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Buriticá Project
NI 43-101 Technical Report
Feasibility Study
Antioquia, Colombia

Effective Date: February 24, 2016

Report Date: March 29, 2016 (Filing Date)

Prepared for:

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This report entitled Buriticá Project NI 43-101 Technical Report Feasibility Study, Antioquia, Colombia, effective as of February 24, 2016 was prepared and signed by the following authors:

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NOTICE

JDS Energy & Mining, Inc. prepared this National Instrument 43-101 Technical Report, in accordance with Form 43-101F1, for Continental Gold 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.

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This report contains "forward-looking information" as defined in applicable securities laws. Forward-looking information includes, but is not limited to, statements with respect to gold price and exchange rate assumptions, cash flow forecasts, projected capital and operating costs, metal or mineral recoveries, mine life and production rates, potential plans and operating performance, the estimation of the tonnage, grades and content of deposits, the extent of mineral resource and Mineral Reserves estimates, potential production from and viability of the properties, the timing and receipt of necessary permits and project approvals for future operations, access to project funding, and exploration results, and is based on current expectations that involve a number of business risks and uncertainties. Often, but not always, forward-looking information can be identified by the use of words such as "plans", "expects", "is expected", "budget", "scheduled", "estimates", "continues", "forecasts", "projects", "predicts", "intends", "anticipates" or "believes", or variations of, or the negatives of, such words and phrases, or statements that certain actions, events or results "may", "could", "would", "should", "might" or "will" be taken, occur or be achieved. Forward-looking information involves known and unknown risks, uncertainties and other factors which may cause the actual results, performance or achievements to be materially different from any of the future results, performance or achievements expressed or implied by the forward-looking information. These risks, uncertainties and other factors include, but are not limited to, the assumptions underlying the mineral resource and mineral reserve estimates not being realized, decrease of future gold prices, cost of labour, supplies, fuel and equipment rising, the availability of financing on attractive terms, actual results of current exploration, changes in project parameters, exchange rate fluctuations, delays and costs inherent to consulting and accommodating rights of local communities, title risks, regulatory risks and uncertainties with respect to obtaining necessary permits or delays in obtaining same, and other risks involved in the gold production, development and exploration industry, as well as those risk factors discussed in CGI's latest Annual Information Form and its other SEDAR filings from time to time. Forward-looking information is based on a number of assumptions which may prove to be incorrect, including, but not limited to, the availability of financing for CGI's production, development and exploration activities; the timelines for CGI's exploration and development activities on the property; the availability of certain consumables and services; assumptions made in mineral resource and mineral reserve estimates, including geological interpretation grade, recovery rates, price assumption, and operational costs; and general business and economic conditions. All forward-looking information herein is qualified by this cautionary statement. Accordingly, readers should not place undue reliance on forward-looking information. CGI and the authors of this technical report undertake no obligation to update publicly or otherwise revise any forward-looking information whether as a result of new information or future events or otherwise, except as may be required by applicable law.

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This technical report contains certain non-International Financial Reporting Standards measures. Such measures have non-standardized meaning under International Financial Reporting Standards and may not be comparable to similar measures used by other issuers.

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1 Executive Summary

1.1 Introduction

This report summarizes the results of the Feasibility Study (FS) completed by JDS Energy & Mining Inc. (JDS) as commissioned by Continental Gold Inc. (CGI) for the Buriticá Project, and was prepared in accordance with the Canadian Securities Administrators' National Instrument 43-101 and Form 43-101F1, collectively referred to as NI 43-101.

Buriticá is a precious metals resource project located in Antioquia, Colombia. The Project will exploit two mineralized vein systems (Yaraguá and Veta Sur) over a 14 year production period by way of a multiple ramp access underground mine, whole ore cyanide leach processing facility capable of processing 3,000 tonnes per day, dry-stacked filtered Tailing Storage Facility and related infrastructure.

1.2 Location, Access and Ownership

The Buriticá Project is located approximately 72 km northwest of Medellín in the Antioquia Department of north-western Colombia. Road access to site from the city is via a two-hour drive on paved Highway 62. The terrain in the Buriticá Project area is mountainous characterized by steep-sided valleys and subdued peaks with elevations ranging from around 600 m, to the east in the Cauca River valley, to a maximum of 2,200 m above mean sea level (masl). The mean average temperatures are relatively constant throughout the year ranging from 17°C to 26°C, depending on elevation, with a mean annual rainfall of 1.69 m with dry periods in January and February.

CGI has all necessary surface rights to conduct proposed mining operations described in this technical report. To the extent known by JDS, there are no option agreements or joint venture terms in place for the property, nor obligations on land covered by claims comprising the property.

1.3 History, Exploration and Drilling

Gold was mined in the Buriticá area since before the arrival of Spanish colonialists in the 17th century. The Spanish continued mining, principally from placer and colluvial deposits; however, high-grade veins were worked in shallow underground artisanal operations that continue to the present day. Notwithstanding the long mining history, there is no known historical resource or reserve estimates for the Buriticá Project area.

Grupo de Bullet S.A. (Bullet) held the main concessions over Buriticá prior to CGI's purchase in 2007 and over the last 20 years undertook development of the Yaraguá prospect. CGI currently operates a 35 t/d mine and processing facility at Yaraguá which has been in operation since the early 1990s. As part of the mine development and ongoing exploration, CGI has also developed three commercial-scale mine access-ways; the Higabra Tunnel, and the Veta Sur and Yaraguá ramps which are part of the proposed mine plan.

Exploration activities in the Buriticá Project conducted prior to and during 2012 consisted of topographic and geological mapping, aerial magnetic and radiometric surveys, geochemical soil surveys and other surface sampling, underground mapping and channel sampling, and drilling at both Yaraguá and Veta Sur.

From 2012 to 2015, CGI continued a program of systematic channel sampling of the underground openings, results of which indicate that the high gold grades are continuous along strike and within the vertical range sampled for several of the Yaraguá vein sets. The channel samples are considered representative and have been incorporated into the data set for the mineral resource estimate.

Drilling in 2014 and 2015 focused on converting Inferred resource into the Measured and Indicated categories, as well as growing the overall mineral resource.

Table 1.1: Verified Buriticá Database as of May 11, 2015

Sampling Type	Area	Drill Holes/ Channels	Number Samples	Metres Sampled
Surface Drill hole	Yaraguá, Veta Sur and Laurel	391	141,775	162,664
Underground Drill hole	Yaraguá and Veta Sur	345	115,011	108,339
Channel samples, surface samples	Yaraguá and Veta Sur	4084	11,032	7,215
Total		**4,820**	**267,818**	**278,218**

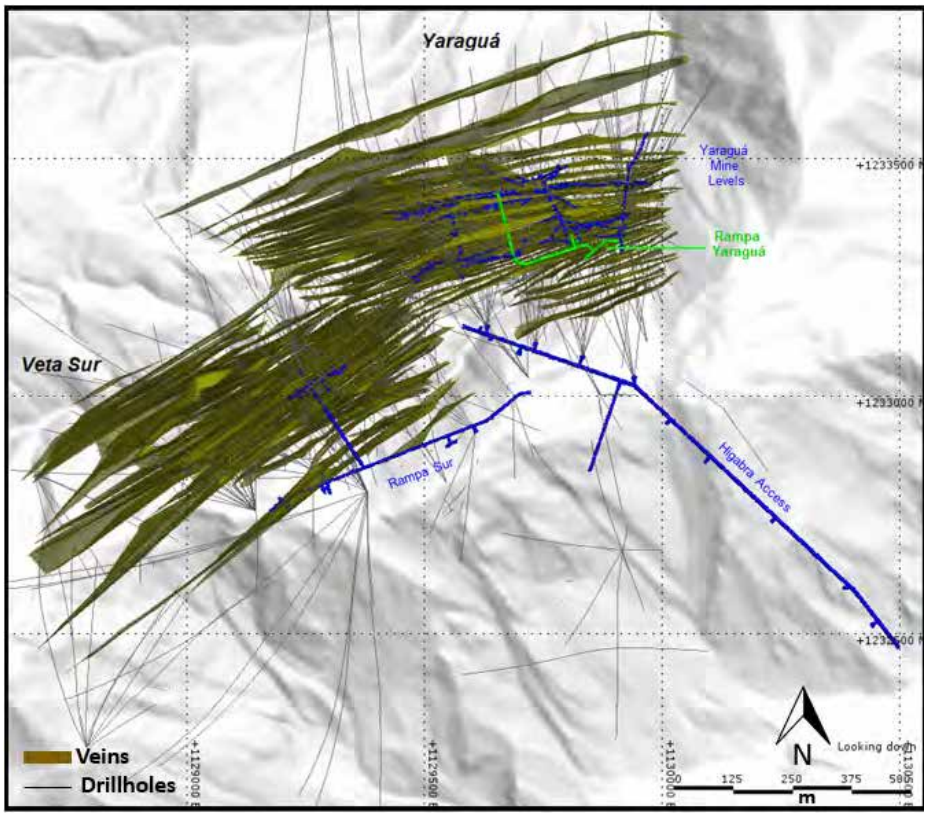
Source: JDS, 2016

1.4 Geology & Mineralization

Buriticá is a porphyry related epithermal carbonate base metal (CBM) gold narrow vein/breccia system. Gold is the primary economic metal. Mineralization occurs in swarms of narrow sheet-like structure envelopes that are called "veins" in this report and also occurs in the hanging wall or footwall. There are two principal orientations of these vein swarms that form the Veta Sur and Yaraguá deposits (see Figure 1.1).

The Buriticá Project area is transected (and geologically partitioned) by a set of regionally extensive north-south to north-northwest trending faults. To the east of the mineralized systems, the steeply-dipping Tonusco Fault system cuts off the intrusion complex and related hydrothermal alteration envelopes. A set of east-dipping faults to the west of and possibly related to the Tonusco Fault cut across the mineralization but appear to exhibit relatively small displacements.

Figure 1.1: Orientation of Veins at Yaraguá and Veta Sur



Source: CGI, 2015

1.5 Mineral Resource Estimates

The current mineral resource estimate (Mining Associates Ltd., Independent Technical Report and Resource Estimate On The Buriticá Gold Project 2015, Buriticá Gold Project, Colombia, May 2015) is an update to the 2014 estimate accompanied by the Technical report prepared by Mining Associates (MA), and utilized a similar but more advanced estimation methodology on the larger and more extensive database. The cut-off date for information used in the 2015 MA report, and thus the effective date of the mineral resource estimate is 11 May 2015. Mineral Resources that are not reserves do not have demonstrated economic viability.

