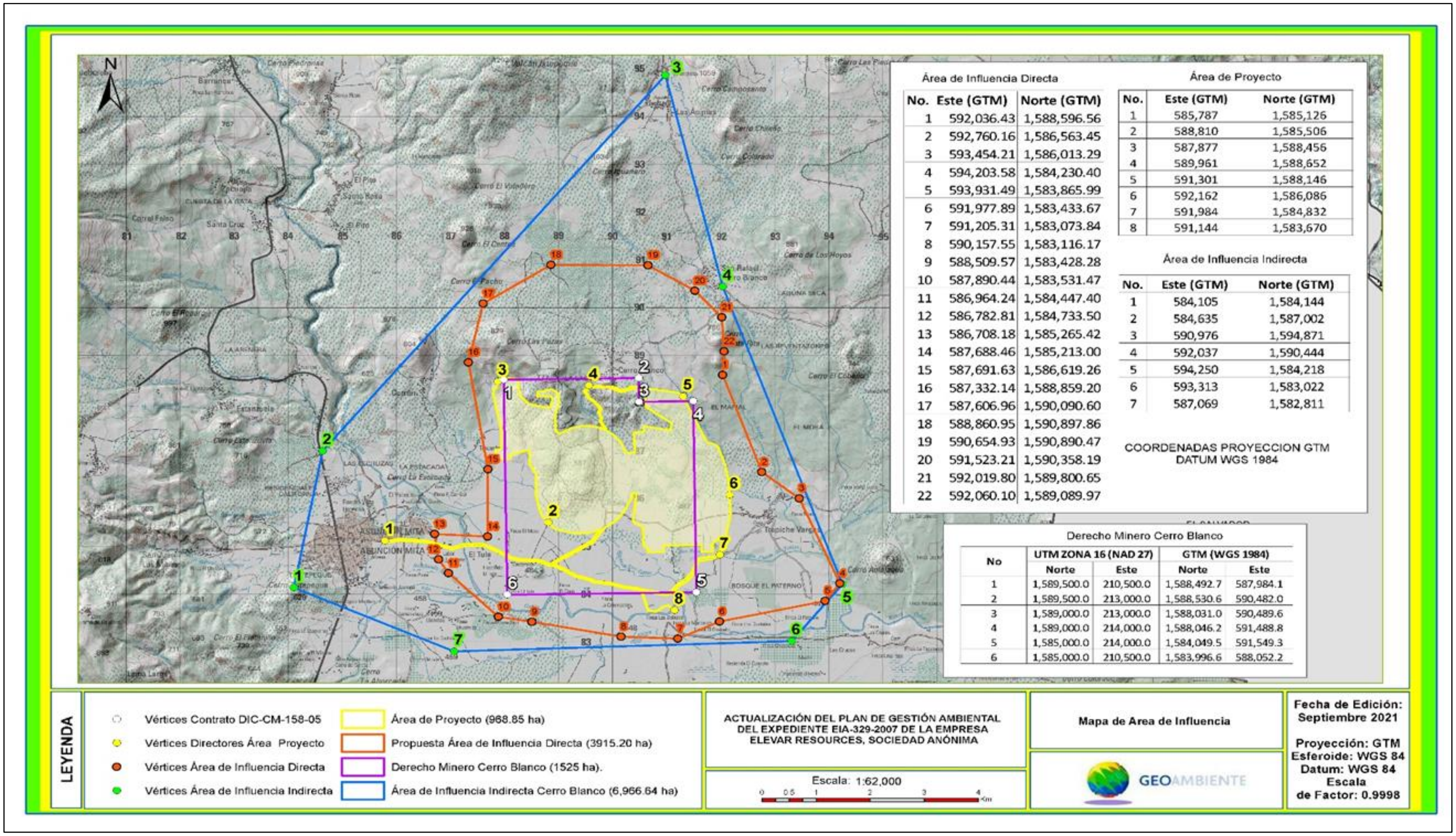
Various environmental studies and ongoing monitoring activities have been performed at the Cerro Blanco site since the granting of the environmental permit in 2007 (ESIA). An update to the permit through a permit amendment application (APGA – Actualizacion del Plan de Gestion Ambiental) has been submitted to Ministerio de Ambiente y Recursos Naturales (MARN) to support the open pit scenario. In addition, it is noted that supplemental baseline environmental and social studies were provided to support the development of the new Project.

The APGA will include environmental and social management plans that are based on current international good practice.

### 20.1.1 ESIA Areas of Influence

The area of influence assessment was updated in 2021 based on three specific areas, as shown in Figure 20-1. The ESIA will define a Project area of 968 ha including the pit, waste rock and dry stack tailings facilities, process plant and its attached service buildings.  The direct area of influence within an irregular polygon is 3,915 ha in area.

**Figure 20-1: ESIA Areas of Influence**




Source: Bluestone (2021).

The indirect area of influence includes the mining title lands (exploration areas plus the direct area of influence) and several communities (total area of 6,967 ha).

### 20.1.2 Water Management

Water management infrastructure at the Project site includes an existing water treatment plant (WTP), pipelines, settling ponds, collection and cooling channels and ponds, groundwater wells, and related minor facilities. Monitoring of surface and groundwater is done regularly and is an ongoing activity. The on-site water management and monitoring requirements are well understood, as the environmental team have been collecting data for the past 10 years. The approved ESIA (2007) indicates that metal constituents, including aluminum, arsenic, iron and manganese are naturally occurring in the surface and groundwater and are commonly found throughout the region.

The existing WTP was installed in 2011 and is capable of treating up to 341 $m^3$/h. The approved discharge location for treated water is the Quebrada Tempisque. The WTP is specifically designed for removal of arsenic using co-precipitation with ferric salt. The treatment process includes chemical oxidation, pH adjustment, ferric iron addition, and solids separation to achieve removal of dissolved arsenic. A complete monitoring and sampling program has been in place since 2011, with regular monthly compliance reports submitted to and approved by MARN. To date, there have been no incidents of non-compliance.

Water management for the Project will focus on separating non-contact from contact waters. Contact water management will include the installation of new diversion ditches, collection ditches, collection ponds, settling ponds, groundwater pumping wells, water treatment plant and discharge outfall.

As part of the Project, installation of a new water treatment plant is planned to treat process, mine infiltration water, groundwater from dewatering wells and contact runoff. It will have a larger capacity than the existing plant and its process will be upgraded to enable the removal of additional contaminants (refer to Section 18 for details).

The new water treatment plant will be located away from the open pit blast radius. It will have two treatment streams: the first stream will treat the contact water consisting of collected surface runoff and process plant excess and will be designed for an original capacity of 600 $m^3$/h, increasing to 1,200 $m^3$/h in Year 4 and thereafter, and the second stream will treat the non-contact groundwater pumped from dewatering wells and will be designed for a capacity of 1,065 $m^3$/h. The contact water treatment plant will include secondary cyanide removal, cooling, pH adjustment, metal precipitation nitrogen removal, solids separation with clarification and filtration. The non-contact groundwater treatment plant will include cooling, precipitation with ferric sulphate, and solid separation via filtration. Treated water will be primarily re-used as service water/process water, with excess discharged in the Ostúa River.

The monitoring program of surface water and groundwater will be adjusted to the new infrastructure to ensure compliance with permitted discharge requirements.

#### 20.1.2.1 Water Quality

The environmental monitoring program includes continuous sampling of surface water and groundwater. The monitoring performed at the discharge point to date shows that the concentrations of all parameters are within the criteria outlined by MARN guidelines for effluents.

With regard to future Project discharge, the plant will be designed to comply with the effluent quality criteria established by Guatemalan regulations and the IFC Environmental, Health, and Safety (EHS) Mining Effluent Guidelines. For more information on discharge criteria, see Table 20.1.

**Table 20.1: Discharge Criteria**

| Parameters | Units | Discharge Standards | |
|---|---|---|---|
| | | MARN Government Agreement 236-2006 | IFC (EHS, 2007) |
| Temperature | °C | TCR ±7 (26.89)1 | < 3 (increment) |
| pH | - | 6 to 9 | 6 to 9 |
| Grease & Oils | mg/L | 10 | 10 |
| Total Suspended Solids | mg/L | 100 | 50 |
| Biological Oxygen Demand (BOD) | mg/L | 100 | 50 |
| Chemistry Oxygen Demand (COD) | mg/L | | 150 |
| Colour | Pt-Co Scale | 500 | N/A |
| Floating Matter | A/P | absent | N/A |
| Fecal Coliforms | most probable number/100 mL | 4000 | N/A |
| Total nitrogen | mg/L | 20 | N/A |
| Total Cyanide | mg/L | 1 | 1 |
| Total Arsenic | mg/L | 0.1 | 0.1 |
| Cadmium Total | mg/L | 0.1 | 0.05 |
| Total Copper | mg/L | 3 | 0.3 |
| Hexavalent Chromium | mg/L | 0.1 | 0.1 |
| Total Phosphorus | mg/L | 10 | 2 |
| Mercury Total | mg/L | 0.01 | 0.002 |
| Total Nickel | mg/L | 2 | 0.5 |
| Total Lead | mg/L | 0.4 | 0.2 |
| Zinc Total | mg/L | 10 | 0.5 |
| Iron | mg/L | N/A | 2.0 |

#### 20.1.2.2 Surface Water Management

Proposed surface water management infrastructure is intended to minimize the volume of precipitation and runoff that comes into contact with areas that could negatively impact water quality (see Section 18.5 for details). Runoff water from these areas is classified as “contact” water, while water that is diverted away from these areas is classified as “non-contact” water. Contact water is either reused in the process plant or sent to the WTP for treatment. Treated water from the WTP is monitored for quality prior to discharge. Non-contact water is also monitored prior to discharge to ensure its quality meets discharge criteria.

The Project continues to improve the management of non-contact and contact water.

#### 20.1.2.3 Groundwater Management

To effectively dewater the mine, the current dewatering plan includes a combination of surface dewatering wells installed along the periphery of the pit, horizontal drains which will report to in-pit sumps.

Up to eight dewatering wells are planned over the LOM for a total pumping capacity of 1,065 $m^3/h$ which will be conveyed to the non-contact groundwater treatment plant. In addition, up to 481 m3/h of groundwater infiltration is estimated through horizontal drains in the pit walls which will report to in pit sumps where it will mix with precipitation and runoff, and then be directed to the contact WTP for treatment. Once treated, a portion of the water will be reused in the process plant and for mine services, while the remainder will be pumped to the approved discharge point at the Ostúa River.

### 20.1.3 Waste Rock

The waste rock facility (see Section 18.14 for details) is designed to store 170 Mt (108 $Mm^3$) of waste rock. Additional testwork, including static and kinetic tests on various representative samples of waste rock, were completed to establish the geochemical properties of the waste rock and tailings and inform the design and management plan of the facility. A total of 312 samples underwent static ABA testing, 31 of which were submitted for kinetic humidity cell testing (HCT), as detailed in Section 18.14.2.

Based on the Mine Environmental Neutral Drainage (MEND) classification, testwork results available as of the date of this report indicate 36% of the waste rock is classified as potentially acidogenic (PAG), 15% is classified as uncertain, and 49% as non-PAG.

The interpretation of the HCT results for the waste rock were mostly consistent with the ABA static test results. Most of the PAG samples selected for HCT based on the ABA testing have produced acidic leachate. These samples have also produced metal constituent concentrations exceeding IFC mining effluent quality guidelines in some samples including arsenic, copper, iron, mercury, nickel, and zinc. Exceedances have only been measured in acidic samples; there is no current evidence of neutral metal leaching.

Given the results from the geochemical characterization, Cerro Blanco Gold Project has identified the need to implement controls to mitigate and reduce the risk of possible contamination by waste rock seepage.

The first component of the waste rock management plan is to centralize the waste into one designated area. This allows the implementation of necessary controls and monitoring programs. The second component is to limit the amount of water coming in contact with the waste, as the water is the transport mechanism for the constituents of concern. This is achieved by centralizing the waste and diverting non-contact runoff away from the storage area. Direct precipitation becomes the only source of water available to transport any potential contaminants, which limits volume of seepage and associated water to manage.

There are different types of control applied in the mining industry to address acidity and metal leaching (ARDML) conditions, and the selection of the control depends on many factors including site conditions, regulatory context, tonnage of waste to manage, footprint available, geochemical characterization, availability of material, operability, certainty, and cost.

Based on these factors and a preliminary assessment of the factors above, limestone addition has been identified as an appropriate method to control ARDML at Cerro Blanco. Further evaluation of this method and consideration of alternative approaches will be conducted in the detailed engineering phase of the Project.

Limestone naturally offers neutralization potential due to its high calcite content and reactivity. It is commonly used to neutralize acidic conditions and precipitate metal in the mining industry and in water treatment processes. As acidic water comes into contact with the limestone, it dissolves the calcite, causing the pH and alkalinity to rise as per the following equation:

$$CaCO3 + H^{+} \longrightarrow Ca^{2+} + HCO3^{-}$$

The objective is to increase the neutralization potential of the mass of waste rock well above NPR> 2, shifting the classification of the waste rock from PAG to non-PAG. Through continued application as required during operations, excess neutralization potential will buffer the acidity created by the oxidation of sulphide and promote conditions that inhibit metal dissolution.

Limestone will be sourced locally (existing or future) and applied in layers at the base of the waste rock facility with additional layers placed as the facility is constructed in accordance with the mixing ratio, that is determined based on the calcite content of the limestone source. The limestone will be placed by using the dozer fleet already assigned to the waste dump.

Diversion channels will be constructed to channel the non-contact water away from the waste rock facility footprint. The surface runoff will be captured by a perimeter channel and directed to the water treatment plant. The treatment of contact water remains part of waste rock management despite limestone addition to provide contingency for the reaction lag time and limestone dosing optimization period.

### 20.1.4 Tailings

The DSTF is described in Section 18.13 and summarized in the subsections below.

#### 20.1.4.1 DSTF Design Features

Tailings generated from the process plant will be dewatered through a filtration process before being transported to the DSTF. The ultimate DSTF footprint covers a total area of 132 ha south of the process plant and is designed to accommodate about 54.4 Mt of filtered tailings and some 168.5 Mt of waste rock, with a 220 Mt of total design capacity. The ultimate height of the current design is 175 m and maximum slope of material stacked on the embankment is 3:1.

A designated contact water management system is designed to collect runoff from the DSTF to prevent potential surface water contamination. Contact water, including seepage collected from the foundation drains, will be pumped to the WTP for treatment prior to discharge.

#### 20.1.4.2 Tailings Characterization

As discussed in Section 18.13.4.3, fifteen tailings samples were submitted for static ABA testing and six samples were submitted for kinetic humidity cell testing as part of the 2020-2021 characterization program. Results indicate that while some samples classify as PAG based on ABA testing, none have generated acidic pH or metals during the kinetic testwork, with the exception of mercury. As stated previously, the tailing samples available were not fully processed by CIP. The use of CIP is likely to remove a significant portion, if not all, mobile mercury from the tailings material. An additional tailings sample reflecting the complete ore process flowsheet, including CIP, was subjected to geochemical characterization through static and kinetic testing and mercury was not detected in the shake flask extraction (SFE). Circumneutral leachate from PAG material suggests that NP is readily available and can neutralize acid generated by sulphide oxidation.

Consistent with the approved ESIA, tailings are considered to be non-acidogenic and non-metal leaching. Kinetic tests on various representative tailings samples are ongoing and will be completed prior to detailed engineering to confirm this assumption.

### 20.1.5 Flora & Fauna

An Environmental Baseline Study originally conducted for the 2007 ESIA was updated during 2021 to include the new mining method approach. The on-site environmental team is responsible for conducting periodic monitoring of the direct and indirect areas of influence. A flora and fauna monitoring program has been implemented and is a useful tool in evaluating the potential changes to the ecosystem and species as a result of Project activities. Information gathered through the monitoring program allows for adequate mitigation measures to be put in place, but thus far through preventive plans there has been no apparent impact from the Project on biological populations in the region. One of these preventive plans is a biological conservation management plan that was implemented to protect the natural environment and species, as discussed in Section 20.1.5.5.

The most recent flora and fauna (terrestrial and aquatic) monitoring was conducted in April and July 2021 by Geoambiente to capture wet and dry seasonal variation. The monitoring results do not indicate any significant impact to the species monitored or their conditions as a result of mining activities. Flora and fauna monitoring activities will continue through the construction and operations phases of the Project to mitigate impact on native species.

A summary of the recent flora and fauna monitoring results is provided below. These were based on sampling points located within the direct, indirect and external areas of influence.

#### 20.1.5.1 Flora

The Project is located in sub-tropical dry forest and tropical dry forest areas where the vegetation is dominated by graminoid savanna, deciduous shrubs, and deciduous forests with many species of microfoliate composite leaves. During the last flora study (July 2021), 142 species distributed in 62 floristic families were observed.

The botanic species were identified for each of the three different land cover types that exist in the area of influence.

1. Pasture: 91 species were collected, which corresponded to 38 botanical families. Fabaceae was the dominant family with 18 species, of which 7% were shrubs, 30% lianas, forbs/herbaceous species, and 63% tree species characteristic of seasonally dry forests.
2. Agroecosystems: This environmental system was the most diverse of the three, with 107 species collected, corresponding to 42 botanical families. Fabaceae was the richest, of which 10% were shrubs, 25% herbaceous, lianas, epiphytes or climbing plants, and 65% arboreal species.
3. Forest: This ecosystem had the lowest richness, with 86 species corresponding to 37 botanical families collected. Fabaceae was the richest, of which 9% were shrubs, 23% grasses, lianas, epiphytes or climbing plants, and 68% tree species.

#### 20.1.5.2 Terrestrial Fauna

The terrestrial fauna monitoring performed at the sampling points in April and July 2021 registered 89 species. The most frequently observed species were birds (65%) and mammals (16%), followed by minor numbers of herpetofauna species (19%). These results are consistent with the previous years of monitoring.

#### 20.1.5.3 Aquatic Fauna

Geoambiente performed aquatic fauna monitoring in 2021 during the dry and wet seasons with data collected and compiled from nine monitoring stations. There are no significant changes relating to the physicochemical parameters between the two seasons.

The monitoring of aquatic invertebrates collected 3,967 organisms representing 37 families. Among the Insect class, the dominant orders were Coleoptera (18.2%), Ephemeroptera (8.75%), Trichoptera (7.3%), Diptera (6.25%) and Odonata (3.6%); the rest of the orders found were less than 1% of the structure of aquatic entomofauna, except for Anelids (2.5%) and Mollusks (51.5%).

The most recent results from 2021 indicate an electrical conductivity between 250 and 2,600 µS/cm (dry season) and 140 to 1560 µS/cm (wet season); a pH range of 7.7 to 9.2 (dry season) and 7.6 to 8.1 (wet season) and temperature ranges between 25.1°C to 30.8°C (dry season) and 23.9°C to 29.6°C (wet season).

