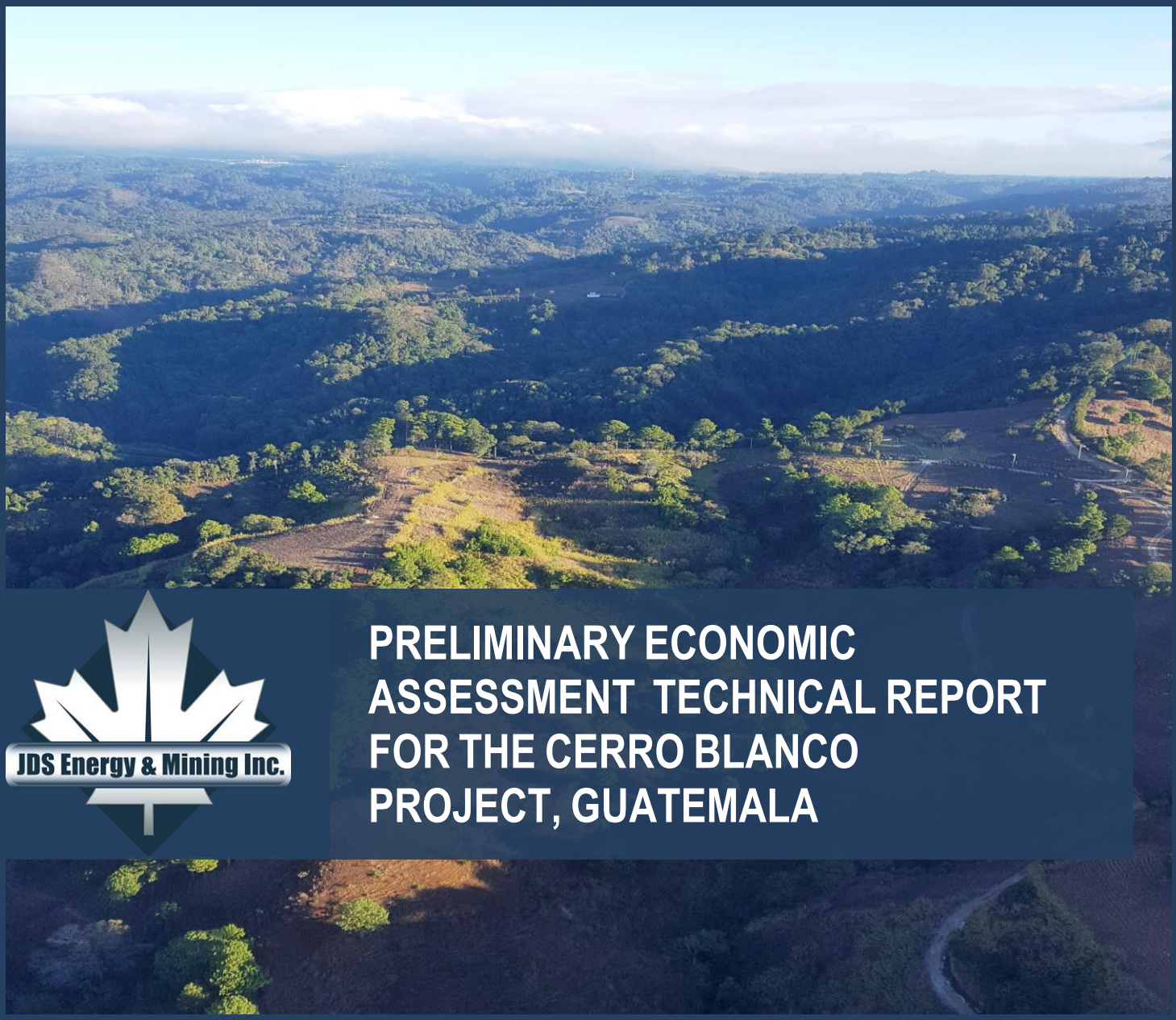

PRELIMINARY ECONOMIC ASSESSMENT TECHNICAL REPORT FOR THE CERRO BLANCO PROJECT, GUATEMALA

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NOTICE

JDS Energy & Mining, Inc. prepared this National Instrument 43-101 Technical Report, in accordance with Form 43-101F1, for Bluestone Resources Inc. The quality of information, conclusions and estimates contained herein is based on: (i) information available at the time of preparation; (ii) data supplied by outside sources, and (iii) the assumptions, conditions, and qualifications set forth in this report.

Bluestone Resources Inc. filed this Technical Report with the Canadian Securities Regulatory Authorities pursuant to provincial securities legislation. Except for the purposes legislated under provincial securities law, any other use of this report by any third party is at that party's sole risk.

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1 Executive Summary

1.1 Introduction

This preliminary economic assessment (PEA) was produced for Bluestone Resource Inc. (Bluestone) or the 'Company' by JDS Energy & Mining Inc. (JDS) with the assistance of Kirkham Geosystems Ltd., Knight Piésold Ltd. (KP), SRK Consulting (U.S.) Inc. (SRK), and Stantec Consulting Ltd.

On January 4, 2017, Bluestone entered into an agreement with Goldcorp Inc. (Goldcorp) to acquire 100% of Minerales Entre Mares de Guatemala, S.A. (Entre Mares), Goldcorp's indirect wholly-owned Guatemalan subsidiary which holds a 100% interest in the Cerro Blanco gold Project (Cerro Blanco or the Project). On successful closure of the deal, Entre Mares shall be a wholly-owned subsidiary of Bluestone, a Canadian company headquartered in Vancouver, British Columbia.

The purpose of this study is to document a new Mineral Resource Estimate and results of preliminary designs and economics of the Project. The structure and content of this report uses National Instrument 43-101, Standards of Disclosure for Mineral Projects (NI 43-101) guidelines.

This PEA is preliminary in nature, it includes inferred mineral resources (3% of mine plan tonnes) that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves, and, as such, there is no certainty that the PEA results will be realized. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

1.2 Project Description

PEA Concept:

Mining:	1,250 t/d UG longhole and drift & fill
Processing:	Crushing-Grinding-Leach-Merrill Crowe
LOM Mill Feed:	4 Mt @ 8.1 g/t Au, 28.0 g/t Ag
Plant Recovery:	91.3% Au, 87.8% Ag
Avg Payable Au:	105,000 oz/yr
Mine Life:	9 years

The Cerro Blanco Project is a gold development Project that, since 2006, has had over US$170 M in investment for the construction of:

- Two underground (UG) declines from surface;
- 3.2 km of lateral underground development;
- Four ventilation raises;
- 17 dewatering wells; and
- Significant surface infrastructure including shop, warehouse, office and water treatment facility.

The Project will be developed as an underground mining operation but also has the potential to be mined as an open pit. The Project is envisioned to be a 1,250 tonne per day (t/d), ramp access, underground gold mine using longhole stoping and drift and fill mining methods.

The mine will extend to a depth of approximately 360 m below surface, or 210m above mean sea level (amsl). The mine experiences significant hot water inflow that must be drawn down prior to mining. In addition to the existing surface dewatering wells, a series of new dewatering wells are planned to draw down the water table around the deposit, in advance of the mine plan. A portion of the mine water (1,500 US gallons per minute (g/m)) will be cooled, treated, and discharged under the existing permit allowances, and the balance (minimum of 2,500 g/m) will be disposed of through a series of new reinjection well in accordance with permit requirements.

The processing plant will utilize crushing, grinding, leaching and zinc precipitation (Merrill Crowe) processes to recover gold and silver as doré. Approximately 105,000 oz of gold will be recovered annually, with higher value zones being mined early in the Project life. The mine life will be about nine years based on current defined resources.

Tailings will be filtered and deposited in an engineered dry stack tailings facility (DSTF). A portion of the tailings and all waste rock will be deposited back underground as paste backfill and cemented rock fill (CRF).

The site is currently accessible with an all-weather gravel road and has 4 MW of diesel-generated power on-site. A new 5.5 km long access road and power transmission line will be installed prior to start-up.

1.3 Property Description

The Project is located in southeast Guatemala approximately 160 km by road from the capital, Guatemala City. It is located in in the Jutiapa District, approximately 9 km west of the border with El Salvador. The nearest town to the Project is Asunción Mita, a community of about 25,000 people and approximately 5 km from the Project. The Project covers a concession which totals 15.25 km^2 and lies entirely in the municipality of Asunción Mita, in the Jutiapa District.

Figure 1.1: Property Location

Source: Goldcorp (2014)

1.4 Ownership

Minerales Entre Mares de Guatemala, S.A. (Entre Mares) is currently a wholly-owned subsidiary of Goldcorp. Bluestone entered into an agreement dated January 4, 2017 (the "Agreement") with Goldcorp to acquire 100% of Entre Mares, Goldcorp's indirect wholly-owned Guatemalan subsidiary which holds a 100% interest in Cerro Blanco. Bluestone and Goldcorp intend to execute a formal definitive agreement within 45 days of signing the Agreement, upon which Entre Mares will become Bluestone's indirect wholly-owned Guatemalan subsidiary.

1.5 History, Exploration and Drilling

As of February 2012, Goldcorp had drilled approximately 95 holes for a total of 21,727 m on the Cerro Blanco property, since the acquisition from Glamis Gold in November 2006. Historical drilling totals 522 holes and 155,557 m for exploration purposes. Table 1.1 provides a summary of the historical drilling on the property.

Table 1.1: Summary of Drilling at Cerro Blanco

Year	Company	Holes Drilled	Metres
1998	Mar-West	9	1,340
1999	Glamis/Mar-West	48	7,217
2000	Glamis Gold	18	3,525
2001	Glamis Gold	0	0
2002	Glamis Gold	23	6,525
2003	Glamis Gold	0	0
2004	Glamis Gold	46	9,659
2005	Glamis Gold	120	29,065
2006	Glamis Gold	67	15,129
2007	Goldcorp	47	12,375
2008-2009	Goldcorp	0	0
2010	Goldcorp	9	9
2011	Goldcorp	39	9,352
2012	Goldcorp	96	21,370
Total		**522**	**115,557**

Source: Goldcorp (2014)

1.6 Geology & Mineralization

The Cerro Blanco district consists of localized precious metal veins and small stockwork deposits which occur over an area of more than 20 km in diameter. The Cerro Blanco gold deposit occurs within the largest mineralized area in a 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.

The current gold resource occurs under a small hill and is confined within a region of about 400 m by 800 m. The gold deposit is characterized by both high angle and low angle banded chalcedony veins, locally with calcite replacement textures.

High angle mineralized faults and discontinuous stockwork zones host some of the highest gold grades. 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.

1.7 Metallurgical Testing and Mineral Processing

Metallurgical test work was conducted on samples from the Cerro Blanco deposit between April 1999 and January 2012 by Kappes, Cassiday & Associates (KCA). JDS reviewed the metallurgical testing procedures and results for the purposes of this PEA.

The test work consisted of agitated cyanide leach tests and gravity/leach tests on mineralized material and rotary drill hole samples. Additional testing in the 2011 test program, reported by KCA, included:

- Mineralogical Study by Carson GeoMin;
- Grindability and abrasion index determinations by Phillip Enterprises;

- Solid-liquid separation tests by Pocock Industries; and
- Cyanide detoxification study by CyPlus.

The test work data that was specifically analyzed by JDS was for samples with gold grades in the range of 5 to 20 g/t; the test work that was considered included the lower and upper limits of mined mineralized material to be delivered from underground to the process plant. Table 1.2 below lists the estimated recoveries and processing factors.

Major conclusions from the review of the test programs were:

- A moderately fine grind will be required to maximize the extraction of both gold and silver. A grind size of 80% passing (P_{80}) 75 µm was selected for design purposes based on the various bottle roll tests;
- Gravity concentration and flotation tests as pre-concentration stages gave low metal recoveries, compared to "whole ore leaching" (note: "whole ore leaching" refers to a metallurgical process and does not imply the confirmed presence of ore);
- The leach time required is 96 hours; and
- The estimated gold extraction will be 92.4% and the estimated silver extraction is 88.9% in the leaching circuit. The final recoveries, taking into account solution losses, are estimated to be 91.3% and 87.8% for gold and silver, respectively.

Table 1.2: Estimated Metallurgical Recoveries and Mineral Processing Factors

Description	Units	Value
Au recovery	%	91.3
Ag recovery	%	87.8
Bond Work Index	**kWh/t**	**19.9**

Source: JDS (2017)

1.8 Mineral Resource Estimate

The PEA is based on an Indicated and Inferred Mineral Resource estimate undertaken by Garth Kirkham, P.Geo., of Kirkham Geosystems Ltd., a qualified person as defined by NI 43-101 and independent of the Company.

The Cerro Blanco deposit comprises several steeply dipping zones extending a strike length of more than 1,200 m and to a depth of more than 450 m. Solids were created based on mineralized intersections to create numerous zones in two sets orienting at azimuths of 110° and 140° which are used to constrain the interpolation of gold and silver grades.

The updated Mineral Resource Estimate incorporates more than 517 drill holes totalling 114,992 m. More than 1.24 Moz of gold and 4.47 Moz of silver are contained in the Indicated Mineral resource category. Cerro Blanco also contains more than 0.04 Moz of gold and 0.03 Moz of silver in the Inferred Mineral resource category.

The base case estimate in Table 1.3 below provides a summary of the mineral resource statement for Cerro Blanco.

Table 1.3: Mineral Resources by Classification

Class	Tonnes	Au (g/t)	Ag (g/t)	Au Ounces	Ag Ounces
Indicated	3,731,000	10.22	36.5	1,242,000	4,468,000
Inferred	243,000	6.02	4.2	47,000	33,000

Source: Kirkham Geosystems Ltd. (2017)

Notes:

1. CIM definitions were followed for Mineral Resources.
2. Effective date: February 7, 2017. All Mineral Resources have been estimated in accordance with Canadian Institute of Mining and Metallurgy and Petroleum (“CIM”) definitions, as required under NI 43-101.
3. Mineral resources reported demonstrate reasonable prospect of eventual economic extraction, as required under NI 43-101. Mineral resources are not Mineral Reserves and do not have demonstrated economic viability.
4. Underground Mineral Resources are reported at a cut-off grade of 3.75 g/t Au. Cut-off grades are based on a price of US$1,250/oz gold, US$16/oz silver and a number of operating cost and recovery assumptions, including a reasonable contingency factor.
5. Numbers are rounded.
6. The Mineral Resources may be affected by subsequent assessment of mining, environmental, processing, permitting, taxation, socio-economic and other factors.

1.8.1 Mineral Resource Estimation Methodology

The Mineral Resources were estimated using commercial mine-modelling and geostatistical software by the independent qualified person. The deposit was segregated into multiple estimation domains based on geologic models for each of the mineralized veins. Gold and silver block grades were estimated from grade limited composited samples in multiple passes. The Mineral Resources were estimated using inverse distance to the third power interpolation. Search ellipse anisotropy and orientation were based on the orientation of the vein solids models. Specific gravities were assigned to individual rock types and assigned on a block by block basis.

1.9 Mining

Cerro Blanco is proposed to be mined as an underground operation using a combination of longhole stoping (LH) and drift and fill (DF) mining methods with cemented paste and rock backfill to reach a target production rate of 1,250 t/d over a mine life of nine years that will extract 4.0 Mt of mineralized material. Long hole (LH) stoping will account for about 88% of total production, and the remaining 12% will come from DF and development. The Cerro Blanco deposit will be accessed from surface via a series of ramps, and all mineralized material and waste rock will be trucked out of the mine. One additional ventilation raise will be required in addition to the four existing ventilation raises. The underground ramps and ventilation raises will be necessary to circulate the required amount of air through the Cerro Blanco underground workings.

Dewatering, ventilation, and cooling are important aspects mine design at Cerro Blanco. The water table in the immediate mine area will be lowered by a series of existing and new surface dewatering wells to be maintained below the active mining levels. Any remaining water underground will be captured and pumped to surface through collection at underground sumps. The quantity of air moving through the mine will greatly exceed the ventilation requirements to ensure proper ventilation for the reduction of diesel particulate matter. The excess air will be used to reduce the exposed rock temperature, decrease the refrigerated cooling requirement, and increase worker comfort. In-situ rock temperatures range currently from 100 to 120°C in non-dewatered areas but the rock is quickly cooled to 30 – 50^{o}C upon dewatering and ventilation. Portable spot coolers will be installed in active production and development areas where workers will be outside of heavy equipment and at the face. The portable spot coolers will decrease the air temperature to 28°C.

Indicated and Inferred Mineral Resources were included in the mine design and schedule optimization process. Inferred Resources are considered too speculative geologically to have economic considerations applied to them to be categorized as Mineral Reserves, and there is no certainty that the Inferred Resources would be upgraded to a higher resource category. The PEA LOM plan contains approximately 96.6% Indicated and 3.4% Inferred Resources. No Mineral Resources have been classified in the Measured category.

The mine production schedule is shown in Table 1.4.

Table 1.4: Mine Production Schedule

Parameter	Unit	Totals	Year									
			-1	1	2	3	4	5	6	7	8	9
Mill Feed Material	kt	**3,981**	14	456	456	456	456	456	456	456	456	317
Mining/milling Rate	t/d	**1,249**	38	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,243
Diluted Au Grade	g/t	**8.14**	9.89	10.29	10.75	9.49	10.54	7.65	6.39	6.67	5.78	4.6
Contained Au Ounces	koz	**1,042**	4	151	158	139	155	112	94	98	85	47
Diluted Ag Grade	g/t	**27.95**	39	45.93	42.6	37.46	14.84	19.9	18.14	32.28	11.3	29.11
Contained Ag Ounces	koz	**3,577**	17	674	625	550	218	292	266	474	166	297
Lateral Waste Dev't	m	**12,357**	1,730	1,941	1,615	1,746	798	982	1,192	678	1,372	302
Lateral Mineralized Dev't	m	**14,492**	427	1,155	1,239	1,333	2,188	2,028	1,836	2,303	1,130	853
Total Development	m	**26,849**	2,157	3,096	2,854	3,079	2,986	3,010	3,028	2,981	2,502	1,155
Total Dev't Rate	m/day	**7.6**	5.91	8.48	7.82	8.44	8.18	8.25	8.3	8.17	6.86	4.53
Vertical Development	m	**630**	311	68	26	157	-	39	29	-	-	-
Waste Placed as Backfill	kt	**1,292**	5	134	133	156	152	160	151	166	141	94
Paste Fill	km^3	**1,633**	8	200	200	174	180	177	188	172	193	141

Source: JDS (2017)

1.10 Recovery Methods

The processing plant will have a capacity of 1,250 t/d. The plant will consist of the following unit operations:

- 3-stage crushing;
- Primary and secondary grinding;
- Cyanide leaching and counter current decantation (CCD) thickening;
- Cyanide destruction;
- Merrill Crowe and gold refining to doré; and
- Tailings filtration and disposal in the surface tailings facility or underground as paste backfill.

The leach circuit will have a retention time of 96 hours and sodium cyanide (NaCN) consumption is expected to be 1.85 kg/t.

The results of the metallurgical test work, together with financial evaluation data, were used to develop metallurgical design criteria, which in turn were used to design the process facility for the Project. From the test work, it can be concluded that the Cerro Blanco mineralized material can be treated using conventional mineral processing techniques for the recovery of gold and silver.

The grinding circuit product size is targeted at approximately P_{80} of 75 µm. The crushing circuit will operate at an availability of 65% while the milling, leaching and gold recovery circuits will operate 24 hours per day, 365 d/y at an availability of 92% processing 1,250 t/d.

1.11 Dewatering

Dewatering of the Cerro Blanco mine is an important factor in the mine development, because of the high temperature groundwater and thermal gradients. The average static groundwater-level elevation in the Project area is approximately 460 m above mean sea level (amsl). Due to dewatering activities using surface wells and underground sumps the water level has dropped to about 440 masl currently This level is being maintained using three surface wells and the underground sumps with a combined rate of about 600 g/m. New groundwater pumping wells will be required in order to dewater the planned mine to lowest proposed mining elevation of 210 masl called for in the mine plan.

Historical estimates for mine dewatering have ranged from 2,000 g/m to as high as 10,000 g/m for various mine plans and have advanced as more data becomes available to better document the interconnections of the various fault systems to the regional flow system. A revised mine plan has recently been developed by JDS that results in the northern area of the mine reaching a level of 210 masl within five years of mine start-up. In the southern area, the planned mine level is 270 masl, which is also attained within five years. The change in planned mining depth reduces the overall dewatering demands at Cerro Blanco and has allowed focused dewatering strategies to be developed.

Initial estimates of dewatering discharge rates to meet the needs of the current mine plan indicate that a minimum cumulative discharge of 4,000 g/m (3,500 g/m from surface wells and 500 g/m from mine sumps) will allow the dewatering to be advanced ahead of the mine development. This volume does not consider any contact water that will be managed at the mine site. The number of wells required to achieve the desired dewatering will include 10 of the 17 existing wells and 14 new dewatering wells, pumping an average of approximately 145 g/m each. Higher dewatering rates may be required to achieve the 210 masl dewatering level within the required time frame depending on aquifer storage and regional connection through fault zones. These details are not known at this time and require further assessment and analysis to better confirm the ultimate dewatering requirements. The 3500 g/m dewatering rate is based on overcoming the existing storage within the aquifer and targeted dewatering along fault zones that provide connection to the regional flow system. If regional influence is significant, dewatering rates could be over twice the focused dewatering rates of 3500 g/m and would require more wells to achieve dewatering. The increase in dewatering rates would have an effect on the need for additional injection wells and/or water treatment capacity based on the anticipated water quality. This will need to be further evaluated in the next study phase.

The estimated depths of the new dewatering wells are 450 m per dewatering well, requiring approximately 6,300 m of drilling. In order to monitor and manage the dewatering program, a series of 10 vibrating wire piezometers will be installed, to track changes in groundwater levels and subsurface temperatures at the mine site. The piezometers will be drilled with a depth comparable to the dewatering wells (i.e. 450 m each). Approximately 1,500 g/m of the extracted water will be treated and discharged to nearby surface drainages, while the remaining minimum of 2,500 g/m will be cooled and injected into the near surface aquifer system. A total of 10 sites for reinjection wells have been identified, with each reinjection well drilled to a depth of approximately 150 m (1,500 m total drilling). Water quality, especially high iron content in the groundwater discharged to the injection wells could cause well fouling and present a future issue with the ability to inject water efficiently. A more detailed

evaluation of water quality and suitability of the water to meet injection requirements and regulatory approvals should be completed during the next study phase.

The implementation of new reinjection wells is covered by the Cerro Blanco existing permits. The PEA has a provision for one additional injection well, in order to give the flexibility of re-injecting seepage and runoff water, collected from the DSTF facility during rain events.

1.12 Infrastructure

The PEA plans on the installation of the following key infrastructure elements to support the mine and process facilities:

- A new 5.5 km long access road that will tie into the Pan-American Highway (CA1), approximately 3 km north of Asunción Mita. A new 80 m long bridge will be installed over the El Achotal River. This road will also provide year-round access to the small community of El Cerron which currently does not have sufficient road access;
- A 7.5 km 69 kV overhead power line and on-site substation;
- 3 MW emergency generator power plant (existing1.5 MW new, 1.5 MW existing);
- Upgrades to the existing bulk storage tanks for diesel;
- New process and fresh/fire water;
- Upgrades to existing facilities, including the truck shop, bio-digester, dewatering cooling ponds and storage warehouse; and
- Upgrades to the North and South portal haul roads.

With respect to domestic/sanitary wastewaters, the mine site is already equipped with a biological wastewater treatment unit and septic field, which will be expanded prior to the start of mining operations.

Since the mine plan uses all of the waste rock as CRF or aggregate in paste backfill, there is no requirement for surface storage of waste rock and no allowance is included for this in the PEA.

A concept level design for a filtered DSTF was completed. The facility is to be located near the process plant and was designed for on surface storage of a portion of the tailings, using filtered (dry stack) tailings placement techniques. The remainder of tailings will be used for underground paste backfill. Design features include foundation preparation, foundation groundwater drainage, starter confining embankment and structural outer shell and a water management pond to collect runoff and seepage. Ditches will be constructed around the perimeter of the DSTF to divert the upper catchment areas and minimize the direct contact water catchment area. The facility will be shaped and capped at closure to prevent erosion of the tailings surface, while blending the structure into the surrounding topography.

1.13 Environment and Permitting

The due diligence review conducted as part of this PEA shows that Cerro Blanco has all the necessary permits to proceed with production, subject to operations adhering to the conditions of the existing permits. The PEA review showed that the proposed new access road is not currently covered by the existing permits; an EIA will have to be prepared and submitted to the local authority.

The PEA due diligence review shows that Cerro Blanco was maintained under Goldcorp's Corporate Social Responsibility (CSR) and their environmental standards. Cerro Blanco is supporting a comprehensive Permits Register and its environmental management and reporting is based on Goldcorp's "Sustainability Excellence Management System".

With respect to the mine closure, the Permits Register shows that, under Guatemalan regulation, the closure plan shall be presented to the Authorities three years in advance of the mine ceasing operation. Accordingly, Cerro Blanco has not yet filed a closure plan which is yet to be prepared should the study move to the next phase.

Other specific environmental aspects (waste rock, tailings, water management, and closure) are addressed in Sections 17 and 20 of this report.

1.14 Operating and Capital Cost Estimates

1.14.1 Capital Cost Estimate

The capital cost estimate was compiled using a combination of quotations, database costs and factors. The estimate is derived from engineers, contractors, and suppliers who have provided similar services to existing operations and have demonstrated success in executing the plans set forth in the study.

The summary capital cost estimate is shown in Table 1.5. The sustaining capital is carried over operating Years 1 through 9, and closure costs are projected from Year 10 to Year 15, predominantly to accommodate longer term water quality testing.

Given that assumptions have been made due to a lack of available engineering information, the accuracy of the estimate and/or ultimate construction costs arising from the engineering work cannot be guaranteed. The target accuracy of the estimate is -20/+30%.

Table 1.5: Summary of Capital Cost Estimate

Capital Costs	Pre-Production M$	Sustaining/ Closure M$	Total M$
Mining	40.8	80.5	121.3
Site Development	5.6	0	5.6
Primary Crushing	4.2	0	4.2
Process Plant	31.5	0	31.5
Tailings Management	4.5	4.6	9.2
On-Site Infrastructure	8.9	0.1	9.0
Off-Site Infrastructure	12.5	9.3	21.8
Project Indirects	15.2	0	15.2
EPCM	12.2	0	12.2
Owner's Costs	14.3	0	14.3
Subtotal	**149.8**	**94.5**	**244.2**
Contingency	21.0	10.8	31.8
Closure	0	6.9	6.9
Total Capital Costs	**170.8**	**112.2**	**283.0**

Source: JDS (2017)

1.14.2 Operating Cost Estimate

The operating cost estimate (OPEX) is based on a combination of experience, reference projects, budgetary quotes and factors as appropriate with a PEA study.

Preparation of the OPEX is based on the JDS philosophy that emphasizes accuracy over contingency and utilizes defined and proven Project execution strategies.

The operating cost estimate in this study includes the costs to mine, process the mineralized material to produce doré, infrastructure, and general and administrative expenses (G&A). These items total the operating costs and are summarized in Table 1.6. The target accuracy of the operating cost is -25/+30%. No allowance for inflation or contingency has been applied.

The operating cost estimate is broken into three major sections:

- Underground Mining;
- Processing; and
- General & Administrative.

The total operating unit cost is estimated to be $92.58/t processed. Average annual, total LOM and unit operating cost estimates are summarized in Table 1.6.

Table 1.6: Summary of Operating Cost Estimate

Operating Costs	$/t milled	LOM $M
Mining	49.06	195
Processing	26.26	105
G&A	17.26	69
Total	**92.58**	**369**

Source: JDS (2017)

The main OPEX component assumptions are outlined in Table 1.7, and Table 1.8 lists the OPEX parameters used.

Table 1.7 Main OPEX Component Assumptions

Item	Unit	Value
Electrical power cost	$/kWh	0.10
Average power consumption	MW	11.2
Overall power consumption (all facilities)	kWh/t processed	215
Diesel cost (delivered)	$/litre	2.50
LOM average manpower (including contractors, excluding corporate)	employees	300

Source: JDS (2017)

Table 1.8: PEA Operating Parameters

Parameter	Unit	Value
Resource Processed	Mt	4.0
Mill Average Daily Production	t	1,250
Mill Average Annual Production	Mt	0.46
Average Gold Mill Grade	g/t	8.14
Average Silver Mill Grade	g/t	27.95
Gold Contained	koz	1,042
Silver Contained	koz	3,577
Gold Produced	koz	952
Silver Produced	koz	3,141
Gold Recovery	%	91.3
Silver Recovery	%	87.8
Average Gold Production Years 1 and 2	koz/year	141
Average Silver Production Years 1 and 2	koz/year	570
Initial Capital Cost (including average 14% contingency, excluding working capital)	US$M	170.8
Sustaining Capital Cost (including average 11% contingency, excluding working capital)	US$M	105.4
Life of Mine Capital (including average 13% contingency)	US$M	276.1

Source: JDS (2017)

1.15 Economic Analysis

This PEA is preliminary in nature, it includes inferred mineral resources (3% of mine plan tonnes) that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves, and, as such, there is no certainty that the PEA results will be realized. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

1.15.1 Main Assumptions

Table 1.9 outlines the terms used in the economic analysis.

All monetary values are nominal 2017 US dollars, unless otherwise stated.

Table 1.9: Economic Assumptions

Off-site Costs and Payables	Item	Unit	Estimated Value
Payables for Doré	Gold	%	99.8
	Silver	%	99.0
Doré Refining Costs	Gold	US$/payable oz	5.00
	Silver	US$/payable oz	0.50
Transportation Costs		US$/payable Au oz	3.73
Guatemalan Government Royalty		% NSR	1.0

Off-site Costs and Payables	Item	Unit	Estimated Value
Goldcorp Royalty		% NSR	1.0

Source: JDS (2017)

Table 1.10 outlines the metal prices and FX rates used in the economic analysis.

Table 1.10 Net Smelter Return Assumptions

Assumptions	Unit	Value
Au Price	US$/oz	1,250
Ag Price	US$/oz	16.00
FX Rate	GTQ:US$	7.55
	C$:US$	1.28

Source: JDS (2017)

1.15.2 Results

The economic results for the Project, based on the assumptions outlined in Section 1.14.1 are outlined in Table 1.11.

Table 1.11: Economic Results

Parameter	Unit	Pre-Tax Results	After-Tax Results
$NPV_{0\%}$	US$M	551	440
$NPV_{5\%}$	US$M	405	317
IRR	%	53	44
Payback Period	Production years	1.6	1.8

Source: JDS (2017)

1.15.3 Sensitivities

A simplistic sensitivity analysis was performed to determine which factors most affect the Project economics and is discussed in Section 23. Each variable evaluated was tested using the same sensitivity values, although some may be more likely to experience significantly more fluctuation in value over the LOM (i.e. CAPEX versus metal prices). The confidence attributed to each variable in this study does not factor into the sensitivity analysis, the inter-correlation between certain variables, and for this reason is considered a simplistic approach to determine which variable would most affect the economic results of the Project.

Sensitivity analyses were performed on metal prices, mill head grade, CAPEX, and OPEX as variables. The value of each variable was changed plus and minus 25% independently while all other variables were held constant. The Project shows the most sensitivity to metal price, grades and recovery. The results of the sensitivity analyses are shown in Table 1.12.

Table 1.12: Sensitivities Analyses

Variable	Pre-Tax $NPV_{5\%}$ (M$)			After-Tax $NPV_{5\%}$ (M$)		
	-25% Variance	0% Variance	25% Variance	-25% Variance	0% Variance	25% Variance
Metal Price/Head Grade	168	405	641	118	317	479
Recovery	170	405	640	119	317	N/A
OPEX	475	405	335	380	317	254
CAPEX	468	405	341	380	317	254

Source: JDS (2017)

1.16 Project Development

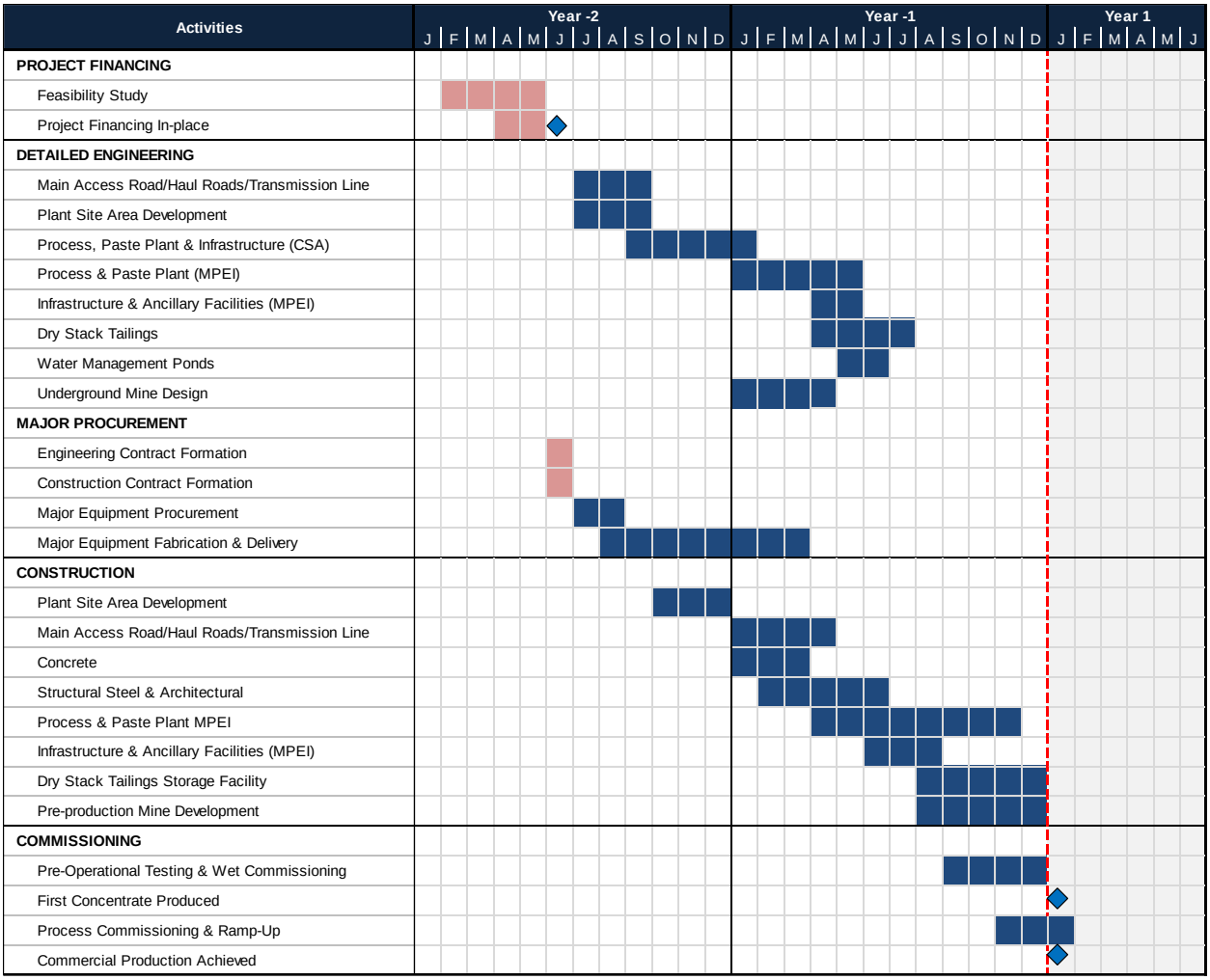
The overall construction and commissioning period for the Project is estimated to be approximately 18 months from start of site development to first gold pour. New access road and powerline permitting and all Project engineering, will take place in Year -2 of the Project. All other work will take place in late Year -2 and Year -1. For additional details, refer to the preliminary development schedule shown in Figure 1.2.

During Year -1, the underground mine design will be completed, and underground development will commence. The construction of new dewatering and reinjection wells will be staged as per the following to defer capital:

- Year -1: six dewatering wells, six reinjection wells;
- Year 1: six dewatering wells, five reinjection wells; and
- Year 2: two dewatering well.

Permanent mine surface infrastructure will also be installed during Year -1, while underground development will be ongoing. Commissioning of the process plant will take place in the latter half of Year -1 as construction activities ramp down.

Figure 1.2: Preliminary Schedule



Source: JDS (2017)

1.17 Conclusions

Based on the assumptions used in the PEA, the Cerro Blanco Project indicates a rapid payback, high margin, underground mining Project with robust economics in the current gold price environment. Industry-standard mining and processing methods were used in the study and the QPs are not aware of any fatal flaws that encumber the Project from undergoing further economic studies, financing and ultimately development.

JDS recommends that Bluestone proceed with a feasibility stage engineering study upon the closing of the proposed acquisition.

This PEA is preliminary in nature, it includes inferred mineral resources (3% of mine plan tonnes) that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves, and, as such, there is no certainty that the PEA results will be realized. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

1.17.1 Risks

It is the conclusion of the QPs that the PEA summarized in this technical report contains adequate detail and information to support the potentially positive economic results. The PEA proposes the use of industry-standard equipment and operating practices. To date, the QPs are not aware of any fatal flaws for the Project.

The most significant potential risks associated with the Project consists of the hot water management that will be encountered in the mine dewatering effort and socio-political resistance to the development of the planned mine in Guatemala. The latter is a common risk to most mining projects and can be mitigated, at least to some degree, with adequate planning and pro-active management The risk associated with water management is not entirely unknown due to the presence of existing dewatering wells and continued dewatering, treatment and discharge of underground water.

1.17.2 Opportunities

The main opportunities identified for the Project are listed below:

- Utilization of existing assets at the Marlin mine – as part of the Agreement with Goldcorp, Bluestone has the right of first refusal on Goldcorp's nearby Marlin mine in Guatemala, which is ceasing operations in the first quarter of 2017;
- Sourcing of used equipment;
- Evaluation of open pit scenarios;
- Ore sorting to improve feed grade to plant;
- Optimization of mine plan;
- Potential growth of resource estimate though additional exploration;
- Use of coarser processing grind size to reduce the grinding power draw; and
- Implementation of power generation in the cooling of the mine water.

1.17.3 Recommendations

JDS recommends in proceeding directly to a Feasibility Study and initiating key early works as follows:

- Kick-off on-site hydrogeology test work to gain a better understanding of the ground fault mapping and ground water model to refine dewatering requirements and injectability of excess water, both in terms of quantity and quality;
- Commence the paste plant test work;
- Carry our geotechnical investigations and testing for the proposed DSTF and underground mine;
- Source used equipment for mine and processing plant;
- Carry out evaluation of assets at Marlin mine; and
- Proceed immediately with preparing the EIA and permitting for the new access road, and the permitting for the new overhead power line, as required.

2 Introduction

2.1 Basis of Technical Report

Bluestone Resources Inc. (Bluestone) commissioned JDS Energy & Mining Inc. (JDS) to complete a preliminary economic assessment (PEA) for the Cerro Blanco Project (Cerro Blanco or the Project) located in the South of Guatemala, Jutiapa District, near the town of Asunción Mita.

On January 4, 2017, Bluestone entered into an agreement (the Agreement) with Goldcorp Inc. (Goldcorp) to acquire 100% of the Cerro Blanco Gold Project (Cerro Blanco) and Mita Geothermal Project (Mita Geothermal) in Guatemala. This report summarizes the results of the Cerro Blanco PEA. The Mita Geothermal Project is not considered in this report but was part of the overall acquisition from Goldcorp.

The purpose of this study is to document an updated Mineral Resource Estimate, mine design and preliminary economics of the Cerro Blanco Project. Goldcorp had previously completed an internal study on the Project in 2013 and updated the results in 2014. These previous studies did not comply with National Instrument 43-101 guidelines; however, in preparation of the PEA, any relevant information was reviewed and reused where deemed appropriate by the QPs.

The structure and content of this report uses National Instrument 43-101, Standards of Disclosure for Mineral Projects (NI 43-101) guidelines.

This report summarizes the work carried out by several consultants and the scope of work for each company is listed below, and combined, makes up the total Project scope.

JDS Energy & Mining Inc. (JDS) scope of work included:

- Compiling the technical report including information provided by other consulting companies;
- Establishing an economic framework for the PEA;
- Mine engineering, design and scheduling;
- Developing a conceptual flowsheet, specifications and selection of process equipment;
- Designing required site infrastructure, identifying proper sites, plant facilities and other ancillary facilities;
- Estimating mining, process plant and G&A OPEX and CAPEX for the Project;
- Establishing gold and silver recovery values for doré production on-site;
- Evaluating status of current permits;
- Preparing a financial model and conducting an economic evaluation including sensitivity and Project risk analyses; and
- Interpreting the results and making conclusions that lead to recommendations to improve Project value and reduce risks.

Knight Piésold Ltd. (KP) scope of work included:

- Dry stack tailings facility (DSTF) design and tailings management;

SRK Consulting (US) Inc. (SRK)

- Geotechnical recommendations for underground mine design;

Kirkham Geosystems Ltd. (Kirkham)

- Mineral Resource Estimate.

Stantec Consulting Inc. (Stantec)

- Dewatering and Injection.

2.2 Qualifications and Responsibilities

The results of this PEA 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 respective QPs. The QPs are being paid a fee for their work in accordance with normal professional consulting practice.

The following individuals, by virtue of their education, experience and professional association, are considered QPs as defined in the NI 43-101, and are members in good standing of appropriate professional institutions/associations. The QPs are responsible for the specific report sections as follows in Table 2.1.

Table 2.1: QP Responsibilities

QP	Company	QP Responsibility/Role	Site Visit	Report Section(s)
Maz Mohaseb, P.Eng.	JDS	Project Management, Environmental/ Permitting, CAPEX, OPEX, Economic Analysis	May 18th, 2017	1-5, 17.4.14, 18.17,18.19, 19-27
Michael Makarenko, P.Eng.	JDS	Mining Methods	Feb.2nd, 2017	15,16 (except 16.3 & 16.4)
Robert Gutowski, P.Eng.	JDS	Infrastructure	No site visit	18 (except 18.17, 18.19 & 18.20)
Kelly McLeod, P. Eng.	JDS	Metallurgical Recoveries/ Recovery Method	No site visit	13,17
Garth Kirkham, P.Geo.	Kirkham	Mineral Resource Estimate	May 8th, 2017	6-12, 14
Ken Embree, P.Eng.	KP	Tailings Management	Feb.2nd, 2017	18.20
Michael Levy, P.E.	SRK	Geotechnical	Feb.2nd, 2017	16.3
Craig Johnston, P.Geo.	Stantec	Dewatering and Injection	No site visit	16.4

Source: JDS (2017)

In accordance with NI 43-101 guidelines, QPs Michael Makarenko, Ken Embree and Mike Levy visited the Cerro Blanco Project on February 2nd, 2017, Garth Kirkham visited the site on May 8th, 2017 and Maz Mohaseb visited the site on May 18th, 2017. The other QPs relied upon the observations of the QP personnel that visited the site.

2.3 Units, Currency and Rounding

The units of measure used in this report are as per the International System of Units (SI) or “metric” except for Imperial units that are commonly used in industry (e.g., ounces (oz.) and pounds (lb.) for the mass of precious and base metals, US gallons per minute (g/m) for water flow rates).

All dollar figures quoted in this report refer to US dollars (US$ or $) unless otherwise noted.

Frequently used abbreviations and acronyms can be found in Section 29.

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

2.4 Sources of Information

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

- Discussions with Goldcorp'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 Goldcorp;
- Previous studies completed by Goldcorp; and
- Additional information from public domain sources.

3 Reliance on Other Experts

The QPs opinions contained herein are based on information provided by Bluestone Resources and others throughout the course of the study. The QPs have taken reasonable measures to confirm information provided by others and take responsibility for the information.

Non-QP specialists relied upon for specific advice are listed below along with extent of involvement, dates when input was provided and specific sections of the report the input applies to.

- Raul Recinos – Goldcorp's Environmental Manager at Cerro Blanco;
    - Mr. Recinos provided a current copy of the Cerro Blanco Permit Register and details on the existing Water Treatment Plant and site Waste Management Plan on January 16th and 17th, 2017
- Marcel Pineau, P.Eng., Ph.D. – Permitting & Environmental Manager at JDS Energy & Mining;
    - Mr. Pineau reviewed and validated the information provided Mr. Recinos, and provided input to sections 4.5, 4.6, 17.4.7, 18.5.1, 18.5.2 and 20 of this report between January and March, 2017
- Hhan Olsen, P.G. – Principal Hydrogeologist at Stantec Consulting Services Inc.;
    - Mr. Olsen has been involved with the Cerro Blanco site since supervising onsite drilling of dewatering wells in March of 2013 and completed the dewatering rate estimates. He provided input to section 16.4 of this report between February and March, 2017
- Indi Gopinathan, P.Eng., CPA, CMA – Certified Professional Accountant at JDS Energy & Mining.
    - Ms. Gopinathan provided input to the cash flow model related to Guatemalan taxation requirements summarized in section 23.3 of this report in March, 2017

The QPs used their experience to determine if the information from previous reports was suitable for inclusion in this technical report and adjusted information that required amending. The QPs take responsibility for the work provided by the experts listed above.

The QPs did not review any licenses, permits, work contracts, or perform an independent verification of land title and tenure. The QPs have not verified the legality of any underlying agreement(s) that may exist concerning the licenses or other agreement(s), such as royalty agreements, between third parties.

4 Property Description and Location

4.1 Location

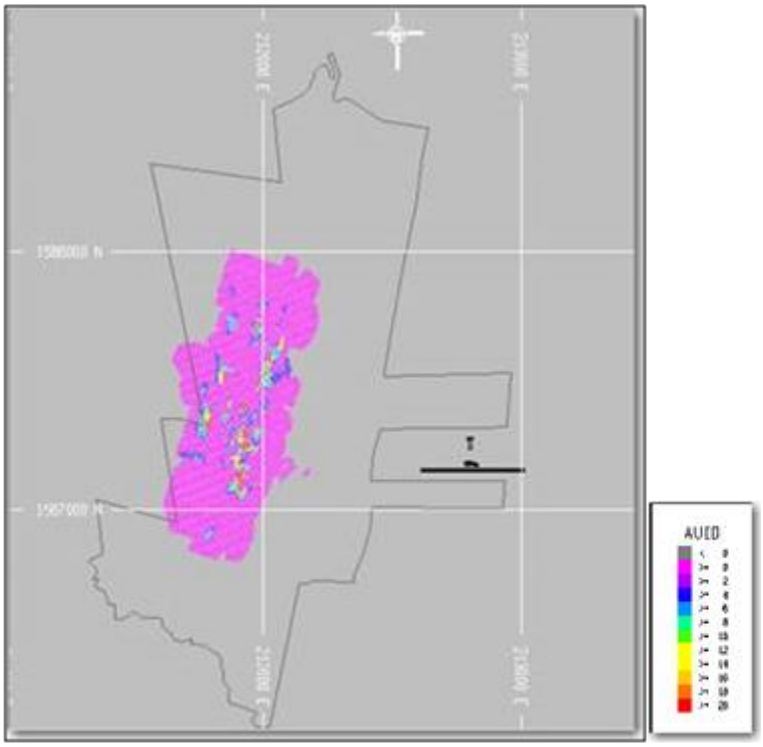
The Project is in southeast Guatemala approximately 160 km by road from the capital, Guatemala City, as indicated in Figure 4.1. It is located in in the Jutiapa District, approximately 9 km west of the border with El Salvador. The nearest town to the Project is Asunción Mita, a community of about 25,000 people located approximately 5 km from the Project. The Project covers a concession which totals 15.25 km^2 and lies entirely in the municipality of Asunción Mita, in the Jutiapa District.

Figure 4.1: Project Location Map showing country

Source: Goldcorp (2014)

The location of the Mineral Resources relative to the property boundary is shown in Figure 4.2. A very small portion of the low grade resource (below cut-off grade) and existing development fall outside of the property boundary.

Figure 4.2: Location of Mineral Resources Relative to Tenements and Property Boundary



Source: Source: Kirkham Geosytems (2017)

4.2 Mineral Tenure

4.2.1 Underlying Agreements Specific to the Resource and Land Tenure

In Guatemala, natural resources are property of the state. Consequently, mining exploitation only requires the resolution of the Ministry of Mining and the Ministry of the Environment before completing all legal issues. Land tenure in Guatemala is regulated by the constitutional law which states that all citizens have the right to obtain land as private property. When the mining companies have a specific target for mining exploration or exploitation, they pay for the land rights by legal agreements under the civil law and the constitutional law.

4.2.2 Mineral Rights

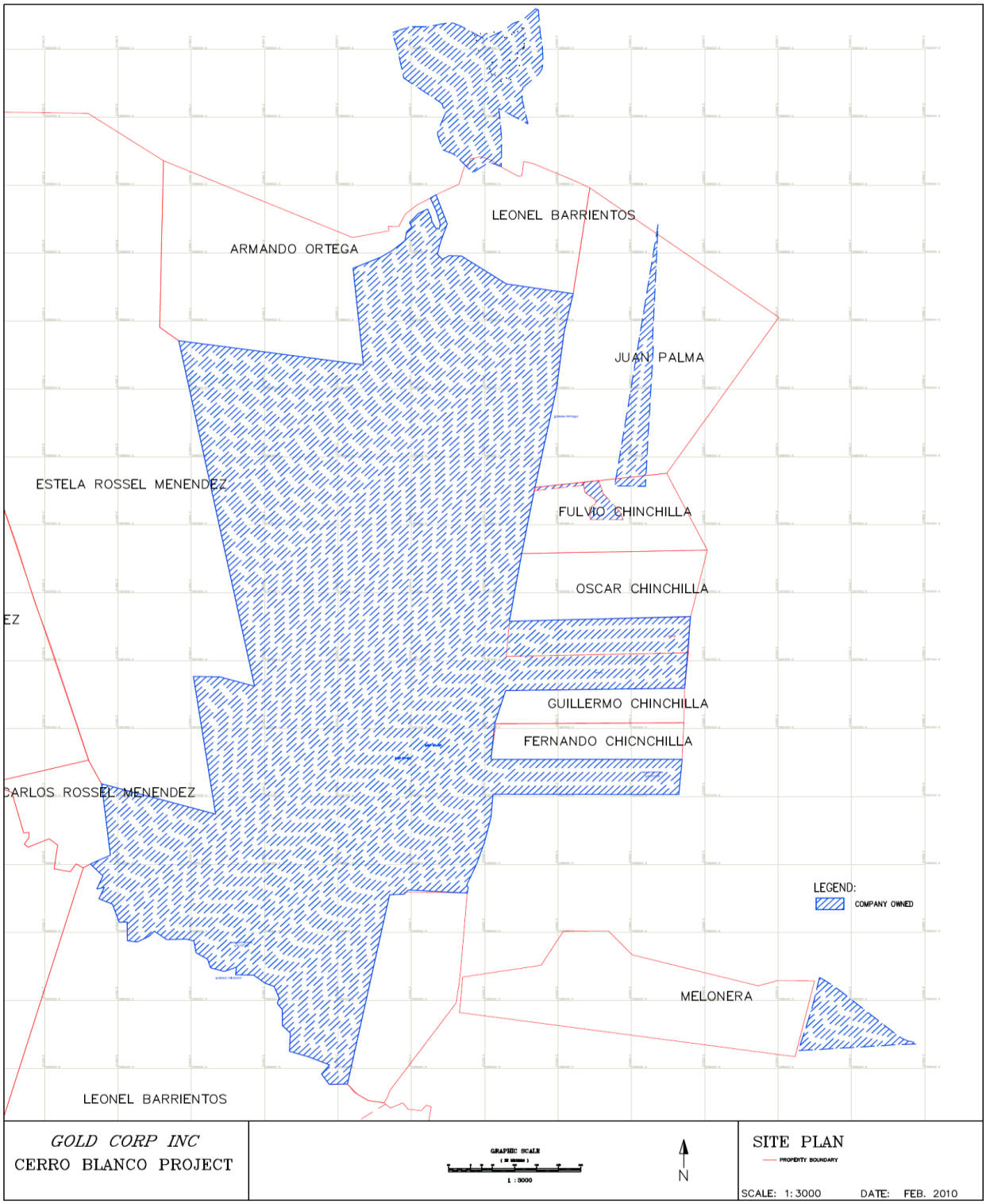
Guatemalan mining law provides for three types of licenses for reconnaissance, exploration and exploitation. A reconnaissance license has a six month term with the possibility of renewal for an additional six month period. The reconnaissance license can cover an area from 500 to 3,000 km^2 and, upon application, can be converted to an exploration license. An exploration license covers and area up to 100 km^2, has a term of three years and can be extended for two additional two-year terms. 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 exploitation license follows an exploration license and has a 25-year term that can be extended for a second 25-year term. It covers a maximum of 20 km^2. Surface rights fees are payable with all types of licenses and a royalty of up to 4% is payable at the exploitation stage, shared between the State and the Municipality where the Project is situated. The Cerro Blanco exploration concession was awarded to Entre Mares de Guatemala, S.A. on 12 November 1997. The concession covers an area of 39 km^2. Entre Mares applied for the conversion to an exploitation license which was granted in September of 2007.

Entre Mares has an exploitation license granted in 2007 for 25 years identified as LEXT-031-05. This license includes the water rights and rights of way associated with the property.

4.2.3 Land Strategy and Property Ownership

Entre Mares has purchased the surface over the expected deposit and additional land for plant facilities. Figure 4.3 shows the current land ownership in the immediate area around the Project.

Figure 4.3: Current Land Ownership Surrounding the Project Area



Source: Goldcorp (2014)

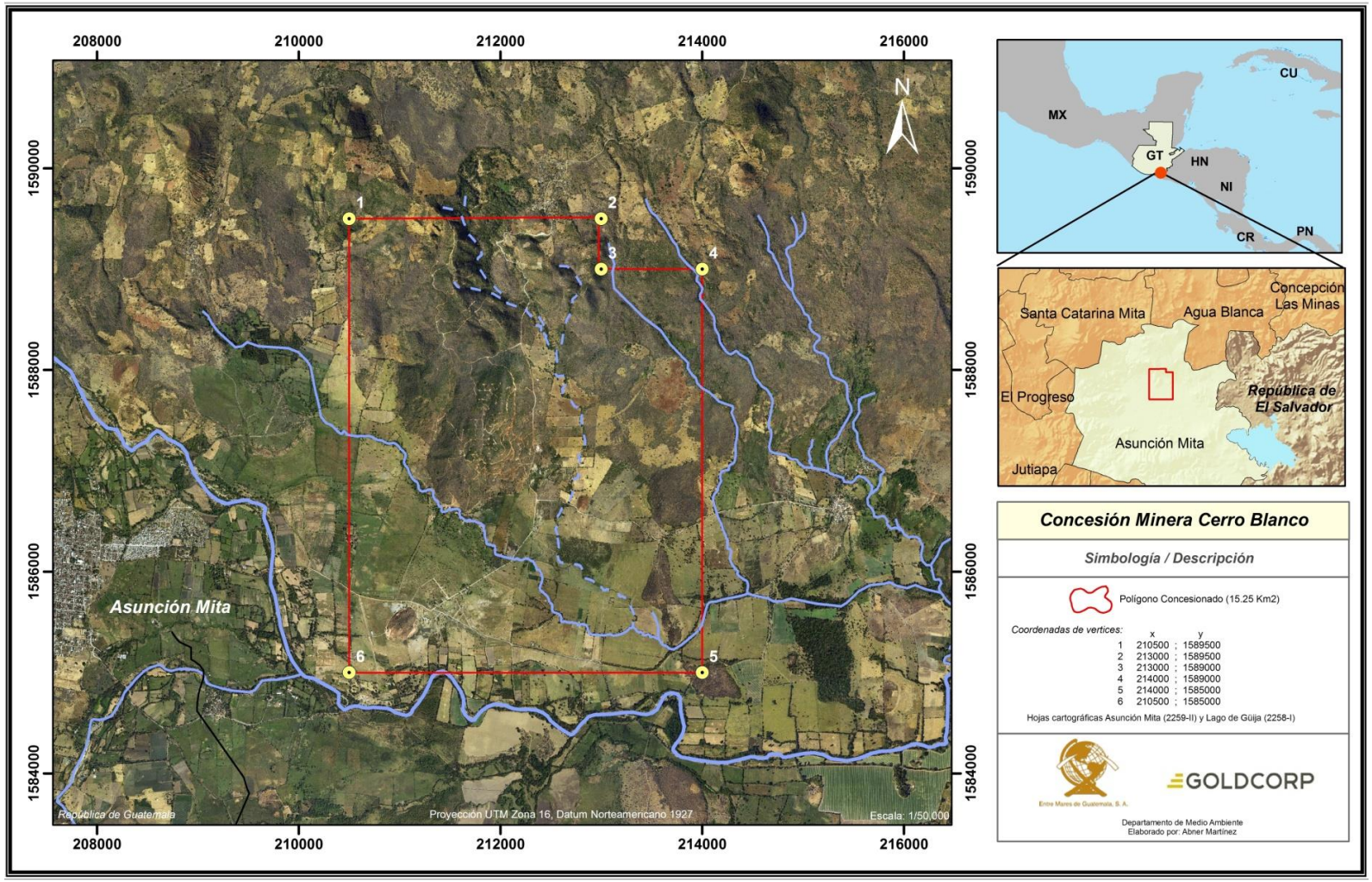
4.3 Mining Rights

The Project will be developed as an underground mining operation. Entre Mares owns the mining rights for Cerro Blanco and has the required approved permits and water rights to develop the property.

The exploitation decree covers all the mining claims which are contiguous. The coordinates of the 15.25 km^2 area are recorded in Decree DIC-CM-158-05 and are shown in Figure 4.5. The perimeter of the area is described as having the following vertices in the maps named Asunción Mita (2259-II) and Lago de Guija (2258-I):

- Vertex 1, north: 1589500.00, east: 210500.00;
- Vertex 2, north: 1589500.00, east: 213000.00;
- Vertex 3, north: 1589000.00, east: 213000.00;
- Vertex 4, north: 1589000.00, east: 214000.00;
- Vertex 5, north: 1585000.00, east: 214000.00;
- Vertex 5, north: 1585000.00, east: 214000.00; and
- Vertex 6, north: 1585000.00, east: 210500.00.

Figure 4.4: Current Land Ownership Surrounding the Project Area



Source: Goldcorp (2014)

4.4 Project Agreements

In 2007, the government of Guatemala (under the Mining Law, Decree Number 114-97) awarded Entre Mares an exploitation licence in the name of “Proyecto Minero Cerro Blanco”. This gave the Company the exclusive rights to mine gold and silver inside the boundaries of the approved concession. The licence was granted for a period of 25 years from the day that the notification was received.

The Project covers a concession which is 15.25 km^2. This area is defined in DIC-CM-158-05 issued on 28 June 2005. The area lies entirely in the municipality of Asunción Mita, in the Jutiapa District.

4.4.1 Liens, Charges and Encumbrances

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 km 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 31 December.

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 the 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, 14 August 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 and Considerations

To the extent known as a result of the PEA due diligence review, the Project is not subject to environmental liabilities other than the obligations attached to the existing permits. The Cerro Blanco Permits Register shows that all applicable permits obligations have been fulfilled.

4.6 Permit Requirements

As noted in Section 1.12, Cerro Blanco has all necessary permits to proceed with development of the underground mine and construction of the process facilities, subject to future operations adhering to the conditions of the exiting permits. One element of the Project not covered at this point by the existing permits is the proposed access road for which an EIA will have to be prepared and submitted to the local regulatory body.

4.7 Property Risks

During the PEA review, no significant factors and risks were identified that may affect access, title, or the right or ability to perform work on the property.

5 Accessibility, Climate, Local Resources, Infrastructure & Physiography

5.1 Accessibility

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

Guatemala has 400 km of coastline, with a deep-water port (Porto Quetzal) on the Pacific Ocean. On the Caribbean coast the port of Santo Tomás de Castillo has a modern facility. Both ports are connected by good highway access to the Cerro Blanco Project. Guatemala claims territorial waters extending out 22 km, plus an exclusive economic zone of 370 km offshore. Hurricanes and tropical storms sometimes affect the coastal regions.

There are five main ports in Guatemala.

The Atlantic ports are:

- Puerto Santo Tomás de Castilla (containers); and
- Puerto Barrios.

The Pacific ports are:

- Puerto San José (liquids);
- Puerto Quetzal (multi-use); and
- Puerto Champerico (fishing).

Puerto Santo Tomás de Castilla is the most important port on the Atlantic coast of Guatemala. It is a cargo terminal, which can handle many kinds of cargo such as containers, 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 Project site is approximately 440 km.

Puerto Quetzal is the most important port on the Pacific Coast and 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 and Physiography

The climate and vegetation are typical of a tropical dry forest environment. The elevation is between 450 masl and 560 masl. The wet season is from May through to October. The average annual rainfall is 1,250 mm. Daily highs reach 41°C and lows reach 10°C. The average annual pan evaporation rate is 1,440 mm with an annual average humidity of 64%.

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. The rain mainly occurs during the winter months (May through October). Maximum and minimum recorded temperatures are 41°C and 10°C respectively. Most of the vegetation in the area loses its foliage because of a lack of precipitation to support growth during the winter months.

The Project occurs within a south-southwest trending bedrock ridge that extends from higher ground to the north, outward into the basin and flood plain 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 masl 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 volcanoes are within sight of the Project area; to the north is the Ixtepeque volcano, to the northwest is the Suchitan, and to the southwest is the Las Viboras volcano.

The 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

Source: JDS (2017)

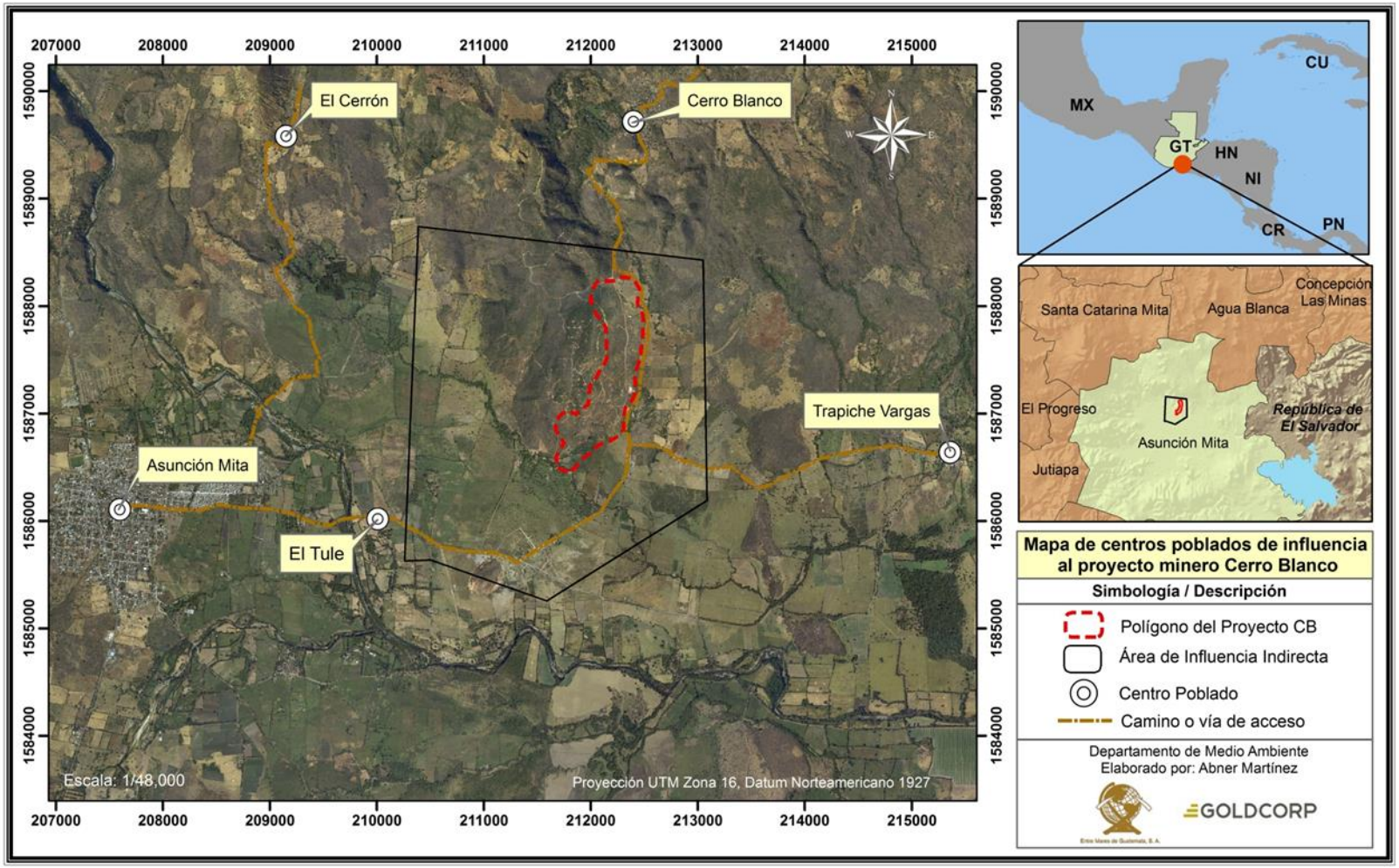
5.3 Local Resources and Infrastructure

The Project is situated in proximity to a number of communities, the largest one being Asunción Mita. The newly constructed La Baranaca substation is located a few kilometres south of Mita.