Table 1.2: Combined Mineral Resources above 3 g/t COG

Combined Yaraguá and Veta Sur Mineral Resources above 3 g/t Au cut-off, May 11, 2015							
Category	**M tonnes**	**Au g/t**	**Ag g/t**	**AuEq g/t**	**Au Moz**	**Ag Moz**	**AuEq Moz**
Measured	0.89	19	55	19.9	0.54	1.58	0.57
Indicated	12	10.2	32	10.7	3.94	12.4	4.14
M and I	12.89	10.8	34	11.4	4.48	13.98	4.71
Inferred	15.6	9	29	9.5	4.5	14.7	4.8

Source: CGI, 2015

Table 1.3: Yaraguá Mineral Resources above 3 g/t COG

Yaraguá Mineral Resources above 3 g/t gold cut-off grade, May 11, 2015							
Category	**M tonnes**	**Au g/t**	**Ag g/t**	**AuEq g/t**	**Au Moz**	**Ag Moz**	**AuEq Moz**
Measured	0.59	18.9	45	19.6	0.36	0.84	0.37
Indicated	7.82	9.2	30	9.7	2.3	7.44	2.43
M and I	8.41	9.8	31	10.3	2.66	8.28	2.8
Inferred	8.8	9.1	30	9.6	2.6	8.4	2.7

Source: CGI, 2015

Table 1.4: Veta Sur Mineral Resources above 3 g/t COG

Veta Sur Mineral Resources classified above 3 g/t gold cut-off grade, May 11, 2015							
Category	**M tonnes**	**Au g/t**	**Ag g/t**	**AuEq g/t**	**Au Moz**	**Ag Moz**	**AuEq Moz**
Measured	0.3	19.2	76	20.5	0.19	0.7	0.2
Indicated	4.17	12.2	37	12.8	1.63	5	1.71
M and I	4.48	12.6	40	13.3	1.82	5.7	1.91
Inferred	6.8	8.9	29	9.4	2	6.3	2.1

Note – Reported tonnage and grade figures have been rounded from raw estimates to reflect the order of accuracy of the estimate. Minor variations may occur during the addition of rounded numbers. There have been no assumptions made as to metal prices or recoveries in this mineral resource estimate other than gold equivalents that are calculated for AuEq = Au + Ag/60. M in Figures and Tables are in millions.

Source: CGI, 2015

1.6 Mineral Reserve Estimates

The effective date for the mineral reserve estimate contained in this report is July 31, 2015. Mineral Reserves are included in Total Mineral Resources except for the dilution component of Mineral Reserves that was not reported in Mineral Resources. Mineral Resources include vein domains only, while Mineral Reserves include metal contained in dilution tonnes comprised of “halo” resource blocks classified as an indicated resource outside of the vein domains.

The Qualified Person (QP) has not identified any risk including legal, political, or environmental that would materially affect potential Mineral Reserves development except for the following: 1) Continued unauthorized mining activities; and, 2) successfully securing from the Colombian government the required permits for Project development and operation. The QP is not aware of unique characteristics related to this Project that would prevent the granting of such permits.

Table 1.5: Yaraguá Mineral Reserve Estimate

Category	M tonnes	Au g/t	Ag g/t	Au Moz	Ag Moz
Proven	0.45	20.5	47.6	0.3	0.69
Probable	8.38	7	20.8	1.89	5.61
Total P&P	8.83	7.7	22.2	2.19	6.3

Notes: Based on a 3.8 g/t cut-off grade, US$950 per ounce gold price, and COP:US$ exchange rate of 2,850. Rounding of some figures may lead to minor discrepancies in totals.

Source: CGI, 2015

Table 1.6: Veta Sur Mineral Reserve Estimate

Category	M tonnes	Au g/t	Ag g/t	Au Moz	Ag Moz
Proven	0.23	22.2	84.7	0.16	0.62
Probable	4.66	9.1	25.4	1.36	3.8
Total P&P	4.89	9.7	28.1	1.52	4.42

Notes: Based on a 4.0 g/t cut-off grade, US$950 per ounce gold price, and COP:US$ exchange rate of 2,850. Rounding of some figures may lead to minor discrepancies in totals.

Source: CGI, 2015

Table 1.7: Total Mineral Reserve Estimate

Category	M tonnes	Au g/t	Ag g/t	Au Moz	Ag Moz
Proven	0.68	21.1	60	0.46	1.31
Probable	13.04	7.8	22.5	3.25	9.41
Total	13.72	8.4	24.3	3.71	10.72

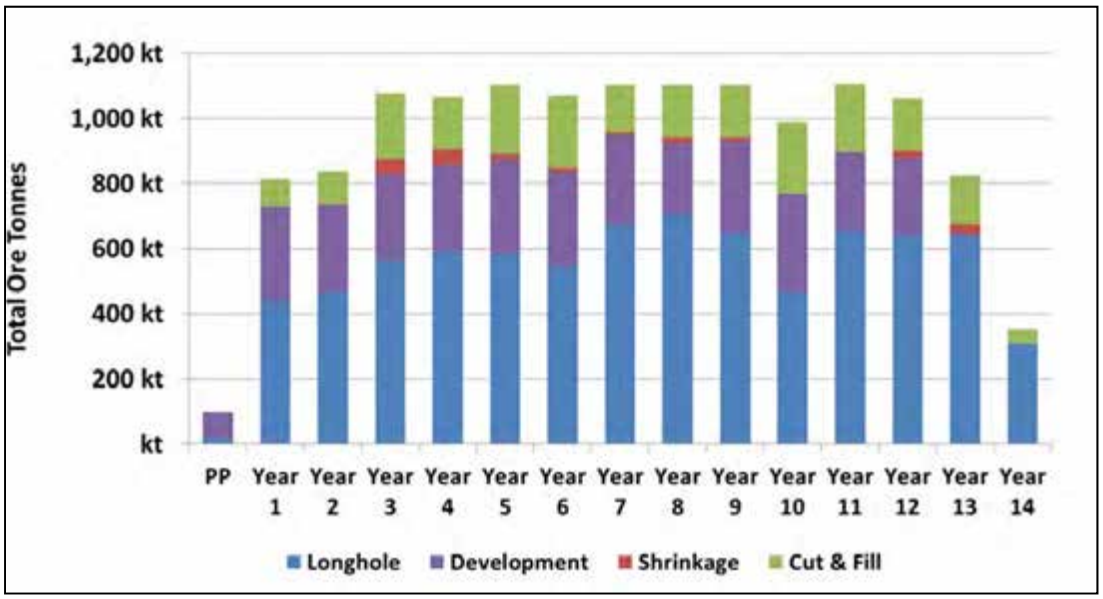
Notes: Based on cut-off grades of 3.8 g/t for Yaraguá and 4.0 g/t for Veta Sur, $950 per ounce gold price, and COP:US$ exchange rate of 2,850. Rounding of some figures may lead to minor discrepancies in totals.

Source: CGI, 2015

1.7 Mining

The FS mine plan is based on a multiple ramp access underground mining operation initially producing 2,100 ore tonnes per day ("t/d"), ramping up to 3,000 t/d by year three.

Figure 1.2: Breakdown of Mining Methods at Buriticá

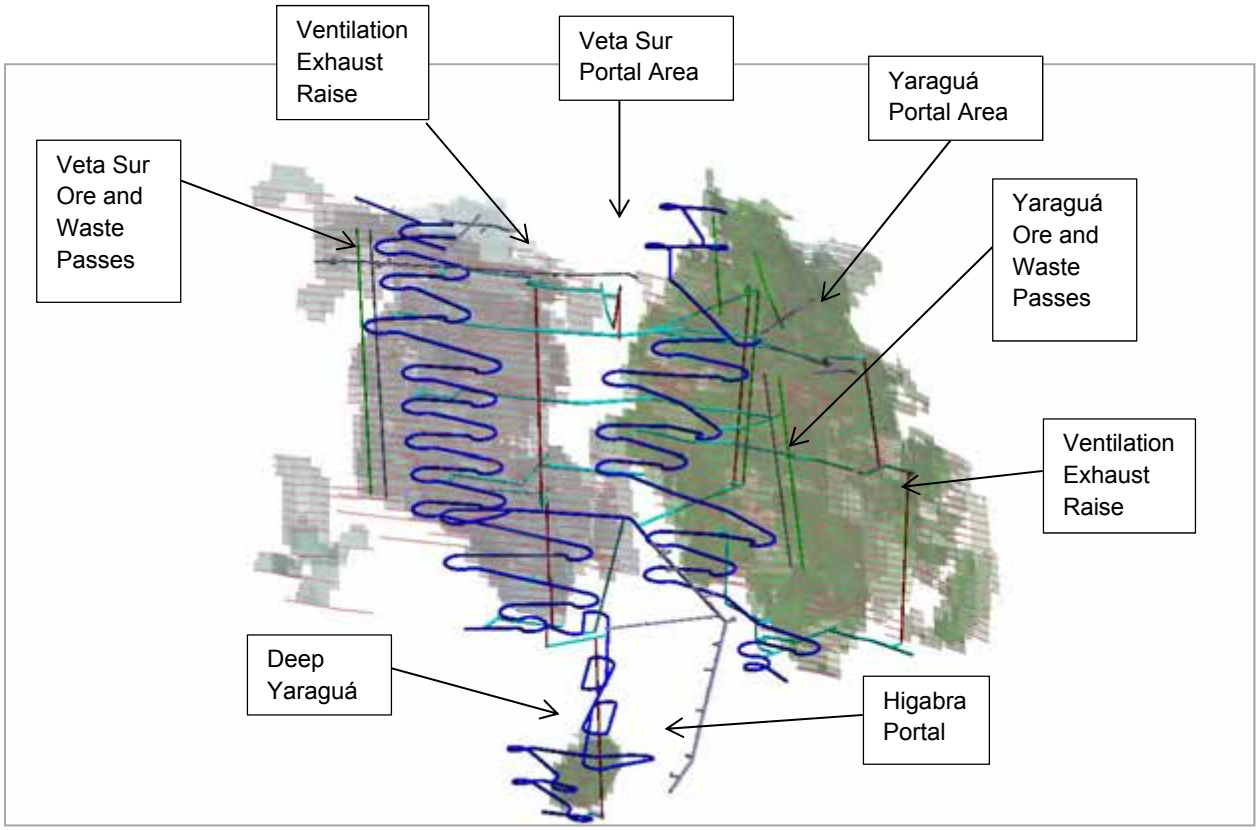


Source: JDS, 2016

The mine is designed to initially develop two high-grade zones to minimize pre-production development capital and maximize early revenues. Mining methods selected for the Buriticá Project were chosen to maintain mining flexibility and selectivity for the various ground conditions anticipated. The majority of Mineral Reserves will be mined by longhole open stoping (58% stoping, plus 25% stope development) on 15 m sublevels, and overhand cut and fill (15%). Some shrinkage stope extraction will be used for narrower, isolated veins. Average diluted stope width is 2.4 m; however, widths can exceed 10 m where veins have been combined. Minimum mining width is 1.0 m. Paste backfill made from a mixture of tailing and cement will be the primary backfill material, with unconsolidated waste rock being used in the cut and fill stopes. An internal ore-pass system will direct ore and waste to the Higabra tunnel (already constructed by CGI), the main haulage level which will daylight adjacent to the process plant.