#### 20.1.5.4 Fish

During the dry season, 139 specimens were collected distributed among 12 species, the dominant species being *Poecilia butleri* (92 individuals). During the wet season, 146 specimens distributed among 14 species were observed, dominated by *Astyanax aeneus* (138 specimens).

#### 20.1.5.5 Flora & Fauna Management Plan

The on-site environmental team, in conjunction with Bluestone's corporate initiatives, has implemented a specific Flora and Fauna Management Plan to protect both flora and fauna environments and species (Entre Mares de Guatemala, S.A., Julio 2018, Manejo de Flora y Manejo de Fauna).

The main objective is to protect the native species and potentially-threatened native species, and to optimize on-site natural growth and habitat conditions. Before any on-site disturbance, the environmental team first inspects the area to evaluate the flora and fauna species potentially affected. If potentially endangered species are identified (such as orchids, tillandsias, cactus or pitayas), a specific relocation protocol is implemented. This includes identification of the impacted species' geolocation and local conditions in order to optimize their future habitat by ensuring the new location will offer similar conditions and minimize stress due to relocation.

The species relocations are carefully carried out in a private reserve (El Pedernal) where periodic monitoring is performed. The species collected on site can thus continue to grow/live in a protected environment.

### 20.1.6 Cultural & Archaeological Resources

As outlined in the approved ESIA, along with Bluestone's corporate initiatives, the on-site environmental team members are identifying, monitoring and mitigating potential impact to any cultural and archaeological artifacts. Similar to the fauna and flora management plan, a defined inspection procedure is performed prior to any activity, along with consultation with external archaeological experts, if required. To date, no cultural and archaeological resources have been identified within the direct area of influence.

### 20.1.7 Environmental Monitoring

The on-site environmental team is currently operating a comprehensive environmental monitoring network consisting of 26 monitoring stations for water quality located within and outside of the Project boundaries. As part of the commitments under the approved ESIA, Bluestone performs monthly monitoring of water quality, air quality and noise levels in the direct and indirect areas of influence. There are six stations for monitoring ambient air quality and noise in the villages around the Project done on a quarterly basis. All the monitoring reports are prepared and presented to the Authorities (MARN, Ministerio de Energia y Mina (MEM), Ministerio de Salud Jutiapa and Ministerio de Ambiente Jutiapa) for review and approval.

The environmental baseline study prepared for the 2007 ESIA and updated in 2021 included air quality or noise parameters for six points located at the nearest receptors (villages), including $PM_{10}$ and greenhouse gases, with results compared to the ambient standards of World Bank EHS Guidelines; no exceedances have been detected. Since October 2019, quarterly monitoring for air quality and noise conducted in the six villages around the Project have shown a normal air quality for rural areas. Noise levels exceed the night standards (World Bank guidelines) due to the natural conditions in the area (wind, birds, insects, road transit, and other human activities), but none that are specifically associated with the Project. In addition, internal monitoring is currently conducted on a monthly basis at six different sampling stations strategically located in the direct and indirect areas of influence.

In September 2021, the company installed automatic monitoring device systems in three locations around the Project for air quality, noise and vibration monitoring, and a real time flowmeter in the current discharge structure to Tempisque Creek to ensure compliance with the authorized flow. The data from this network confirms the results of the quarterly monitoring.

### 20.1.8 Environmental Management Plan

The ESIA for the Project includes an Environmental Management Plan (EMP), which has been reviewed and updated as part of the open pit scenario. Based on the environmental monitoring data collected at site over the past decade, the current environmental procedures have been carefully reviewed and updated to ensure improvement and increased effectiveness to satisfy all local regulations and incorporate international good practice. The new EMP also includes sections relating to the corporate Health, Safety and Environment Program (HSE) and the Emergency Response Plan (ERP).

Bluestone is advancing in the documentation of the environmental management system, creating new procedures for “non-conformities” and action plans, as well as several specific standards to prevent, control and mitigate any potential impacts. This system goes beyond the local regulations contemplated in the ESIA to ensure compliance with the good international practice in the gold mining sector.

## 20.2 Permitting

The Exploitation permit and the approved ESIA received in 2007 and associated permits allow the Project to proceed with underground mine development and construction of the process facilities provided future operations adhere to the existing permit requirements. Since the design has been updated and optimized to be exploited as surface mining operation, specific permits will be required for approval in order to be aligned with the updated Project design.

Two new infrastructure items, the access road/bridge and powerline are not covered by any previous studies or permits. These require new baseline studies, ESIAs, and permit applications to be submitted to MARN for approval, with input from the following Guatemalan authorities: MEM, Consejo Nacional de Areas Protegidas (CONAP), Instituto Nacional de Bosques (INAB), Ministerio de Salud y Asistencia Social (Ministry of Health & Social Assistance), and the local municipality of Asunción Mita.

Table 20.2 provides a summary of the main environmental permit amendments required and the new environmental permits necessary for the Project. The Cerro Blanco Gold Project permit register indicates all applicable permit commitments have been fulfilled to date.

**Table 20.2: Main Permit Amendments & New Permit Required**

| Project Component | Action Required |
|---|---|
| Water Management | ESIA Amendment |
| Increase of Exploitation Rate (Change of mining Method): Open Pit, Processing Plant and Waste Rock Management | ESIA Amendment |
| DSTF New Location | ESIA Amendment |
| New Access Road / Bridge | New ESIA and Permit |
| New Powerline | New ESIA and Permit |

Source: Bluestone (2021).

## 20.3 Mine Closure

The approved ESIA (2007) includes a conceptual mine closure and reclamation plan. The recently submitted permit amendment also includes a conceptual closure plan. Under Guatemalan regulations, a mine closure plan does not need to be submitted to the authorities until three years before the ed mine closure. Nevertheless, the requirements of the updated closure plan are summarized in Table 20.3, and a benchmark closure cost has been assumed and included in Section 21.

Opportunities for concurrent or progressive reclamation during construction of the combined WRF/DSTF will be considered when the ultimate footprint is completed.

The surface of the DSTF will be contoured so that it will shed precipitation rather than impound it. Topsoil that is stockpiled from the DSTF footprint during construction will be stored and spread over the surface of the DSTF at closure. Native grass seed mixture will be planted to reduce erosion.

It is anticipated that the open pit will naturally flood via runoff and groundwater infiltration once all pumping and diversion have ceased, and that a pit lake will form. Modelling work will be completed to determine if discharge is anticipated and to predict its quality for compliance with discharge limits.

**Table 20.3: Mine Closure Requirements**

| Component | Phase | Description |
|---|---|---|
| Safety | Progressive & Definitive | HSE program to be specifically developed and put in place prior to closure/decommissioning activities. |
| | | Adequate HSE training will be provided prior to performing any closure/decommissioning activities. |
| | | Any contractor engaged in closure activities must have HSE policy reviewed and approved by the Owner. |
| Mine | Progressive | Waste dumps will be gradually reclaimed as the final dump slopes are established. |
| | Definitive | Removal of all equipment and material. |
| | | Slopes above the expected flood line will be contoured. |
| | | The pit will be allowed to flood. |
| Infrastructure | Definitive | Infrastructure will be removed, with exception of roads (including bridges). |
| | | Concrete foundations will remain in place and be covered with locally sourced fill and indigenous vegetation. Surface will be graded to prevent water accumulation. |
| Process Plant | Definitive | HAZMAT abatement will take place, if required, prior to closure. |
| | | Adequate cleaning of infrastructure and drainage of piping before demolition. |
| | | Recycling, reuse, and reclamation of materials will be evaluated prior to closure phase to avoid disposal in landfills. |
| | | Safe demolition of the infrastructure and proper waste disposal. |
| | | Concrete foundations will remain in place and be covered with locally sourced fill and indigenous vegetation. Surface will be graded to prevent water accumulation. |
| Water Treatment Plant (WTP) | Progressive | Adequate cleaning of infrastructure and piping before demolition. |
| | | Recycling, reuse, and reclamation of the infrastructure will be evaluated prior to closure phase to avoid disposal. |
| | Definitive | The WTP will be kept in operation for five years after mine closure. |
| | | Safe demolition of the infrastructure and proper waste disposal. |
| | | Concrete foundations will remain in place and be covered with locally sourced fill and indigenous vegetation. Surface will be graded to prevent water accumulation. |
| Piping, Ponds & Tanks | Progressive & Definitive | Ponds, tanks and pipes will be emptied and cleaned. |
| | | Contaminated material will be cleaned out and disposed of as per Guatemalan regulations. |
| | | Recycling and reuse will be prioritized over disposal of equipment. |
| | | Removal of all overland pipelines and tanks. Any concrete foundations will be left in place, covered with locally sourced fill and indigenous vegetation. |
| | | Ponds will be backfilled and revegetated. |
| Switchyard & Power Distribution | Progressive & Definitive | All services will be de-energized prior to decommissioning. |
| | | Equipment will be cleaned and adequately disposed of. |
| | | All surface powerlines and poles on site will be dismantled and disposed of. |
| | | Concrete foundations will remain in place and be covered with locally sourced fill and indigenous vegetation. Surface will be graded to prevent water accumulation. |
| Administration Offices & Ancillary Buildings | Definitive | None of the buildings will be kept in place after the closure phase. |
| | | Administration offices and all other buildings (including explosives storage warehouse) will be decommissioned and demolished. |
| | | Cleaning and decontamination procedures will include equipment/waste management disposal/recycling prior to decommissioning. |
| | | Concrete foundations will remain in place and be covered with locally sourced fill and indigenous vegetation; surface will be graded to prevent water accumulation. |
| Dry Stack Tailings Facility (DSTF) | Definitive | Once mining operations have ceased, the DSTF will be closed, covered with growth media and revegetated with appropriate native species. |
| | | All surface piping, mechanical equipment and electrical services associated with the DSTF will be decommissioned and disposed of. |
| | | Soils around the facility will be tested for potential contamination. |
| | | Final reclaimed profile will respect the site specific landform objectives. |
| Waste Rock Facility | Progressive & Definitive | Final slopes will be contoured. |
| | | Natural soil cover will be put in place and vegetation with indigenous species planted over the impacted area. |
| Wells | Progressive & Definitive | All dewatering and injection wells will be grouted and sealed; wellhead piping will be removed, and concrete pads will remain in place and be covered with locally sourced fill and Indigenous vegetation. |

Source: GMS (2021).

## 20.4 Social Environment & Relationships with Communities

Similar to the environmental management program, Bluestone Resources has maintained a comprehensive community relationship register and reporting system. The social management processes were recently reviewed, and a comprehensive social management system was developed using information available from the Project database, applicable reports, and on-site Community Relations team.

Bluestone is committed to developing Cerro Blanco Gold Project with best international practices in line with its corporate sustainable and transparency objectives.

In 2021, Bluestone updated to the Economic and Social Baseline Study (INSUCO) to provide a current socio-economic context for the Project. The Social Baseline Study has been updated and key information has been included in the ESIA recently submitted (November 2021) to the authorities for approval. The Social Baseline Study (SBS) update included the most recent primary data obtained from each household, and was supported through face-to-face meetings with representatives of local organizations and communities. In addition, relevant project social aspects and stakeholder engagements were reviewed and updated based on international best practices and the relevant IFC Performance Standards. In 2021, Bluestone developed and initiated the implementation of a new social management system that includes a database to register all community engagements, activities, and key information related to the community relationships. The social management system is used to track and monitor performance against key CSR objectives.

### 20.4.1 Social Baseline Study Update

The primary objective of the Social Baseline Study update was to obtain current information regarding the socio-economic status in the Project's area of influence. The scope of the study covers the seven rural villages in the direct and indirect area of influence of the Project, and a portion of the town of Asunción Mita (capital of the municipality, also referred to as the "Urban Area"),

Field visits were conducted between March and May 2021, which included a census of 699 households in the rural area. In addition, workshops, semi-structured interviews and general observations were conducted and completed in both the urban and rural areas. Official Guatemalan documentation was also reviewed to understand the context of the study area and establish historical trends.

#### 20.4.1.1 Asunción Mita (Urban Area)

Asunción Mita is characterized as a municipality in constant growth. In 2016, it had already exceeded the population projections made in 2011 for 2025. Its inhabitants are mostly children and adolescents. The population is almost evenly split between men and women, with a slight predominance of women. Typical families in this urban area are consistent with the trends in other urban areas within the department of Jutiapa: traditional nuclear families of three to four members, with a male head of household. The town has more than 4,500 homes. Buildings are characterized by block walls with sheet metal roof and floors of cement or granite.

The primary industry in the urban area of Asunción Mita is commercial trade. This has evolved in part due to the large population of migrants that have relocated to the town. Agricultural activities, such as melon production for export, and services such as masonry and plumbing, are common and act as important economic drivers in the region. The population of the Urban Area is ladino, characterized as a heterogeneous population which expresses itself in Spanish as a maternal language, possesses specific cultural traits of Hispanic origin mixed with indigenous cultural elements, and dresses in a style commonly considered as Western.

In Asunción Mita, 28 registered small local community organizations called Consejos Comunitarios de Desarrollo (COCODEs) manage public budgets and formulate projects to meet community needs. During the SBS update, meetings were held with representatives from the 20 active COCODEs in the Urban Area.

The Urban Area also has a municipal council called Consejo Municipal de Desarrollo (COMUDE), which includes the mayor, councillors, and other municipal officials. The COMUDE mandate is to ensure projects identified by the COCODEs are carried out. An independent civil organization dedicated to environmental monitoring, called Asociacion de Monitoreo Ambiental Regional (AMAR) works closely with the Bluestone Project team and local representatives. Additional social organizations identified include the School Parents' Board and Catholic, Mormon and Evangelical churches.

Bluestone Project team members are actively involved with local organizations and communities to inform the population regarding project activities and development strategies. In order to respond to the concern around lack of specific knowledge regarding the Project, COCODE representatives and the public have been invited to visit the Cerro Blanco Project site. These site visits have proved to be very successful and will continue until construction activities commence.

In 2018, approximately two-thirds of the population over the age of 14 was either employed or looking for work. Earnings for both men and women are typically far below minimum wage. There is a clear gender gap in the labour market where women are less represented. Absence of contracts and proof of payment or social security are indicators of informal employment. There is no evidence of child labour.

#### 20.4.1.2 Surrounding Villages (Rural Area)

The Bluestone Project team maintains active contact, meetings, and discussions with all the villages surrounding the mine.

The majority of villages in the area studied have small populations (i.e., less than 400). See Table 20.4 below. The population characteristics of the villages are generally in line with those in the Urban Area. There is no significant immigration in the Rural Area, while there is emigration to the United States and the interior of Guatemala. Although the information collected represents a low percentage of migrant population within the Rural Area, there is daily commuting to the town of Asunción Mita for work which is not considered as migration by the village population.

**Table 20.4: Community Survey Results**

| Community | Total Dwellings | Surveyed Dwellings | Total Population | Surveyed Population |
|---|---|---|---|---|
| Cerro Blanco | 98 | 82 | 374 | 211 |
| Trapiche Vargas | 153 | 114 | 584 | 360 |
| El Tule | 3 | 3 | 13 | 9 |
| La Lima | 33 | 28 | 125 | 104 |
| El Cerrón | 104 | 80 | 395 | 264 |
| San Rafael CB | 79 | 75 | 301 | 206 |
| Las Ánimas | 73 | 54 | 280 | 130 |
| Urban Area | 4,921 | 263 | 18,504 | 591 |
| **Total** | **5,464** | **699** | **20,576** | **1,875** |

Source: Bluestone (2021).

The majority of the employed rural population earns below the national minimum wage. There is a significant availability of workforce in the villages, as 7 out of 10 residents are of working age. However, the employment rate of approximately 50% is below the national rural average. Formal employment is almost non-existent at only 2%, and 87% of the rural population have not completed secondary school.

### 20.4.2 Social Management Plan

The Project has refined the existing Social Management Plan, and there has been a dedicated Community Relations team at site since the start of the Project to manage social aspects and obligations.

The Social Management Plan was reviewed and updated to comply with current IFC Performance Standards on Environmental and Social Sustainability. This updated Social Management Plan satisfies all ongoing IFC requirements, including the following good practice guidance manuals:

- Grievance Mechanism
- Communication Plan
- Conjuncture Monitoring
- Local Relationship
- Social Commitments Monitoring
- Contractors and Subcontractor Monitoring
- Social Impact Evaluation
- Continuous Training
- Social Investment
- Corporate Social Alignment

- Community Requests and Donations
- Participative Socio-Environmental Monitoring
- Recruitment: Talent and Human Resources Selection
- Local Procurement
- Land Access.

### 20.4.3 Stakeholder Engagement

A stakeholder engagement plan has been designed and implemented by Bluestone to comply with the IFC Performance Standards. Since Project inception, the community relations team has been engaged with local stakeholders on an ongoing basis. Specific meetings designed to involve all local stakeholders were completed in 2018 during a Social Baseline Study update and all participants were invited to visit the Project site. A calendar of presentations and guided site visits have been developed, allowing all stakeholders the opportunity to observe the current activities occurring on-site and meet with the on-site team. These visits are intended to be participative and interactive. All concerns or questions raised are registered by the site team, and improvement and/or corrective actions implemented, when necessary.

### 20.4.4 Social Management System

The Project social management system has been reviewed and improved during the FS. Dedicated software (Staketracker) has been implemented as a tool to register, monitor and report all aspects of the social management program. Specific management processes have been developed and implemented, such as a grievance mechanism, in order to formalize the responses to social concerns. This has improved the access to information for the on-site team and resulted in more efficient monitoring and reporting of activities.

### 20.4.5 Visitor Center

In line with corporate policies, Bluestone intends to provide interpretive and educational information to the local communities. A visitor center is currently open to the public where detailed information about the Project is presented through various interactive media tools. The visitor center is located in Asunción Mita where the community relations office is based. A representative of the community relations team is permanently available to answer questions, assist visitors in understanding the Project details, and to address any topics of concerns.

# 21. CAPITAL & OPERATING COSTS

Life-of-mine project capital costs are estimated at $749.9 million over three distinct phases, as follows:

- Pre-production Capital Costs – This phase includes all costs to develop the property to an average mill production rate of 11,747 kt/d. Initial capital costs total $571.5 million (including $60.7 million for contingency and $47.5 million in pre-production revenue), which will be expended over a 25-month pre-production design, construction, and commissioning period.
- Sustaining Capital Costs – This phase includes all costs related to the acquisition, replacement, or major overhaul of assets during the mine life required to sustain operations. Sustaining capital costs are estimated to be $140.3 million and do not include contingency. Sustaining costs are expended in operating Years 1 through 14.
- Closure Costs – This phase includes all costs related to the closure, reclamation, and ongoing monitoring of the mine after operations. Closure costs total $38.1 million and do not include contingency. Closure costs are all incurred in Year 14 through 18.

Note that certain costs incurred prior to the full start of development of the project are considered to be sunk costs and are not included in the estimate of the above three phases.

## 21.1 Capital Cost Summary

Table 21.1 presents the capital estimate summary for the pre-production phase and sustaining capital costs in U.S. dollars with no escalation or value-added tax (VAT) included.