Farmers and cattle ranchers make up most of the local work force. There is no record of any previous mining in the area; however, with the closure of Goldcorp's Marlin mine in 2017, it is anticipated that a large contingent of Guatemalan trained labour will be available for employment at Cerro Blanco. In addition, Entre Mares will hire local people and train them. The mine work force will live in the surrounding communities and be transported to and from the mine site by buses.

Figure 5.2 shows the local population centers in close proximity to the Project site.

Figure 5.2: Population Centers near the Project Area



Source: Goldcorp (2014)

6 History

6.1 Management and Ownership

On January 4, 2017, Bluestone entered into an agreement with Goldcorp Inc. (Goldcorp) to acquire 100% of Minerales Entre Mares de Guatemala, S.A. (Entre Mares), Goldcorp's indirect wholly-owned Guatemalan subsidiary which holds a 100% interest in Cerro Blanco. On successful closure of the deal, Entre Mares shall be a wholly-owned subsidiary of Bluestone, a Canadian company headquartered in Vancouver, British Columbia.

6.2 Exploration History

Mar-West Resources Ltd. (Mar-West), has been working in adjacent Honduras since 1995. They expanded their gold prospecting activities into southern Guatemala in 1997; and found and they sampled they densely silicified boulders, in some cases cut by chalcedonic veins, on their first trip through the area. The Cerro Blanco property was identified by Mar-West during a reconnaissance evaluation of alteration areas near the active hot springs, and an exploration concession was granted in 1997. The hot springs sinter area was soil sampled, trenched and surveyed with IP-resistivity-CSAMT geophysics. Subsequent traverses over the hill at Cerro Blanco yielded surface rock assays of 1 g/t Au to 3 g/t Au. Mar-West drilled nine holes from April to June 1998. These initial reverse circulation (RC) holes tested near surface open pit potential, and went only to depths of 100 to 150 m. At least seven holes contained one or more intercepts of 5 to 15 m with 1 Au to 5 g/t Au, and with the occasional 10 to 20 g/t Au interval.

In October 1998, Mar-West 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. Drilling was advanced to Hole 19 and geophysical orientation surveys were completed.

Mar-West (1998) and Glamis (1999 to 2006) drilled a total of 233 (42,711 m) RC and DDH.

Goldcorp became the sole proprietor of the Cerro Blanco Project through the purchase of Glamis in November 2006.

More development of the underground mine, including extensive mapping and drilling programs, was done since, with the construction of two ongoing with two portals (North and South) and ventilation raises.

Exploration activities at the Cerro Blanco property by Goldcorp included:

- Surface soil geochemistry (gold);
- Surface rock geochemistry (gold);
- Surface geological mapping;
- Underground geological mapping from the exploration declines;
- Underground chip sampling (gold) from the exploration declines; and
- Surface and underground diamond drilling (since 2007).

6.3 Data Validation History

Core logging, sampling, and QA/QC procedures were reviewed with Goldcorp geological staff. Ten core samples were collected from 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 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 in half again with the quarter sample sent for analysis and the other quarter returned to the core racks. Table 6.1 summarizes the samples selected for core logging review and verification sampling.

Table 6.1: Verification Samples

Drill Hole ID	Duplicate Sample No.	Original Sample No	From (m)	To (m)	Deposit	Metal Analyzed	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)

Samples were sawed and bagged under Golder supervision and were transported off-site via helicopter and plane, and then ground transported to the 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. No issues were identified.

As part of the drill hole collar survey verification, Golder visited the North and South deposit locations. GPS surveys (Garmin etrex) were taken of selected drill hole collar locations and then compared against Cerro Blanco's final collar surveys completed in Cerro Blanco's Trimble system. All data collected is in UTM WGS 1984 Zone 16N and the accuracy of the Garmin GPS is approximately 10 m. The Cerro Blanco site surveys are in UTM NAD27 Zone 16N, and the data collected was converted to this system.

Comparison of the Golder surveys with the Cerro Blanco collar survey data shows an average difference of within 3 m on the Eastings, with maximum differences of +6.8 m and -6.7 m. However, Golder was unable to match the Northing data received from Goldcorp. All Northing data was off by an average of 112 m. Golder believed that there was a conversion issue when converting from WGS 1984 to UTM NAD27 Zone 16 N, and that the Cerro Blanco collar locations were likely accurate due to the consistent nature of the differences.

Table 6.2 is a list of the drill hole collar surveys completed by Golder.

Table 6.2: Drill Hole Collar Survey (NAD 27 Zone 16N)

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

Source: Goldcorp (2014)

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

Approximately 5% of the drill holes (20 holes) were subjected to data verification checks by Golder. The 20 selected holes (summarized in Table 6.3) 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 scatter plots; and
- Review of down hole survey measurements to identify anomalous changes to hole orientation.

Table 6.3: Drill Holes Selected for Data Verification

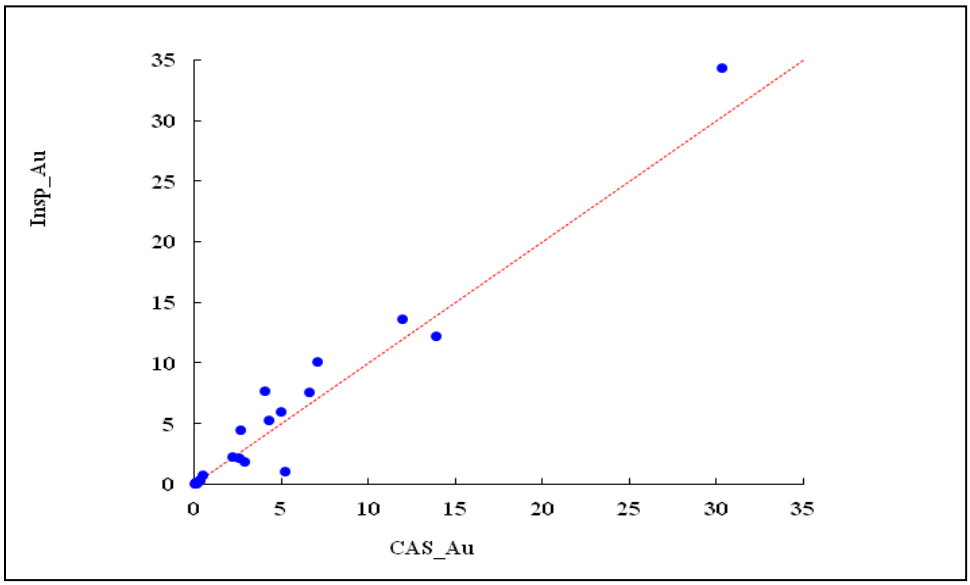
Drill Hole ID	
CB-012	CB-200
CB-016	CB-227
CB-063	CB-244
CB-078	CB-247
CB-095	CB-305
CB-10-02	CB-309
CB-120	CB-314
CB-142	CB-345
CB-146	CB-357
CB-151	CB-362

Source: Goldcorp (2014)

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 scatter plots were generated for each of the 20 holes. With the exception of a few outliers, the majority of the data compared well. Figure 6.1 illustrates an example of the XY scatter plots used to compare assay results.

Figure 6.1: Example of XY Scatter Plot for Hole CB34



Source: Goldcorp (2014)

6.4 Historic Resources

No reserves have been reported for Cerro Blanco to date. However, Indicated and Inferred Resources have been reported in 2008 for the Cerro Blanco Project at an 8.0 g/t gold equivalent cut-off grade isas follows in Table 6.4.

Table 6.4: Indicated and Inferred Mineral Resource Estimate

Resource Type [1]	Tonnes (kt)	Gold Grade (g/t)	Contained Gold (koz)	Silver Grade (g/t)	Contained Silver(koz)	Contained Equivalent Ounces of Gold [2]
Indicated	2,500	15.65	1,266	72	5,826	1,324
Inferred	1,400	15.3	665	59.6	2,589	691

Source: Voorhees (2008)

1) The Mineral Resources have been calculated in accordance with definitions adopted by the Canadian Institute of Mining, Metallurgy and Petroleum on August 20, 2000. Employees of Glamis Gold Ltd. under the supervision of James S. Voorhees, Executive Vice President of Operations and Chief Operating Officer have prepared these calculations.

2) The conversion of silver ounces to gold equivalent ounces is at a ratio of 100 silver ounces to one gold equivalent ounce.

In addition, Mineral Resources were reported as in-situ resources at cut-off grades of 1.0 g/t Au and 4.0 g/t Au in 2014. Blocks were classified based on drill spacing (sample distances) and the number of drill holes. In-situ Mineral Resources are summarized in Table 6.5. No wire framing was performed around these blocks.

Table 6.5: Cerro Blanco In-Situ Mineral Resources

In-situ Mineral Resources (veins)					
COG (g/t)	Mt	Au g/t	Ag g/t	Au Moz	Ag Moz
Indicated In-situ* Mineral Resources					
1	19	3.38	13.9	2.06	8.5
4	3.85	9.71	35.2	1.2	4.4
Inferred In-situ* Mineral Resources					
1	2.3	3.1	8.5	0.22	0,6
4	0.33	10.8	19.8	0.11	0.2

* Reported in-situ Mineral Resources do not consider mineral availability by underground or open pit mining methods

Source: Goldcorp (2014)

In 2016, Goldcorp listed resources for Cerro Blanco within its Annual Report. The Company stated that “Cerro Blanco is one of the highest grade pre-production, fully permitted gold projects in the world: Goldcorp’s most recent public resource statements outline historical resources of 2.05 Mt grading 12.69 g/t for 840,000 oz of gold in the Indicated category, as well as 0.75 Mt grading 9.34 g/t for 230,000 oz of gold in the Inferred category.”

The Indicated and Inferred resources are historical estimates and use the categories set out in NI 43-101. These resources are effective as of June 30, 2016 and are disclosed in Goldcorp's press release dated October 26, 2016. Resources were estimated using US$1,400/oz AU and US$20/oz Ag. 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 Setting and Mineralization

7.1 Introduction

Major tectonic forces have shaped the landscape of Guatemala. The country is roughly cut in half (north to south) by the triple juncture boundary of the North American, Caribbean, and Cocos tectonic plates. Extensive regional faults systems run parallel (east-west) to this feature such as the Cuilco-Chixay-Polochic fault system, the Motagua fault system, and the Jocotan Fault system. Nearer the Cerro Blanco deposit, other major regional structures such as the Jalpatagua and Ipala faults are responsible for shaping of the land and trend roughly north-northeast.

The north half of Guatemala contains abundant Cretaceous and Paleozoic age sediments overlying Mesozoic igneous rocks. The southern half of Guatemala lies within a volcanic cordillera with most of the major eruptive events occurring in the Tertiary and Quaternary periods. The Cerro Blanco Project lies within the volcanic province with the principle rock units being andesite, rhyolite, dacite and Cretaceous aged sediments. These units are in turn overlain by younger aged sinters, limestone, and alluvium as well as locally derived volcanic sediments and basalt.

The Cerro Blanco Project area is comprised of Cretaceous sediments, tertiary volcanics, sub-volcanics, sub-intrusives, volcaniclastic sediments, sinter and silicified sediments. The veined and gold bearing sinter and related silicified sediments occurs as a broad, elongate, north-northwest trending zone overlying a zone of structural weakness which narrows to the north and broadens to the south.

The mineralized zone is flanked to the west by relatively near flat-lying Cretaceous limestone, argillaceous to siliceous, fine to coarse volcaniclastics, and younger basalts. To the east, the zone is flanked by Cretaceous sediments and tuffaceous rocks. To the north is flanked by younger basalts and rhyolite and to the south by alluvium. Rhyolite domes and young basalt underlie the extreme northeast potion of the district.

Active hot springs occur at the conjunction of one of the west-northwest structures and a north-northwest structure along the southeast margin of the main zone. A small hill (Cerro Limon) just to the east of the active hot spring is underlain by well-layered, biotite-bearing, felsic tuff which hosts gold-bearing veins with similar attitudes as those found at Cerro Blanco.

7.2 Regional and Local Geology

Cerro Blanco is located in southern Guatemala, within an active volcanic arc, which begins close to the Guatemala-Mexico border and continues, more or less unbroken, along the Pacific side of Central America into central Costa Rica.

Central America comprises several disparate crustal blocks that slid together during Upper Cretaceous (70 to 90 million years ago) as they collided northward into the North American craton. 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 70 and 66 million years ago along the east-west-striking Polochic-Montagua fault system that crosses southern Guatemala (see Figure 7.1).

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

Figure 7.1: Cerro Blanco Regional Structure Map

Source: Google Maps (2014)

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 produced 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 that accumulated tuffaceous sediments continuously from Eocene (about 55 million years) to present (Donnelly, et al., 1990).

Principal periods of Andean-style calc-alkaline volcanism on 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 10 to 6 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 by about 7 million years and continues to drop today. Bimodal, rhyolite-basalt volcanism was initiated during this extension event and by 7 ALT million years was widespread throughout the western half of the Chortis block.

A large percentage of Central American gold deposits, including Marlin and Cerro Blanco, occur within a narrow belt that parallels the western Central American coast from southern Guatemala through Panama. This metallogenic belt follows the volcanic arc. Precious metal deposits are clearly related in space and time to Miocene-Pliocene extensional tectonics and associated bimodal volcanism. Published age dates cluster between 4 and 8 million years. An ^{40}Ar-^{39}Ar age date of adularia from Cerro Blanco returned a date of 2.69±0.01 million years in age.

7.2.1 Local Geology

The Cerro Blanco deposit is a classic hot springs-related, low sulphidation quartz-chalcedony-adularia-calcite vein system. It was localized along a complex fault intersection 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.

Wall rock alteration largely 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.

7.2.1.1 Lithology

Figure 7.2 shows approximate rock ages, the lithological column, and approximate thickness estimates of rock units. Intense hydrothermal alteration and complex faulting in the deposit area make these first approximations only.

Figure 7.2: Cerro Blanco Lithologies

		Post-Mineral Cover Rock-Quaternary	
		Qc	Colluvial and soil cover
		Qb	Basalt flows
		Cross-Cutting Intrusive Rock, Breccia, and Veins	
		bi	Basalt intrusive dikes and/or plugs
		vt	Vein
		cbx	Collapse Breccia. Multi-stage, polimictic collapse/hydrothermal breccia that contains Mita Group, Ad, and Vt clasts
		rp	Quartz eye rhyolite dikes and flow
		ad	Vesicular andesite dikes
		dp	Dacite flor
		gr	Granite (crosscuts Tca and Tcb)
		Salinas Tuff-Volcaniclastic Sequence-Pliocene	
≤150 m	**≤50 m**	Ss	Bedded chalcedonic silica sinter
		Svc	Stratified volcaniclastic sand to pebble conglomerate
		Srt	Quartz eye rhyolite tuff
		Sfx	Phreatic breccia (60 to 80% chaotic heterolithic clasts in rock flour matrix)
		Slt	Crystal lithic tuff
		Sct	Ash tuff
	≤40 m	Scgl	Volcaniclastic cobble conglomerate (clasts to 15 to 20 cm)-basal member
		Mita Group Volcano-Sedimentary Sequences-Paleocene/Eocene	
≥800 m		*Continental Clastic Sequence-Subinal equivalent*	
	>200 m	Mss	Coarse arkosic sandstone with poorly sorted pebble conglomerate horizons and beds (containing metamorphic quartz clasts)
		Tuffaceous Sequence	
	100 to 200 m	Mat	Andesitic crystal lithic tuff
		Mlt	Fine grained crystal tuff
		Mbt	Fine grained tuff and volcaniclastic unit with locally distinctive concretionary texture (accretionary lapilli?)
		Shallow Marine Sequence	
	100 to >150 m	Msc	Laminated calcareous siltstone
		Mls	Fosiliferous micritic limestone
		Volcaniclastic Complex (former Td Dacite)	
	200 to >300 m	Mcv	>80% Quartz latite crystal lithic tuff <10% Volcaniclastic breccia <10% Andesite porphyry flows
		Tempisque Volcanic Complex (Pre-Mita)-Cretaceous-Tambor Fm?	
		Tcb	Basalt-dominated
		Tca	Andesite-Dacite-dominated

Source: Goldcorp (2014)

Tempisque Complex Tca &Tcb

The Tempisque Complex is exposed in two road cuts and in several quebradas (gullies) north of fault EW-8 (Figure 7.1). The lower, Tca, andesite-dominated unit is strongly sheared and strongly chlorite-epidote-calcite altered in all exposures. It may include a limestone component. Cataclastic textures are apparent in drill hole CB-285. Fine grained andesite and granite dikes intrude Tca along faults.

Tcb unconformably overlays Tca as seen in road cuts near 9250N 1700E. Tcb comprises a sequence of basalts and intermediate tuffs. It shows only minor to moderate tectonic effects and is weakly to moderately clay-altered. These contrasts with the underlying Tca may indicate that the Tcb would be more appropriately placed in the Mita Group. Quaternary basalt unconformably overlays the Tcb north of 9500N.

Relationships seen in outcrop and drill hole CB285 lead to the conclusion that the Tc complex represents the oldest rock in the Project area. It could conceivably be part of the Cretaceous Tambor Formation which is reported to contain oceanic basalt, andesite, and pelagic limestone (Donnelly, et al., 1990). No drill hole reaches the Tc south of fault EW 8.

Mita Group – Lower Volcaniclastic Complex Mcv (formerly Dacite)

The lower Mita Group volcaniclastic sequence comprises thick quartz latite, pumice-rich crystal lithic tuff units (±80%), volcaniclastic breccias/conglomerates (≥10), and lesser andesite flows (≤10%).

Mita Group Shallow Marine Sequence Mls and Msc

An alternating sequence of shallow, fossiliferous marine limestone, Mls, and laminated calcareous siltstone, Msc, was deposited on top of the Mcv volcaniclastic sequence. North of 7500N, Glamis drilled several large Msc blocks up to 200 m thick within the Mcv. It is not clear from current drilling if these represent structural slivers or primary sedimentary lenses deposited deep in the Mcv sequence. The deposit model suggests that the Mls and Msc units are at least 100 to 150 m thick. These units are important mineralizes material hosts at Cerro Blanco.

Mita Group Tuffaceous Sequence Mbt, Mlt, and Mat

This is a dominantly tuffaceous sequence comprising fine grained volcaniclastics and crystal tuff units and overlays the shallow marine sequence. The fine grained tuff and volcaniclastic unit Mbt contains distinctive round concretionary textures (bolas) which often look like accretionary lapilli. A distinctive crystal lithic tuff unit, Mat, marks the top of the sequence. Mat is a useful marker bed in portions of the model. The Tuffaceous sequence also contains several 1 m to 15 m thick sand-pebble conglomerate lenses, similar to the overlying Mss unit. Structural complications in the deposit model make it difficult to confirm the thickness of this sequence. It appears to be in the range of 100 to 200 m thick. The Tuffaceous sequence tuffs and volcaniclastics are important economic mineral hosts at Cerro Blanco.

Mita Group Continental Clastic Sequence, Mss

The Mss, a hematite-rich arkosic sandstone-pebble conglomerate unit, caps the Mita Group in the Cerro Blanco area. This unit is likely correlative with the Subinal Formation, a Paleocene-Eocene red-bed sequence documented through southern Guatemala.

Salinas Tuff – Volcaniclastic Sequence

The Salinas tuff-volcaniclastic sequence unconformably overlays the Mita Group. This sequence is interpreted to be upper Miocene-Pliocene in age, based on the rhyolite-basalt rock association and the regional tectonic history. As such, Oligocene and Miocene calc-alkaline volcanism, widespread through much of Central America, is not represented or preserved, in the Cerro Blanco area. The Salinas group was deposited into and fills small, structurally controlled grabens in the deposit area.

The basal alluvial conglomerate (Scgl) is a useful marker horizon through the southern half of the model. The conglomerate is locally inter-bedded with and overlain by a series of volcaniclastic sand (Svc), ash tuff (Sct), lithic tuff (Slt), phreatic breccia (Sfx), and quartz-rhyolite tuff beds (Srt). Individual

beds are quite lenticular and discontinuous. The phreatic breccia seems to thicken dramatically to the east, at the expense of sedimentary rock types. Rhyolite tuff increases in thickness northward.

A large Srt/rp rhyolite body dominates the Salinas sequence in the northeastern portion of the model (7600N to 7800N) and is interpreted as a flow dome. The rhyolite shows contorted flow banding and possible carapace breccias, and is cut by hydrothermal breccias and stock work chalcedony veinlets. Most or all of the Salinas tuff and phreatic breccia beds are likely related to the rhyolite flow dome eruption.

Cross sections suggest that the flow dome erupted on top of, and post-dates, the basal Scgl conglomerate beds. The conglomerate also contains flow-banded rhyolite cobbles. This relationship indicates the possibility that blocks of rhyolite were subsequently incorporated into ongoing conglomerate deposition, which thickens below, and to the south of, the flow dome. The geological model and rock relationships further suggest that rhyolite dome eruption was likely the igneous event most closely tied, in space and time, to the mineralizing event at Cerro Blanco.

The upper 50 m to 80 m of the Salinas sequence comprises inter-bedded Sfx, Svc, and chalcedonic sinter. Locally, at least three distinct (stacked) sinter horizons, ranging from several metres to 40 m thick, developed. The sinter horizons often show lenticular aspect and appear to thicken above modelled faults, particularly NE-1 and the NW faults. Petrified tree and plant parts are common.

Cross-Cutting Intrusive Rock and Breccia

Granite (gr) has been encountered in the Far North drill area it intrudes the Tca. The granite is almost certainly related to the large group of 85 to 35 million year old plutons that intruded the Central America/North America suture zone. It demonstrates that Cerro Blanco is within the limits of the Cretaceous suture-related tectonic/structural domain. The granite is unaltered and unmineralized in drill intercepts to date.

Unaltered, fine grained, locally vertically-flow-banded igneous rocks underlie several dome-type features northeast of Cerro Blanco. These are interpreted to be dacite flow domes (dp).

Fine grained vesicular andesite dikes (ad) are numerous and intrude many of the northwest faults. They are particularly common in the northern half of the deposit model where they form a “dike swarm” in the footwall, on the west side of NE-1. Andesite dikes are tentatively identified in the Salinas sequence where they intrude the basal conglomerate and possibly higher into the sequence. This also places the andesite dike intrusive event close in space and time with the mineralizing event at Cerro Blanco.

A quartz eye rhyolite dike intrudes a northwest trending fault near 8550N 1740E, just above the Far North target zone. The dike is coarser grained but otherwise very similar to the rhyolite dome rock just north of the main gold zone.

Large collapse hydrothermal breccias (Cbx) formed south of 7300N and to a lesser extent near 7700N. These breccias are multi-phase, heterolithic, generally clast-supported breccias that contain predominantly Mita Group sediment clasts but also significant Mcv crystal lithic tuff clasts, andesite dike clasts, and gold bearing vein clasts. They are locally cross-cut by non-brecciated gold rich veins. This paradoxical relationship demonstrates that the Cbx formed during, and as an active consequence of, multiple mineralizing hydrothermal events. Cbx formed largely within, and at the expense of, the Mita calcareous sediment and tuffaceous units just above their contact with Mcv. Locally, Cbx related breccia dikes invade/cross-cut the Scgl conglomerates. The major Cbx bodies developed along, and

on the hanging wall side of, fault NE 1 (Figure 7.1). The Mcv-Mls contact and northwest faults provided secondary controls. East-west faults cut and displace Cbx, but it is not clear if they influenced its formation.

Two unaltered basalt occurrences are located between the main and Far North targets. These are interpreted as basalt intrusives (bi).

7.2.1.2 Structure

Table 7.1 summarizes the principal fault sets at Cerro Blanco. Geographical and geological factors, including the paucity of natural outcrop, limited road cuts outside the central drill area, and the fact that the Salinas volcaniclastic sequence post-dates and obscures most faults through the core area, present challenges to development of a clear and confident structural interpretation. There remains significant uncertainty in many areas.

Table 7.1: Summary of Important Structural Sets at Cerro Blanco

Fault Set	Structural Description
East-West ±10°	Paleocene-Eocene sedimentary basin-bounding, south-dipping normal faults. Reactivated, but recent movement mostly post-dates vein deposition.
N10° to 15°E	Ipala-parallel, graben-related normal faults. Primary vein control at Cerro Blanco.
N20° to 30°W	Provided principal control for andesite dike and quartz-rhyolite dike and dome emplacement. Primary vein control at Cerro Blanco.

Source: Goldcorp (2014)

Surface exposures, drill holes, and deposit modelling combine to indicate the following characteristics for each fault set:

- East-West Faults
    - All exposures of Tc Complex rocks are north of EW-8;
    - The northernmost drill holes, including CB285, cut thin Qb basalt cover. They then penetrate Mbt tuffs and volcaniclastic sediments, then pass directly into Tca, and finally bottom in granite. The deeper Mita Group volcaniclastics, sediments, and tuffs are missing, a stratigraphic section over 500 m thick. Likewise, south of EW-7, great thicknesses of Mcv crystal lithic tuff and volcaniclastic rock are encountered. No drill holes have bottomed the Mcv south of EW-7. These relationships imply that the east-west faults, specifically EW-8, EW-7, and probably others, are older basin-bounding structures that controlled the deposition and distribution of the Mita Group. The major movement on these faults is likely Paleocene-Eocene (45 to 65 million years in age);
    - East-west faults EW-0, EW-1, EW-3, EW-4, and EW-8 are the only faults that definitively cut and displace Ss sinter. Modelled vein zones also terminate against the east-west faults. However, it does not appear that veins follow the east-west faults. Very few drill holes cross the east-west faults so this has yet to be confirmed. The east-west faults were reactivated relatively recently and represent the only significant post-mineral fault activity; and
    - East-west faults strike within 10° of east-west and appear to be consistently normal faults, south side down.

- N10° to 15°E Faults
    - Fault NE-1 strikes N10°E and dips variably 40° to 60° East, east side down roughly 100 m, based on [marginally believable] stratigraphic correlation across the model (see 7200N);
    - NE-1 is the single most important control of hydrothermal activity and vein deposition in the Cerro Blanco deposit. NE-1 defines the overall trend of the deposit. Only rarely do significant veins continue into the west-footwall side of NE-1. Drill holes that pass west into the footwall side of NE-1 generally encounter relatively less altered to unaltered rock. The Cbx collapse/hydrothermal breccia formed along and in the hanging wall of NE-1. Recent work and underground mapping have demonstrated that the northeast trending faults were primary vein controls throughout most of the deposit;
    - Andesite dikes appear to occasionally intrude and follow the NE-1 faults, but they are not common;
    - NE-1 controls/forms the western margin of the Salinas sequence volcaniclastic wedge through the deposit area. It was therefore active predominantly prior to deposition of the Salinas sequence; and
    - The upper sinter horizons pass across the NE-1 fault projection without obvious displacement. There is no evidence for post-sinter or post-mineral movement on NE-1.
- N20° to 30°W Faults
    - A well developed northwest fault set is important throughout the deposit area. Northwest faults were mapped in road cuts north of the deposit. They are reported (Howell, 2003) from the Cerro Limon area southeast of Cerro Blanco. In outcrop, they strike N20° to 30°W with steep dips;
    - The Cerro Blanco deposit model shows a strong fault set of similar strike but with moderate northeast dips (faults NW-1 through NW-8). Movement/displacement on this fault set mostly predates Salinus Sequence deposition. The model interprets displacement of basal Scgl along northwest faults only between 7100N and 7250N (and this is open to alternative interpretation);
    - The northwest faults display significant offset on the Mcv crystal lithic tuff contact and, in general, apparent offsets on this contact define the northwest faults;
    - Andesite dikes commonly follow northwest faults, particularly in the footwall-west side of NE-1;
    - A quartz-rhyolite dike follows a northwest fault near the Far North target zone. The projection of that same northwest fault to the southeast lines up with the rhyolite dome where it crosses the NE-1 fault;

- The northwest trending faults were an important vein control, especially on the west side of the deposit. Northwest striking veins are well developed in the hanging wall of NE-1. Vein zone thicknesses and grades are often best near northwest fault intersections with NE-1. Only rarely do well-mineralized veins continue into the footwall of NE-1 (along northwest faults); and
- There is no evidence for post-sinter or post-mineral movement on the northwest faults.

The relationship between northeast and northwest faults is not well defined. NE-1 is clearly the dominant, throughgoing fault in the main deposit area; however, the northwest fault set is no less important in that they show significant offsets and controlled andesite and rhyolite intrusion. These two structural sets host most of the high-grade veins at Cerro Blanco.

7.3 Deposit Geology

The Cerro Blanco deposit is a classic hot springs-related, low sulphidation quartz-chalcedony-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 flow dome eruption.

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

A massive “silica cap” developed along the axis of the deposit and is the dominant alteration feature. Dense opaline silica-flooded phreatic breccia is inter-bedded with chalcedonic silica sinter from the present surface to depths of ±100 m. Fine grained quartz-pyrite flooded replacement horizons continue down another 100 to 200 m to the ±300 m elevation level. In total, the “silica cap” is 200 m to 300 m deep, 400 m wide and continuous over a kilometre north-south strike length. Silicification gives way outward and upward to clay-pyrite dominated assemblage. Below an elevation of ±450 m clay-silica-pyrite alteration grades downwards into sericite-clay-chlorite-calcite-pyrite assemblages. Disseminated pyrite, where unoxidized, is commonly 3% to 6% and locally reaches 10% by volume in the calcareous siltstone and fine grained tuffaceous units.

Wall rock alteration, to a large extent, determines geotechnical rock hardness and also 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.

7.3.1 Mineralization

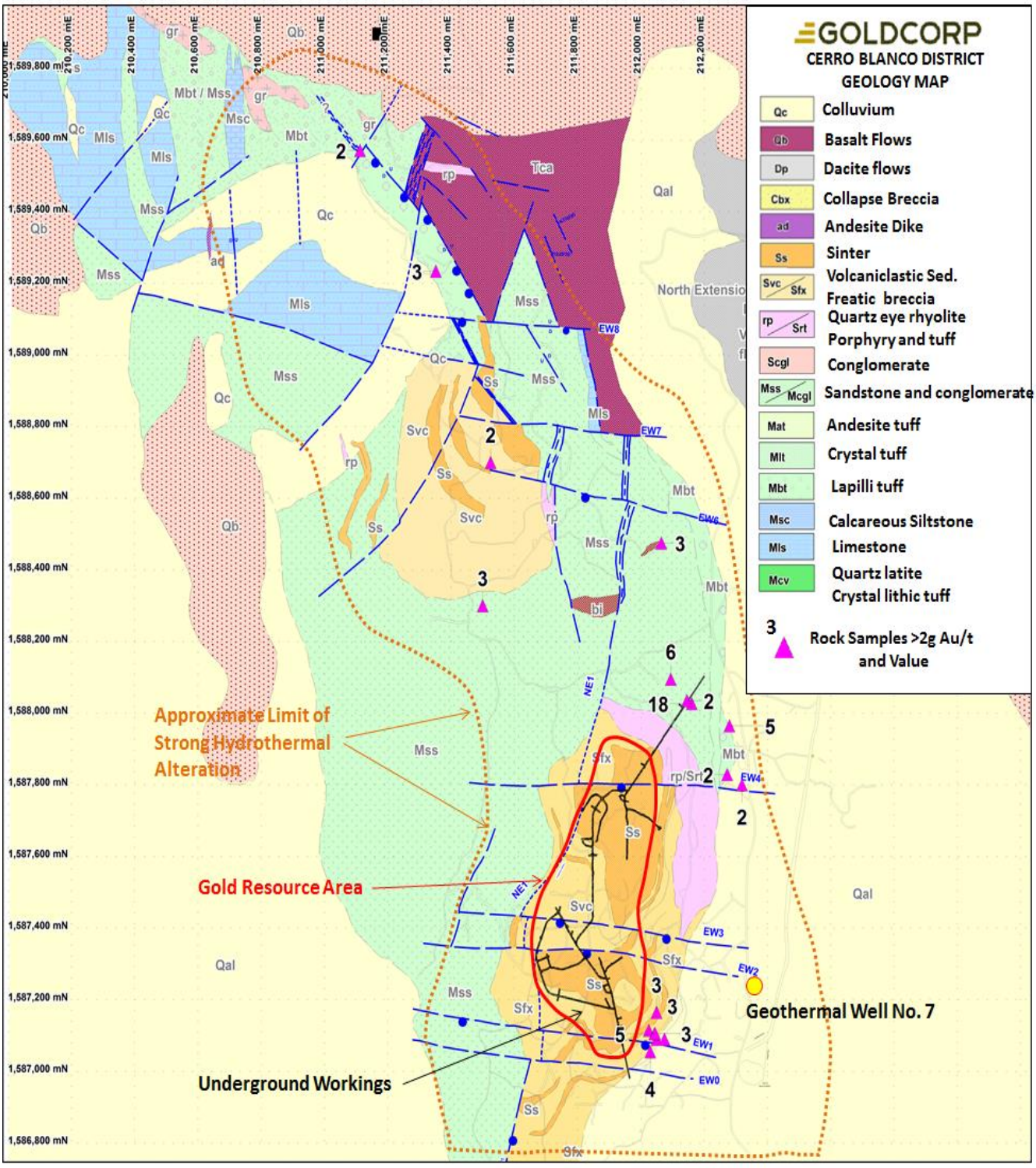
The Cerro Blanco district consists of localized precious metal veins and small stockwork deposits which occur over an area more than 20 km in diameter. The Cerro Blanco gold deposit occurs within

the largest mineralized area in a hydrothermal alteration zone covering an area about 5 km long and 1 km wide. This zone exhibits the effects of strong, pervasive hot spring type hydrothermal alteration.

The current gold resource occurs under a small hill and is confined within an area about 400 m by 800 m (Figure 7.3). The gold deposit is characterized by both high angle and low angle banded chalcedony veins, locally with calcite replacement textures. High angle mineralized faults and discontinuous stockwork zones host some of the highest gold grades. 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. Exposures are poor and locally covered by alluvium and post-mineral rocks. Veins outside the resource area tend to be narrow and host narrow high-grade shoots. They are, so far, of limited economic interest. Gold bearing structures extend at least 1 km south and southwest of the deposit under valley fill and post-mineral rocks. The southern area has good exploration potential for additional veins. However, exploration in this area was discontinued due to the presence of boiling water flashing to steam.

Geothermal well No. 7 located about 0.5 km southwest of the deposit encountered a 27 m zone averaging 6.3 g/t Au and 22 g/t Ag at a depth of 634 m. The upper 6 m of this zone averages 23.9 g/t Au and 79 g/t Ag.

Figure 7.3: Cerro Blanco District Geology Map and Exploration Potential



Source: Goldcorp (2014)

7.3.1.1 Vein Zones

A petrographic description of four vein zones, from 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 nearly 300 m elevation range. Bladed calcite or pseudomorphs after bladed calcite were seen in all four samples. Bladed calcite is a rapid deposition 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. The only significant depth-related textural or mineralogical change noted was an increase in quartz relative to chalcedony (Thompson, et al., 2006). Figure 7.4 shows a high-grade intercept in drill hole CB-309.

Figure 7.4: Banded Chalcedony-Calcite-Adularia-Sulphide Vein with Visible Gold from a 1.5 m Thick Interval in Drill Hole CB-309 with 218 g/t Au and 525 g /t Ag

Source: Goldcorp (2014)

Opaque minerals include pyrite, marcasite, acanthite, miargyrite (one sample), and visible gold. Disseminated grains occur in most textures of quartz, chalcedony, adularia, and calcite. Sulphides and visible gold also fill interstices between adularia, quartz, and chalcedony (Thompson, et al., 2006).

Only a fraction of mineralized material stage chalcedony-quartz-adularia-calcite veins contains high-grade gold, but these vein zones envelop the majority of high-grade intercepts and thus appear to offer a legitimate first order guide to productive vein locations and shape. Other observations include the following:

- Vein zones are best developed throughout the model between elevations of 300 m 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 calcareous sediments and overlying tuffs;
- Vein zones tend to narrow and steepen at depth in the Mcv, but high gold grades locally persist at least down to the 200 m elevation, notably in the southern third of the model. There are several locations north of 7400N where 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;

- Most vein zones stop at the Salinas contact. The principal vein-controlling faults, both the NE-1 and northwest faults, as discussed above, appear to show minor or no offset within the Salinas sequence; and
- To the extent that gold bearing chalcedonic veins continue upward into the Salinas volcaniclastics and sinter, vein orientations may be dramatically different from those beneath the contact.

There is ample drill hole evidence to demonstrate that vein zones follow the NE-1 and numerous NW-1 through NW-8 faults/fault trends. There are also likely to be steep northwest faults (with veins).

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.

7.3.1.2 Hydrothermal Alteration

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.

The most impressive alteration feature at Cerro Blanco is the large “silica cap” that typically begins at or below 400 m elevation and continues upward to the surface. 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 200 m deep for at least a kilometre in strike. Within 50 m to 100 m of the surface, silicification is manifested by opaline silica flooding in the fragmental Svc and Sfx units. At depth, very fine grained quartz replacement of Mita 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 clay alteration above 400 m or 450 m elevation. At deeper levels, silicified zones grade outward and downward into large volumes of clay-sericite-pyrite±calcite (Asp) alteration in Mita Group sediment and tuff. Pyrite contents are commonly in the range of 3% to 6%, locally reaching 10%.

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%.

The Cbx collapse/hydrothermal breccia displays diverse alteration assemblages. The alteration mineralogy often varies from clay-sericite to clay to chlorite-calcite to silica-dominant over short distances due to structural and permeability controls. These mixed assemblages suggest that hydrothermal conditions fluctuated dramatically and often; never permitting any particular mineral assemblage to attain equilibrium.

There are several large volumes of Mita Group Msc and Mls that show strong carbonate removal with clay alteration near the southern end of the model (7000N to 7150N). These carbonate depleted zones

occur adjacent/peripheral to large Cbx bodies. This rock that has a density around 2 g/ml probably has a very low rock strength.

The hydrothermal alteration model is a better general indicator of rock hardness within and near the mineralized zones than the actual lithologies. Estimates of rock hardness are presented in Table 7.2. These estimates do not account for faults and/or fractures. However, the overlaying of RQD values will develop a better overall rock quality model.

Table 7.2: Rock Hardness by Alteration

Hardness	Alteration Assemblage	Code
5	Sinter	
4 to 4.5	Silicified rock	(S2-3)
3.5 to 4	Quartz-sericite-pyrite	(QS2-3)
3 to 3.5	Chlorite-calcite-pyrite	(P2-3)
2.5 to 3	Clay-sericite-pyrite	(ASp2-3)
2 to 2.5	Moderate	(A2)
1 to 2	Strong clay (A3)	

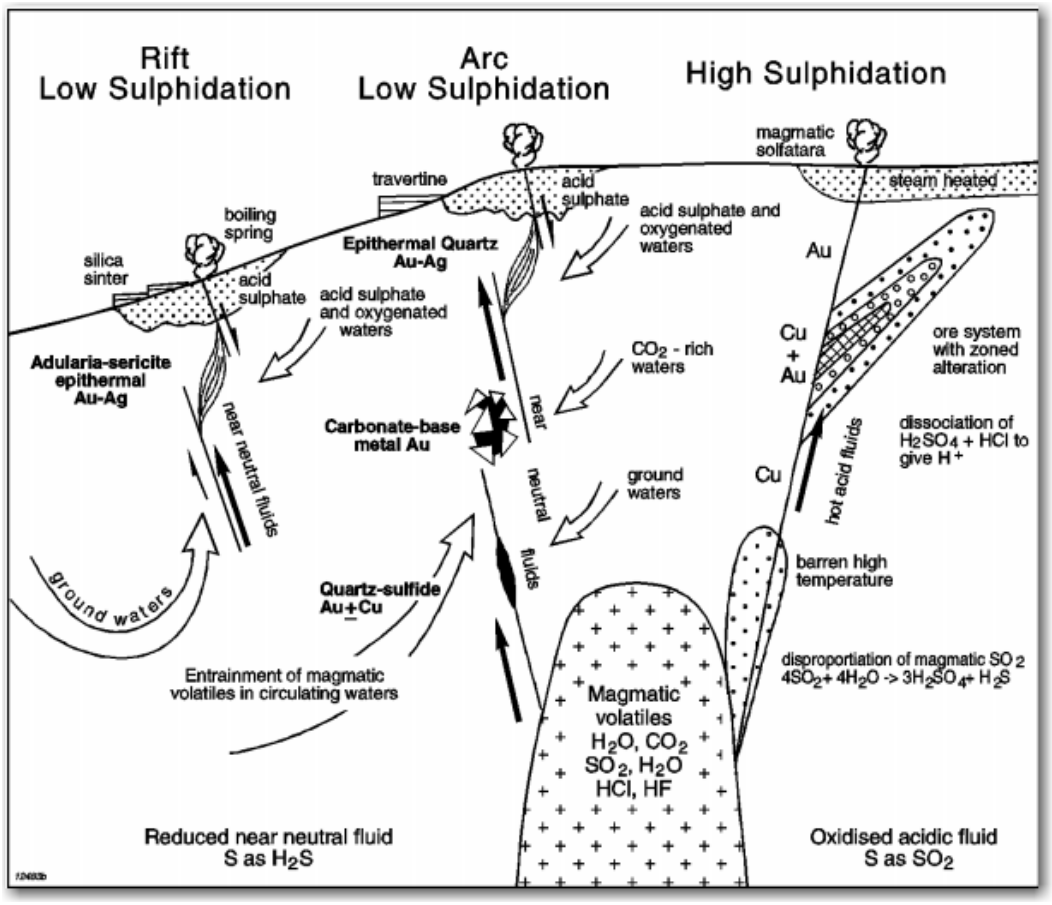
Source: Goldcorp (2014)

8 Deposit Types

8.1 Deposit Models

The Cerro Blanco deposit is a classic hot springs-related, low sulphidation quartz-chalcedony-adularia-calcite vein system. In the group of arc low sulphidation gold deposits the term epithermal is only incorporated in the epithermal quartz gold-silver deposits, which occur at highest crustal levels most removed from Inferred intrusion source rocks, however the adularia-sericite epithermal gold-silver deposits can be described as rift low sulphidation. This latter group occurs as the banded fissure veins comprising quartz (chalcedony), adularia and quartz pseudomorphing carbonate gangue minerals. 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 depressurised geothermal fluids, are characterized by intensely silicified matrix and generally anglular fragments including sinter, host rock and local surficial plant material. Although sinter deposits formed distal to fluid upflows, commonly associated with eruption breccias, tend to be barren with respect to gold and may be anomalous in other elements such as tungsten, 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) groundwaters is also favoured as a mechanism for the development of characteristic bonanza gold-silver grades. Adularia-sericite vein systems are silver rich with Au:Ag 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 throughgoing 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.

9 Exploration

There are no current exploration activities to report on the property.

10 Drilling

10.1 Drilling Summary

As of February 2012, Goldcorp had drilled approximately 95 holes for a total of 21,727 m on the Cerro Blanco property since the acquisition of Glamis Gold in November 2006. Table 10.1 summarizes the historical drilling on the property. There are currently no drilling results conducted on behalf of Bluestone Resources Inc.

Table 10.1: Summary of Drilling at Cerro Blanco

Year	Company	Holes Drilled	Metres
1998	Mar-West	9	1,340
1999	Glamis/Mar-West	48	7,217
2000	Glamis Gold	18	3,525
2001	Glamis Gold	0	0
2002	Glamis Gold	23	6,525
2003	Glamis Gold	0	0
2004	Glamis Gold	46	9,659
2005	Glamis Gold	120	29,065
2006	Glamis Gold	67	15,129
2007	Goldcorp	47	12,375
2008-09	Goldcorp	0	0
2010	Goldcorp	9	9
2011	Goldcorp	39	9,352
2012	Goldcorp	96	21,370
Total		**522**	**115,557**

Source: Goldcorp (2014)

A total of 517 holes were included in the block model. This drilling database contains 75,678 unique assay intervals representing a total core length of 114,974 m. Final drilling was completed by Goldcorp's own drill crew, using a Boart Longyear LM-75 drill rig. The drill rig was previously used for the surface drilling from 2009 through 2012 and had been converted to an underground drill rig. Drilling started with H-sized core and was reduced to N and B when ground conditions deteriorated.

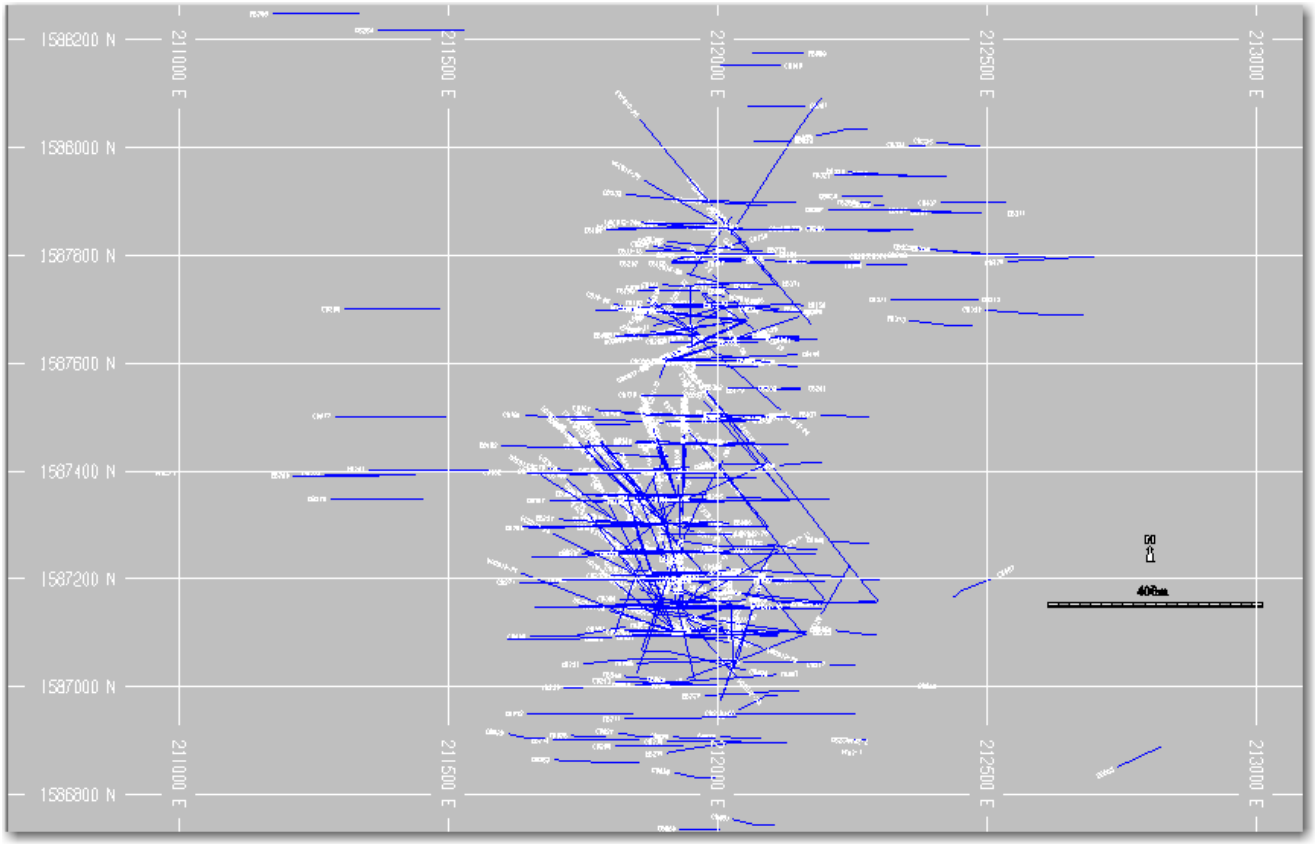
Surface drill hole collars are located in the field by the geologist using a GPS system based on UTM NAD 27 Zone 16N. Down hole surveying was completed by the drillers using the Reflex EZ-Shot system to determine hole dip and azimuth. Surveys were taken approximately every 50 m and provided to the geologist and entered in an Excel database. The Reflex EZ-Shot system is a single shot magnetic survey instrument. The local magnetic declination at the time of survey was positive 3˚ East.

The drill data consists of a combination of reverse circulation and diamond drill core, with many holes collared using RC and then switching to NQ size core. Collar data from these historical programs were surveyed with a differential GPS system, and down hole survey measurements were taken with either a single shot Sperry-Sun camera system or with a multi-shot Flexit instrument.

The majority of drill holes were not drilled perpendicular to the strike and dip of the veining and therefore do not represent true thickness of the veining. 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). The exact orientation of veining remains speculative in much of the deposit.

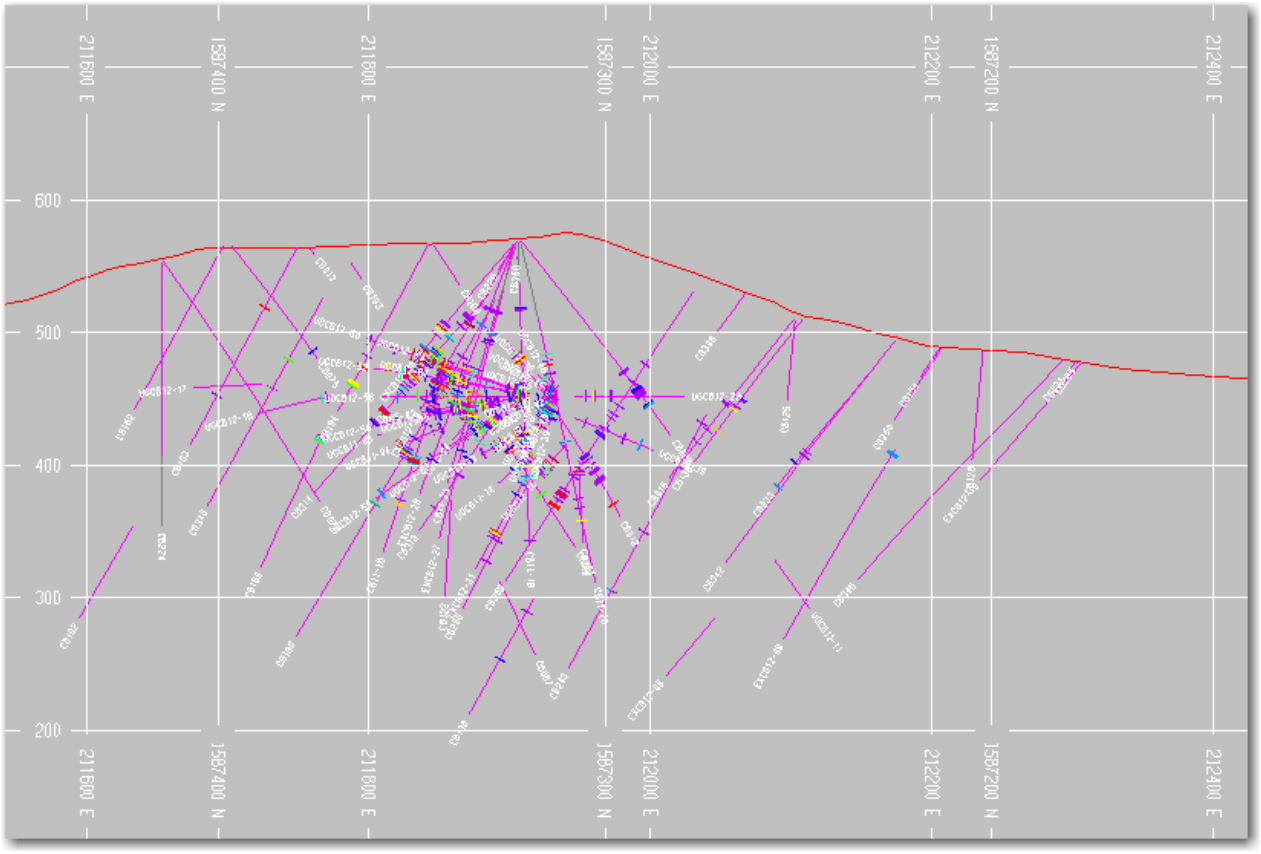
Figures 10.1 shows a plan view of drill hole locations and Figures 10.2 and 10.3 shows representative section views of the drilling along with gold assay data and topography.

Figure 10.1: Plan View of Drill Hole Locations



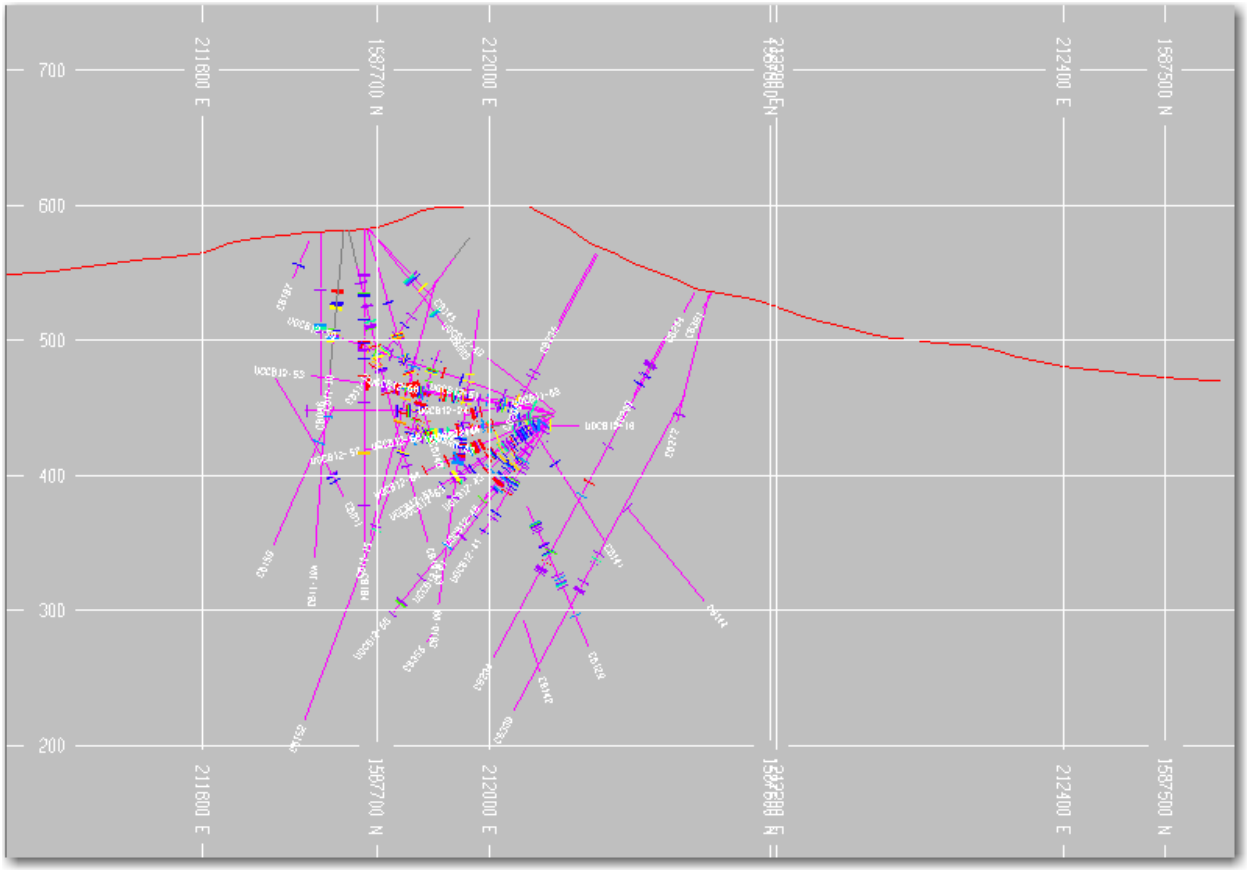
Source: Kirkham Geosytems (2017)

Figure 10.2: Section View (azimuth 110°) with Drill holes and Gold Data Along with Topography



Source: Kirkham Geosytems (2017)

Figure 10.3: Section View (azimuth 110°) with Drill holes and Gold Data Along with Topography



Source: Kirkham Geosytems (2017)

Table 10.2 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 do not represent the true thickness.

Table 10.2: Gold and Silver Samples from the Drill Hole Database

Hole	From	To	Length (m)	Au (g/t)	Ag (g/t)
CB-012	99.5	108.5	9	13.7	46.5
CB-012	141.5	147.5	6	12.9	75.8
CB-012	195.5	198.5	3	3	8
CB-012	236	237.5	1.5	13	6
CB-016	192.55	195.35	2.8	3.3	0
CB-063	88.5	99	10.5	4.4	25.7
CB-063	114	126	12	3.2	21
CB-063	183	186	3	7.7	20
CB-063	196.5	199.5	3	4.2	25
CB-063	207	210	3	18.7	20
CB-063	225	228	3	37.3	75
CB-063	241.5	244.5	3	5.1	3.5
CB-078	158.2	161.4	3.2	3.4	4.1
CB-078	242.1	245.1	3	3.5	4.7
CB-078	248.1	273.75	25.65	66.1	42.2
CB-078	299.25	303.75	4.5	4.7	17.7
CB-078	338.25	345.75	7.5	10.8	17.4
CB-095	155	158	3	3.7	204.9
CB-095	179	182	3	17.8	7.4
CB-095	233	236	3	88	98.6
CB-10-02	117.5	120.3	2.8	14.7	79.5
CB-10-02	135.75	139.5	3.75	12.9	91.8
CB-10-02	146	149	3	9.5	79.6
CB-10-02	168.86	173	4.14	26.2	144.8
CB-10-02	197	200	3	20.3	19.9
CB-120	219	238.5	19.5	17.5	20.3
CB-120	246	249	3	8.8	20.6
CB-142	163.5	171.5	8	16	72.2
CB-142	196.2	204.5	8.3	19.2	11.7
CB-142	302.75	306	3.25	19.3	14.3
CB-146	80.3	86	5.7	14	196.8
CB-146	109	112.4	3.4	10.3	78.9
CB-146	118.9	130	11.1	70.4	226.3
CB-146	139	143	4	12.4	35.4
CB-146	149	152	3	3.7	8
CB-146	156	159	3	21.1	30.6
CB-146	182	185	3	4.2	2.5
CB-151	162.4	165.5	3.1	25.6	152.8
CB-151	172.9	179.3	6.4	13.6	24.7
CB-151	327.5	330.5	3	5	5.5
CB-200	117	120	3	5.7	26

Hole	From	To	Length (m)	Au (g/t)	Ag (g/t)
CB-200	144	147	3	5	13
CB-200	152	161	9	7.5	13.6
CB-200	165	168.5	3.5	16.7	212.9
CB-227	117.34	124.96	7.62	15.4	20.6
CB-227	131	134	3	5.6	22
CB-244	90	99	9	10.3	57
CB-244	139.5	142.5	3	4.2	4
CB-244	234	237	3	22.5	21
CB-247	135	138	3	3.5	25.5
CB-247	159	162	3	4	4.5
CB-247	231	234	3	6.8	15.7
CB-247	240	243	3	28.6	98.5
CB-305	86	90	4	5	9.5
CB-305	138	141.5	3.5	5.5	21.3
CB-309	128.5	132	3.5	3.5	8.6
CB-309	183	186.7	3.7	130.1	304.6
CB-309	193.5	196.5	3	40.3	17
CB-314	99.5	102.5	3	5.3	11
CB-314	111.5	119.5	8	8.3	19.9
CB-314	124.5	127.5	3	24.2	113.6
CB-314	131.5	134.5	3	13.6	30.7
CB-314	140.5	143.5	3	11.8	45
CB-314	151.5	154.5	3	3.7	15
CB-314	175.5	178.5	3	85.6	386.9
CB-314	186	189	3	4.2	12.5
CB-345	231.7	234.7	3	13.1	20.8
CB-345	315.5	318.5	3	5.8	6.7
CB-357	63	66	3	5.5	33.3
CB-357	140	143	3	3.4	2.7
CB-357	159	162.5	3.5	4	2.7
CB-357	184	187	3	3.6	22
CB-357	192.5	195.5	3	46.4	126.3
CB-357	200	206.2	6.2	12.6	6.3
CB-357	217.5	220.8	3.3	4.3	5
CB-362	128.5	131.5	3	4.2	6
CB-362	219	222.2	3.2	4.5	6

Source: Goldcorp (2014)

11 Sample Preparation, Analyses and Security

11.1 Sampling Method and Approach

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

Prior to Goldcorp taking ownership of the Cerro Blanco Project in November 2006, all previous drilling, sampling, and assaying was under the control of Glamis (Voorhees, 2005).

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

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

- Double tube HQ core in the upper reaches of the hole switching to double tube NQ core deeper in the hole;
- RC 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; and
- Reverse circulation for the entire hole.

Sampling completed by Glamis included both DDH and RC. Core recovery was good and employed a telescoping drilling approach where larger-sized diameter drilling was completed in the upper part of the drill hole (HQ size) and reduced to NQ size deeper in the drill hole. In some holes, RC was completed in the upper part of the hole and then switched to diamond drilling when mineralization was encountered.

Rotary samples collected from the 4¾ inch, face-sampling, hammer-drilled RC were initially collected in a five-gallon bucket. The weight is 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, one half is 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 a pre-labelled plastic sample bags, one for assay and the other was stored. An air hose and nozzle was 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.

The DD core was collected and placed in wooden core boxes. The core was washed to obtain a clean surface for geologic 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, was placed in pre-labelled plastic bags.

The other half being retained for inspection or additional tests as warranted. Splits from the core holes were shipped to a facility operated by CAS labs in Honduras. Unused core was retained for inspection on-site.

All sample preparation and analyses were conducted by CAS Laboratories in Tegucigalpa, Honduras. Samples were transported from Cerro Blanco to the laboratory in Tegucigalpa, Honduras by CAS personnel. Reject samples and pulps were stored at the CAS facility. Samples were analyzed for gold, using a 30 gram pulp with Fire Assay, AA finish. Samples that ran over 1.0 g/t Au from this method were re-analyzed for both gold and silver using a 30 gram pulp, Fire Assay with Gravimetric finish.

Standards and blanks were inserted into the samples by Glamis. Glamis employed a frequency of 1 in 20 pulps for gold and silver analysis to the Chemex Laboratories in Nevada. A selection of coarse reject samples was sent for duplicate analysis at Inspectorate (Nevada) laboratory in Reno for gold, silver, and multi-element ICP analyses.

The drill samples were initially quick-logged to locate and mark significant changes in volcanic stratigraphy. Then each volcanic unit is described, 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 logging was done daily and transcribed on to 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 capturing data in four tables: lithology, alteration and sulphide type and geotechnical. 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 as type and angles taken related to core axis are displayed in an area as a graphical representation. The geotechnical data recorded was RQD data for the core portion of the hole.

Both independent laboratories used in the Cerro Blanco 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.

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 Au and Ag analysis in addition to coarse rejects.

11.1.2 Sampling Preparation, Analyses and 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 are developed by drilling larger diameter core at the early stage of the hole (HQ) and reduced to NQ size and BQ size if drilling conditions are difficult. Drill recovery is 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 core (surface and underground) were stored in wooden labelled boxes from the drill and transported to the surface core logging facility at Cerro Blanco surface core facility.

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 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 are 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 Microsoft Excel files. This data was then entered in MapInfo and MineSight™ software for geological modelling.

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

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

The quality control and quality assurance (QA/QC) program employed at the Cerro Blanco Project was under the direction of Goldcorp which included blank samples inserted by Goldcorp's geologists prior to shipping to Inspectorate (Guatemala) at a frequency of 1 in 25 sample submission. Duplicate samples included all pulps submitted by Goldcorp to Inspectorate which were being assayed at Marlin mine. No duplicates of coarse rejects or standards had been included in the QA/QC program at Cerro Blanco. However, it is recommended that duplicates of the coarse rejects be analysed and compared, and 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 Project site. The Cerro Blanco site is fully controlled by perimeter fencing and security on the property. All samples removed from site were under the control of Inspectorate Laboratories.

11.2 Quality Assurance and Quality Control

At the Cerro Blanco Project, inserting quality control samples takes place in the core shack before samples are shipped to the lab. These samples are routinely inserted and are used to check for accuracy and cleanliness in the analytical laboratory.

Field blanks are non-mineralized material inserted into the sample series every 25 samples (4%). Field blanks are inserted to test for any potential carry-over contamination which might occur in the crushing phase of sample preparation, because of poor cleaning practices. A total of 1,390 blanks where analysed with 558 performed at Laboratori Inspectorate, 302 at Laboratori CAS Honduras and 530 at the Marlin Mine laboratory. Analysis of the Inspectorate blanks resulted in 5 fails or 0.01% with 1 re-failing on resample. This appears to be the result of sample misclassification as both original and resample are relatively high grade. The CAS Honduras results showed 8 fails or 0.03% with 4 of those failing on resample. Therefore, there may have been some cleaning issues at CAS Honduras although it was not wide spread and not significant. The blanks from the Marlin Mine laboratory resulted in 14 fails or 0.03% which is not significant and considering that the Marlin Mine assaying was utilized for fast turnaround to guide the program and not for resource estimation purposes, does not pose an issue.

Core duplicate samples are 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 are quarter-spilt cores sampled on site and sent to Labratori Inspectorate and Labratori CAS Honduras, respectively. A total of 1060 samples with gold values > 2 grams per tonne were selected in the drillhole database through hole CB-222. Of those, a total of 797 samples were submitted for check analyses with 618 samples having been 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 shows 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 shows 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, probably attributable in 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 appear to show a relatively small grade bias which should be further investigated should the company continue to use this laboratory in the future.