The majority of the Mineral Reserves in the Yaraguá and Veta Sur vein systems are located above the elevation of the Higabra tunnel, providing an advantageous gravity scenario for ore and waste movement and dewatering.

Figure 1.3: Underground Development at Buriticá



Source: JDS, 2015,

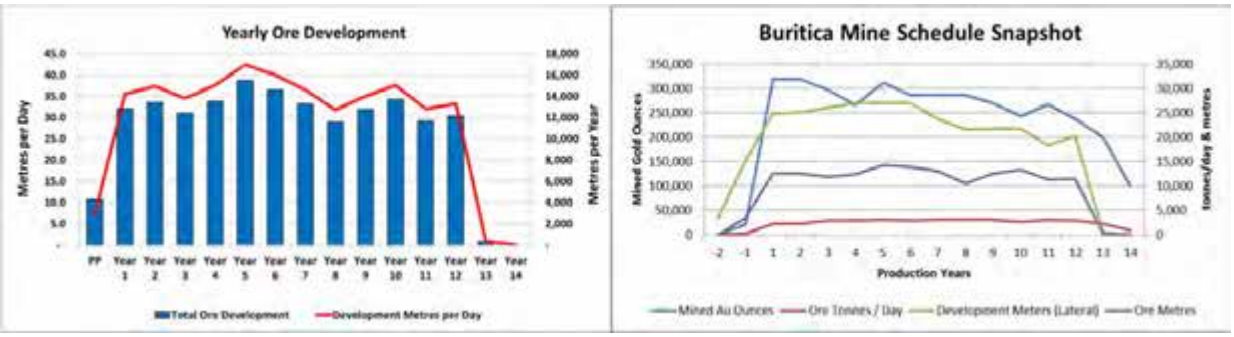
A comprehensive geomechanical characterization program was completed by SRK in 2015 to bring the overall characterization of the Project to FS level (SRK 2016). The selection of mining methods is driven primarily by geotechnical considerations, interpreted geometry of the veins and distribution of estimated precious metal values within those veins.

The permanent ventilation network will include a main exhaust fan, installed underground in a dedicated gallery, capable of delivering 19,200 m^3/m. Fresh air is drawn in from the three portals and directed via the main ramps to active mining levels with the use of regulators. A series of 4.0 m and 5.0 m diameter exhaust raises will collect and direct exhaust air to the main fans and out of the mine.

Dewatering of all levels above the Higabra access will occur by gravity to the greatest extent possible. Dewatering of levels below Higabra will be enabled by the establishment of drainage galleries and collection sumps along and proximal to the main decline. Dewatering is a critical activity for development below the Higabra level and the mine development schedule is designed to allow time for dewatering.

The principal method of backfilling for Buriticá will be with paste backfill comprised of filtered tailing, water, and cement in varying proportions. The use of cemented backfill enables secondary stopes along strike to be mined directly adjacent to the primary backfilled stopes without a rib pillar, and prevents sterilization of adjacent veins. Loose rock fill (LRF), and lesser amounts of Cemented Rock Fill (CRF), will be used.

Figure 1.4: Annual Ore Development and Mine Schedule



Source: JDS, 2016

1.8 Metallurgical Testing and Mineral Processing

A considerable amount of testing has been undertaken on the Buriticá Project prior to the Feasibility Study as previously summarized in the 2014 PEA (M3, NI 43-101 Technical Report Preliminary Economic Assessment, Antioquia, Colombia, November 17, 2014). The 2015 metallurgical test program included mineralogy, comminution, gravity separation, flotation, cyanidation, cyanide destruction and solid/liquid separation studies. Historical and Feasibility Study test work results indicate that the mineralization responded well to flotation and to cyanide leaching for precious metal extraction.

For the feasibility program, samples from 100 drill composites from 83 drill holes were used to create 45 variability composites representative of the feasibility mine plan. The "Year 1 to 5" optimization composite was prepared from 23 of the variability composites and included intercepts from 46 holes.

Grindability test work indicated that the hardest 80th Percentile Bond Work Index was 17.6 kWh/t and the SMC A*b was 28, which places the ore in the hard to very hard classification for comminution.

Given test work results for gravity concentration and leaching of gravity tailing from the variability test work program, the equations below were created to calculate the recoveries of materials to be processed throughout the life of the mine.

1.9 Recovery Equations

For Veta Sur, the recovery relationships proposed for gold grades between 3.0 and 24.0 g/t Au and silver grades between 5.0 and 105.0 g/t Ag, based on the feasibility data would be:

- Gold Recovery (%) = 95.627 – 0.006861 x Arsenic (ppm); and
- Silver Recovery (%) = 64.408 – 0.4317 x Silver Grade (g/t).

For Yaraguá the recovery relationships proposed for gold grades tested between 1.0 and 93.0 g/t Au and silver grades between 3.0 and 190.0 g/t Ag, based on the feasibility data would be:

- Gold Recovery (%) = 102.4 – 1.0672 x Fe (%); and
- Silver Recovery (%) = 72.864 – 0.2787 x Silver Grade (g/t).

1.10 Recovery Methods

The process plant utilizes conventional technology and equipment, which are standard to the industry. The process plant is designed to process 3,000 t/d, or 1,095,000 tonnes per year (t/y) at 91% availability, operating for 365 days per year.

The process design follows these steps: Crushing > Grinding > Gravity Concentration > Cyanide Leach > Counter-Current Decantation (CCD) > Merrill Crowe > On-site refining to doré bars. Treated tailing will be dewatered by filtering prior to disposal and process water will be recycled to minimize environmental impact

The process mass balance was developed for the Buriticá process using MetSim. The process simulation assumed the recoveries and grades based on completed test work as shown in Table 1.8.

Table 1.8: Metal Production Schedule

Parameter	Unit	Au	Ag
Mine Head Grades	g/t	8.5	24.2
Gravity Recovery	%	40 to 90	9
Leach Extraction	%	90	56
CCD Wash Efficiency	%	99	99
Merrill Crowe Extraction	%	99	99
Overall Extraction	%	96	60
Soluble Loss	%	0.5	0.5
Overall Plant Recovery	%	95	59
Metal Production	kg/d	24	43
Metal Production	Troy oz/y	284,000	503,000

Source: M3, 2016

The ore will be processed by a jaw crusher followed by a semi-autogenous (SAG) mill in closed circuit with a pebble crusher and a ball mill to a target grind size of 80% passing (P80) 74 µm. Combined SAG and ball mill discharge will feed the gravity circuit. Gravity concentrate will feed the cleaner table to produce a gravity concentrate that can be refined directly.

All gravity tailing will be classified by a hydro-cyclone system with overflow feeding the trash screen to leaching circuit. Leaching will be performed with cyanide and oxygen on agitated tanks with pH control by milk of lime. The slurry from the leaching circuit will flow through a CCD system of four high rate thickeners of 17 m diameter each. The solution from CCD will follow to Merrill Crowe system for gold recovery and slurry will be treated for cyanide destruction prior to thickening and filtration in preparation for proper disposal.

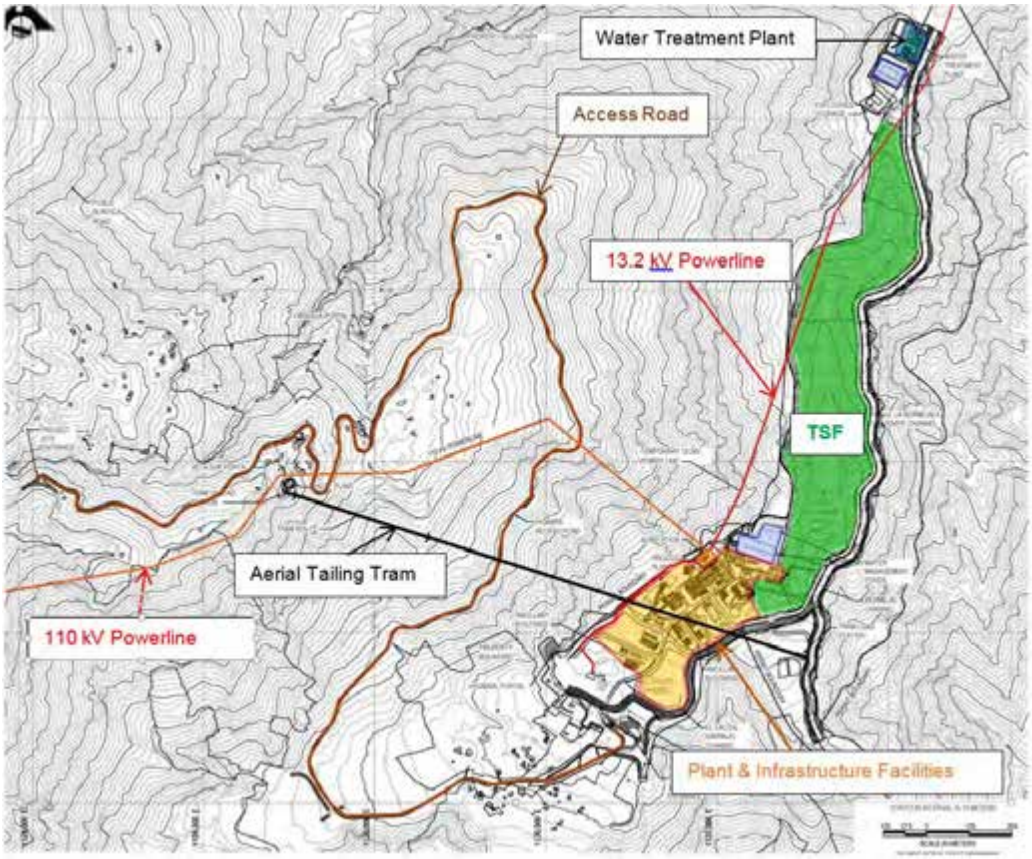
1.11 Infrastructure

The key support infrastructure includes the following (see Figure 1.5):

- A 5.9 km access road between the existing property entrance and the Higabra Valley, originating at the paved road leading to Buriticá;
- Process plant with security, administration, and personnel facilities;
- Paste backfill plant for providing cemented paste to the underground workings;
- Aerial Tram for transporting tailing from the process plant to the paste backfill plant;
- Mine support facilities including mobile equipment maintenance, mine personnel facilities, and shotcrete mixing plant;
- Explosive storage facility;
- Utility infrastructure for the site: water, sewer, fire protection, & communications;
- 13.2 kV grid power supply for the pre-production phase;
- 110 kV power transmission line connected to the EPM national electricity grid;
- Mine water sediment settling ponds and water treatment plant;

- Surface water handling infrastructure to manage local streams and runoff from the facilities; and
- Tailing Storage Facility (TSF) for filtered tailing and waste rock in a lined area.