**Table 21.1: Capital Cost Summary**

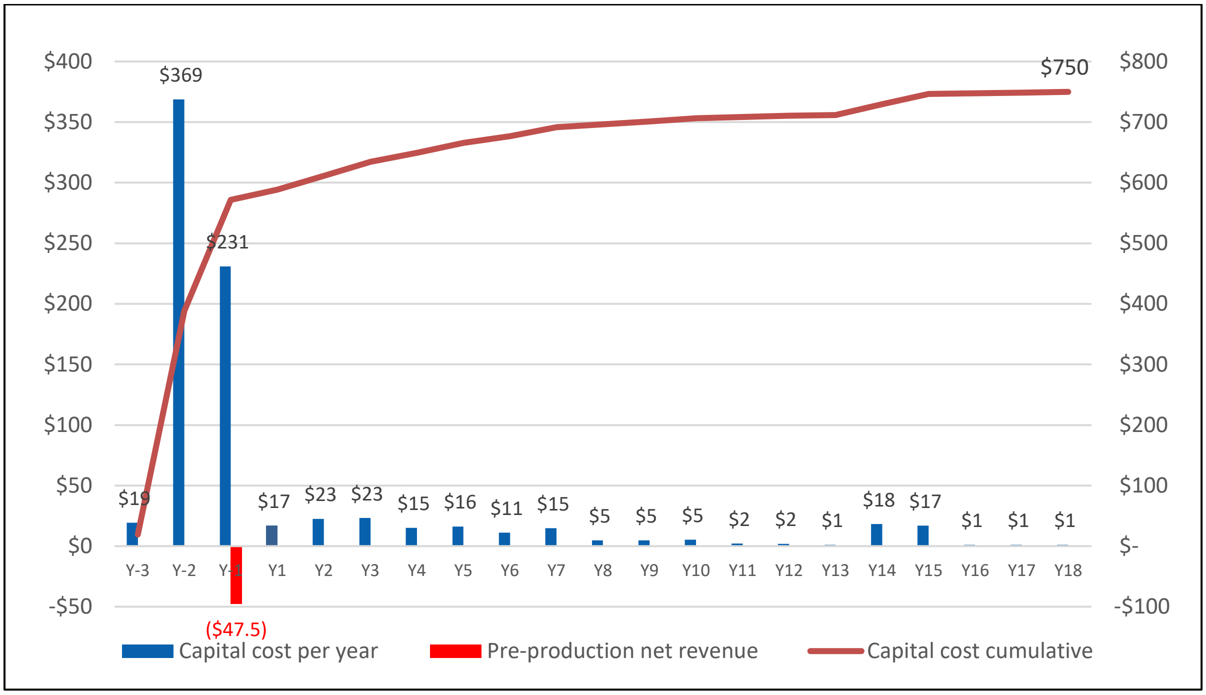
| WBS Area | WBS Description | Pre-Production Cost (US$M) | Sustaining/Closure Cost (US$M) | Project Total Cost (US$M) |
|---|---|---|---|---|
| **100** | **Infrastructure** | **39.6** | **7.0** | **46.6** |
| 110 | Roads, Bridges & Fencing | 6.9 | 1.9 | 8.8 |
| 120 | Mine Infrastructure | 7.3 | 1.3 | 8.7 |
| 130 | Support Infrastructure | 8.3 | - | 8.3 |
| 140 | Camp Facilities | 7.2 | - | 7.2 |
| 160 | Process Plant Infrastructure | 6.8 | - | 6.8 |
| 170 | Fuel Systems Storage | 0.9 | - | 0.9 |
| 180 | Stockpile Pads | 2.1 | 3.8 | 5.9 |
| **200** | **Power & Electrical** | **38.8** | **0.3** | **39.1** |
| 210 | Main Power Generation | 10.8 | - | 10.8 |
| 220 | Secondary Power Generation | 1.5 | - | 1.5 |
| 230 | Water Management Electrical Room | 1.5 | 0.2 | 1.7 |
| 240 | Service Electrical Room | 0.1 | - | 0.1 |
| 260 | Process Plant Electrical Rooms | 18.8 | - | 18.8 |
| 270 | Overhead Distribution Line | 1.9 | 0.1 | 2.1 |
| 280 | Automation Network | 1.9 | - | 1.9 |
| 290 | IT Network & Fire Detection | 2.0 | - | 2.0 |
| 202 | Electrical Duct Bank | 0.2 | - | 0.2 |
| **300** | **Water Management** | **52.0** | **39.9** | **91.9** |
| 310 | Fresh water Intake / Wells | 0.3 | - | 0.3 |
| 320 | Water Ponds & Water Management | 13.1 | 5.3 | 18.4 |
| 330 | Domestic Water (Cost Code Account) | 0.8 | - | 0.8 |
| 340 | Sewage (Cost Code Account) | 1.9 | - | 1.9 |
| 350 | Fire Protection (Cost Code Account) | 3.6 | - | 3.6 |
| 360 | Effluent Water Treatment | 24.2 | 16.8 | 41.0 |
| 370 | Dry Stack Tailings Facility | 1.1 | 3.0 | 4.0 |
| 380 | Dewatering & Injection Wells | 7.0 | 14.9 | 21.9 |
| **400** | **Surface Operations** | **14.4** | **-** | **14.4** |
| 410 | Surface Operation Equipment | 11.5 | - | 11.5 |
| 430 | Concrete Batch Plant | 0.1 | - | 0.1 |
| 480 | Aggregate Plant | 2.8 | - | 2.8 |
| **500** | **Mining** | **42.3** | **40.3** | **82.6** |
| 540 | Mine Infrastructure | 0.5 | 3.9 | 4.4 |
| 550 | Mine Equipment | 41.8 | 36.4 | 78.2 |
| **600** | **Process Plant** | **136.9** | **-** | **136.9** |
| 602 | Pipe Rack | 2.2 | - | 2.2 |
| 603 | Underground Services | 0.2 | - | 0.2 |
| 604 | Run-of-Mine Pad & Mechanically Stabilized Earthwork (MSE) Wall | 1.3 | - | 1.3 |
| 605 | Final Grading | 0.3 | - | 0.3 |
| 610 | Comminution | 53.2 | - | 53.2 |
| 640 | Lixiviation | 29.5 | - | 29.5 |
| 650 | Reagents | 3.6 | - | 3.6 |
| 660 | Refinery | 9.7 | - | 9.7 |
| 680 | Tailings Management | 29.6 | - | 29.6 |
| 690 | Process Plant Services | 7.4 | - | 7.4 |
| **700** | **Construction Indirect** | **66.3** | **-** | **66.3** |
| 710 | Engineering, Construction Management, Project Management | 20.1 | - | 20.1 |
| 720 | Construction Offices, Facilities & Services | 1.7 | - | 1.7 |
| 730 | Shops | 0.7 | - | 0.7 |
| 740 | Construction Equipment & Tools | 15.1 | - | 15.1 |
| 760 | Energy | 10.6 | - | 10.6 |
| 790 | External Engineering | 18.0 | - | 18.0 |
| **800** | **General Services – Owner's Costs** | **77.8** | **-** | **77.8** |
| 810 | G&A Departments | 21.3 | - | 21.3 |
| 820 | Logistics / Taxes / Insurance | 28.3 | - | 28.3 |
| 830 | Operating Expenses | 15.1 | - | 15.1 |
| 840 | Environmental | 6.4 | - | 6.4 |
| 850 | Health & Safety | 3.2 | - | 3.2 |
| 860 | Site Insurance | 3.4 | - | 3.4 |
| **900** | **Pre-Production, Start-up, Commissioning** | **90.2** | **52.9** | **143.0** |
| 910 | Mining Pre-Production | 52.6 | 52.9 | 105.5 |
| 920 | DSTF Preprod | 1.8 | - | 1.8 |
| 950 | Process Plant Pre-Production | 29.2 | - | 29.2 |
| 960 | First Fill, Spares & Consumables (8% of Process Plant Mechanical Equipment) | 6.6 | - | 6.6 |
| **955** | **Pro-production Revenue** | **-47.5** | **-** | **-47.5** |
| **991** | **Project Contingency** | **60.7** | **-** | **60.7** |
| | **Closure Costs** | **-** | **38.1** | **38.1** |
| | **Project Total Cost** | **571.5** | **178.4** | **749.9** |

Source: GMS (2021).

## 21.2 Capital Cost Profile

All capital costs for the project have been distributed against the development schedule to support the economic cash flow model. Figure 21-1 presents an annual life-of-mine capital cost profile that includes Years 14 through Year 18 (closure).

**Figure 21-1: Capital Cost Profile (Million per Year)**




Source: Bluestone 2022

## 21.3 Scope of the Estimate

The project capital cost estimate includes all costs to develop and sustain the project at a commercially operable status. The estimate does not include costs related to operating consumables inventory purchased before commercial production; these costs are considered within the working capital estimate. Sunk costs and Owner's reserve accounts are not considered in the FS estimates or economic cash flows.

## 21.4 Key Estimate Assumptions

The following key assumptions were made during the development of the capital cost estimate:

- The capital estimate is based on a self-perform construction strategy.
- Open pit mining activities will be self-performed. The Owner's team will immediately start with mine development activities at Year -1.

- Surface construction (including earthworks) activities will be self-performed, except for specific scopes of work that require contractors to be hired.

## 21.5 Key Estimate Parameters

The base date of the capital cost estimate is Q4 2021. The initial capital expenditure duration is planned over a period of 25 months, from March 2023 to the end of March 2025 (including a five-month ramp-up period). The capital cost estimate is aligned with a self-perform owner-managed project delivery model.

The base date of the capital estimate is Q4 2021, with no escalation applied for future costs. Proposals and quotations supporting the FS estimate were received in Q3 and Q4 2021.

Metric units are used throughout the capital estimate, except for pipe sizes which are provided in inches (nominal pipe size).

All capital costs are expressed in U.S. dollars (USD or US$). Table 21.2 presents the exchange rates used for costs estimated in foreign currencies and the portions of the capital costs estimated in those currencies.

**Table 21.2: Currency Exchange Rates**

| USD | Exchange Rates | Currency |
|---|---|---|
| 1 USD = | 1.32 | CAD |
| | 7.69 | GTQ |

Source: GMS (2021).

A deterministic estimating methodology was used for the capital cost estimate, which is considered to be in a Class 3 category as established by the Association for the Advancement of Cost Engineering International (AACEi). This determines the level of confidence with respect to the project definition and defines the accuracy of the estimate at -10% to +15% before contingency, with a probability of underrun selected at P85.

The capital cost estimate includes all costs associated with the development of the project up to commercial production.

## 21.6 Mine Capital Cost

### 21.6.1 Introduction

The mine capital cost estimate provides an evaluation of the capital expenditure required to develop the mining infrastructure (roads) and acquire the mining mobile equipment. Capital cost estimates are based on a combination of budgetary quotes from equipment suppliers, bottom-up cost build-ups, and the previous experience of the mining team at establishing detailed requirements to efficiently operate a mine of this size. Table 21.3 summarizes the mine pre-production capital costs for the project.

**Table 21.3: Mine Pre-Production Capital Costs**

| Capital Costs | Total (US$M) |
|---|---|
| Mine Infrastructure | 0.5 |
| Mine Equipment | 41.8 |
| Total Mining (excluding Contingency) | 42.3 |

Source: GMS (2021).

### 21.6.2 Surface Construction Costs

Surface construction costs include site development, mineral processing plant, tailings management facility, and on-site and off-site infrastructure. These cost estimates stem from an FS level of engineering of the various infrastructure and are primarily based on material and equipment quotations received during Q3 and Q4 2021, workhours based on preliminary construction quantities, and detailed equipment lists. Pricing for main equipment was primarily derived from quoted sources, with some factors applied for bulk material and minor cost elements.

Table 21.4 presents a summary basis of estimate for the various commodity types within the surface construction estimates.

Sustaining capital costs primarily include what is required for DSTF construction and mine maintenance rebuilds.

DSTF earthworks quantities were developed from FS-level engineering drawings by the design engineer (NewFields). Database unit rates have been received from national contractors and benchmarked against projects recently constructed in the region. These rates have been applied to the engineered quantities.

## 21.7 Indirect Cost Estimate

Indirect costs are classified as costs not directly accountable to a specific cost object.

Table 21.5 presents the subjects and basis for the indirect costs within the capital cost estimate.

In addition to project and construction management costs, the detailed engineering estimate includes costs associated with further geotechnical analysis work (refer to Section 18.3).

**Table 21.4: Surface Construction Basis of Estimate**

| Commodity | Basis |
|---|---|
| Access Roads & Bridge | Material take-offs developed based on general arrangements by GMS. Local contractor quote for bridge. |
| Bulk Earthworks | Model volumes from 3D grading model. Database productivity unit rates for bulk excavation and fill. Material take-offs for surface drainage, water management ponds and temporary roads from engineering drawings. National contractor quotations. |
| Concrete | Material take-offs from FS-level engineering and 3D model and quoted rates from multiple, local and international suppliers. Workhour estimates based on MTOs and estimated productivities. |
| Structural Steel | Material take-offs from FS-level engineering and quoted rates from multiple, local and international suppliers. Workhour estimates based on MTOs and estimated productivities. |
| Pre-Engineered Buildings, Modular Buildings and Warehouses | Buildings sized according to general arrangements, with quotations for overall building structures. Estimates for lighting, small power, electrical/control rooms, and fire detection established by GMS. Material take-offs established from FS-level engineering. |
| Mechanical Equipment | All major mechanical equipment quoted by multiple vendors. Installation hours based on equipment sizes and productivity estimates. |
| Piping | In the process plant, MTOs for large bore pipelines equal to or greater than 2" NPS and factored values based on mechanical equipment cost for smaller diameters. For other infrastructures and buildings, material take-offs for small bore piping. |
| Electrical and Instrumentation | Major electrical equipment list prepared and average cable runs prepared in neat line material take-offs. Major instruments list prepared and priced using quotations submitted by vendors. Minor instruments were factored, although majority were included in vendor packages. |
| Dewatering | Number of wells based on dewatering plan. Costs based on quoted rates from experienced local drilling contractor. Dewatering pump costs quoted by multiple vendors. |
| Power Transmission Line and Major Substations | Quantities developed based on general arrangements and site layouts. A combination of quoted and database costs applied from similar projects and local contractor quoted for bidding pricing to execute the transmission line. |

Source: GMS (2021).

**Table 21.5: Indirect Cost Basis of Estimate**

| Commodity | Basis |
|---|---|
| On-Site Contract Services | Heavy lift crane services based on estimated durations and quoted rates for crane services. |
| Owner`s & Contractor Field Indirect | Estimated by first principles and including the items below. |
| | Time based cost allowance for general construction site services (temporary power, heating, contractor support, etc.) applied against the surface construction schedule. |
| | Construction offices and wash car facilities. |
| | Safety training, tools and equipment. |
| | Environmental cost. |
| | Materials management and warehouse operations. |
| | Site services, maintenance and temporary services. |
| | General administration costs. |
| | Human resources costs. |
| | Social responsibility and legal costs. |
| | Surveying and quality assurance. |
| | Communications. |
| | Contractor facilities and related cost. |
| | Construction team facilities, fuel. |
| Freight & Logistics | Quoted freight for major mechanical equipment. Remainder of equipment is factored (11%) for freight and logistics related to the materials and equipment required for the remainder of international bulk materials and equipment. Factor excludes mining equipment as prices quoted include delivery to site. |
| Vendor Representatives | Estimated by first principles, assessing the equipment supply packages and vendor services hours required for commissioning equipment. |
| Capital Spares | Based on material take-offs provided by vendors with updated pricing. |
| Start-up & Commissioning | Included under pre-production costs 900 WBS series team costs (material and consumables). |
| First Fills | Based on requirements determined by engineering/process and quoted pricing for reagents and grinding media. |
| Detailed Engineering & Procurement | Detailed labour estimate based on deliverables for engineering and drafting, and time based on project management services required to oversee project development. Costs are based on a self-perform execution strategy. A schedule of rates was applied against a staffing plan aligned with the project schedule. |
| Project & Construction Management | Staffing plan built up against the development schedule for project management, health and safety, construction management, field engineering, project controls, contract administration and supply chain. Costs are based on a self-performed execution strategy. A schedule of rates was applied against a staffing plan aligned with the project schedule. |

Source: GMS (2021).

## 21.8 Owner's Cost Estimate

Owner's costs are capitalized in the initial capital costs during the construction phase. Owner's costs for the project start in Month 1 of Year -3 of the capital cost cash flow. Any Owner's costs prior to this are assumed to be within the Owner's approved budget expenses and are considered sunk costs.

### 21.8.1 Process Plant Operations

The following processing related costs are included in the pre-production costs:

- management, technical, operations, and maintenance labour employed during the construction phase
- first fills of consumables and reagents, and initial consumption during process commissioning to initiate operations
- energy costs for power consumed during process commissioning and start-up activities.

### 21.8.2 Water Treatment Plant Operation

The following cost elements are included in the pre-production costs for operation of the water treatment plant:

- technical and operations labour
- maintenance and parts
- power consumption
- reagents, consumables, and third-party services.

### 21.8.3 Dewatering Wells Operations

The following costs elements are included in the pre-production costs for operations of dewatering wells:

- supervision, technical and operations labour
- maintenance
- power consumption
- reagents, consumables, and third-party services.

### 21.8.4 Pre-Production G&A – Labour

Costs for general and administrative labour are included for the following sectors:

- Accounting
- General Management

- Community Relations
- Health and Safety
- Environmental
- Human Resources
- Supply Chain
- Security
- Site Services and Facilities Maintenance
- Legal
- Information Technology
- Corporate Affairs.

#### 21.8.5 Pre-Production G&A – Equipment

Costs for Owner site support equipment usage are included for the following sectors:

- Supply Chain
- Site Services
- Health, Safety, and Environment
- Administration / Management.

#### 21.8.6 Pre-Production G&A – Expenses & Services

Costs for general and administrative (G&A) expenses and fees are included for the following sectors:

- Health, Safety and Medical Supplies
- Staff Safety Equipment
- Environmental Services, Fees, and Outside Laboratory Costs
- Human Resources (Training, Recruitment)
- Construction Insurance
- Community Relations and Programs
- Legal and Regulatory, including Property Tax
- External Consulting
- IT and Communications
- Site Office Costs
- Office Lease and Services for Guatemala City

- Waste Disposal
- Existing Infrastructure Power and Maintenance.

### 21.8.7 Pre-Production Mining Costs

The pre-production mining costs were developed by GMS from first principles and are consistent with the basis of the operating costs. Pre-production mining costs are presented in Table 21.6.

**Table 21.6: Pre-Production Mining Costs**

| Capital Costs | Total ($M) |
|---|---|
| Operating Costs | 42.3 |
| Fuel | 7.2 |
| **Total Pre-Production Mining Costs** | 49.5 |

Source: GMS (2021).