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 is recommended that a QA/QC program be implemented during all future drill programs that include the insertion and analysis of standards in addition to blanks and duplicates along with umpire assays.

It is the opinion of the QP, Garth Kirkham, P.Geo., that the adequacy of the sample preparation, security, and analytical procedures employed are to industry standards and are appropriate for mineral resource estimation.

12 Data Verification

A site visit to the Cerro Blanco property was completed by Michael Makarenko (JDS), Mike Levy (SRK) and Ken Embree (KP) on February 1st, 2017. Maz Mohaseb (JDS) visited the site on May 18th, 2017. The purpose of these visits was to fulfill the requirements specified under NI 43-101 and to familiarize with the property. These site visits consisted of an underground tour of non-mineralized development headings as well as an inspection of the surface core logging, sampling, and storage areas.

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

The tour of the offices showed a clean, well-organized, professional environment. On-site staff led the author through the chain of custody and methods used at each stage of the logging and sampling process. All methods and processes are up to industry standards and reflect best practices, and no issues were identified. The core is accessible and the core is stored in covered racks. However, going forward it is recommended that the Company refurbish some of the core facilities, clean and remove broken pulp bags. In addition, it would be recommended that the core be re-arranged for easier access and analysis along with creating a core map.

The author selected 16 drill holes from the database and they were laid out at the core storage area. Site staff supplied the logs and assay sheets for verification against the core and the logged intervals. The data correlated with the physical core and no issues were identified. In addition, the author toured the complete core storage facilities, selecting and reviewing core throughout. No issues were identified. However, some of the high-grade sections have been re-sampled multiple times and, as a result, there is minimal core remaining for viewing. It is recommended that in future drilling campaigns that a minimum of half core be retained. The results of the data verification performed independent of the Company in May 2014 are detailed in Section 6.3. The author accepts the analysis and results and believes that this information may be relied upon.

The author is confident that the data and results are valid based on the site visit and inspection of all aspects of the Project, including methods and procedures used. It is the opinion of the independent author that all work, procedures, and results have adhered to best practices and industry standards required by NI 43-101. No duplicate samples were taken by the author to verify assay results, but the author is of the opinion that the work was being performed by a well-respected, large, multi-national company that employs competent professionals that adhere to industry best practices and standards.

The data verification process did not identify any material issues with the Cerro Blanco sample/assay data. The author is satisfied that the assay data is of suitable quality to be used as the basis for this resource estimate.

Mining data was verified during the site visit and through review of previous studies completed for the site. Any studies referred to were thoroughly reviewed, updated and revised as required to align with the PEA mine design and mine plan. All mining data was verified and is adequate for the PEA Technical Report as required by NI 43-101 guidelines.

Regarding metallurgical data, the first step taken to verify the data was to review the metallurgical reports prepared by Kappes, Cassiday & Associates (KCA) based on metallurgical samples from Cerro Blanco. The sample locations were summarized in metallurgical test programs conducted by KCA for Glamis and Goldcorp between 1999 and 2012. The samples were received as bags/buckets of material, RC drill chip rejects and ½ split HQ and PQ core. Some of the samples were composed at the Cerro Blanco property and shipped to KCA and others were prepared by KCA at their laboratory. The drill core provided to KCA was clearly identified by drill hole and interval of each sample allowing the material to be identified within the ore zone. The samples ranged in head grade from under 10 g/t up to 50 g/t gold and represented different geological rock types. The drill holes used for metallurgical test work 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. It is the QP's opinion that there is sufficient data and test work to estimate metallurgical recoveries and define the flowsheet at a PEA level.

Regarding permitting and environmental data, the first verification step by the QP was to request a copy of the current permits register which was issued by the site Environmental Manager in Spanish. The QP obtained an English translation of the register through a certified Translator. The formal English version of the permit register was then distributed to the PEA project team for a detailed review. The Cerro Blanco mine site has existing water management infrastructure currently being operated under existing permits issued by the Guatemala Authorities. All data related to permitting, water management and environment are adequate for the Technical Report as required by NI 43-101 guidelines.

No limitations or failures to conduct data verification were identified by the QPs in the preparation of this Technical Report.

13 Mineral Processing and Metallurgical Testing

13.1 Introduction

For the purposes of preparing this PEA, JDS reviewed the metallurgical testing procedures and results as reported by Kappes, Cassiday & Associates (KCA) in the Cerro Blanco 2013 Update Feasibility Study, Section 16.0, Metallurgy Rev F and Appendix I Metallurgy of the study report.

Metallurgical test work was conducted between April 1999 and January 2012 and no further test work has been reported on the Cerro Blanco deposit since 2012.

This PEA includes the metallurgical test work and results in Sections 13.2 and 13.4-13.7 as reported by KCA, and subsequent analysis and interpretation by JDS in Sections 13.8 and 13.9 of this report.

- The metallurgical testing in the 2011 test program, reported by KCA, also included:
- Solid-Liquid Separation (SLS) tests on leached material from the 2011 tests, by Pocock Industrial Inc.;
- Abrasion index, and rod and ball mill Bond Work Indices by Phillips Enterprises Inc;
- Mineralogical study by Carson GeoMin; and
- Cyanide detoxification study by CyPlus GmbH.

The term “whole ore leaching” used in this section describes a metallurgical process and does not imply the confirmed presence of ore.

13.2 Metallurgical Test Reports as Reported

The Project metallurgical test work consisted of agitated cyanide leach tests and gravity/leach tests on mined and rotary drill hole samples. During this time period, a mineralogical study was also performed by Carson GeoMin Inc., and grindability and abrasion index determinations were completed by Phillip Enterprises. Table 13.1 summarizes the individual test reports that KCA has analyzed.

Table 13.1: Summary of Metallurgical Reports

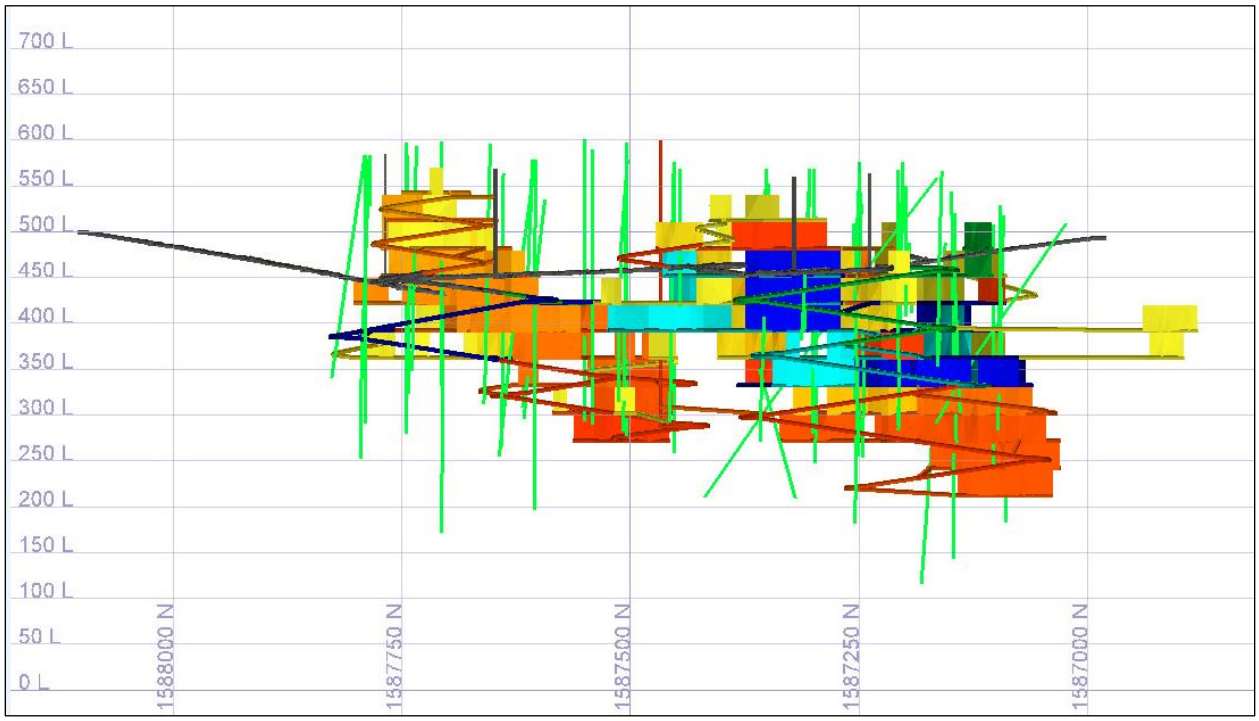
Samples Received	Report Date	Report Title	Tests	Issued By:
12 Mar, 1999	08 Apr. 1999	Cerro Blanco Project, Results of Cyanide Leach Tests	Bottle Rolls on As-Received Material. Cyanide Soluble on Pulverized Material	KCA
Sep. 1999	12 Jan. 2000	Cerro Blanco Project, Results of Cyanide Bottle Roll Tests	Cyanide Bottle Rolls	KCA
10 Aug. 2000	24 Aug. 2000	Cerro Blanco Project, Bottle Roll Tests	Cyanide Bottle Rolls	KCA
14 Aug. 2002	08 Oct. 2002	Cerro Blanco Project, Results of Leaching Tests and Gravity Concentration Tests	Gravity/Leach Tests Direct Leach Tests	KCA
	Aug. 2005	Cerro Blanco North Zone Samples for Met Testing at SGS-Lakefield	Gravity plus Leach of Tails "Whole Ore Leach"	SGS-Lakefield
31 Oct. 2005	15 Dec. 2005	Cerro Blanco Project	Cyanide Bottle Roll Tests	KCA
	29 Dec. 2005	Mineralogy of Ore Composites and Related Cyanide Tailings from the Cerro Blanco Gold Project	Mineralogical Study	Carson GeoMin
31 Oct. 2005	16 Jan. 2006	Cerro Blanco Project	Cyanide Bottle Roll Leach Test	KCA
22 Mar. 2006	21 Apr. 2006	Cerro Blanco Project	Cyanide Bottle Roll Leach Test	KCA
29 Apr. 2011	11 Jun. 2011	Comminution Tests, Cerro Blanco	Grindability and Abrasion Tests	Phillips Enterprises
15 Apr. 2011	25 Jan. 2012	Cerro Blanco Project, Report of Metallurgical Test Work, January 2012	Cyanide Bottle Roll Tests, Detoxification Tests	KCA

Source: KCA (2013)

13.3 Sample Selection

Figure 13.1 illustrates the holes that were drilled for the metallurgical test programs completed by KCA with respect to the new mine plan as represented by the green intercepting lines. No additional drilling for metallurgical sampling is anticipated. The drill holes are considered spatially representative and provide variability with respect to gold and silver grades.

Figure 13.1: Metallurgical Sample Drill Holes



Source: JDS (2017)

13.4 Pre-October 2005 Metallurgical Test work Review

13.4.1 March-April 1999 Cyanide Leach Tests

KCA received 44 samples in March 1999, which ranged from fine clay-like material to a maximum particle size of 19 mm. A portion of each sample was split out for a bottle roll test. The remaining material was pulverized (minus 0.15 mm) and a split was taken for Fire Assay of gold and silver. A cyanide soluble analysis was also run on the pulverized head material.

Table 13.2 summarizes the gold extraction results of bottle roll tests performed on the as-received material and gold and silver extractions of the pulverized material.

Table 13.2: Summary of April 1999 Bottle Roll Tests

Bottle Roll Test	Calculated Head (g Au/t)	Gold Extraction (%)	NaCN Consumption(kg/t)	$Ca(OH)_2$ Consumption (kg/t)
As-received (-19 mm), Average	7.39	51	0.31	1.10
Pulverized, Average	7.42	41.17	85	84

Source: KCA (2013)

13.4.2 September 1999-January 2000 Cyanide Bottle Roll Tests

In early September 1999, KCA received samples that consisted of 11 5-gallon buckets of ½ split HQ core material and one large bag containing eight separate samples of RC drill chip reject.

Based upon the geology presented by Glamis Gold, the eight (8) RC Chip intervals were compiled into four separate composites for bottle roll leach tests. The four composites were separated into the following geological descriptions: Breccia with veining (BwV), Lithic tuff (LT), Fine tuff (FT), Veins in breccia tuff (VBT). A series of bottle roll leach tests were conducted on the four composites which investigated: the gold recovery versus grind size, gold recovery via carbon in leach (CIL) tests, and cyanide concentration versus gold recovery.

Fourteen bottle roll tests were conducted that examined the % passing 75 µm grind size versus gold and silver extraction. The extraction for gold and silver increased for all composites as the material finer. NaCN consumption increased as the grind became finer.

Bottle roll leach tests with carbon added (CIL) were completed on each individual composite sample. Table 13.3 summarizes the results of these tests.

Table 13.3: Summary of January 2000 Bottle Roll CIL Tests

Sample Description	Pass 75 µm (%)	Calculated Head (g Au/t)	Calculated Head (g Ag/t)	Gold Extraction (%)	Silver Extraction (%)	NaCN Consumption (kg/t)	$Ca(OH)_2$ Consumption (kg/t)
VBT	96	1.56	11.5	91	81	3.34	2.00
FT	98	1.45	7.2	95	92	2.89	2.00
LT	98	0.53	1.9	81	47	3.60	2.00
BwV	98	0.80	8.8	66	88	3.48	2.00

Source: KCA (2013)

Bottle roll leach tests were completed on each composite in which the level of NaCN was maintained at varying concentrations between 0.25 and 2.50 g NaCN per L of solution. Results indicated showed the extraction of gold and silver was not significantly improved with above 1.00 g NaCN. Composites VBT and FT has extractions in the high 90's, LT in the mid 80's and BwV in the mid 60's.

13.4.3 August 2000 Bottle Roll Tests

In August 2000, nine composite drill cutting samples were delivered to the KCA facility. The samples were composed at the Cerro Blanco property prior to shipping to KCA. After drying, each composite sample was crushed to a nominal 1.70 mm. The results of the bottle roll tests are summarized on the following Table 13.4.

Table 13.4: Summary of August 2000 Bottle Roll Tests

Sample Description	Pass 75 µm (%)	Calculated Head (g Au/t)	Calculated Head (g Ag/t)	Gold Extraction (%)	Silver Extraction (%)	NaCN Consumption (kg/t)	$Ca(OH)_2$ Consumption (kg/t)
CB 63, 88.5-139.5 m	68	2.48	13.1	90	89	0.26	3.8
CB 63, 139.5-243 m	65	1.96	5.7	84	75	0.19	3
CB 64, 16.5-63 m	48	1.47	6.8	79	70	0.23	3.5
CB 64, 103.5-129 m	77	1.63	8.7	90	84	0.16	3.5
CB 64, 129-174 m	60	1.54	7.3	97	74	0.21	3.3
CB 65, 115.5-205 m	74	0.57	3.2	70	57	0.19	3
CB 66, 127.5-163.5 m	69	2.02	8.1	91	83	0.11	3
CB 67, 105-172.5 m	70	2.22	8	87	82	0.15	3
CB 67, 172.5-238.5 m	74	2.73	9.5	82	85	0.18	3

Source: KCA (2013)

13.4.4 Aug-October 2002 Results of Leaching and Gravity Concentration Tests

In August 2002, KCA received six 5-gallon buckets of samples from Cerro Blanco. Each bucket contained individual intervals designated for a composite sample (six composite samples were generated). The composites were test at 212 µm and 106 µm with and without gravity before leaching. The results indicated that gravity did not significantly improve gold extraction. Leach extraction was improved with the finer grind size.

13.4.5 August 2005 Gravity and "Whole Ore Leach" Tests

In August 2005, SGS-Lakefield conducted gravity plus leach of tailings and "whole ore leach" (WOL) tests. The tests were performed on four high-grade composite samples. Similar to the test work in 2002 gravity followed by leach did not significantly improve overall gold and silver extractions compared to WOL.

13.5 Post October 2005 Metallurgical Tests Completed at KCA

From October 2005 to 2011, three different campaigns of cyanidation tests (plus other tests such as detoxification) have been completed at the KCA metallurgical laboratories.

13.5.1 October-December 2005 Cyanide Bottle Roll Tests

A total of 45 samples were delivered to KCA on October 2005. Each sample had a portion that was pulverized to P_{80} of 75 µm. A bottle roll leach test was then performed on each pulverized portion.

Table 13.5 shows the results for the leach kinetic test, which are grouped by gold feed grade as follows:

< 10 g/t Au,10 – 20 g/t Au and 20 - 50 g/t Au.

KCA also reported the calculated incremental extractions for various leach times between 48 and 96 hours as presented in Table 13.6.

Table 13.5: November 2005 Tests - Summary of Extraction Results

		Average Extraction %											
		0 h Leach	2 h Leach	4 h Leach	8 h Leach	24 h Leach	48 h Leach	72 h Leach	96 h Leach	Grade (g/t)	Reagent Usage (kg/t)		No. of Tests
											NaCN	Lime	
Samples: Head Grade < 10 Au g/t													
Au	% Extraction – Avg.	0	35.6	53.1	67.6	81.2	83.8	85.0	87.5	5.61	0.65	1.69	13
	Incremental % Extraction	0	35.6	17.5	14.5	13.6	2.6	1.2	2.5	-	-	-	-
Ag	% Extraction – Avg.	0	31.9	41.6	49.5	58.4	63.8	64.2	67.7	27.02	-	-	-
	Incremental % Extraction	0	31.9	9.7	7.9	8.9	5.4	0.4	3.5	-	-	-	-
Samples : Head Grade 10.1 to 20 Au g/t													
Au	% Extraction – Avg.	0	29.4	45.4	63.6	84.4	90.0	91.4	95.0	13.32	0.98	1.40	5
	Incremental % Extraction	0	29.4	16.0	18.2	20.8	5.6	1.4	3.6	-	-	-	-
Ag	% Extraction – Avg.	0	23.2	35.8	51.0	69.4	76.8	79.2	86.0	78.62	-	-	-
	Incremental % Extraction	0	23.2	12.6	15.2	18.4	7.4	2.4	6.8	-	-	-	-
Samples: Head Grade 20.1 to 50 Au g/t													
Au	% Extraction – Avg.	0	31.2	49.3	67.5	89.2	92.3	93.8	94.0	30.53	0.47	1.33	6
	Incremental % Extraction	0	31.2	18.1	18.2	21.7	3.1	1.5	0.2	-	-	-	-
Ag	% Extraction – Avg.	0	28.8	42.5	56.7	74.0	79.7	83.5	86.8	52.25	-	-	-
	Incremental % Extraction	0	28.8	13.7	14.2	17.3	5.7	3.8	3.3	-	-	-	-
Samples: Head Grade > 50.1 Au g/t													
Au	% Extraction – Avg.	0	29.3	43.5	60.8	77.0	81.3	83.2	89.8	99.13	0.72	1.20	6
	Incremental % Extraction	0	29.3	14.2	17.3	16.2	4.3	1.9	6.6	-	-	-	-
Ag	% Extraction – Avg.	0	19.7	28.2	40.5	54.8	61.8	67.5	74.3	241.45	-	-	-
	Incremental % Extraction	0	19.7	8.5	12.3	14.3	7.0	5.7	6.8	-	-	-	-
Overall Average:													
Au	% Extraction	-	-	-	-	-	-	-	90.55	30.58	0.68	1.47	30
Ag	% Extraction	-	-	-	-	-	-	-	75.83	83.55	-	-	-

Source: KCA (2013)

Table 13.6: November 2005 Tests - Increase in Extraction with Leaching Time

	Average Extraction %		Incremental % Extraction		
	48 h Leach	96 h Leach	48 to 72	72 to 96	48 to 96
Samples: Head Grade < 10 Au g/t					
Au %	83.8	87.5	1.2	2.5	3.7
Ag %	63.8	67.7	0.4	3.5	3.9
Samples: Head Grade 10.1 to 20 Au g/t					
Au %	90.0	95.0	1.4	3.6	5.0
Ag %	76.8	86.0	2.4	6.8	9.2
Samples: Head Grade 20.1 to 50 Au g/t					
Au %	92.3	94.0	1.5	0.2	1.7
Ag %	79.7	86.8	3.8	3.3	7.1
Samples: Head Grade > 50.1 Au g/t					
Au %	81.3	89.8	1.9	6.6	8.5
Ag %	61.8	74.3	5.7	6.8	12.6

Source: KCA (2013)

13.5.2 January 2006 Bottle Roll Leach Tests

Bottle roll tests were requested by Glamis Gold on samples from the December 2005 test program. Glamis Gold selected three (3) samples for an additional bottle roll test. The original bottle roll tests were conducted on material that was pulverized to a P_{80} of 75 µm. These additional tests were conducted on material that was pulverized to P_{80} of 45 µm. The data suggests that increasing the leach time increases the overall gold recovery of the sample. Although the newer bottle rolls were done at a finer grind size, a close look at the time vs. extraction data in the report shows that after four days of leaching at the finer grind size, it did not make a significant difference in gold extraction compared to the 4 days of leaching at the coarser grind size.

13.5.3 April 2006 Bottle Roll Leach Tests

Fifty-five samples were delivered to KCA in March 2006 and were prepared for bottle roll leach tests. Table 13.7 summarizes the results based on head grade. The incremental change in extraction with increased leach times is shown in Table 13.8.

Table 13.7: March 2006 Tests - Summary of Extraction Results

		Average Extraction %									Grade (g/t)	Reagent Usage (kg/t)		Avg. Grind %	No. of Tests
		0 h Leach	2 h Leach	4 h Leach	8 h Leach	24 h Leach	48 h Leach	72 h Leach	96 h Leach	168 h Leach		NaCN	Lime	= 75 µm	
Samples: Head Grade < 10 Au g/t															
Au	% Extraction – Avg.	0	34.3	55.3	69.0	84.2	92.8	94.7	95.5	95.5	5.74	0.74	2.33	95.5	6
	Incremental % Extraction	0	34.3	21.0	13.7	15.2	8.6	1.9	0.8	0.0	-	-	-	-	-
Ag	% Extraction – Avg.	0	30.3	44.7	53.7	68.8	77.0	79.0	81.8	85.3	33.00	-	-	-	-
	Incremental % Extraction	0	30.3	14.4	9.0	15.1	8.2	20.0	2.8	3.5	-	-	-	-	-
Samples : Head Grade 10.1 to 20 Au g/t															
Au	% Extraction – Avg.	0	24.6	38.8	50.6	78.4	90.2	93.9	95.4	96.5	15.35	0.89	2.27	92.3	11
	Incremental % Extraction	0	24.6	14.2	11.8	27.8	11.8	3.7	1.5	1.1	-	-	-	-	-
Ag	% Extraction – Avg.	0	20.4	31.8	42.2	59.2	72.5	78.5	83.8	90.3	79.00	-	-	-	-
	Incremental % Extraction	0	20.4	11.4	10.4	17.0	13.3	6.0	5.3	6.5	-	-	-	-	-
Samples: Head Grade 20.1 to 50 Au g/t															
Au	% Extraction – Avg.	0	30.0	49.3	65.8	82.1	90.8	93.9	95.7	96.1	27.90	1.00	2.00	93.3	9
	Incremental % Extraction	0	30.0	19.3	16.5	16.3	8.7	3.1	1.8	0.4	-	-	-	-	-
Ag	% Extraction – Avg.	0	22.1	32.7	42.7	59.2	70.6	76.2	82.9	89.4	138.20	-	-	-	-
	Incremental % Extraction	0	22.1	10.6	10.0	16.5	11.4	5.6	6.7	6.5	-	-	-	-	-
Samples: Head Grade > 50.1 Au g/t															
Au	% Extraction – Avg.	0	25.0	46.9	66.0	80.3	90.6	95.5	96.0	96.9	116.87	1.02	2.09	89.7	11
	Incremental % Extraction	0	25.0	21.9	19.1	14.3	10.3	4.9	0.5	0.9	-	-	-	-	-
Ag	% Extraction – Avg.	0	22.5	41.9	59.2	73.8	80.1	83.1	86.2	90.8	156.10	-	-	-	-
	Incremental % Extraction	0	22.5	19.4	17.3	14.6	6.3	3.0	3.1	4.6	-	-	-	-	-
Overall Average:		-	-	-	-	-	-	-	-	-	-	0.93	2.16	92.3	37
Au	% Extraction	-	-	-	-	-	-	-	-	97	47.03	-	-	-	-
Ag	% Extraction	-	-	-	-	-	-	-	-	89	91.21	-	-	-	-

Source: KCA (2013)

Table 13.8: March 2006 Tests - Increase in Extraction with Leach Time

Leaching Time (h)	Average % Extraction			Incremental % Extraction				
	48 h Leach	96 h Leach	168 h Leach	48 to 72 h Leach	72 to 96 h Leach	48 to 96 h Leach	96 to 168 h Leach	48 to 168 h Leach
Samples: Head Grade < 10 Au g/t								
Au	92.8	95.5	95.5	1.9	0.8	2.7	0.0	2.7
Ag	77.0	81.8	85.3	2.0	2.8	4.8	3.5	8.3
Samples: Head Grade 10.1 to 20 Au g/t								
Au	90.2	95.4	96.5	3.7	1.5	5.2	1.1	6.3
Ag	72.5	83.8	90.3	6.0	5.3	11.3	6.5	17.8
Samples: Head Grade 20.1 to 50 Au g/t								
Au	90.8	95.7	96.1	3.1	1.8	4.9	0.4	5.3
Ag	70.6	82.9	89.4	5.6	6.7	12.3	6.5	18.8
Samples: Head Grade > 50.1 Au g/t								
Au	90.6	96.0	96.9	4.9	0.5	5.4	0.9	6.3
Ag	80.1	86.2	90.8	3.0	3.1	6.1	4.6	10.7

Source: KCA (2013): JDS 2017

13.5.4 April 2011-January 2012 Bottle Leach Tests

In April 2011, KCA received six pallets from the Cerro Blanco Project. The pallets contained a total of 55 cloth bags containing ½ split HQ and PQ core material from five samples. A portion from each sample was then utilized in the generation of a Master Composite. The amount of each individual sample utilized to generate the Master Composite was determined by Cerro Blanco personnel. The individual samples and Master Composite were utilized in metallurgical testing.

Head analyses were completed on each individual sample as well as the Master Composite. Portions of the head material were assayed for gold and silver content. A portion of each sample was also assayed semi-quantitatively by means of an ICAP-OES for an additional series of elements and for whole rock constituents. In addition to these semi-quantitative analyses, each sample was assayed by quantitative methods for carbon, sulphur and mercury. Cyanide soluble analyses were completed on a portion of each sample. The head analyses for gold and silver are summarized in Table 13.9.

Table 13.9: Summary of Head Analysis - Gold & Silver

KCA Sample No.	Description	Average Assay, g Au/t	Average Assay, g Ag/t
48901	MbT	9.40	46.25
48902	Mcv	4.47	5.79
48903	Svc	6.52	44.71
48904	Msc	5.07	38.79
48905	Cbx	4.59	18.06
48907	Master Composite	7.70	37.86

Source: KCA (2013)

13.5.4.1 Individual Composites

Five bottle roll leach tests were performed on a portion of each individual composite, one test for each composite. The material was milled to a target P_{80} of 53 µm (KCA Sample Nos. 48901-48905). The test parameters for these tests were optimized based on the tests performed on the Master Composite. Table 13.10 summarizes the results of these initial bottle roll tests.

The average gold extraction percentage was 90% with a range of 80% to 94%. The average silver extraction percentage was 86% with a range of 80 to 90%. The average NaCN consumption was 5.09 kg/t of material with a range of 3.91 to 7.68 kg/t of material. The average hydrated lime consumption was 1.10 kg/t of material with a range of 1.00 to 1.50 kg/t. Each individual composite was utilized in a bottle roll leach test in order to determine gold content and leaching behavior.

The gold calculated head values of the individual composites ranged from 4.25 to 6.16 g/t of gold. While four of the five individual composites exhibited gold extractions of above 90%, only 80% of the gold was extracted from the Msc composite (KCA Sample No. 48904). No correlation between gold content and extraction percentage appeared to exist.

The silver calculated head values of the individual composites ranged from 8.14 to 51.83 g/t Ag of material. The silver extraction percentages ranged from 80 to 90%. No correlation between silver content and extraction percentage appeared to exist.

It should be noted that the Mbt sample (KCA Sample No. 48901) contained the highest contents for both gold and silver of all of the individual samples (6.164 Au g/t and 51.83 Ag g/t).

Table 13.10: Summary of Cyanide Bottle Roll Leach tests on Individual Composites

Sample Description	Grind Time, Min	Target NaCN, g/L	P_{80} Grind Size, µm	Calculated Head, g Au/t	Extracted Au (%)	Calculated Head, g Ag/t	Extracted Ag (%)	Leach Time, days	NaCN Consumption, kg/t	$Ca(OH)_2$ Consumption kg/t
Mbt	68	1	61	6.16	94	51.83	90	4	5.14	1.00
Mcv	67	1	58	4.25	93	8.14	80	4	4.55	1.00
Svc	83	1	65	5.39	93	34.61	86	4	7.68	1.50
Msc	69	1	77	4.62	80	41.21	86	4	3.91	1.00
Cbx	83	1	53	5.76	92	29.52	84	4	4.18	1.00
Averages	**-**	**1**	**63**	**5.24**	**90**	**33.06**	**86**	**4**	**5.09**	**1.10**

Source: KCA (2013)

13.5.4.2 Comminution Test Work

Portions of material from each of the individual samples were submitted to Phillips Enterprises, LLC in Golden, Colorado for comminution testing. Test work was completed to determine the abrasion index and the rod mill and ball mill work indices for each sample submitted. The results of the test work are summarized in Table 13.11.

Table 13.11: Physical Test Work - Grinding Indices

KCA Sample No.	Description	Abrasion Index	Rod Mill Work Index (kWh/t)	Ball Mill Work Index (kWh/t)
48901	MbT	0.1931	17.08	20.27
48902	Mcv	0.1035	13.91	16.37
48903	Scv	0.3280	18.26	22.24
48904	Msc	0.3286	16.90	21.45
48905	Cbx	0.2461	15.52	18.95
Averages		**0.2399**	**16.33**	**19.86**

Source: Phillips Entreprises Project No. 112014 (2011)

13.5.4.3 Bottle Roll Leach Test Work

A total of 48 milled bottle roll leach tests were completed on portions of the Master Composite (KCA Sample No. 48907). Twenty-six bottle roll leach tests were utilized for specific testing parameters in order to optimize gold and silver leaching. The test parameters were milling time, NaCN level, milling solids loading, leach time and dissolved oxygen (DO) levels. The tests were completed and organized according to the test parameter. In addition, 22 milled bottle roll leach tests were completed on portions of the Master Composite for detoxification testing as well as solid/liquid separation.

From the bottle rolls only 23 were leached for 96 hours. The other bottle roll tests were leached for one to two days depending on the type of test. As with the 2005 and 2006 bottle roll tests described in earlier Sections, the extractions were averaged for each of the kinetic samples taken, i.e., after a leaching period of two, four, eight, 24, 48, 72 and 96 hours. Unlike the previous bottle rolls from 2005 and 2006, the samples head grades were all under 10 g/t Au, therefore all of these samples will be averaged within the same group. The results of the extraction results are summarized in Table 13.12 and Table 13.13.

Table 13.12: January 2012 Tests – Summary of Extraction Results

		Average Extraction %								Grade (g/t)	Reagent Usage (kg/t)		No. of Tests
		0 h Leach	2 h Leach	4 h Leach	8 h Leach	24 h Leach	48 h Leach	72 h Leach	96 h Leach		NaCN	Lime	
Samples: Head Grade < 10 Au g/t													
Au	% Extraction – Avg.	0	9.0	27.0	51.6	82.9	88.3	90.0	91.9	5.93	2.37	1.00	23
	Incremental % Extraction	0	9.0	18.0	24.6	31.3	5.4	1.7	1.9	-	-	-	-
Ag	% Extraction – Avg.	0	11.8	29.5	46.9	72.0	81.2	84.0	88.1	34.84	-	-	-
	Incremental % Extraction	0	11.8	17.7	17.4	25.1	9.2	2.8	4.1	-	-	-	-

Source: KCA (2013)

Table 13.13: January 2012 Tests – Increase in Extraction with Leaching Time

	Average Extraction %		Incremental % Extraction		
	48 h Leach	96 h Leach	48 to 72 h Leach	72 to 96 h Leach	48 to 96 h Leach
Samples: Head Grade < 10 Au g/t					
Au %	88.3	91.4	1.7	1.9	3.6
Ag %	81.2	87.0	2.8	4.1	6.9

Source: KCA (2013)

By analyzing the results, the same conclusions can be made about the mineralized material that were made from the 2005 test results: gold leaches faster than silver, silver still seems to be leaching after 96 hours, and that gold leaching is essentially complete after 96 hours.

13.5.4.4 Grind versus Extraction

The first series of analyses performed on the Master Composite were utilized in order to optimize the grind size of the material.

For gold extraction, the optimum grind size appeared to be a target P_{80} of 75 µm. The gold extraction did not noticeably change as the grind size decreased beyond the target P_{80} of 75 µm. For example, the test utilizing material milled for 70 minutes (calculated P_{80} of 77 µm) exhibited a gold extraction of 94% (KCA Test No. 48913 C). Meanwhile, the test utilizing material milled for 90 minutes (P_{80} of 55 µm) exhibited a gold extraction of 94% (KCA Test No. 48916 B).

For silver extraction, the optimum grind size appeared to be a target P_{80} of 53 µm. Contrary to the gold extraction, the silver extraction increased steadily as the grind size decreased beyond the target P_{80} of 75 µm. For example, the test utilizing material milled for 70 minutes (calculated P_{80} of 77 µm) exhibited a silver extraction of 84% (KCA Test No. 48913 C).

Meanwhile, the test utilizing material milled for 90 minutes (calculated P_{80} of 55 µm) exhibited a silver extraction of 88% (KCA Test No. 48916 B).

The average gold extraction percentage was 92% with a range of 86 to 94%. The average silver extraction percentage was 87% with a range of 78 to 99%. The average NaCN consumption was 1.31 kg/t of material with a range of 0.61 to 1.81 kg/t of material. The average hydrated lime consumption was 0.95 kg/t of material with a range of 0.75 to 1.25 kg/t of material.

The gold extraction appeared to reach a limit once the grind size achieved a calculated P_{80} of 85 µm, with the extraction percentages reaching the 92 to 94% range with no discernible trend with decreasing material size. However, the silver extraction did not reach 90% until a calculated grind size of P_{80} of 71 µm.

13.5.4.5 Cyanide Level Analyses

The second series of analyses performed on the Master Composite were completed in order to determine the optimum NaCN concentration in the leach solutions for the Cerro Blanco material. The material was tested at a target P_{80} of 75 µm and a target grind size of P_{80} of 53 µm. For gold extraction, the optimum NaCN concentration appeared to be 1 g NaCN per L of solution. The gold extraction did not noticeably change as the NaCN concentration increased.

For example, the test utilizing material milled for 70 minutes (calculated P_{80} of 76 µm) and a NaCN concentration of 1 g/L exhibited a gold extraction of 92% (KCA Test No. 48914 B). Meanwhile, the test utilizing material milled for 70 minutes (P_{80} of 73 µm) and a NaCN concentration of 3 g/ L exhibited a gold extraction of 93% (KCA Test No. 48914 D).

For silver extraction, the optimum NaCN concentration appeared to be 3 g NaCN per L of solution. Contrary to the gold extraction, the silver extraction increased slightly as the NaCN concentration increased. For example, the test utilizing material milled for 70 minutes (calculated P_{80} of 76 µm) and a NaCN concentration of 1 g/L exhibited a silver extraction of 88% (KCA Test No. 48914 B). Meanwhile, the test utilizing material milled for 90 minutes (P_{80} of 56 µm) exhibited a silver extraction of 91% (KCA Test No. 48916 D).

For the material milled for 70 minutes (calculated P_{80} of 75 µm), the average gold extraction percentage was 92% with a range of 88 to 93%. The average silver extraction percentage was 90% with a range of 88 to 91%. The average NaCN consumption was 1.81 kg/t of material with a range of 1.36 to 2.80 kg/t of material. The hydrated lime consumption for all of these tests was 1.00 kg/t of material.

For the material milled for 85 minutes (calculated P_{80} of 53 µm), the range of gold extraction percentages was 93 to 95%. The range of silver extraction percentages was between 88% and 99%. The average NaCN consumption was 1.90 kg/t of material with a range of 1.63 to 2.14 kg/t of material. The hydrated lime consumption for all of these tests was 1.00 kg/t of material.

13.5.4.6 Leach Time versus Extraction

The third series of analyses performed on the Master Composite were utilized in order to optimize the leaching time of the milled product. The slurry was then utilized for leach testing and leached for one, one and a half, two and four days.

For gold extraction, the optimum leaching time appeared to be 36 hours (one and a half days). The gold extraction did not noticeably change as the leach time increases beyond 36 hours.

For example, the test utilizing material leached for 36 hours (calculated P_{80} of 45 µm) exhibited a gold extraction of 95% (KCA Test No. 48917 D). Meanwhile, the test utilizing material leached for 48 hours (calculated P_{80} of 55 µm) also exhibited a gold extraction of 95% (KCA Test No. 48917 E).

For silver extraction, the optimum leaching time appeared to be 48 hours (two days) at a NaCN concentration of 3 g/L. The silver extraction changed slightly as the leach time increases beyond 36 hours. For example, the test utilizing material leached for 36 hours (calculated P_{80} of 45 µm) exhibited a silver extraction of 92% (KCA Test No. 48917 D). Meanwhile, the test utilizing material leached for 48 hours (calculated P_{80} of 55 µm) exhibited a silver extraction of 93% (KCA Test No. 48917 E).

For the material utilizing 1 g/L NaCN solution, the gold extraction percentage was 92% for both tests. The silver extraction percentage was 91% and 96%. The average NaCN consumption was 3.64 kg/t of material. The hydrated lime consumption for all tests was 1.00 kg/t of material. See Table 13.14.

From the tests completed on material milled with 50% solids and 3 g/L of NaCN, gold extraction percentage is similar when comparing one, one and a half, and two days of leaching (average 0% increase). However, the silver extraction percentage increased when comparing one to one and a half days (average 5% increase), but did not increase when comparing one and a half to two days (average <1% increase).

13.5.4.7 Increased Dissolved Oxygen Testing

The fourth series of analyses performed on the Master Composite were utilized in order to study the effect of increasing the dissolved oxygen (DO) levels to the leaching study.

For the material utilizing 1 g/L NaCN solution, the gold extraction percentages were 91% for DO levels greater than 20 mg/L and 92% for levels between 10 to 15 mg/L. The silver extraction percentages were 83% for DO levels greater than 20 mg/L and 80% for levels between 10-15 mg/L. The average NaCN consumption was 1.00 and 1.25 kg/t of material. The hydrated lime consumption for both tests was 1.50 kg/t of material.

These tests did not show a significant benefit to gold and silver extraction when increasing the DO levels.

Table 13.14: Dissolved Oxygen Testing

KCA Test No.	DO mg/L	Target NaCN, gpL	p80 Grind Size, µm	Calculated Head, g Au/t	Extracted Au	Calculated Head, g Ag/t	Extracted Ag	Leach Time, days	NaCN Consumption, kg/t	$Ca(OH)_2$ Consumption kg/t
48927A	>20	1	47	6.65	91%	44.44	83%	2	1.25	1.50
48927B	10-15	1	44	5.80	92%	33.20	80%	2	1.00	1.50

Source: KCA (2013)

13.5.4.8 Detoxification Bottle Roll Test

The bottle roll leach test was conducted on a milled portion of the Master Composite material for detoxification test purposes. The sample was detoxed for three hours. The reagent amounts for the detoxification process are presented in Table 13.15.

Table 13.15: Initial Detoxification Testing Amount of Detox Reagents

Slurry Volume, kg	100% H_2O_2, kg/t	100% H_2SO_4, kg/t	$Na_2S_2O_4$, kg/t	$CuSO_4$, kg/t	$Ca(OH)_2$, kg/t
5.0	0.161	1.274	2.80	1.53	1.50

Source: KCA (2013)

From the final solution, 100 ml were split out and analyzed by KCA for pH, total and cyanide WAD (CN_{WAD}), copper and iron. The remaining solution was submitted to Western Environmental Testing Laboratories (WETLABS) for a final detox analysis. The cake was also submitted to WETLABS for meteoric water mobility testing (MWMT).

The similar solution measurements made by both KCA and WETLABS are presented in Table 13.16.

The final detox and MWMT analyses are presented in Table 13.17.

Table 13.16: Initial Detoxification Testing - KCA & WETLABS Solution Detox Analysis

Laboratory	pH	Total Cyanide, mg/L	WAD Cyanide, mg/L	Cu, mg/L	Fe, mg/L
KCA	7.97	0.167	0.109	1.14	0.02
WETLABS	8.15	0.360	0.025	1.10	<0.01

Source: KCA (2013)

Table 13.17: Final Detox Analysis and MWMT – Profile II Analysis

Profile II Wet Chemistry	Units	Final Detox Solution	Final Cake Analysis MWMT
Alkalinity, Total	mg/L as $CaCO_3$	230	60
Bicarbonate	mg/L as $CaCO_3$	290	73
Carbonate	mg/L as $CaCO_3$	<1.0	<1.0
Hydroxide	mg/L as $CaCO_3$	<1.0	<1.0
Aluminum	mg/L	<0.045	0.077
Antimony	mg/L	0.033	0.021
Arsenic	mg/L	0.020	0.0067
Barium	mg/L	0.036	0.041
Beryllium	mg/L	<0.0010	<0.0010
Bismuth	mg/L	<0.10	<0.10
Boron	mg/L	<0.10	<0.10
Cadmium	mg/L	<0.0010	<0.0010
Calcium Chloride	mg/L	270	460
Chloride	mg/L	120	26
Chromium	mg/L	0.045	0.012
Cobalt	mg/L	0.054	0.014
Copper	mg/L	1.1	3.7
Cyanide (WAD)	mg/L	0.025	6.2*
Cyanide (Total)	mg/L	0.36	--
Fluoride	mg/L	7.4	<1.0
Gallium	mg/L	<0.10	<0.10
Iron	mg/L	<0.010	0.28
Lead	mg/L	0.0078	<0.020
Lithium	mg/L	0.12	<0.10
Magnesium	mg/L	12	14
Manganese	mg/L	<0.0050	<0.0050
Mercury	mg/L	0.054	0.089
Molybdenum	mg/L	1.5	0.46
Nickel	mg/L	0.016	0.012

Profile II Wet Chemistry	Units	Final Detox Solution	Final Cake Analysis MWMT
Nitrate + Nitrite Nitrogen	mg/L	9.2	0.98
pH	pH Units	8.15	7.73
Phosphorus	mg/L	<0.50	<0.50
Potassium	mg/L	90	<0.10
Scandium	mg/L	<0.10	<0.10
Selenium	mg/L	0.19	0.053
Silver	mg/L	0.29	6.0
Sodium	mg/L	1300	370
Strontium	mg/L	1.5	1.6
Sulphate	mg/L	3300	2000
Thallium	mg/L	<0.0010	<0.020
Tin	mg/L	<0.10	<0.10
Titanium	mg/L	<0.10	<0.10
Total Dissolved Solids	mg/L	4700	3300
Vanadium	mg/L	0.012	0.021
Zinc	mg/L	<0.010	<0.010

*Note: MWMT significantly higher than detox solution
Source: KCA (2013)

13.5.4.9 Solids/Liquids Separation Testing

Two bottle roll leach tests were conducted on a milled portion of the Master Composite material for solid/liquid separation test purposes. The combined tailings material was subject to detoxification and the slurry was packaged and submitted to Pocock Industrial, Inc. for detoxification analysis and solid/liquid separation analysis. KCA also delivered a “Fresh Milled” sample to Pocock that had not been leached.

13.5.4.10 Pocock Results

SLS tests were conducted on “Fresh Milled” and “Leached & Detoxed” samples delivered from KCA. The purpose for conducting the test work was to generate data for SLS equipment design and sizing criteria. All testing was conducted by Pocock Industrial at their laboratory facility located in Salt Lake City, Utah during October of 2011.

A brief summary of some of the equipment sizing criteria and recommendations gleaned from the testing program follows:

- The minimum flocculant dose anticipated varied by individual sample and thickener type or application desired, but was in the overall range of 35 to 55 g/t for Fresh Milled and 30 to 55 g/t for Leached & Detoxed materials in the tested pH range;
- For conventional thickener sizing Pocock recommended a minimum unit area design basis of 0.30 to 0.40 $m^2/t/d$ for the Fresh Milled, and 0.25 to 0.35 $m^2/t/d$ for the Leached & Detoxed material;

- Dynamic thickening tests conducted on the samples indicated a hydraulic net feed loading rate design basis in the maximum range of 3.1 – 4.3 $m^3/m^2 \cdot hr$ for both the Fresh Milled and Leached & Detoxed materials for optimal performance;
- The overall maximum underflow density range for the Leached & Detoxed material is 53 to 57% based on fully sheared data (but this could be limited to 53 to 55% with rake torque considerations based on un-sheared data); and
- Pressure filter testing for the Leached & Detoxed material based on a tonnage throughput of 1,000 t/d indicates a minimum sizing requirement of about 152 chambers for a horizontal recess plate type press (with 1,500 mm plates, and 15 mm recess (30 mm full chamber)) with no cake wash to achieve 18.3% moisture (or 145 chambers to achieve 18.9% moisture with pH adjusted to 10.5).

13.6 Mineralogy

In December 2006, samples were sent to Carson GeoMin for a mineralogical examination of the mineralized material.

Results are detailed in Carson Geomin Report CB-01-15/12/05 dated December 29, 2005

The following extracts from the Carson Geomin Report are provided for clarity.

“Most of the gold occurs in electrum; SEM analyses done on single electrum grains in two mineralized rock samples and one tailings sample indicate that the electrum at Cerro Blanco probably contains approximately 60% gold and 40% silver.”

“…. a total of 122 electrum grains (locked+liberated+attached/exposed) were observed in the four “ore” composites; 121 of these are in the the <1-30 micron size-range, and one coarser (liberated) grain is 18x104 microns”.

“…. Four tailings samples were observed to contain a total of 10 electrum grains ranging from 1 to 16 microns; all of these are locked in quartz + clay. Electrum grains locked in siliceous gangue therefore contain most of the gold lost to the tailings.

13.7 CyPlus Cyanide Destruction Test Work

The following information is taken from the KCA Updated Feasibility Study of 2013; this data was used to select the detox area process and reagent requirements and consumption.

Three tests reported were Caros Acid, SO_2/Air and a combination of both, CombinOx.

The total cyanide in each of the samples tested was approximately 300mg/L. The CombinOx process was successful in reducing the CN_{WAD} to a concentration of < 0.5 mg/L and < 1.0 mg/L total $CN_{[Total]}$. Table 13.18 provides a summary of the results reported from each of the three cyanide destruction processes tested.

Based on the CyPlus test work the CombinOx process and reagent consumptions as previously reported was the preferred option for the detoxification process.

Table 13.18: Summary of Results of Cyanide Destruction Tests

Method	Solution Analysis, mg/L						Reagent Additions: g/g CN[Wad] x Stoichiometry & Retention Time
	CN[Wad]	CN[Total]	Cu	Ni	Fe	pH	
Caro's Acid, Test No.5							
Feed	301.50	305.10	16.40	0.45	1.28	10.50	100% H_2O_2, 3; 100% H_2SO_4, 6; $Ca(OH)_2$, 9.8;
Final	0.70	1.00	0.07	0.03	0.11	9.10	retention time = 0.5 h
SO_2/Air; Test No.5							
Feed	294.00	305.50	18.40	0.40	4.10	10.50	SO_2, 4.5; $Ca(OH)_2$, 2.1; Cu_2+ ppm, 25;
Final	0.05	1.20	0.10	0.11	0.41	8.50	retention time = 2.0 h
CombinOx, Test No.6							
Feed	305.60	323.71	20.04	0.75	6.48	10.50	100% H_2O_2, 0.5; SO_2, 1.2; 100% H_2SO_4, 3.8;
Final	0.32	0.77	0.40	0.05	0.16	8.50	$Ca(OH)_2$, 3.3; Cu_2+, 12.5 ppm; retention time = 2.5 h

Source: KCA (2013)

13.8 Relevant Results

13.8.1 Grade and Recovery

A review of the test results as shown in Table 13.19 reported by KCA was used to examine the grade and recovery relationship for both gold (Figure 13.2) and silver (Figure 13.3).

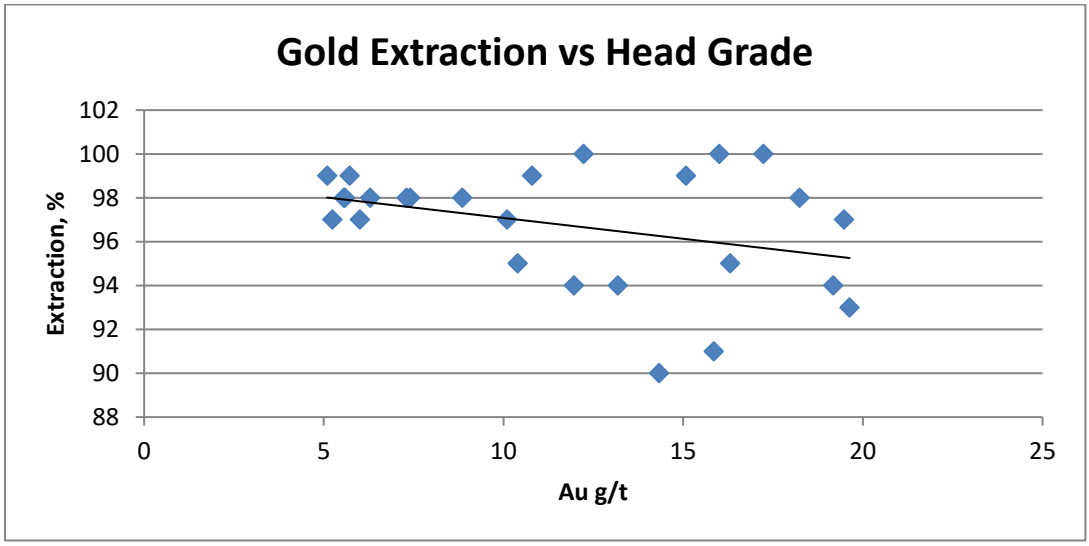
There is a slight correlation between gold head grade and extraction in the head grade range of five to 20g/t Au; however results are very scattered with an extraction range of 64 to 99% for the 96 to hour leach time.

Table 13.19: Gold and Silver Extraction vs. Head Grade

KCA Test #	Head Grade g/t	Gold Extraction, %				Head Grade g/t	Silver Extraction, %			
		24h	48h	72h	96h		24h	48h	72h	96h
34937A	6.00	92.00	96.00	97.00	97.00	26.90	72.00	76.00	77.00	77.00
34937B	7.40	92.00	94.00	96.00	98.00	41.00	69.00	74.00	75.00	75.00
34937D	7.30	93.00	94.00	96.00	98.00	34.70	68.00	69.00	71.00	73.00
34937E	10.10	92.00	93.00	95.00	97.00	48.70	71.00	74.00	74.00	77.00
34938B	5.09	78.00	93.00	96.00	99.00	59.60	70.00	83.00	86.00	89.00
34938A	19.48	82.00	86.00	94.00	97.00	97.70	55.00	71.00	78.00	82.00
34938C	6.29	85.00	93.00	96.00	98.00	71.20	71.00	85.00	87.00	89.00
34939B	19.17	61.00	85.00	100.00	94.00	136.90	34.00	67.00	75.00	83.00
34939C	13.18	52.00	70.00	89.00	94.00	97.30	58.00	71.00	79.00	87.00
34940D	5.58	92.00	97.00	97.00	98.00	4.70	74.00	76.00	79.00	79.00
34940E	10.79	82.00	96.00	97.00	99.00	28.30	75.00	76.00	80.00	86.00
34942C	16.00	95.00	97.00	100.00	100.00	31.20	84.00	85.00	87.00	88.00
34942E	5.24	76.00	96.00	97.00	97.00	55.60	90.00	94.00	96.00	97.00
34943A	12.23	99.00	96.00	98.00	100.00	12.50	65.00	69.00	69.00	71.00
34943B	8.85	91.00	93.00	96.00	98.00	8.40	78.00	82.00	82.00	84.00
34943E	5.72	96.00	98.00	98.00	99.00	55.50	73.00	83.00	83.00	84.00
24944A	17.23	95.00	96.00	97.00	100.00	151.40	65.00	82.00	88.00	93.00
34944C	10.39	89.00	96.00	95.00	95.00	33.20	59.00	76.00	82.00	88.00
34944D	16.31	93.00	96.00	98.00	95.00	15.10	83.00	83.00	84.00	89.00
34945A	19.63	87.00	90.00	88.00	93.00	112.00	40.00	54.00	59.00	70.00
34945B	5.56	87.00	97.00	97.00	98.00	26.20	48.00	57.00	59.00	64.00
34946C	11.96	95.00	100.00	98.00	94.00	7.60	84.00	86.00	86.00	86.00
34946D	15.85	87.00	89.00	93.00	91.00	11.60	84.00	86.00	86.00	88.00
34946E	18.24	59.00	92.00	97.00	98.00	59.20	61.00	81.00	89.00	91.00
34947A	14.33	75.00	89.00	88.00	90.00	258.70	38.00	61.00	76.00	89.00
34947E	15.08	56.00	86.00	94.00	99.00	18.00	81.00	85.00	92.00	97.00
Average	**11.65**	**83.90**	**92.60**	**95.70**	**96.80**	**57.82**	**67.20**	**76.30**	**79.90**	**83.60**

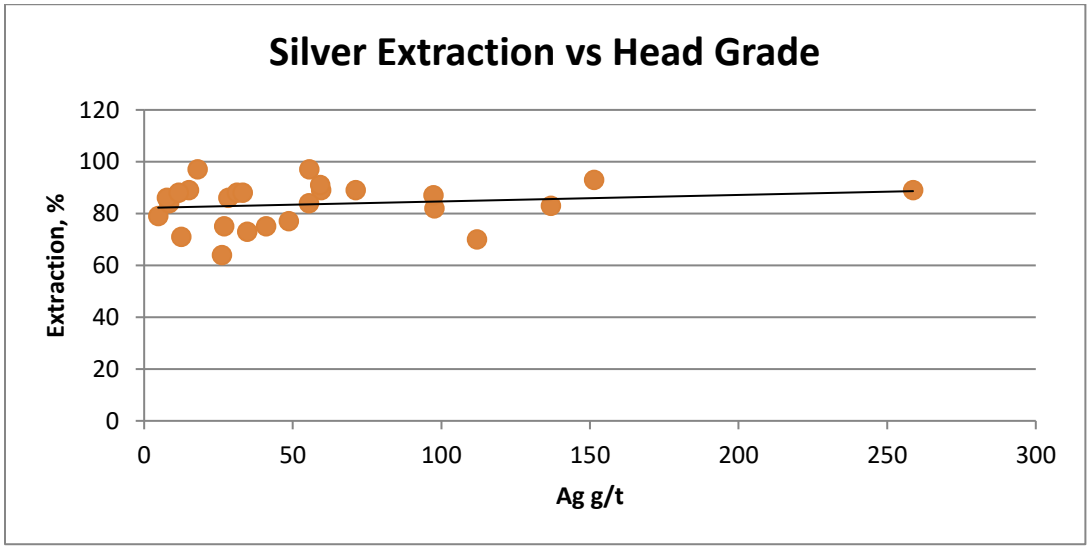
Source: KCA (2013)

Figure 13.2: Gold Extraction versus Head Grade (Au)



Source: JDS (2017)

Figure 13.3: Silver Extraction versus Head Grade (Ag)



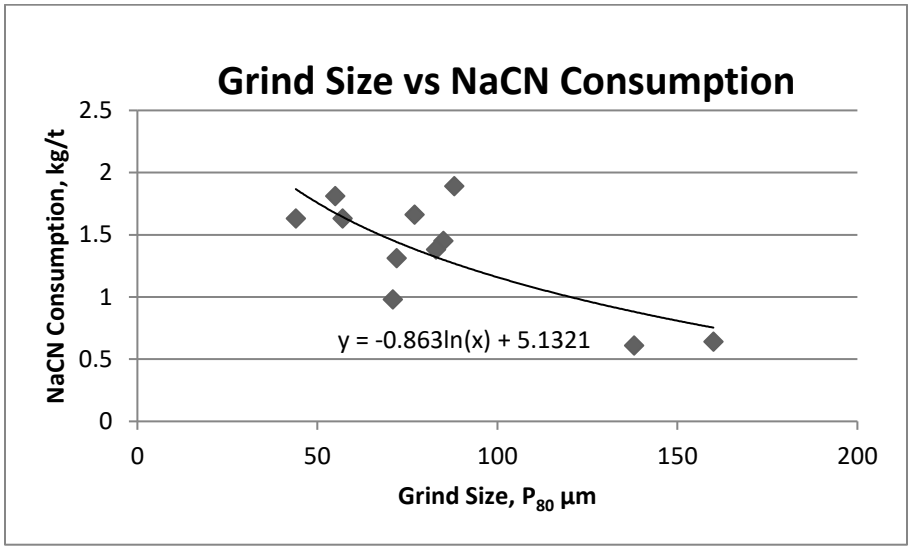
Source: JDS (2017)

13.8.2 Cyanide Consumption

A review was undertaken by JDS of the data reported by KCA from a series of test programs to find a correlation between grind size and NaCN consumption. The test work data that was specifically analyzed was for samples with gold grades in the range 5 to 20 g/t, this being considered the lower and upper limits of mined mineralized material delivered to the process plant, the reported expected mine grade being approximately 8 to 12 g/t gold. The leach time for these tests was 96 hours.

Data reported in Table 13.22 was used to develop a graphical representation of NaCN consumption versus grind size (see Figure 13.4). The range of grind size was P_{80} 44 to 160 µ and a range in NaCN consumption of 0.61 to 1.89 kg/t

Figure 13.4: Grind Size versus NaCN Consumption



Source: JDS (2017)

The NaCN consumption was calculated at various grind sizes as shown in Table 13.20. Based on a grind size P_{80} of 75 µm the estimated NaCN consumption is 1.41 kg/t.

Table 13.20: Cyanide Consumption at Various Grind Sizes

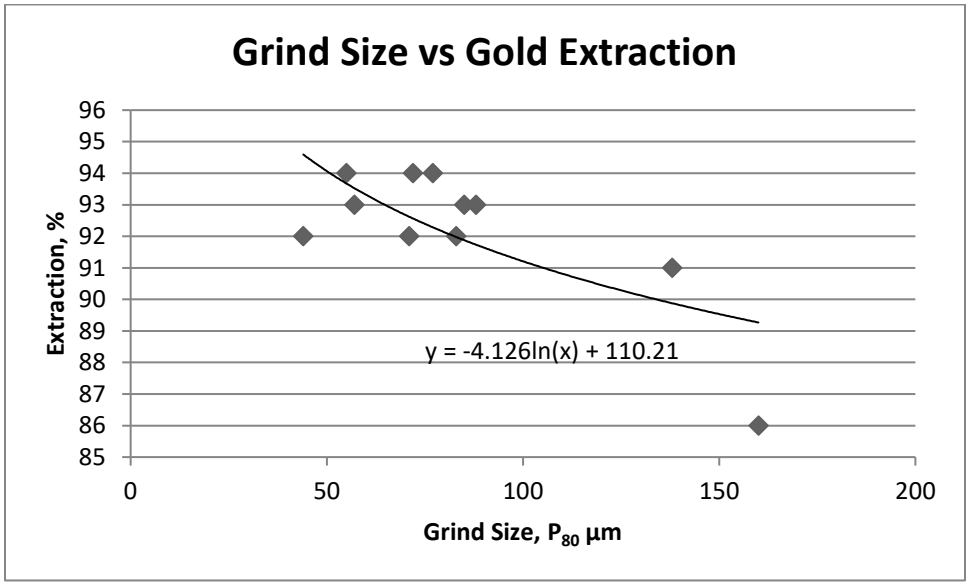
Grind Size, P_{80} µm	Calculated NaCN kg/t
60	1.60
75	1.41
90	1.25

Source: JDS (2017)

13.8.3 Grind Size

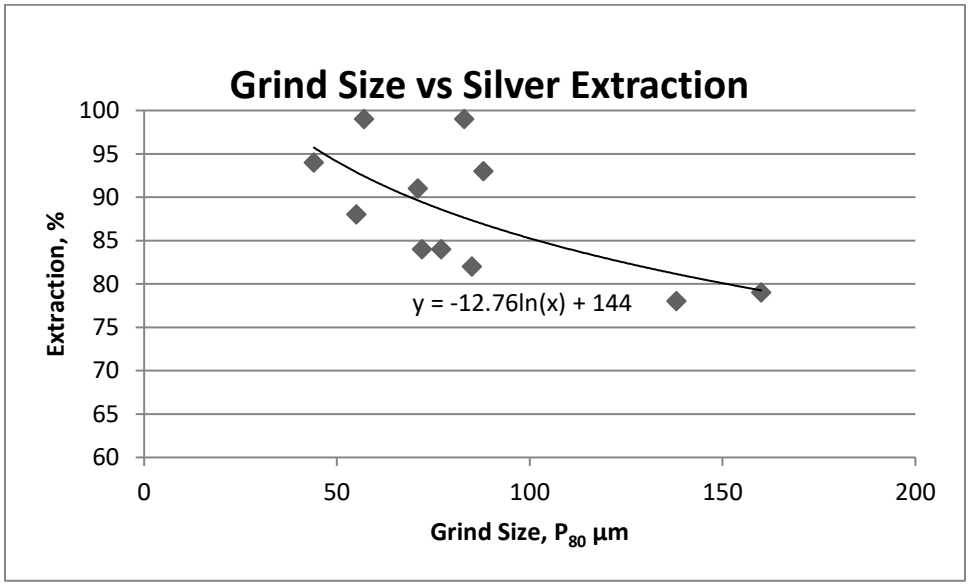
Test work data was used to develop grind versus extraction graphs for both gold and silver. Figure 13.5 and Figure 13.6 show the relationship between grind size and extraction for gold and silver, respectively.

Figure 13.5: Grind Size versus Gold Extraction



Source: JDS (2017)

Figure 13.6: Grind Size versus Silver Extraction



Source: JDS (2017)

Based on the above data, the gold and silver recoveries were calculated for 60, 75 and 90 micron grind size respectively, Table 13.21.

Table 13.21: Calculated Gold and Silver Extractions at Various Grind Sizes

Microns	Au Extraction %	Ag Extraction %
60	93.3	91.8
75	92.4	88.9
90	91.6	86.6

Source: KCA (2013)

13.8.4 Leach Time

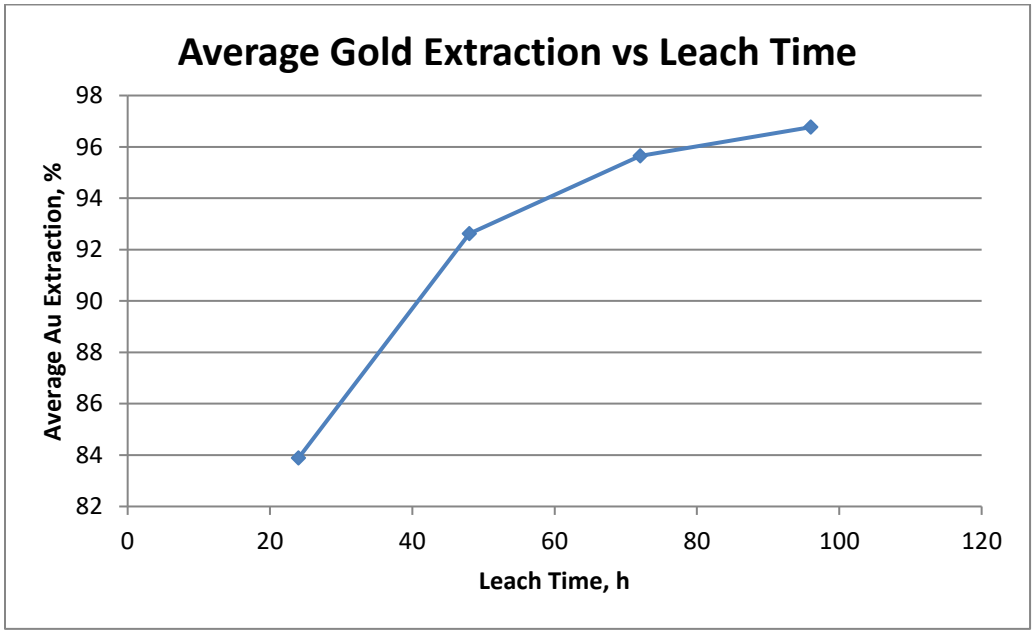
Test results reported by KCA in the Cerro Blanco 2013 Update Feasibility Study report were reviewed by JDS. For the purposes of this PEA, test work data that was specifically analyzed was for samples with gold grades between 5g/t and 20 g/t, this being in line with the expected mine grade of approximately 8 to 12 g/t and a leach time of 96 hours. Some results indicated 100% gold/silver recovery and this data was considered unreliable and thus excluded from this analysis.