Figure 1.5: General Site Layout



Source: JDS, 2016

1.12 Environment and Permitting

CGI currently operates the Yaraguá underground mine, which is located in the same area as the proposed Buriticá Project. Yaraguá presently operates under a global environmental license granted by the regional environmental authority of the department of Antioquia, Corantioquia.

Environmental studies for a 2,000 t/d operation, (the Buriticá Project) began in 2010 with baseline studies, hydrogeological studies in 2011 and 2012, and preparation of an Environmental Impact Assessment (Estudio de Impacto Ambiental) (EIA), which was submitted to Corantioquia on December 23, 2013.

On September 15, 2015, the company announced that it had requested the National Government of Colombia to assume the responsibility of reviewing an updated application for the Buriticá Project as a Project of National Strategic Interest (PINES) Project. Consequently, the company withdrew its application for the modification of the EIA from Corantioquia and filed a new EIA with ANLA on January 20, 2016. Since then, a subsequent court ruling removed ANLA as exclusive permitting authority over PINES projects.

Baseline environmental and monitoring studies have been conducted to support the ongoing operation and EIA application.

Baseline surface water quality generally meets discharge standards except in the upper section of Quebrada La Mina and more recently Quebrada El Sauzal where the majority of the evaluated water quality parameters exceed Colombian surface water discharge limits, most likely due to artisanal mining activity.

Surface water quality discharge standards became more stringent with Colombia Resolution 631, 2015 which will be in effect as the Buriticá Project moves into production and all contact and mine discharge water will need to be treated. Groundwater from mine dewatering has naturally elevated levels of chloride and sulphate and will require treatment to meet new discharge criteria. Other contact waters predominantly originating from the TSF, ore stockpile and process plant areas, will be directed to collection ponds for treatment and/or returned to the process plant, or discharged.

Forestry surveys to identify species of concern have been conducted. No impacts to construction or operations, beyond typical mitigation measures, or exchanges have been identified. The Project area vegetative cover is mainly comprised of a mosaic of crops, pastures, and natural spaces (15%), riparian forest (40%), and other grasslands (45%).

In response to informal community meetings, residents generally realize the potential of formal job creation in the area as well as the potential improvements for transportation resulting from the Project. Residents also see the operation as a welcome alternative to certain third party mining, which has been creating environmental and social concerns including criminal activity. Residents had many questions concerning interruption to established community paths, and mitigation measures being implemented for the Project. Ongoing dialog with both residents and Environmental and Municipal representatives will continue, and be part of the social management plan.

A concern for residents is illegal mining, which involves primarily people from outside the Buriticá area. As well, residents display environmental and social concerns and unease due to criminal activity and challenges to the respect for the rule-of-law. Cooperative programs to legitimize several credible associates in a mining formalization program along with government initiatives to remove illegal mining activities from the area are in place to address community concerns.

1.13 Capital Costs

Project capital costs total US$662M, consisting of the following:

Initial Capital Costs – includes all costs to develop the property to a sustainable production of 2,100 t/d. Initial capital costs total $389M and are expended over a 35-month pre-production construction and commissioning period; and

Sustaining Capital Costs – includes all costs related to expansion of production to 3,000 t/d and the acquisition, replacement, or major overhaul of assets required to sustain operations. Sustaining capital costs total $273M and are expended in operating years 1 through 13.

Table 1.9: Level 1 Capital Estimate Detail

Description	Initial Capital (US$ M)	Sustaining Capital (US$ M)	Total Capital (US$ M)
Site Development	10.9	-	10.9
Underground Mining	86.5	178.3	264.8
Processing Facilities	97.6	11.6	109.2
Tailing & Waste Rock Management	7.7	16.1	23.8
Off-Site Infrastructure	10.0	12.7	22.7
On-Site Infrastructure	45.3	18.7	64.0
Project Indirect Costs	28.5	9.1	37.5
Engineering & EPCM	27.8	0.6	28.4
Owners Costs	21.8	4.3	26.1
Taxes	17.7	14.0	31.7
Contingency	35.4	7.0	42.4
Total	**389.2**	**272.5**	**661.7**

Note: estimates in Q1 2016 dollars with no escalation
Note: Subtotals/totals may not match due to rounding
Source: JDS, 2016

1.14 Reclamation & Closure

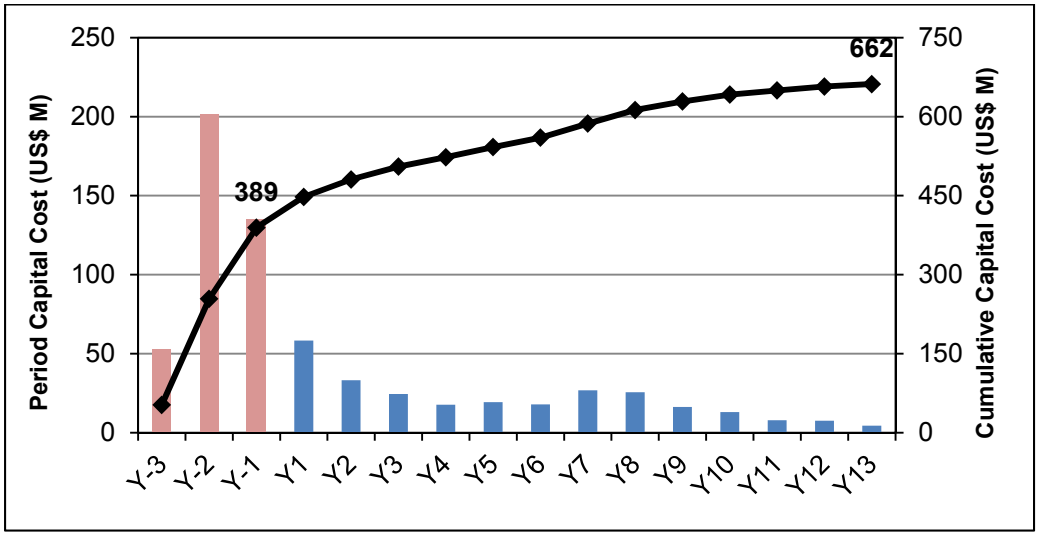
Progressive closure costs (the TSF soil cover) are incurred between Years 5 and 14. Demolition and final closure activities are incurred at the end of Year 14 and through Year 20 with the exception of ongoing water treatment, which has been assumed to continue in perpetuity.

Table 1.10: Reclamation & Closure Cost Summary

Category	Total Cost (US$ M)	%
Progressive Closure (activities occurring during operations)	0.8	5
Demolition & Closure	8.6	49
Ongoing Monitoring & Maintenance	8.0	46
Total	**17.5**	**100**

Source: JDS, 2016

Figure 1.6: Life of Mine Capital Cost Profile



Source: JDS, 2016

1.15 Operating Costs

Life of mine (LOM) operating costs for the Project average $100.50/t processed. The operating costs exclude off-site costs (such as shipping and refining costs), taxes, and government royalties. These cost elements are used to determine the net smelter return (NSR) in the economic model, and are discussed elsewhere.

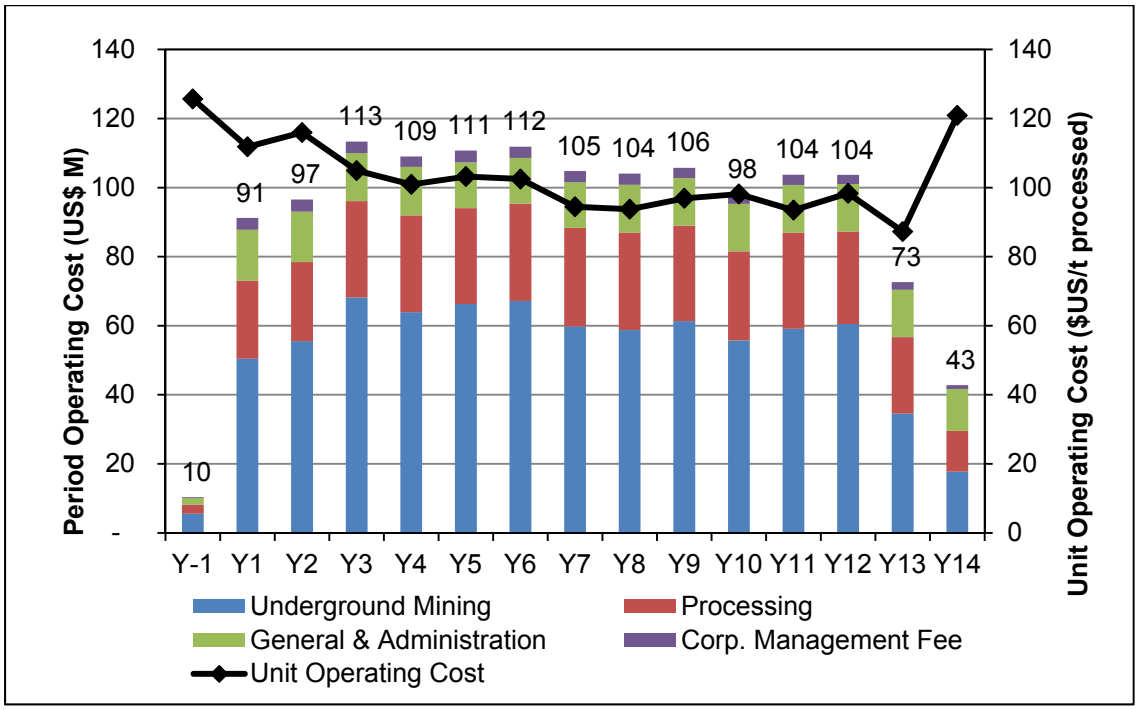
Table 1.11: Operating Cost Summary

Sector	Average US$ M/year	Life of Mine $US M	$/t processed
Underground Mining	56.3	784.7	57.21
Processing	25.8	358.8	26.16
General & Administration	13.9	193.8	14.13
Corporate Management Fee	3.0	41.3	3.01
Total Mine Operating Costs	**99.0**	**1,378.6**	**100.50**

Source: JDS, 2016

All operating costs are included in the economic cash flow model according to the production schedule.

Figure 1.7: Life of Mine Operating Cost Profile



Source: JDS, 2016

1.16 Economic Analysis

Based on the FS findings, it can be concluded that the Project is economically viable at estimated metal prices with an after-tax IRR of 31.2% and an NPV of $860M at a 5% discount rate.