## 21.9 Closure Cost Estimate

Closure costs have been similarly built up from first principals based on the planned disturbance and assuming typical closure, reclamation, and monitoring activities for an open pit mine that align with Guatemalan requirements and industry standards. Activities include the following:

- removal of all fixed equipment
- removal of all surface infrastructure and buildings
- process plant demolition
- closure and revegetation of the DSTF and WRF
- power transmission line and substation removal
- re-vegetation and seeding
- ongoing site monitoring
- WTP operation during two years after the initial closure phase
- landfill reclamation.

## 21.10 Contingency

Contingency has been applied to the estimate on an area and discipline basis as a deterministic allowance by assessing the level of confidence of the scope definition, supply cost and installation cost, and then applying Monte Carlo iteration analysis. The overall recommended pre-production contingency resulted in approximately 11% of direct, indirect, and Owner's costs.

## 21.11 Mining Sustaining Capital Costs

Sustaining capital for the mine includes additional equipment purchases and replacement of shorter-life ancillary equipment for a total of US$36.4 million. Major repairs to the main production fleets, which represent US$52.9 million over the life of mine, are also capitalized.

Mine sustaining capital costs are presented in Table 21.7.

**Table 21.7: Mine Sustaining Capital**

| Item | Total ($M) | Y1 | Y2 | Y3 | Y4 | Y5 | Y6 | Y7 | Y8 | Y9 | Y10 | Y11 | Y12 | Y13 | Y14 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Equipment Purchase | 36.4 | 6.6 | 0.7 | 3.6 | 5.8 | 3.7 | 4.9 | 1.3 | 1.6 | 2.3 | 3.2 | 1.5 | 1.0 | 0.0 | 0.0 |
| Major Repairs | 52.9 | 2.5 | 8.3 | 4.5 | 7.3 | 6.4 | 5.4 | 7.6 | 3.3 | 2.5 | 2.1 | 0.7 | 0.9 | 1.1 | 0.3 |
| **Total** | **89.2** | **9.1** | **9.0** | **8.1** | **13.1** | **10.1** | **10.3** | **8.9** | **4.9** | **4.8** | **5.3** | **2.2** | **1.9** | **1.1** | **0.3** |

Source: GMS (2021).

## 21.12 Capital Estimate Exclusions

Exclusions from the capital cost estimate include, but are not limited to, the following:

- working capital (included in the financial model)
- escalation
- financing costs
- currency fluctuations
- lost time due to severe weather conditions beyond those expected in the region
- lost time due to force majeure
- additional costs for accelerated or decelerated deliveries of equipment, materials or services resultant from a change in project schedule
- warehouse inventories, other than those supplied in initial fills, capital spares, or commissioning spares
- any project sunk costs (studies, exploration programs, early works etc.)
- value-added tax (VAT)
- closure bonding
- COVID-19 impact escalation cost.

## 21.13 Operating Cost Estimate

A summary of the operating cost estimate is provided in Table 21.8. The estimate includes the costs to mine and process the mineralized material to produce doré, along with site services to maintain the site, and general and administrative (G&A) expenses. The costs are expressed in U.S. dollars with no allowance for inflation. The target accuracy of the operating cost estimate is -10/+15%.

The operating cost estimate is broken into four major sections:

- open pit mining
- processing
- site services
- G&A.

The total operating unit cost is estimated to be US$29.56/t processed. Average annual, total life-of-mine, and unit operating cost estimates are also summarized in Table 21.8. The unit rates include tonnes mined during pre-production.

**Table 21.8: Summary of Operating Cost Estimate**

| Operating Costs | $/t Milled | Life of Mine ($M) |
|---|---|---|
| Mining | 10.42 | 550 |
| Processing | 12.97 | 685 |
| Site Services | 2.73 | 144 |
| G&A | 3.43 | 181 |
| **Total** | **29.56** | **1,560** |

Source: GMS (2021).

The main operating cost component assumptions are shown in Table 21.9.

**Table 21.9: Main Operations Components Assumptions**

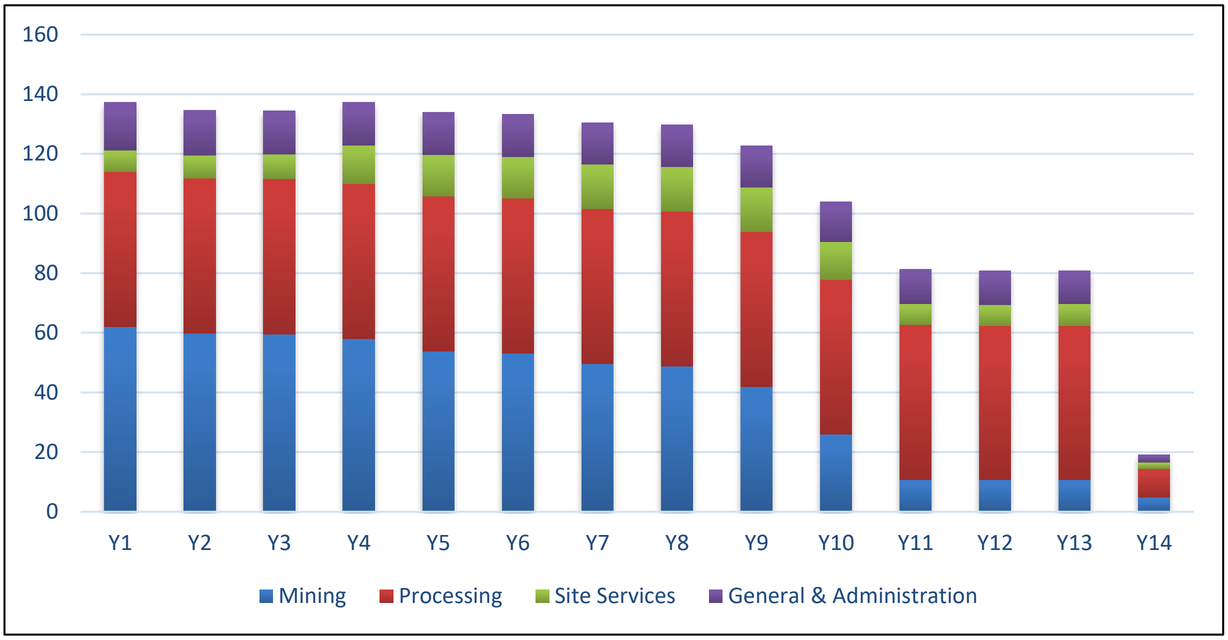
| Item | Unit | Value |
|---|---|---|
| Electrical Power Cost | $/kWh | 0.08 |
| Total Operating Load | MW | 37.7 |
| Peak Overall Power Consumption (all Facilities) | kWh/t processed | 41.8 |
| Diesel Cost (Delivered) | $/L | 0.65 |

Source: GMS (2021)

### 21.13.1 Operating Cost Profile

The operating costs for the project have been estimated against the development schedule to support the economic cash flow model. Figure 21-2 presents the annual life-of-mine operating cost profile.

**Figure 21-2: Operating Cost Profile (Million per Year)**




Source: GMS (2021).

### 21.13.2 Operational Labour Rate Build-up

Operational labour rates have been estimated by applying legal and discretionary burdens against base labour rates. Wage scales were defined and applied to the various operational positions based on skill level and expected salary. Elevar's human resources personnel were involved in the development and verification of the operational labour rates.

The Owner's team averages 545 positions during operations. Levels will fluctuate depending on the amount of yearly development.

### 21.13.3 Mining Operating Costs

Table 21.10 presents the breakdown of mining costs by department, and Table 21.11 presents the five major cost drivers for the mine department.

The mine operating costs are estimated from first principles for all mine activities. Equipment hours required to meet the production needs of the life-of-mine plan are based on productivity factors or equipment simulations. Each piece of equipment has an hourly operating cost that includes operating and maintenance labour, fuel and lube, maintenance parts, tires (if required) and ground engaging tools (if required). A budgetary request for proposal (RFP) process has been completed for the mine equipment and associated operating costs, fuel, tires, explosives and accessories, etc.

**Table 21.10: Mine Operating Costs**

| Mining Category | Unit Cost ($/t Processed) | Total Cost ($M) | Y1 | Y2 | Y3 | Y4 | Y5 | Y6 | Y7 | Y8 | Y9 | Y10 | Y11 | Y12 | Y13 | Y14 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Mine Operations | 0.27 | 14.3 | 2.2 | 2.2 | 1.7 | 1.2 | 1.2 | 1.2 | 1.2 | 1.2 | 1.2 | 0.8 | 0.0 | 0.0 | 0.0 | 0.0 |
| Mine Maintenance Admin. | 0.30 | 15.8 | 1.5 | 1.4 | 1.3 | 1.3 | 1.3 | 1.3 | 1.3 | 1.3 | 1.3 | 1.2 | 0.6 | 0.6 | 0.6 | 0.6 |
| Mine Geology | 0.15 | 8.1 | 0.8 | 0.8 | 0.8 | 0.8 | 0.8 | 0.8 | 0.8 | 0.8 | 0.7 | 0.4 | 0.2 | 0.2 | 0.2 | 0.2 |
| Mine Engineering | 0.28 | 14.8 | 1.9 | 1.5 | 1.5 | 1.9 | 1.5 | 1.5 | 1.5 | 1.3 | 1.1 | 1.1 | - | - | - | - |
| Grade Control | 0.24 | 12.8 | 1.6 | 1.5 | 1.5 | 1.4 | 1.4 | 1.4 | 1.3 | 1.3 | 1.0 | 0.4 | - | - | - | - |
| Drilling | 0.57 | 30.1 | 3.5 | 3.5 | 3.5 | 3.5 | 3.4 | 3.3 | 3.3 | 3.1 | 2.2 | 0.8 | - | - | - | - |
| Blasting | 2.37 | 125.2 | 18.7 | 16.0 | 16.0 | 16.0 | 13.0 | 12.6 | 11.1 | 10.5 | 7.9 | 3.4 | - | - | - | - |
| Pre-Split Drilling & Blasting | 0.16 | 8.3 | 0.3 | 0.7 | 0.8 | 0.7 | 0.8 | 0.7 | 1.0 | 1.2 | 1.0 | 1.1 | - | - | - | - |
| Loading | 0.71 | 37.7 | 4.3 | 4.4 | 4.4 | 4.4 | 4.3 | 4.2 | 4.1 | 3.9 | 2.8 | 1.0 | - | - | - | - |
| Hauling | 2.22 | 117.2 | 11.1 | 12.2 | 12.5 | 12.2 | 11.7 | 11.6 | 11.8 | 12.3 | 11.0 | 5.7 | 1.6 | 1.6 | 1.6 | 0.3 |
| Dump Maintenance | 0.30 | 16.0 | 1.5 | 1.5 | 1.5 | 1.5 | 1.5 | 1.5 | 1.5 | 1.5 | 1.0 | 0.5 | 0.5 | 0.5 | 0.5 | 0.5 |
| Road Maintenance | 0.48 | 25.3 | 2.4 | 2.4 | 2.4 | 2.4 | 2.4 | 2.4 | 2.4 | 2.4 | 2.3 | 1.6 | 0.5 | 0.5 | 0.5 | 0.5 |
| Dewatering | 0.09 | 4.7 | 0.9 | 0.4 | 0.4 | 0.4 | 0.4 | 0.4 | 0.5 | 0.5 | 0.5 | 0.5 | - | - | - | - |
| Support Equipment | 0.42 | 22.3 | 2.0 | 2.0 | 2.0 | 2.0 | 2.0 | 2.0 | 2.0 | 2.0 | 1.9 | 1.1 | 0.8 | 0.8 | 0.8 | 0.8 |
| Rehandling | 0.09 | 4.7 | 0.2 | 0.2 | 0.2 | 0.2 | 0.1 | - | - | - | 0.3 | 0.6 | 0.9 | 0.9 | 0.9 | 0.2 |
| **Total Mining Cost** | **8.66** | **457.3** | **53.0** | **50.8** | **50.5** | **49.9** | **45.7** | **45.0** | **43.9** | **43.2** | **36.3** | **20.4** | **5.2** | **5.2** | **5.2** | **3.2** |
| **Total Cost/t Mined** | | **2.5** | **2.5** | **2.4** | **2.4** | **2.4** | **2.2** | **2.3** | **2.2** | **2.4** | **2.7** | **4.4** | **-** | **-** | **-** | **-** |
| | | | | | | | | | | | | | | | | |
| Tailing Rehandling | 1.41 | 74.6 | 5.6 | 5.6 | 5.6 | 5.6 | 5.6 | 5.6 | 5.6 | 5.6 | 5.6 | 5.6 | 5.6 | 5.6 | 5.6 | 1.7 |
| Limestone Management | 0.34 | 17.9 | 3.3 | 3.5 | 3.5 | 2.5 | 2.5 | 2.6 | 0.1 | - | - | - | - | - | - | - |
| **Total Cost/t Processed** | **-** | **10.4** | **15.5** | **15.0** | **14.9** | **14.5** | **13.5** | **13.3** | **12.4** | **12.2** | **10.5** | **6.5** | **2.7** | **2.7** | **2.7** | **6.4** |

Source: GMS (2022)

**Table 21.11: Mine Operating Cost by Account**

| Top Mining Components | Total Cost ($M) | Y1 | Y2 | Y3 | Y4 | Y5 | Y6 | Y7 | Y8 | Y9 | Y10 | Y11 | Y12 | Y13 | Y14 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Explosive - Bulk Product | 120.5 | 16.6 | 13.9 | 13.9 | 13.9 | 11.0 | 10.6 | 9.2 | 8.7 | 6.3 | 2.4 | 0.0 | 0.0 | 0.0 | 0.0 |
| Maintenance Parts | 105.8 | 9.8 | 10.1 | 10.2 | 10.1 | 9.8 | 9.6 | 9.7 | 9.7 | 8.3 | 4.6 | 1.8 | 1.8 | 1.8 | 0.9 |
| Diesel/Fuel | 89.3 | 7.9 | 8.5 | 8.6 | 8.4 | 8.2 | 8.1 | 8.2 | 8.3 | 7.2 | 4.1 | 1.3 | 1.3 | 1.3 | 0.7 |
| Salaries | 73.3 | 7.0 | 7.0 | 6.5 | 6.0 | 6.0 | 6.0 | 6.1 | 5.8 | 5.3 | 4.0 | 0.9 | 0.9 | 0.9 | 0.9 |
| Tires | 26.5 | 2.3 | 2.5 | 2.6 | 2.6 | 2.5 | 2.4 | 2.4 | 2.5 | 2.3 | 1.3 | 0.4 | 0.4 | 0.4 | 0.1 |
| Subtotal Top Five | 415.4 | 43.6 | 42.1 | 41.7 | 41.0 | 37.4 | 36.7 | 35.6 | 35.0 | 29.5 | 16.3 | 4.4 | 4.4 | 4.4 | 2.6 |
| Percentage of Total | 82% | 82% | 83% | 83% | 82% | 82% | 82% | 81% | 81% | 81% | 80% | 86% | 86% | 86% | 84% |

Source: GMS (2022)

The average mining cost during operations is estimated at US$2.53 per tonne mined including re-handling costs. The mining costs are lower than average during the early years and increase with increased haulage distances and pit deepening in the later years. This operating cost estimate excludes capital repairs, which are treated as sustaining capital.

Blasting is the major mining cost activity, representing 27% of total costs, followed by hauling (26%) and loading (8%). Loading and hauling for stockpile re-handling is also captured as a separate activity cost.

Maintenance parts is the dominant cost by element, representing 23% of total costs, followed by bulk explosive (21%), diesel (19%) and salaries (15%).

#### 21.13.4 Processing Operating Costs

Processing operating costs were estimated to include all activities to produce unrefined gold and silver doré on site. The crushing and process plants are designed for a throughput of 11,747 t/d. Process operating costs are summarized in Table 21.12.

**Table 21.12: Processing Plant Operating Costs**

| Processing Category | Unit Cost ($/t processed) | Life-of-Mine Cost ($M) |
|---|---|---|
| Labour | 0.71 | 37.2 |
| Power | 5.11 | 269.6 |
| Maintenance and Consumables | 3.60 | 189.9 |
| Processing Reagents | 3.41 | 179.8 |
| Others | 0.09 | 8.5 |
| **Total Processing Operating Cost** | **12.97** | **684.9** |

Source: GMS (2022).

##### 21.13.4.1 Mineral Processing Labour

Milling operations and maintenance staffing levels have been built up based on experience at similar operations. Labour costs are based on a fully burdened staffing wage.

##### 21.13.4.2 Mineral Processing Power

Electrical power consumption has been based on the equipment connected loads, discounted for operating time and the anticipated operating load level.

The total estimated annual process plant energy consumption is 255.4 MWh/a. At an estimated power cost of $0.08/kWh, the life-of-mine power cost is $269.6 million ($5.11/t processed).

#### 21.13.4.3 Grinding Media & Liners

Liners for the grinding mills and primary crusher have been estimated based on vendor quotes, ore hardness, and experience at similar operations. Grinding media consumption for the grinding mills have been estimated on a kilogram-per-tonne basis, mill power draw, and abrasion index. Budgetary quotations for liners and grinding media were received from equipment vendors and local suppliers.

#### 21.13.4.4 Reagents

Milling reagent consumption rates have been determined from the metallurgical test data. Unit pricing is based on budgetary quotations from local suppliers obtained through the site team. Table 21.13 presents a summary of reagent requirements and costs.

**Table 21.13: Reagent Requirements & Costs**

| Reagents | Reagent Consumption | Unit Reagent Cost (US$/t) | Annual Cost (US$M) |
|---|---|---|---|
| Sodium Cyanide | 1,208 t/y | 3,038 | 3.7 |
| Hydrated Lime | 6,840 t/y | 143 | 1.0 |
| Carbon | 120 t/y | 2,930 | 0.4 |
| Sodium Hydroxide | 225 t/y | 781 | 0.2 |
| Hydrochloric Acid | 533 t/y | 457 | 0.3 |
| Copper Sulphate | 840 t/y | 2,514 | 2.1 |
| Sodium Metabisulphate | 3,120 t/y | 1,194 | 3.7 |
| Lead Nitrate | 1,000 t/y | 980 | 1.0 |
| Flocculant | 240 t/y | 4,760 | 1.2 |
| Antiscalant | 2,100 L/y | 4,148 | 0.01 |
| **Total** | | | **13.6** |

Source: GMS (2022).

#### 21.13.4.5 Maintenance Parts

Annual maintenance parts costs have been factored at a rate of 4% of the direct capital costs of the equipment within each area.

### 21.13.5 Site Services Operating Costs

The site services operating cost summary is show in Table 21.14.

**Table 21.14: Site Services Operating Costs**

| Sector | Average US$M/a | Life of Mine US$M | US$/t Processed |
|---|---|---|---|
| Water Treatment Plant | 7.67 | 107.4 | 2.03 |
| Sewage Treatment Plant | 0.11 | 1.6 | 0.03 |
| Dewatering Wells | 2.00 | 28.1 | 0.53 |
| Domestic Water Treatment Plant | 0.06 | 0.9 | 0.02 |
| Surface Water Collection System | 0.46 | 6.5 | 0.12 |
| **Total Site Services** | **10.31** | **144.4** | **2.73** |

Source: GMS (2022).