The reported grind size for these tests was approximately 92% passing 75 µm, which is estimated to be an approximate P_{80} of 60 µm.

These results confirm that a leach time of approximately 96 hours is required to achieve gold extractions of greater than 90%.

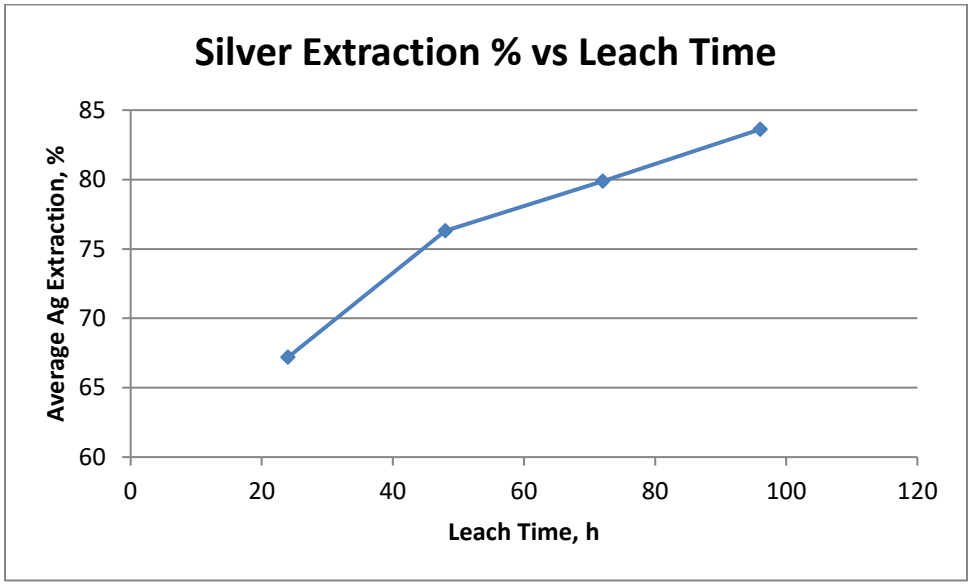
The average gold and silver extractions are presented graphically in Figure 13.7 and Figure 13.8.

Figure 13.7: Average Gold Extraction versus Leach Time (<20g/t Au grade)



Source: JDS (2017)

Figure 13.8: Silver Extraction versus Leach Time



Source: KCA (2013)

13.8.5 Mercury

The mercury values reported in the Profile II and MWMT analyses from the detoxification test data were considered high. Therefore, due to the uncertainty and limited amount of samples tested for mercury, a retort has been included in the process plant flowsheet (refinery area).

13.9 Process Design Criteria and Metallurgical Projections

Based on a review of the metallurgical test work as reported by KCA, the following process design criteria were developed by JDS for the Cerro Blanco Project. Table 13.23 summarizes the metallurgical projection and design parameters.

Table 13.22: Metallurgical Projection and Major Process Design Parameters

Parameter	Unit	Value
Gold Grade	g/t	4.74*
Silver Grade	g/t	18.2*
Average Gold Extraction	%	92.4
Average Silver Extraction	%	88.9
Soluble Losses	%	1
Gold Recovery	%	91.4
Silver Recovery	%	87.9
Ball Mill Work Index (BWi)	kWh/t	19.9
Abrasion Index	g	0.2399 (max 0.33)
Target Grind	P_{80} µm	75
Leach Retention Time	hours	96
Thickener Underflow Density	%	50 - 55
NaCN Consumption in Leach	kg/t	1.41
Leach Lime Consumption,	kg/t	1.00
Flocculant Consumption per Thickener	kg/t	50

*Note: The gold and silver grades were based on the mine design by JDS, 2017
Source: JDS (2017)

14 Mineral Resource Estimates

14.1 Introduction

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

The Cerro Blanco deposit comprises several steeply dipping zones extending a strike length of more than 1,200 m and to a depth of more than 450 m. Solids were created based on mineralized intersections to create numerous zones in two sets, orienting at azimuths of 110° and 140°, which are used to constrain the interpolation of gold and silver grades.

The updated Mineral Resource Estimate incorporates more than 517 drill holes totalling 114,992 m. There is more than 1.24 Moz of gold and 4.47 Moz of silver contained in the Indicated Mineral Resources. The Project also contains more than 0.04 Moz of gold and 0.03 Moz of silver in the Inferred Mineral Resource category.

The Mineral Resource Estimate for the Cerro Blanco deposit is reported at a base case above a 3.75 g/t Au cut-off, as tabulated below in Table 14.1.

Table 14.1: Mineral Resource Statement above 3.5 Au Cut-off – effective date February 7, 2017

Classification	Tonnes ('000)	Au (g/t)	Ag (g/t)	Au Oz ('000)	Ag Oz ('000)
Indicated	3,731	10.22	36.50	1,242	4,468
Inferred	243	6.02	4.20	47	33

Source: Kirkham Geosystems (2017)

The mineral resource statement is subject to the following:

- 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 (NI 43-101), with an effective date of February 7, 2017;
- Mineral resources reported demonstrate reasonable prospect of eventual economic extraction, as required under NI 43-101. Mineral resources are not Mineral Reserves and do not have demonstrated economic viability;
- Underground Mineral Resources are reported at a cut-off grade of 3.75 g/t Au. Cut-off grades are based on a price of US$1,250/oz gold, US$16/oz silver and a number of operating cost and recovery assumptions, plus a contingency;
- Numbers are rounded; and
- The Mineral Resources may be affected by subsequent assessment of mining, environmental, processing, permitting, taxation, socio-economic and other factors.

14.1.1 Data

The drill hole database was supplied in electronic format (i.e. MS Excel and Access) by Goldcorp. This included collars, down hole surveys, lithology data and assay data (i.e. Au g/t, Ag g/t and down hole from and to intervals in metric units). Lithology data was provided as lithology group and description along with abbreviated alpha-numeric and numeric codes as described in Table 14.2.

Table 14.2: Table of Lithology Units and Codes

LITH	LITH CODE	LITH Group	LITH Description
Qc	10	Post-Mineral Cover Rock - Quaternary	Colluvium
Qb	11		Basalt Flows
Bi	20	Cross-Cutting Rocks	Basaltic Intrusive Dikes and/or plugs
Cbx	30		Collapse Breccia
Gr	40		Granite
ad	50		Andesite Dike
Rp	60		Quartz Eye Rhyolite
Vt	70		Vein
Stock	71		Stockwork
Hbx	72		Hydrothermal Breccia
RF	80		Rhyolite Flow
Ss	90	Salinas Group	Sinter
Svc	91		Volcanic Sediments
Srt	92		Quartz Eye Rhyolite
Sfx	93		Treatic Breccia
Slt	94		Siltstone
Sct	95		Ash Tuff
Scgl	96		Conglomerate
Mss	100	Mita Group	Sandstone
Mat	101		Adesite Tuff
Mlt	102		Crystal Tuff
Mbt	103		Lapilli Tuff
Msc	104		Calcareous Limestone
Mls	105		Limestone
Mcv	106		Quartz Latite Crystal Lithic Tuff
Mcgl	0		Conglomerate
Tcb	110	Tempisque Volcanic Complex	Basalt-dominated
Tca	111		Andesite-dominated

Source: Kirkham Geosystems (2017)

A total of 517 drill holes were supplied for the complete Cerro Blanco Project area. Validation and verification checks were performed during import to insure no overlapping intervals, typographic errors and anomalous entries. None were found.

14.1.2 Data Analysis

Table 14.3 shows statistics of gold and silver assays for each of the lithologic units listed in Table 14.2. There are 75,667 gold assays or 112,112 m total which average 0.57 g/t and 109,014 silver assays or 109,014 m total which average 3.66 g/t. It is interesting to note that the maximum gold grade is 434.8 g/t while the maximum silver grade is 8,656.7 g/t.

Table 14.3: Statistics for Weighted Gold and Silver Assays for the Cerro Blanco Deposit

	Au/Ag	#Values	Length (m)	Maximum (g/t)	Mean (g/t)	CV
Ob	Au	118	199	0.5	0.03	2.3
	Ag	118	199	2.0	1.03	0.2
Qc	Au	679	1,106	1.4	0.05	2.3
	Ag	646	1,057	16.0	0.96	1.0
Qb	Au	129	214	-	0.01	0.3
	Ag	129	214	1.0	0.83	0.4
Bi	Au	1	2	-	0.02	-
	Ag	1	2	1.0	1.00	-
Cbx	Au	2,647	3,510	166.1	0.64	6.8
	Ag	2,641	3,500	265.2	3.38	3.0
Gr	Au	371	682	0.2	0.01	1.5
	Ag	371	682	2.3	0.81	0.4
Ad	Au	1,496	2,257	314.0	0.45	13.9
	Ag	1,424	2,149	801.2	2.84	7.7
Rp	Au	1,935	2,847	8.1	0.21	2.3
	Ag	1,846	2,717	100.0	1.86	2.6
Vt	Au	1,950	2,129	434.8	9.71	2.7
	Ag	1,949	2,128	4,677.8	44.17	4.1
Stock	Au	1,400	1,936	148.7	1.47	4.1
	Ag	1,400	1,936	336.0	7.39	2.7
Hbx	Au	477	547	266.1	1.06	10.8
	Ag	477	547	969.0	6.37	6.7
RF	Au	2,845	4,486	96.4	0.34	8.1
	Ag	2,729	4,316	8,656.7	7.36	23.4
Ss	Au	3,413	5,378	15.7	0.30	2.0
	Ag	3,192	5,046	187.8	1.80	2.5
Svc	Au	11,860	17,657	60.6	0.45	3.0
	Ag	11,343	16,908	1,346.9	3.43	4.2

	Au/Ag	#Values	Length (m)	Maximum (g/t)	Mean (g/t)	CV
Srt	Au	1,080	1,420	16.5	0.28	2.2
	Ag	1,080	1,420	88.0	2.57	2.0
Sfx	Au	1,368	2,343	194.7	0.34	10.7
	Ag	1,367	2,341	267.4	2.59	4.3
Slt	Au	249	392	9.2	0.41	2.2
	Ag	181	283	74.0	2.07	3.3
Sct	Au	203	340	2.0	0.18	1.7
	Ag	198	333	18.0	1.42	1.3
Scgl	Au	1,091	1,727	157.4	0.64	5.2
	Ag	1,080	1,714	1,552.0	3.94	8.0
Mss	Au	6,241	9,464	368.3	0.26	15.3
	Ag	6,040	9,167	2,405.9	2.12	12.3
Mat	Au	2,858	4,573	105.6	0.46	7.6
	Ag	2,783	4,472	467.3	2.85	5.2
Mlt	Au	3,136	4,590	56.6	0.37	5.6
	Ag	3,091	4,521	348.7	2.32	4.4
Mbt	Au	9,769	14,096	99.9	0.50	5.0
	Ag	9,376	13,541	1,076.3	3.75	5.3
Msc	Au	2,945	3,944	65.9	0.33	7.3
	Ag	2,945	3,944	624.6	2.23	5.4
Mls	Au	1,496	2,064	46.0	0.44	5.6
	Ag	1,459	2,009	280.9	2.96	3.5
Mcv	Au	15,250	23,079	287.1	0.31	10.2
	Ag	14,998	22,738	430.1	1.58	4.2
Tcb	Au	31	49	-	-	0.9
	Ag	31	49	1.0	0.39	1.0
Tca	Au	615	1,067	1.4	0.03	3.1
	Ag	615	1,067	13.0	0.79	1.2
Total	**Au**	**75,653**	**112,099**	**434.8**	**0.57**	**8.5**
	Ag	**73,510**	**109,000**	**8,656.7**	**3.65**	**12.5**
All	**Au**	**75,667**	**112,112**	**434.8**	**0.57**	**8.5**
	Ag	**73,524**	**109,014**	**8,656.7**	**3.66**	**12.5**

Source: Kirkham Geosystems (2017)

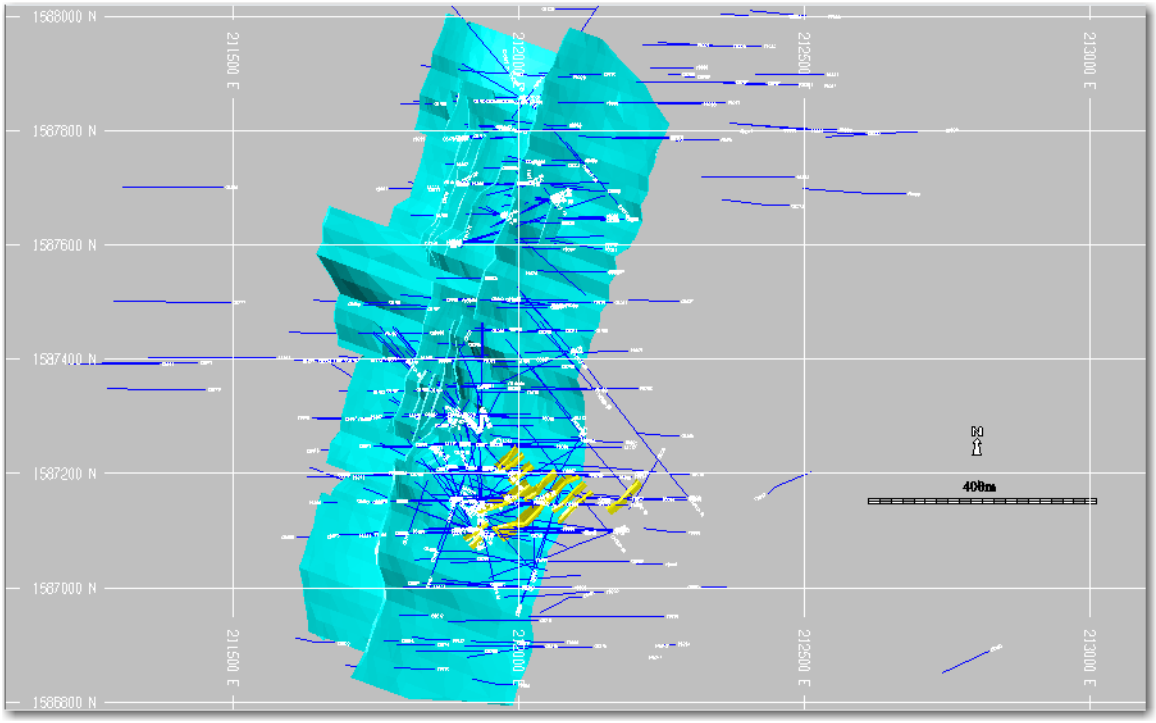
The table above shows intervals that intersect the high grade which are primarily encountered within the Vt or code 70, as would be expected. The Vt unit has 1,950 assay intersections, with a total 2,129 m and average grade of 9.71 g/t Au and 44.17 g/t Ag. The coefficient of variation is relatively high with 2.7 for Au and 4.1 for Ag.

14.1.3 Geology Model

A solid model of the vein zones within the Cerro Blanco deposit as shown in plan (Figure 14.1) and section (Figure 14.2) was supplied by Goldcorp, which created from sectional interpretations which subsequently wireframed, and were based on a combination of lithology and gold grades.

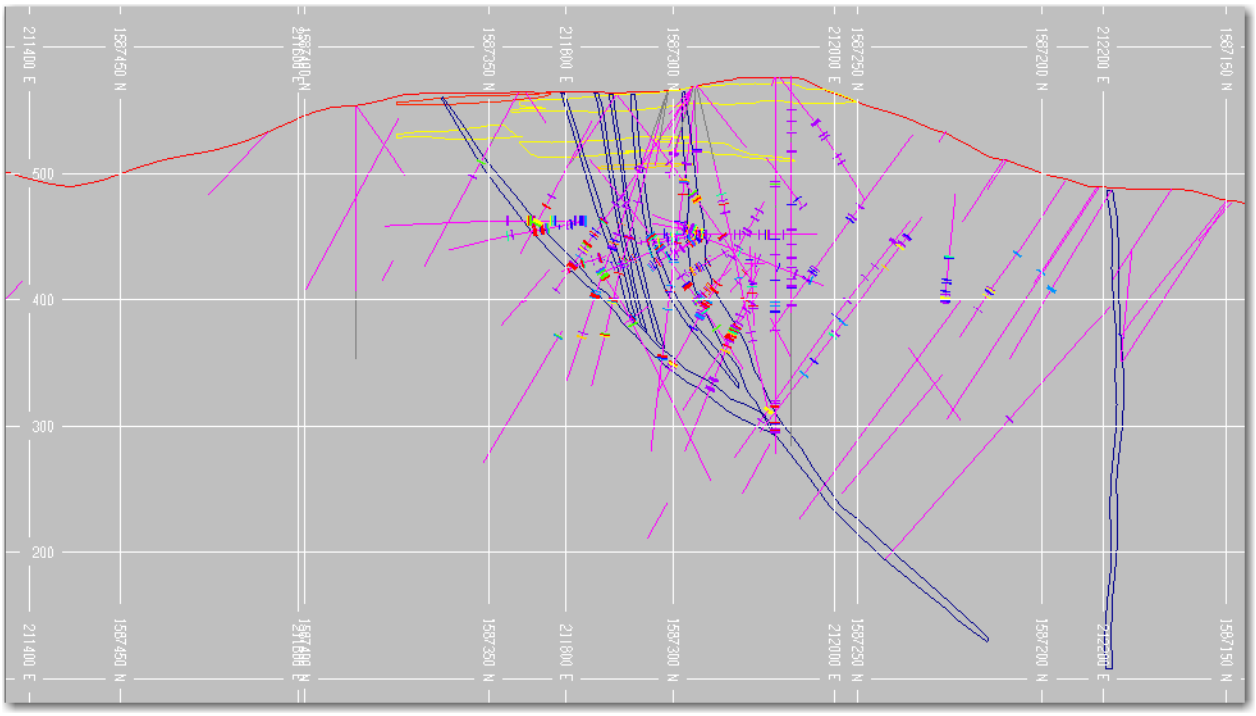
Intersections were inspected and the solid was then manually adjusted to match the drill intercepts. Once the solid models were edited and complete, they were used to code the drill hole assays and composites for subsequent statistical and geostatistical analysis. The solid zones were utilized to constrain 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 derived from the strike and dip of the mineralized zone.

Figure 14.1: Plan View of Drill Holes and Vein Solids (light blue=main and splays veins, yellow= 140 veins)



Source: Kirkham Geosystems (2017)

Figure 14.2: Section View of Drill Holes, Vein Solids and Sinter (yellow)



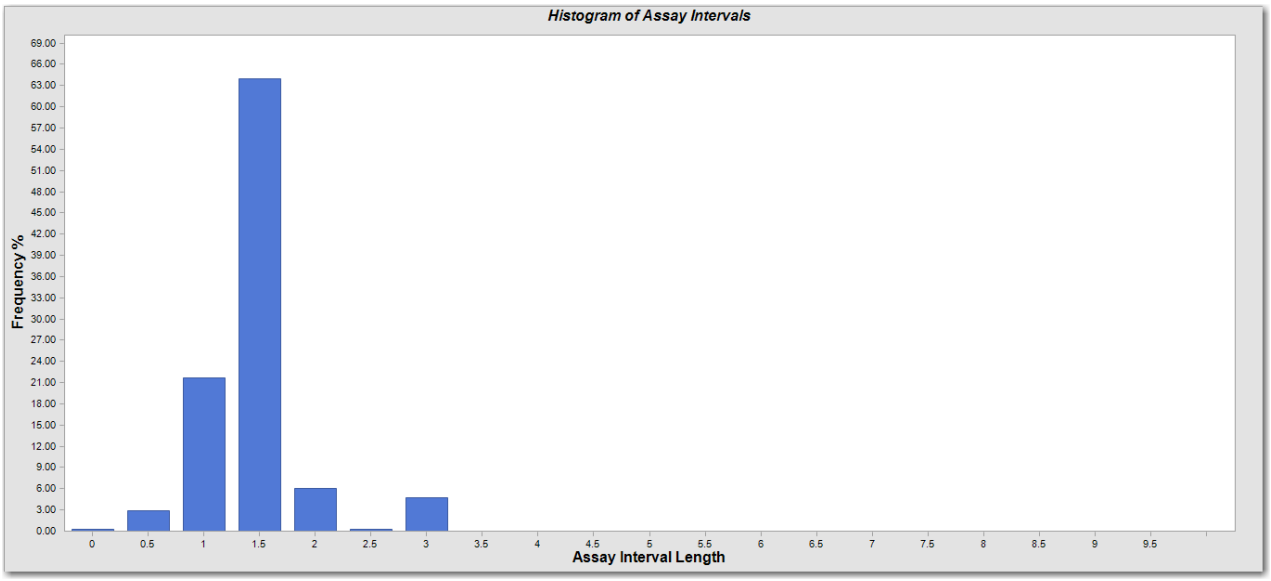
Source: Kirkham Geosystems (2017)

The solids were coded into the composite database by priority, with the vein solids being highest, then the sinter unit and lastly the surrounding waste.

14.1.4 Composites

It was determined that the 2 m composite lengths offered the best balance between supplying common support for samples and minimizing the smoothing of the grades. Figure 14.3 shows a histogram illustrating the distribution of the assay interval lengths with 63% of the data having interval lengths of 1.5 m. The 2 m sample length also was consistent with the distribution of sample lengths. It should be noted that although 2 m is the composite length, any residual composites of length greater than 1 m and less than 2 m remained to represent a composite, while any composites residuals less than 1 m were combined to the composite above.

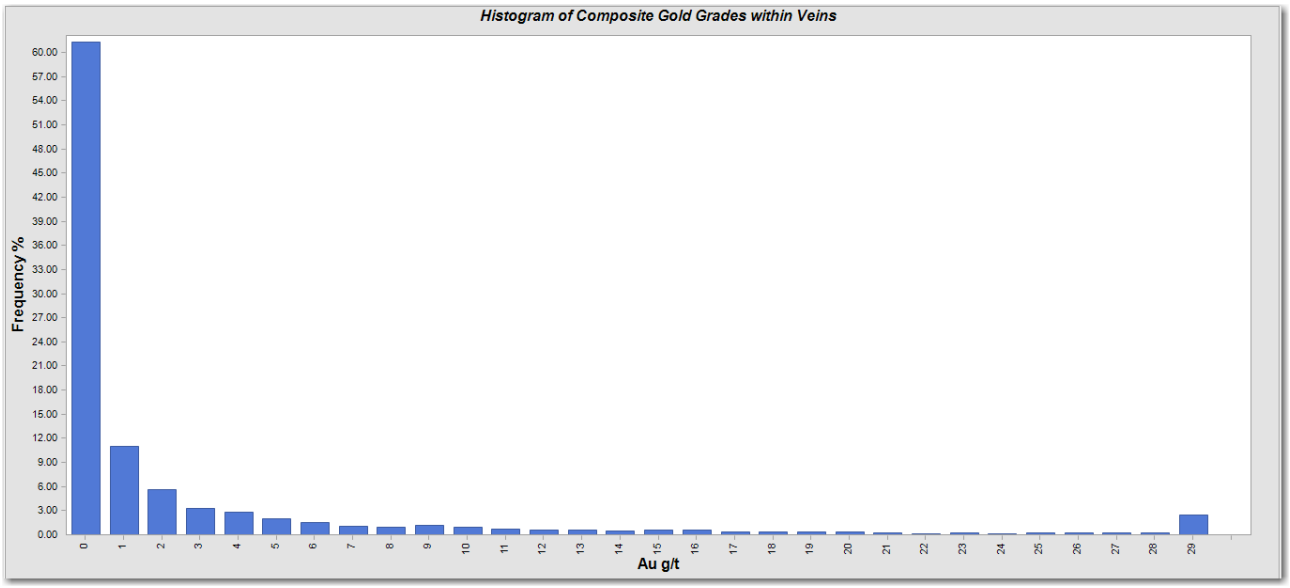
Figure 14.3: Histogram of Assay Interval Lengths



Source: Kirkham Geosystems (2017)

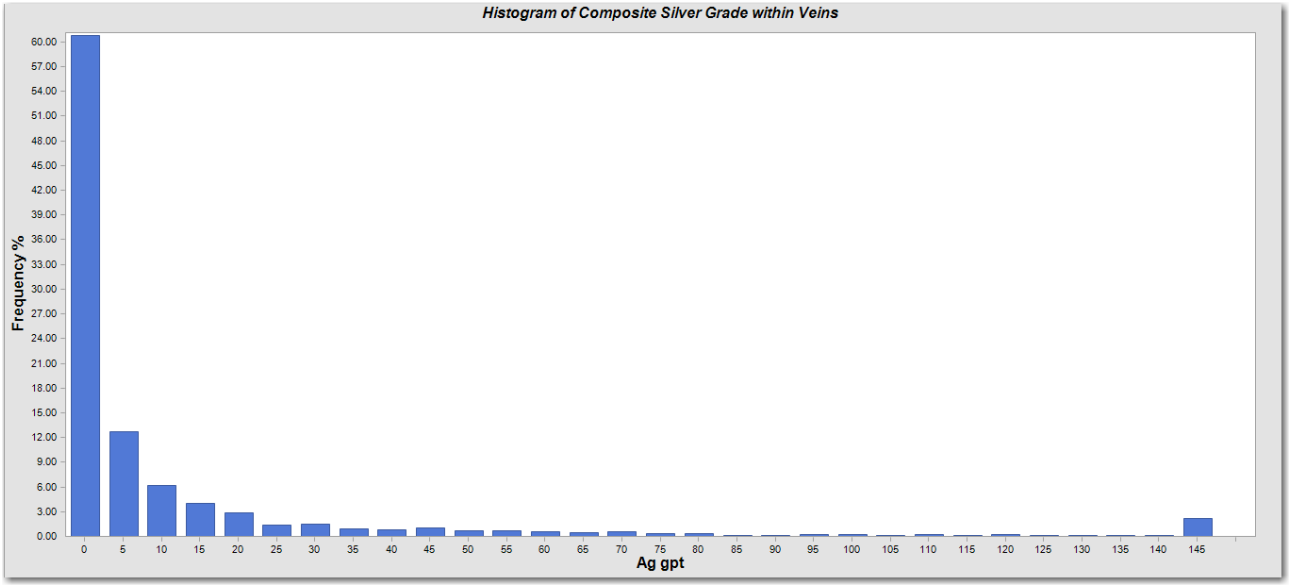
Figure 14.4 and Figure 14.5 show histograms of the gold and silver composite values, respectively. The composite data demonstrate log-normal distributions in both cases.

Figure 14.4: Histogram of Au Composite grades with Vein Zones



Source: Kirkham Geosystems (2017)

Figure 14.5: Histogram of Ag Composite grades with Vein Zones



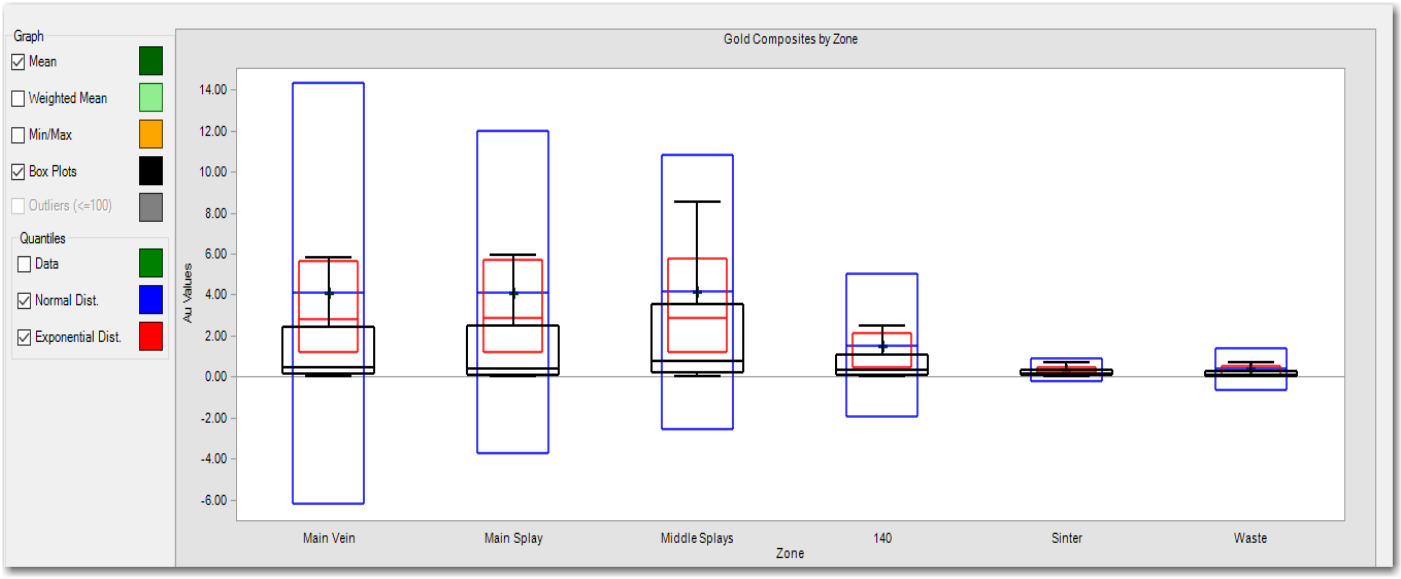
Source: Kirkham Geosystems (2017)

Figure 14.6 and Table 14.4 show the box plots for the grouped composite data, which were then subsequently grouped for the estimation process. It is evident that although the Main vein, Main Splay and the Middle Splays are very similar and could be considered to be the same unit. Table 14.4 and Table 14.5 show the basic statistics for the 2 m Au and Ag composite grades within the mineralized domains, respectively.

There are a total of 3,968 composites with the 110 Veins (Main, Main Splay and Middle Splays) and 538 composites in the 140 Veins. Note that the term 110 and 140 are indicative of the strike direction of each of the groups. The sinter unit has 2,117 composites and the waste has 36,911 composites.

The weighted average Au grades for the 110 zones range between 4.11 g/t to 4.19 g/t with CV's being between 2.37 and 3.72. Also for 110 Veins, Ag grades range from 16.14 g/t and 24.08 with CV's being between 3.21 to 5.08. The weighted average Au grades for the 140 zones 1.56 g/t with CV being between 3.35 while Ag grades average 6.46 g/t with CV being 2.37. CV's or variability is typically high for precious metal deposits primarily due to the nugget nature particularly within epithermal veins; however, the CV's in this case are still relatively high. Grade limiting or cutting will further reduce the CV's.

Figure 14.6: Box Plot of Au Composites



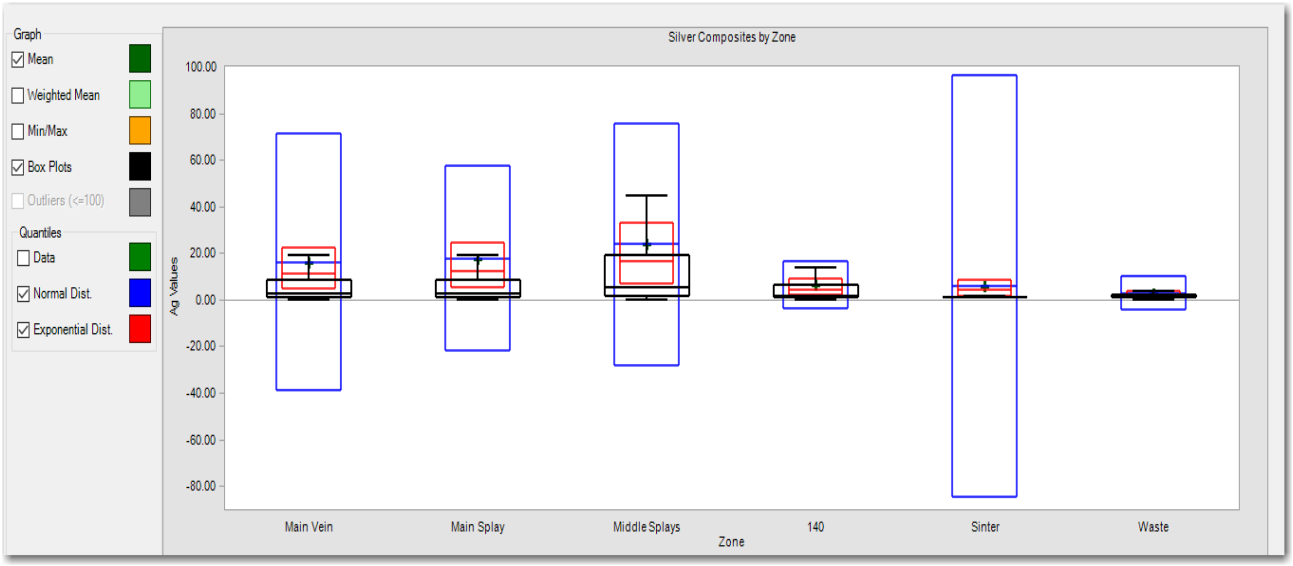
Note: Blue box is Normal distribution; Red box is log-normal; black is data range with mean grade
Source: Kirkham Geosystems (2017)

Table 14.4: Au Composite Statistics Weighted by Length

	Main Vein	Main Splay	Middle Splays	140	Sinter	Waste
Num Samples	1,622	808	1530	538	2117	36,911
Min (g/t)	0.002	0.005	0.002	0.01	0	0
Max (g/)	350.535	122.2	155.05	69.921	26.33	113.646
Mean (g/t)	4.0688	4.1166	4.1457	1.5265	0.3134	0.3779
Weighted Mean (g/t)	4.1076	4.1306	4.1866	1.557	0.3092	0.3765
Weighted CV	3.7176	2.7741	2.3662	3.3543	2.5043	3.832

Source: Kirkham Geosystems (2017)

Figure 14.7: Box Plot of Ag Composites



Note: Blue box is normal distribution; red box is log-normal; black is data range with mean grade
Source: Kirkham Geosystems (2017)

Table 14.5: Ag Composite Statistics Weighted by Length

Unit	Main Vein	Main Splay	Middle Splays	140	Sinter	Waste
Number of Samples	1613	801	1,519	502	1979	35,996
Min (g/t)	0.1	0.1	0.1	0.1	0.1	0
Max (g/t)	2,401.20	937.6	2,157.30	171.6	5,527.40	1,192.00
Mean (g/t)	15.99	17.53	23.82	6.37	5.96	2.72
Weighted Mean (g/t)	16.14	17.47	24.08	6.46	6.04	2.71
Weighted CV	5.08	3.33	3.21	2.37	22.47	3.9

Source: Kirkham Geosystems (2017)

The box plots show that only the 900 zone and the Inferno deposit (200 zone) are of potentially economically mineable grades especially in consideration of the mining cut-off grades employed at Minto Mine. The box plots also show that the 900 zone and Inferno deposit do not appear to be related however, the 100, 300, 500 and 800 could potentially have some correlation.

14.1.5 Evaluation of Outlier Assay Values

During the estimation process, the influence of composites greater than 140 g/t Au, and 600 g/t Ag have been limited as illustrated by the breaks in the cumulative frequency plots shown in Figures 14.7 and 14.8, respectively.

Table 14.6: Au and Ag Values Used for Grade Limiting

Zone	Value	g/t
110	Au Cut	140
	Ag Cut	600
140	Au Cut	25
	Ag Cut	80
Sinter	Au Cut	5
	Ag Cut	60
Waste	Au Cut	40
	Ag Cut	400

Source: Kirkham Geosystems (2017)

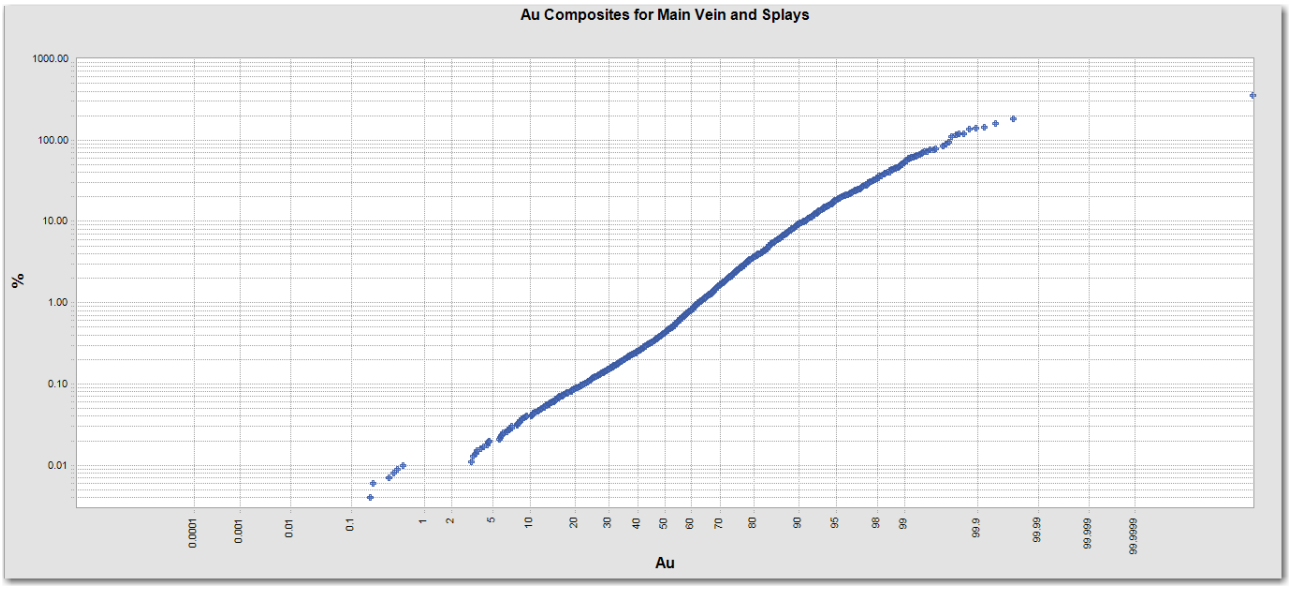
In the case of the Au composites, values higher than 5.2% were cut, while in the case of Ag, grades greater than 13 g/t Ag and 3 g/t Au were cut, as illustrated by the breaks in the cumulative frequency plots shown in Figures 14.8 and 14.9, respectively. The distance restriction for grade limiting was 20 m range. In other words, composite grades greater than the threshold amounts were not used in the estimation of blocks, if those high-grade composites are outside the respective distance from that block.

Tables 14.7 and 14.8 show the effect of cutting the outlier grades to the level recommended above. This shows that the average copper grades are reduced by 4% and the CV's are reduced to ≤ 1.0, illustrating that the grade limiting strategy is effective. Similar results for gold and silver support the grade limiting strategy for those elements also.

Table 14.7: Cut vs. Uncut Comparisons

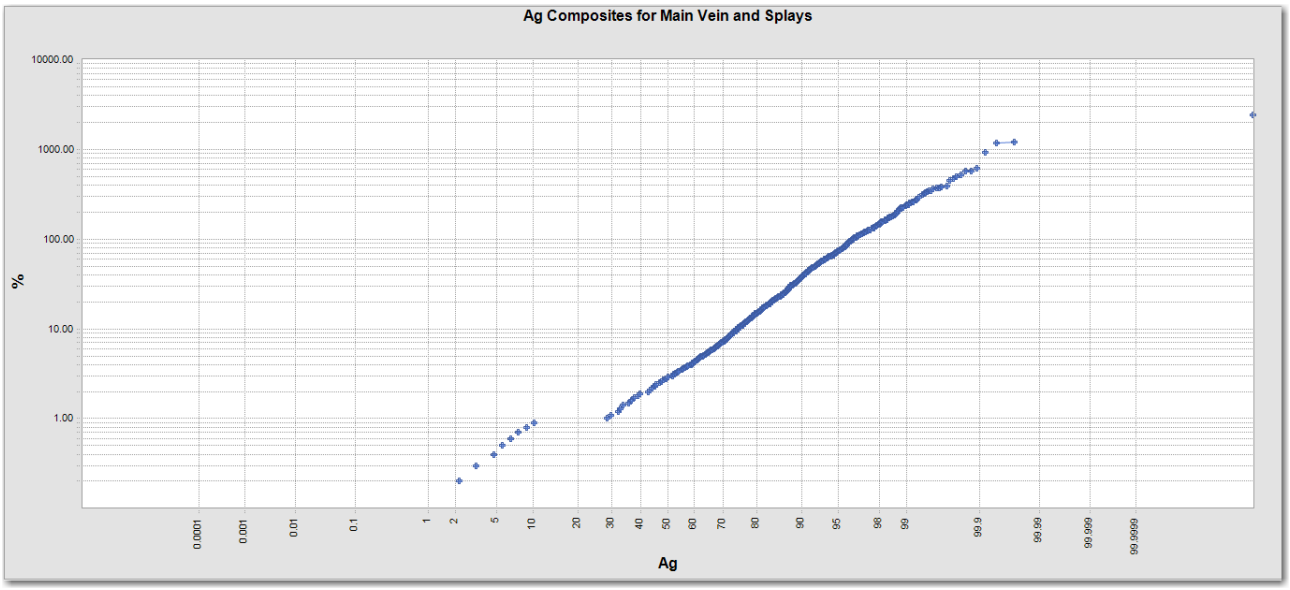
Zones	Value	Maximum (g/t)	Mean (g/t)	CV	%Diff
110	Au	350.54	4.14	3.1	
	Ag	2,401.2	19.5	3.9	
	Au Cut	140	4.06	2.8	-1.9
	Ag Cut	600	18.1	2.8	-6.9
140	Au	69.9	1.56	3.4	
	Ag	171.6	6.5	2.4	
	Au Cut	25	1.37	2.5	-11.8
	Ag Cut	80	6.0	2.0	-6.7
Sinter	Au	26.33	0.31	2.5	
	Ag	5,527.4	6.0	22.5	
	Au Cut	5	0.29	1.5	-5.4
	Ag Cut	60	1.8	2.3	-70.9
Waste	Au	113.65	0.38	3.8	
	Ag	1,192.0	2.7	3.9	
	Au Cut	40	0.37	3.4	-0.7
	Ag Cut	400	2.7	3.0	-0.7

Figure 14.8: Au Cumulative Frequency Plot



Source: Kirkham Geosystems (2017)

Figure 14.9: Ag Cumulative Frequency Plot



Source: Kirkham Geosystems (2017)

14.1.6 Specific Gravity Estimation

Table 14.8 shows the SG assignment by zone. It is recommended that future work programs should include SG measurements to expand the density distributions. The SG are assigned by vein which includes all 110 and 140 veins, the sinter unit and then the host rocks, which were assigned code 97 and 98. In addition, there is a silica cap that overprints the deposit which has a relatively low specific gravity of 2.1 gm/mm^3.

Table 14.8: SG Zone Assignments

Domain/Rock	Density (gm/mm^3)
Veins	2.5
Sinter	2.54
Silica cap	2.1
Host rocks	2.59

Source: Kirkham Geosystems (2017)

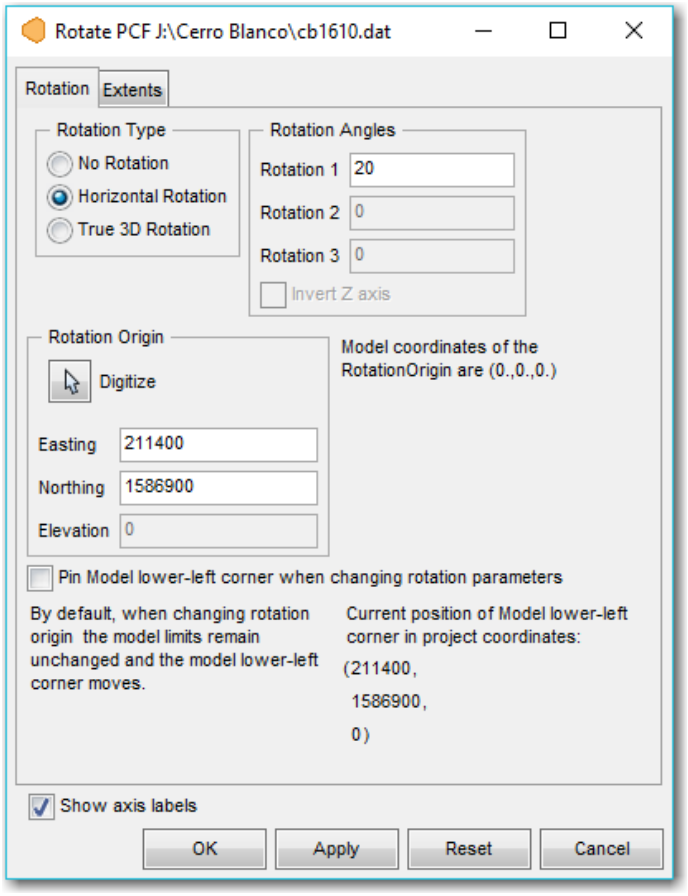
14.1.7 Variography

Experimental variograms and variogram models in the form of correlograms were generated for Au and Ag grades. However, the individual zones do not have sufficient data to generate meaningful variogram results. For this reason, it was decided at this time to use inverse distance to the third power as the interpolator.

14.1.8 Block Model Definition

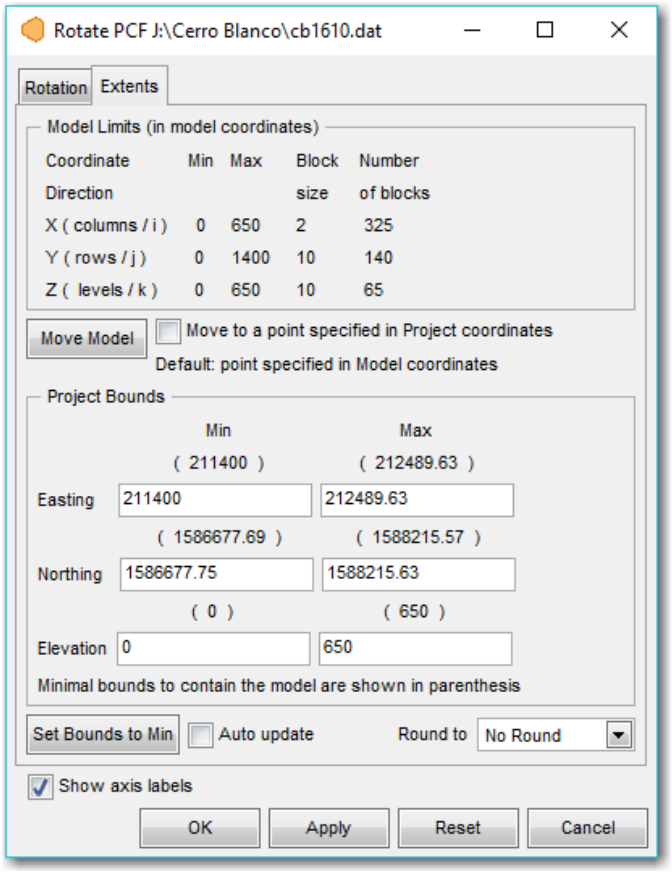
The block model used for estimating the resources was defined according to the origin and orientation as shown in Figure 14.10 and the limits specified in Figure 14.11. The block model is orthogonal and non-rotated reflecting the orientation of the deposit. The block size chosen was 10 m by 10 m by 2 m, roughly reflecting drill hole spacing. Note that MineSight^{TM} uses the centroid of the blocks as the origin.

Figure 14.10: Origin and Orientation for the Cerro Blanco Block Model



Source: Kirkham Geosystems (2017)

Figure 14.11: Extents and Dimensions for the Cerro Blanco Block Model



Source: Kirkham Geosystems (2017)

14.1.9 Resource Estimation Methodology

The estimation plan includes the following items:

- Mineralized zone code and percentage of modelled mineralization in each block;
- Estimated SG based on rock type code;
- Estimated block Au and Ag grades by inverse distance to the third power; and
- Estimated Au and Ag waste grades for partial blocks.

For the seven domains that make up the Cerro Blanco deposit, the search ellipsoids were oriented along the major direction of continuity of the solids. Table 14.11 summarizes the search ellipse dimensions and orientations for each of the estimation passes.

Table 14.9: Search Ellipse Parameters for the Cerro Blanco Deposit

Pass	Major Axis (m)	Semi-Major Axis (m)	Minor Axis (m)	1st Rotation Angle Azimuth (°)	2nd Rotation Angle Dip (°)	3rd Rotation Angle (°)
Main	100	100	25	110	-65	0
Main Splays	150	150	50	110	-40	0
Middle Splays	100	100	25	110	-65	0
140 Veins	100	100	25	140	-90	0
Sinter	100	100	15	0	0	0
Host Hangingwall	100	100	15	140	-90	0
Host Footwall	100	100	15	110	-65	0

Source: Kirkham Geosystems (2017)

14.1.10 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 manual approximate estimates of tonnage to determine reasonableness; and
- 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; and
- Analysis of Relative Variability Index, which quantifies variability and relative error on a block by block basis within the deposit, in addition to assisting with the classification of resources.

14.1.11 Mineral Resource Classification

Mineral resources were estimated in conformity with generally accepted CIM "Estimation of Mineral Resource and Mineral Reserve Best Practices" Guidelines (2003). Mineral resources are not Mineral Reserves and do not have demonstrated economic viability.

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 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. (Kirkham), an "independent qualified person" as defined by National Instrument 43-101.

Drill hole spacing is sufficient for preliminary geostatistical analysis and evaluating spatial grade variability. Kirkham is therefore of the opinion that the amount of sample data is adequate to demonstrate very good confidence of the grade estimates in the deposit.

The estimated blocks were classified according to:

- Confidence in interpretation of the mineralized zones;
- Number of data used to estimate a block;
- Number of composites allowed per drill hole;
- Distance to nearest composite used to estimate a block; and
- Average distance to the composites used to estimate a block.

The classification of resources was based primarily upon distance to the nearest composite; however, all of the quantitative measures, as listed above, were inspected and taken into consideration. In addition, the classification of resources for each zone was considered individually by virtue of their relative depth from surface and the ability to derive meaningful geostatistical results.

Blocks were classified as Measured if they were within 25 m of a composite, had an average distance of all composite used less than 45 m and were interpolated with a minimum of two drill holes. Note that there were no blocks classified as Measured resources. Blocks were classified as Indicated if the nearest composite greater than 20 m, but less than 40 m away.

In addition, the number of drill holes used for the estimate, the average distance of all composites used and the number of composites was considered. The remaining blocks were classified as Inferred; however, all of these blocks were within a maximum of 100 m of the nearest composite and had at least two drill holes contributing to the estimate of that block.

To ensure continuity, the boundary between the Indicated and Inferred categories were contoured and smoothed eliminating outliers and orphan blocks.

14.1.12 Sensitivity of the Block Model to Selection Cut-off Grade

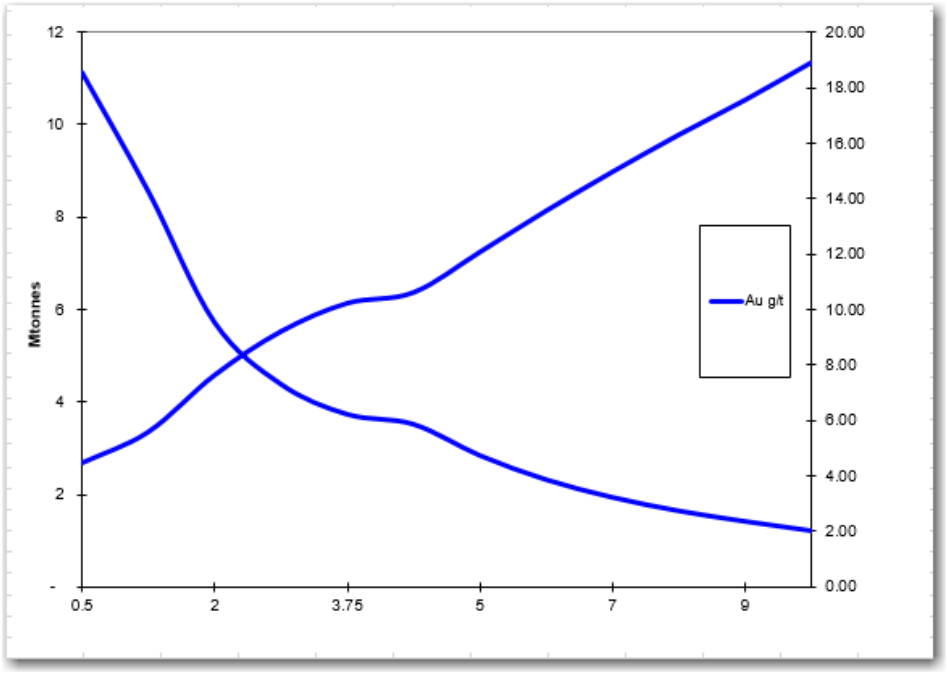
The Mineral Resources are sensitive to the selection of cut-off grade. Table 14.12 shows tonnage and grade in the Cerro Blanco deposit, at different Au cut-off grades. The reader is cautioned that these values should not be misconstrued as a mineral reserve. The reported quantities and grades are only presented as a sensitivity of the resource model to the selection of cut-off grade. Au and Ag grade-tonnage curves for different resource categories are presented in Figure 14.10 and Figure 14.11. Figures 14.12 and 14.13 show section views of the block model with drill holes and estimation domains for Au and Ag, respectively. Figures 14.14 through 14.21 also shows long section views of the Au and Ag block models for the Main, Main Splay, Middle Splays and 140 veins.

Table 14.10: Cerro Blanco Deposit – Sensitivity analyses of Tonnage along with Au and Ag Grades at Various Au Cut-off Grades

Class	Cut-off	Tonnes ('000)	Au g/t	Ag g/t	Au Oz ('000)	Ag Oz ('000)
Indicated	0.5	11,144	4.45	17.1	1,494	5,641
	1	8,566	5.57	21.0	1,447	5,399
	2	5,724	7.62	27.9	1,338	4,873
	3	4,385	9.20	33.1	1,242	4,468
	3.75	**3,731**	**10.22**	**36.5**	**1,242**	**4,468**
	4	3,516	10.61	37.7	1,152	4,093
	5	2,840	12.07	42.4	1,061	3,730
	6	2,315	13.56	47.0	973	3,378
	7	1,932	14.96	51.7	896	3,101
	8	1,638	16.30	55.9	828	2,843
	9	1,411	17.56	60.2	768	2,635
	10	1,208	18.91	65.4	708	2,453
Inferred	0.5	939	2.80	4.6	186	619
	1	802	3.15	4.8	168	517
	2	439	4.59	3.8	129	313
	3	321	5.37	3.9	110	235
	3.75	**243**	**6.02**	**4.2**	**47**	**33**
	4	226	6.18	4.2	92	195
	5	186	6.55	4.3	80	168
	6	100	7.64	5.6	61	139
	7	64	8.26	5.9	50	120
	8	30	9.19	6.3	40	106
	9	15	9.96	6.7	33	97
	10	4	11.35	7.3	28	89

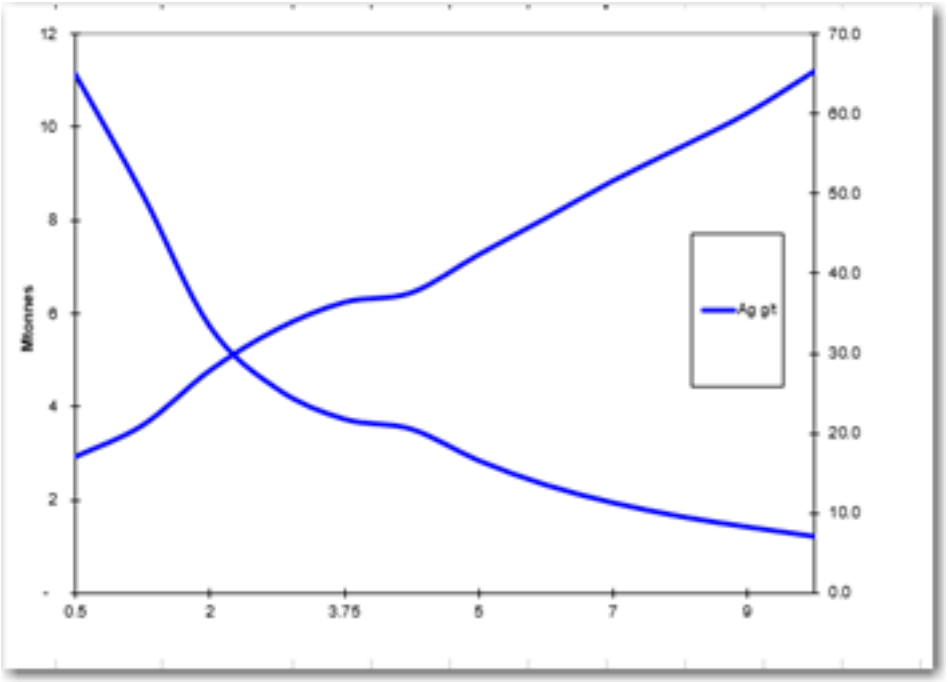
Source: Kirkham Geosystems (2017)

Figure 14.12: Au Grade-Tonnage Curve



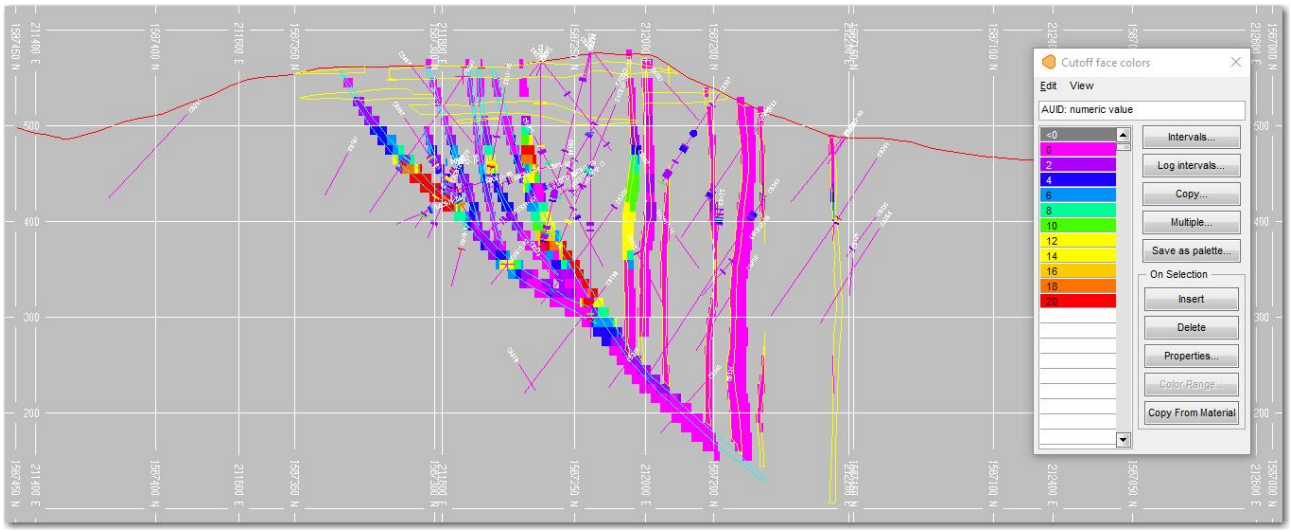
Source: Kirkham Geosystems (2017)

Figure 14.13: Ag Grade-Tonnage Curve



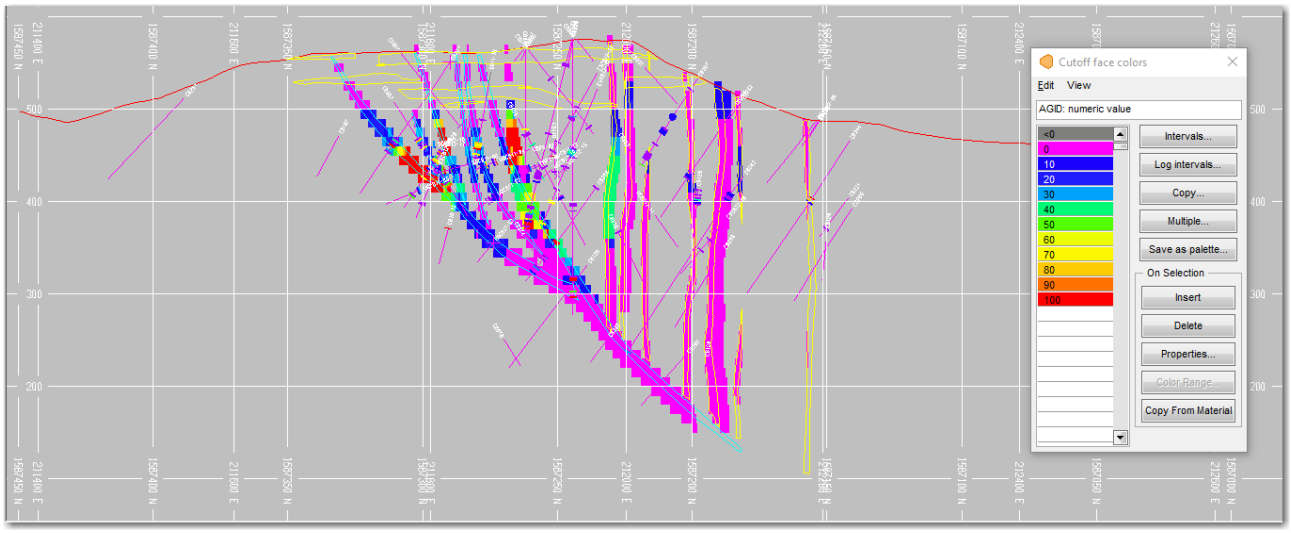
Source: Kirkham Geosystems (2017)

Figure 14.14: Section View of Au



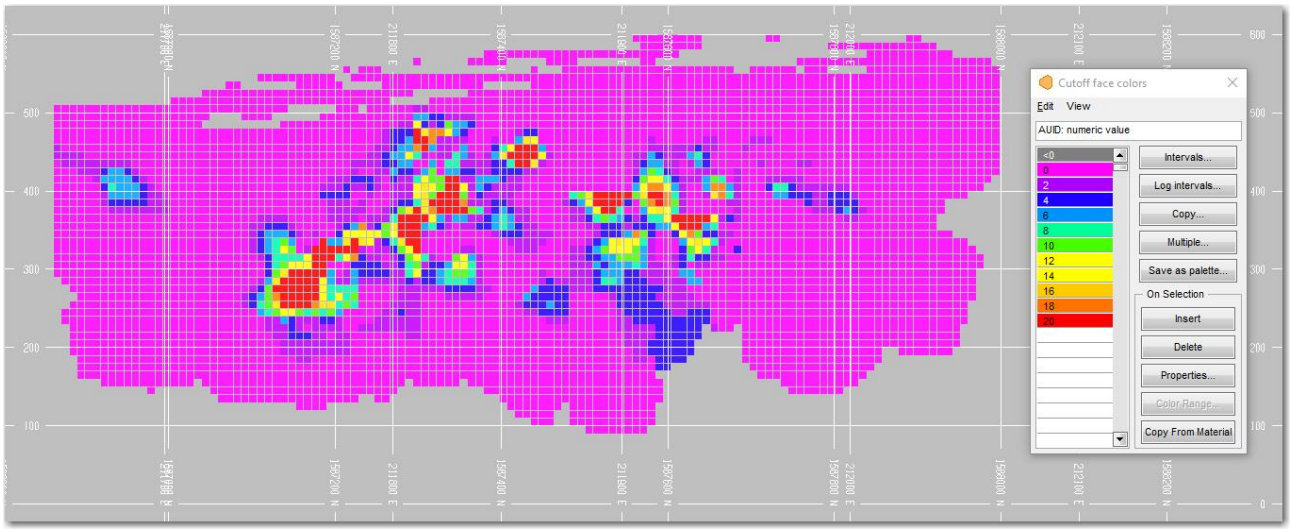
Source: Kirkham Geosystems (2017)

Figure 14.15: Section View of Ag Block Model



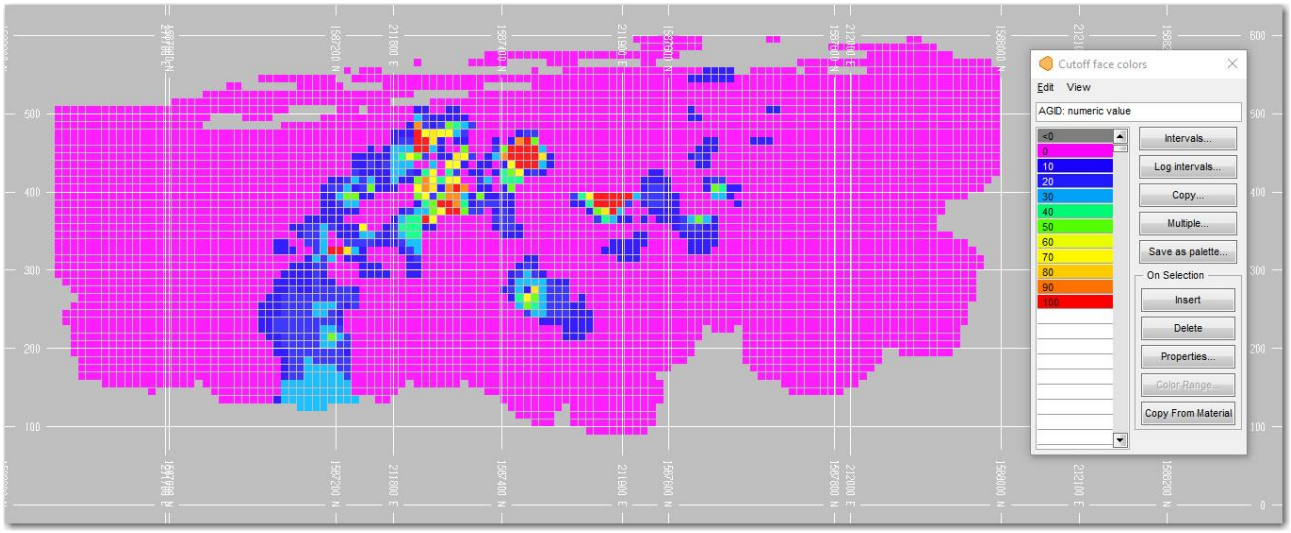
Source: Kirkham Geosystems (2017)