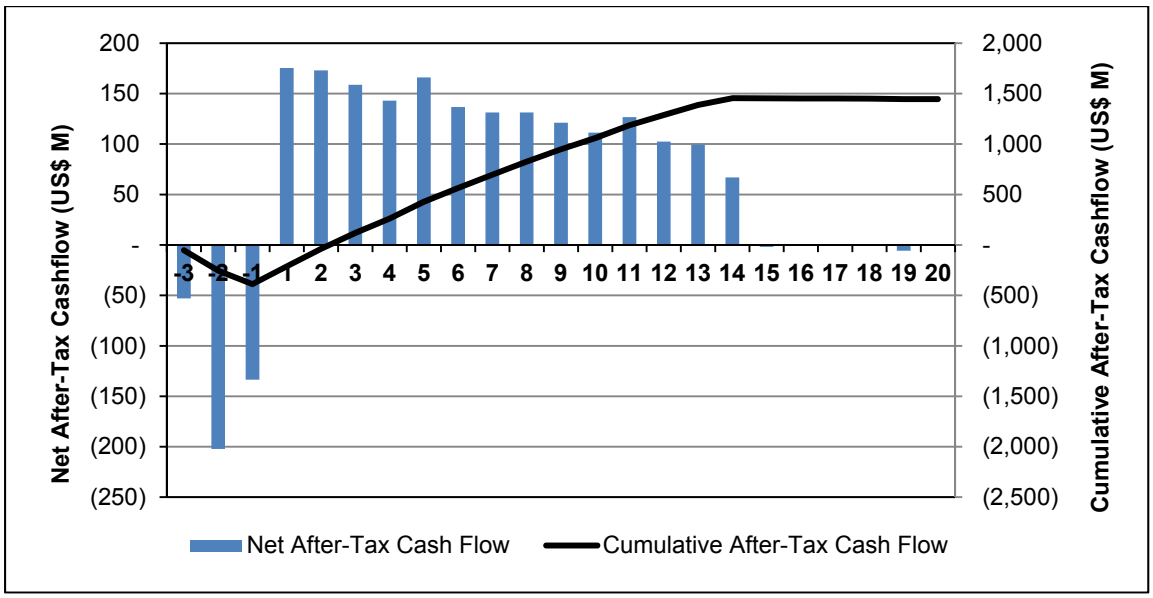
Table 1.12: Summary of Economic Results

Parameter	Unit	Value
Gold Price	US$/ounce	1,200
Silver Price	US$/ounce	15.00
Exchange Rate	COP:US	2,850
Pre-Tax NPV5%	US$ M	1,263.3
Pre-Tax IRR	%	38.0
Pre-Tax Payback	years	1.9
Total Taxes	US$ M	636.2
Effective Tax Rate	%	33
After-Tax NPV5%	US$ M	860.2
After-Tax IRR	%	31.2
After-Tax Payback	years	2.3
Break-Even After-Tax Gold Price	US$/ounce	634

Payback is calculated on annual cash flows without considering discount rates or inflation.

Source: JDS, 2016

Figure 1.8: Annual and Cumulative After-Tax Cash Flows



Source: JDS, 2016

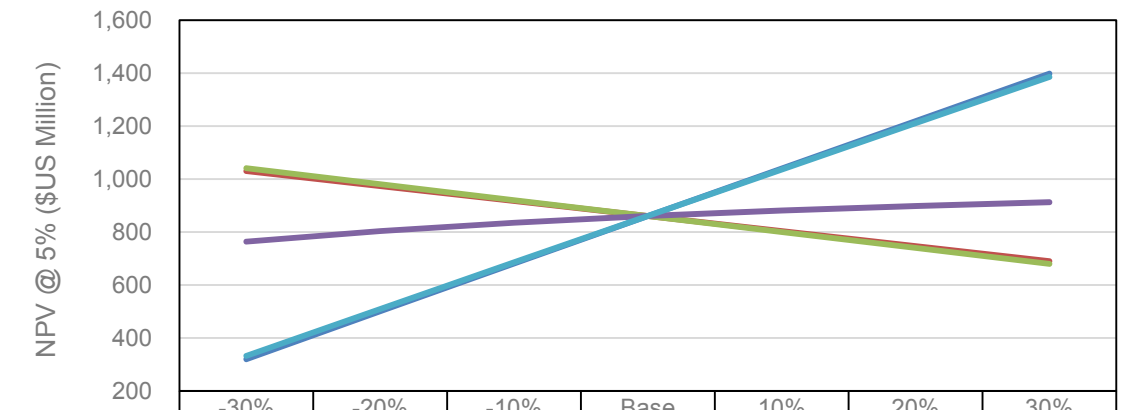
Table 1.13: Cash Cost Summary

Item	Parameter	LOM Cost (US$M)	Unit Cost (US$/Payable Au Oz)
A	Pre-Production Capital Costs	389.2	112
B	Life of Mine Sustaining Capital Costs	272.5	78
C	Closure Cost (net of Salvage Value)	10.0	3
D	Operating Costs	1,378.6	395
E	Refining and Transportation	14.9	4
F	Royalties	137.2	39
G	Silver Credits	(96.1)	(28)
	Total Cash Cost, net Ag Credits (D+E+F+G)	1,434.6	411
	All-in Sustaining Cost, net Ag Credits (B+C+D+E+F+G)	1,717.1	492
	All-in Sustaining & Construction Costs, net Ag Credits (A+B+C+D+E+F+G)	2,106.3	604

Source: JDS, 2016

To determine Project value drivers, a sensitivity analysis was performed on the Net Present Value (NPV) and Internal Rate of Return (IRR). The Project proved to be most sensitive to changes in the metal pricing and gold grades, and least sensitive to changes in exchange rate.

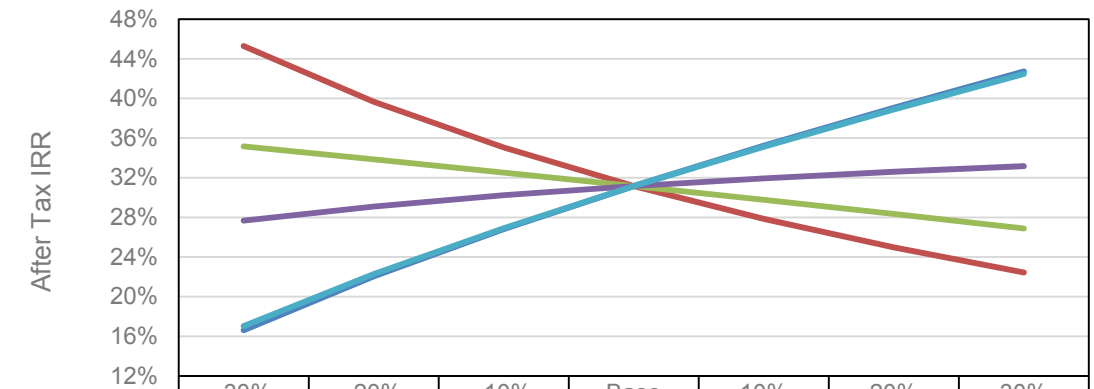
Figure 1.9: After-Tax NPV5% Sensitivities



	-30%	-20%	-10%	Base	10%	20%	30%
Price	320	501	681	860	1,040	1,219	1,398
CAPEX	1,030	974	917	860	804	747	690
OPEX	1,041	981	920	860	800	740	680
FX	764	804	835	860	881	898	912
Au Grade	333	510	685	860	1,035	1,211	1,386

Source: JDS, 2016

Figure 1.10: After-Tax IRR Sensitivities



	-30%	-20%	-10%	Base	10%	20%	30%
Price	16.6%	22.0%	26.8%	31.2%	35.2%	39.1%	42.7%
CAPEX	45.3%	39.7%	35.0%	31.2%	27.8%	25.0%	22.4%
OPEX	35.2%	33.9%	32.5%	31.2%	29.8%	28.3%	26.9%
FX	27.7%	29.1%	30.2%	31.2%	31.9%	32.6%	33.2%
Au Grade	17.0%	22.3%	26.9%	31.2%	35.1%	38.9%	42.5%

Source: JDS, 2016

1.17 Conclusions

Results of this Feasibility Study demonstrate that the Buriticá Project warrants development due to its positive, robust economics.

It is the conclusion of the QPs that the FS summarized in this technical report contains adequate detail and information to support a Feasibility level analysis. Standard industry practices, equipment and design methods were used in this Feasibility Study and except for those outlined in this section, the report authors are unaware of any unusual or significant risks, or uncertainties that would affect Project reliability or confidence based on the data and information made available.

For these reasons, the path going forward must continue to focus on obtaining the environmental permit modification approval, while concurrently advancing key activities that will reduce project execution time.

Risk is present in any mineral development project. Feasibility engineering formulates design and engineering solutions to reduce that risk common to every project such as resource uncertainty, mining recovery and dilution control, metallurgical recoveries, political risks, schedule and cost overruns, and labour sourcing.

Potential risks associated with the Buriticá Project include:

- **Environmental Permit Modification –** The environmental permit modification, not yet approved, is of paramount importance, and further delays will increase project execution time. Without the permit modification approval, the Project cannot proceed and failure to secure the necessary permits could stop or delay the Project. In place is a project design that gives appropriate consideration to the environment and local communities.
- **Reserve** - A potential unknown identified by the QP is the extent of unauthorized and illegal mining activities in the Yaraguá and Veta Sur areas. Unauthorized mining activities have been discovered, on occasion, adjacent to Continental's underground workings in the Yaraguá and Veta Sur deposits. In late July 2015, CGI surveyed the extent of unauthorized mine workings in areas that were accessible. These areas are excluded from the Mineral Reserves; however, the extent of impact in the upper areas of the reserve may not be fully quantified.
- **Management of Unauthorized Mining Activities** – The unauthorized mining activity near the Project has the potential to disrupt and delay construction activities, and to deplete the resource. The government of Colombia has the responsibility and has committed to address and manage this situation including actions to remove these illegal activities.
- **Groundwater** - Groundwater inflows below the Higabra level have been modeled and the mine development plan addresses estimated flows as well as potential variations. Increases in the actual amount of groundwater encountered would impact development costs. Drilling for drainage, and operational definition drilling included in the mine plan will help to identify specific water bearing zones with higher than expected flows to establish control and/or management procedures. As well, initiating certain development earlier in the mine life to allow more time for dewatering may prove cost effective.
- **Stability of natural slopes** –The steep topography, high rainfall, and seismic hazard level in the Higabra Valley suggests the potential for mass movements and remnants of past events are evident. The access road, power lines, and facilities located in the valley bottom would be at risk. CGI has undertaken slope deposit mapping (based partly on ultra-high resolution Light Detection and Ranging (LiDAR), topography and surface morphologies) and geotechnical studies throughout all areas of current planned infrastructure in the Buriticá Project and has concluded that risks from major mass movements are acceptable.
- **Comminution** – The wall rock is much harder than the vein material and it appears that increased waste rock in the metallurgical samples increases the comminution parameters. Grade control and proper mining execution when implemented will maintain minimal unplanned dilution, which would minimize potential impacts on grade, throughput, and operating costs. Continued test stoping at Yaraguá mine will help to verify dilution estimates.
- **Plant Feed Blend** - Determination of the amount of plant feed that contains increased levels of arsenic needs to be refined, particularly from Veta Sur where it indicates a decrease in the gold recovery. It is recommended that additional study applying the variability data to date and geometallurgical models be used to optimize mine production scheduling and blending techniques to more fully understand and possibly improve gold recovery.
- **Geomechanical Conditions –** Comprehensive studies were done to accurately estimate anticipated ground conditions. There is a risk that a larger percentage of the ore must be

extracted using cut and fill (C&F) rather than the longhole method resulting in higher costs. If this situation were to occur, the sensitivity analyses show that Buriticá project economics continue to justify mine and mill development.