#### 21.13.5.1 Total Site Services

General site services cover the cost of water treatment, non-contact water, dewatering wells services and surface water collection system maintenance.

#### 21.13.5.2 Water Treatment Plant

The water treatment plant operating cost includes all costs related to the treatment of site dewatering for use in the process or discharge to the environment. Water treatment operating costs are based on the unit power and reagent consumptions and provisions for maintenance services from the existing operations at site. The unit costs for chemicals, consumables, and power have been used to estimate the planned increases in water treatment plant throughput.

#### 21.13.5.3 Dewatering

The dewatering operating costs include the costs to manage, operate and maintain the dewatering wells, and associated conveyance systems. The costs are based on the dewatering schedule designed for the project.

### 21.13.6 General & Administration Operating Costs

A summary of general and administration (G&A) operating costs is provided in Table 21.15.

**Table 21.15: General & Administration (G&A) Operating Cost Summary**

| Sector | Average US$M/a | Life of Mine US$M | US$/t Processed |
|---|---|---|---|
| General Management | 0.5 | 7.2 | 0.14 |
| Health & Safety | 0.9 | 12.6 | 0.24 |
| Accounting | 0.7 | 8.9 | 0.17 |
| Information Technology | 1.2 | 16.5 | 0.31 |
| Supply Chain | 1.8 | 24.5 | 0.46 |
| Human Resources | 0.5 | 6.1 | 0.12 |
| Site Services | 2.6 | 35.3 | 0.67 |
| Community & Social Responsibility (CSR) | 0.9 | 12.1 | 0.23 |
| Security | 1.5 | 20.1 | 0.38 |
| Environment | 1.9 | 25.8 | 0.49 |
| Corporate Affairs | 0.4 | 5.9 | 0.11 |
| Legal | 0.5 | 6.2 | 0.12 |
| **Total G&A** | **13.4** | **181.0** | **3.43** |

Source: Bluestone (2022).

#### 21.13.6.1 G&A Labour

G&A staffing levels have been prepared based on a current staff plan for the Cerro Blanco Gold project, consultation with Bluestone management, and experience at similar operations. Labour costs are based on fully burdened staffing wages.

#### 21.13.6.2 G&A Services & Expenses

G&A services and expenses have been estimated in consultation with current Bluestone management and by taking into consideration similar operations. Major items are based on current budgeted operating costs of Cerro Blanco in Guatemala. Minor items are factored based on estimate parameters, such as numbers of employees, or are general allowances based on experience with other projects and input from Bluestone.

### 21.13.7 Taxes

Value-added tax (VAT) applies to goods and services provided in Guatemala; however, VAT is fully recoverable for metal producers. As such, no provisions for VAT are included in the operating cost estimate. However, VAT has been accounted for in the financial model.

### 21.13.8 Contingency

No operating cost contingency provision has been included in the estimate.

# 22. ECONOMIC ANALYSES

An economic model was developed to estimate annual cash flows and Project sensitivities. Pre-tax estimates of Project values were prepared for comparative purposes, while after-tax estimates were developed and are likely to approximate the true investment value. It must be noted, however, that tax estimates involve many complex variables that can only be accurately calculated during operations and, as such, the after-tax results are only approximations.

Sensitivity analyses were performed for variations in metal prices, head grades, operating costs, capital costs, and discount rates to determine their relative importance as Project value drivers.

This technical report contains forward-looking information regarding Projected mine production rates, construction schedules, and forecasts of resulting cash flows. The mill head grades are based on sufficient sampling that is reasonably expected to be representative of the realized grades from actual mining operations. Factors such as the ability to obtain or amend permits to construct and operate a mine, or to obtain major equipment or skilled labour on a timely basis to achieve the assumed mine production rates at the assumed grades, may cause actual results to differ from those presented in this economic analysis.

The estimates of capital and operating costs have been developed specifically for this Project and are summarized in Section 21 of this report. The economic analysis has been run with no inflation (constant dollar basis).

## 22.1 Life-of-Mine Summary & Assumptions

A summary of the mine plan and payable metals produced is provided in Table 22.1. Other economic factors are listed below.

- A discount rate of 5% is applied.
- The analysis is performed in nominal 2021 dollars.
- Revenues, costs, taxes are calculated for each period in which they occur rather than actual outgoing/incoming payment.
- Working capital is calculated as three months of consumables, one week of plant inventory, and two weeks of accounts receivable.
- Results are presented based on 100% ownership.
- No management fees or financing costs (equity fund-raising was assumed) are applied.
- The model excludes all pre-development and sunk costs (i.e., exploration and resource definition costs, engineering fieldwork and studies costs, environmental baseline studies costs, financing costs, etc.).

**Table 22.1: Life-of-Mine Summary**

| Parameter | Unit | Value |
|---|---|---|
| Total Material Mined | Mt | 199.3 |
| Ore Processed | Mt | 53.9 |
| Strip Ratio | waste:ore | 2.7 |
| Mill Average Daily Production | kt | 11,747 |
| Mill Average Annual Production | Mt/a | 4.00 |
| Average Gold Mill Grade | g/t | 1.64 |
| Average Silver Mill Grade | g/t | 7.27 |
| Gold Contained | koz | 2,846 |
| Silver Contained | koz | 12,602 |
| Gold Recovered | koz | 2,645 |
| Silver Recovered | koz | 10,617 |
| Gold Recovery | % | 93% |
| Silver Recovery | % | 84% |
| Peak Gold Production | koz/year | 347 |
| Avg. Gold Production Years 1 - 4 | koz/year | 297 |
| Avg. Silver Production Years 1 - 4 | koz/year | 1,356 |
| Initial Capital Cost | US$M | 572 |
| Sustaining Capital Cost | US$M | 140 |
| Total Life of Mine Capital | US$M | 712 |
| Closure Cost | US$M | 38.1 |

Source: GMS (2022).

Table 22.2 outlines the metal prices and exchange rate assumptions used in the economic analysis. The gold and silver price is in line with recently released comparable technical reports.

**Table 22.2: Metal Prices & Exchange Rates**

| Assumptions | Unit | Value |
|---|---|---|
| Gold Price | US$/oz | 1,600 |
| Silver Price | US$/oz | 20.00 |
| FX Rate | GTQ:USD | 7.69 |
| | CAD:USD | 1.32 |

Source: GMS (2021).

The reader is cautioned that the metal prices and exchange rates used in this study are only estimates based on recent historical performance. There is no guarantee that they will be realized if the Project is taken into production.

## 22.2 Net Smelter Return Parameters

Mine revenue is derived from the sale of doré bars into the international marketplace. No contractual arrangements for refining currently exist. Table 22.3 indicates the net smelter return (NSR) parameters that were used in the economic analysis.

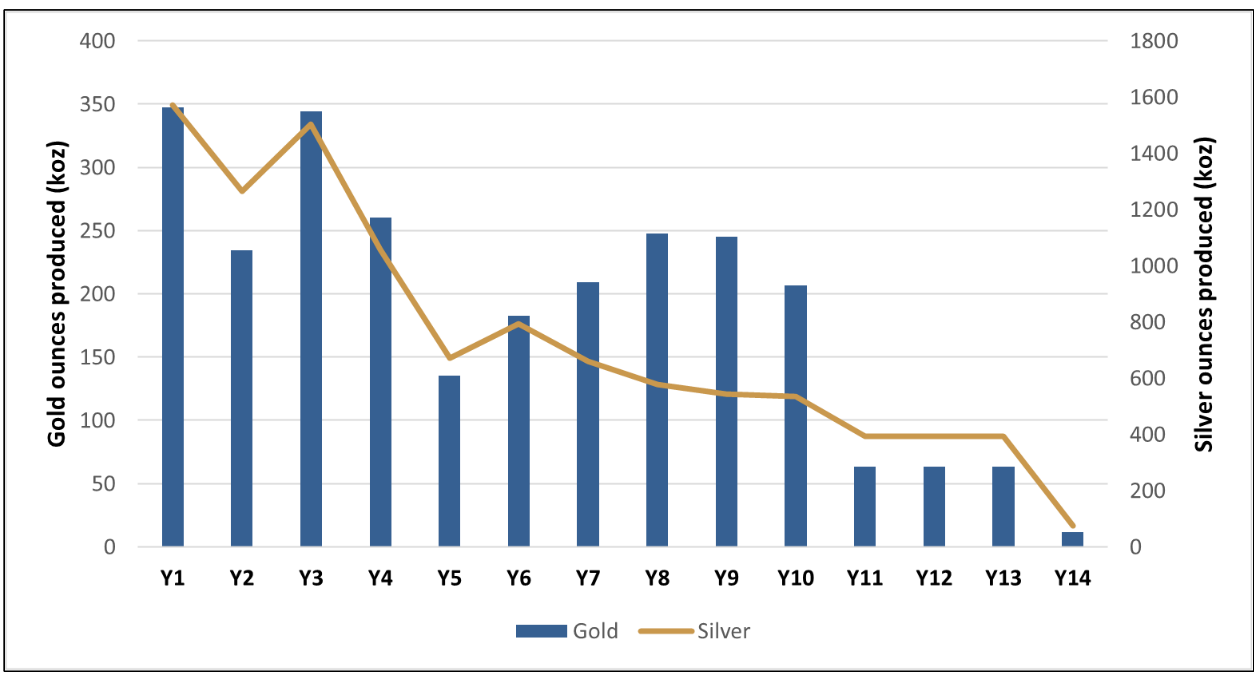
**Table 22.3: NSR Parameters**

| Parameter | Unit | Value |
|---|---|---|
| Mine Operating Days | days/year | 365 |
| Gold Recovery | % | 93.0 |
| Silver Recovery | % | 84.3 |
| Gold Payable | % | 99.92 |
| Silver Payable | % | 99.5 |
| Gold Refining Charge | US$/payable oz | 1.00 |
| Silver Refining Charge | US$/payable oz | 0.50 |

Source: GMS (2020).

Figure 22-1 shows the grade and the amount of gold and silver recovered during the mine life. A total of 2,645 koz of gold and 10,617 koz of silver is Projected to be produced over the life of mine. Gross Project revenues are divided into 95% and 5% for gold and silver, respectively.

**Figure 22-1: Life-of-Mine Payable Gold & Silver**




Source: GMS (2022).

## 22.3 Taxes

The Project has been evaluated on an after-tax basis to provide a more indicative, but still approximate, value of the potential Project economics. A tax model was prepared by Wentworth Taylor, an independent tax consultant, and reviewed by Bluestone personnel. Current tax pools were used in the analysis. The tax model contains the assumptions listed below and in Table 22.4:

- the lesser of 25% on net income for tax purposes or 7% on gross income
- no state tax
- VAT modelled with a net zero impact due to expected VAT credits and status as exporter
- withholding taxes assumed to be 5% on royalties
- stamp tax (captured within contingency)
- capital cost allowance based on units of production and at specific rates in the Tax Act
- exploration carry-forwards of $149 million applied on a straight-line basis over five years.

Total taxes for the Project amount to US$286 million.

**Table 22.4: Royalty Assumptions**

| Parameter | Unit | Value |
|---|---|---|
| Guatemalan Government Royalty | % NSR | 1.00 |
| Third Party Royalty | % NSR | 1.05 |

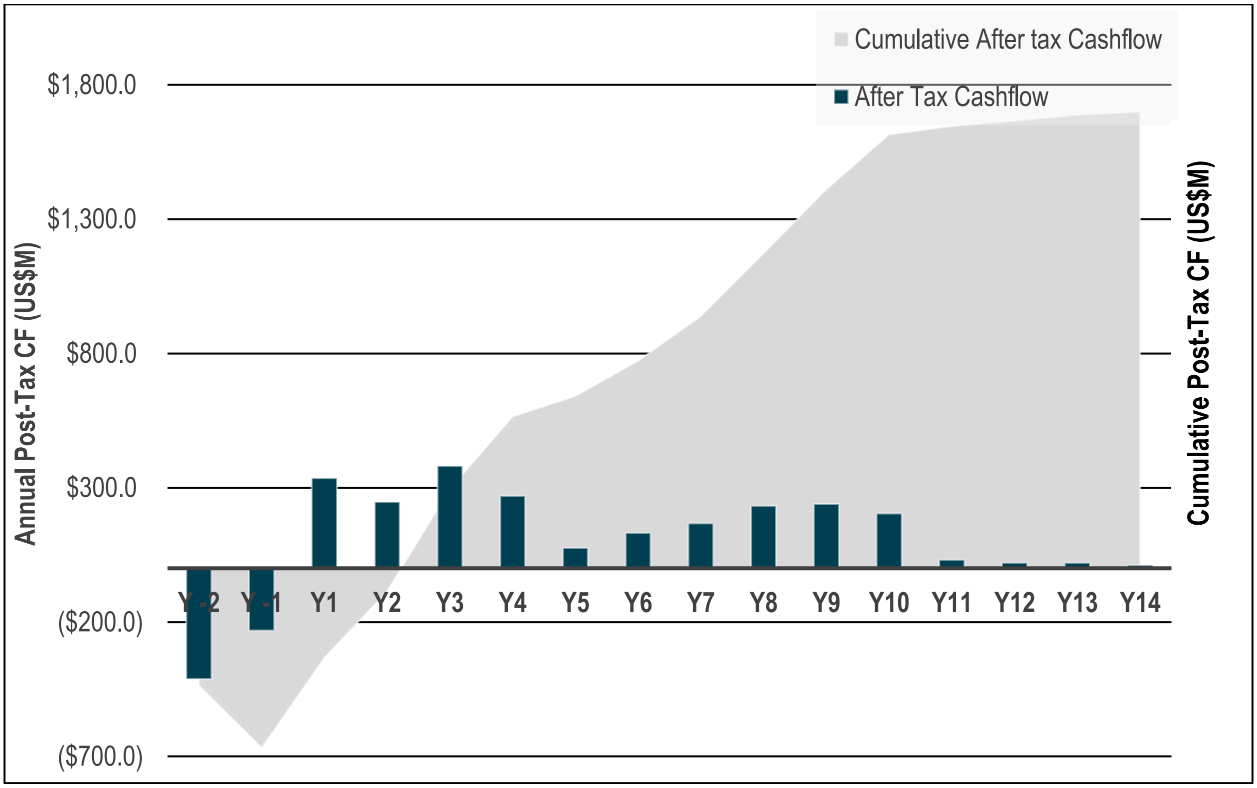
Note: 1.05% royalty has been grossed up to account for country withholding tax.
Source: Bluestone (2021).

## 22.4 Results

The Cerro Blanco Gold Project has a post-tax IRR of 30% and a net present value using a 5% discount rate (NPV5%) of $1,047 million using the metal prices described in Section 22.1. Figure 22-2 shows the Projected cash flows, and Table 22.5 summarizes the economic results of the Cerro Blanco Gold Project.

The life-of-mine, all-in sustaining cost (AISC) and AISC (net of by-product) is US$709/oz and US$629/oz, respectively. The straight AISC cost is calculated by adding the refining, transport, royalty, operating, and sustaining and closure costs together and dividing by total payable ounces of gold. This calculation does not consider the value of silver. The AISC (net of by-product) is a similar calculation. It adds the refining, transportation, royalty, operating and sustaining and closure costs, but subtracts the value of the silver before dividing by total payable ounces of gold.

**Figure 22-2: Annual After-Tax Cash Flow**




Source: GMS (2022).

**Table 22.5: Summary of Results**

| Parameter | Unit | Result |
|---|---|---|
| AISC* | US$/oz | 709 |
| AISC (Net of By-product)** | US$/oz | 629 |
| **Capital Costs** | | |
| Pre-Production Capital | $M | 619.0 |
| Pre-Production Contingency | $M | 60.7 |
| Pre-Production Revenue | $M | 47.5 |
| Sustaining and Closure Capital | $M | 178.4 |
| Sustaining and Closure Contingency | $M | 0.0 |
| Total Sustaining and Closure Capital | $M | 178.4 |
| Working Capital | $M | 32.2 |
| Pre-tax Cash Flow | LOM $M | 2,571 |
| | $M/a | 185.1 |
| Taxes | LOM $M | 286.3 |
| After-Tax Cash Flow | LOM $M | 2,349 |
| | $M/a | 169.1 |
| **Economic Results** | | |
| Pre-Tax NPV 5% | US$M | 1,265 |
| Pre-Tax IRR | % | 37.3% |
| Pre-Tax Payback | years | 1.7 |
| After-Tax NPV 5% | US$M | 1,047 |
| After-Tax IRR | % | 30.3% |
| **After-Tax Payback** | **years** | **2.2** |

Notes: *All-in sustaining cost is calculated as (refining & shipping costs + royalties+ operating costs + sustaining and closure capital)/payable gold ounces. **All-in sustaining cost (net of by-product) is calculated as (refining & shipping costs + royalties+ operating costs + sustaining capital + closure capital - payable silver value) / payable gold ounces. Source: GMS (2022).

## 22.5 Sensitivities

A univariate sensitivity analysis was performed to examine which factors most affect the Project economics when acting independently of all other cost and revenue factors. Each variable evaluated was tested using the same percentage range of variation, from -25% to +25%, although some variables may actually experience significantly larger or smaller percentage fluctuations over the life of mine. For instance, the metal prices were evaluated at a ±25% range to the base case, while the recovery and all other variables remained constant. This may not be truly representative of market scenarios, as metal prices may not fluctuate in a similar trend. The variables examined in this analysis are those commonly considered in similar studies—their selection for examination does not reflect any particular uncertainty.