Figure 14.16: Long Section View of Au Block Model for Main Vein



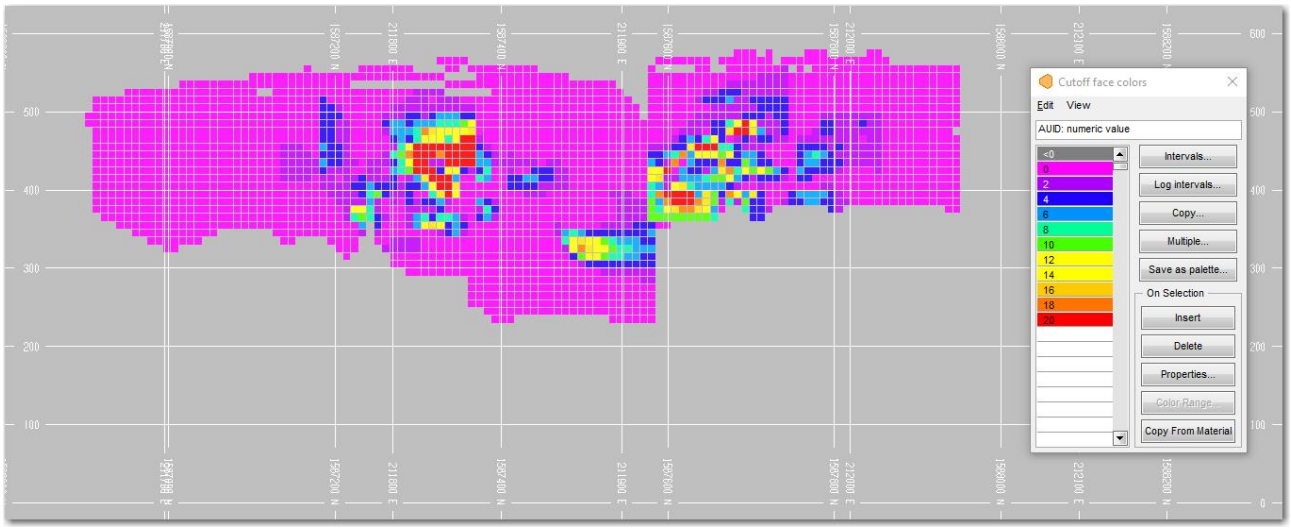
Source: Kirkham Geosystems (2017)

Figure 14.17: Long Section View of Ag Block Model for Main Vein



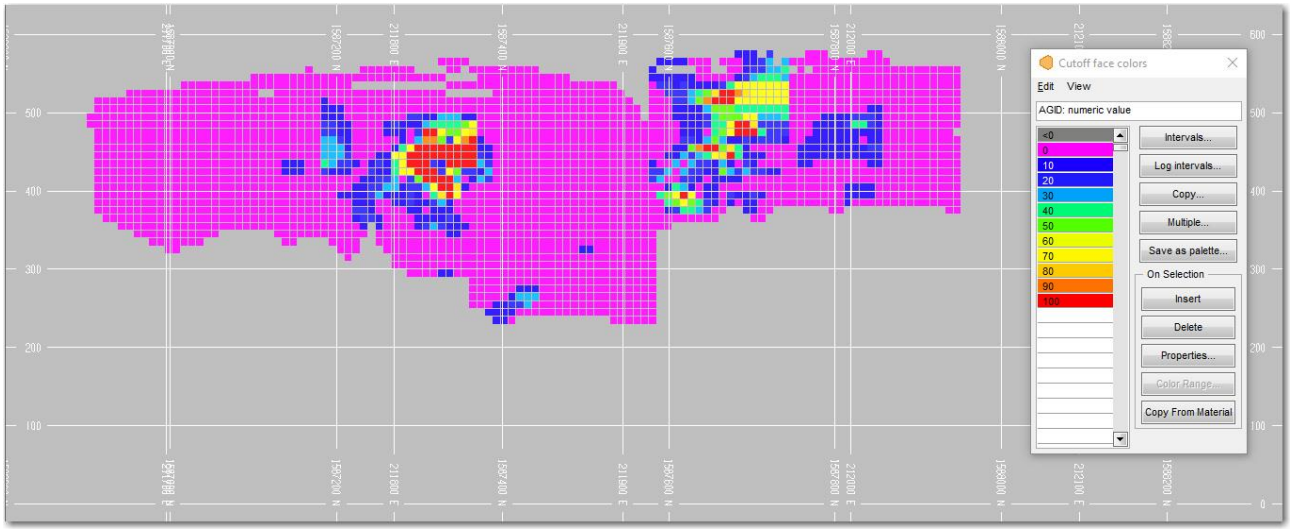
Source: Kirkham Geosystems (2017)

Figure 14.18: Long Section View of Au Block Model for Main Splay Vein



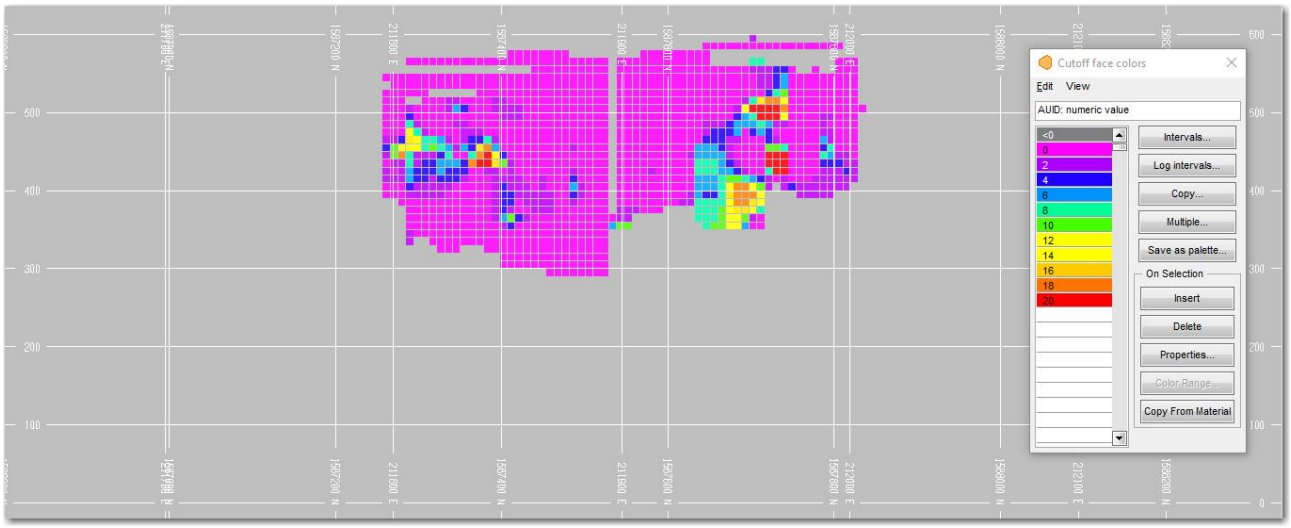
Source: Kirkham Geosystems (2017)

Figure 14.19: Long Section View of Ag Block Model for Main Splay Vein



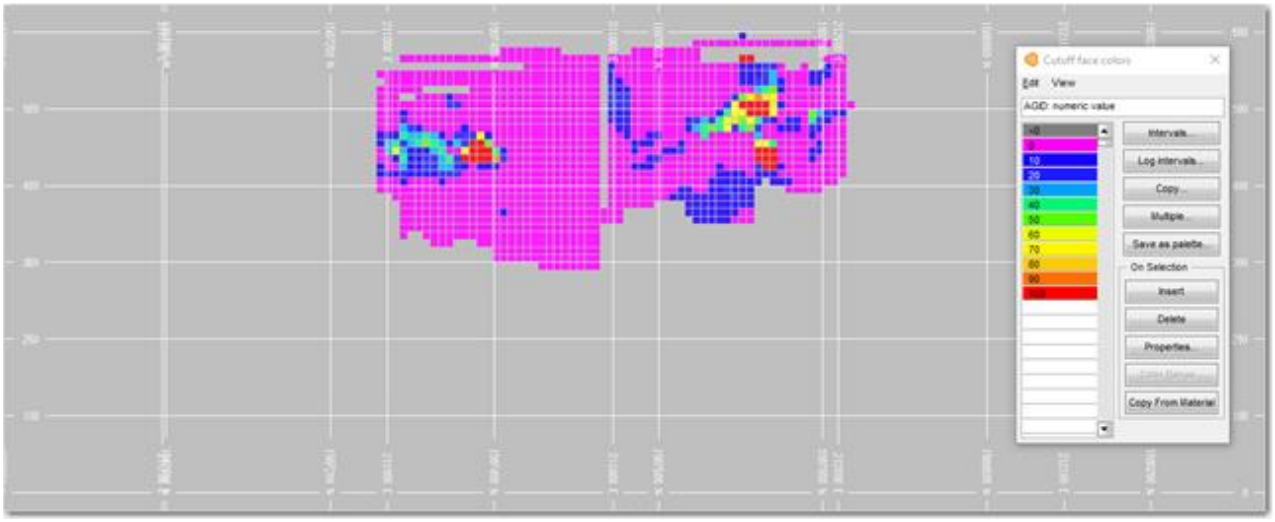
Source: Kirkham Geosystems (2017)

Figure 14.20: Long Section View of Au Block Model for Middle Splay Veins



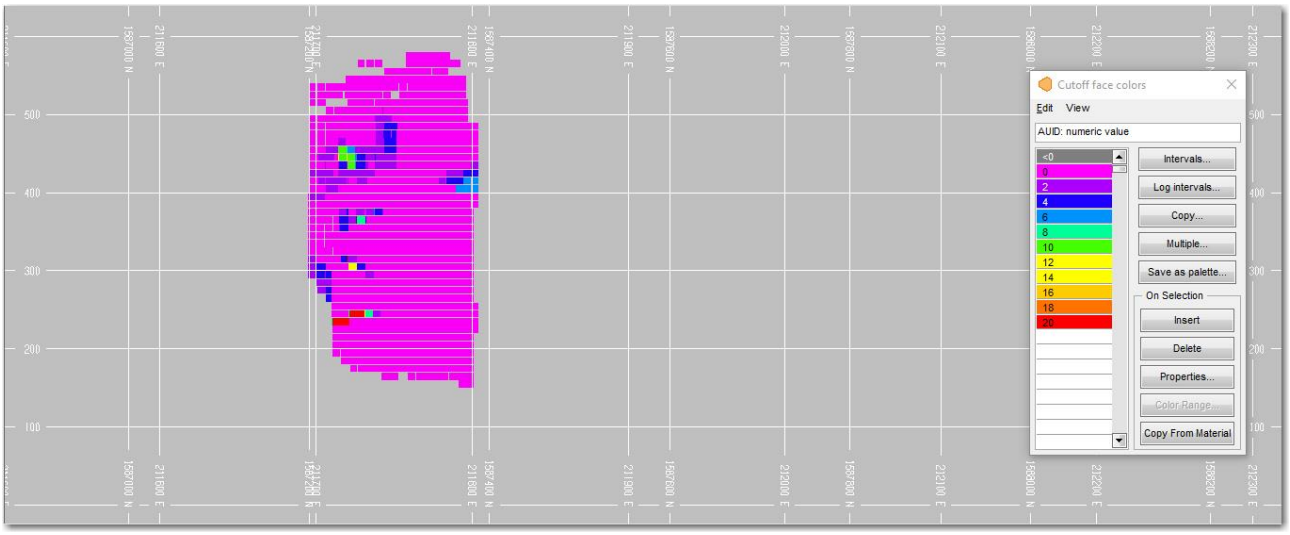
Source: Kirkham Geosystems (2017)

Figure 14.21 Long Section View of Ag Block Model for Middle Splay Veins



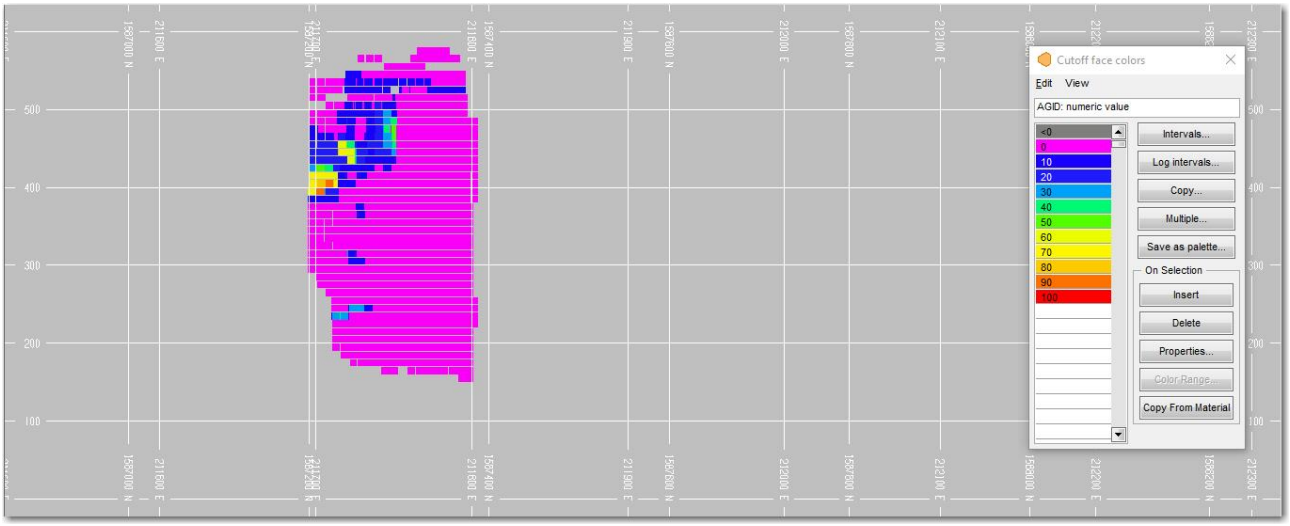
Source: Kirkham Geosystems (2017)

Figure 14.22: Long Section View of Au Block Model for 140 Veins



Source: Kirkham Geosystems (2017)

Figure 14.23: Long Section View of Ag Block Model for 140 Veins



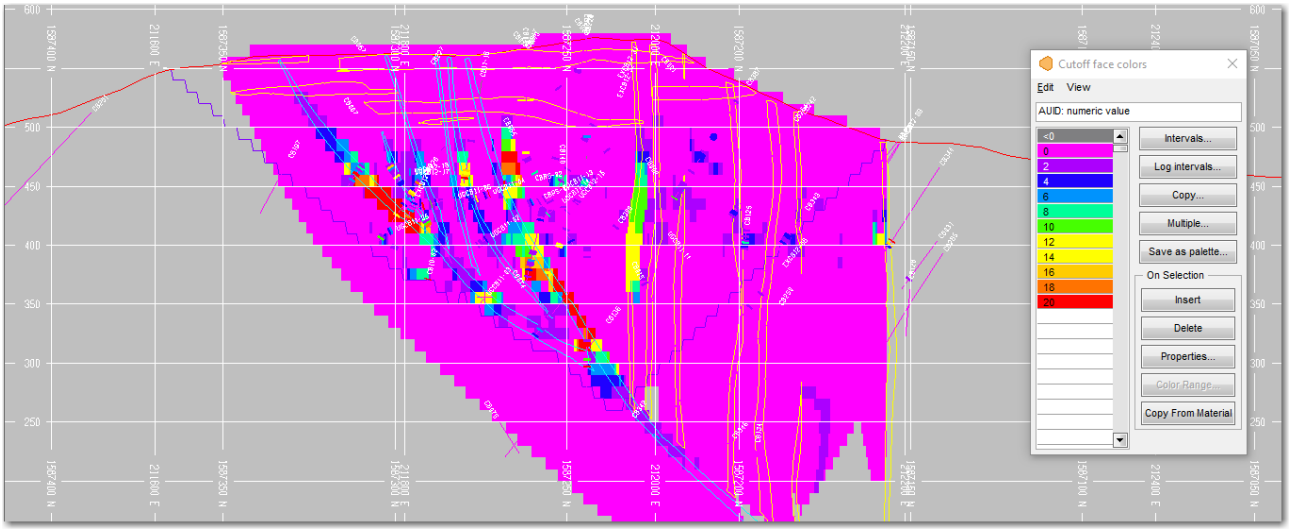
Source: Kirkham Geosystems (2017)

The Cerro Blanco deposit could also be considered potentially open pit minable purely from a resource standpoint. 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 economic assumptions as shown in Figure 14.22. The economic assumptions include the following: $1,250/oz. Au, and $16.00/oz. Ag, 85% recovery, $2.00/t mining costs, $1.50/t G&A costs, $16.50/t milling costs, and a pit slope of 45°.

The pit optimization results are used solely for the purpose of testing the "reasonable prospects for eventual economic extraction" and do not represent an attempt to estimate Mineral Reserves.

Table 14.13 shows tonnage and grade in the Cerro Blanco deposit and includes all units, veins, sinter and host rocks, at different Au cut-off grades. The reported quantities and grades are only presented as a sensitivity of the resource model to the selection of cut-off grades. Au and Ag grade-tonnage curves for different resource categories are presented in Figure 14.23 and Figure 14.24.

Figure 14.24: Long Section View of Gold Equivalent Block Model with Optimized Pit and Drill Hole Data



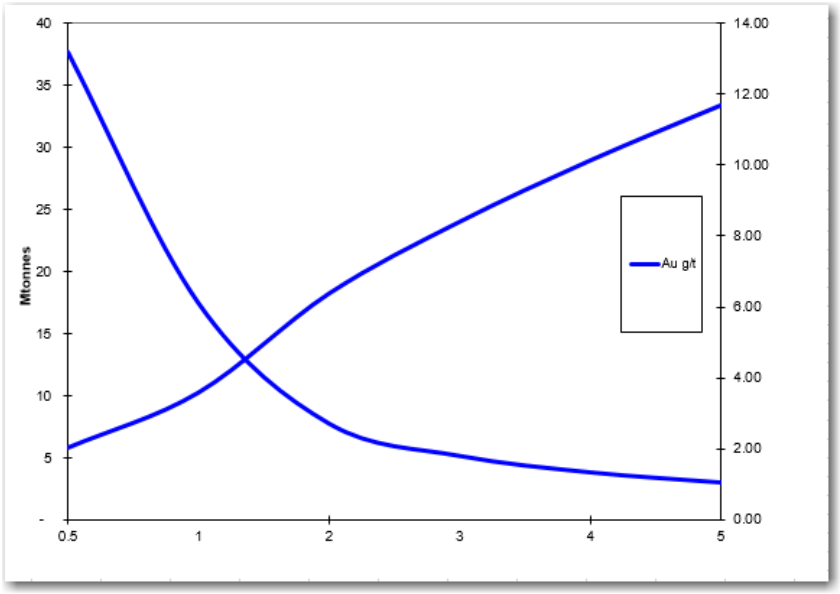
Source: Kirkham Geosystems (2017)

Table 14.11: Cerro Blanco Deposit – Sensitivity Analyses of Tonnage and Grades at Various Au Cut-off Grades

Class	Cut-off	Tonnes ('000)	Au	Ag	Au Ounces ('000)	Ag Ounces ('000)
Indicated	0.5	37,679	2.04	9.3	2,466	11,315
	1	17,437	3.59	14.6	2,012	8,190
	2	7,722	6.38	24.2	1,584	6,013
	3	5,112	8.40	31.2	1,381	5,129
	4	3,787	10.13	37.2	1,233	4,527
	5	2,972	11.68	42.6	1,116	4,073
Inferred	0.5	653	1.24	4.3	26	90
	1	259	2.09	3.5	17	29
	2	96	3.38	3.6	10	11
	3	57	4.08	3.7	7	7
	4	21	5.00	4.4	3	3
	5	6	5.79	4.4	1	1

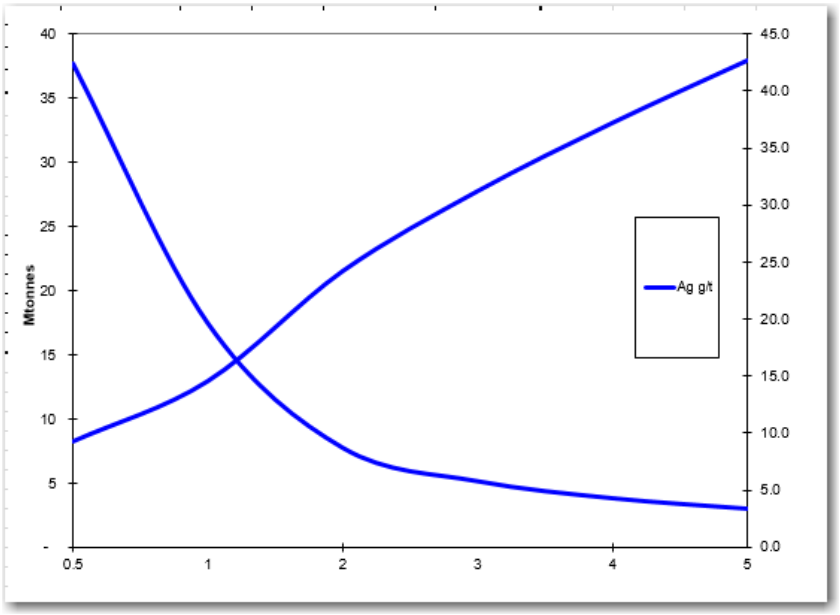
Source: Kirkham Geosystems (2017)

Figure 14.25: Au Grade-Tonnage Curve*



*For the Indicated Class based on Table 14.13
Source: Kirkham Geosystems (2017)

Figure 14.26: Ag Grade-Tonnage Curve*



*For the Indicated Class based on Table 14.13
Source: Kirkham Geosystems (2017)

15 Mineral Reserve Estimates

Mineral resources are not Mineral Reserves and have no demonstrated economic viability. This preliminary economic assessment does not support an estimate of Mineral Reserves, since a pre-feasibility or Feasibility Study is required for reporting of mineral reserve estimates. This report is based on potentially mineable material (mineable tonnes and/or mineable resources).

Mineable tonnages were derived from the resource model described in the previous section. Measured, Indicated and Inferred resources were used to establish mineable tonnes.

Inferred mineral resources are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as Mineral Reserves, and there is no certainty that all or any part of the Mineral Resources or mineable tonnes would be converted into Mineral Reserves.

16 Mining Methods

16.1 Introduction

The potentially mineable resources at the Cerro Blanco deposit will be extracted using a combination of longhole stoping (LH) and drift and fill (DF) underground mining methods with paste and rock backfill. The proposed mine plan is expected to reach a target production rate of 1,250 t/d over a total mine life of 10 years. LH stoping will account for about 88% of total production and the remaining 12% will come from development material and DF. The Cerro Blanco deposit will be accessed from surface via a series of declines, and all mineralized material and waste rock will be trucked out of the mine via these declines. In addition to the four existing ventilation raises, a new ventilation raise will be required. To supplement the ventilation provided by the raises, as the ramps are being driven, shorter internal ventilation drop raises will ensure air at the active development face.

Indicated and Inferred Mineral Resources were included in the mine design and schedule optimization process. Inferred Resources are considered too speculative geologically to have economic considerations applied to them to be categorized as Mineral Reserves, and there is no certainty that the Inferred Resources will be upgraded to a higher resource category. The PEA LOM plan contains approximately 97% Indicated and 3% Inferred Resources. No Mineral Resources have been classified in the Measured category.

16.2 Deposit Characteristics

High-grade mineralization at the Cerro Blanco deposit occurs in steeply dipping, stacked, sub-parallel irregular lenses, which generally strike 20° to the east. The dip of the deposit varies from 40° to nearly vertical. Each lens ranges in thickness from 2 to 10 m in the middle and generally tapers at the edges. The spacing between lenses is inconsistent, but ranges from 0 to 50 m. Generally, lower-grade mineralization surrounds the higher-grade lenses.

The strike length of the main portion of the potentially economic resource is 700 m with a small division between the North and South portions of the mine. The deepest mining level is 340 m below surface (210 mL) and the highest mining level is 40 m below surface (540 mL), meaning the total vertical extent of the potentially economic resource is 330 m. Perpendicular to strike, the potentially mineable resource ranges from 50 m at the lowest levels to 300 m near the surface.

16.3 Geotechnical Analysis and Recommendations

16.3.1 Rock Mass Characterization

A geotechnical investigation was carried out by Golder in 2011 and 2012 to characterize rock mass conditions in support of underground mine design. A combined total of 16 core holes were drilled through the deposit and geotechnically logged and sampled for laboratory strength testing as part of the program. Point load testing was also conducted on core retrieved from the geotechnical drill holes. The original Golder report(s) and data were not available for review as part of the PEA. Assessments of rock mass characteristics are based on summaries of the work that were contained in past studies carried out by Goldcorp for the Project which were reviewed and verified by the QP.

Table 16.1 contains a summary of the key rock mass properties derived by Golder from the investigation.

Table 16.1: Summary of rock mass properties by rock type (after Goldcorp, 2014)

Lithology	Code	Avg. UCS (MPa)	Avg. RQD (%)	*Avg. Q'	*Q' Rating	Avg. RMR'76	RMR'76 Rating
Mita Sandstone	Mss		37	12.4	*Good*	66.4	*Fair*
Mita Siltstone	Msc		76	74.7	*V. Good*	78.1	*Good*
Mita Limestone	Mls	186.9	39	13.8	*Good*	67.1	*Good*
Collapsed Breccia	Cbx	95.7	50	10	*Good*	65	*Good*
Rhyolite	Rp	132.5	9	0.6	*V. Poor*	46.6	*Fair*
Mita Dacites	Mcv	88.3	73	33.3	*Good*	72.8	*Good*
Vein	Vt		30	5.5	*Fair*	61.6	*Good*
Salinas sinter	Ss		43	31.1	*Good*	72.4	*Good*
Salinas Tuffs (conglomerate)	Svc	132.3	49	15.9	*Good*	68	*Good*
Salinas Tuffs (phreatic breccia)	Sfx	68.7	64	11.8	*Good*	66.1	*Fair*
Salinas Volcanic (conglomerate)	Scgl	157.1	72	39.8	*Good*	74	*Good*
Mita Tuffs (andesitic crystalline tuffs)	Mat	49.3	38	9.3	*Fair*	64.5	*Good*
Mita Tuffs (fine grained crystal tuffs)	Mlt		64	25.1	*Good*	71	*Good*
Mita Tuffs (fine grained volcaniclastic)	Mbt	64.3	39	9.3	*Fair*	64.5	*Fair*

[1] Intact rock uniaxial compressive strength.
[2] Q values were calculated as a function of RMR using Barton equation rather than directly from core logging.
Source: SRK (2017)

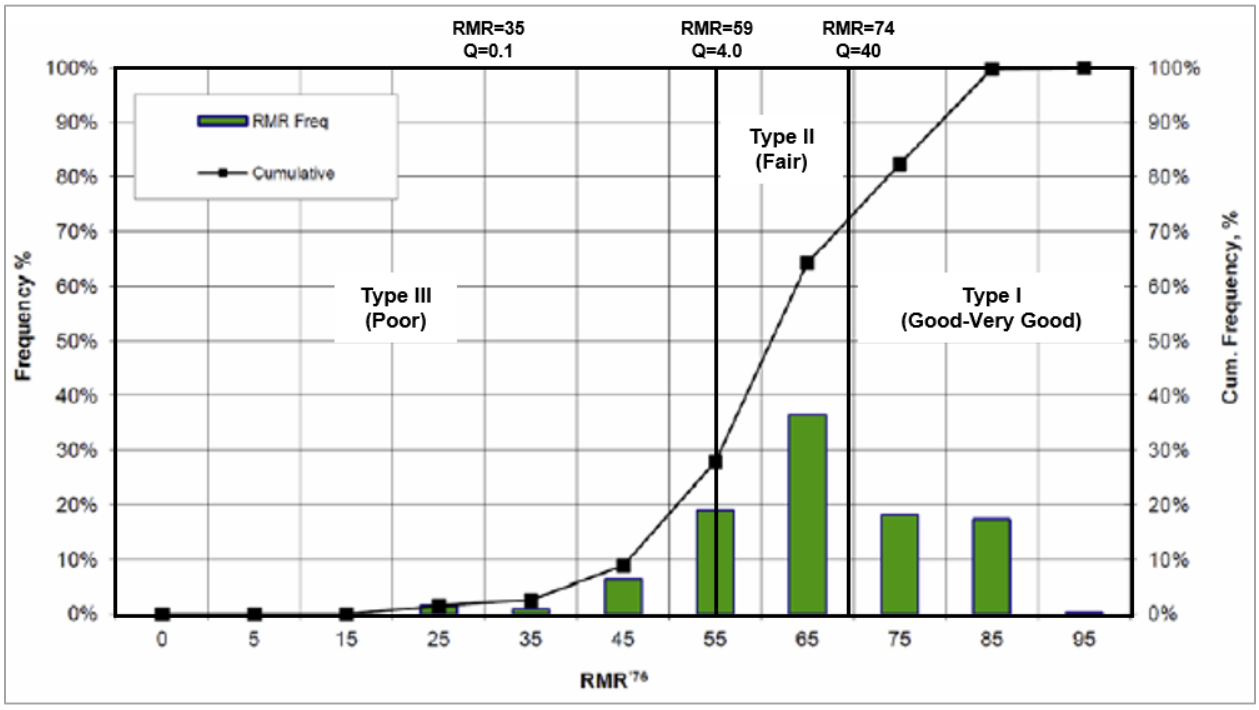
With the exception of the rhyolite which appears to represent a small portion of the overall rock mass in the mine area, the available data indicates reasonably similar rock mass quality between the various lithologies. Overall, the rock mass at Cerro Blanco is anticipated to be Fair to Good quality according to both the Bieniawski (1976) and Barton (1974) rock mass rating systems. Given the similarities in rock quality between the different lithologies and that a 3D lithologic model is not currently available to allow separation of the different lithological units spatially, the combined distribution of rock mass properties was used as the basis for the PEA stope design and ground support analyses.

Figure 16.1 contains a histogram and cumulative frequency plot of the RMR (Bieniawski, 1976) data developed by Golder from the 2011 and 2012 investigations. Given that industry-standard empirical stope design and ground support analyses are based on the Barton Q' value, the RMR values were converted to Q' using the Bienawski (1989) equation.

The RMR cumulative frequency plot was then divided into three ground types for analysis based on rock mass quality:

- Type I (Good to Very Good quality with Q' greater than 40);
- Type II (Fair quality with Q' between 4.0 and 40); and,
- Type III (Very Poor to Poor quality with Q' less than 4.0).

Figure 16.1: Distribution of RMR'76 and Ground Condition Categories



Source: Golder (2012)

Beyond major mineralization controlling structures, the structural geology of the Project does not appear to be well understood yet. The previous geotechnical core was not able to be oriented due to the excessively high temperatures encountered in the drill holes and mapping of discontinuity patterns in the existing workings has not yet been conducted. A geologic structural mapping program will be necessary as the Project advances.

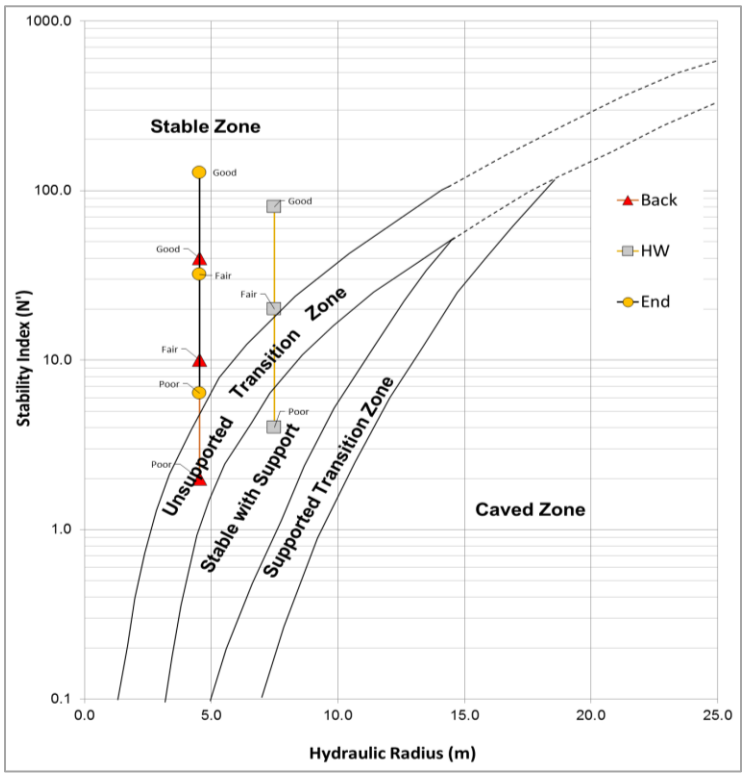
16.3.2 Empirical Stope Design Analyses

Empirical stope design analyses are based on a series of stability graphs where the stability number (N') is plotted on the vertical axis against the hydraulic radius (wall area divided by wall perimeter) of the particular stope wall or back being evaluated on the horizontal axis. The stability number is calculated based on the rock mass quality (Q-system), geologic structure and induced stress conditions.

The estimated stability condition is then determined from the graphs, depending on the empirical method used. The various stability graph methods have slight variations in the position of the stability lines and the terms used to describe the stability zones but overall the approaches are similar.

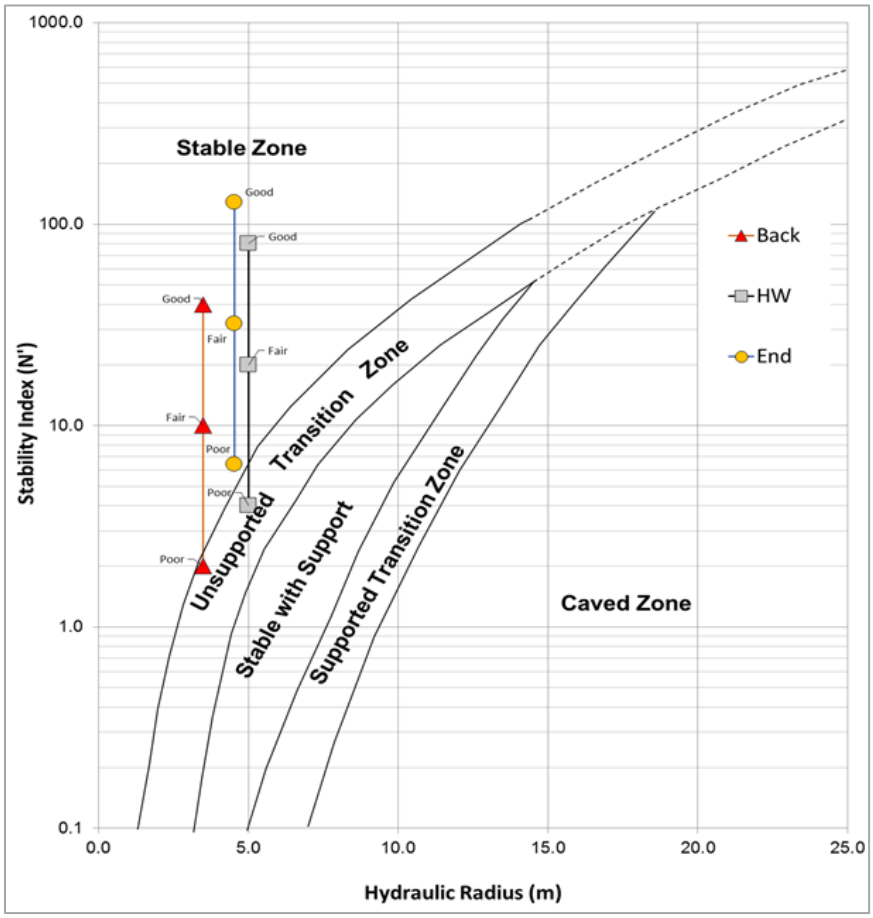
Limiting stope dimensions were estimated using the Potvin (2001) method for the various rock mass conditions anticipated. Figures 16.2 and 16.3 contain stability graphs for the average long hole (LH) stope width of 13 m, height of 30 m and lengths of 30 m and 15 m, respectively. Each stope wall dimension (hydraulic radius) was analyzed with the average Q' value for the Type I (Good), Type II (Fair) and Type III (Poor) ground types. Figure 16.3.4 contains a similar plot for the 4 m x 4 m x 200 m long DF stopes. All cases were analyzed assuming the average hanging wall dip of 60°.

Figure 16.2: Empirical Stope Design Chart - 30 m long LH stopes



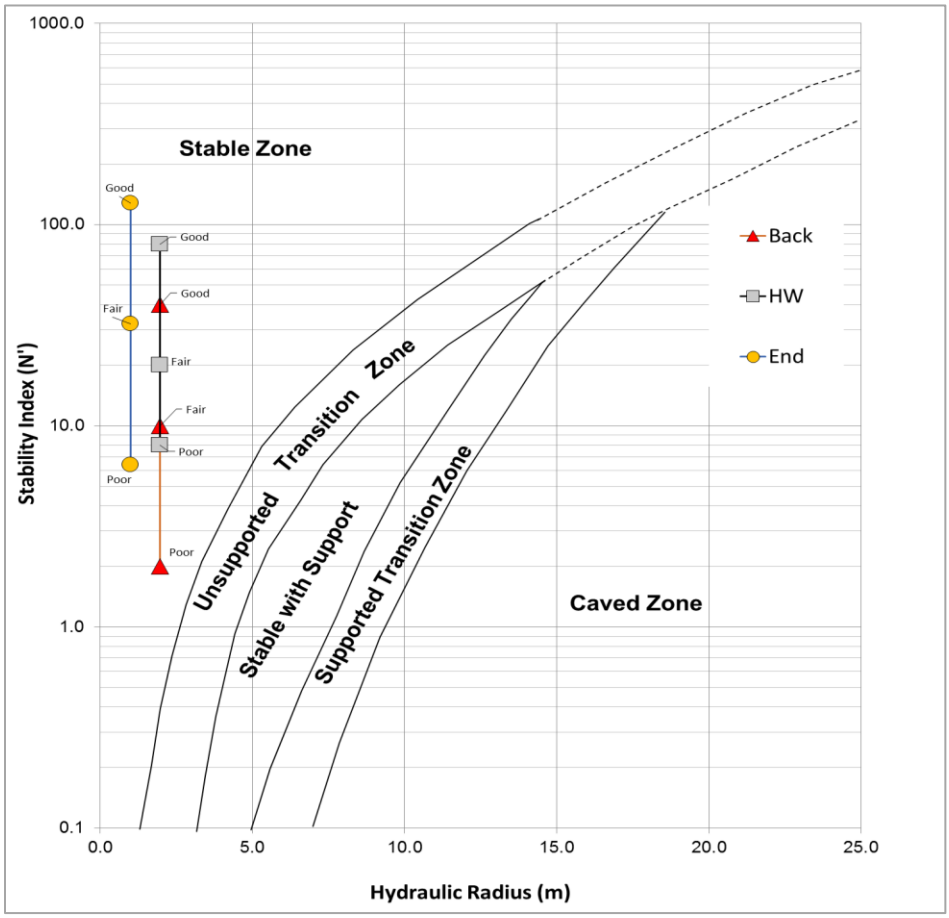
Source: Potvin (2001) & SRK (2017)

Figure 16.3: Empirical Stope Design Chart - 15 m long LH stopes



Source: Potvin (2001) & SRK (2017)

Figure 16.4: Empirical Stope Design Chart - 4 m x 4 m x 200 m DF stopes



Source: Potvin (2001) & SRK (2017)

The following can be concluded from the empirical stope design charts:

- The LH stope hanging wall and back are likely to be unstable when in Type III (Poor) ground conditions (Figure 16.3.2). Approximately 25% of stopes are anticipated to be in Type III ground conditions based on the cumulative frequency plot of rock quality data (Figure 16.3.1);
- For the 25% of the stopes that are anticipated to be in Type III (Poor) ground conditions, the length of the stopes will be reduced to 15 m from the 30 m base case. The stope backs and hanging walls are anticipated to be stable without support in the Type III ground conditions with a 15 m length due to the reduction in hydraulic radius; and,
- The stope end walls are generally expected to be stable without support for all three ground types for the LH stopes and the DF stopes.

16.3.3 Ground Support

Based on the range of anticipated rock quality (Q' values) as well as the size and expected life and use of the various mine openings, ground support recommendations were developed according to the Barton (1974) criteria. The Q-system also takes into account the life and use of the opening (ex. man-entry or equipment only) with the excavation support ratio (ESR) parameter. The ESR is used to adjust the design span, in order to obtain the equivalent span for use in the Q Support Diagram; in effect it imposes a higher factor of safety on critical structures with long life (such as an underground nuclear power station with an ESR rating of 0.5 to 0.8) than on temporary tunnels (such as temporary mine workings with an ESR rating of 2 to 5).

Different ESR values were used for primary (permanent man-entry) and secondary (temporary) stope and development openings. The ground support class recommended for each type of mine opening and ground type is summarized in Tables 16.2, 16.3 and 16.4. Bolts for long-term support are recommended to have corrosion protection and be fully grouted along their length. The grout can be either cement or resin depending on the expected water flow and water chemistry.

Table 16.2: Ground support Recommendations for 4.5 m x 4.5 m Primary Ramp System (ESR of 1.6)

Support Class	Rock Quality	Percent of Ramp (%)	Support Type
Type I	Good	25	Systematic support not required. Spot bolting with mesh as needed.
Type II	Fair	50	2.4 m long fully grouted #7 rebar on 1.5 m ring spacing and 1.5 m within the ring with welded wire mesh in back and shoulders.
Type III	Poor	25	2.4 m long fully grouted #7 rebar on 1.5 m ring spacing and 1.5 m within the ring with 6 cm shotcrete in back and walls.

Source: SRK (2017)

Table 16.3: Ground Support Recommendations for 4 m x 4 m Attack Ramps, Drives and DF Mining (ESR of 3)

Support Class	Rock Quality	Percent of Mine (%)	Support Type
Type I	Good	25	Systematic support not required. Spot bolting with mesh as needed.
Type II	Fair	50	1.8 m long fully grouted #7 rebar on 1.2 m ring spacing and 1.2 m within ring down in back and shoulders. Welded wire mesh is anticipated to be necessary in 50% of the areas.
Type III	Poor	25	1.8 m long fully grouted #7 rebar on 1.2 m ring spacing and 1.2 m within ring down in back and shoulders with welded wire mesh.

Source: SRK (2017)

Table 16.4: Ground Support Recommendations for 13 m x 4 m LH Stopes (ESR of 3)

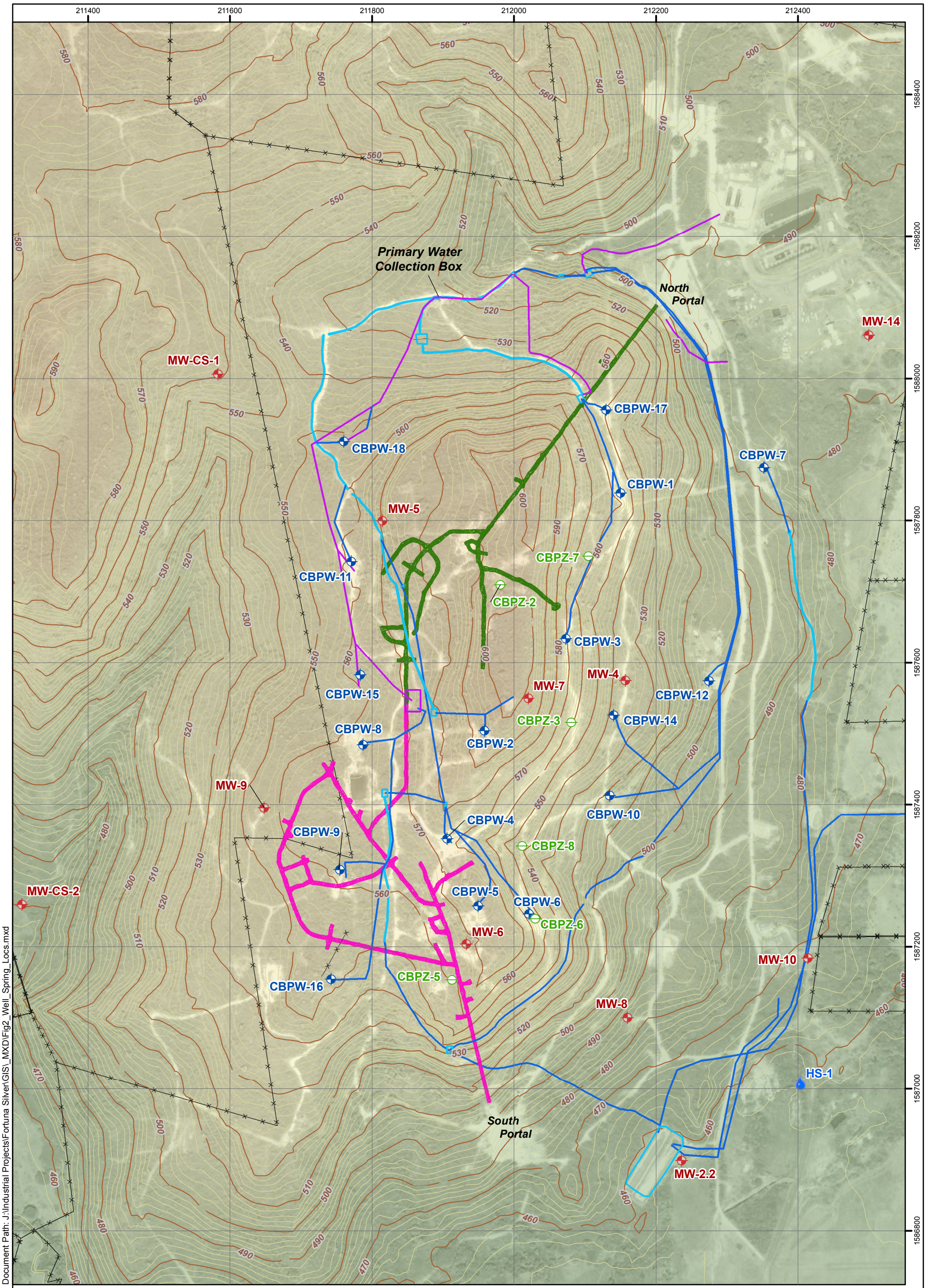
Support Class	Rock Quality	Percent of Mine (%)	Support Type
Type I	Good	25	Systematic support not required. Spot bolting with mesh as needed.
Type II	Fair	50	3.2 m long fully grouted #7 rebar on 3 m ring spacing and 3 m within ring in back and shoulders with welded wire mesh.
Type III	Poor	25	Reduce to 15 m long and use Type II ground support.

Source: SRK (2017)

16.4 Hydrogeology Analysis and Recommendations

Dewatering of the Cerro Blanco mine has been a limiting economic factor in mine development because of high temperature groundwater and thermal gradients. The average static groundwater-level elevation in the Project area is approximately 462 masl. Mine dewatering activities from pumping wells and two in tunnel sump pumps (combined discharge of about 600 gallons per minute (g/m)) have lowered the static groundwater-level to approximately 440 m at the mine as of 2013. The existing location of the portals, dewatering wells and monitoring wells are shown in Figure 16.5.

Figure 16.5: Existing Location of Portals, Dewatering Wells and Monitoring Wells



Legend

- Pumping Well
- Piezometer
- Monitoring Well
- North Tunnel
- South Tunnel
- Hot Spring
- Drainage Pipe
- Cooling Channels
- HDPE Pipe
- Property Boundary
- Ground Surface Contour
  - 2 m interval
  - 10 m interval

N, W, E, S

0 50 100 150 Meters

Coordinate System: UTM, NAD 27, Zone 16N

Title: Well and Spring Locations	
Project: Cerro Blanco	Figure: 2
	Date: Jan 2017
Client: JDS Energy & Mining Inc.	Project No:
	MWH now part of Stantec

The current conceptual model suggests that as previous dewatering efforts at Cerro Blanco increased, groundwater levels declined, but remained within the lower Salinas tuff-volcaniclastic sequence and upper Mita Group units. The lower hydraulic heads created by pumping likely enhanced gradient- and thermal-driven recharge through faults and interconnected fractures. There is little information pertaining to hydrogeology of the faults to determine if they provide higher permeability conduits to flow or act as barriers limiting connection between fault zones and the overall effect on long term water quality within the area. Regional and intermediate groundwater flow systems between the Ipala Volcano and the Rio Paz shear zone likely also contribute to gradient-driven flow to Cerro Blanco under pumping conditions. The regional contribution of groundwater is not fully understood and needs to be further evaluated as part of the future dewatering infrastructure installation as evidence suggest that a broad-relatively flat drawdown cone will result and could impact potential water uses and discharges to the natural environment..

16.4.1 Evaluation of Dewatering Rates and Number of Locations

Historical estimates for mine dewatering have ranged from 2,000 g/m to as high as 10,000 g/m for various mine plans and have advanced as more data becomes available to better document the interconnections of the various fault systems to the regional flow system.

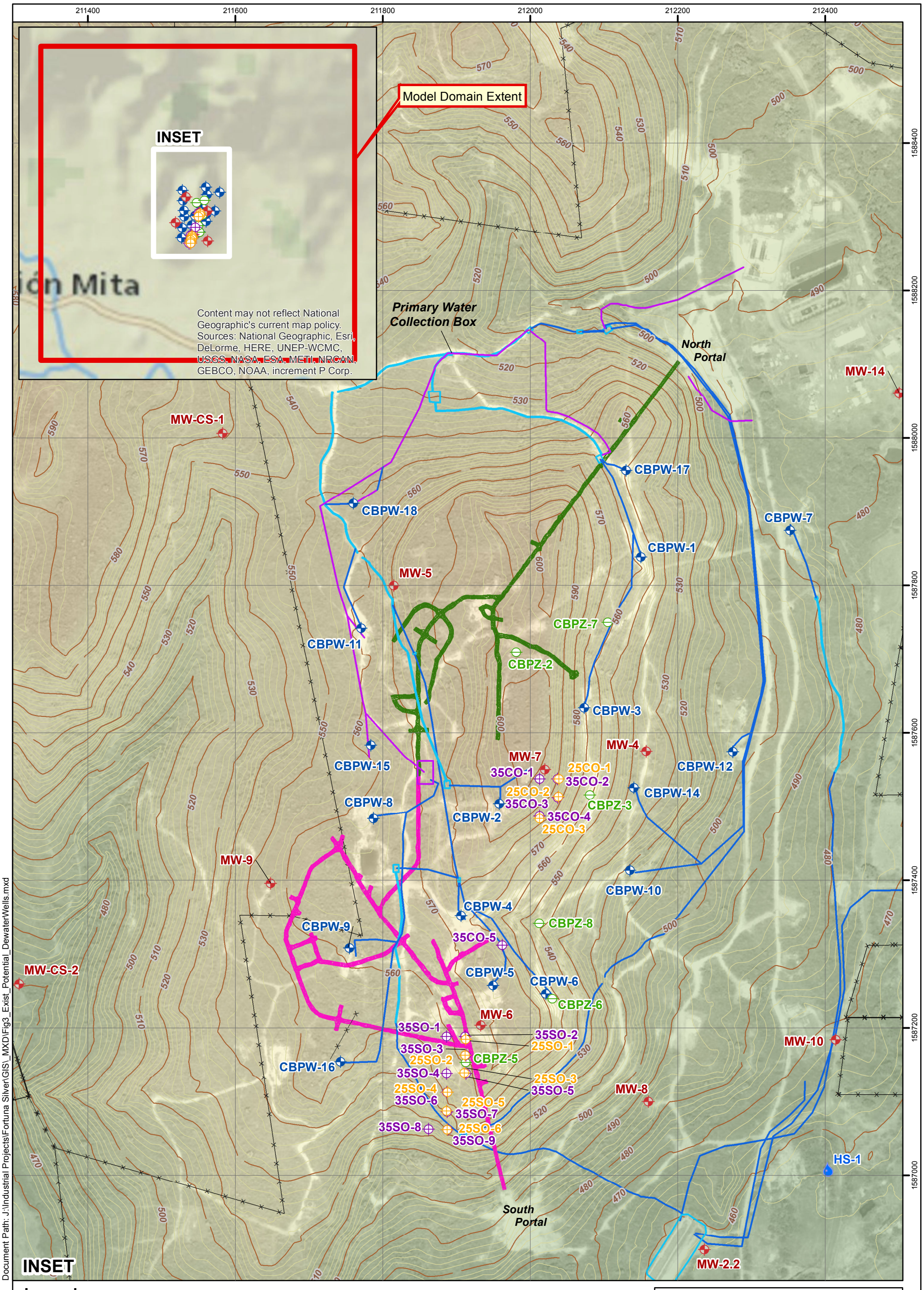
A revised mine plan has recently been developed by JDS that results in the northern area of the mine reaching a level of 210 m within five years of mine start-up. In the southern area, the planned mine level is 270 m, which is also attained within five years. The change in planned mining depth reduces the overall dewatering demands at Cerro Blanco and has allowed focused dewatering strategies to be developed.

As part of the preliminary dewatering evaluation, a simplified numerical groundwater flow model was used to simulate potential impacts associated with groundwater pumping for mine dewatering. The numerical model (MODFLOW) used a single unconfined layer, with 462 m of saturated thickness, and a grid of 200 rows by 200 columns (40,000 total cells) with a cell size of 25 x 25 x 500 m. The model assumed homogeneous aquifer characteristics, with a hydraulic conductivity of 0.07285 m/day and a storage coefficient of 0.028. These hydrologic parameters are based on the average values derived from testing and analyses conducted previously by MWH (2014). A general head boundary condition was used to simulate flux into and out of the model along all periphery model cells (a total of 796 general head cells). No surface recharge or evapotranspiration was simulated. This model provides a simplified tool for the calculation of dewatering rates and volumes over time and will require further development and calibration to refine dewatering estimates and injection strategies.

Dewatering wells were simulated at combined rates of 2,500 g/m and 3,500 g/m to evaluate the timeframe for dewatering areas around the current mine plan. The location of the dewatering wells for both scenarios are shown in Figure 16.6 (note: 2,500 g/m wells are called “25” series and 3,500 g/m wells are called “35” series). While both dewatering rates as predicted by the model reach the desired dewatering levels, the 2,500 g/m rate does not achieve the dewatering in the desired timeframe planned for mine advancement. Therefore, a dewatering rate of 3,500 g/m was selected for a preliminary rate to be used in this PEA. The model-predicted water levels over time are shown in Figure 16.7 with key timeframes for dewatering levels based on the mine plan provided by JDS. Higher dewatering rates may be required to achieve the 210 masl dewatering level within the required time frame depending on aquifer storage and regional connection through fault zones. These details are not known at this time and require further assessment and analysis to better confirm the ultimate

dewatering requirements. The 3500 g/m dewatering rate is based on overcoming the existing storage within the aquifer and targeted dewatering along fault zones that provide connection to the regional flow system. If regional influence is significant, dewatering rates could be over twice the focused dewatering rates of 3500 g/m and would require more wells to achieve dewatering. The increase in dewatering rates would have an effect on the need for additional injection wells and/or water treatment capacity based on the anticipated water quality. This will need to be further evaluated in the next study phase.

Figure 16.6: Existing and New Dewatering Well Locations



Legend

- Pumping Well (PW)
- Piezometer
- Monitoring Well
- Proposed PW (3500 gpm)
- Proposed PW (2500 gpm)
- Hot Spring
- Drainage Pipe
- Cooling Channels
- HDPE Pipe
- Property Boundary
- gpm — gallons per minute
- North Tunnel
- South Tunnel
- Ground Surface Contour
  - 2 meter interval
  - 10 meter interval

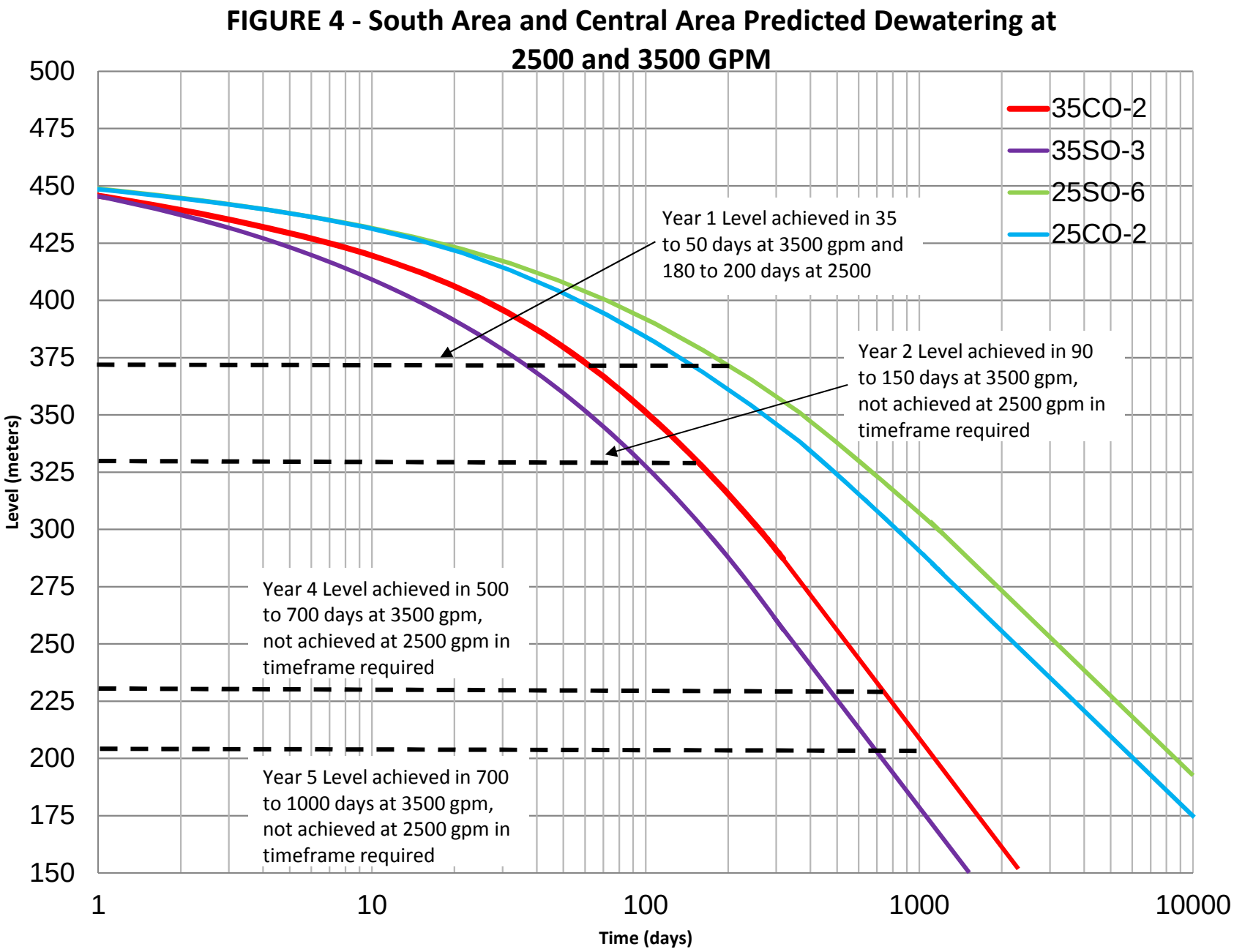
N, W, E, S

0 48 96 144 Meters

Coordinate System: UTM, NAD 27, Zone 16N

Title: Existing and Potential Dewatering Well Locations	
Project: Cerro Blanco	Figure: 3
	Date: Jan 2017
Client: JDS Energy & Mining Inc.	Project No:
	MWH, now part of Stantec

Figure 16.7: Simulation Hydrograph – South Area and Central Area Predicted Dewatering at 2,500 and 3,500 g/m



Source: Stantec (2017)

The 3,500 g/m scenario uses 10 existing wells and 14 new dewatering wells pumping an average of approximately 145 g/m each (see Table 16.1). The 10 existing wells were selected because of their proximity to the mineralized zones where dewatering is required; however, the current condition of these wells is not known. Further evaluation of the existing capacity and condition is required and either redevelopment maybe required, or if conditions have deteriorated beyond recovery, new wells maybe required. The new wells will likely be 12-inch diameter casing in an 18-inch diameter borehole. Telescoping perforations starting at 26-inches will likely be required for the drilling depths. It is recommended that ten piezometers be installed in small diameter boreholes (3.345-inch diameter) and equipped with vibrating wire line transducers and temperature sensors to monitor pressure changes and temperature changes in the rock as part of the dewatering program. With the new dewatering wells, these can be phased over the first two years of planned operations. Year -1 and Yea 1 would require six wells installed each and the remaining two wells could be installed in Year 2.

While the model shows the dewatering being reached within the desired timeframe for mining at a rate of 3,500 g/m, it does not account for the thermal effects that will be at site, heterogeneities of the aquifer and faults that may connect the mine site to regional groundwater flow systems, and seasonal variations in groundwater recharge. Observations from site suggest that the system is highly compartmentalized, which may necessitate the installation of additional wells to achieve the dewatering condition. The degree of compartmentalization will need to be measured with vibrating wire line piezometer installed in the mine area to assess pressures and temperatures. In addition, a more regional assessment of the potential effects of dewatering on the groundwater resources of the area will be required as it is expected that a broad and flat drawdown cone will result from the proposed dewatering.

Furthermore, potential changes in water quality over time will need to be better understood as this will impact the need for treatment to meet discharge criteria or for suitability of the water for reinjection, The water quality assessment will need to consider the potential effects of dewatering on overall water quality and water resources in the area.

Finally, it is anticipated that precipitation will continue to supply approximately 400 to 600 g/m of water to the shaft, which would need to be dewatered through sumps. Overall, the water modelling suggests that the mine plan should expect to handle (through treatment or injection) a minimum water volume in the range of 4,000 g/m.