The FS has highlighted several opportunities to increase mine profitability and project economics, and reduce identified risks.

- **Inferred Resources** – Inferred resources are not included in the production schedule; however, a plan to infill drill specific areas could increase Measured and Indicated resource, especially in the footwall and hanging wall, and for the zone between Yaraguá main and Yaraguá deep. This resource increase could significantly improve Project economics. Operational definition drilling will test inferred resources as part of the production sequence. Identification of additional resources would have a compounding positive effect in that the development per ore tonne as well as vertical mining advance rate would be decreased. Additionally, mine plans for specific areas, such as those below the Higabra level would change dramatically with the addition of resource, by allowing more methodical development below the Higabra and allocating development and water handling costs over a larger reserve base.
- **Mine Grade Strategy** – Cut-off grade trade-off evaluations indicated that a COG higher than the 3.8 g/t for Yaraguá and 4.0 g/t for Veta Sur could provide equal or better project economics. Comprehensive mine design and scheduling would be required to validate any potential benefits, and to determine if an alternative plan with higher early year cut-off grades (COG) would provide increase upfront cash flow and decrease risk without diminishing the mineable resource. The mining strategy is flexible and depending on market conditions at the time of commissioning, opportunity exists to adjust the mine plan.
- **Increased Gold Recovery** - Potential exists to increase gravity gold recovery using alternative methods such as intensively leaching gravity concentrate and/or regrinding gravity concentrate.
- **Grind Size** - There exist potential economic benefits of improving recovery using a finer primary grind size.
- **Flotation of a bulk concentrate followed by regrind and cyanidation** - The opportunities with this approach trade additional complexity for a coarser primary grind and indications of initial test work for overall gold and silver recovery improvements, and potential benefits of two tailing streams – a non-sulphide un-leached coarse tailing and a reduced tonnage of finer sulphide tailing, decreased plant footprint and capital requirements.
- **Ultra-Filtration Water Treatment** – Capital and operating costs are relatively high due to the large water volumes treated, and the process produces a brine waste product, which is crystalized to a solid in the FS operations plan. Alternatives for disposing of the brine such as a liquid in the paste backfill, or investigating methods to reduce the brine volume, could significantly reduce costs.
- **Alternative TSF construction material** – Stability considerations require that a binder be used to increase TSF stability due to a limited supply of development waste rock. Alternative material for buttress and cover could be sourced from an optimized TSF foundation excavation design. Engineering to determine the best excavated material balance compared to binder addition may result in significant cost savings.

- **Construction Costs** - Civil construction capital cost may be reduced by additional engineering to better balance cut and fill of the plant site and water management infrastructure.
- **Concrete Reduction** - Opportunity exists to reduce concrete quantities with more geotechnical investigation during detailed engineering.
- **Powerline Line Finance Cost** - The 110 kV powerline supply and installation cost are financed in the economic model at 15%. Opportunities to reduce this cost impact to project economics need to be investigated.

1.18 Recommendations

Due to the positive, robust economics, it is recommended to expediently advance the Buriticá Project to construction and development, and then production. The recommended development path is to continue efforts to obtain the environmental permit modification approval while concurrently advancing key activities that will reduce project execution time. Associated project risks are manageable, and identified opportunities can provide enhanced economic value.

The Project exhibits robust economics with the assumed gold price, COP exchange rate, and consumables pricing. The risks are acceptable as well. Value engineering and recommended fieldwork should be advanced in preparation of permit approval in order to de-risk the construction schedule and minimize costs.

From project risks and opportunities, the following were identified as critical actions that have the potential to strengthen the Project and further reduce risk and should be pursued as part of the project development plan.

Regarding the Section 1.17 opportunities, recommendations are as follows:

- **Inferred Resources** – Inferred resources are not included in the production schedule; a plan should be developed to infill drill specific areas that could potentially provide significant improvement to Project economics.
- **Mine Grade Strategy** – A trade-off study evaluating project economics for various mine cut-off grades should be advanced. The study requires mine design and production schedules using different cut-off grades in order to compare results and subsequent recommendations.
- **Increased Gold Recovery** – Comparative economic trade-off evaluations should be performed to determine if there is potential to increase gravity gold recovery using alternative methods such as intensively leaching gravity concentrate and/or grinding gravity concentrate to improve recovery.
- **Grind Size** – Additional studies are warranted to evaluate the potential economic benefits of improving recovery using a finer primary grind size.
- **Flotation of a Bulk Concentrate followed by Regrind and Cyanidation** - A high level trade-off study comparing Flotation to Whole Ore Leach (WOL) was completed based on historic metallurgical test work. The results demonstrated economic, and operational advantages of WOL. It is recommended that these results be verified with current test results.

- **Ultra-Filtration Water Treatment** – Due to the relatively high capital and operating costs for water treatment, a study is warranted to determine alternative methods to dispose of the brine waste product, or reduce the brine volume.
- **Alternative TSF construction material** – The benefits to using excavated alluvial material to reduce binder requirements while maintaining required stability performance should be assessed. The assessment would need to include evaluating TSF foundation excavation design. As part of this work, modeling the deposit by alteration type to better plan production and disposal of Potentially Acid Generating/Non-Acid Generating (PAG/NAG) waste rock types is also recommended.
- **Construction Costs -** Civil construction cost may be reduced by additional engineering to better balance cut and fill of the plant site and water management infrastructure perhaps reducing the CAPEX of plant earthworks. This opportunity should be targeted in the detailed engineering phase.
- **Concrete Reduction** – Existing geotechnical site characterization needs to be assessed, and then a program planned to obtain required information to design foundations for major mill equipment and TSF.
- **Powerline Line Finance Cost** – A comprehensive study to develop powerline installation and power supply finance alternatives is needed to fully assess potential reductions to capital and operating costs.

2 Introduction

2.1 Basis of Feasibility Study

This Feasibility Study was compiled by JDS for CGI. This report summarizes the results of the Feasibility Study and was prepared in accordance with the Canadian Securities Administrators' National Instrument 43-101 and Form 43-101F1.

2.2 Scope of Work

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

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

- FS project management;
- Compilation of the report, including data and information provided by other consulting companies;
- Project setting, history and geology description;
- Mine planning;
- Metallurgical testwork program;
- Environmental permitting application preparation;
- CAPEX and OPEX estimation;
- Preparation of a financial model to enable economic evaluation; and
- Interpretation of results and recommendations to improve value and reduce risks.

Mining Associated Ltd. (MA) scope of work included:

- Mineral Resource Estimate.

M3 Engineering & Technology Corp. (M3) scope of work included:

- Recovery methods;
- Site layout and select infrastructure design; and
- Plant site water management infrastructure design.

Robertson GeoConsultants Inc. (RGC) scope of work included:

- Tailing Storage Facility (TSF) design; and
- Select water management structure design for TSF location.

SRK Consulting Inc. (SRK) scope of work included:

- Geotechnical assessment of the Buriticá underground mine workings.

Minefill Services Inc. (Minefill) scope of work included:

- Paste backfill plant design.

Schlumberger Water Services (SWS) scope of work included:

- Geochemistry of mine water and TSF contact water.

Membrane Development Specialists:

- Water treatment plant design.

2.3 Qualified Person Responsibilities and Site Inspections

The Qualified Persons (QPs) preparing this report are specialists in the fields of geology, exploration, mineral resource and Mineral Reserve estimation and classification, geotechnical, environmental, permitting, metallurgical testing, mineral processing, processing design, capital and operating cost estimation, and mineral economics.

None of the QPs or any associates employed in the preparation of this report has any beneficial interest in CGI and neither are they insiders, associates, or affiliates. The results of this report are not dependent upon any prior agreements concerning the conclusions to be reached, nor are there any undisclosed understandings concerning any future business dealings between CGI and the 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. The QPs are responsible for specific sections as follows:

Table 2.1: QP Responsibilities

QP Name	Company	QP Responsibility/Role	Report Section(s)
Wayne Corso, P.E.	JDS Energy & Mining Inc.	Executive Summary, Introduction, Reliance on Other Experts, Property Description, Accessibility and Physiography, History, Infrastructure, Market Studies, Capital and Operating Cost Estimates, Economic Analysis, Adjacent Properties, Other Relevant Data and Information, Interpretations and Conclusions, Recommendations, References, Units of Measurement and Abbreviations	1; 2; 3; 4; 5; 6; 18;19, 21; 22; 23; 24; 25; 26; 27; 28; 29
Greg Blaylock, P.Eng.	JDS Energy & Mining Inc.	Mineral Reserve Estimate, Mining Methods, Infrastructure, Mining Capital and Operating Cost Estimate	15; 16; 21.3; 22.3
Austin Hitchins, P.Geo	JDS Energy & Mining Inc.	Geologic Setting and Mineralization, Deposit Types, Exploration and Drilling	7; 8; 9; 10
Stacy Freudigmann, P.Eng.	JDS Energy & Mining Inc.	Mineral Processing and Metallurgical Testing	13
Mike Creek, P.E.	JDS Energy & Mining Inc.	Environmental Studies and Permitting	20
Jack Caldwell, P.E.	Robertson Geoconsultants Ltd.	Tailing Management Facility	18.6
Andrew J. Vigar, FAusIMM	Mining Associates Ltd.	Sample Preparation, Analysis and Security, Data Verification, Mineral Resource Estimate	11;12;14
Mike Levy, P.E.	SRK Consulting Inc.	Geotechnical Analysis/Mine Geomechanics	16.3
David Stone	MineFill Services Inc.	Backfill Plant	
Laurie Tahija, MMSA	M3 Engineering and Technology Corp.	Recovery Methods	17

Source: JDS, 2016

QP visits to the Buriticá property were conducted as follows:

- Wayne Corso visited the site April 8-9, 2015;
- Greg Blaylock visited site April 8-9, 2015, in May 2015, & October 2015;
- Austin Hitchins visited site April 8-9, 2015;

- Mike Creek visited site October 1-2, 2015;
- Jack Caldwell visited site in July, 2015;
- David stone visited the site on June 10, 2016;
- Andrew J. Vigar visited the site February 24-27, 2015;
- Michael Levy visited the site in April 8-9 and October 23–24, 2015;
- Laurie Tahija visited the site on February 13, 2012;
- Stacey Freudigmann did not visit the site.