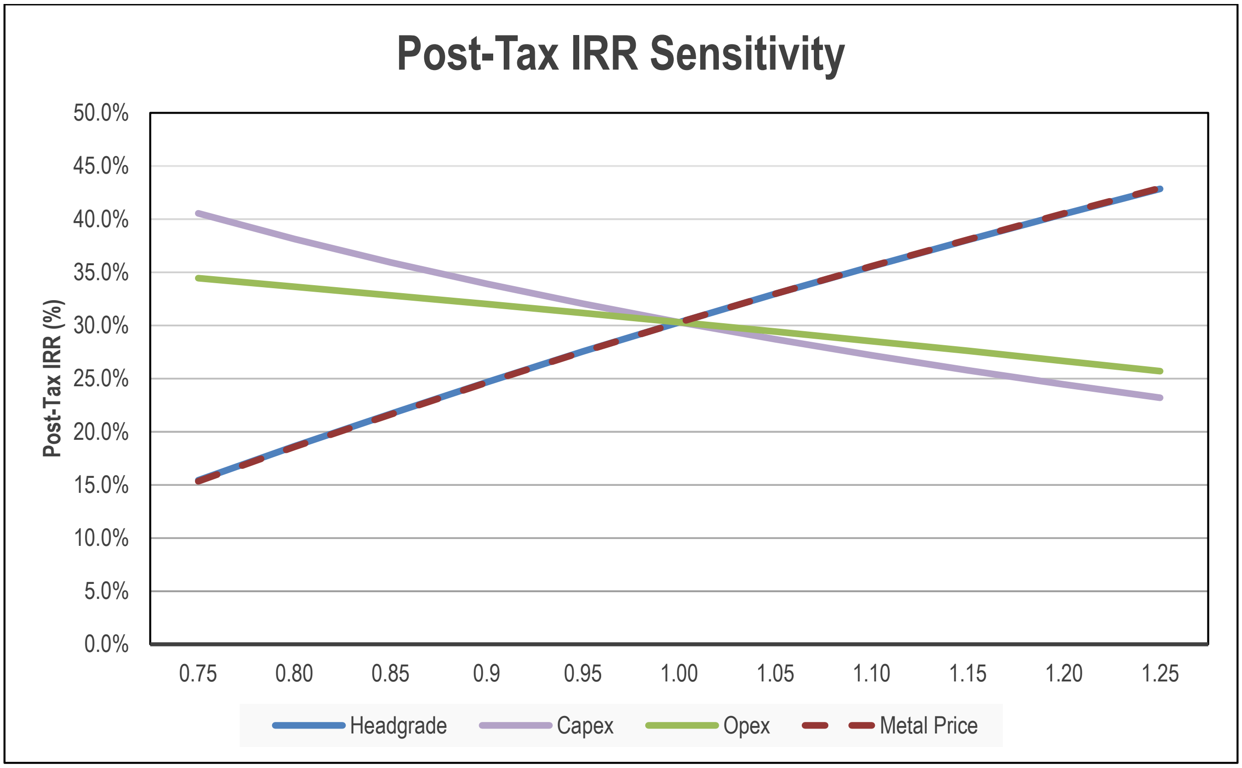
Notwithstanding the above limitations to the sensitivity analysis, which are common to studies of this nature, the analysis revealed that the Project is most sensitive to metal prices and head grade. The Project showed the least sensitivity to capital costs. Table 22.6 and Figure 22-3 show the results of the sensitivity tests.

**Table 22.6: Pre-Tax & After-Tax Sensitivity Results on NPV @ 5%**

| Variable | After-Tax NPV5% ($M) | | |
|---|---|---|---|
| | -25% Variance | 0% Variance | 25% Variance |
| Metal Price | $355 | $1,047 | $1,739 |
| Mill Head Grade | $359 | $1,047 | $1,735 |
| Operating Cost | $1,297 | $1,047 | $788 |
| Capital Cost | $1,222 | $1,047 | $872 |

Source: GMS (2022).

**Figure 22-3: Post-Tax IRR Sensitivity**




Source: GMS (2022).

The economic cash flow model for the Project is illustrated in Table 22.7.

**Table 22.7: Economic Cash Flow Model**

| Description | Unit | Pre-Production Total | Production Total | LOM Total | Y -3 | Y -2 | Y -1 | Y1 | Y2 | Y3 | Y4 | Y5 | Y6 | Y7 | Y8 | Y9 | Y10 | Y11 | Y12 | Y13 | Y14 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| **METAL PRICES & F/X RATE** | | | | | | | | | | | | | | | | | | | | | |
| Gold (Au) | US$/oz | 1,600 | 1,600 | 1,600 | 1,600 | 1,600 | 1,600 | 1,600 | 1,600 | 1,600 | 1,600 | 1,600 | 1,600 | 1,600 | 1,600 | 1,600 | 1,600 | 1,600 | 1,600 | 1,600 | 1,600 |
| Silver (Ag) | US$/oz | 20.00 | 20.00 | 20.00 | 20.00 | 20.00 | 20.00 | 20.00 | 20.00 | 20.00 | 20.00 | 20.00 | 20.00 | 20.00 | 20.00 | 20.00 | 20.00 | 20.00 | 20.00 | 20.00 | 20.00 |
| USD:CAD | x:x | 0.76 | 0.76 | 0.76 | 0.76 | 0.76 | 0.76 | 0.76 | 0.76 | 0.76 | 0.76 | 0.76 | 0.76 | 0.76 | 0.76 | 0.76 | 0.76 | 0.76 | 0.76 | 0.76 | 0.76 |
| **MINING** | | | | | | | | | | | | | | | | | | | | | |
| **PRODUCTION** | | | | | | | | | | | | | | | | | | | | | |
| Total Tonnage Mined | kt | 18,781 | 180,502 | 199,283 | - | 1,650 | 17,130 | 21,000 | 21,000 | 21,000 | 21,000 | 20,583 | 20,000 | 19,583 | 18,435 | 13,283 | 4,619 | - | - | - | - |
| Waste Rock Mined | | 15,577 | 129,810 | 145,387 | - | 1,618 | 13,959 | 13,837 | 14,412 | 14,317 | 16,160 | 16,389 | 14,763 | 14,425 | 12,927 | 9,333 | 3,247 | - | - | - | - |
| Ore Tonnage Mined | | 3,204 | 50,692 | 53,896 | | 32 | 3,172 | 7,163 | 6,588 | 6,683 | 4,840 | 4,194 | 5,236 | 5,158 | 5,508 | 3,950 | 1,372 | - | - | - | - |
| Strip Ratio | | | 2.56 | 2.70 | | | | 1.9 | 2.2 | 2.1 | 3.3 | 3.9 | 2.8 | 2.8 | 2.3 | 2.4 | 2.4 | | | | |
| Au Grade | g/t | | | 1.64 | | 0.54 | 0.84 | 1.96 | 1.32 | 1.90 | 1.79 | 1.10 | 1.29 | 1.48 | 1.68 | 2.16 | 3.58 | | | | |
| Contained Au | troy koz | | | 2,846 | - | 0.6 | 85.6 | 452.0 | 279.3 | 407.6 | 279.1 | 148.9 | 217.5 | 245.9 | 297.2 | 274.0 | 157.9 | - | - | - | - |
| Ag Grade | g/t | | | 7.27 | - | 1.81 | 3.76 | 10.72 | 8.30 | 10.15 | 8.09 | 6.05 | 6.50 | 5.32 | 4.61 | 4.96 | 7.07 | | | | |
| Contained Ag | troy koz | | | 12,602 | - | 2 | 383 | 2,470 | 1,759 | 2,181 | 1,258 | 815 | 1,094 | 883 | 816 | 629 | 312 | - | - | - | - |
| **MINERAL PROCESSING** | | | | | | | | | | | | | | | | | | | | | |
| **PRODUCTION** | | | | | | | | | | | | | | | | | | | | | |
| Ore Milled | kt | 1,107 | 52,789 | 53,896 | - | - | 1,107 | 4,000 | 4,000 | 4,011 | 4,000 | 4,000 | 4,000 | 4,011 | 4,000 | 4,000 | 4,000 | 4,011 | 4,000 | 4,000 | 755 |
| Operating Days | day | 151 | 4,836 | 4,987 | | | 151 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 365 | 91 |
| Average Plant Throughput | Kt/d | 7,333 | 11,865 | 11,747 | - | - | 7,333 | 10,959 | 10,959 | 10,989 | 10,959 | 10,959 | 10,959 | 10,989 | 10,959 | 10,959 | 10,959 | 10,989 | 10,959 | 10,959 | 8,300 |
| Au Grade | g/t | | 1.65 | 1.64 | | | 1.04 | 2.87 | 1.95 | 2.83 | 2.16 | 1.14 | 1.52 | 1.75 | 2.07 | 2.04 | 1.73 | 0.56 | 0.56 | 0.56 | 0.56 |
| Contained Au | troy koz | | 2,809 | 2,846 | | | 37.0 | 368.7 | 250.2 | 365.5 | 277.8 | 146.1 | 195.6 | 225.7 | 266.2 | 262.2 | 222.6 | 71.7 | 71.5 | 71.5 | 13.5 |
| Ag Grade | g/t | | 7.33 | 7.27 | | | 4.51 | 14.55 | 11.75 | 13.88 | 9.81 | 6.29 | 7.36 | 6.08 | 5.32 | 5.02 | 4.96 | 3.66 | 3.66 | 3.66 | 3.66 |
| Contained Ag | troy koz | | 12,441 | 12,602 | | | 161 | 1,871 | 1,511 | 1,789 | 1,262 | 808 | 947 | 784 | 684 | 646 | 637 | 472 | 470 | 470 | 89 |
| **RECOVERY** | | | | | | | | | | | | | | | | | | | | | |
| Au | % | - | 93.2% | 93.0% | - | - | 78.0% | 94.2% | 93.9% | 94.2% | 93.9% | 92.6% | 93.4% | 92.6% | 93.1% | 93.6% | 92.9% | 88.5% | 88.5% | 88.5% | 88.5% |
| Ag | % | - | 84.3% | 84.3% | - | - | 83.0% | 84.5% | 84.1% | 84.5% | 84.3% | 83.3% | 84.2% | 84.8% | 84.8% | 84.6% | 84.4% | 83.8% | 83.8% | 83.8% | 83.8% |
| **METAL RECOVERED** | | | | | | | | | | | | | | | | | | | | | |
| Au | koz | 29 | 2,616 | 2,645 | - | - | 29 | 347 | 235 | 344 | 261 | 135 | 183 | 209 | 248 | 245 | 207 | 63 | 63 | 63 | 12 |
| Ag | koz | 133 | 10,484 | 10,617 | - | - | 133 | 1,580 | 1,271 | 1,512 | 1,064 | 674 | 798 | 665 | 579 | 547 | 538 | 395 | 394 | 394 | 74 |
| **SALES & NSR** | | | | | | | | | | | | | | | | | | | | | |
| **PAYABLE METALS** | | | | | | | | | | | | | | | | | | | | | |
| Au Payable | % | 99.9% | 99.9% | 99.9% | - | - | 99.9% | 99.9% | 99.9% | 99.9% | 99.9% | 99.9% | 99.9% | 99.9% | 99.9% | 99.9% | 99.9% | 99.9% | 99.9% | 99.9% | 99.9% |
| | koz | 29 | 2,614 | 2,643 | - | - | 29 | 347 | 235 | 344 | 261 | 135 | 183 | 209 | 248 | 245 | 207 | 63 | 63 | 63 | 12 |
| | US$M | $46.2 | $4,182.8 | $4,229.0 | - | - | 46.2 | 555.4 | 375.5 | 550.6 | 416.9 | 216.2 | 292.0 | 334.3 | 396.1 | 392.4 | 330.5 | 101.4 | 101.2 | 101.2 | 19.1 |
| Ag Payable | % | 99.5% | 99.5% | 99.5% | - | - | 99.5% | 99.5% | 99.5% | 99.5% | 99.5% | 99.5% | 99.5% | 99.5% | 99.5% | 99.5% | 99.5% | 99.5% | 99.5% | 99.5% | 99.5% |
| | koz | 133 | 10,432 | 10,564 | - | - | 133 | 1,572 | 1,264 | 1,504 | 1,059 | 670 | 794 | 661 | 576 | 544 | 535 | 393 | 392 | 392 | 74 |
| | US$M | $2.7 | $208.6 | $211.3 | - | - | 2.7 | 31.4 | 25.3 | 30.1 | 21.2 | 13.4 | 15.9 | 13.2 | 11.5 | 10.9 | 10.7 | 7.9 | 7.8 | 7.8 | 1.5 |
| **Total Payable Metals** | **US$M** | **$48.8** | **$4,391.4** | **$4,440.2** | **-** | **-** | **48.8** | **586.9** | **400.8** | **580.7** | **438.1** | **229.6** | **307.9** | **347.5** | **407.6** | **403.3** | **341.2** | **109.3** | **109.0** | **109.0** | **20.6** |
| **REFINING CHARGES** | | | | | | | | | | | | | | | | | | | | | |
| Dore Refining Charge | US$M | $0.1 | $7.8 | $7.9 | - | - | 0.1 | 1.1 | 0.9 | 1.1 | 0.8 | 0.5 | 0.6 | 0.5 | 0.5 | 0.5 | 0.5 | 0.3 | 0.3 | 0.3 | 0.0 |
| Total Shipping Costs | US$M | $0.2 | $15.7 | $15.9 | - | - | 0.2 | 2.3 | 1.8 | 2.2 | 1.6 | 1.0 | 1.2 | 1.0 | 1.0 | 1.0 | 0.9 | 0.6 | 0.5 | 0.5 | 0.1 |
| Audit Services | US$M | $0.0 | $0.7 | $0.7 | - | - | 0.0 | 0.1 | 0.1 | 0.1 | 0.1 | 0.0 | 0.0 | 0.1 | 0.1 | 0.1 | 0.1 | 0.0 | 0.0 | 0.0 | 0.0 |
| **Total Shipping & Refining Charges** | **US$M** | **$0.3** | **$24.2** | **$24.5** | **-** | **-** | **0.3** | **3.5** | **2.7** | **3.4** | **2.4** | **1.5** | **1.8** | **1.6** | **1.6** | **1.5** | **1.4** | **0.8** | **0.8** | **0.8** | **0.2** |
| **Net Smelter Return (before Royalties)** | **US$M** | **$48.5** | **$4,367.2** | **$4,415.7** | **-** | **-** | **48.5** | **583.3** | **398.1** | **577.3** | **435.6** | **228.1** | **306.1** | **345.9** | **406.0** | **401.8** | **339.8** | **108.5** | **108.2** | **108.2** | **20.4** |

| | | | | | | | | | | | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| **ROYALTIES** | | | | | | | | | | | | | | | | | | | | | |
| Third Party NSR | % | 1.05% | 1.05% | 1.05% | - | - | 1.05% | 1.05% | 1.05% | 1.05% | 1.05% | 1.05% | 1.05% | 1.05% | 1.05% | 1.05% | 1.05% | 1.05% | 1.05% | 1.05% | 1.05% |
| | US$M | $0.5 | $46.0 | $46.5 | - | - | 0.5 | 6.1 | 4.2 | 6.1 | 4.6 | 2.4 | 3.2 | 3.6 | 4.3 | 4.2 | 3.6 | 1.1 | 1.1 | 1.1 | 0.2 |
| Guatemalan Government Royalty | % | 1.00% | 1.00% | 1.00% | - | - | 1.00% | 1.00% | 1.00% | 1.00% | 1.00% | 1.00% | 1.00% | 1.00% | 1.00% | 1.00% | 1.00% | 1.00% | 1.00% | 1.00% | 1.00% |
| | US$M | $0.5 | $43.9 | $44.4 | - | - | 0.5 | 5.9 | 4.0 | 5.8 | 4.4 | 2.3 | 3.1 | 3.5 | 4.1 | 4.0 | 3.4 | 1.1 | 1.1 | 1.1 | 0.2 |
| **Total Royalties** | US$M | $1.0 | $89.9 | $90.9 | - | - | 1.0 | 12.0 | 8.2 | 11.9 | 9.0 | 4.7 | 6.3 | 7.1 | 8.4 | 8.3 | 7.0 | 2.2 | 2.2 | 2.2 | 0.4 |
| **Net Smelter Return (inc. Royalties)** | **US$M** | **$47.5** | **$4,277.3** | **$4,324.9** | **-** | **-** | **47.5** | **571.3** | **389.9** | **565.4** | **426.6** | **223.4** | **299.8** | **338.8** | **397.7** | **393.5** | **332.8** | **106.2** | **105.9** | **105.9** | **20.0** |
| **OPERATING COSTS** | | | | | | | | | | | | | | | | | | | | | |
| Mining | US$/t mined | | $10.42 | | - | - | - | $15.49 | $14.97 | $14.86 | $14.50 | $13.47 | $13.29 | $12.36 | $12.21 | $10.48 | $6.49 | $2.69 | $2.69 | $2.69 | $6.40 |
| | US$M | | $549.9 | | $0.0 | $0.0 | $0.0 | $62.0 | $59.9 | $59.6 | $58.0 | $53.9 | $53.1 | $49.6 | $48.8 | $41.9 | $26.0 | $10.8 | $10.8 | $10.8 | $4.8 |
| Processing | US$/t processed | | $12.97 | | - | - | - | $12.99 | $12.99 | $12.99 | $13.00 | $13.00 | $13.00 | $12.99 | $13.00 | $12.99 | $12.98 | $12.96 | $12.91 | $12.91 | $12.80 |
| | US$M | | $684.9 | | $0.0 | $0.0 | $0.0 | $52.0 | $52.0 | $52.1 | $52.0 | $52.0 | $52.0 | $52.1 | $52.0 | $52.0 | $51.9 | $52.0 | $51.6 | $51.6 | $9.7 |
| Site Services | US$/t processed | | $2.73 | | - | - | - | $1.81 | $1.91 | $2.04 | $3.23 | $3.45 | $3.48 | $3.69 | $3.72 | $3.75 | $3.18 | $1.73 | $1.74 | $1.83 | $2.75 |
| | US$M | | $144.4 | | $0.0 | $0.0 | $0.0 | $7.2 | $7.7 | $8.2 | $12.9 | $13.8 | $13.9 | $14.8 | $14.9 | $15.0 | $12.7 | $6.9 | $6.9 | $7.3 | $2.1 |
| General & Administration | US$/t processed | | $3.42 | | - | - | - | $4.04 | $3.79 | $3.60 | $3.60 | $3.55 | $3.54 | $3.50 | $3.51 | $3.46 | $3.34 | $2.91 | $2.86 | $2.78 | $3.46 |
| | US$M | | $180.6 | | $0.0 | $0.0 | $0.0 | $16.2 | $15.1 | $14.4 | $14.4 | $14.2 | $14.2 | $14.0 | $14.0 | $13.9 | $13.3 | $11.7 | $11.4 | $11.1 | $2.6 |
| **Total Operating Costs** | **US$/t processed** | | **$29.56** | | **-** | **-** | **-** | **$34.35** | **$33.67** | **$33.50** | **$34.34** | **$33.48** | **$33.31** | **$32.54** | **$32.45** | **$30.69** | **$25.98** | **$20.28** | **$20.20** | **$20.22** | **$25.41** |
| | **US$M** | | **$1,560.2** | | **$0.0** | **$0.0** | **$0.0** | **$137.4** | **$134.7** | **$134.4** | **$137.4** | **$133.9** | **$133.2** | **$130.5** | **$129.8** | **$122.8** | **$103.9** | **$81.4** | **$80.8** | **$80.9** | **$19.2** |
| **Net Operating Cash Flow** | **US$M** | **$47.5** | **$2,717.1** | **$2,764.7** | **$0.0** | **$0.0** | **$47.5** | **$433.9** | **$255.2** | **$431.0** | **$289.3** | **$89.5** | **$166.6** | **$208.3** | **$267.9** | **$270.8** | **$228.9** | **$24.9** | **$25.1** | **$25.1** | **$0.8** |
| *AISC (Net Credits)* | *US$/oz* | | *$629* | | | | | *$399* | *$609* | *$415* | *$547* | *$1,058* | *$748* | *$674* | *$538* | *$515* | *$518* | *$1,242* | *$1,233* | *$1,222* | *$3,057* |
| *Operating Margin* | *%* | | *62%* | | | | | *74%* | *64%* | *74%* | *66%* | *39%* | *54%* | *60%* | *66%* | *67%* | *67%* | *23%* | *23%* | *23%* | *4%* |
| **CAPITAL COSTS** | | | | | | | | | | | | | | | | | | | | | |
| **INITIAL & SUSTAINING** | | | | | | | | | | | | | | | | | | | | | |
| Infrastructure | US$M | 39.6 | 11.1 | 50.8 | 7.2 | 30.9 | 1.5 | 3.6 | 1.3 | 4.2 | 2.0 | 0.0 | - | - | - | - | - | - | - | - | - |
| Power & Electrical | US$M | 38.8 | - | 38.8 | 2.5 | 16.8 | 19.5 | - | - | - | - | - | - | - | - | - | - | - | - | - | - |
| Water Management | US$M | 52.0 | 39.9 | 91.9 | - | 43.2 | 8.8 | 4.3 | 12.2 | 10.8 | - | 6.1 | 0.9 | 5.7 | - | - | - | - | - | - | - |
| Surface Operations | US$M | 14.4 | - | 14.4 | - | 10.9 | 3.5 | - | - | - | - | - | - | - | - | - | - | - | - | - | - |
| Mining | US$M | 42.3 | 89.2 | 131.6 | - | 26.0 | 16.3 | 9.1 | 9.0 | 8.2 | 13.1 | 10.1 | 10.2 | 9.0 | 4.9 | 4.8 | 5.3 | 2.2 | 1.9 | 1.1 | 0.3 |
| Process Plant | US$M | 136.9 | - | 136.9 | 4.3 | 103.4 | 29.2 | - | - | - | - | - | - | - | - | - | - | - | - | - | - |
| Construction Indirects | US$M | 66.3 | - | 66.3 | 1.9 | 50.1 | 14.4 | - | - | - | - | - | - | - | - | - | - | - | - | - | - |
| Owner's Costs | US$M | 77.8 | - | 77.8 | 2.9 | 41.0 | 33.9 | - | - | - | - | - | - | - | - | - | - | - | - | - | - |
| Pre-Production Start-up, Commissioning | US$M | 90.2 | - | 90.2 | 0.8 | 20.3 | 69.0 | - | - | - | - | - | - | - | - | - | - | - | - | - | - |
| **Subtotal - Pre-Contingency** | **US$M** | **$558.3** | **$140.3** | **$698.7** | **$19.6** | **$342.6** | **$196.1** | **$17.0** | **$22.6** | **$23.2** | **$15.0** | **$16.3** | **$11.1** | **$14.7** | **$4.9** | **$4.8** | **$5.3** | **$2.2** | **$1.9** | **$1.1** | **$0.3** |
| Contingency | US$M | $60.7 | $0.0 | $60.7 | - | $31.3 | $29.3 | - | - | - | - | - | - | - | - | - | - | - | - | - | - |
| Contingency (% of Total Costs) | % | 11% | 0% | 9% | 0% | 9% | 15% | 0% | 0% | 0% | 0% | 0% | 0% | 0% | - | - | - | - | - | - | - |
| **Total Initial & Sustaining Capital** | **US$M** | **$619.0** | **$140.3** | **$759.4** | **$19.6** | **$374.0** | **$225.4** | **$17.0** | **$22.6** | **$23.2** | **$15.0** | **$16.3** | **$11.1** | **$14.7** | **$4.9** | **$4.8** | **$5.3** | **$2.2** | **$1.9** | **$1.1** | **$0.3** |
| **SALVAGE & CLOSURE** | | | | | | | | | | | | | | | | | | | | | |
| Closure Costs | US$M | **-** | **$34.8** | **$34.8** | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | 17.9 |
| Monitoring Costs | US$M | **-** | **$3.3** | **$3.3** | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - |
| **Total - Closure Costs** | **US$M** | **-** | **$38.1** | **$38.1** | **-** | **-** | **-** | **-** | **-** | **-** | **-** | **-** | **-** | **-** | **-** | **-** | **-** | **-** | **-** | **-** | **$17.9** |
| **TOTAL** | | | | | | | | | | | | | | | | | | | | | |
| **Total Capital Costs** | **US$M** | **$619.0** | **$178.4** | **$797.4** | **$19.6** | **$374.0** | **$225.4** | **$17.0** | **$22.6** | **$23.2** | **$15.0** | **$16.3** | **$11.1** | **$14.7** | **$4.9** | **$4.8** | **$5.3** | **$2.2** | **$1.9** | **$1.1** | **$18.2** |
| **WORKING CAPITAL** | | | | | | | | | | | | | | | | | | | | | |
| Changes in Working Capital | US$M | **$32.2** | **($32.2)** | **$0.0** | $2.5 | $7.5 | $22.2 | $24.4 | ($22.7) | $7.4 | ($5.9) | ($8.6) | $3.3 | $1.7 | $2.5 | ($0.0) | ($2.2) | ($9.2) | $0.0 | ($0.0) | ($22.9) |