Table 16.5: Dewatering Well Key Parameters

Well	Approx. Pumping Rate to Achieve Planned Dewatering (g/m)	Dewater Well Depth for Planning (m)	Dewater Borehole Diameter (inches)	Casing and Screen Diameter (inches)	Easting[1] (m)	Northing (m)
CBPW-2	85	Existing	Existing	Existing	Existing	Existing
CBPW-3	93	Existing	Existing	Existing	Existing	Existing
CBPW-4	141	Existing	Existing	Existing	Existing	Existing
CBPW-5	141	Existing	Existing	Existing	Existing	Existing
CBPW-6	125	Existing	Existing	Existing	Existing	Existing
CBPW-8	146	Existing	Existing	Existing	Existing	Existing
CBPW-9	141	Existing	Existing	Existing	Existing	Existing

Well	Approx. Pumping Rate to Achieve Planned Dewatering (g/m)	Dewater Well Depth for Planning (m)	Dewater Borehole Diameter (inches)	Casing and Screen Diameter (inches)	Easting[1] (m)	Northing (m)
CBPW-11	292	Existing	Existing	Existing	Existing	Existing
CBPW-14	141	Existing	Existing	Existing	Existing	Existing
CBPW-15	130	Existing	Existing	Existing	Existing	Existing
35-CO-1	180	450	18	12	212012.5	1587538
35-CO-2	165	450	18	12	212037	1587538
35-CO-3	125	450	18	12	212037.3	1587512
35-CO-4	175	450	18	12	212012.5	1587487
35-CO-5	165	450	18	12	211962.1	1587312
35-SO-1	145	450	18	12	211886.4	1587188
35-SO-2	125	450	18	12	211911.6	1587188
35-SO-3	135	450	18	12	211912.8	1587163
35-SO-4	150	450	18	12	211886.4	1587138
35-SO-5	125	450	18	12	211912.8	1587138
35-SO-6	150	450	18	12	211886.8	1587112
35-SO-7	125	450	18	12	211887.5	1587086
35-SO-8	150	450	18	12	211862.3	1587062
35-SO-9	150	450	18	12	211888.3	1587062
Total	3500	6300				

[1] – NAD1927UTMZn16N

Source: Stantec (2017)

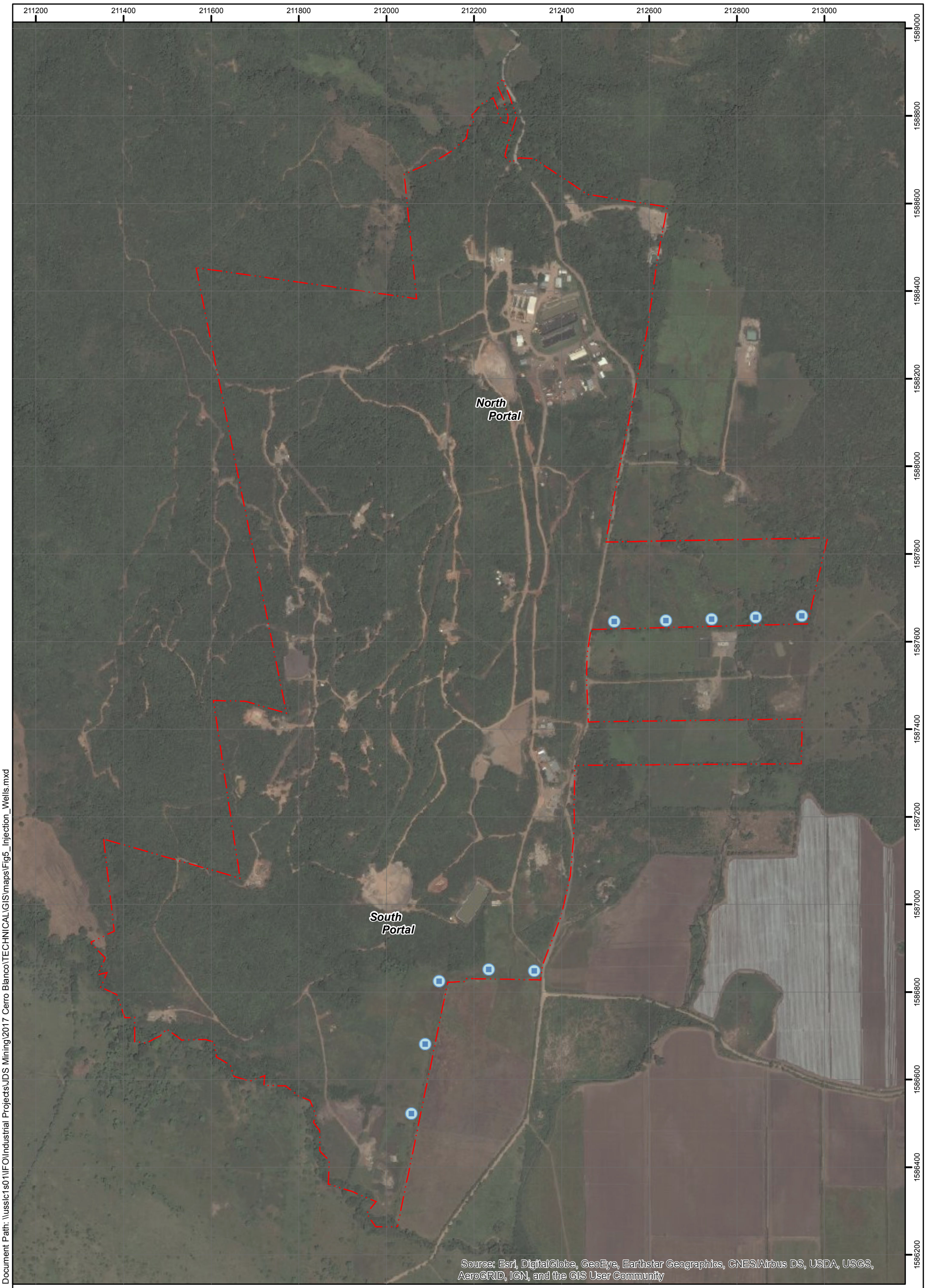
16.4.2 Injection Wells

The existing water treatment plant at Cerro Blanco can treat 1,500 g/m. Considering the dewatering rates are expected to be 3,500 g/m and an additional 500 g/m for mine sump water, total water treatment volumes from dewatering will be approximately 4,000 g/m in the peak dewatering periods. This volume does not consider any contact water that will be managed at the mine site. This means a total of approximately 2,500 g/m of dewatering water will need to be managed by other means than the current water treatment plant. It is recommended that 10 injection wells be installed that are capable of managing approximately 250 g/m each and will be confirmed during future injection well testing. The injection wells would be drilled to a depth of about 150 m using a 12-inch diameter casing in an 18 inch diameter borehole and located on the mine property. The preliminary locations of the injection wells are shown in Figure 16.8. The injection wells are planned to be located on the south side of the mine so that groundwater withdrawals from dewatering have less impact on the local community and in order to limit pull back of water into the dewatering zone. The injection wells will likely be much shallower than the dewatering wells, and will be more focused on the shallow alluvium and upper fractured bedrock. One additional injection well is planned to handle the water from the dry

stack tailings facility (DSTF) runoff pond. As with the dewatering wells, injection wells can be phased-in with six planned in Year -1 and five in Year 1. Water quality, especially high iron content in the groundwater discharged to the injection wells could cause well fouling and present a future issue with the ability to inject water efficiently. A more detailed evaluation of water quality and suitability of the water to meet injection requirements and regulatory approvals should be completed during the next study phase.

Further test work will be carried out in the next phase of engineering to confirm the shallower depth of injection wells will not adversely impact ground water quality in surrounding communities and the efficiency of the injection system.

Figure 16.8: Preliminary Location of Injection Wells



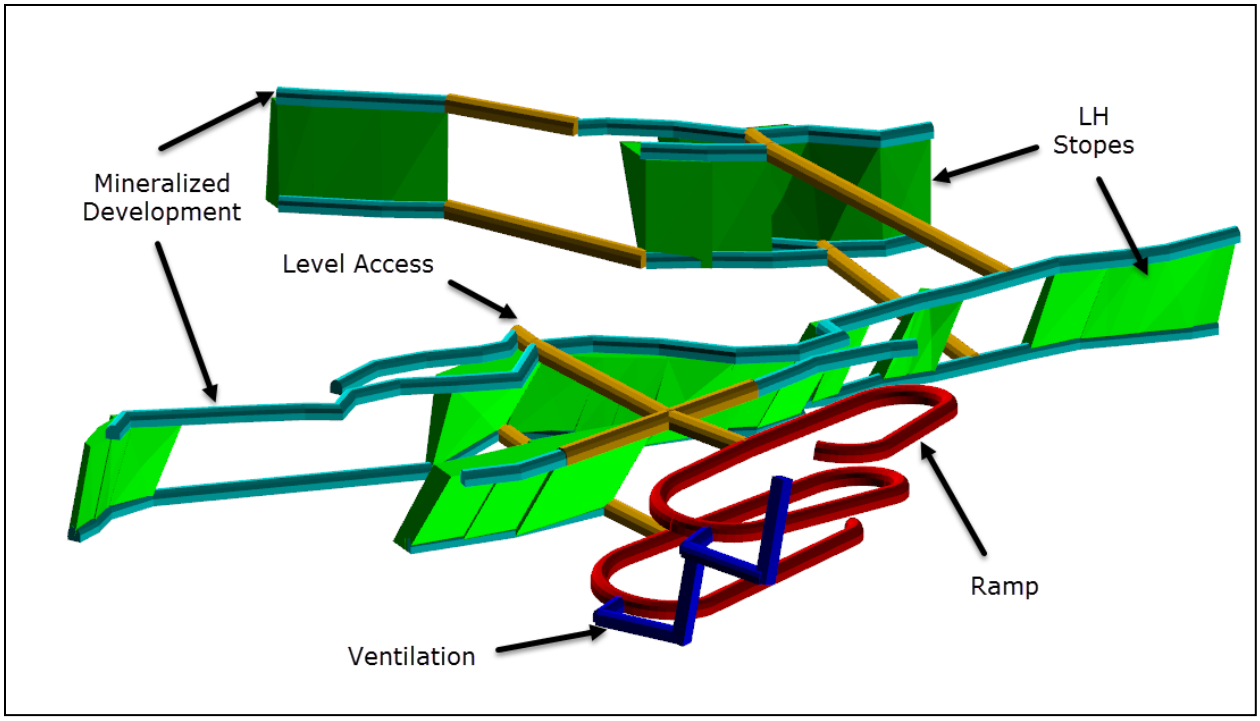
Source: Stantec (2017)

16.5 Mining Methods

JDS selected longhole stoping (LH) as the primary mining method for the areas which are thickest and most continuous and DF for the flatter and narrower areas. Both methods will use a combination of cemented paste backfill and rock backfill. Areas amenable to LH stoping were selected first, and DF was selected for the remainder. LH stoping accounts for about 89% of the total mined resources in the mine plan while cut and fill accounts for 7%. The remaining 4% mined resource comes from development in the mineralized areas. Mining of the Cerro Blanco deposit is designed and scheduled such that multiple mining blocks will be in production simultaneously, thereby allowing the mine to reach target production rate before the spiral decline has reached its ultimate depth. However, each individual mining block will be mined in an overhand (bottom-up) manner. A typical level arrangement is shown in Figure 16.9.

The total average mining dilution calculated for the overall deposit was 14%.

Figure 16.9: Perspective View of a Typical Mining Level (330 Level)



Source: JDS (2017)

16.5.1 Drift and Fill Mining

The DF method is typically used for mineralization that is irregular in shape, has a shallow dip, and has relatively high value mineable material. It is a preferred mining method in poor or uncertain ground conditions, since the span of the openings can be kept small and backfill is used for support of the hanging wall.

DF is a development-intensive mining method, in which a footwall drive is established across the entire length of the mining zone and production drifts are mined parallel to the footwall drive in a primary-secondary sequence. After the primary drift is mined, it is backfilled with cemented fill, so that the secondary drift may be mined alongside. The walls of the secondary drift expose the cemented backfill of the primary drifts.

The minimum drift dimensions for DF mining at Cerro Blanco were assumed to be 5 m wide by 5 m high. Mining recovery for the DF zone is 95%.

16.5.2 Longitudinal Longhole Stoping

LH stoping is a semi-selective and productive underground mining method, and is well suited for steeply dipping or massive deposits with good continuity. It is typically one of the most productive and lower-cost mining methods applied across many different styles of mineralization. Longitudinal LH stoping will be the mining method used at Cerro Blanco for the thickest, most continuous portions of the potentially mineable resource. This method can achieve high production rates as multiple stopes can be in operation at once on a level depending on the level geometry.

The LH stopes at Cerro Blanco were designed to be up to 30 m long, 30 m tall, and no wider than 15 m (average width of 11.5 m). The extraction sequence will generally be overhand (bottom-up) and will retreat towards the level access. Each stope is accessed by a 5 by 5 m cross-cut above and below. In cases where the stoping block accessed by a set of crosscuts is wider than 15 m and will be divided into multiple stopes, the stopes will be extracted and filled with paste backfill. Parallel extraction drifts are required when the stope exceeds 15 m in width.

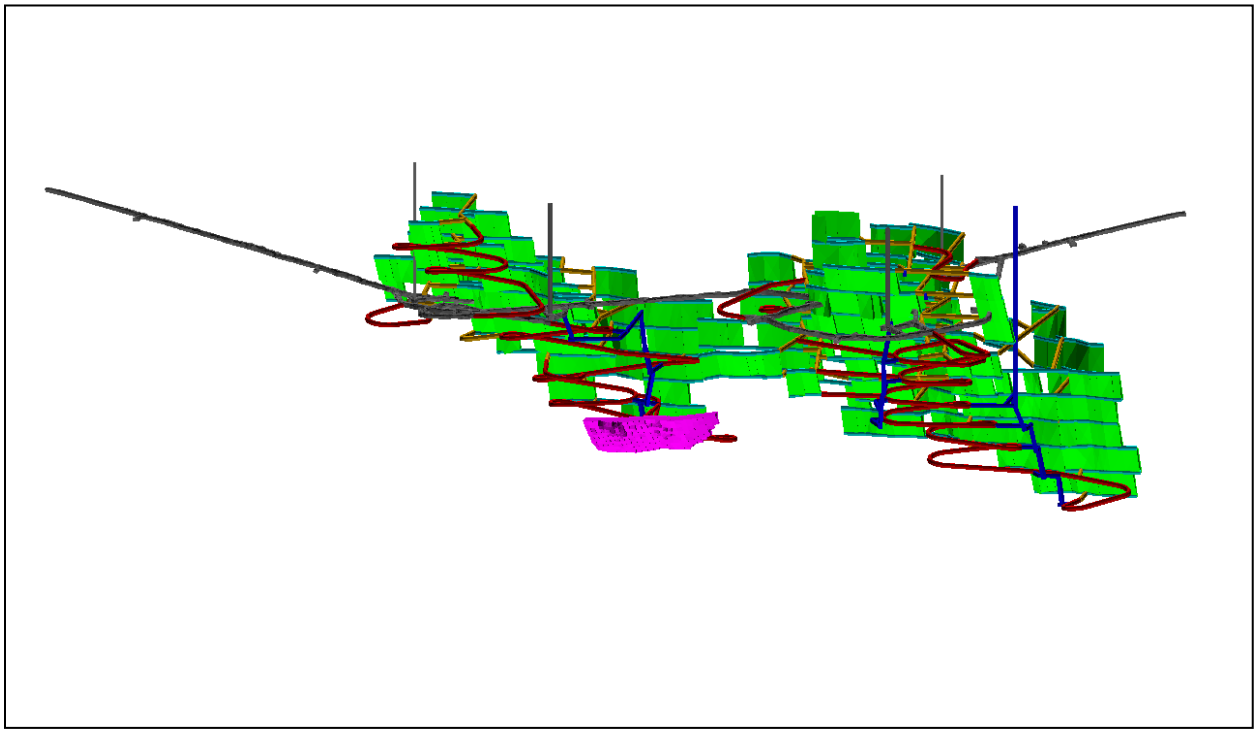
The total mining dilution for longhole stopes was calculated to be 14% by mass for primary and secondary stopes combined. The underlying assumptions were 0.50 m overbreak on the hanging wall and footwall. Dilution grades were pulled from the geologic block model. The minimum stope width is 2.5 m before dilution.

16.6 Mine Design

16.6.1 Design and Optimization

Mine planning for the Cerro Blanco Project was conducted by JDS using Maptek Vulcan 3D and Minemax iGantt software. A long section of the complete mine design is shown in Figure 16.10.

Figure 16.10: Mine Long Section



Source: JDS (2017)

Mine design was carried out based on a gold cut-off grade calculation (COG) for each block in the resource model. The COG was calculated based on estimated gold price, metallurgical recoveries, mining, processing, and general and administrative (G&A) operating costs, and mining dilution (see Table 16.2).

Table 16.6: Cut-off Grade Calculation Inputs

Parameter	Unit	Value
Revenue, Smelting & Refining		
Gold price	US$/oz Au	1,250
Payable metal	%	99.8
TC/RC/Transport	US$/oz Au	1
Royalty	US$/oz Au	87.26
Gold value per ounce	US$/oz	1,159
Gold value per gram	US$/g	37.27
OPEX Estimates		
Mining	US$/t milled	55
Processing	US $/t milled	34.7
G&A	US $/t milled	10.45
Infrastructure	US $/t milled	10.95
Sustaining CAPEX	US $/t milled	0
Total OPEX estimate	US $/t milled	111.1
Recovery and Dilution		
Process Recovery	%	93
Dilution	%	15
Cut-off Grade		
In-situ cut-off Au grade	g/t	3.7

Source: JDS (2017)

Vulcan Stope Optimizer software was used to create optimized stope shapes. Stope shapes must meet certain design constraints such as dip, width, and COG. Once the LH stopes were designed, the remaining areas were designed using DF parameters. The cut-off grade used to optimize the production LH stopes and DF mining areas was 3.75 g/t gold. After the LH and DF production areas were designed, the remaining mine development and access was designed. Minemax iGantt was used to optimize the mine production schedule by maximizing the NPV, subject to constraints including maximum lateral development rates, maximum production rates, maximum backfill rates, minimum backfill cure times, and extraction sequence.

16.6.2 Access

The Cerro Blanco deposit will be accessed via two ramps: one ramp for the north portion of the deposit, and one for the south. The ramps will be used to haul mineralized material and waste and used as general access. The decline will also be used as a fresh airway.

The North and South portals are currently located at 496 and 490 masl, respectively. The south decline will descend to a final depth of approximately 280 m below the existing portals to an elevation of 210 masl, and the north decline will reach a final elevation of 270 masl.

The dual ramps will connect to the existing underground workings and utilize the current portals. The north portal is located close to the proposed mill location to minimize mine to mill haulage.

The size of the decline was selected according to required clearances for the chosen mobile equipment and required ventilation during development and production. It was determined that a 4.5 m wide by 4.5 m high profile would be suitable for a 30 t haul truck. The declines will be driven at a -15% gradient. Level access crosscuts and attack ramps are planned to be developed off the decline at a 4.5 m by 4.5 m profile.

Level access crosscuts are designed to be located every 30 m vertically along the spiral decline to provide access to the potentially mineable resource. Attack ramps will provide access from the access drifts directly to each mining level. The DF areas will be accessed via attack ramps with a maximum gradient of +/- 15%. Once a given level has been completely mined and backfilled, the back of the attack ramp access is planned to be slashed down and a ramp will be constructed with the slashed rock to access the next cut above.

16.7 Mine Services

16.7.1 Mine Ventilation

The ventilation system for the Cerro Blanco operation has been designed to dilute and remove dust, diesel emissions, blast fumes and provide cooling of the mine workings. The ventilation network was modelled using Ventsim software. The Cerro Blanco deposit requires additional air to pull away excess heat and control air temperatures.

A total of five ventilation raises are required to ventilate the mine. The Cerro Blanco Project currently has four ventilation raises, therefore only one new ventilation raise is required. As the declines are being developed, a series of ventilation drop raises will be developed concurrently. Three ventilation raises are planned to be used as exhaust raises. The remaining two raises are planned to be utilized for fresh air intake, along with the two portal ramps. The return air raises (RAR) will be required to keep air velocity on the ramp at or below 6 m/s. The fresh air raise (FAR) will also act as a means of secondary egress. The new raise will be developed using a raise climber at a 4 m by 4 m profile. Lateral ventilation drifts at 4 m by 4 m profile will be required to follow the decline and connect the ventilation circuits to the decline and level access.

Minimum airflow requirements were based on expected diesel emissions of the underground mining fleet required at peak mine production. Additional airflow is used underground for general cooling. The power rating of each piece of equipment was determined, and the utilization factors representing the equipment in use at any time, were applied to estimate the amount of air required. Equipment specified for site has undergone testing by either MSHA or CANMET to determine the ventilation requirements to dilute the engine emissions to a safe working level. The volume of air required for ventilating the diesel emissions is 165 m^3/s. An additional 25 m^3/s is used for cooling and with a 10% safety factor, the final airflow requirements for the mine were calculated at 205 m^3/s.

Auxiliary fans will be used to ventilate the advancing development and active production levels. Fresh air will be sourced from the FAR and distributed using the auxiliary fans through ventilation ducting to the active mine areas.

In order to control the underground ventilation temperatures, underground mining equipment will be purchased with sealed and air conditioned cabs. When working at the active face, portable spot coolers will be used. A chiller plant located on surface will pump cold water through insulated pipes to the spot coolers used at the active faces underground to cool the air to approximately 28° C.

16.7.2 Water Supply

Service water will be required mainly for drilling, dust suppression and washing of development faces. Water will be supplied from a service water tank located close to the portal and will be gravity fed to the underground work areas via 100 mm diameter pipelines. Pressure reduction valves will be installed along the decline as required. The service water tank will be refilled with cooled underground mine water or externally sourced water.

16.7.3 Dewatering

Inflows into the underground workings were estimated at 25 L/s for Year 1 through 3 and 38 L/s for Year 4 onwards to the end of the mine life.

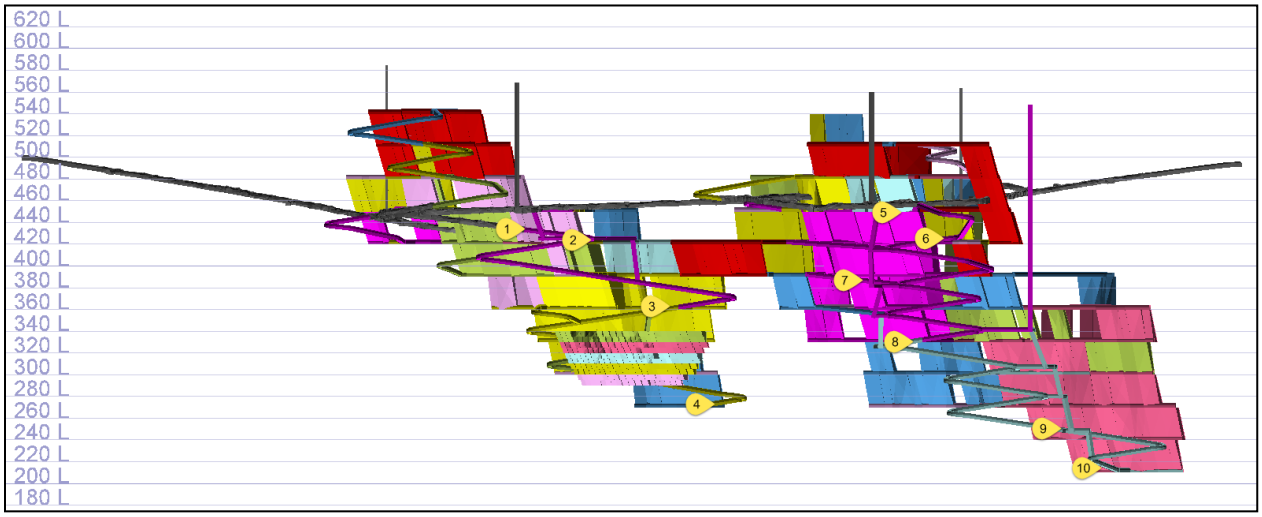
The mine dewatering system is designed as two standalone systems, the northern system and southern system. Both systems have been designed to accommodate a peak flow of 30 L/s, and will use a combination of 4-inch piping and 4-inch drill holes drilled between levels to transport water. A summary of the dewatering system is summarized in Table 16.3. The location of the sumps and pumps are show in Table 16.7 and Figure 16.11.

Table 16.7: Underground Dewatering System

Sump No.	Section	Elevation (masl)	Pumps to	Pump (kw)	Booster (kw)
1	Northern	412	Surface via North Ramp	104	56
2	Northern	420	N Main Sump (up and around)	43	
3	Northern	360	N Main Sump	43	
4	Northern	275	360 Sump	104	56
5	Southern	440	Surface via south Ramp	104	
6	Southern	420	South Main	43	
7	Southern	330	S 420	56	
8	Southern	330	S 390	104	
9	Southern	250	S 330	104	56
10	Southern	210	S 250	104	

Source: JDS (2017)

Figure 16.11: Sump and Pump Station Locations



Source: JDS (2017)

16.8 Unit Operations

16.8.1 Drilling

Development headings are planned to be driven with electro-hydraulic two-boom jumbos. Blast holes with 48 mm diameter will be drilled to a depth of 4.88 m. The advance per round is assumed to be 4.4 m. It is envisioned that one jumbo could drill between two to three rounds per shift.

Production drilling for the longhole stopes will be performed by longhole drills. Blast holes with 89 mm diameter will be drilled in a fan pattern from the overcut to the undercut.

16.8.2 Blasting

Development rounds will be charged by an explosives and ANFO loader. Lifter holes will be loaded with bulk emulsion. Blasting is planned to be initiated by non-electric (NONEL) detonators.

For longhole production blasting, bulk emulsion will be used together with NONEL detonators and Pentex boosters.

16.8.3 Ground Support

After mucking and scaling is complete, ground support will be installed by a mechanized scissor bolter. Typical ground support in access development is planned to consist of 2.4 m long resin rebar bolts in the back and in the walls at a spacing ranging from 1.5 x 1.5 m in moderate and poor ground. Welded wire mesh will be installed in all ground conditions. The anticipated breakdown between good, moderate, and poor is 25%, 50%, and 25%, respectively. In intersections, 3.0 m bolts will be used for deep ground support.

It was assumed that 25% of the development will be in poor ground conditions, which would require shotcrete. A shotcrete machine will be used to apply shotcrete at 60 mm thickness.

16.8.4 Mucking

Blasted material from development headings will be mucked with a 6.0 yd^3 (10 t) LHD directly to a haul truck or to a remuck bay. Broken material from longhole stopes will be mucked by remote control LHD.

16.8.5 Hauling

30 t haul trucks will drive on the decline to surface, where they will dump the material on mineralized material or waste stockpiles in close proximity to the portal.

Haulage profiles for all production levels were generated to calculate equipment hours for the fleet.

16.8.6 Backfill

The selected mining methods require the placement of backfill for full extraction of the mineralized zones. Stopes require the use of cemented paste backfill to provide stability to exposed backfill walls when mining the adjacent stopes. The use of paste backfill will also minimize the storage requirements for process plant tailings on surface. The paste will be mixed at a paste plant and pumped through pipelines underground to the stopes. A cement content of 8% was assumed for cemented paste fill of primary stopes. Due to the high mica content of the mineralized material, 46% of the paste recipe will be crushed and screened waste rock, and 46% will be tailings from the plant. Further test work will be required to determine the optimum cement content, curing time and achievable backfill strength.

Underground development waste may be used for un-cemented backfill in attack ramps and remote stopes to minimize waste haulage to surface.

16.9 Mine Equipment

The mobile equipment fleet to support the mining operation is summarized in Table 16.8.

Table 16.8: Mobile Equipment Fleet (average number of units)

Underground Equipment	Year -1	Year 1	Year 5
Haulage Truck - 30 tonne	2	4	5
LHD (10 t/6 yd)	2	5	5
Drill Jumbo - 2-Boom	2	2	2
Bolter	2	2	2
Explosives Truck	1	2	2
Drill Longhole	1	2	2
Scissor Lift	1	2	2
Shotcrete Sprayer	1	1	1
Personnel Carrier	1	2	2
Lubrication Service Truck	1	1	1
Boom Truck	1	1	1
Transmixer	1	1	1
Motor Grader	1	1	1
Utility Vehicle	1	1	1

Underground Equipment	Year -1	Year 1	Year 5
Backhoe with Rock Breaker	1	1	1
Telehandler	1	1	1
Mechanics Truck	2	2	2
Supervisors/Tech Services	4	6	6

Source: JDS (2017)

16.10 Mine Personnel

The underground mine is planned to operate on two 12-hour shifts (day shift / night shift), 365 days per year with four crews on rotation. Two crews will be on-site at any time with the other crews off-site on break. Both hourly mining and maintenance personnel and salaried supervisors and technical staff will work on the same four days on, four days off rotation.

Hourly personnel were estimated based on development and production rates, operation productivities and maintenance requirements.

Underground mining personnel requirements are summarized in Table 16.9.

Table 16.9: Underground Mine Operations Personnel

Position	Avg. Quantity	Hourly/Salary
Mining Management		
Mine Manager	1	Salary
Mining Superintendent	1	Salary
Maintenance Superintendent	1	Salary
Technical Services Superintendent	1	Salary
Mine Foreman	1	Salary
Mine Clerk	1	Salary
Mining Operations (Production)		
Shift Supervisor	4	Salary
Blasting Supervisor	4	Salary
Trainer	10	Hourly
Blaster	7	Hourly
Development Services/Shotcrete	6	Hourly
Waste Development Miner	7	Hourly
LH Production Miner	7	Hourly
Scooptram Operator (Large)	19	Hourly
Haul Truck Operator	18	Hourly
Bolter Operator	7	Hourly
Jackleg/Stoper Miner	8	Hourly
Grader Operator	4	Hourly
Mining Operations (Services)		
Paste Plant Operators	8	Hourly
Backfill Miner	8	Hourly
Backfill Helper	8	Hourly
Mine Electrician	8	Hourly
Mine Maintenance		
Maintenance Supervisor	1	Hourly
Maintenance Planner	1	Hourly
Heavy Equipment Mechanic	16	Hourly
Mechanic Helper	4	Hourly
Welder	8	Hourly
Electric/Hydraulic Mechanic	4	Hourly
Mining Technical Services		
Senior Mine Engineer	1	Salary
Geotechnical Engineer	1	Salary
Chief Geologist	1	Salary
Ventilation Engineer	1	Salary

Position	Avg. Quantity	Hourly/Salary
Mine Surveyor	2	Salary
Surveyor Helper	2	Salary
Geologist	4	Salary
Sampler	4	Salary
Short Term Mine Planner	1	Salary
Project Engineer	1	Salary
Long-Term Mine Planner	1	Salary
Technician	2	Salary
Total Underground	**194**	

Source: JDS (2017)

16.11 Mine Production Schedule

Mine scheduling for the Cerro Blanco Project was conducted by JDS using Minemax iGantt software. The scheduler seeks to optimize the NPV of a 1,250 t/d operation subject to constraints of development rates, production rates, and backfill rates.

Underground production was considered to have started as soon as first mineralization is mined. Mining blocks with higher profitability (net $/t) mineralization were targeted in the early stages of the mine life to optimize Project economics.

Annual mine production statistics are provided in Table 16.10.

Table 16.10: Annual Mineralized Material, Waste and Backfill Schedule

Year	Unit	TOTAL	Year									
			-1	1	2	3	4	5	6	7	8	9
Mineralized Material Mined												
Tonnes	kt	3,981	14	456	456	456	456	456	456	456	456	317
Avg Au Grade	g/t	8.1	9.9	10.3	10.8	9.5	10.5	7.7	6.4	6.7	5.8	4.6
Avg Ag Grade	g/t	27.9	39	45.9	42.6	37.5	14.8	19.9	18.1	32.3	11.3	29.1
Contained Metal												
Au	koz	1,042	4	151	158	139	155	112	94	98	85	47
Ag	koz	3,577	17	674	625	550	218	292	266	474	166	297
Waste Mined	kt	674	85	103	87	82	45	58	71	43	70	30
Backfill Placed	kt	1,292	5	134	133	156	152	160	151	166	141	94

Source: JDS (2017)

16.12Mine Development Schedule

The development schedule was based on estimated cycle times for jumbo development.

All waste development during pre-production is shown as capital development.

During the production phase, the decline, ventilation drifts and raises are considered sustaining capital development, but crosscuts and drifting on the levels were included in the operating costs.

Annual development metres are summarized below in Table 16.11.

Table 16.11: Annual Development Schedule

Year	Unit	TOTAL	Year									
			-1	1	2	3	4	5	6	7	8	9
Capital Lateral	m	6,586	1,260	1,221	913	1,263	73	382	524	353	509	87
Capital Vertical	m	630	311	68	26	157	0	39	29	0	0	0
Operating Lateral	m	20,263	897	1,875	1,941	1,816	2,913	2,628	2,504	2,628	1,993	1,068

Source: JDS (2017)

17 Process Description/Recovery Methods

The process design criteria and flowsheets were developed based on the metallurgical test work results from historical test work programs, as described in Section 13, using industrial design factors as noted. The test work has shown that Cerro Blanco mineralization can be treated using three stage crushing, grinding, leaching, CCD thickening and Merrill Crowe for the recovery of gold and silver doré.

The plant will process material at a rate of 1,250 t/d with an average LOM head grade of 8.14 g/t gold and 27.95 g/t silver. Based on test work, the overall LOM metal recoveries are expected to be approximately 91.3% for gold and 87.8% for silver. The grinding circuit product size is targeted at approximately P_{80} 75 µ. The flowsheet includes 3-stage crushing, 2-stage ball mill grinding, leaching, CCD thickening, Merrill Crowe and gold refining. The tailings will be filtered and trucked as dry stack tailings to the surface tailings facility or used for underground paste backfill. The crushing circuit will operate at an availability of 65%. The milling and leaching circuits will operate 24 hours per day, 365 days per year at an availability of 92%.

17.1 Introduction

The plant will consist of the following unit operations:

- 3-stage crushing and fine mineralized material storage;
- Primary and secondary grinding;
- Cyanide leaching and CCD thickening;
- Cyanide destruction;
- Merrill Crowe and gold refining; and
- Tailings filtering for dry stack placement or used underground as paste backfill.

17.2 Process Design Criteria

The process design criteria and mass balance detail the annual production capabilities, major mass flows and capacities, and plant availability. Consumption rates for major operating and maintenance consumables can be found in the operating cost estimate described in Section 22. Key process design criteria are outlined in Section 13 and Table 17.1.

Table 17.1: Process Design Criteria

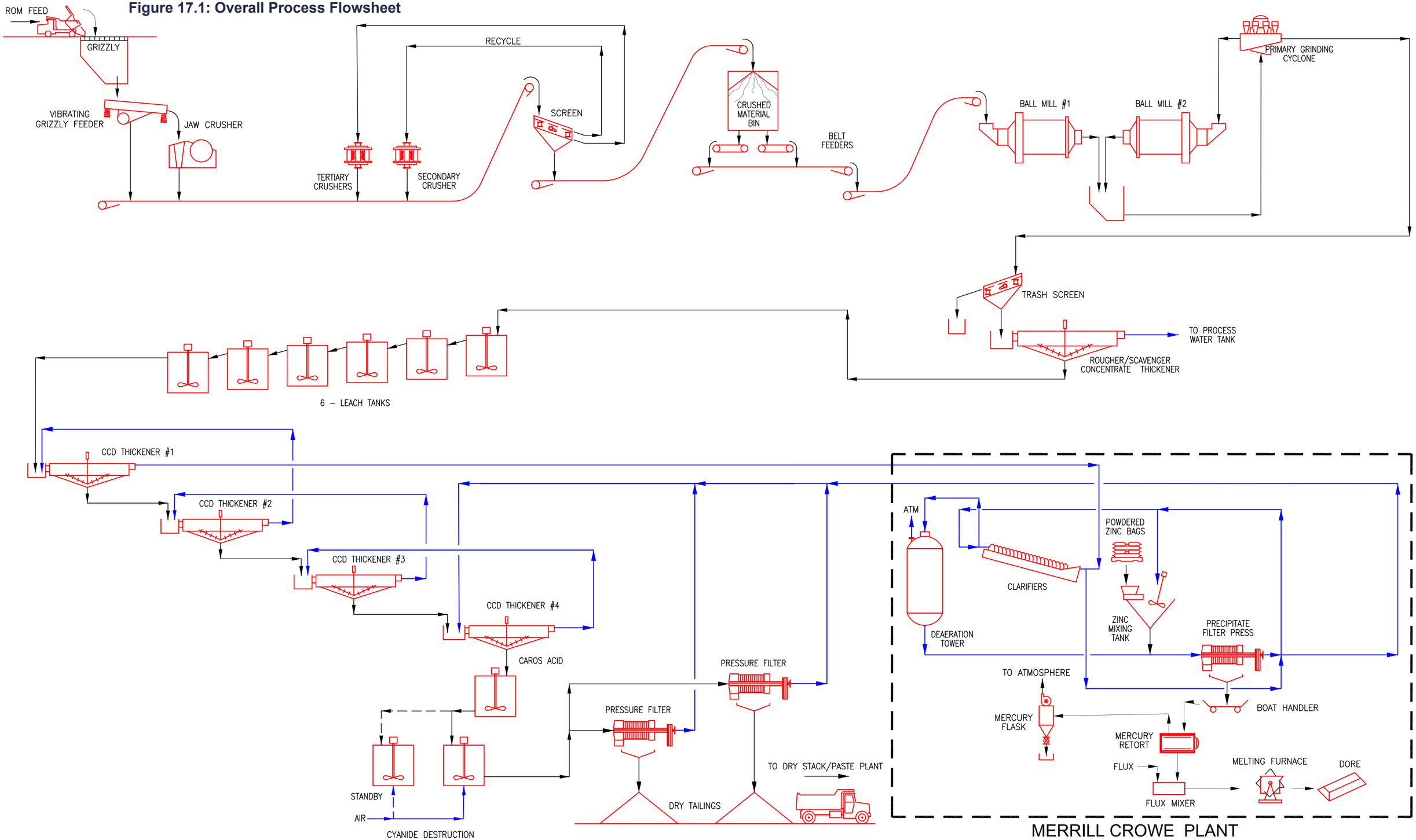
Area	Criteria	Unit	Nominal Value
General	Gold	g/t	8.14
	Silver	g/t	27.95
	Daily throughput	t/d	1,250
	Process plant availability	%	92
	Overall Gold recovery	%	91.3
Crusher System	Availability/Utilities	%	65
	Crusher work index	kWh/t	5.4
	Number of crushing stages	-	3
	Crushing System product size (P_{80})	mm	8 to 10
Fine Mill Feed Storage	Bin Capacity (live)	t	400
	Bin Capacity (live)	h	8
Grinding	Bond Ball mill work index	kWh/t	19.9
	Ball #1 & 2 mill size	m dia. x m long	3.7 x 5.5
	Ball #1 & 2 installed power	kW	933
	Ball mill #1 product size (P_{80})	µm	500
	Final product size (P_{80})	µm	75
Pre-leach Thickening	Thickener loading rate	$m^3/m^2/h$	3.10
	Thickener underflow density	% w/w	50
	Thickener Diameter	m	8
Leaching and CCD Thickening	Leach retention	h	96
	Number of leach tanks	#	6
	Leach tank dimensions	m dia. x m high	12 x 13
	Leach slurry feed rate	m^3/h	90
	Number of CCD Thickeners	#	4
	Thickener diameter	m	12
	Thickener underflow density	%	50
	CCD#1 thickener O/F to Merrill Crowe	m^3/h (design)	200
Cyanide Destruction	Discharge solution CNWAD	mg/L	<1.0
	Caro's Acid Tank	#	1
	Design Residence time	h	2.0
	Number of Tanks	#	2 (1 operational, 1 standby)
Tailings Filtration	Stock tank residence time	h	8
	Filtration Rate (Similar Projects)	$kg/m^2/h$	500
	Filter Cake Moisture Content	%	18

Source: JDS (2017)

17.3 Plant Design

A summary of the process flowsheet is shown in Figure 17.1.

Figure 17.1: Overall Process Flowsheet



REV. No.	ISSUED FOR	REVISION DESCRIPTION	DR.	CHK.	APP.	DATE

REV. No.	ISSUED FOR	REVISION DESCRIPTION	DR.	CHK.	APP.	DATE	REFERENCE DRAWING	REF. DWG No.

SCALE: None	DATE:
DESIGNED: J.D.S.	Jan 15 2017
DRAWN: B. Wong	Jan 15 2017
CHECKED:	
APPROVED:	

BLUESTONE RESOURCES INC.

JDS Energy & Mining Inc.

TITLE
BLUESTONE RESOURCES
CERRO BLANCO PEA – 1,250 tph
SIMPLIFIED PROCESS FLOWSHEET

PROJECT No.	DRAWING No.	REV No.
	100–PF–001	

17.4 Process Plant Description

17.4.1 Primary Crushing

Material from the underground mining operations will feed a vibrating grizzly - primary jaw crusher system, which will produce a crushed product size P_{100} of 115 mm.

Material will be stockpiled near the jaw crusher or direct dumped through a static grizzly into a dump pocket. Stockpiled run of mine (ROM) material will be re-handled by a front-end loader and fed into the crusher. The material will discharge through a static grizzly into a feed hopper. Oversize material from the static grizzly will be removed, for later size reduction, using a rock breaker.

A vibrating grizzly feeder will draw material from the dump pocket at a rate of 80 t/h. The vibrating grizzly oversized material will discharge directly into the primary jaw crusher. The undersized material will bypass the crusher and will be fed directly onto the secondary screen feed conveyor.

17.4.2 Secondary Crushing & Screening

The secondary screen feed conveyor will collect primary crushing circuit product and feed a triple deck vibrating screen. The oversize material from the top two decks, at a size of +30 mm, will be conveyed to the secondary crusher surge bin.

Material from the secondary crusher feed bin will be reclaimed by a belt feeder controlled by a variable speed drive into the secondary cone crusher. The secondary crusher will reduce the material to a nominal product size of approximately P_{80} of 28 mm with a closed side setting (CSS) of 25 mm.

17.4.3 Tertiary Crushing

Material from the third (lower) deck of the secondary screen will feed the tertiary crusher. The tertiary crusher will reduce the material to a nominal product size of approximately P_{80} of -10 mm with a CSS of 10 mm. Both the secondary and tertiary cone crusher products will be recycled back to the secondary screen, together with the jaw crusher circuit products.

17.4.4 Crushed Material Storage Bin

The secondary screen undersize, with a final product size P_{80} of 8 mm will be conveyed to the crushed material storage bin. The storage bin will have a capacity of 400 t, equivalent to approximately eight hours live capacity. Two belt feeders under the bin will draw material to feed the primary ball mill. Each belt feeder is capable of providing the total throughput of 57 t/h to the primary ball mill

17.4.5 Grinding

The grinding circuit will consist of a primary ball mill followed by a secondary ball mill and gravity recovery circuit, operating in closed circuit with a hydrocyclone cluster. The grinding circuit will operate at a nominal throughput of 57 t/h (new feed), and produce a final particle size P_{80} of 75 µm. The ball mills will both be 3.5 m in diameter by 5.5 m in length driven by 933 kW motors.

Water will be added to the primary ball mill to maintain the slurry charge in the mill at a constant density of 70%. Slurry will overflow from the ball mill to a trommel screen, attached to the ball mill discharge end. The ball mill trommel screen oversize will overflow into a trash bin for removal from the system.

The slurry from the primary ball mill will combine with the secondary ball mill discharge in the cyclone feed pumpbox. The hydrocyclone cluster will classify the feed slurry into coarse and fine fractions. The coarse underflow will feed the secondary ball mill for additional grinding. The fine cyclone overflow will flow by gravity to the pre-leach thickener. The ball mill hydrocyclones have been designed with a 300% recirculating load to the secondary ball mill.

17.4.6 Pre-leach Thickening

The cyclone overflow slurry will flow by gravity to the vibrating trash screen for removal of trash material.

The undersize from the screen will flow by gravity to the pre-leach thickener. Flocculant solution (anionic polyacrylamide) will be added to the thickener feed to promote the settling of fine solids. The thickener, which is 8 m in diameter, will thicken the slurry to approximately 50% solids. Thickener overflow will flow by gravity into the process water tanks to be used as make-up water in the grinding circuit and the thickener underflow will be pumped to the leach circuit.

17.4.7 Cyanide Leaching and CCD Thickening

The pre-leach thickener underflow will be pumped to six 12.0 m diameter by 12.0 m high mechanically agitated leach tanks. The circuit will be operated as a single leach train providing 96 hours of residence time. All leach tanks will be located outside and adjacent to the main process building.

Lime slurry will be added at a rate of 5.3 kg/t to the first and second leach tanks to maintain protective alkalinity at a design pH of 10.5 - 11.0, to prevent evolution of hydrogen cyanide gas (HCN). Sodium cyanide at a concentration of approximately 3,000 ppm will be used to leach the gold and silver into solution. Oxygen will be sparged to the bottom of the leach tanks to promote the leach reaction.

The slurry from the last leach tank will be pumped to a series of four 12 m diameter CCD thickeners. The slurry will flow from the first thickener through to the last thickener. The overflow solution will flow countercurrent to the slurry flow to recover the majority of the soluble metals as pregnant solution at the first thickener overflow. The thickener overflow solution will report to the Merrill Crowe circuit and the thickener underflow to the cyanide destruction circuit.

17.4.8 Merrill Crowe and Refinery

The solution from the first CCD thickener will be pumped through a clarifier filter to remove any residual particulate, and then oxygen will be removed from the clear solution by passing it through a vacuum de-aeration column. Zinc dust will be added to the clarified, de-aerated solution, to precipitate the gold and silver.

The solution containing the gold and silver precipitate produced will be filtered. The filter solids will be dried and then smelted in an induction furnace to produce doré. The barren filtrate solution will circulate back to the last thickener in the CCD train as the wash solution.

17.4.9 Cyanide Destruction

The cyanide destruction circuit will consist of two mechanically agitated tanks, each with a capacity to handle the full slurry flow for the required residence time of 1.5 hours. Cyanide will be destroyed using the SO_2/Air process.

The cyanide destruction circuit will treat thickened slurry from the CCD circuit, process spills from various contained areas and process solution bleed streams.

Process air will be sparged from near the bottom of the tanks, under the agitator impeller. Lime slurry will be added to maintain the optimum pH of 8.0 – 8.5, and copper sulphate ($CuSO_4$) will be added as a catalyst, maintaining approximately 100 ppm concentration in the solution. Sodium metabisulphite (SMBS) will be dosed into the system as a solution as the source of SO_2. This system has been designed to reduce the total cyanide concentration to less than 1.0 mg/L CN_{WAD} prior to transfer to the tailings stock tank.

17.4.10 Tailings Management

Cyanide destruction tailings will be pumped to an 8-hour stock tank. The stock tank will feed one of two pressure filters to produce a cake with approximately 18% moisture. The filtered cake will discharge from the filter onto a concrete slab where a front-end loader (FEL) will be used to load the material either into trucks (for haulage to the dry stack tailings facility) or a hopper to feed the paste backfill plant.

17.4.11 Supply

An instrument and plant air system with compressors, dryers, filters, and receivers will be provided and located with the process air compressors in a compressor room inside the plant building. Compressed air will be used in the plant for instrument air, backup for the leach circuit and cyanide destruction. Blowers will be used to supply air to the leach circuit.

17.4.12 Reagents Handling & Storage

Reagents consumed within the plant are planned to be prepared and distributed via the reagent handling systems. The reagents that will be mixed on-site are NaCN, $CuSO_4$, SMBS, flocculant and lime. All reagent areas will be bermed with sump pumps which transfer spills to the cyanide destruction stock tank, with the exception of the flocculant and cyanide. Flocculant spills will be returned back to the flocculant storage tank and cyanide spills will be pumped to the leach circuit. The reagents will be mixed, stored and then delivered to the leaching, thickening and cyanide destruction circuits with dosage controlled by flow metres and manual control valves. The capacity of the various reagent storage tanks will be sized for one day consumption. The reagents will be delivered in dry form with the exception of antiscalant, which will be delivered as a solution.

17.4.13 Water Supply and Consumption

The following types of water will be used in the process plant:

- **Process water:** overflow water from the pre-leach thickener and tailings filter will be used as process water. This water will have a low gold concentration and will be used predominantly in the grinding circuit to dilute slurry to the required densities.
- **Make-up water:** the water system has flexibility to supply make-up water to the process from the following two sources:
    - Fresh ground water; and/or
    - Treated water from the discharge of the water treatment plant (WTP).

17.4.14 Water Management

At the Cerro Blanco mine site, water management is linked to the mine dewatering plan. This section addresses water management; however, the reader is referred to Section 16.4 for more specific details of the mine dewatering plan.

As presented in Section 16.4, the proposed mine plan includes the addition of 14 new dewatering wells to an average depth of 450 m, in addition to the 10 existing operational dewatering wells at Cerro Blanco. The preliminary design of the underground mine dewatering plan is such that a total of 24 wells (existing + new) will be extracting up to a maximum of 3,500 g/m of mine water, with an additional 500 g/m contribution expected from the underground mine sumps. Under current permits, 1,500 g/m is allowed to be sent to the existing WTP for treatment and discharge. The excess mine water that cannot report to the WTP will be reinjected through 10 new reinjection wells with a capacity of 250 g/m each. The installation and implementation of these new proposed injection wells is covered by the Cerro Blanco existing permits.

This PEA proposes to add one additional injection well in order to allow for flexibility to re-inject seepage and runoff water collected from the DSTF facility during rain events (refer to the DSTF infrastructure in Section 17.4.6).

17.4.14.1 Existing Cerro Blanco Water Treatment Plant (WTP)

The existing Cerro Blanco WTP is a modern facility designed for maximum discharge to the environment (receiving creek) of 1,500 g/m. The WTP is based on a physio-chemical treatment process designed to meet World Bank effluent guidelines for the discharge of treated water.

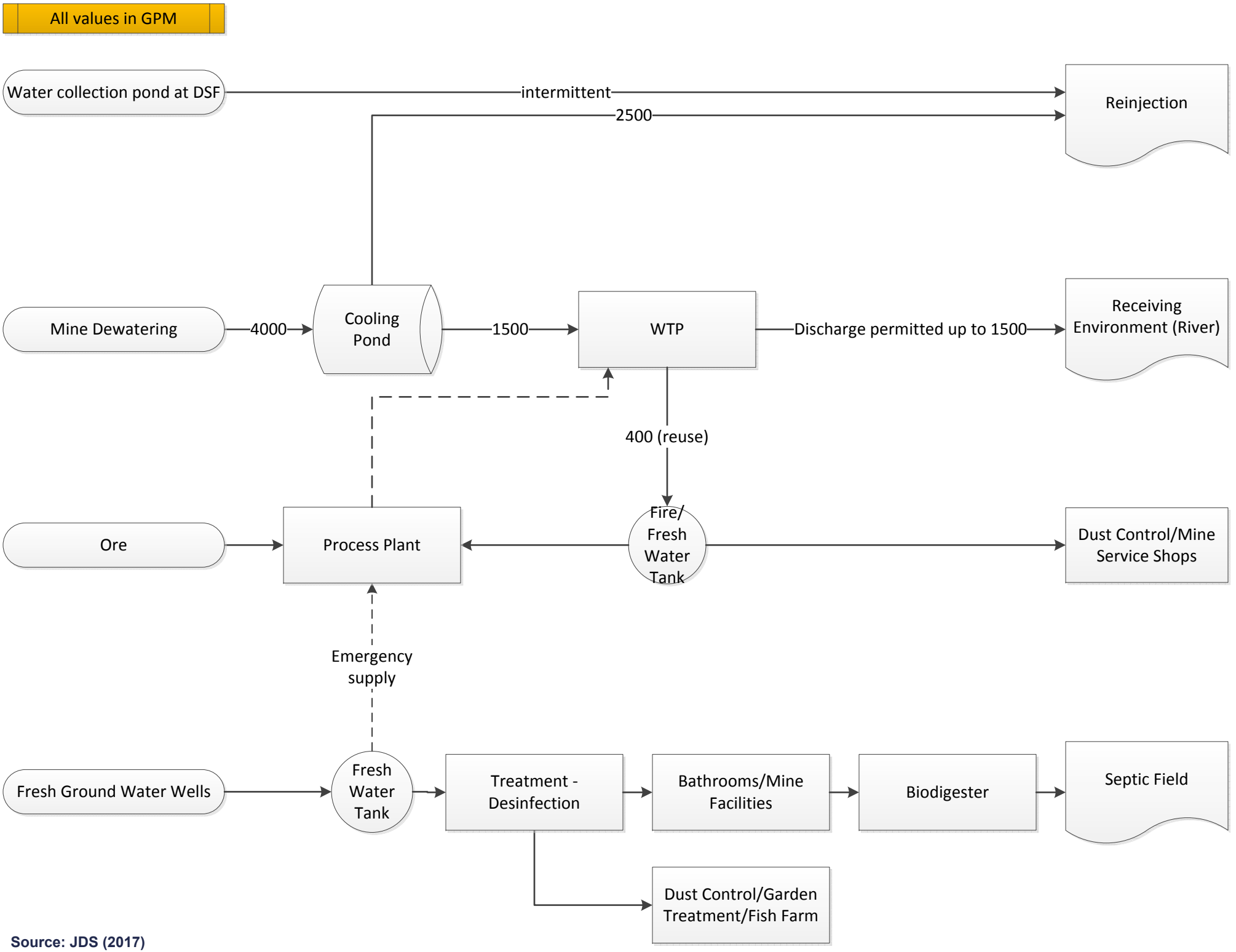
The function of the WTP is to receive and to treat up to 1,500 g/m of mine water that is extracted from the underground mine during dewatering. As seen in Figure 17.4 the Cerro Blanco water management system allows for the flexibility of re-using treated water from the WTP instead of discharging it. This optionality is important and will be maintained for the proposed future mine operation. The Project also recently received approval for its management plan for the treatment of sludge from the WTP.

17.4.14.2 Existing Cerro Blanco Site Water System

The diagram in Figure 17.4 presents the site-wide water system on which Cerro Blanco was permitted. The information collected as a part of this PEA demonstrated that this site water system is in

accordance with the current Cerro Blanco operations. A detailed site water balance will be developed in the next engineering design phase for both wet and dry seasons, in order to integrate the proposed mine operating conditions and requirements into the current water management system.

Figure 17.4: Site water balance



Source: JDS (2017)

17.4.14.3 Site Drainage

As part of the future engineering work, improvements to the site drainage infrastructures will be investigated in order to manage storm runoff in accordance with the specifics of the revised site configuration for the future operation. This will require a review and update of the site surface hydrology, with emphasis on heavy rain events that are expected to occur during the wet season. Presently, the Cerro Blanco site has an open channel to accommodate flash flood conditions. Runoff water during rain events generally leaves the site through gullies and natural depressions.

17.4.14.4 Domestic Wastewaters

The mine site is already equipped (under the existing permits) with a biological wastewater treatment unit and septic field that has the flexibility to be extended.

17.4.14.5 Potable Water

Currently, potable water is delivered to the site by a local water supplier. Other options for potable water will be assessed in the next study phase and reconciled with the potable water demand by the workforce during planned construction and operation.

17.4.15 Process Plant Personnel

Table 17.3 lists the process plant personnel requirements.

Table 17.2: Process Plant Personnel

Position	Staff/Hourly	Quantity
Operations & Maintenance		
Mill Superintendent	Staff	1
Mill General Foreman	Staff	1
Mill Shift Foreman	Staff	4
Crushing Operators	Hourly	4
Grinding Mill Operators	Hourly	4
Control Room Operators	Hourly	4
Flotation Operators	Hourly	4
Concentrate Dewatering & Handling	Hourly	4
Reagent Prep	Hourly	2
General Labourers (Shared)	Hourly	4
Subtotal - Operations & Maintenance		**32**
Process Technical Services		
Plant Metallurgist	Staff	2
Chief Chemist	Staff	1
Senior Metallurgical Engineer	Staff	1
Assay Technicians	Staff	2
Metallurgical Technician	Staff	2
Sample Preparation	Hourly	4

Subtotal - Process Technical Services		**12**
Total Process Plant Personnel		**44**

Source: JDS (2017)

18 Project Infrastructure and Services

18.1 General

The Project envisions construction of the following key infrastructure items:

- 5.5 km (new build) site access road (including an 80 m long bridge);
- 7.5 km 69 kV power line;
- On-site 69 kV to 13.2 kV substation;
- Process plant building;
- Primary crusher pad;
- Emergency power generator(s);
- Communications system upgrade;
- Bio-digester upgrade;
- 14 new dewatering wells;
- 11 new reinjection wells;
- Additional cooling pond;
- Warehouse;
- Truck shop upgrades;
- Fresh/Fire water tank;
- Process water tank;
- Bulk fuel storage upgrade;
- Dry stack tailings facility (DSTF);
- 1.0 km South Portal haul road; and
- 1.0 km North Portal and DSTF haul road.

18.2 General Site Layout

The overall site arrangement is shown in Figure 18.1.

The proposed site layout has been configured for optimal construction access and operational efficiency. Primary buildings have been located to allow easy access from the site access road and utilize existing topography to minimize bulk earthworks volumes. The primary crusher has been located as close as safely possible to the portal and at an elevation that facilitates mill feed conveying. Existing roads have been upgraded and reused wherever possible.

18.3 Site Access Road

The 2007 EIA limits site access for heavy vehicle and chemical transportation through the town of Asunción Mita. A new 5.5 km long access road is proposed to tie into the Pan-American Highway (CA1) approximately 3 km north of Asunción Mita. A new 80 m long bridge will be installed over the El Achotal River. This will also provide year-round access to the small community of El Cerron, which currently does not have sufficient road access.

18.4 Power Supply and Distribution

The anticipated peak operational electrical demand will be approximately 11 MW. Electrical power will be supplied from the nearby substation located in La Baranca. A 7.5 km 69 kV overhead power line and on-site substation will need to be constructed. The existing substation at La Baranca will require a spur and local disconnect.

18.5 Construction Power

Existing standalone generators will supply the 1.5 MW of power required during the construction phase.

18.6 Process Plant

The proposed process plant overall footprint is approximately 135 x 70 m. It contains grinding, leaching & CCD tanks, Merrill Crowe, filtration and detox, refinery, cyanide destruction, reagents preparation, dry stack tailings filtration, paste backfill plant and electrical rooms. Full enclosures will be provided for the milling, Merrill Crowe, refinery, electrical rooms, reagent preparation, paste backfill plant and electrical rooms. Open air facilities will include leaching, CCD tanks and cyanide destruction.

18.7 Primary Crusher

The primary crusher will not require a covered building. A 5 m tall Mechanically Stabilized Earth (MSE) wall will be constructed to allow loader access to the crusher dump pocket. The proposed crusher location has been optimized to allow easy access from the North and South portals while minimizing conveying lengths.

18.8 On-Site Power Generation

In the event of an on-site power outage, 3 MW of backup generation will be required. This will allow for underground mine dewatering, reinjection and critical process equipment to continue running. Currently, there is a 1.5 MW diesel generator installed on-site that could be used for operational backup. During operations, a second 1.5 MW diesel generator will be added to provide sufficient emergency power. During a power outage, a total draw of 3 MW is required.

18.9 Dewatering and Reinjection

Currently there are 10 existing dewatering wells that are reusable. In order to adequately remove groundwater, 14 additional dewatering wells will need to be installed. Each of these 14 wells will be 450 m deep and 12 ” in diameter. An additional cooling pond downstream of the existing south cooling pond will be constructed to accommodate for the increased flow. The estimated peak dewatering rate is 3,500 g/m from surface wells and 500 g/m from underground sumps.

The existing water treatment plant is permitted for 1,500 g/m. The remaining water that cannot be treated and safely discharged will need to be reinjected through a series of 11 new reinjection wells. These wells will be 150 m deep and 12“ in diameter. Each well is capable of accepting 250 g/m. One (1) of these reinjection wells is intended as a backup in the event of a 1 in 100 year storm.

18.10Truck Shop, Warehouse, Mine Dry and Administration Buildings

The existing mine dry and administration buildings will be utilized for operations. The existing truck shop will receive minor upgrades to accommodate the increased fleet size. A new 15 x 20 m warehouse will be constructed to complement the existing warehouse.

18.11On-Site Water Tanks

A new dual-purpose fresh/fire water tank will be erected with a 640,000 L capacity. Tank internal risers on all non-firewater suction lines will ensure a minimum volume of 470,000 L. This capacity will allow for approximately two hours of firefighting capability.

A new process water tank with 170,000 L capacity will also be erected adjacent to the fresh/fire water tank.

18.12 Bulk Fuel Storage and Delivery

The existing fuel storage facility will be expanded by adding a 50,000 L diesel tank with dispensing unit inside a contained and bermed area. This volume will provide approximately 48 hours of emergency genset fuel supply.

18.13 Haul Roads

The existing North and South portal access roads will be significantly upgraded to accommodate increased traffic. Both roads will be increased to 22 m wide and the north access road will be extended to provide access to the dry stack tailing facility.

18.14 Communications / IT

The existing communications tower currently provides sufficient access to stream data to and from site. Minor upgrades to the on-site infrastructure will be required.

18.15 First Aid / Emergency Services

A qualified nurse or first aid attendant will be provided on-site. The first aid room will be located besides the administration building. The ambulance and fire truck will be parked at the ready outside the process plant.

Buildings will be equipped with smoke, carbon monoxide and heat detectors appropriate and chemical fire extinguishers.

18.16 Explosives Storage and Magazines

The existing explosives storage and magazine will be used throughout the mine life.

18.17 Sewage Treatment

The existing bio-digester and field bed will be expanded to process the increased sewage rates during construction and operations.

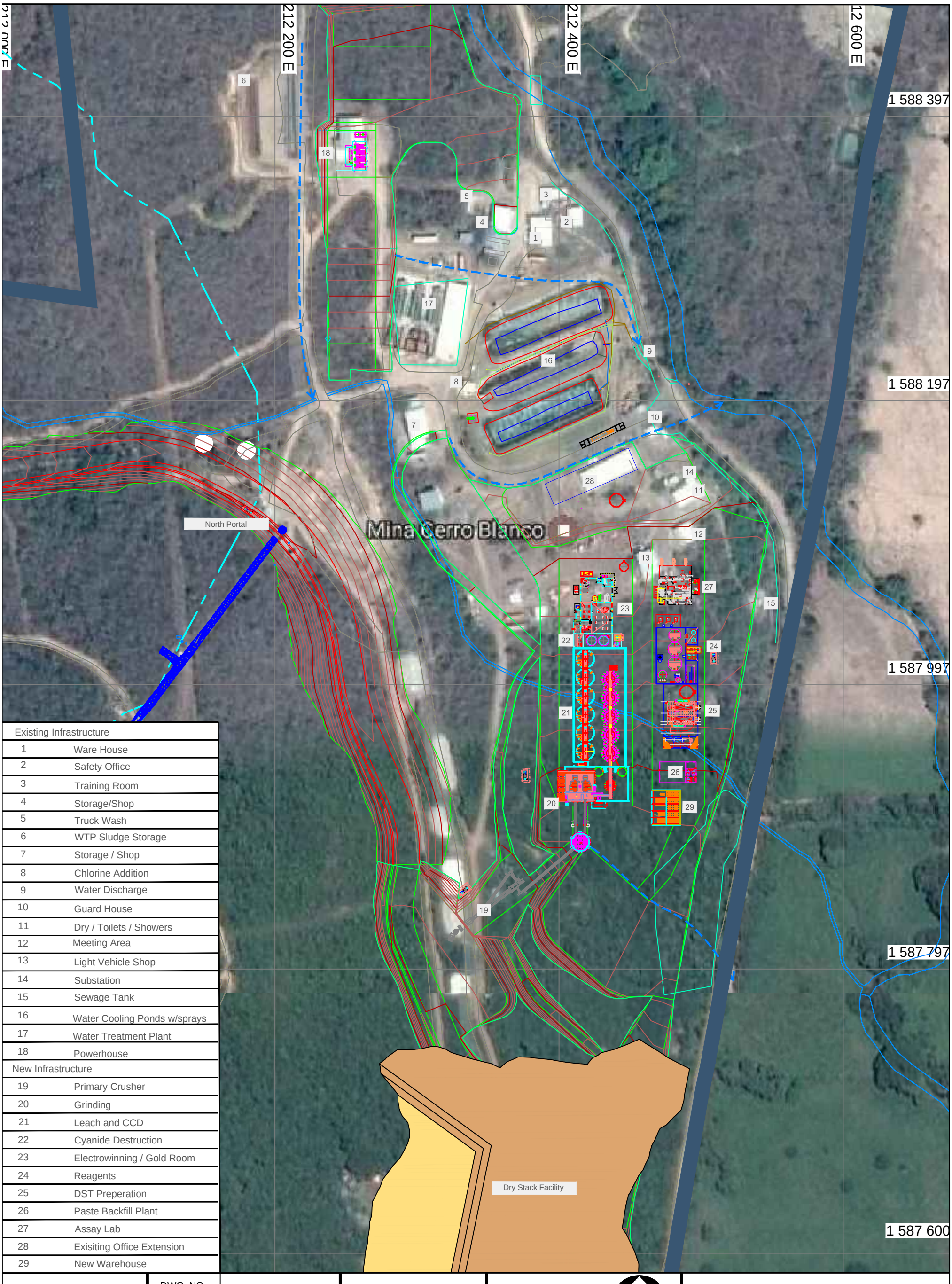
18.18 Freight

Two different options are proposed for shipping major equipment to site. The first would be from Los Angeles to Puerto Quetzal. The second would utilize Houston to Puerto Santo Thomas. From either port the materials would travel via truck to site.

18.19 Water

The water system diagram in Section 17.4.7 (water management) demonstrates that Cerro Blanco benefits from the flexibility of two sources of water supply: groundwater and reuse of treated water from the WTP. Identifying the optimal combination of these two sources of water supply will be assessed as part of the engineering work to follow.