2.4 Sources of Information

The sources of information include data and reports supplied by CGI personnel as well as documents cited throughout the report and referenced in Section 28. In particular, background Project information was also sourced from the most recent technical report published by MA in August 2015.

2.5 Units, Currency and Rounding

Unless otherwise specified or noted, the units used in this technical report are metric. Every effort has been made to clearly display the appropriate units being used throughout this technical report. Currency is in US dollars (US$ or $).

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

3 Reliance on Other Experts

The QP's opinions contained herein are based on information provided by CGI and others throughout the course of the study. The QPs have taken reasonable measures to verify information provided by others and take responsibility for the information.

Non-QP specialists relied upon for specific advice are:

- Richard Boehnke, P. Eng, JDS Energy & Mining Ltd.;
- Allen Anderson, P.E., President Allen R. Anderson Metallurgical Engineer Inc.;
- Chris Copley, JDS Energy & Mining Ltd.;
- Michael Rosko, P.Geo. Montgomery & Associates Consultants Ltd.;
- Larry Lien, Membrane Development Specialists;
- Keith Dagel P.E., Project Manager, M3 Engineering and Technology;
- David Stone P.E., Minefill Services Inc.;
- Martin Williams BSc. PhD, Schlumberger Water Services; and
- Tobias Roetting BSc. PhD, Schlumberger Water Services.

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.

4 Property Description and Location

4.1 Property Area

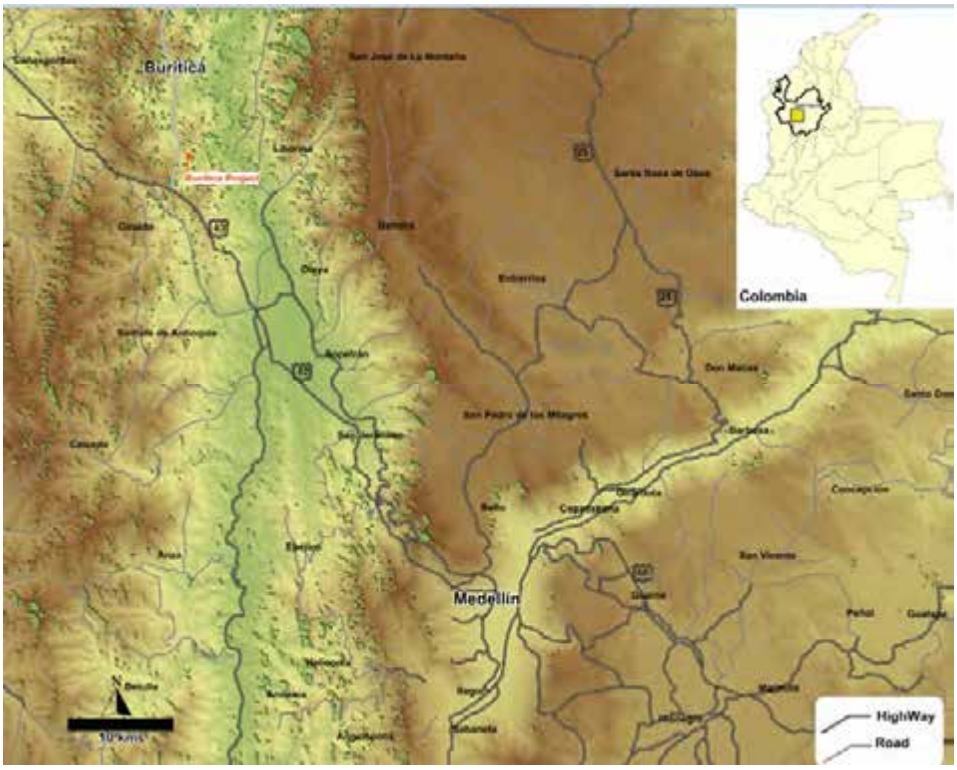
The Buriticá Project comprises 64 tenements and applications with technical studies covering an area of approximately 61,784 ha. CGI has 100% ownership (directly or by means of assignment agreements) over the tenements and applications. Granting of these applications is pending and subject to approval from the mining authority.

Both the Yaraguá and Veta Sur mineral systems lie within the tenement known as License 7495. License 7495 is directly owned by CGI and has commodity rights to precious metals, copper minerals, lead minerals, zinc minerals and concentrates. CGI has filed for an addition of minerals to License 7495, comprising aggregate and construction materials. This request is currently under review by the mining authority.

4.2 Property Location

The Buriticá Project (centred on 400538 E, 746204 N, Zone 18 N, and at elevations from 600 to 2,200 m) is located approximately 72 km northwest of the major city of Medellín in the Antioquia Department of north-western Colombia. The Buriticá Project is located approximately 2 km south of the town of Buriticá (Figure 4.1). The area is accessible by a paved road from Medellín.

Figure 4.1: Project Location and Local Access

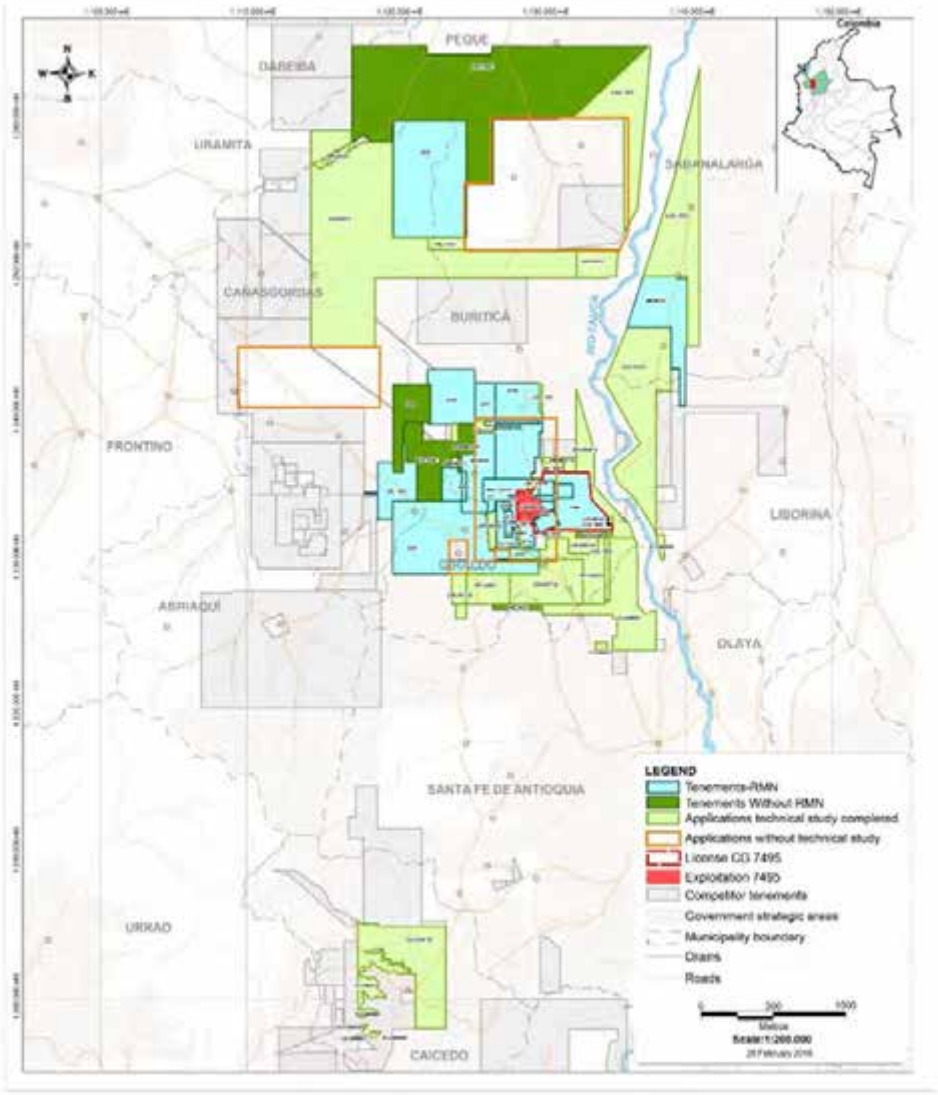


Source: CGI, 2015

4.3 Property Tenure

Figure 4.2 shows the location of the Buriticá Project tenements and applications, License 7495 and competitor tenements and government strategic areas. The Buriticá Project includes the Yaraguá mine that had previously been operated as a small-scale mine by predecessors of CGI. It is now utilized for underground exploration development and a bulk sample testing operation. None of the competitor tenements or the government strategic areas have negative impact on CGI´s tenements and applications.