| CASH FLOWS | | | | | | | | | | | | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| **Net Pre-Tax Free Cash Flow** | **US$M** | **($603.7)** | **$2,571.3** | **$1,967.3** | **($22.7)** | **($376.4)** | **($205.6)** | **$392.6** | **$255.3** | **$400.4** | **$280.1** | **$81.8** | **$152.2** | **$191.9** | **$260.5** | **$266.0** | **$225.7** | **$31.9** | **$23.2** | **$23.9** | **$5.5** |
| **Cumulative Pre-Tax Free Cash Flow** | **US$M** | | | | **($22.7)** | **($376.4)** | **($582.0)** | **($189.4)** | **$65.9** | **$466.3** | **$746.4** | **$828.2** | **$980.4** | **$1,172.3** | **$1,432.8** | **$1,698.8** | **$1,924.5** | **$1,956.5** | **$1,979.7** | **$2,003.6** | **$2,009.1** |
| VAT Payable | | | | $239.7 | $1.9 | $36.3 | $22.7 | $15.7 | $16.3 | $16.2 | $16.0 | $15.8 | $15.1 | $15.5 | $14.3 | $13.5 | $11.6 | $9.0 | $8.9 | $8.8 | $2.1 |
| Income Taxes | US$M | $3.4 | $282.8 | $286.3 | - | - | $3.4 | $41.1 | $28.1 | $40.6 | $30.7 | $6.3 | $21.6 | $24.3 | $28.5 | $28.2 | $23.9 | $3.4 | $2.4 | $2.6 | $1.1 |
| VAT Receivable | | | | $239.7 | - | - | - | - | $36.0 | $36.6 | $36.5 | $16.0 | $15.8 | $15.1 | $15.5 | $14.3 | $13.5 | $11.6 | $9.0 | $8.9 | $8.8 |
| **Net After-Tax Free Cash Flow** | **US$M** | **($668.4)** | **$2,349.8** | **$1,681.0** | **($23.6)** | **($412.7)** | **($231.8)** | **$335.8** | **$247.0** | **$380.3** | **$269.9** | **$75.7** | **$131.3** | **$167.2** | **$233.1** | **$238.6** | **$203.8** | **$31.1** | **$20.9** | **$21.4** | **$11.1** |
| **Cumulative After-Tax Free Cash Flow** | **US$M** | | | | **($23.6)** | **($436.3)** | **($668.0)** | **($332.3)** | **($85.3)** | **$295.0** | **$564.9** | **$640.6** | **$771.9** | **$939.1** | **$1,172.2** | **$1,410.8** | **$1,614.6** | **$1,645.7** | **$1,666.6** | **$1,688.0** | **$1,699.1** |

| ECONOMIC INDICATORS | | | | |
|---|---|---|---|---|
| **PRE-TAX** | | | | |
| **Pre-Tax IRR** | **%** | | | **37.3%** |
| **Pre-Tax Payback** | **Years** | | | **1.7** |
| **Pre-Tax NPV @ 0%** | **US$M** | | | **$1,967** |
| **Pre-Tax NPV @ 5%** | **US$M** | | | **$1,265** |
| **POST-TAX** | | | | |
| **After-Tax IRR** | **%** | | | **30.3%** |
| **After-Tax Payback** | **Years** | | | **2.2** |
| **After-Tax NPV @ 0%** | **US$M** | | | **$1,681** |
| **After-Tax NPV @ 5%** | **US$M** | | | **$1,047** |

# 23. ADJACENT PROPERTIES

There are no adjacent properties.

# 24. OTHER RELEVANT DATA & INFORMATION

## 24.1 Project Execution Plan

### 24.1.1 Introduction

The Project Execution Plan (PEP) describes the standard procedures that will be used during project execution. The procedures can be used by the project team in their daily project activities, from planning and initiation through execution and close-out. The information in the PEP helps the project team design and construct facilities that satisfy the Owner's functional requirements.

### 24.1.2 Preparation, Revision, & Control of the PEP

The Project Management Team is responsible for preparing the PEP in alignment with the Owner's required methods of project execution. The company's senior management is responsible for approving the PEP, and the project director is responsible for communicating its required use to project members. The project managers are responsible for executing and carrying through the PEP within their groups. The PEP is comprised of several sections and provides detailed information about the following:

- project organization
- roles and responsibilities
- practices and procedures
- reference systems.

### 24.1.3 Summary

The integrated project management team (IPMT) will be created to lead the execution of the Cerro Blanco Gold Project using a self-perform approach (Owner personnel and a contracted project management team). The plan is for the IPMT to lead the project execution and construction of all on-site infrastructure and the process plant. Mine development will also be self-performed by the Owner Mine Team. Off-site infrastructure, including the access road, bridge and powerline, will be built by a contractor under the supervision of the IPMT.

The project team reports into the project director and works in unison to achieve the project objectives through the efficient use of the Owners' equipment, material and personnel, and to minimize difficulties common to commissioning and start-up.

The IPMT shares the responsibility for planning and executing the Project. Schedule development begins at the management level schedule (highest) and drills down through the project/control levels. The management level schedule is used to establish work goals and overall time frames for the scope of work. The project schedule is a statement of project objectives. A Level 1 schedule (Figure 24-1) contains the least amount of detail but is sufficient for management to evaluate and track the main project milestones captured in the plan.

The management level schedule defines:

- established goals or milestones
- major elements of work to be performed, the duration of each, and their relationship to one another
- responsibility assignments for accomplishing project objectives.

Figure 24-2 shows the project workforce (full-time equivalent) profile during the pre-production phase of the Cerro Blanco Gold Project. This number represents all direct and indirect workers required in the mining, processing, surface operations, general services and construction departments, but does not include specialty contractors.

The project team will use a quality assurance / quality control (QA/QC) system in all phases of the Project (engineering, manufacturing of equipment, procurement, construction, commissioning and start-up). The system will include audits, certification, factory inspection, and destructive and non-destructive testing during construction. It will also provide traceability from origin for each component of the Project. The detailed procedures and practices will be developed with the project QA/QC team integrating the project document control process.

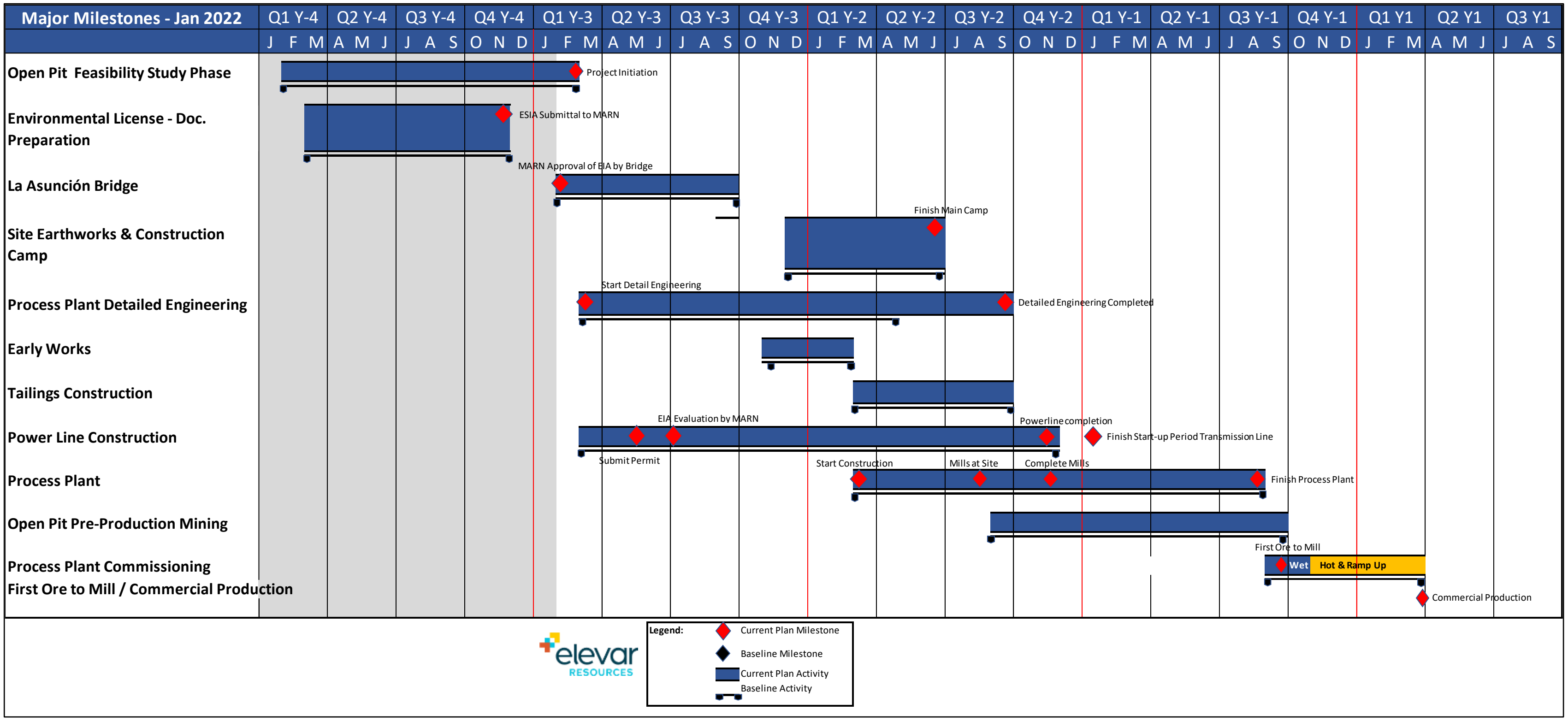
The operations team will be recruited during construction and will work with the IPMT during pre-commissioning, commissioning, start-up, and handover of the facilities. This will be a structured and planned process that will require strong coordination between the project personnel and operations team for a gradual transfer of responsibility.

All general service departments will be staffed with Owner employees to service the construction efforts. These departments will become fully operational early in the construction period with a trained workforce and established programs and service providers. As construction is demobilized, the service departments will continue seamlessly into operations.

The general service departments are as follows:

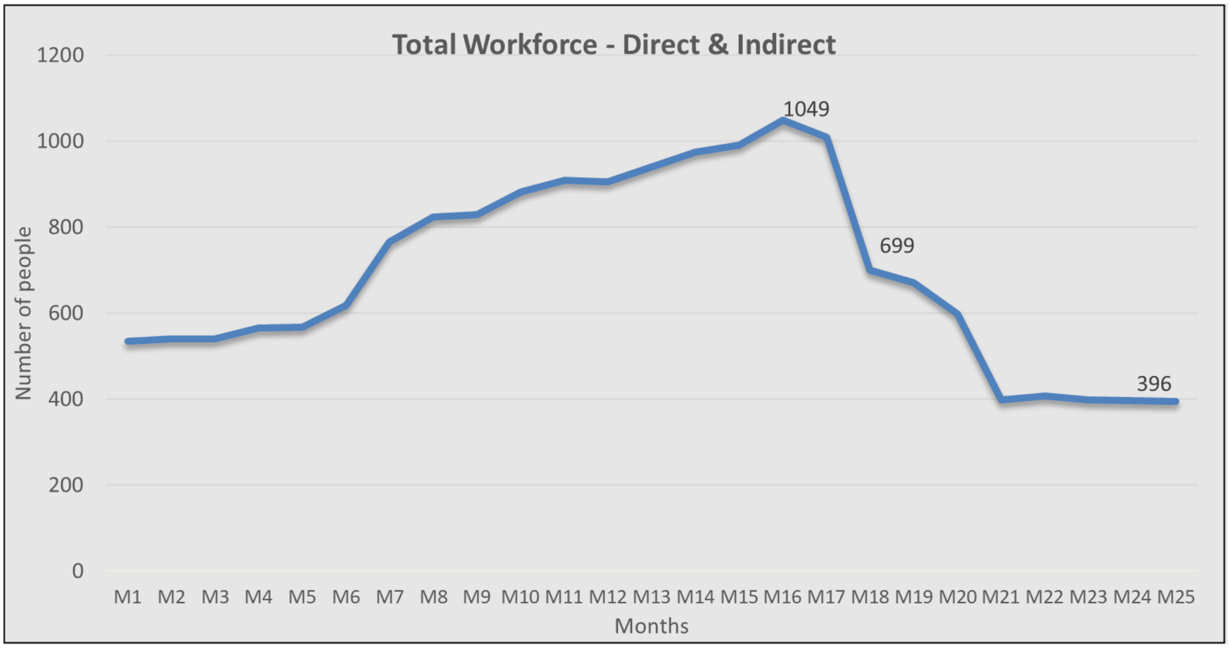
- Human Resources (HR)
- Information Technology (IT)
- Site Services
- Accounting
- General Administration
- Health and Safety (HS)
- Security
- Community Relations
- Environment
- Corporate Affairs
- Legal Services
- Finance
- Supply Chain Management (SCM).

**Figure 24-1: Cerro Blanco Gold Project Schedule – Level 1**




Source: GMS (2021).

**Figure 24-2: Construction Direct & Indirect Workers (Full-Time Equivalent)**




Source: GMS (2022).

The Owner mining team will consist of operations, maintenance and technical services and will be a critical component of the overall Owner project team.

The project team headed by the project director and general manager will provide overall site management during construction. The Owner operations team will take full control of the operation after commercial production is achieved. Based on the recommended ramp-up criteria, the Cerro Blanco Gold Project will reach its design capacity and gold recoveries six months after concluding the commissioning phase. The commissioning phase will last for two months at 20% of the design throughput. The ramp-up period will be carried out from months 20 through 25 (Table 24.1).

**Table 24.1: Cerro Blanco Ramp-up Criteria**

| Month | Period | % of Designed Throughput | Ore/Waste | Au Recovery % |
|---|---|---|---|---|
| -2 | Commissioning | 20 | Waste | |
| -1 | Commissioning | 20 | Waste | |
| 1 | Ramp-up | 40 | Low Grade | 50 |
| 2 | Ramp-up | 55 | Low Grade | 60 |
| 3 | Ramp-up | 70 | Low Grade | 70 |
| 4 | Ramp-up | 80 | Low Grade | 80 |
| 5 | Ramp-up | 90 | High Grade | 90* |
| 6 | Ramp-up | 100 | High Grade | 90 |
| 7 | Full Production | 100 | High Grade | 90 |
| 8 | Full Production | 100 | High Grade | 90 |
| 9 | Full Production | 100 | High Grade | 90 |

* The 90% recovery is referential. The exact gold recovery is being defined based on the recovery curves of the different ores fed to the plant and integrated into the production plan.