Figure 18.1: General Plant Layout



THIS DRAWING HAS NOT BEEN PUBLISHED BUT RATHER HAS BEEN PREPARED BY JDS FOR USE BY THE CLIENT NAMED IN THE TITLE BLOCK SOLELY IN RESPECT OF THE CONSTRUCTION, OPERATION AND MAINTENANCE OF THE FACILITY NAMED IN THE TITLE BLOCK AND SHALL NOT BE USED FOR ANY OTHER PURPOSE OR FURNISHED TO ANY OTHER PARTY WITHOUT THE EXPRESS CONSENT OF JDS

DWG. NO.	REFERENCE DRAWINGS

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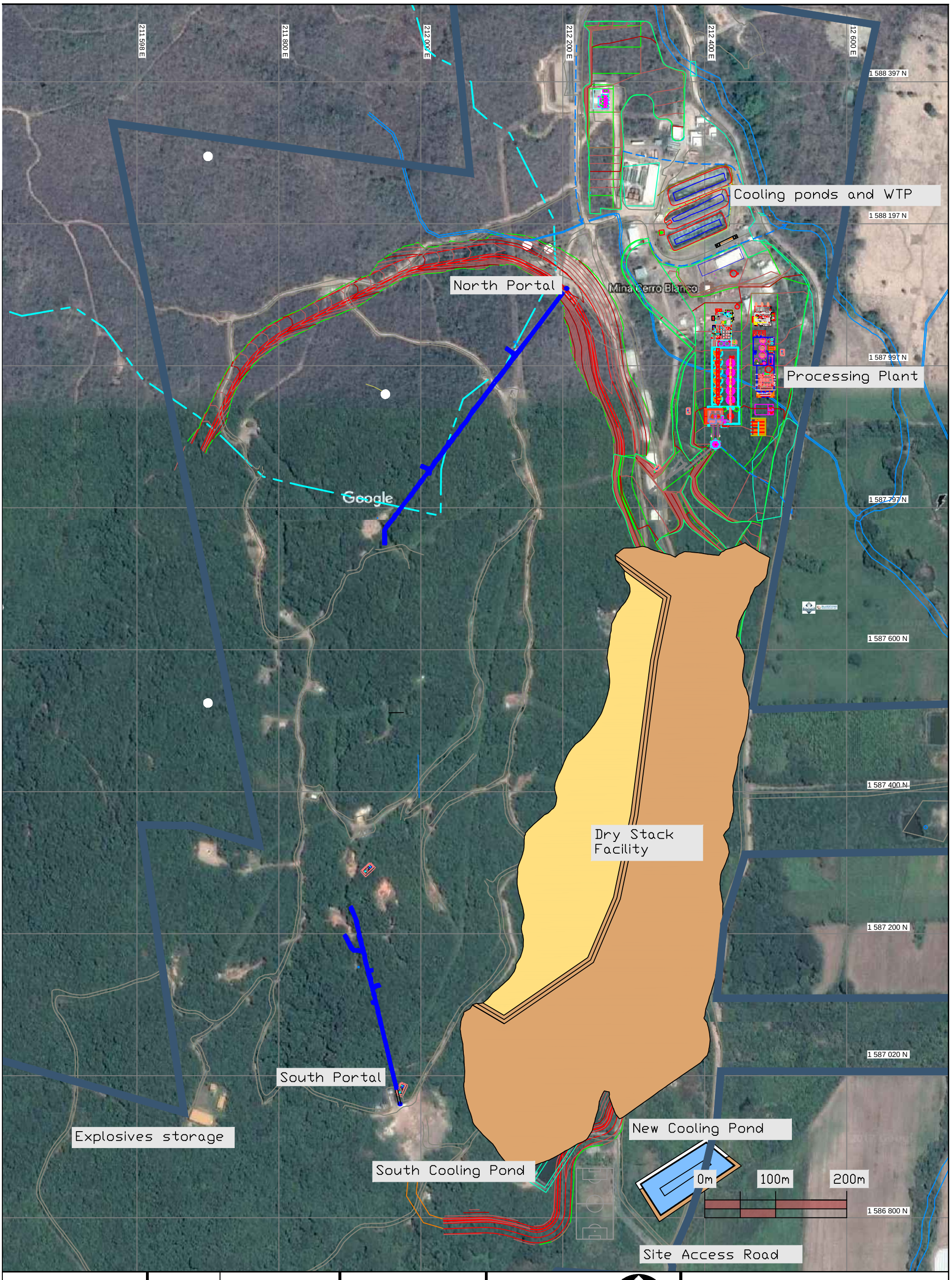
JDS Energy & Mining Inc.

BLUESTONE RESOURCES INC.

Cerro Blanco
General Site Layout

PROJECT NUMBER	DRAWING NUMBER	REVISION No.

Figure 18.2: Overall Site Layout



THIS DRAWING HAS NOT BEEN PUBLISHED BUT RATHER HAS BEEN PREPARED BY JDS FOR USE BY THE CLIENT NAMED IN THE TITLE BLOCK SOLELY IN RESPECT OF THE CONSTRUCTION, OPERATION AND MAINTENANCE OF THE FACILITY NAMED IN THE TITLE BLOCK AND SHALL NOT BE USED FOR ANY OTHER PURPOSE OR FURNISHED TO ANY OTHER PARTY WITHOUT THE EXPRESS CONSENT OF JDS

Legend	Description
	Property Boundary

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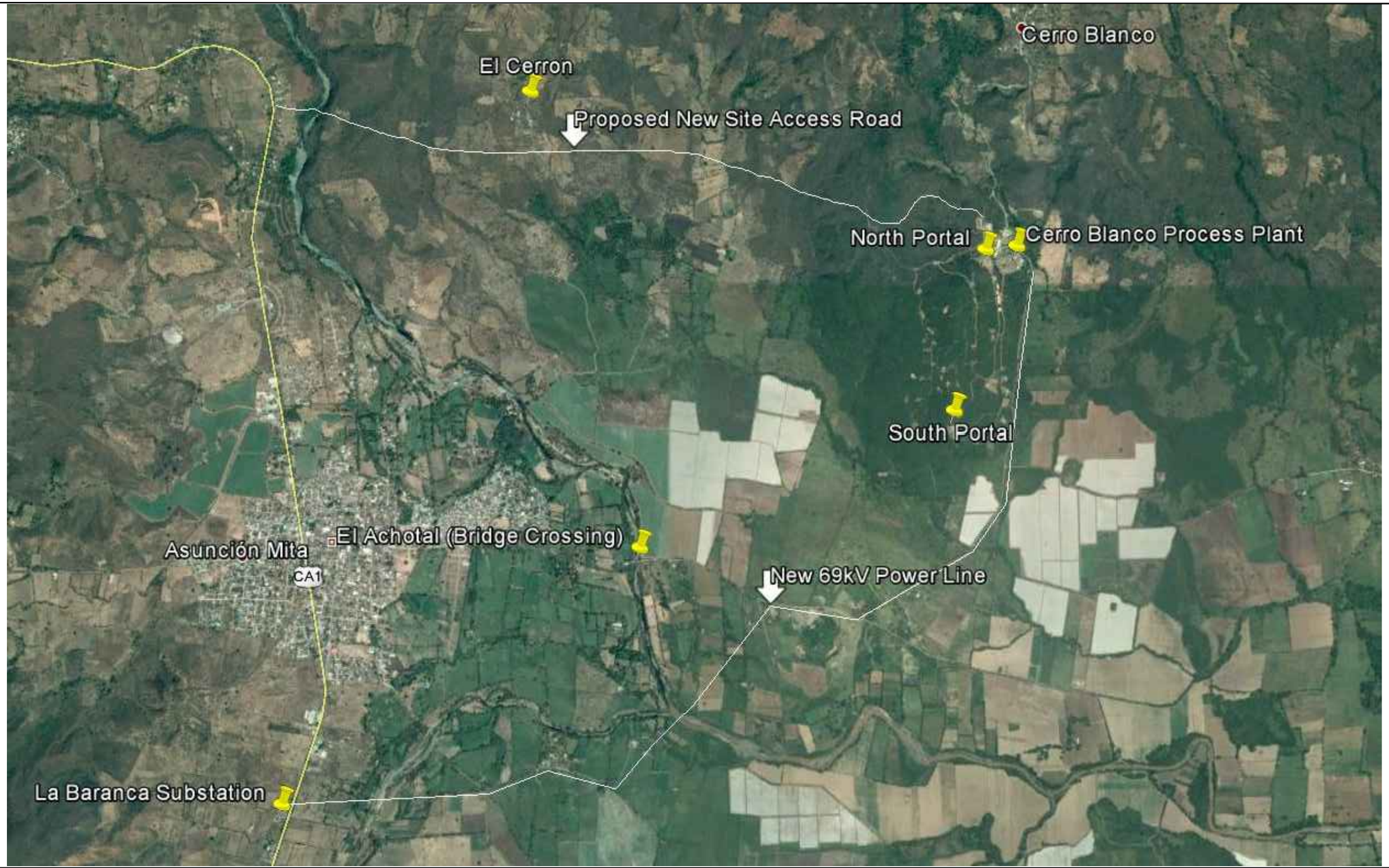
JDS Energy & Mining Inc.

BLUESTONE RESOURCES INC.

Cerro Blanco
General Site Layout

PROJECT NUMBER	DRAWING NUMBER	REVISION No.

Figure 18.3: Location of Major Infrastructure



THIS DRAWING HAS NOT BEEN PUBLISHED BUT RATHER HAS BEEN PREPARED BY JDS FOR USE BY THE CLIENT NAMED IN THE TITLE BLOCK SOLELY IN RESPECT OF THE CONSTRUCTION, OPERATION AND MAINTENANCE OF THE FACILITY NAMED IN THE TITLE BLOCK AND SHALL NOT BE USED FOR ANY OTHER PURPOSE OR FURNISHED TO ANY OTHER PARTY WITHOUT THE EXPRESS CONSENT OF JDS

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BLUESTONE RESOURCES INC.

Cerro Blanco Overall Map

PROJECT NUMBER	DRAWING NUMBER	REVISION No.

Source: Google Maps & JDS (2017)

18.20 Tailings Management Facility

18.20.1 Introduction

Knight Piésold Ltd. (KP) prepared the design of the tailings management facility for the Project. The design utilizes filtered tailings delivered from the process plant and stored on surface in a DSTF.

18.20.2 DSTF Location

The mine plan includes processing approximately 4.0 Mt of mineralized material and storing 2.9 Mt of tailings on surface in the DSTF. The remaining tailings will be used for paste backfill in the underground mine. Previous studies on tailings management for the Project based on a larger tailings quantity identified two possible locations for the DSTF:

- The hillside south of the process plant area adjacent to the eastern property boundary; and
- A valley to the southwest of the plant site area, off the current property boundary.

The DSTF design completed for this study utilizes site 1. This site is close to the mine infrastructure and is within the property boundary. The DSTF in this location is shown on Figure 18.4.

Table 18.1 summarizes the main design criteria for the DSTF.

Table 18.1: DSTF Design Criteria Summary

Description	Unit	Value
Life of Mine	years	9
Mill Throughput (Tailings Production)	t/d	1,250
Total Tailings Capacity	Mt	2.9
Filtered Tailings Dry Density	t/m^3	1.6
Total Tailings Volume	Mm^3	1.8
Outer Shell Downstream Slope		4H:1V
Outer Shell Width	m	5
Outer Shell Upstream Slope		2H:1V
Compacted Tailings Surface Grade		1% (min)
Tailings Geochemistry		NPAG

Source: KP (2017)

18.20.3 DSTF Design Concept

18.20.3.1 General

KP prepared a conceptual level DSTF design to accommodate the filtered tailings placement scheduled for surface disposal at the site. Filtered tailings will be placed and compacted in thin lifts (0.3 to 0.5 m) using the mine haul fleet and a vibratory compactor. For additional stability, a 5 m wide compacted outer shell zone of earth/rockfill is included. This shell zone will be constructed using suitable materials excavated during foundation preparation activities within the DSTF footprint. A starter confining embankment is required and this will form part of the outer shell zone.

Additional design features for the DSTF include foundation preparation for tailings placement, a foundation drain, and a water management pond at the base of the facility to collect contact water runoff and seepage from the foundation drain. Ditches will be constructed around the perimeter of the DSTF footprint to divert the upper catchment areas and minimize inflows. The DSTF plan and sections are shown on Figures 18.4 and 18.5, respectively.

18.20.3.2 Subsurface Preparation

Detailed geotechnical investigations were previously completed to support the design for a DSTF in the valley southwest of the process plant site and west of the current DSTF site. The investigations included 10 drill holes and 25 test pits, and identified layers of topsoil, overburden, and saprolitic tuff that would be excavated for foundation improvement purposes. It was concluded that up to 5 m of near surface material may need to be removed from structural zones of the DSTF in this valley.

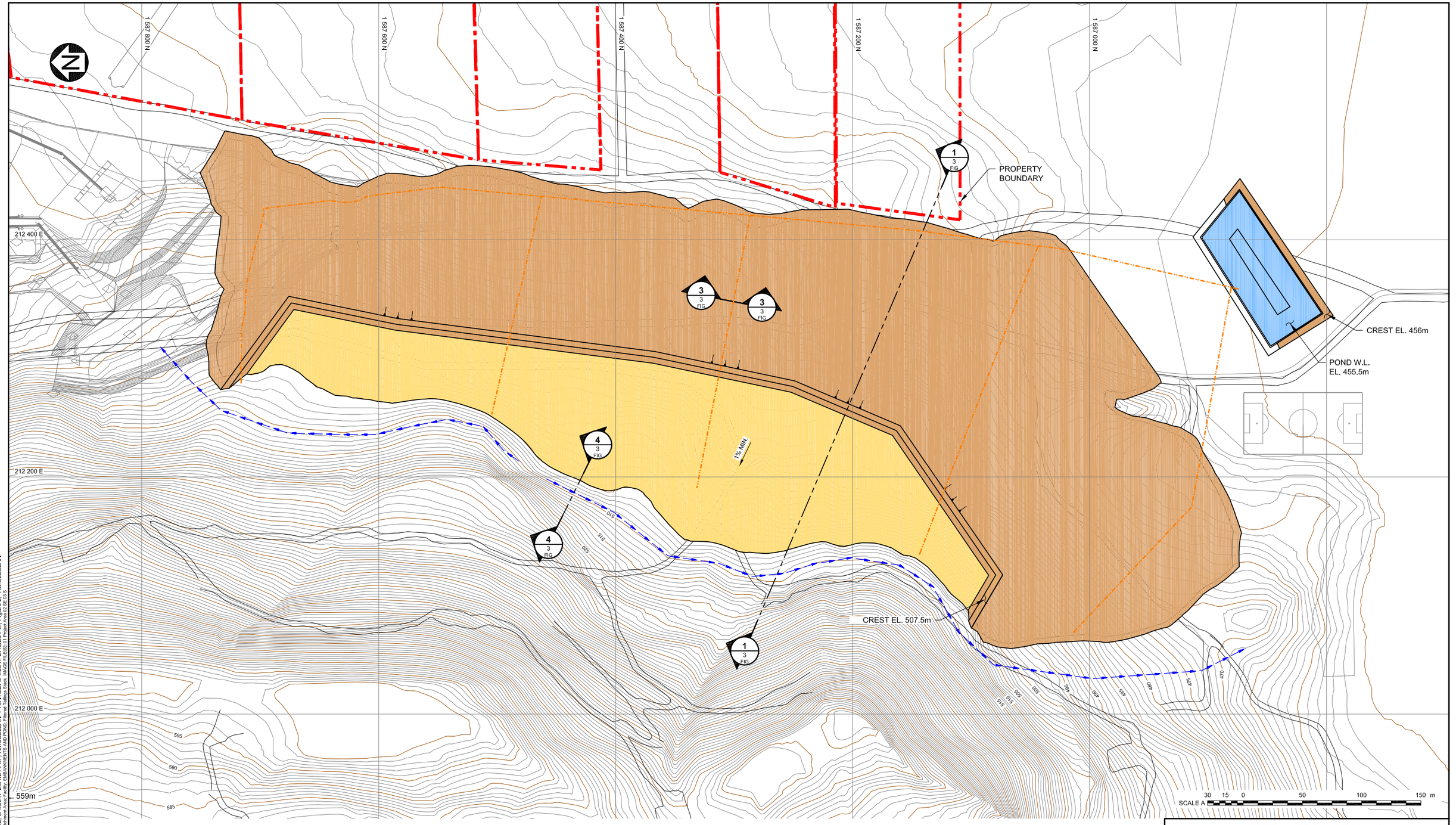
No site investigations have been completed in the current DSTF site and KP has adopted similar assumptions for foundation preparation. However, it has been assumed that there is a thinner sequence of material to be removed based on a recent site visit and the steeper topography in the area.

For this PEA it has been assumed that up to 3 m of near surface material may need to be removed from the full footprint of the DSTF as follows:

- Topsoil: 0.3 m;
- Overburden:1.0 m; and
- Saprolitic tuff: 1.7 m.

The excavated materials will be stockpiled on-site. A potential stockpile area is located in the small valley immediately west of the process plant site. Suitable excavated materials will be used to construct the outer shell zone, as mentioned above.

Figure 18.4: Dry Stack Tailings Facility (DSTF) Plan



LEGEND:

- DIVERSION CHANNEL
- SEEPAGE COLLECTION UNDER DRAIN
- EXISTING ROADS
- PROPERTY BOUNDARY

NOTES:

1. COORDINATE GRID IS UTM NAD 27, ZONE 16.
2. CONTOUR INTERVAL IS 1 METRES.
3. DIMENSIONS AND ELEVATIONS ARE IN METRES, UNLESS NOTED OTHERWISE.

REV	DATE	DESCRIPTION	DESIGNED	DRAWN	REVIEWED
B	01MAR'17	ISSUED WITH LETTER	JEF	RM	KDE

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CERRO BLANCO PROJECT

DSTF PLAN

Knight Piésold CONSULTING

P/A NO. VA201-561/1 | REF NO. VA17-00374

FIGURE 2 | REV B

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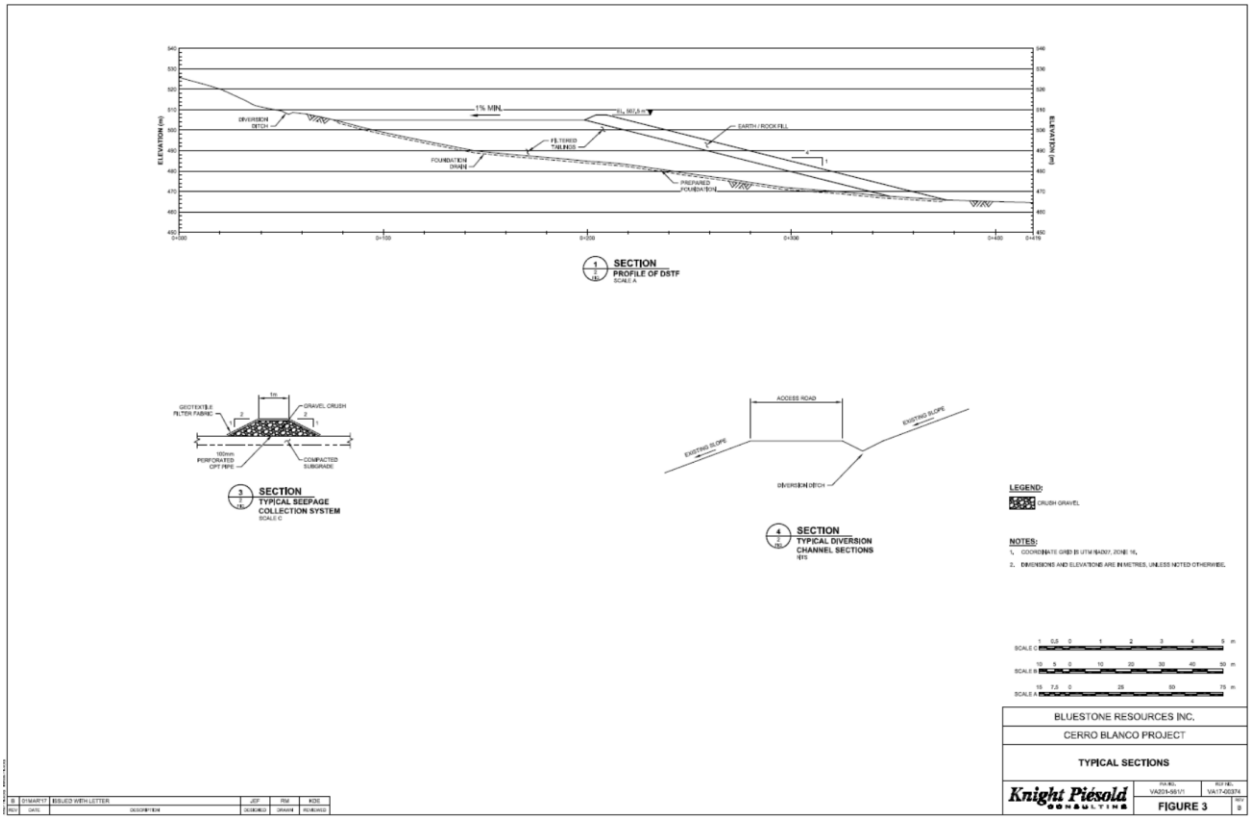
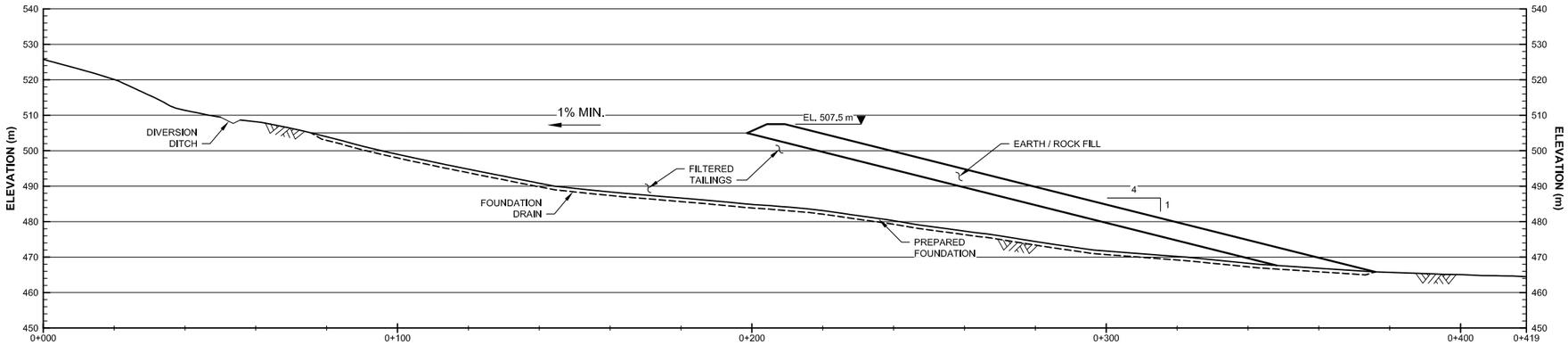
Source: KP (2017)

Figure 18.5: Dry Stack Tailings Facility (DSTF) Typical Sections

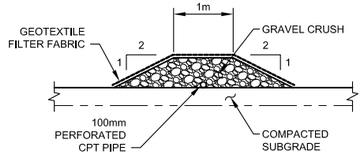


Source: KP (2017)

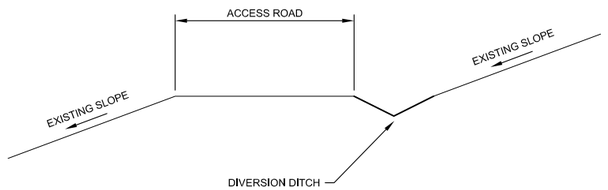
Figure 18.5: Dry Stack Tailings Facility (DSTF) Typical Sections



1 / 2 FIG **SECTION**
PROFILE OF DSTF
SCALE A



3 / 2 FIG **SECTION**
TYPICAL SEEPAGE COLLECTION SYSTEM
SCALE C



4 / 2 FIG **SECTION**
TYPICAL DIVERSION CHANNEL SECTIONS
NTS

LEGEND:

CRUSH GRAVEL

NOTES:

1. COORDINATE GRID IS UTM NAD27, ZONE 16.
2. DIMENSIONS AND ELEVATIONS ARE IN METRES, UNLESS NOTED OTHERWISE.

SCALE C: 1 0,5 0 1 2 3 4 5 m
SCALE B: 10 5 0 10 20 30 40 50 m
SCALE A: 15 7.5 0 25 50 75 m

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CERRO BLANCO PROJECT

TYPICAL SECTIONS

Knight Piésold CONSULTING

P/A NO. VA201-561/1 | REF NO. VA17-00374

FIGURE 3 REV B

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Source: KP (2017)

18.20.4 Seepage Collection System

A seepage collection system will be provided for the DSTF. This system will include an HDPE lined water management pond located downstream of the DSTF. Seepage collection drains will be installed beneath the filtered tailings to direct contact water to the water management pond (when exposed) and to collect seepage or filtration through placed tailings. The water management pond has been designed to store the 100-year, 24-hour rain event (130 mm) falling directly on the DSTF. The layout of the seepage collection drains is shown on Figure 18.4. Typical cross sections for the seepage collection drains are shown on Figure 18.5.

18.20.4.1 Surface Water Management

A water management system will be constructed to minimize water in the DSTF and prevent clean runoff from coming in contact with placed tailings. Diversion ditches will be constructed around the perimeter of the DSTF to direct the flow of non-contact water runoff past the facility. The diversions are expected to encounter bedrock and overburden. In locations with sound bedrock, the ditches will not require erosion protection material. In more erodible materials, riprap may be included to protect the ditches. A typical cross section for the diversion ditches is shown on Figure 18.5.

In the event water starts to accumulate on the DSTF it will drain away from the outer shell due to the sloped tailings surface. This water will be pumped down to the water management pond and released, as appropriate.

18.20.4.2 Closure

The DSTF will be shaped and capped at closure to prevent erosion of the tailings surface, while blending the structure into the surrounding topography.

18.20.4.3 Materials and Quantities

KP developed the materials and quantities for the DSTF, with the following key assumptions:

- The placed density of the compacted tailings was assumed to be 1.6 t/m^3, and
- The 100-year, 24-hour rain event of 130 mm was used for water management design calculations.

18.20.5 Conclusions

KP prepared this conceptual (PEA) level design for the DSTF. The DSTF is located within the property boundary and the design layout provides sufficient storage capacity and meets the water management requirements. The following are recommendations to consider for future studies:

- The DSTF layout may be optimized to store a refined tailings quantity;
- Water management plans can be optimized to incorporate the staged development of the DSTF;
- Site water storage requirements can be reviewed and updated based on mine infrastructure demand;
- Geotechnical, hydrogeological and hydrological data collection and interpretation should be undertaken for the DSTF site to support more detailed design;
- Stability analyses are required for the DSTF, incorporating test results on tailings and foundation materials; and
- A detailed construction plan and schedule should be developed to provide input for construction management and cost estimates.

19 Market Studies and Contracts

19.1 Market Studies

No preliminary market study was completed on the potential sale of doré from the Cerro Blanco Project. The terms used in the economic analysis are considered to be in line with the current market conditions. The indicative terms were reviewed and found to be acceptable by QP Maz Mohaseb, P.Eng.

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: NSR Assumptions

Off-site Costs and Payables	Item	Unit	Estimated Value
Payables for Doré	Gold	%	99.8
	Silver	%	99.0
Doré Refining Costs	Gold	US$/payable oz	5.00
	Silver	US$/payable oz	0.50
Transportation Costs		US$/payable Au oz	3.73

Source: JDS (2017)

19.2 Contracts

No contractual arrangements for shipping, port usage, or refining exist at this time. Furthermore, no contractual arrangements have been made for the doré at this time.

19.3 Royalties

The Cerro Blanco 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 royalty terms. Total royalties for the Project amount to US$ 25 M over the life of mine.

Table 19.2: Royalty Assumptions

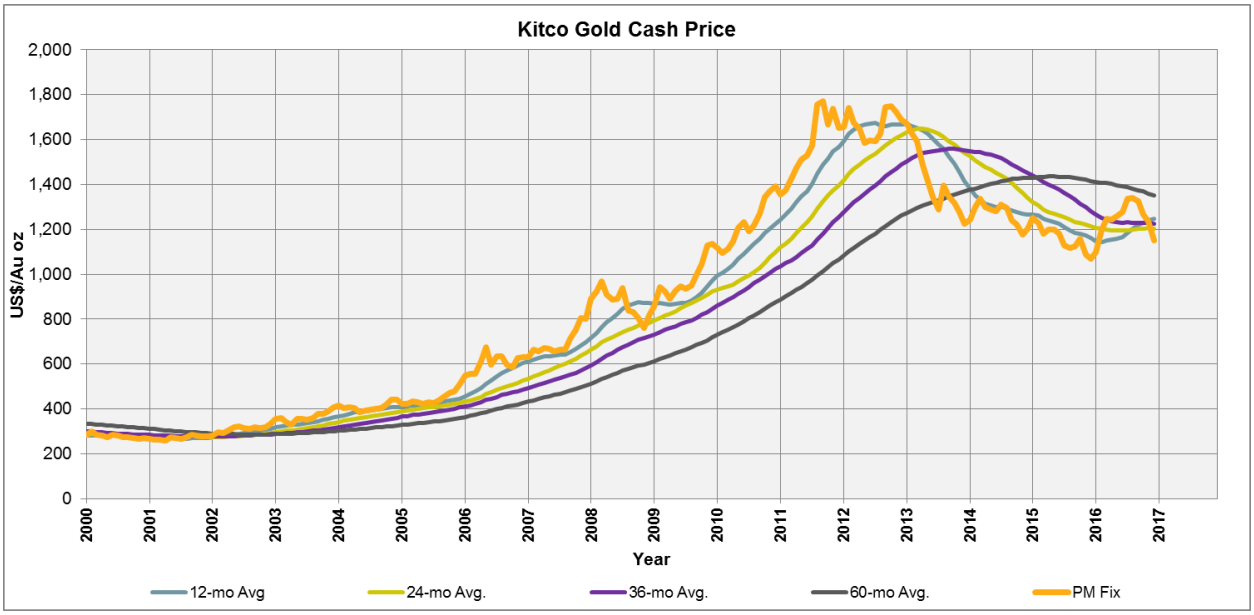
Parameter	Unit	Value
Guatemalan Government Royalty	% NSR	1.0
Goldcorp Royalty	% NSR	1.0

Source: JDS (2017)

19.4 Metal Prices

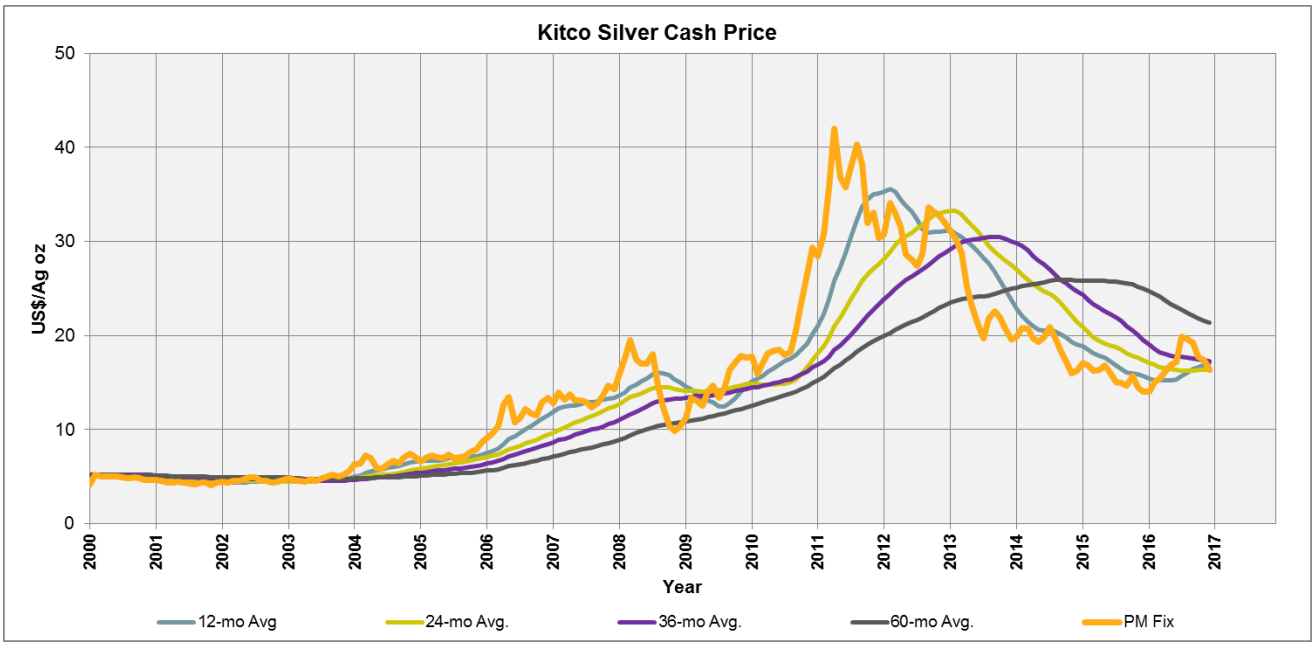
The precious metal markets are highly liquid and benefit from terminal markets around the world (London, New York, Tokyo, and Hong Kong). Historical gold and silver prices are shown in Figure 19.1 and Figure 19.2.

Figure 19.1: Historical Silver Price



Source: Kitco (2017)

Figure 19.2: Historical Silver Price



Source: Kitco (2017)

The gold and silver prices selected were based on the 12- and 18 month average respectively as at December 2016 and are in line with recently released comparable Technical Reports. The US$:C$ FX rate selected was based on the 24-month average as at December 2016 and recently released comparable Technical Reports. 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 exchanges rates used in the economic analysis.

It must be noted that metal prices are highly variable and are driven by complex market forces and are extremely difficult to predict.

Table 19.3: Metal Prices and Exchange Rates

Assumptions	Unit	Value
Au Price	US$/oz	1,250
Ag Price	US$/oz	16.00
FX Rate	GTQ:US$	7.55
	C$:US$	1.28

Source: JDS (2017)

20 Environmental, Permitting and Community

20.1 Permitting

The PEA due diligence review shows that Cerro Blanco has all necessary permits to proceed with the development of the underground mine and construction of the process facilities, subject to the future operation to adhere to the conditions of the exiting permits. The review shows that Cerro Blanco has been operating so far under Goldcorp's high CSR and environmental standards. Cerro Blanco has been maintaining a comprehensive Permits Register which shows that all applicable permit commitments have been fulfilled over time. Cerro Blanco environmental management is based on Goldcorp "Sustainability Excellence Management System".

Cerro Blanco has maintained a comprehensive permit register, which shows the status of each permit/license. Table 20.1 summarizes the permits and their status.

Table 20.1: Current Permits/Licenses and Expiration Dates

Licence/Permit	Resolution	Validity Date
Mining	Resolution No.1942 MEM	2032
Tracking and Surveillance Licence Category A	2613-2007/ECM/LP	Every 3 years with bond
EIA approval	2613-2007/ECM/LP MARN	The duration of the Project life
Discharge Abatement Cerro Blanco Project and Environmental Management Plan	511/DIGARN/ECM/caml	Indefinite
Property Registry	31 October 2007	2032
Cerro Blanco Building Permit Municipality As. Mita	29 December 2007	Indefinite
Forestry License	No. 40-2205-155-1.6-2007	2017
Handling and disposal of sludge	Resolution 00244-2016-DIGARN/FACD/gamc	Expires 2019
Resolution: no pre-Hispanic or paleontological remains in the Project area	Opinion No. 002/mc.2008 Department of Pre-Hispanic and Colonial Monuments.	Undefined
Diesel Tank Operating Licence, Own Consumption	Lic No. 0627	2017
Licence for operation and management of Explosives	1942	Undefined
Other resolutions of environmental documents, from previously acquired commitments	2007	Undefined

Source: JDS (2017)

The proposed access road is an element of the Project that is not covered by any existing permits. An EIA will have to be prepared and submitted to the Authorities for approval of this infrastructure item.

This PEA plan brings positive environmental impacts to the Project. These are summarized below:

- The PEA preliminary mine plan was developed such that all the waste rock can be returned to the underground mine as backfill. Therefore, no potentially acid generating waste rock will be left on surface after closure. The PEA mine plan also eliminated, during the first year of operation, the two existing waste rock piles at the Cerro Blanco site, by returning this material to the underground mine;
- The PEA mine plan was developed so that the underground mine can accommodate up to 50% of the tailings as paste backfill. This design approach translates into minimizing the storage of dry tailings on surface; and
- The existing site water system offers the flexibility and the potential to maximize the reuse of treated water from the WTP during the dry season.

20.2 Environmental Monitoring

Cerro Blanco is currently operating and reporting on a comprehensive environmental monitoring network consisting of 26 monitoring stations for water quality within and outside of the Project boundaries. There are also nine stations for monitoring ambient air quality. Annual reports are prepared and presented to the Authorities to report on the results of monitoring, Health & Safety and CSR.

20.3 Mine Closure

The permit register shows that, under Guatemalan regulation, the closure plan must be submitted three years before the mine ceases operations. Accordingly, no formal closure plan as yet been filed with the Authorities at this point. This study did however estimate a preliminary closure cost estimate based on available information. Closure costs will be further detailed and developed in the next stage of engineering study.

20.4 Social Environment and Relationships with Communities

Cerro Blanco has been reporting on an annual basis regarding environmental monitoring, health & safety, social engagement and sustainability initiatives in the form of an “Informe Annual”. For the purposes of this PEA, the social environment of the Cerro Blanco mine site has been reviewed using information available from the Cerro Blanco database and reports.

The Cerro Blanco mine site is located near the town of Asunción Mita, Jutiapa District, in eastern Guatemala. The population of Asunción Mita is mainly of Spanish origin with no indigenous communities. The town of Asunción Mita offers all basic services such as hotels, restaurants, a hospital, banking, gas stations, a public market, construction services and mechanical shops. The main employment in Asunción Mita is in agriculture and cattle followed by daily labour; industrial employment is minimal.

Entre Mares has established contacts and relationships with the municipal Authorities, with the regional political-administrative organization "*Mancomunidad del Lago de Guija",* and with the provincial government. Entra Mares has been pursuing social engagement, including but not limited to: home visits, community meetings and strategic coordination with government and non-government organizations..

It is reported that "mining" is supported by the municipal Authorities and is accepted by the communities as an important contributor to the sustainable development of the region.

There are no resettlement issues with the Cerro Blanco Project.

21 Capital Cost Estimate

LOM Project capital costs total $283 M, consisting of the following distinct phases:

- Pre-production capital costs – includes all costs to develop the property to a 1,250 t/d production. Initial capital costs total $171 M and are expended over a 18-month pre-production construction and commissioning period;
- Sustaining capital costs – includes all costs related to the acquisition, replacement, or major overhaul of assets during the mine life required to sustain operations. Sustaining capital costs total $105 M and are expended in operating Years 1 through 9; and
- Closure Costs – includes all costs related to the closure, reclamation, and ongoing monitoring of the mine post operations. Closure costs total $7 M, and are primarily incurred in Year 10, with costs extending into Year 15 for ongoing monitoring activities.

The capital cost estimate was compiled using a combination of quotations, database costs, and database factors. Once compiled the overall cost estimate was top-down benchmarked against similar operations.

Table 21.1 presents the capital estimate summary for initial and sustaining capital costs in Q1 2017 dollars with no escalation.

21.1 Capital Cost Summary

Table 21.1: Capital Cost Summary

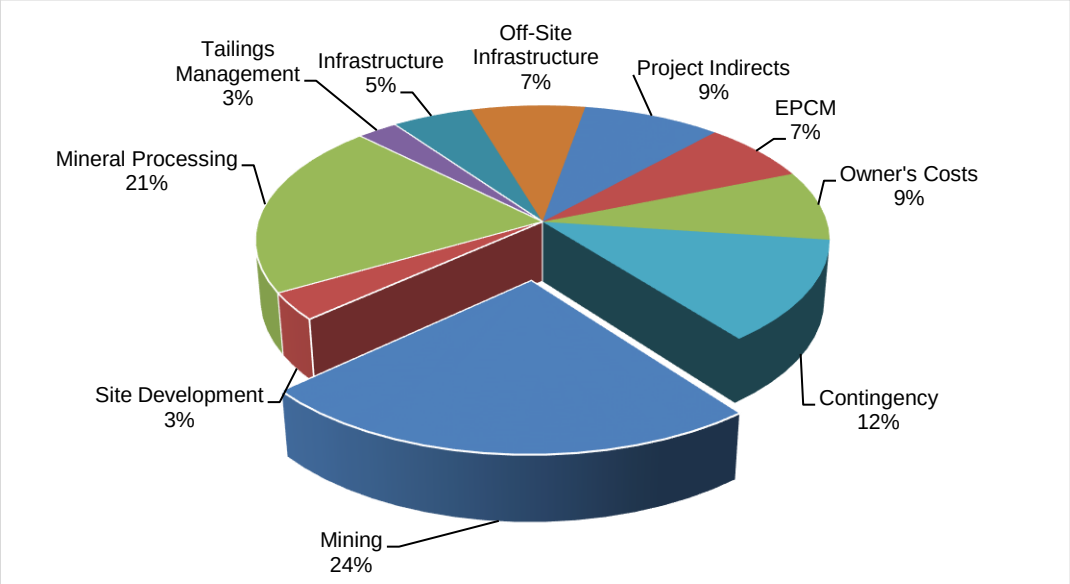
Capital Costs	Pre-Production (M$)	Sustaining/ Closure (M$)	Total (M$)
Mining	40.8	80.5	121.3
Site Development	5.6	0	5.6
Primary Crushing	4.2	0	4.2
Process Plant	31.5	0	31.5
Tailings Management	4.5	4.6	9.2
On-Site Infrastructure	8.9	0.1	9.0
Off-Site Infrastructure	12.5	9.3	21.8
Project Indirects	15.2	0	15.2
EPCM	12.2	0	12.2
Owner's Costs	14.3	0	14.3
Subtotal	**149.8**	**94.5**	**244.2**
Contingency	21.0	10.8	31.8
Closure	0	6.9	6.9
Total Capital Costs	**170.8**	**112.2**	**283.0**

*totals may not add due to rounding

Source: JDS (2017)

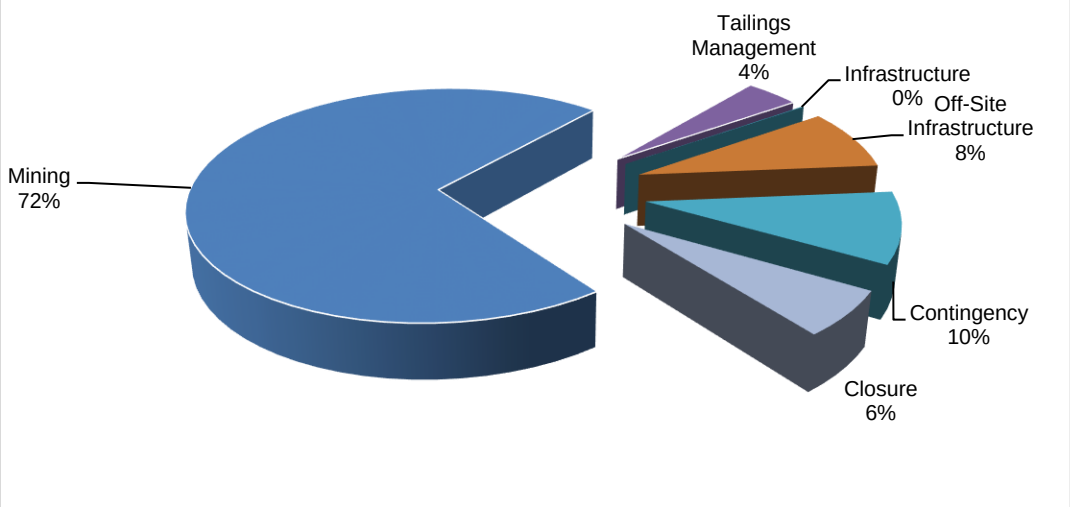
Figure 21.1 and Figure 21.2 present the capital cost distribution for the pre-production and sustaining phases. As typical with underground operations, the majority of sustaining capital costs relate to underground lateral and vertical development. In addition, due to the geothermal nature of the Project, the sustaining capital costs include a significant amount of reinjection well drilling and dewatering wells.

Figure 21.1: Distribution of Initial Capital Costs



Source: JDS (2017)

Figure 21.2: Distribution of Sustaining Capital Costs

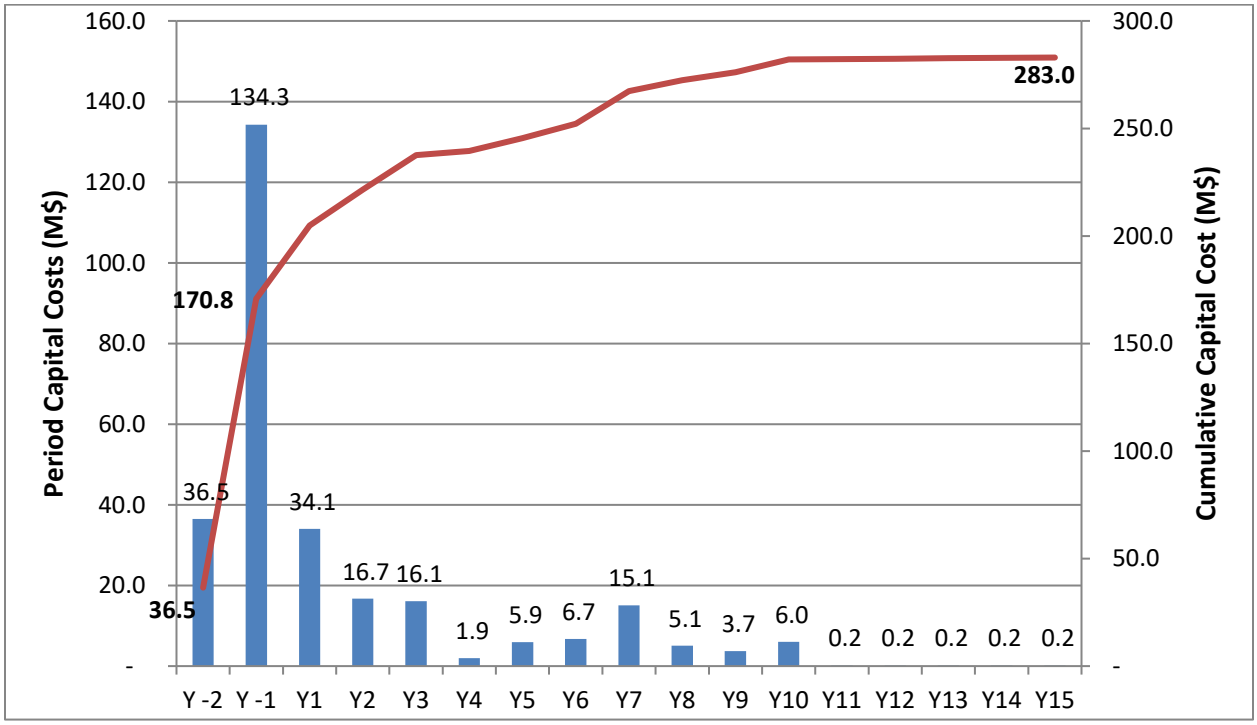


Source: JDS (2017)

21.2 Capital Cost Profile

All capital costs for the Project have been distributed against the development schedule in order to support the economic cash flow model. Figure 21.3 presents an annual life of mine capital cost profile including closure years (Year 10-15).

Figure 21.3: Capital Cost Profile



Source: JDS (2017)

21.3 Key Estimate Assumptions

The following key assumptions were made during development of the capital estimate:

- Underground mine development activities will be performed by the Owners forces; and
- All surface construction (including earthworks) will be performed by contractors.

21.4 Key Estimate Parameters

The following key parameters apply to the capital estimates:

- Estimate Class: The capital cost estimates are considered Class 4 estimates (-20%/+30%). The overall Project definition is estimated to be 10%;
- Estimate Base Date: The base date of the estimate is January 2017. No escalation has been applied to the capital cost estimate for costs occurring in the future;
- Units of Measure: The International System of Units (SI) is used throughout the capital estimate; and
- Currency: All capital costs are expressed in US Dollars (US$). Portions of the estimate were estimated in Canadian Dollars (C$) and converted to US Dollars at a rate of CA$1.00: US$0.78.

21.5 Basis of Estimate

21.5.1 Mine Capital Costs

Capital cost estimates are based on a combination of budgetary quotes from equipment suppliers, in-house cost databases and similar mines in Guatemala. Table 21.2 summarizes the underground mine capital cost estimate.

Table 21.2: Mine Capital Costs

Capital Costs	Pre-Production (M$)	Sustaining/ Closure (M$)	Total (M$)
Underground Mobile Equipment	12.0	17.3	29.3
Underground Infrastructure	13.5	17.5	31.0
Capital Development	9.6	45.8	55.3
Capitalized Production Costs	5.8	0	5.8
Total Mining (excl. Contingency)	**40.8**	**80.5**	**121.3**

Source: JDS (2017)

21.5.1.1 Underground Mobile Equipment

Underground mining equipment quantities and costs were determined through buildup of mine plan quantities and associated equipment utilization requirements. Budgetary quotations or database costs have been carried and applied to the required quantities.

21.5.1.2 Underground Infrastructure

Design requirements for underground infrastructure were determined from design calculations for ventilation, dewatering, and material handling.

Budgetary quotations or database costs were used for major infrastructure components. Allowances have been made for miscellaneous items, such as initial PPE, radios, water supply, refuge stations, and geotechnical investigations. Acquisition of underground infrastructure is timed to support the mine plan requirements.

21.5.1.3 Capital Development

Capital development includes the labour, fuel, equipment usage, power, and consumables costs for lateral and vertical development required for underground access to stopes, and underground infrastructure.

21.5.1.4 Capitalized Production Costs

Capitalized production costs are defined as mine operating expenses (operating development, stoping, mine maintenance, and mine general costs) incurred prior to the introduction of feed to the processing facilities and the commencement of Project revenues. They are included as a pre-production capital cost.

The basis of these costs is described in Section 22, operating costs, as they are estimated in the same manner. Capitalized production costs are included in the asset value of the mine development and are depreciated over the mine life within the financial model.

21.5.2 Surface Construction Costs

Surface construction costs include site development, crushing plant, mineral processing plant, tailings management facility, on-site and off-site infrastructure. These cost estimates are primarily based on database costs, with factors applied for minor cost elements. Table 21.3 presents a summary basis of estimate for the various commodity types within the surface construction estimates.

Table 21.3: Surface Construction Basis of Estimate

Commodity	Basis
Access Roads	Database costs per kilometre for road upgrades or construction, based on assessed difficulty of construction for each section Database costs for bridges
Bulk Earthworks	Model volumes from preliminary 3D grading model Database unit rates for bulk excavation and fill Bulk allowances for surface drainage and temporary roads
Concrete	Quantities developed based on building sizes outlined in general arrangements and cross checked against similar projects in high seismic areas Budgetary quotes for unit rates in Guatemala from local contractors in the region
Structural Steel	Quantities developed based on building sizes outlined in general arrangements and cross checked against similar projects in high seismic areas Budgetary quotes for unit rates in Guatemala from local contractors in the region
Pre-Engineered Buildings	Database unit rates ($/m^2) applied against the building sizes outlined in the general arrangements Database allowances for lighting, small power, electrical/control rooms, and fire detection

Commodity	Basis
Modular Buildings & Warehouses	Database costs from similar projects in the area for the process building and main ancillary buildings such as the mine dry, administration offices, mine maintenance building, and mine warehouse
Mechanical Equipment	A combination of quoted and database costs from similar projects A combination of actual install hours based on equipment size and database factors applied against mechanical equipment costs for installation
Piping	Database factors applied against mechanical equipment costs
Electrical and Instrumentation	Database factors applied against mechanical equipment costs
Power Transmission Line and Major Sub-stations	A combination of quoted and database costs from similar projects who engaged an experienced line contractor Quantities developed based on general arrangements and site layouts

Source: JDS (2017)

21.5.2.1 Surface Construction Sustaining Capital

Sustaining capital costs are included in the estimate for continued construction of the Tailings Management Facility. The balance of the facility is evenly constructed from Year 1 to 9 of operations. In addition, the sustaining capital costs include six new dewatering wells and five reinjection wells during Year 1 and two new dewatering wells during Year 2.

21.6 Indirect Cost Estimate

Indirect costs are those that not directly accountable to a specific cost object. Table 21.4 presents the subjects and basis for the indirect costs within the capital estimate.

Table 21.4: Indirect Cost Basis of Estimate

Commodity	Basis
Heavy Equipment	Factor (1.5%) of direct costs for heavy equipment rental (i.e. 100 t + crane)
Contractor Field Indirects	Factor (8.5%) for the following items:
	Time based cost allowance for general construction site services (temporary power, heating & hoarding, contractor support, etc.) applied against the surface construction schedule
	Construction offices and ablution facilities
	Combination of diesel and transmission line construction power
	Contractor mobilization
Freight & Logistics	Factor (10%) for freight and logistics related to the materials and equipment required for the crushing plant, mineral processing plant, on-site and off-site infrastructure. Factor excludes mining equipment as prices are FOB site
Vendor Representatives	Factor (2%) of direct costs for the provision of vendor services for commissioning equipment
Capital Spares	Factor (5%) of direct costs for spare parts
Start-up and Commissioning	Factored (3%) of direct costs for start-up and commissioning support
First Fills	Factor (2%) of direct costs for first fills (oils, lubes, mill liners, etc.)
Detailed Engineering & Procurement	Factor (6.5%) applied against direct and indirect hours for engineering management, detailed design, drawings, and major equipment procurement
Project & Construction Management	Staffing plan built up against the development schedule for Project management, health and safety, construction management, field engineering, Project controls, and contract administration
	Database unit (hourly) rates

Source: JDS (2017)

21.7 Owners Cost Estimate

Owner's costs are items that are included within the operating costs during production. These items are included in the initial capital costs during the construction phase and capitalized. The cost elements described below are described in more detail within Section 22.

- Pre-production General & Administration: Costs of the Owner's labour and expenses (safety, finance, security, purchasing, management, etc.) incurred prior to commercial production.
- Pre-production Milling: Costs of the Owner's processing labour, power, and consumables incurred before declaration of commercial production; and
- Surface Support: Costs of the Owner's surface support labour, maintenance, and equipment usage; costs for contract water supply and waste removal prior to commercial production.

In addition, an allowance of 3% of direct plus indirect costs was carried for miscellaneous Owner's Costs.

21.8 Closure Cost Estimate

Closure costs have been estimated based on the typical closure, reclamation, and monitoring activities for an underground mine in Guatemala. Activities include:

- Removal of all surface infrastructure and buildings;
- Closure and capping of the tailings management facility;
- Closure of the underground mine portals;
- Access road closure;
- Power transmission line and substation removal;
- Re-vegetation and seeding; and
- Ongoing site monitoring.

The majority of closure costs are incurred immediately following completion of operations (Year 10). Monitoring activities are anticipated to extend to Year 15.

Table 21.5: Closure Estimate Summary

Item	Estimated Cost (M$)
Closure Costs	4.9
Monitoring	1.0
Subtotal	**5.9**
Contingency	1.0
Total Closure	**6.9**

Source: JDS (2017)

21.9 Contingency

A blended contingency of 13% was applied to the LOM capital costs of the Project. LOM Project contingency amounts to $32 M.

21.10Capital Estimate Exclusions

The following items have been excluded from the capital cost estimate:

- Working capital (included in the financial model);
- 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, etc.);
- Provincial sales tax;
- Closure bonding; and
- Escalation cost.

22 Operating Cost Estimate

The OPEX estimate is based on a combination of experience, reference Project, budgetary quotes and factors as appropriate with a preliminary study.

The operating cost estimate in this study includes the costs to mine, process the mineralized material to produce doré, and general and administrative expenses (G&A). These items total the Project operating costs and are summarized in Table 22.1. The target accuracy of the operating cost is -25/+30%.

The operating cost estimate is broken into three major sections:

- Underground mining;
- Processing; and
- General & administrative.

The total operating unit cost is estimated to be US$92.58/t processed. Average annual, total LOM and unit operating cost estimates are summarized in Table 22.1. Figure 22.1 illustrates the operating cost distribution.

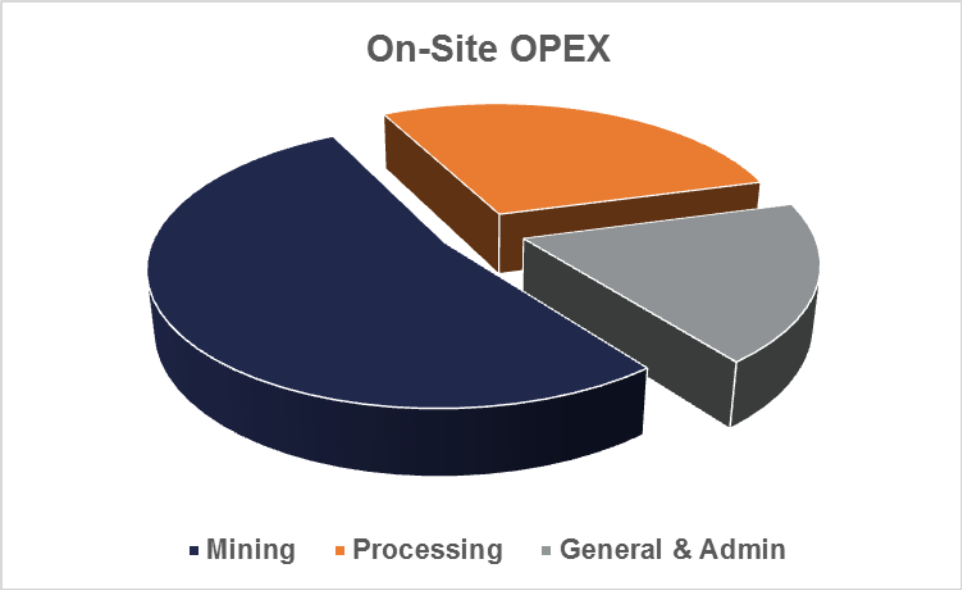
Operating costs are expressed in US dollars. No allowance for inflation has been applied.

Table 22.1: Breakdown of Estimated Operating Costs

Operating Costs	Avg Annual (M$)	$/t processed	LOM (M$)
Mining	22	49.06	195
Processing	12	26.26	105
G&A	8	17.26	69
Total	**41**	**92.58**	**369**

Totals may not add due to rounding
Source: JDS (2017)

Figure 22.1: Operating Cost Distribution



Source: JDS (2017)

The main operating cost component assumptions are shown in Table 22.2.

Table 22.2: Main OPEX Component Assumptions

Item	Unit	Value
Electrical Power Cost	$/kWh	0.10
Overall Power Consumption (all facilities)	kWh/t processed	215
Diesel Cost (delivered)	$/litre	2.50
LOM Average Manpower	employees	300

Source: JDS (2017)

22.1 Operations Labour

This section provides an overview of total workforce and the methods used to compile the labour rates.

Table 22.3 summarizes the total planned workforce during Project operations.

Table 22.3: Summary of Personnel

Department	Total Persons Employed (Peak)
Mining	114
Processing	90
G&A	70
Project Team	2
Construction	40
Total Personnel - All Areas	**316**

Source: JDS (2017)

Labour is a significant portion of annual operating cost. Labour rates include base wage and allowances for overtime, night shift, insurance, tax, and benefits.

Labour burdens were assembled using first principles and range from 34 to 42%. The following items are included in the burdened labour rates:

- Unscheduled Overtime at 10%;
- Social security at 12.67%;
- Yearly Christmas bonus at 1 month salary per year;
- Yearly Bonus 14 at 1 month salary per year;
- Vacation pay at 4%;
- Statutory Holiday pay at 3%;
- Yearly Insurance Payments of $54.00/employee/month; and
- Monthly savings funds at 13%.

22.2 Mine Operating Cost Estimate

Mining operating costs listed in Table 22.4 are averaged over the life of mine. During the mine life, operating costs on a per tonne basis range from a low of $41.31 to a high of $61.61 in Years 3 and 4, respectively. The increase of operating cost increase is due to additional development in Year 4.

Mine operating costs have been built up from using a combination of first principle engineering and equivalent Project scaling.

Table 22.4: Mining Operating Cost Estimate

Parameter	Unit	Value
Lateral Waste Development	US$/t processed	6.79
Production	US$/t processed	19.74
Backfill	US$/t processed	16.64
Mine Maintenance	US$/t processed	1.97
Mine General	US$/t processed	3.92
Total	**US$/t processed**	**49.06**

Source: JDS (2017)

22.3 Processing Operating Cost Estimate

Process operating costs were developed using labour rates based on operating mines in the area and sufficient personnel to operate the process plant, factored maintenance cost, budget quotes for consumables and a factored power requirement. Process operating costs are summarized below in Table 22.5. Costs are subdivided into operating categories.

Table 22.5: Processing Operating Cost by Category

Category	$M/Yr	$/t processed
Labour	2	4.72
Power & Fuel	3	6.48
Equipment Maintenance & Consumables (Reagents, Media, Liners and other Wear Parts)	7	14.96
Grand Total by Activity	**12**	**26.26**

Source: JDS (2017)

Process labour includes burden for salaried and hourly employees to account for in-country benefits, training, production bonus and potential ex-patriot benefits & costs.

Equipment maintenance was calculated by applying a factor of 4% to major process equipment cost. Costs for media were determined using engineering calculations based on mill power draw, abrasion index and vendor quotes for media as a cost per tonne. Reagent requirements from recent test work and budget quotes from vendors and mines operating in the area were used to calculate the cost of reagents. Mill liners and wear parts for major equipment were based on vendor recommended requirements and quotes.

Power costs were calculated from the total installed power assuming $0.10/kWh.

22.4 General and Administration Operating Cost Estimate

General and Administration costs include all on-site activities including but not limited to dry stack tailings (DST) haulage, personal protection services, water treatment plant operation, site services equipment and labour, office operating costs and associated labour. The summary of costs is shown in Table 22.6, averaged over the life of mine.

Table 22.6: G&A Personnel Requirement

Parameter	US$/t processed	Average US$M/a	LOM (US$M)
G&A Labour	3.75	2	15
G&A Items - On-site	13.52	6	54
Total Operating Cost – G&A	**17.26**	**8**	**69**

Source: JDS (2017)

22.4.1 G&A Labour Requirements

Table 22.7 lists the site supervision and support personnel requirements and costs.

Table 22.7: G&A Labour Requirements & Costs

G&A Labour	Salary/Hourly	Loaded Pay (US$)	Quantity
Mine/General Manager	Salary	243,459	1
Site Administrator	Salary	12,956	1
Accounting & Taxes Manager	Salary	20,340	1
Accounting/Payroll Coordinator	Salary	15,417	2
Human Resources Manager	Salary	22,802	1
Human Resources Clerk	Salary	12,956	1
Trainer	Salary	15,417	2
Community Relations Manager	Salary	74,493	1
Community Relations Coordinator	Salary	15,417	1
IT/Telecom. Technician	Salary	22,802	1
Procurement/Contracts/Logistics Manager	Salary	25,263	1
Procurement/Contracts Agent	Salary	20,340	1
Warehouse Operators	Salary	15,417	1
Multi-Equipment Operators	Salary	15,417	2
Health, Safety, and Security Manager	Salary	49,878	1
Health & Safety Coordinator	Salary	22,802	2
First Aid Attendant/Nurse	Salary	7,591	4
Environmental Manager	Salary	49,878	1
Environmental Technician	Salary	15,417	1
Environmental Coordinator	Salary	25,263	2
Protective Services Officials	Salary	12,956	30
Site Services Foreman	Salary	37,571	2
Carpenters	Salary	15,417	1
WTP Operator	Salary	15,417	4
Multi-Equipment Operator	Salary	15,417	4
Skilled Labourers	Salary	11,725	2
Total G&A Labour			**71**

Source: JDS (2017)

23 Economic Analysis

An engineering economic model was developed to estimate annual cash flows and sensitivities of the Project. 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.

Univariate 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 as part of this study. 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 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 materially 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 and Section 22 of this report (presented in 2017 dollars). The economic analysis has been run with no inflation (constant dollar basis).

It must be noted that this PEA is preliminary in nature and includes the use of inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves, and there is no certainty that the results of the preliminary economic assessment will be realized. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

23.1 LOM Summary & Assumptions

The summary of the mine plan and payable metals produced is outlined in Table 23.1.

Table 23.1: LOM Summary

Parameter	Unit	Value
Mine Life	Years	9
Resource Mined	kt	3,981
Au Grade	g/t	8.14
Ag Grade	g/t	27.95
Processing Rate	kt/d	1.2
Au Payable	koz	950
	koz/a	105
Ag Payable	koz	3,109
	koz/a	344

Other economic factors include the following:

- Discount rate of 5% (sensitivities using other discount rates have been calculated);
- Closure cost of $7 M;
- Nominal 2017 dollars;
- Revenues, costs, taxes are calculated for each period in which they occur rather than actual outgoing/incoming payment;
- Working capital calculated as one month of operating costs (mining, processing, infrastructure and G&A) in Year 1;
- Results are presented on 100% ownership;
- No management fees or financing costs (equity fund-raising was assumed); and
- 3% of the mine plan tonnes are classified as Inferred Resources.

The model excludes all pre-development and sunk costs up to the start of detailed engineering (i.e. exploration and resource definition costs, engineering fieldwork and studies costs, environmental baseline studies costs, financing costs, etc.).

Table 23.2 outlines the metal prices and exchange rate assumptions used in the economic analysis. The gold and silver prices selected were based on the 12-month average as at December 2016 and are 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 and there is absolutely 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 23.2: Metal Prices and Exchange Rates

Parameter	Unit	Value
Au Price	US$/oz	1,250
Ag Price	US$/oz	16.00
FX Rate	GTQ:US$	7.50
	C$:US$	1.28

Source: JDS (2017)

23.2 NSR Parameters

Mine revenue is derived from the sale of doré bars into the international marketplace. No contractual arrangements for refining exist at this time.

Table 23.3 indicates the NSR parameters that were used in the economic analysis.

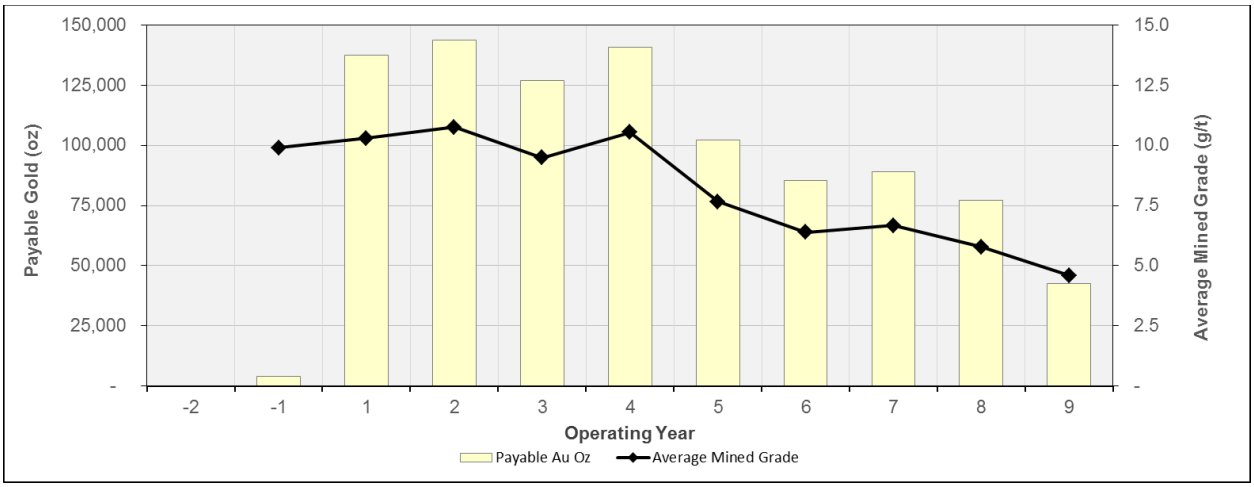
Table 23.3: NSR Parameters

Parameter	Unit	Value
Mine Operating Days	days/a	365
Au Recovery	%	91.3
Ag Recovery	%	87.8
Au Payable	%	99.8
Ag Payable	%	99.0
Au Refining Charge	US$/pay oz	5.00
Ag Refining Charge	US$/pay oz	0.50

Source: JDS (2017)

Figure 23.1 shows the grade and the amount of gold recovered during the mine life. A total of 950 koz of gold, and 3,109 koz of silver are projected to be produced over the life of mine. Gross Project revenues are divided as follows: 96% gold, 4% silver.

Figure 23.1: LOM Payable Gold



Source: JDS (2017)

23.3 Taxes

The Project has been evaluated on an after-tax basis in order to provide a more indicative, but still approximate, value of the potential Project economics. A tax model was prepared by JDS and reviewed by an industry contact with applicable Guatemalan mineral tax regime experience. Current tax pools were used in the analysis. The tax model contains the following assumptions:

- Income Tax: the greater of 25% on net income or 7% on gross income;
- There are no state taxes;
- VAT modelled with a net zero impact due to expected VAT credits and status as exporter;
- Withholding Taxes: 15% on royalties and professional fees, 5% on freight;
- Property Taxes: 0.9% annually on land value;
- Stamp Tax (captured within contingency);
- Capital cost allowance applied on units of production basis to approximate blended CCA rate (~11%); and
- Tax loss carry forward of $135 M from pre-2013 exploration losses applied on a unit of production basis. Losses after 2013 are not eligible.

Total taxes for the Project amount to US$111 M.

23.4 Royalties

The Cerro Blanco Project is subject to two royalties – both of which have been included in the economic analysis and cash flow model. Table 23.3 outlines the royalty terms. Total royalties for the Project amount to US$ 25 M over the life of mine.

Table 23.4: Royalty Assumptions

Parameter	Unit	Value
Guatemalan Government Royalty	% NSR	1.0
Goldcorp Royalty	% NSR	1.0

23.5 Results

At this preliminary stage, the Project is economically viable with a pre-tax IRR of 53% and a net present value using a 5% discount rate ($NPV_{5\%}$) of $405 M using the metal prices described in Section 23.1.

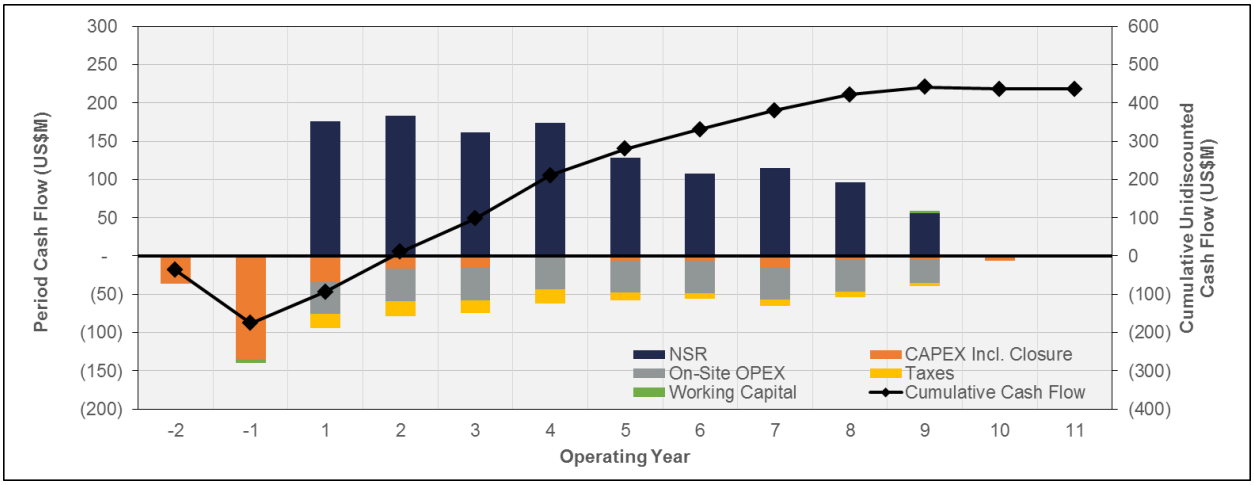
Figure 23.2 shows the projected cash flows, and Table 23.4 summarizes the economic results of the Cerro Blanco Project.

The pre-tax break-even gold price for the Project is approximately US$690/oz, based on the LOM plan presented herein, and a silver price of US$16.00/oz. This is the gold price at which the Project NPV @ 5% discount rate is zero.

The life of mine all-in sustaining cost (AISC) and AISC (net of by-product) is US$542/oz, and US$490/oz respectively. The straight AISC cost is calculated by adding the refining, transport, royalty,

operating, and sustaining and closure costs together and dividing by the total payable ounces of gold. This metric does not consider the value of silver in the calculation. 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 23.2: Annual After-Tax Cash Flow



Source: JDS (2017)

Table 23.5: Summary of Results

Summary of Results	Unit	Value
Cash Cost (Net of By-product)*	US$/oz	477
AISC Cash Cost (Net of By-product incl. Sustaining and Closure CAPEX)**	US$/oz	490
Capital Costs		
Pre-Production Capital	M$	150
Pre-Production Contingency	M$	21
Total Pre-Production Capital	**M$**	**171**
Sustaining & Closure Capital	M$	101
Sustaining & Closure Contingency	M$	11
Total Sustaining & Closure Capital	M$	112
Total Capital Costs Incl. Contingency	**M$**	**283**
Working Capital	M$	4
Pre-Tax Cash Flow	LOM M$	551
	M$/a	61
Taxes	LOM M$	111
After-Tax Cash Flow	LOM M$	440
	M$/a	49
Economic Results		
Pre-Tax $NPV_{5\%}$	**M$**	**405**
Pre-Tax IRR	**%**	**53**
Pre-Tax Payback	**Years**	**1.6**
After-Tax $NPV_{5\%}$	**M$**	**317**
After-Tax IRR	**%**	**44**
After-Tax Payback	**Years**	**1.8**

Source: JDS (2017)

*Cash Cost (Net of By-product) is calculated as: (Refining & shipping costs + royalties+ operating costs - payable Ag value)/payable gold ounces.

**AISC Cash Cost (Net of By-product incl. Sustaining and Closure CAPEX) is calculated as: (Refining & shipping costs + royalties+ operating costs + sustaining capital + closure capital - payable Ag value)/payable gold ounces.

23.6 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 LOM. 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 noted limitations to the sensitivity analysis, which are common to studies of this sort, the analysis revealed that the Project is most sensitive to metal prices, head grade and recovery costs. The Project showed the least sensitivity to capital and operating costs. Table 23.6 and Figure 23.3 show the results of the sensitivity tests, while Table 23.7 shows the NPV at various discount rates.

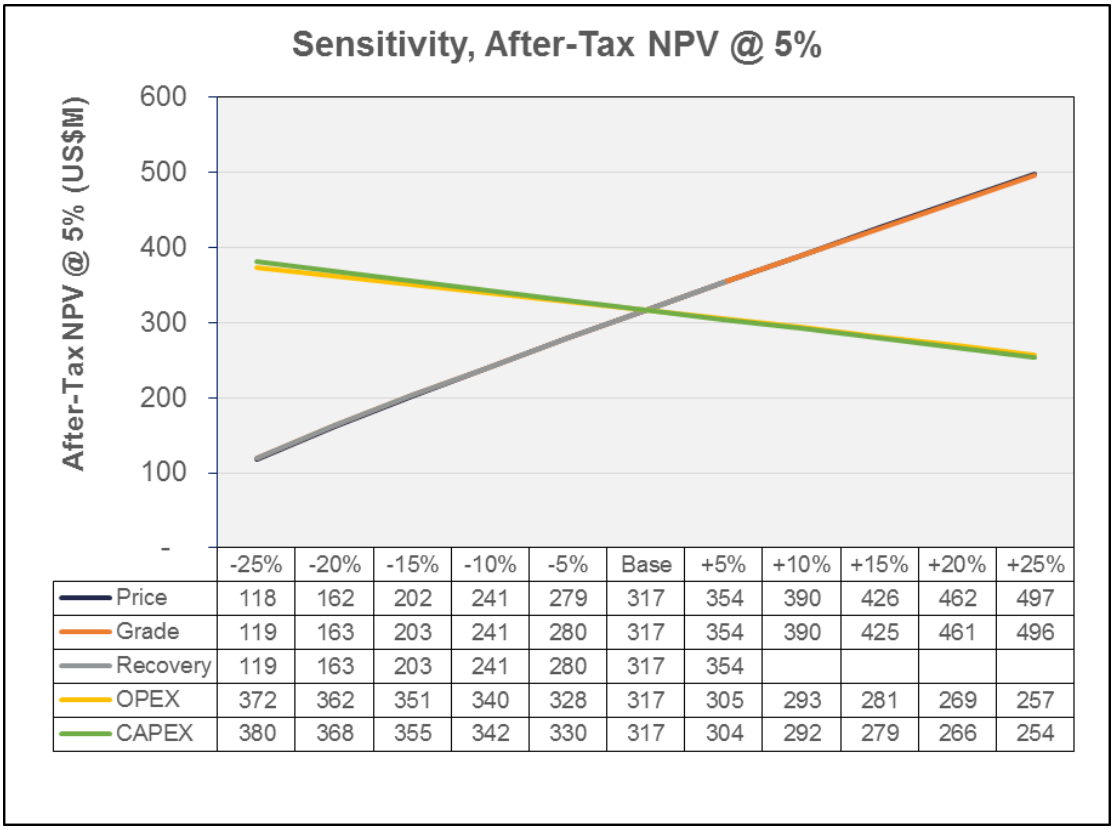
Table 23.7 displays the key Project economic indicators and their sensitivity to gold price.

Table 23.6: Pre-Tax and After-Tax Sensitivity Results on NPV @ 5%

Variable	Pre-Tax $NPV_{5\%}$ (M$)			After-Tax $NPV_{5\%}$ (M$)		
	-25% Variance	0% Variance	25% Variance	-25% Variance	0% Variance	25% Variance
Metal Price/Head Grade	168	405	641	118	317	497
Recovery	170	405	N/A	119	317	N/A
OPEX	475	405	335	372	317	257
CAPEX	468	405	341	380	317	254

Source: JDS (2017)

Figure 23.3: After-Tax $NPV_{5\%}$ Sensitivity



	-25%	-20%	-15%	-10%	-5%	Base	+5%	+10%	+15%	+20%	+25%
Price	118	162	202	241	279	317	354	390	426	462	497
Grade	119	163	203	241	280	317	354	390	425	461	496
Recovery	119	163	203	241	280	317	354				
OPEX	372	362	351	340	328	317	305	293	281	269	257
CAPEX	380	368	355	342	330	317	304	292	279	266	254

Source: JDS (2017)

Table 23.7: Project Net Present Value at Various Discount Rates

Discount Rate (%)	Pre-Tax NPV (M$)	After-Tax NPV (M$)
0	551	440
5	405	317
6	381	297
7	359	278
8	338	261
10	300	229
12	266	200

Source: JDS (2017)

Table 23.8: Cerro Blanco Economics at Various Gold Prices

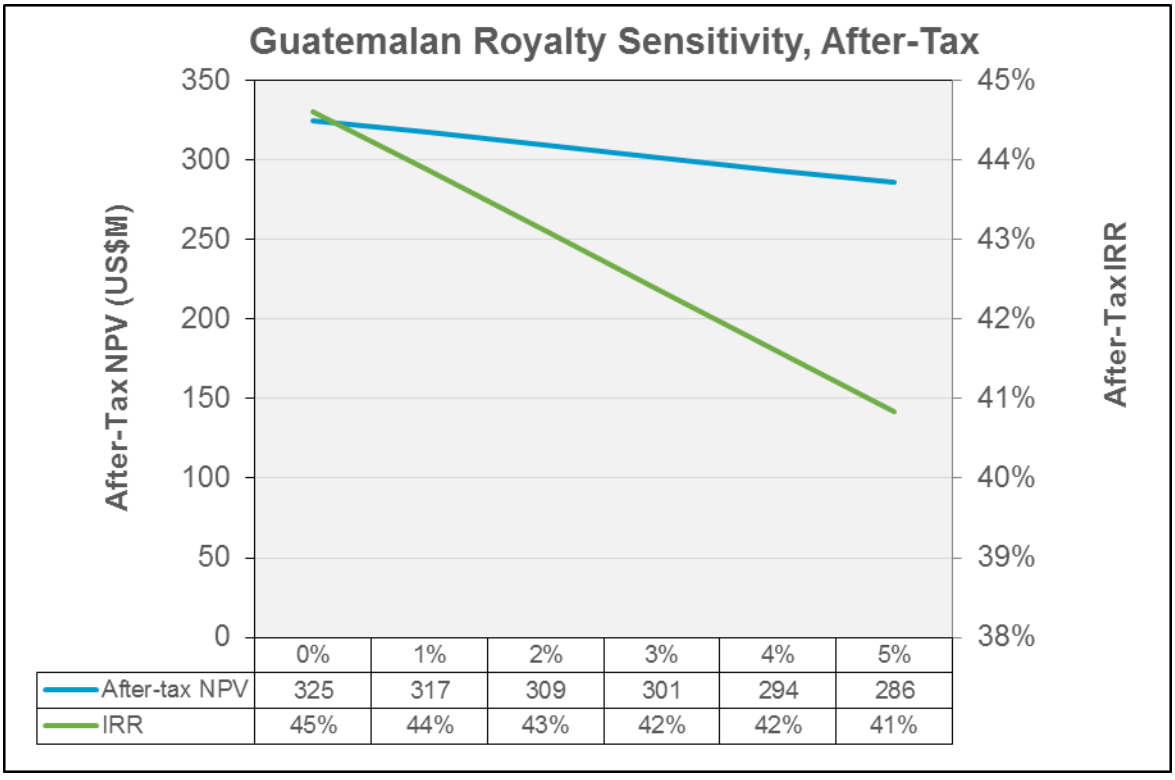
Gold Price	$/oz	1,000	1,100	1,200	1,250	1,300	1,400	1,500
Pre-Tax NPV 5%	$M	216	291	367	405	443	518	594
After-Tax NPV 5%	$M	162	226	287	317	346	404	462
Pre-Tax IRR	%	34	42	50	53	57	64	71
After-Tax IRR	%	28	35	41	44	47	52	58
Pre-Tax Payback	years	2.3	1.9	1.7	1.6	1.5	1.3	1.2
After-Tax Payback	years	2.6	2.2	2	1.8	1.8	1.6	1.5

Source: JDS (2017)

Note: This analysis was completed at a constant silver price of US$16.00/oz.

A separate analysis on royalty sensitivity was completed to assess the Project's exposure to changes in governmental royalty percentages. Figure 23.4 shows the Affter-tax NPV @ 5% and IRR sensitivity to the royalty between the range of 0% to 5%. The economic cash flow model for the Project is illustrated in Figure 23.5.

Figure 23.4: Guatemalan Government Royalty Sensitivity



Source: JDS (2017)

Figure 23.5: Cash Flow Model

Item	Source	Unit	Pre-Production Total	Production Total	Life of Mine Total	Y -2	Y -1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13	Y14	Y15	Y16	Y17
KEY PARAMETERS																								
Metal Prices																								
Gold (Au)	link (K0)	US$/oz	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250
Silver (Ag)	link (K0)	US$/oz	16.00	16.00	16.00	16.00	16.00	16.00	16.00	16.00	16.00	16.00	16.00	16.00	16.00	16.00	16.00	16.00	16.00	16.00	16.00	16.00	16.00	16.00
Exchange Rates																								
USD : CAD	link (K0)	x:x	0.78	0.78	0.78	0.78	0.78	0.78	0.78	0.78	0.78	0.78	0.78	0.78	0.78	0.78	0.78	0.78	0.78	0.78	0.78	0.78	0.78	0.78
UNDERGROUND MINING																								
Total Production																								
Resources	link	ktonnes	14	3,967	3,981		14	456	456	456	456	456	456	456	456	317	-	-	-	-	-	-	-	-
Au Grade	link	g/t	9.89	8.14	8.14		9.89	10.29	10.75	9.49	10.54	7.65	6.39	6.67	5.78	4.60	-	-	-	-	-	-	-	-
Contained Au	link	troy koz	4	1,038	1,042		4	151	158	139	155	112	94	98	85	47	-	-	-	-	-	-	-	-
Ag Grade	link	g/t	39.00	27.91	27.95		39.00	45.93	42.60	37.46	14.84	19.90	18.14	32.28	11.30	29.11	-	-	-	-	-	-	-	-
Contained Ag	link	troy koz	17	3,560	3,577		17	674	625	549	218	292	266	474	166	297	-	-	-	-	-	-	-	-
MINERAL PROCESSING																								
Total Production																								
Resource Processed	link	ktonnes	14	3,967	3,981	-	14	456	456	456	456	456	456	456	456	317	-	-	-	-	-	-	-	-
Operating Days	link (O2)	day	11	3,285	3,296	-	11	365	365	365	365	365	365	365	365	365	-	-	-	-	-	-	-	-
Average Plant Throughput	calc	ktpd	1.2	1.2	1.2	-	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	0.9	-	-	-	-	-	-	-	-
Recovery																								
Au	link (K0)	%	91.3%	91.3%	91.3%	-	91.3%	91.3%	91.3%	91.3%	91.3%	91.3%	91.3%	91.3%	91.3%	91.3%	-	-	-	-	-	-	-	-
Ag	link (K0)	%	87.8%	87.8%	87.8%	-	87.8%	87.8%	87.8%	87.8%	87.8%	87.8%	87.8%	87.8%	87.8%	87.8%	-	-	-	-	-	-	-	-
Metal Recovered																								
Au	calc	koz	4	948	952	-	4	138	144	127	141	102	86	89	77	43	-	-	-	-	-	-	-	-
Ag	calc	koz	15	3,125	3,141	-	15	592	549	482	191	256	234	416	146	260	-	-	-	-	-	-	-	-
SALES & NSR																								
Payable Metals																								
Au Payable	link (K0)	%	99.8%	99.8%	99.8%	-	99.8%	99.8%	99.8%	99.8%	99.8%	99.8%	99.8%	99.8%	99.80%	99.80%	-	-	-	-	-	-	-	-
	calc	koz	4	946	950	-	4	138	144	127	141	102	85	89	77	43	-	-	-	-	-	-	-	-
	calc	US$M	5.0	1,182.1	1,187.2	-	5.0	171.9	179.6	158.6	176.1	127.8	106.8	111.4	96.6	53.4	-	-	-	-	-	-	-	-
Ag Payable	link (K0)	%	99.0%	99.0%	99.0%	-	99.0%	99.0%	99.0%	99.0%	99.0%	99.0%	99.0%	99.0%	99.00%	99.00%	-	-	-	-	-	-	-	-
	calc	koz	15	3,094	3,109	-	15	586	543	478	189	254	231	412	144	258	-	-	-	-	-	-	-	-
	calc	US$M	0.2	49.5	49.7	-	0.2	9.4	8.7	7.6	3.0	4.1	3.7	6.6	2.3	4.1	-	-	-	-	-	-	-	-
Total Payable Metals	**calc**	**US$M**	**5.3**	**1,231.6**	**1,236.9**	**-**	**5.3**	**181.3**	**188.3**	**166.2**	**179.1**	**131.9**	**110.5**	**118.0**	**99**	**58**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
	calc	**C$M**	**6.8**	**1,579.0**	**1,585.8**	**-**	**6.8**	**232.4**	**241.4**	**213.1**	**229.6**	**169.1**	**141.6**	**151.3**	**126.8**	**73.7**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
Refining Charges																								
Au Refining Charge	link (K0)	US$/pay oz	5.00	5.00	5.00	-	5.00	5.00	5.00	5.00	5.00	5.00	5.00	5.00	5.00	5.00	-	-	-	-	-	-	-	-
	calc	US$M	(0.0)	(4.7)	(4.7)	-	(0.0)	(0.7)	(0.7)	(0.6)	(0.7)	(0.5)	(0.4)	(0.4)	(0.4)	(0.2)	-	-	-	-	-	-	-	-
Ag Refining Charge	link (K0)	US$/pay oz	0.50	0.50	0.50	-	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	-	-	-	-	-	-	-	-
	calc	US$M	(0.0)	(1.5)	(1.6)	-	(0.0)	(0.3)	(0.3)	(0.2)	(0.1)	(0.1)	(0.1)	(0.2)	(0.1)	(0.1)	-	-	-	-	-	-	-	-
Shipping Costs	input	shipments/yr	2	52	52	-	2	52	52	52	52	52	52	52	52	52	-	-	-	-	-	-	-	-
	link (K0)	$/shipment	5,485	5,485	5,485	-	5,485	5,485	5,485	5,485	5,485	5,485	5,485	5,485	5,485	5,485	-	-	-	-	-	-	-	-
	calc	US$M	(0.0)	(2.6)	(2.6)	-	(0.0)	(0.3)	(0.3)	(0.3)	(0.3)	(0.3)	(0.3)	(0.3)	(0.3)	(0.3)	-	-	-	-	-	-	-	-
	link (K0)	$/kg	2.32	2.32	2.32	-	2.32	2.32	2.32	2.32	2.32	2.32	2.32	2.32	2.32	2.32	-	-	-	-	-	-	-	-
	calc	kg	1	133	134	-	0.63	24	23	20	11	12	10	17	7	10	-	-	-	-	-	-	-	-
	calc	US$M	(0.0)	(0.3)	(0.3)	-	(0.0)	(0.1)	(0.1)	(0.0)	(0.0)	(0.0)	(0.0)	(0.0)	(0.0)	(0.0)	-	-	-	-	-	-	-	-
Total Shipping Costs	**calc**	**US$M**	**(0.0)**	**(2.9)**	**(2.9)**	**-**	**(0.0)**	**(0.3)**	**(0.3)**	**(0.3)**	**(0.3)**	**(0.3)**	**(0.3)**	**(0.3)**	**(0.3)**	**(0.3)**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
Audit Services	link (K0)	$/shipment	1,400	1,400	1,400	-	1,400	1,400	1,400	1,400	1,400	1,400	1,400	1,400	1,400	1,400	-	-	-	-	-	-	-	-
	calc	US$M	(0.0)	(0.7)	(0.7)	-	(0.0)	(0.1)	(0.1)	(0.1)	(0.1)	(0.1)	(0.1)	(0.1)	(0.1)	(0.1)	-	-	-	-	-	-	-	-
Total Shipping and Refining Charges	**calc**	**US$M**	**(0.0)**	**(9.8)**	**(9.9)**	**-**	**(0.0)**	**(1.4)**	**(1.4)**	**(1.3)**	**(1.2)**	**(1.0)**	**(0.9)**	**(1.0)**	**(0.8)**	**(0.7)**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
	calc	**C$M**	**(0.1)**	**(12.6)**	**(12.6)**	**-**	**(0.1)**	**(1.8)**	**(1.8)**	**(1.6)**	**(1.5)**	**(1.3)**	**(1.2)**	**(1.3)**	**(1.1)**	**(0.9)**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
Net Smelter Return *(before Royalties)*	**calc**	**US$M**	**5.2**	**1,221.8**	**1,227.1**	**-**	**5.2**	**179.9**	**186.9**	**164.9**	**177.9**	**130.8**	**109.5**	**117.0**	**98.0**	**56.8**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
	calc	**C$M**	**6.7**	**1,566.5**	**1,573.2**	**-**	**6.7**	**230.6**	**239.6**	**211.4**	**228.1**	**167.8**	**140.4**	**150.0**	**125.7**	**72.8**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
Royalties																								
Goldcorp NSR	link (K0)	%	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%	-	-	-	-	-	-	-	-
	calc	C$M	(0.1)	(15.7)	(15.7)	-	(0.1)	(2.3)	(2.4)	(2.1)	(2.3)	(1.7)	(1.4)	(1.5)	(1.3)	(0.7)	-	-	-	-	-	-	-	-
	calc	US$M	(0.1)	(12.2)	(12.3)	-	(0.1)	(1.8)	(1.9)	(1.6)	(1.8)	(1.3)	(1.1)	(1.2)	(1.0)	(0.6)	-	-	-	-	-	-	-	-
Guatemalan Government Royalty	link (K0)	%	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%	-	-	-	-	-	-	-	-
	calc	C$M	(0.1)	(15.7)	(15.7)	-	(0.1)	(2.3)	(2.4)	(2.1)	(2.3)	(1.7)	(1.4)	(1.5)	(1.3)	(0.7)	-	-	-	-	-	-	-	-
	calc	US$M	(0.1)	(12.2)	(12.3)	-	(0.1)	(1.8)	(1.9)	(1.6)	(1.8)	(1.3)	(1.1)	(1.2)	(1.0)	(0.6)	-	-	-	-	-	-	-	-
Total Royalties	calc	US$M	(0.1)	(24.4)	(24.5)	-	(0.1)	(3.6)	(3.7)	(3.3)	(3.6)	(2.6)	(2.2)	(2.3)	(2.0)	(1.1)	-	-	-	-	-	-	-	-
Net Smelter Return *(after Royalties)*	**calc**	**US$M**	**5.1**	**1,197.4**	**1,202.5**	**-**	**5.1**	**176.3**	**183.2**	**161.6**	**174.4**	**128.2**	**107.3**	**114.6**	**96.1**	**55.7**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
	calc	*US$/t*	*369*	*302*	*302*	-	*369*	*386*	*401*	*354*	*382*	*281*	*235*	*251*	*211*	*176*	-	-	-	-	-	-	-	-
	calc	**C$M**	**6.6**	**1,535.1**	**1,541.7**	**-**	**6.6**	**226.0**	**234.8**	**207.2**	**223.6**	**164.4**	**137.6**	**147.0**	**123.2**	**71.4**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
	calc	*C$/t*	*473*	*387*	*387*	-	*473*	*495*	*515*	*454*	*490*	*360*	*302*	*322*	*270*	*225*	-	-	-	-	-	-	-	-
OPERATING COSTS																								
Mining	calc	US$/t processed	49.06	49.06	49.06	-	49.06	49.06	49.06	49.06	49.06	49.06	49.06	49.06	49.06	49.06	-	-	-	-	-	-	-	-
	link (O1)	US$M	(0.7)	(194.6)	(195.3)		(0.7)	(22.4)	(22.4)	(22.4)	(22.4)	(22.4)	(22.4)	(22.4)	(22.4)	(15.6)	-	-	-	-	-	-	-	-
Processing	calc	US$/t processed	26.26	26.26	26.26	-	26.26	26.26	26.26	26.26	26.26	26.26	26.26	26.26	26.26	26.26	-	-	-	-	-	-	-	-
	link (O2)	US$M	(0.4)	(104.2)	(104.5)	-	(0.37)	(12.0)	(12.0)	(12.0)	(12.0)	(12.0)	(12.0)	(12.0)	(12.0)	(8.3)	-	-	-	-	-	-	-	-
General & Administration	calc	US$/t processed	13.24	17.27	17.26	-	13.24	16.43	16.71	16.72	16.72	16.72	16.72	16.72	16.72	24.03	-	-	-	-	-	-	-	-
	link (O3)	US$M	(0.2)	(68.5)	(68.7)	-	(0.18)	(7.5)	(7.6)	(7.6)	(7.6)	(7.6)	(7.6)	(7.6)	(7.6)	(7.6)	-	-	-	-	-	-	-	-
Total Operating Costs	**calc**	**US$/t processed**	**88.57**	**92.59**	**92.58**	**-**	**88.57**	**91.75**	**92.03**	**92.04**	**92.04**	**92.04**	**92.04**	**92.04**	**92.04**	**99.35**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
	calc	**US$M**	**(1.2)**	**(367.3)**	**(368.6)**	**-**	**(1.2)**	**(41.9)**	**(42.0)**	**(42.0)**	**(42.0)**	**(42.0)**	**(42.0)**	**(42.0)**	**(42.0)**	**(31.5)**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
INCOME																								
Net Operating Cash Flow	**calc**	**US$M**	**3.9**	**830.1**	**834.0**	**-**	**3.9**	**134.4**	**141.2**	**119.6**	**132.4**	**86.2**	**65.3**	**72.6**	**54.1**	**24.2**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
	calc	**US$/t**	**280.41**	**209.25**	**209.50**	**-**	**280.41**	**294.65**	**309.40**	**262.19**	**290.16**	**189.01**	**143.23**	**159.21**	**118.54**	**76.24**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
Au Cash Cost	*calc*	*US$/oz*	*342*	*425*	*424*	-	*342*	*341*	*328*	*367*	*332*	*446*	*528*	*509*	*580*	*781*	-	-	-	-	-	-	-	-
Au Cash Cost (net of by-product)	*calc*	*US$/oz*	*282*	*372*	*372*	-	*282*	*273*	*268*	*307*	*310*	*407*	*485*	*435*	*550*	*684*	-	-	-	-	-	-	-	-
Operating Margin	*calc*	%	*74%*	*67%*	*67%*	-	*74%*	*74%*	*75%*	*72%*	*74%*	*65%*	*59%*	*62%*	*55%*	*42%*	-	-	-	-	-	-	-	-

CAPITAL COSTS																							
Initial & Sustaining Capital Costs																							
Mining	link (C0)	US$M	(40.8)	(80.5)	(121.3)	-	(40.8)	(21.9)	(12.4)	(14.1)	(1.3)	(4.9)	(5.6)	(13.2)	(4.1)	(2.9)	-	-	-	-	-	-	-	-
Site Development	link (C0)	US$M	(5.6)	-	(5.6)	(5.6)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Mineral Processing	link (C0)	US$M	(35.8)	-	(35.8)	(3.6)	(32.2)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Tailings Management	link (C0)	US$M	(4.0)	(4.0)	(7.9)	-	(4.0)	(0.4)	(0.4)	(0.4)	(0.4)	(0.4)	(0.4)	(0.4)	(0.4)	(0.4)	-	-	-	-	-	-	-	-
Infrastructure	link (C0)	US$M	(8.8)	(0.1)	(8.9)	(4.6)	(4.2)	(0.1)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Off-Site Infrastructure	link (C0)	US$M	(13.1)	(10.0)	(23.0)	(7.9)	(5.1)	(7.8)	(2.1)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Project Indirects	link (C0)	US$M	(15.2)	-	(15.2)	(5.3)	(9.9)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
EPCM	link (C0)	US$M	(12.2)	-	(12.2)	(4.3)	(7.9)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Owner's Costs	link (C0)	US$M	(14.3)	-	(14.3)	(3.0)	(11.2)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Subtotal - Pre-Contingency	**calc**	**US$M**	**(149.7)**	**(94.5)**	**(244.2)**	**(34.4)**	**(115.3)**	**(30.2)**	**(15.0)**	**(14.6)**	**(1.7)**	**(5.4)**	**(6.1)**	**(13.7)**	**(4.6)**	**(3.3)**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
Contingency	link (C0)	US$M	(21.0)	(10.8)	(31.9)	(2.1)	(18.9)	(3.9)	(1.8)	(1.5)	(0.2)	(0.6)	(0.7)	(1.4)	(0.5)	(0.4)	-	-	-	-	-	-	-	-
Contingency (% of Total Costs)	calc	%	14%	11%	13%	6%	16%	13%	12%	10%	13%	11%	11%	10%	0	0	-	-	-	-	-	-	-	-
Total Initial & Sustaining Capital	**calc**	**US$M**	**(170.8)**	**(105.4)**	**(276.1)**	**(36.5)**	**(134.3)**	**(34.1)**	**(16.7)**	**(16.1)**	**(1.9)**	**(5.9)**	**(6.7)**	**(15.1)**	**(5.1)**	**(3.7)**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
Salvage & Closure Costs																							
Closure Costs	input	US$M	**-**	**(4.9)**	**(4.9)**	-	-	-	-	-	-	-	-	-	-	-	(4.9)	-		-	-	-	-	-
Closure Contingency	calc	US$M	**-**	**(1.0)**	**(1.0)**	-	-	-	-	-	-	-	-	-	-	-	(1.0)	-		-	-	-	-	-
Monitoring Costs	input	US$M	**-**	**(1.0)**	**(1.0)**			-	-	-	-	-	-	-	-	-	(0.2)	(0.2)	(0.2)	(0.2)	(0.2)	(0.2)		
Total - Closure Costs	**calc**	**US$M**	**-**	**(6.9)**	**(6.9)**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**(6.0)**	**(0.2)**	**(0.2)**	**(0.2)**	**(0.2)**	**(0.2)**	**-**	**-**
Total Capital Costs																							
Total Capital Costs	**calc**	**US$M**	**(170.8)**	**(112.2)**	**(283.0)**	**(36.5)**	**(134.3)**	**(34.1)**	**(16.7)**	**(16.1)**	**(1.9)**	**(5.9)**	**(6.7)**	**(15.1)**	**(5.1)**	**(3.7)**	**(6.0)**	**(0.2)**	**(0.2)**	**(0.2)**	**(0.2)**	**(0.2)**	**-**	**-**
WORKING CAPITAL																							
Working Capital & Reserve Accounts																							
Working Capital	calc	US$M	**(3.5)**	**3.5**	**-**	-	(3.5)	-	-	-	-	-	-	-	-	3.5	-	-	-	-	-	-	-	-
CASH FLOWS																							
Pre-Tax																							
Net Pre-Tax Free Cash Flow	calc	US$M	(170.3)	721.4	551.0	(36.5)	(133.9)	100.4	124.4	103.5	130.5	80.3	58.6	57.5	49.0	24.0	(6.0)	(0.2)	(0.2)	(0.2)	(0.2)	(0.2)	-	-
Cumulative Pre-Tax Free Cash Flow	calc	US$M	(170.3)	551.0	551.0	(36.5)	(170.3)	(70.0)	54.5	158.0	288.5	368.8	427.4	484.9	533.9	557.9	551.9	551.7	551.5	551.4	551.2	551.0	551.0	551.0
Post-Tax																								
Income Taxes	link (T1)	US$M	(0.5)	(110.1)	(110.7)	-	(0.5)	(18.5)	(19.6)	(16.1)	(18.2)	(10.6)	(7.7)	(8.4)	(6.9)	(4.0)	-	-	-	-	-	-	-	-
Net After-Tax Free Cash Flow	calc	US$M	(170.9)	611.2	440.3	(36.5)	(134.4)	81.8	104.8	87.4	112.2	69.7	50.9	49.2	42.1	19.9	(6.0)	(0.2)	(0.2)	(0.2)	(0.2)	(0.2)	-	-
Cumulative After-Tax Free Cash Flow	calc	US$M	(170.9)	440.3	440.3	(36.5)	(170.9)	(89.0)	15.8	103.2	215.4	285.1	336.0	385.2	427.3	447.2	441.2	441.0	440.9	440.7	440.5	440.3	440.3	440.3
ECONOMIC INDICATORS																								
Pre-Tax																								
Pre-Tax IRR	**calc**	**%**			**53.3%**																			
Pre-Tax Payback	**calc**	**Years**			**1.6**																			
Pre-Tax NPV @ 0%	**calc**	**US$M**			**551.0**																			
Pre-Tax NPV @ 5%	**calc**	**US$M**			**404.8**																			
Post-Tax																								
After-Tax IRR	**calc**	**%**			**43.9%**																			
After-Tax Payback	**calc**	**Years**			**1.8**																			
After-Tax NPV @ 0%	**calc**	**US$M**			**440.3**																			
After-Tax NPV @ 5%	**calc**	**US$M**			**317.0**																			

Source: JDS (2017)

24 Adjacent Properties

There are no adjacent properties relevant to the scope of this report.

25 Other Relevant Data and Information

There is no other relevant data or information relative to the scope of this report.

26 Interpretations and Conclusions

26.1 Risks

It is the conclusion of the QPs that the PEA summarized in this technical report contains adequate detail and information to support the potentially positive economic result. The PEA proposes the use of industry-standard equipment and operating practices. To date, the QPs are not aware of any fatal flaws for the Project. Table 26.1 identifies what are currently deemed to be the most significant internal Project risks, potential impacts, and possible mitigation approaches. The most significant potential risks associated with the Project are water/heat management, socio-political resistance to development of the mine in Guatemala, operating and capital cost escalation, delays in acquiring access road permit, suitability of filtered tailings for paste backfill, operating and capital cost escalation, unforeseen schedule delays, changes in regulatory requirements, ability to raise financing and a reduction in gold price. These risks are common to most mining projects, many of which may be mitigated, at least to some degree, with additional information, adequate engineering, planning and pro-active management.

Table 26.1: Main Project Risks

Risk	Explanation/Potential Impact	Possible Risk Mitigation
Water/heat Management	The PEA currently employs a simplified numerical groundwater flow model to simulate potential impacts associated with groundwater pumping for mine dewatering. This model does not account for the thermal effects or the heterogeneities of the aquifer.	Additional hydrogeology test work is needed in the next level of the study to develop a more detailed ground water model to understand pumping and injection strategies and to assess effects to water quality
Geology, Resource Modelling and Estimation	All projects benefit from increasing amounts of data and information in order to improve understanding and mitigate risks. However, there is a risk that unknown issues may arise with additional data. It is prudent to continue to improve the quantity and quality of information to decrease risk as much as possible.	Definition drilling in order to further refine and delineate structures and identify any potential problem areas.
Local Socio-political Risk to Mine Development	Guatemala has been resistant to mine development in the past. Potential impact could include delays in construction of mine, access road and overhead power line.	Both Marlin and Escobal mines have successfully been constructed within the last 10 years in Guatemala and are currently operating, so there is existing local support for mining projects.
		Voluntary royalty up to 4% will be considered in future economic models to provide direct benefit to local community.
Delay in Permit Acquisition for Access Road	The ability to secure the access road permit is an important step to ensure timely construction of the access road. This would avoid mine traffic travelling through the local town of Asunción Mita.	The development of close relationships with the local communities, land owners and government along with a thorough Environmental and Social Impact Assessment is required.
CAPEX and OPEX	The ability to achieve the estimated CAPEX and OPEX costs are important elements of Project success.	Further cost estimation accuracy with the next level of study, as well as the active investigation of potential cost-reduction measures would assist in the support of reasonable cost estimates.
	If OPEX increases then the NSR cut-off would increase and, all else being equal, the size of the mineable resource would reduce yielding fewer mineable tonnes.	
Development Schedule Impact due to Regulatory Delays	The Project development and economics could be impacted by any permitting or regulatory delays. A change in schedule would alter the Project economics.	If an aggressive schedule is to be followed, FS field work should begin as soon as possible. Continued discussions with local regulatory bodies are required to ensure avoidance of unforeseen delays in licensing/permitting.

Risk	Explanation/Potential Impact	Possible Risk Mitigation
Suitability of Using Filtered Tailings in paste Backfill	Filtered tailings may not provide suitable strength required by mine plan for paste backfill. The current design uses approximately one third of tailings produced in paste backfill. If these tailings are to be stored on surface, it would increase the size of the proposed DSTF.	Paste test work program is required in the next level of the study.
Ability to Attract Experienced/Trained Local Labour	The ability to attract and retain competent, experienced local labour is a key success factor for the Project.	The early search for trained labour as well as competitive salaries and benefits identify, attract and retain critical local personnel.
	High turnover or the lack of appropriate technical staff at the Project could result in difficulties meeting Project goals.	Further opportunity exists in hiring trained labour currently at Marlin mine which is scheduled for closure in 2017

Source: JDS (2017)

26.2 Opportunities

The main opportunities identified for the Project are listed below in Table 26.2.

Table 26.2: Identified Project Opportunities

Opportunity	Explanation	Potential Benefit
Additional geological/vein models	Refining the geological particularly within the Middle Splays to better delineate, define and refine vein models.	Expand and increase the number and size of the vein sets within the 100 vein sets between the Main and Main Splay veins increasing resources.
Develop further grade continuity and delineation	High-grade structures appear to demonstrate trends that should be further investigated for continuity and extension. Also, assist in refining mining shapes dimensions and orientations.	Increase confidence and continuity for reserve definition and expansion.
Development of Open Pit Mining	Shallow deposit amenable to open pit mining. Opportunity to develop open pit mine after initial start-up as underground operation.	Reduce mining CAPEX and OPEX. Increase in potential recoverable ounces based on initial high level assessment.
Recovery from processing of existing ROM stockpile	Stockpile of approximately 30,000 t exists on-site and is not considered in PEA processing plan.	Additional revenue (currently not included in economic model).
“Ore” Sorting	Ore has right attributes for “ore” sorting (colour, presence of quartz). This would result in more high-grade feed and lower overall feed to processing plant, reducing size of equipment required.	Reduce processing CAPEX and OPEX.
Identify Additional Resources	Potential exists to add to resource estimate through additional exploration. This would include along strike and down dip from the existing structures following high-grade shoots that have potentially economic viability.	Increase in size of deposit and extension of mine life.
Optimize Mine Plan	Optimize the mine plan and stope sequence.	Decrease ramp-up duration and potentially higher grades earlier in the mine life.
Power Generation From Geothermal Wells	Dewatering from hot water wells provides opportunity to generate power using Organic Rankine Cycle (ORC) modules.	Power produced on-site can reduce amount of power to be purchased from grid and reduce overall OPEX.
Sourcing Used Equipment	Purchase used equipment for processing plant, crushers, or UG fleet.	Reduce CAPEX.
Marlin Assets	Bluestone has Right of First Refusal on assets at Marlin mine.	Reduce CAPEX.
Backfill Cement Content	Paste backfill testing may reduce the cement content assumption.	Reduce mining OPEX.

Source: JDS 2017

27 Recommendations

It is recommended that the Cerro Blanco Project proceed to the Feasibility Study (FS) stage to further detail the Project schedule, engineering design, costs and revenue, increase geological confidence in addition to high-grade continuity and to improve accuracy of Project economics. The first phase of the recommended work should include a site hydrogeology test program and permit/EIA kick-off for the new proposed access road. A full list of field and test work related tasks needed to prepare for this study along with an engineering estimate are included in Table 27.1.

Table 27.1: Recommended Field/Test Work in Preparation of FS

Item	Approximate Cost (US$)
Hydrogeology test work and development of detailed ground water model	1,200,000
Access road and power line EIA	500,000
Access road permitting	500,000
Paste plant test work	225,000
Geotechnical investigations for DSTF	150,000
Evaluation of Marlin assets	100,000
Exploration and Feasibility Drill Program	2,900,000
Updated Geological and Resource Modelling and Estimation	100,000
Feasibility Study Engineering	1,800,000
Total	***7,475,000***

Source: JDS (2017)

28 References

Barton N, Lien R, and Lunde J, (1974). Engineering classification of rock masses for the design of tunnel support. Rock mechanics 6(4) 189-236.

Bieniawski Z.T., 1976. “Rock Mass Classification in Rock Engineering.” Proceedings of the Symposium on Exploration for Rock Engineering, Johannesburg. November 1 to 5.

Bieniawski Z.T., (1989) Engineering Rock Mass Classifications, John Wiley & Sons, New York.

Canadian Institute of Mining, Metallurgy and Petroleum (CIM), 2014, CIM Definition Standards on Mineral Resources and Reserves. Adopted by CIM Council May 2014.

Corbett, G.J., 1998, Epithermal Gold for Explorationists, AIG journal, Paper 2002-1, February 2002.

Corbett, G.J. and Leach, T.M., 1998, Southwest Pacific Rim Gold-Copper Systems: Structure, Alteration, and Mineralization: Special Publication, No. 6 of the Society of Economic Geology, pp. 201-205.

Donnelly, T.H., Shergold, J.H., Southgate, P.N. and Barnes, C.J. (1990). Events leading to global phosphogenesis around the Proterozoic Cambrian boundary. Geological Society of America, Bulletin 52: 273-287.

Hedenquist, J.W., Arribas, A., Jr., and Gonzalez-Urien, E., 2000, Exploration for epithermal gold deposits: Reviews in Economic Geology, v. 13, p. 245-277.

James Pindell & Stephen Barrett, 1990, Geological evolution of the Caribbean region; a Plate tectonic perspective, The Caribbean Region., Geological Association of America, The Geology of North America, H, 405-432.

James Pindell & Stephen Barrett, 1990, Geological evolution of the Caribbean region; a Plate tectonic perspective, The Caribbean Region., Geological Association of America, The Geology of North America, H, 405-432.

Potvin, Y. and Hadjigeorgiou, J., (2001) The Stability Graph for Open-Stope Design, Underground Mining Methods: Engineering Fundamentals and International Case Studies, Ed. William Hustrulid, SME, pp 513 to 520.

Voorhees, J.S., Cerro Blanco Technical Report for Glamis Gold Ltd., SEDAR filing, December 2005.

29 Units of Measure, Abbreviations and Acronyms

Symbol/Abbreviation	Description
'	minute (plane angle)
"	second (plane angle) or inches
°	degree
°C	degrees Celsius
3D	three-dimensions
A	ampere
a	annum (year)
ac	acre
Acfm	actual cubic feet per minute
ACK	apparent coherent kimberlite
AISC	All-in sustaining cost
ALT	active layer thickness
amsl	above mean sea level
AN	ammonium nitrate
ARD	acid rock drainage
Au	gold
AWR	all-weather road
B	billion
BD	bulk density
Bt	billion tonnes
BTU	British thermal unit
BV/h	bed volumes per hour
BwV	Breccia with veining
bya	billion years ago
C$	dollar (Canadian)
Ca	calcium
CCD	Counter current decantation
cfm	cubic feet per minute
CIL	Carbon in leach
CIM	Canadian institute of mining and metallurgy
cm	centimetre
cm^2	square centimetre
cm^3	cubic centimetre
COG	Cut-off grade
cP	centipoise
Cr	chromium
CRF	Cemented rock fill
CSS	Closed side setting
Cu	copper
d	day
d/a	days per year (annum)
d/wk	days per week
dB	decibel
dBa	decibel adjusted
DD	Diamond drill
DF	Drift and fill
DGPS	differential global positioning system

Symbol/Abbreviation	Description
DMS	dense media separation
dmt	dry metric ton
DST	Dry stack tailings
DSTF	Dry Stack Tailings Facility
DWT	dead weight tonnes
EA	environmental assessment
EIS	environmental impact statement
ELC	ecological land classification
ERD	explosives regulatory division
ESR	Excavation support ratio
FAR	Fresh air raise
FEL	Front-end loader
FS	Feasibility study
FT	Fine tuff
ft	foot
ft^2	square foot
ft^3	cubic foot
ft^3/s	cubic feet per second
g	gram
G&A	general and administrative
g/cm^3	grams per cubic metre
g/L	grams per litre
g/m	gallons per minute (us)
g/t	grams per tonne
Ga	billion years
gal	gallon (us)
GJ	gigajoule
GPa	gigapascal
GW	gigawatt
h	hour
h/a	hours per year
h/d	hours per day
h/wk	hours per week
ha	hectare (10,000 m^2)
HG	high grade
HLEM	horizontal loop electro-magnetic
hp	horsepower
HPGR	high-pressure grinding rolls
HQ	drill core diameter of 63.5 mm
Hz	hertz
ICP-MS	inductively coupled plasma mass spectrometry
in	inch
in^2	square inch
in^3	cubic inch
IRR	internal rate of return
JDS	JDS Energy & Mining Inc.
K	hydraulic conductivity
k	kilo (thousand)
KCA	Kappes, Cassiday & Associates
kg	kilogram
kg/h	kilograms per hour

Symbol/Abbreviation	Description
kg/m^2	kilograms per square metre
kg/m^3	kilograms per cubic metre
km	kilometre
km/h	kilometres per hour
km^2	square kilometre
kPa	kilopascal
kt	kilotonne
kV	kilovolt
kVA	kilovolt-ampere
kW	kilowatt
kWh	kilowatt hour
kWh/a	kilowatt hours per year
kWh/t	kilowatt hours per tonne
L	litre
L/min	litres per minute
L/s	litres per second
LDD	large-diameter drill
LG	low grade
LGM	last glacial maximum
LH	Long hole
LOM	life of mine
LT	Lithic tuff
m	metre
M	million
m/min	metres per minute
m/s	metres per second
m^2	square metre
m^3	cubic metre
m^3/h	cubic metres per hour
m^3/s	cubic metres per second
Ma	million years
MAAT	mean annual air temperature
MAE	mean annual evaporation
MAGT	mean annual ground temperature
mamsl	metres above mean sea level
MAP	mean annual precipitation
masl	metres above mean sea level
Mb/s	megabytes per second
mbgs	metres below ground surface
Mbm^3	million bank cubic metres
Mbm^3/a	million bank cubic metres per annum
mbs	metres below surface
mbsl	metres below sea level
Mct	million carats
mg	milligram
mg/L	milligrams per litre
min	minute (time)
mL	millilitre
mm	millimetre
Mm^3	million cubic metres
MMER	metal mining effluent regulations

Symbol/Abbreviation	Description
mo	month
MPa	megapascal
MSE	Mechanically Stabilized Earth
Mt	million metric tonnes
MVA	megavolt-ampere
MW	megawatt
MWMT	Meteoric water mobility testing
NAD	North American datum
NG	normal grade
Ni	nickel
NI 43-101	national instrument 43-101
Nm^3/h	normal cubic metres per hour
NQ	drill core diameter of 47.6 mm
OP	open pit
ORC	Organic Rankine Cycle
OSA	overall slope angles
oz	troy ounce
P.Geo.	professional geoscientist
Pa	Pascal
PAG	potentially acid generating
PEA	Preliminary economic assessment
PFS	preliminary Feasibility Study
PGE	platinum group elements
PK	pyroclastic kimberlite
PMF	probable maximum flood
ppb	parts per billion
ppm	parts per million
psi	pounds per square inch
QA/QC	quality assurance/quality control
QP	qualified person
RAR	Return air raises
RC	Reverse circulation
RC	reverse circulation
RMR	rock mass rating
ROM	Run of mine
rpm	revolutions per minute
RQD	rock quality designation
RVK	resedimented volcaniclastic kimberlite
s	second (time)
S.G.	specific gravity
Scfm	standard cubic feet per minute
SEDEX	sedimentary exhalative
SFD	size frequency distribution
SG	specific gravity
SLS	Solid-Liquid Separation
SRK	SRK Consulting (US) Inc.
st/kg	stones per kilogram
st/t	stones per metric tonne
t	tonne (1,000 kg) (metric ton)
t	metric tonne
t/a	tonnes per year

Symbol/Abbreviation	Description
t/d	tonnes per day
t/h	tonnes per hour
TCR	total core recovery
TMF	tailings management facility
tph	tonnes per hour
ts/hm^3	tonnes seconds per hour metre cubed
US	united states
US$	dollar (American)
UTM	universal transverse mercator
V	volt
VBT	Veins in breccia tuff
VEC	valued ecosystem components
VK	volcaniclastic kimberlite
VMS	volcanic massive sulphide
VSEC	valued socio-economic components
w/w	weight/weight
wk	week
wmt	wet metric ton
WOL	Whole ore leach
WRSF	waste rock storage facility
WTP	Water treatment plant
µm	microns
µm	micrometre

APPENDIX A
QP CERTIFICATES

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CERTIFICATE OF AUTHOR

I, Mazyar Mohaseb, P. Eng., do hereby certify that:

1. This certificate applies to the Technical Report entitled “NI 43-101 Preliminary Economic Assessment Technical Report on the Cerro Blanco Project, Guatemala”, with an effective date of February 7, 2017 and a report date of March 20, 2017 as revised on June 2, 2017, (the “Technical Report”) prepared for Bluestone Resources Inc.;

2. I am currently employed as Senior Project Manager with JDS Energy & Mining Inc. with an office at Suite 900 – 999 West Hastings Street, Vancouver, British Columbia, V6C 2W2;

3. I am a Professional Mechanical Engineer registered with the APEGBC, P.Eng. #36771. I am a graduate of the University of British Columbia with a degree in Bachelor’s of Applied Science, Mechanical Engineering, 2002. I have practiced my profession continuously since 2005;

   I have worked on mining projects worldwide since 2007. My roles have included: project management, project engineering, field engineering, procurement, CAPEX and OPEX development, infrastructure planning, mechanical engineering, estimating, and technical report writing.

4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of NI 43-101;

5. I visited the Cerro Blanco project on May 18, 2017;

6. I am responsible for Sections 1 to 5, 17.4.14, 18.17, 18.19 and 19 to 27 of this Technical Report;

7. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101;

8. I have had no prior involvement with the property that is the subject of this Technical Report;

9. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading; and

10. I have read NI 43-101, and the Technical Report has been prepared in accordance with NI 43-101 and Form 43-101F1.

Effective Date: February 7, 2017
Signing Date: June 2, 2017

(original signed and sealed) “Mazyar Mohaseb, P. Eng.”

Mazyar Mohaseb, P. Eng.

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CERTIFICATE OF AUTHOR

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

1. This certificate applies to the Technical Report entitled "NI 43-101 Preliminary Economic Assessment Technical Report on the Cerro Blanco Project, Guatemala", with an effective date of February 7, 2017 and a report date of March 20, 2017 as revised on June 2, 2017 (the "Technical Report") prepared for Bluestone Resources Inc.;

2. I am currently employed as a Senior Engineer, Metallurgy, with JDS Energy & Mining Inc. with an office at Suite 900 – 999 West Hastings Street, Vancouver, British Columbia, V6C 2W2;

3. I am a Professional Metallurgical Engineer registered with the APEGBC, P.Eng. #15868. I am a graduate of McMaster University with a Bachelors of Engineering, Metallurgy, 1984. I have practiced my profession intermittently since 1984 and have worked for the last 10 years consulting in the mining industry in metallurgy and process design engineering;

4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of NI 43-101;

5. I not personally visited the Cerro Blanco project site;

6. I am responsible for Sections 13 & 17 of this Technical Report;

7. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101;

8. I have not had prior involvement with the property that is the subject of this Technical Report;

9. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading; and

10. I have read NI 43-101, and the Technical Report has been prepared in accordance with NI 43-101 and Form 43-101F1.

Effective Date: February 7, 2017
Signing Date: June 2, 2017

(original signed and sealed) "Kelly McLeod, P. Eng."

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CERTIFICATE OF AUTHOR

I, Michael Makarenko, P. Eng., do hereby certify that:

1. This certificate applies to the Technical Report entitled "NI 43-101 Preliminary Economic Assessment Technical Report on the Cerro Blanco Project, Guatemala", with an effective date of February 7, 2017 and a report date of march 20, 2017 as revised on June 2, 2017, (the "Technical Report") prepared for Bluestone Resources Inc.;

2. I am currently employed as Senior Project Manager with JDS Energy & Mining Inc. with an office at Suite 900 – 999 West Hastings Street, Vancouver, British Columbia, V6C 2W2;

3. I am a graduate of the University of Alberta with a B.Sc. in Mining Engineering, 1988. I have practiced my profession continuously since 1988;

   I have worked in technical, operations and management positions at mines in Canada, the United States, Brazil and Australia. I have been an independent consultant for over ten years and have performed mine design, mine planning, cost estimation, operations & construction management, technical due diligence reviews and report writing for mining projects worldwide;

   I am a Registered Professional Mining Engineer in Alberta (#48091) and the Northwest Territories (#1359);

   I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of NI 43-101;

4. I visited the Cerro Blanco project on February 1, 2017;

5. I am responsible for Sections 15, 16 (except 16.3 & 16.4) of this Technical Report;

6. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101;

7. I have had no prior involvement with the property that is the subject of this Technical Report;

8. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading; and

9. I have read NI 43-101, and the Technical Report has been prepared in accordance with NI 43-101 and Form 43-101F1.

Effective Date: February 7, 2017
Signing Date: June 2, 2017

(original signed and sealed) "Michael Makarenko, P. Eng."

Michael Makarenko, P. Eng.

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CERTIFICATE OF QUALIFIED PERSON

Michael Levy, P.E., P.G.

I, Michael E Levy, P.E., P.G., do hereby certify that:

1. I am a Professional Engineer, employed as a Principal Geotechnical Engineer with SRK Consulting (U.S.), Inc. with an office at Suite 600, 1125 17th Street, Denver, CO, 80202;
2. This certificate applies to the technical report titled NI 43-101 Preliminary Economic Assessment Technical Report on the Cerro Blanco Project, Guatemala", with an effective date of February 7, 2017 and a report date of March 20, 2017 as revised on June 2, 2017, (the "Technical Report") prepared for Bluestone Resources Inc. ("the Issuer");
3. I am a registered Professional Engineer in the states of Colorado (#40268), California (#70578) and Arizona (#61372) and a registered Professional Geologist in the state of Wyoming (#3550). I am a current member of the International Society for Rock Mechanics (ISRM) and the American Society of Civil Engineers (ASCE);

   I received a bachelor's degree (B.Sc.) in Geology from the University of Iowa in 1998 and a Master of Science degree (M.Sc.) in Civil-Geotechnical Engineering from the University of Colorado in 2004. I have practiced my profession continuously since March 1999 and have been involved in a variety of geotechnical projects specializing in advanced analyses and design of mine excavations;
4. I visited the Cerro Blanco project on February 1, 2017;
5. I am responsible for preparation of sections 16.3 of the Technical Report;
6. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the National Instrument 43-101;
7. I have had no prior involvement with the property that is the subject of this Technical Report;
8. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101;
9. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1;
10. As of the effective date of the Technical Report and the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Effective Date: February 7, 2017
Signing Date: June 1, 2017

"Original Signed and Sealed"

Michael E. Levy, P.E, P.G.

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gdkirkham@shaw.ca

CERTIFICATE OF AUTHOR

I, Garth David Kirkham, P.Geo., do hereby certify that:

1) I am a consulting geoscientist with an office at 6331 Palace Place, Burnaby, British Columbia.

2) This certificate applies to the Technical Report entitled "NI 43-101 Preliminary Economic Assessment Technical Report on the Cerro Blanco Project, Guatemala", with an effective date of February 7, 2017 and a report date of March 20, 2017 as revised on June 2, 2017 (the "Technical Report") prepared for Bluestone Resources Inc.;

3) I am a graduate of the University of Alberta in 1983 with a B. Sc. I have continuously practiced my profession since 1988. I have worked on and been involved with many similar NI43-101 technical reports including Bralorne, Table Mountain, Monument Bay and Cerro Las Minitas.

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

5) I visited the Cerro Blanco project on May 8, 2017.

6) In the independent report entitled "NI 43-101 Preliminary Economic Assessment Technical Report on the Cerro Blanco Project, Guatemala" with effective date February 7, 2017, I am responsible for Sections for Sections 6, 7, 8, 9, 10, 11, 12 and 14.

7) I have not had prior involvement with the property that is the subject of this Technical Report.

8) I am independent of Bluestone Resources Inc.as defined in Section 1.5 of National Instrument 43-101.

9) I have read the definition of "qualified person" set out in National Instrument 43-101 and certify that by reason of education, experience, independence and affiliation with a professional association, I fulfil the requirements of a Qualified Person as defined in National Instrument 43-101.

10) I am not aware of any material fact or material change with respect to the subject matter of the technical report that is not reflected in the Technical Report and that this technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

11) I have read National Instrument 43-101, Standards for Disclosure of Mineral Properties and Form 43-101F1. This technical report has been prepared in compliance with that instrument and form.

Effective Date: February 7, 2017
Signing Date: June 2, 2017

(original signed and sealed) "Garth Kirkham, P.Geo."

Garth Kirkham, P.Geo.

Stantec Consulting Ltd.
100-300 Hagey Boulevard, Waterloo ON N2L 0A4

CERTIFICATE OF AUTHOR

I, Craig Johnston, P. Geo., do hereby certify that:

1. This certificate applies to the Technical Report entitled “NI 43-101 Preliminary Economic Assessment Technical Report on the Cerro Blanco Project, Guatemala”, with an effective date of February 7, 2017 and a report date of March 20, 2017 as revised on June 2, 2017, (the “Technical Report”) prepared for Bluestone Resources Inc.;

2. I am currently employed as a Professional Geoscientist acting as Vice President, Mining Sector Leader – Canada for Stantec Consulting Ltd. (Stantec) with an office at 100-300 Hagey Boulevard, Waterloo, Ontario, N2L 0A4;

3. I am a graduate of the McMaster University with an Honours B.Sc. in Geology in 1990 and the University of Waterloo with an M.Sc. in Earth Sciences (Hydrogeology) in 1994;

   I have practiced my profession continuously since 1994 and have worked for Stantec over this entire time. I have over 26 years’ experience in geology and hydrogeology and assessing environmental effects related to project developments;

   I am a Professional Geoscientist registered with the Association of Professional Geoscientists of Ontario (P.Geo.), license no. 538;

   I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of NI 43-101;

4. I have not visited the Cerro Blanco project and am relying on the site visit and inspections by Hhan Olsen, non-qualified person supporting the technical assessment under Section 16.4;

5. I am responsible for Section 16.4 of this Technical Report;

6. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101;

7. I have had prior involvement with the property that is the subject of this Technical Report;

8. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading; and

9. I have read NI 43-101 and Form 43-101F1 and have prepared Section 16.4 in accordance with NI 43-101 and Form 43-101F1.

Effective Date: February 7, 2017
Signing Date: June 2, 2017

(original signed and sealed) “Craig Johnston, P. Geo.”

Craig Johnston, M.Sc., P. Geo.
Vice President – Mining Sector Leader
Environmental Services Canada
Stantec Consulting Ltd.

JDS Energy & Mining Inc.
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CERTIFICATE OF AUTHOR

I, Robert Gutowski, P. Eng., do hereby certify that:

1. This certificate applies to the Technical Report entitled “NI 43-101 Preliminary Economic Assessment Technical Report on the Cerro Blanco Project, Guatemala”, with an effective date of February 7, 2017 and a report date of March 20, 2017 as revised on June 2, 2017 (the “Technical Report”), prepared for Bluestone Resources Inc.;

2. I am currently employed as a Senior Engineer, Mechanical/Infrastructure, with JDS Energy & Mining Inc. with an office at Suite 900 – 999 West Hastings Street, Vancouver, British Columbia, V6C 2W2;

3. I am a Professional Mechanical Engineer registered with the APEGBC, P.Eng. #161314. I am also a registered Professional Mechanical Engineer in Alberta P.Eng. #179426. I am a graduate of the University of Ottawa with a Bachelor’s of Science, Mechanical Engineering, 2008. I have practiced my profession continuously in the mining industry since 2008.

   I have worked on mining projects worldwide since 2008. My roles have included: multidiscipline field engineering, engineering management, commissioning management, cost estimation, infrastructure planning for mine developments in Canada, Central and South America.

4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of NI 43-101;

5. I have not personally visited the Cerro Blanco project site;

6. I am responsible for Section 18 (excluding 18.19 and 18.20) of this Technical Report;

7. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101;

8. I have not had prior involvement with the property that is the subject of this Technical Report;

9. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading; and

10. I have read NI 43-101, and the Technical Report has been prepared in accordance with NI 43-101 and Form 43-101F1.

Effective Date: February 7, 2017
Signing Date: June 2, 2017

(original signed and sealed) “Robert Gutowski, P. Eng.”

Robert Gutowski, P. Eng.

CERTIFICATE OF AUTHOR

I, Kenneth Embree, P. Eng., do hereby certify that:

1. This certificate applies to the Technical Report entitled “NI 43-101 Preliminary Economic Assessment Technical Report on the Cerro Blanco Project, Guatemala”, with an effective date of February 7, 2017, and a report date of March 20, 2017 as revised on June 2, 2017, (the “Technical Report”) prepared for Bluestone Resources Inc.

2. I am employed as Managing Principal of Knight Piésold Ltd. with an office at Suite 1400 - 750 West Pender Street, Vancouver, British Columbia, V6C 2T8.

3. I am a Professional Engineer in good standing with the Association of Professional Engineers and Geoscientists of in British Columbia in the area of geological engineering (No. 17439). I am also as a registered Professional Engineer in Ontario (No. 100040332).

4. I am a graduate of the University of Saskatchewan with a B.Sc. in Geological Engineering (1986). I have practiced my profession continuously since 1986. My experience includes tailings and waste and water management for mine developments in Canada, the US and South America.

5. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

6. I visited the Cerro Blanco project on February 2, 2017.

7. I am responsible for Section 18.20 of this Technical Report.

8. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of NI 43-101.

9. I have had no prior involvement with the property that is the subject of this Technical Report.

10. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

11. I have read NI 43-101, and the Technical Report has been prepared in accordance with NI 43-101 and Form 43-101F1.

Effective Date: February 7, 2017
Signing Date: June 2, 2017

(original signed and sealed) “Ken Embree, P. Eng.”

Ken Embree, P. Eng.