# 25. INTERPRETATIONS & CONCLUSIONS

## 25.1 Summary

The Cerro Blanco Gold Project has been optimized and updated as demonstrated by the results and findings provided in this Technical Report. This FS (Table 25.1) shows the results of the project for the current long-term gold price, exchange rate, final pit design and reserves, life-of-mine production schedule, process plant, infrastructure and support facilities design and execution.

This N.I. 43-101 Technical Report reconfirms the technical feasibility and economic viability of the Project based on an open pit mining operation with a life of mine production of 2.6 Moz of gold over an initial 14-year life of mine. It is recommended to advance the Project to the execution phase.

**Table 25.1: Summary of the Economic Metrics of the Cerro Blanco FS**

| Description | Value |
|---|---|
| Gold Price (Base Case) | $1,600/oz |
| Silver Price (Base Case) | $20.00/oz |
| Exchange Rate (Quetzal:USD) | 7.69:1 |
| Exchange Rate (CAD:USD) | 1.32:1 |
| Peak Annual Gold Production | 347,000 oz |
| Average Annual Gold Production (Years 1-4) | 297,000 oz |
| Average Annual Gold Production (LOM) | 197,000 oz |
| Total Gold Production (LOM) | 2,645,000 oz |
| Strip Ratio (Waste:Ore) | 2.7:1 |
| Average Gold Head Grade | 1.64 g/t |
| Average Silver Head Grade | 7.27 g/t |
| Average Gold Recovery | 93.0% |
| Average Silver Recovery | 84.3% |
| Nominal Plant Throughput | 4.0 Mt/a |
| Mine Life | 13.7 years |
| Operating Costs | Mining – $2.53/t mined; Processing – $12.97/t milled; Site Services – $2.73/t milled; G&A – $3.43/t milled |
| Total Operating Costs | $29.56/t milled |
| Cash Costs (LOM Net Credits) | $560/oz Au |
| All-in Sustaining Cash Costs (LOM net Credits)* | $629/oz Au |
| Initial Capital (Including Contingency) | $572 M |
| Sustaining Capital, including Closure Costs | $178 M |
| Average Annual After-Tax Free Cash Flow | $308 M per year (Years 1-4) |
| Total Production After-Tax Free Cash Flow | $2,350 M |
| $NPV_{5\%}$ (Pre-Tax) | $1,265 M |
| IRR (Pre-Tax) | 37% |
| $NPV_{5\%}$ (After-Tax) | $1,047 M (base case) |
| IRR (After-Tax) | 30% (base case) |

* All in Sustaining Cash Costs (net credits) = (operating costs + offsite costs + royalties + sustaining and closure capital – value of payable silver ounces) / payable gold ounces

## 25.2 Geology & Mineral Resources

The mineral resource has a footprint of 800 x 400 m between elevations of 525 and 200 masl. The mineral resource estimate is the result of 141,969 m of drilling by Bluestone and previous operators (totalling 1,256 drill holes and channel samples). The 3.4 km of underground infrastructure allowed for underground mapping, sampling, and over 30,000 m of underground drilling that enhanced the current understanding and validation of the Cerro Blanco geological model. The mineral resource estimate is based on a scenario that considers open pit mining methods and therefore requires improved and refined geological models of the lithological units. These broad mineralized lithologies are host to the high-grade veins that have been the focus of the potential underground mining scenario. The resulting domain models and estimation strategy were designed to accurately represent the grade distribution.

The mineral resource estimate for the Cerro Blanco deposit is reported at a base case above a 0.4 g/t Au cut-off, as tabulated in Table 25.2.

**Table 25.2: Mineral Resource Statement**

| Resource Category | Tonnes (kt) | Au Grade (g/t) | Ag Grade (g/t) | Contained Gold (koz) | Contained Silver (koz) |
|---|---|---|---|---|---|
| Measured | 40,947 | 1.8 | 7.9 | 2,382 | 10,387 |
| Indicated | 22,595 | 1 | 4.2 | 706 | 3,058 |
| Measured & Indicated | **63,542** | **1.5** | **6.6** | **3,089** | **13,445** |
| Inferred | 1,672 | 0.6 | 2.1 | 31 | 112 |
| Below Pit (Indicated)* | 189 | 5.7 | 13.4 | 35 | 82 |
| Stockpile (Measured) | 30 | 5.4 | 22.6 | 5 | 22 |

Notes: The mineral resource statement is subject to the following: 1. All mineral resources have been estimated in accordance with Canadian Institute of Mining and Metallurgy and Petroleum (CIM) definitions, as required under National Instrument 43-101 (N.I. 43-101), with an effective date of June 20, 2021. 2. Mineral resources reported demonstrate reasonable prospect of eventual economic extraction, as required under N.I. 43-101; mineral resources are not mineral reserves and do not have demonstrated economic viability. 3. Cut-off grades are based on a price of US$1,600/oz gold, US$20/oz silver and a number of operating cost and recovery assumptions, plus a contingency. 4. Numbers are rounded. 5. The mineral resources may be affected by subsequent assessment of mining, environmental, processing, permitting, taxation, socio-economic and other factors. 6. An inferred mineral resource has a lower level of confidence than that applying to an indicated mineral resource and must not be converted to a mineral reserve. It is reasonably expected that the majority of inferred mineral resources could be upgraded to indicated mineral resources with continued exploration. 7. *Resources identified below the PEA pit shell that are amenable to underground mining (3.5 g/t cut off). 8. Mineral resources are inclusive of mineral reserves. Source: Kirkham (2021).

## 25.3 Mineral Reserves

The mineral reserves are based on measured and indicated mineral resources, and do not include any inferred mineral resources. Inferred mineral resources contained within the mine design are classified as waste.

Table 25.3 shows that proven and probable mineral reserves are inclusive of mining dilution and ore loss. The total ore tonnage before dilution and ore loss is estimated at 50.1 Mt at an average grade of 1.73 g/t Au and 7.57 g/t Ag. The dilution envelope around the remaining ore blocks results in a dilution tonnage of

3 Mt. The dilution tonnage represents 6% of the ore tonnage before dilution and the dilution grade is estimated from the block model and corresponds to the average grade of the dilution skin.

**Table 25.3: Mineral Reserves Statement**

| Mineral Reserves | Tonnage | Au | | Ag | |
|---|---|---|---|---|---|
| | kt | g/t | koz | g/t | koz |
| Proven | 37,618 | 1.89 | 2,286 | 8.33 | 10,084 |
| Probable | 16,279 | 1.07 | 560 | 4.81 | 2,518 |
| **Proven + Probable** | **53,896** | **1.64** | **2,846** | **7.27** | **12,602** |

Notes: 1. CIM definitions were followed for mineral reserves. 2. Effective date of the estimate is Nov 1, 2021. 3. The Qualified Person for the mill feed estimate is Mathieu Gignac, P. Eng. of G Mining Services Inc. 3. The cut-off grade for mill feed material was estimated using a $1,550/oz gold price, $20/oz silver price and a cut-off grade of 0.5 g/t Au Eq. Other costs and factors used for gold cut-off grade determination were process, G&A, and other costs of $21.17/t, a royalty of $31.6 /oz Au and a gold metallurgical recovery of 91%, and a silver metallurgical recovery of 85%. 4. Bulk density of mineralized material is variable but averages 2.6 t/m$^3$. 5. The average strip ratio is 2.7:1. 6. Tonnages are rounded to the nearest 1,000 tonnes, metal grades are rounded to two decimal places. 7. Tonnage and grade measurements are in metric units; contained gold and silver are reported as thousands of troy ounces 8. The mineral reserves may be affected by subsequent assessment of mining, environmental, processing, permitting, taxation, socio-economic and other factors. 9. Mineral resources are inclusive of mineral reserves. 10. Rounding as required by reporting guidelines and may result in summation differences.

## 25.4 Mining

Mineral reserve estimation was performed by Bluestone Resources and G-Mining Services. Optimization runs were carried out only using measured and indicated mineral resources to define the optimal mining limits. Inferred mineral resources were considered as waste.

The mine design and mineral reserve estimate have been completed to an FS level with the pit designs following a design criterion recommended from pit stability geotechnical study carried out by E-Mining Technologies.

The proven and probable ore reserve estimate at a cut-off grade of 0.50 g/t Au Eq. is as follows:

- Proven mineral reserve: 37,618 kt at 1.89 g Au/t for 2,286 koz of gold in situ
- Probable mineral reserve: 16,279 kt at 1.07 g Au/t for 560 koz of gold in situ
- Total proven and probable reserve: 53,896 kt at 1.64 g Au/t for 2,846 koz of gold in situ.

The mineral reserves incorporate appropriate mining dilution and mining recovery estimations. The average mining dilution is 6.7% at a grade of 0.22 g/t Au with minimal isolated ore blocks resulting in negligeable ore losses.

Mining activities will occur over a period of ~10.5 years, including one year and four months of pre-production activities. The mining rate reaches a steady state of 21 Mt/a in Year 1 of commercial production. The life-of-mine plan includes a total tonnage of 199.3 Mt with 145.4 Mt of waste for a strip ratio of 2.70 (W:O). The mine consists of a main open pit that will be developed in seven phases, and a satellite pit located at the north of the main pit that will be mined at the end of the life of mine.

Mining is planned as a conventional open pit operation using 15 $m^3$ hydraulic excavators and 90 tonne class haul trucks. A bulk mining approach is well suited for the massive ore body, with mining taking place on 10 m high benches. The mine is planned as an Owner mining operation with blasting activities outsourced.

## 25.5 Metallurgical Testing and Mineral Processing

Drill core and coarse rejects of various ore lithologies and waste rock were used for metallurgical testwork. Various composites and blends representing initial mining periods were also tested. Metallurgical and vendor testwork to date has confirmed the flowsheet and equipment selection and sizes.

The process plant design is based on a conventional leach-CIP metallurgical flowsheet to treat gold and silver-bearing ore to produce doré. The flowsheet is based on metallurgical testwork, industry standards, vendor recommendations and conventional unit operations used in the mineral processing industry.

The flowsheet includes a comminution circuit (primary crusher, SAG and ball mill) to produce a primary grind size of $P_{80}$ of 75 µm (design of $P_{80}$ of 53 µm), pre-leach thickener, 2-hour atmospheric pre-oxidation, 36-hour cyanide leach circuit, carbon adsorption via a CIP carousel circuit, 12.5 tonne elution and carbon handling circuit, electrowinning and gold room, and tailings dewatering circuit (thickener and pressure filters) to produce a filtered tailings product (16% to 18% moisture).

Leach extraction results from the metallurgical testwork were used to develop gold and silver recovery curves for the FS and have been estimated overall at 93% for gold and 84% for silver.

## 25.6 Infrastructure

Multiple design reviews of the building locations and type of infrastructures took place during the FS phase to adapt land acquisition and reflect market availability.

Access road design reflects the latest land agreements and topography. A new permanent camp has been introduced from the 2021 preliminary economical assessment (PEA) and is now part of the capital cost estimate. The dry stack tailing facility (DSTF) and water management strategies were updated. The explosives storage area location has been relocated to match the new mine plan and to meet safety distance criteria when blasting. The mine service area is now oriented to fit with topography and facilitate circulation. Three bays of the mine maintenance facility are included in the sustaining capital cost estimate to align with operation requirements.

The electrical network both on and off site has been designed with the latest infrastructure and process equipment updates. Power will be provided at 138 kV from the Jutiapa substation, 38 km west of the project site. The voltage for the site network will be 13.8 kV.

A workshop on “reliability, availability, maintainability, buildability and operability” (RAMBO) took place during the FS to confirm the process plant layout.

## 25.7 Capital and Operating Costs

The initial capital cost is estimated to be $571.5 million net of recoverable taxes and credits; this includes a contingency of $60.7 million, which represents 11% of total expenditures, as well as $47.5 million in pre-production revenue for a five-month ramp-up period after commissioning is completed.

Sustaining costs over the life of mine are estimated at $140.3 million for all costs related to the acquisition, replacement, or major overhaul of assets during the mine life. This estimate does not include contingency.

Closure costs are estimated at $38.1 million. Reclamation and monitoring activities are planned for Years 14 through 19.

The average life-of-mine operating cost is $629/oz of gold or $29.56/t milled. The average life-of-mine all-in sustaining cost (AISC) is $709/oz of gold.

## 25.8 Economic Analysis

The base case economic analysis using a gold price of $1,600/oz has an after-tax NPV 5% of $1,047 million, an IRR of 30.3% and payback period of 2.2 years after the start of production.

## 25.9 Risks

The most significant project risks are summarized below:

- Permitting – While the project is fully permitted and has an approved EIA in place, environmental permit amendments are required for some of the proposed modifications, including the change to an open pit operation, increase in processing rate and discharge flow to the environment. New EIAs and permits are also needed for the powerline, access road and bridge. Although several of these amendments have been submitted, potential delays in approval of permit amendments and/or requirements for new permits could adversely affect the planned schedule.
- Powerline Rights of Way – Until all rights of way have been acquired, this issue remains a risk to the timely completion of the powerline. Negotiations for the powerline rights of way are set to begin in Q1 2022.
- Power Availability – The availability of power from the national grid and ability of Energuate/RECSA/Guatemel and the transmission line contractor IEGSA to deliver it on time need to be confirmed.
- Power Cost – The cost of grid power is based on a market projection and not a power supply agreement. A higher power cost would result in increased operating costs.
- Site Land Negotiations – There is a potential for schedule delays while finalizing negotiated purchases of land and/or rights-of-way for areas impacted by the expanded footprint of the operation.
- COVID-19 – The pandemic may impact the project schedule and cost (restrictions to get to site, availability of personnel, permitting process, procurement delays, equipment costs).

- Commodity Prices (Gold, Silver) – Lower commodity prices will change the size and grade of the potential targets. Conversely, increased commodity prices will improve economics and resources.
- Groundwater – It has been assumed that groundwater dewatering would be achieved using horizontal drains. Pit inflows and deep dewatering wells, flows have been estimated based on an understanding of the groundwater pathways gained during the 2019 Feasibility Study. Further testwork and modelling is required to better understand the groundwater flows and optimize the dewatering approach. Increases in the actual amount of groundwater encountered would impact development costs and/or the schedule. Drilling for drainage, and operational definition drilling included in the mine plan, will help to identify specific water-bearing zones with higher-than-expected flows to establish control and/or management procedures. Initiating well development earlier in the mine life should allow time to better understand mine dewatering.
- Project Schedule – Project components are tightly coupled within the schedule; many must be completed without delay to maintain the desired development targets.
- Inflation – The current market conditions are experiencing abnormal inflation. If the trend continues costs may be increased beyond budgetary predictions.
- Local Procurement – The obligation to purchase in-country may inflate project costs.
- Socio-Political Risk – There is a potential risk of socio-political opposition in the local communities as well as at the national level, which could adversely impact the project schedule.

## 25.10 Opportunities

The main opportunities identified for the project are listed below:

- Mineral Resources – There is the potential for an increase in mineral resources with increased exploration drilling along trend of the deposit.
- Grade Control – As with most vein deposits, effective grade control and proper mining execution to maintain minimal unplanned dilution will minimize potential impacts on grade, throughput, and operating costs. RC definition drilling ahead of blasting will improve the definition of grade boundaries between high-grade veins and low-grade disseminated ore. In combination with geological mapping and sampling at the bench, definition drilling will help minimize unplanned dilution and negative impacts during mining.
- Acid Rock Drainage and Metal Leaching: Developing a waste rock facility design and PAG material handling plan could reduce the acidity and concentration of metals released from the waste rock, thereby reducing treatment costs and environmental footprint. The benefits of such a waste rock facility design should be evaluated.
- Cyanide Destruction – The current process uses the addition of sodium metabisulphite as a source of $SO_2$ to destruction cyanide. This process results in high concentrations of sulphate in the process effluents, which creates the need for a costly sulphate removal system prior to discharge to the environment. It is recommended to investigate alternative cyanide destruction options to minimize the introduction of sulphate in the effluent and eliminate a costly removal system.

- Nitrogen Treatment – The current process assumes a costly nitrogen treatment; a trade-off study should be conducted to examine passive treatment through wetlands.
- Administrative Services – Relocating administrative services to Asuncion Mita would reduce the number of offices required on site.
- Infrastructures – The next phase will allow the project to explore more local procurement opportunities which may reduce costs and help build relationships.

# 26. RECOMMENDATIONS

At a gold price assumption of $1,600/oz, the results of this FS demonstrate that the Cerro Blanco Gold Project warrants advancement due to its positive economics and strong cash flows.

Additional engineering work and mine planning, as well as continuous engagement with local stakeholders, have all improved the understanding of the project and reduced its risk profile.

The path forward for the project is to obtain all the required permit amendments and advance detailed engineering for an efficient project construction mobilization.

## 26.1 Exploration, Geology & Resources

Additional drilling may increase resources and improve understanding and modelling of lithological units. Definition drilling ahead of blasting will improve the definition of grade boundaries between high-grade veins and low-grade disseminated ore, and help minimize unplanned dilution.

A comprehensive brownfields exploration program along trend of the main deposit is recommended to explore for additional gold and silver resources that could potentially extend the project's life.

## 26.2 Metallurgy

Metallurgical testwork to date has confirmed the flowsheet and equipment selection. Additional tests may be required to finalize equipment sizing as follows:

- Caro's acid tests for cyanide destruction as a potential lower cost alternative to the current $SO_2$/air system
- vendor oxygen sparging tests to optimize oxygen injection into pre-oxidation, leaching and cyanide destruction, and finalize the oxygen plant size
- tailings filter cake flowability tests for bin and material handling design.

## 26.3 Processing

It is recommended to complete the following tasks in additional to detailed design activities:

- Implement flowsheet changes as a result of additional test work noted in Section 26.2.
- complete a tailings filter cake handling trade-off study (stockpile vs. bin distribution) to optimize the existing design concepts.

## 26.4 Mining

The following work is recommended in the next study phase:

- proceed to vendor selection for mobile equipment and explosives
- pursue further optimization of open pit stage sequencing to maximize the financial return of the asset.

## 26.5 Infrastructure

The powerline right-of-way should be negotiated, and the incoming powerline route and cost should be confirmed during the next study phase.

## 26.6 Geotechnical

A geotechnical campaign is planned to complete the DSTF detailed design

## 26.7 Water Management

To support the design of the water management infrastructure, the following work is recommended in the next study phase:

- complete the ongoing geochemistry testwork on the waste rock and update the water quality predictions based on the larger dataset
- update the water balance model to reflect the latest mine plan and optimize the pumping capacity vs. storage capacity
- optimize the WTP design and phased approach based on the results of the updated water balance and water quality models
- conduct a long-duration pump test on a deep dewatering well to verify the assumptions of the hydrogeological model and adjust the dewatering plan as required.

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