Figure 4.2: Project Tenements



Source: CGI, 2016

Table 4.1: Summary of CGI Tenements Details and Status

#	Code	Type of Tenement	Date Granted	Expiration Date	Company*	Area (ha)	% Owned by CGI	Minerals Covered
1	3638	Exploration-RMN Decree 2655, 1988	21/12/07	06/04/2013 Note 3	La Peña SOM	4,000.00	100%	Au, Ag & other permissible minerals, in vein and alluvium
2	4246	Exploration Decree 2655, 1988	03/03/15	Note 1	Colombian Development Corporation SAS	904.30	100%	Au, Ag, Cb & other minerals
3	5486	Exploration-RMN Decree 2655, 1988	06/12/11	30/06/19	Majayura SOM	3,250.80	100%	Au, Ag, Cb & other minerals
4	6230	Concession contract-RMN Law 685, 2001	31/12/08	16/2/39	Continental Gold Limited Sucursal Colombia	1,151.23	100%	Au & other concessions
5	6366	Concession contract-RMN Law 685, 2001	25/05/06	13/07/39	Continental Gold Limited Sucursal Colombia	60.70	100%	Precious Metals & Cu
6	6367	Concession contract-RMN Law 685, 2001	25/05/06	12/6/36	Continental Gold Limited Sucursal Colombia	17.95	100%	Precious Metals & Cu
7	6747	Concession contract-RMN Law 685, 2001	05/06/08	28/10/38	Continental Gold Limited Sucursal Colombia	243.24	100%	Au, Ag, Cu, Pb, Zn & other minerals
8	6748	Concession contract-RMN Law 685, 2001	22/05/08	13/7/38	CGL Gran Buriticá SAS	1,862.66	100%	Au, Ag, Cu, Zn & other minerals
9	6977	Concession contract-RMN Law 685, 2001	14/12/07	13/2/38	Continental Gold Limited Sucursal Colombia	347.20	100%	Au, Ag & other minerals
10	6992	Concession contract-RMN Law 685, 2001	28/12/07	21/9/41	Continental Gold Limited Sucursal Colombia	15.75	100%	Au, Ag & other minerals
11	7495	Concession contract Exploration and Exploitation-RMN Law 685, 2001	05/02/13	19/3/43	Continental Gold Limited Sucursal Colombia	1,893.90	100%	Precious Metals & concentrates, Cu & concentrates, Pb & concentrates, Zn & concentrates
12	8133	Concession contract exploitation-RMN Law 685, 2001	10/02/15	30/09/45	Continental Gold Limited Sucursal Colombia	150.09	100%	Precious Metals
13	12713	Exploitation-RMN Decree 2655, 1988	29/12/95	07/11/2014 Note 2	Continental Gold Limited Sucursal Colombia	90.00	100%	Precious Metals in vein and alluvium
14	5486B	Exploration-RMN Decree 2655, 1988	27/08/13	21/05/16	Majayura SOM	27.11	100%	Au, Ag, Cb & other minerals
15	AH5-15431X	Exploration-RMN Decree 2655, 1988	17/02/15	13/05/17	Encenillos SOM	414.76	100%	Au, Ag & other minerals
16	ALN-09371X	Exploration Decree 2655, 1988	19/06/15	Note 1	Colombian Development Corporation SAS	260.59	100%	Au, Ag, Cb & other minerals
17	IG5-10031	Concession contract-RMN Law 685, 2001	16/05/12	5/7/42	CGL Gran Buriticá SAS	325.73	100%	Au & concentrates
18	IHD-11081	Concession contract-RMN Law 685, 2001	19/12/11	20/3/42	CGL Gran Buriticá SAS	45.98	100%	Precious Metals, Zn, Pb, Cu, Mo, & concentrates
19	IJN-14011	Concession contract-RMN Law 685, 2001	09/12/09	11/7/43	CGL Gran Buriticá SAS	1,214.76	100%	Precious Metals, Cu, Zn, Pb, Mo, Pb & concentrates
20	IJN-14281	Concession contract-RMN Law 685, 2001	10/12/09	9/5/41	Continental Gold Limited Sucursal Colombia	840.26	100%	Au, Ag, Cu, Zn, Pt, Mo, Pb & concentrates
21	IJN-14321	Concession contract Law 685, 2001	09/12/09	Note 1	Anglogold Ashanti Colombia S.A	137.62	100%	Au, Ag, Cu, Zn, Pt, Mo & Pb
22	JDO-08592X	Concession contract Law 685, 2001	30/08/11	Note 1	Antioquia SOM	107.50	100%	Precious Metals & concentrates, natural & siliceous sand & gravel, Ti minerals
23	JI8-08231	Concession contract-RMN Law 685, 2001	06/08/12	18/10/42	CGL Gran Buriticá SAS	1,622.10	100%	Au, Pt & concentrates
24	KAQ-10431	Concession contract Law 685, 2001	19/12/11	Note 1	Encenillos SOM	9,821.39	100%	Au, Pt & concentrates
25	KJG-14581	Concession contract Law 685, 2001	19/12/11	Note 1	Frontera SOM	1,144.53	100%	Au, Pt & concentrates

Note 1: Concession contracts have been signed by both Beneficiary and Mining Authority and are in the process of being registered with the Mining Registry or Registro Minero Nacional
Note 2: Exploitation license expired. Request for conversion into concession contract in process
Note 3: Exploration license expired. Evaluating request for exploration extension
Note 4: Tenements and applications under third parties different from CGI, or related companies, are under CGI´s ownership based on assignment agreements pending approval and/or registry from the mining authority.
Source: CGI, 2016

Table 4.2: Summary of CGI Free Area Technical Studies Completed

#	Code	Status	Company	Area (ha)	% Owned By CGI	Minerals Covered
1	IJN-14301	Free area technical study completed	Anglogold Ashanti Colombia S.A	99.09	100%	Au, Ag, Cu, Zn, Pt, Mo & Pb
2	IJN-14302X	Free area technical study completed	Anglogold Ashanti Colombia S.A	32.58	100%	Au, Ag, Cu, Zn, Pt, Mo & Pb
3	JDO-08591X	Free area technical study completed	Antioquia SOM	817.69	100%	Precious Metals & concentrates, natural & siliceous sand & gravel, Ti minerals
4	JDO-08593X	Free area technical study completed	Antioquia SOM	3,456.36	100%	Precious Metals & concentrates, natural & siliceous sand & gravel, Ti minerals
5	JHM-11221	Free area technical study completed	Continental Gold Limited Sucursal Colombia	6.00	100%	Au, Pt & concentrates & other minerals concessions
6	JHR-08071	Free area technical study completed	Costa SOM	31.30	100%	Au, Pt & concentrates
7	JHR-08073X	Free area technical study completed	Costa SOM	203.25	100%	Au, Pt & concentrates
8	JHR-08074X	Free area technical study completed	Costa SOM	268.72	100%	Au, Pt & concentrates
9	JHR-08075X	Free area technical study completed	Costa SOM	24.34	100%	Au, Pt & concentrates
10	JHR-08076X	Free area technical study completed	Costa SOM	143.45	100%	Au, Pt & concentrates
11	JHR-08077X	Free area technical study completed	Costa SOM	137.17	100%	Au, Pt & concentrates
12	JJO-08041	Free area technical study completed	Continental Gold Limited Sucursal Colombia	32.10	100%	Au, Pt & concentrates
13	KAQ-10271	Free area technical study completed	Encenillos SOM	9,891.35	100%	Au, Pt & concentrates & other minerals concessions
14	KAQ-10331	Free area technical study completed	Encenillos SOM	5,546.11	100%	Au, Pt & concentrates
15	KCK-15021	Free area technical study completed	Continental Gold Limited Sucursal Colombia	52.47	100%	Au, Pt & concentrates & other minerals concessions
16	KFC-08031	Free area technical study completed	Escorpion SOM	681.96	100%	Au, Pt & concentrates
17	KFC-08035X	Free area technical study completed	Escorpion SOM	819.86	100%	Au, Pt & concentrates
18	KJ2-08061	Free area technical study completed	Frontera SOM	2,068.07	100%	Au, Pt & concentrates
19	KJ2-08062X	Free area technical study completed	Frontera SOM	40.29	100%	Au, Pt & concentrates
20	KJ2-08064X	Free area technical study completed	Frontera SOM	22.94	100%	Au, Pt & concentrates
21	KJ2-08065X	Free area technical study completed	Frontera SOM	7.23	100%	Au, Pt & concentrates
22	KJ2-08066X	Free area technical study completed	Frontera SOM	65.43	100%	Au, Pt & concentrates
23	KJ2-08067X	Free area technical study completed	Frontera SOM	221.16	100%	Au, Pt & concentrates
24	KJ2-08068X	Free area technical study completed	Frontera SOM	1.82	100%	Au, Pt & concentrates
25	KJ2-08069X	Free area technical study completed	Frontera SOM	73.85	100%	Au, Pt & concentrates
26	KJ2-080610X	Free area technical study completed	Frontera SOM	2,591.69	100%	Au, Pt & concentrates
27	LC9-10481	Free area technical study completed	Continental Gold Limited Sucursal Colombia	3.57	100%	Au, Pt & concentrates
28	LC9-11001	Free area technical study completed	Continental Gold Limited Sucursal Colombia	38.83	100%	Au, Pt & concentrates & other minerals concessions
29	LCP-08025	Free area technical study completed	Continental Gold Limited Sucursal Colombia	15.70	100%	Au, Pt & concentrates
30	OG2-08099	Free area technical study completed	Continental Gold Limited Sucursal Colombia	148.46	100%	Precious Metals & concentrates
31	OG2-081718	Free area technical study completed	Continental Gold Limited Sucursal Colombia	1,456.41	100%	Precious Metals & concentrates
32	OGF-09171	Free area technical study completed	Continental Gold Limited Sucursal Colombia	0.00	100%	Precious Metals & concentrates
33	PEE-16031	Free area technical study completed	Continental Gold Limited Sucursal Colombia	4.40	100%	Precious Metals & concentrates
34	PEF-08231	Free area technical study completed	Continental Gold Limited Sucursal Colombia	120.11	100%	Au, Pt & & concentrates
35	PEL-08021	Free area technical study completed	Continental Gold Limited Sucursal Colombia	0.00	100%	Precious Metals & concentrates
36	PEL-14101	Free area technical study completed	Continental Gold Limited Sucursal Colombia	210.28	100%	Precious Metals & concentrates

Note 1: Free Area Technical Study: No fees apply for initial reconnaissance exploration prior to formal exploration. There is no certainty that the area will be granted.

Note 2: Tenements and applications under third parties different from CGI, or related companies, are under CGI´s ownership based on assignment agreements pending approval and/or registry from the mining authority.
Source: CGI, 2016

Table 4.3: Summary of CGI Applications

#	Code	Status	Company	Area (ha)	% Owned By CGI	Minerals Covered
1	PEN-08011	Application without free area study	Continental Gold Limited Sucursal Colombia	0.00	100%	Au, Pt & concentrates
2	QBH-16351	Application without free area study	Continental Gold Limited Sucursal Colombia	0.00	100%	Au, Pt & concentrates
3	QE6-08002	Application without free area study	Continental Gold Limited Sucursal Colombia	2,500.00	100%	Au, Pt & concentrates

Source: CGI, 2016

Tenement information has been supplied by CGI. JDS has not undertaken any title search or due diligence on the tenement titles or tenement conditions and the tenement's status has not been independently verified by JDS.

4.4 Property Ownership, Rights and Obligations

All CGI's tenements and applications have rights to gold. Other minerals are not covered by the licenses unless the mineral is linked, associated, obtained as a sub-product of the exploitation of gold, or in the event the company files for an addition of minerals (as occurred in License 7495). Thus, if a mineral is linked, associated, obtained as a sub-product of gold, or is granted per an addition request, CGI also has rights to that mineral.

Colombian mining law considers minerals to be considered a sub-product of exploitation when they are extracted together with the mineral of the contract, but their quality and amount would not be economically exploitable in a separate form, and can only be separated by physical or chemical beneficiation processes. The law recognizes that associated minerals form an integral part of the mineralized body, which is the object of the concession contract.

4.4.1 General

Exploration and mining in Colombia is governed by Mining Law 685 of 2001 and its regulatory Decrees (the "2001 Code"). Colombia has several authorities and entities which enforce exploration and mining law:

- Ministry of Mines and Energy (Ministerio de Minas y Energía, MME);
- The Agencia Nacional de Minería (the National Mining Agency) is responsible for the administration of Mineral Resources except where the responsibility is delegated to a different authority, as it occurs in the Antioquia Department which has a Mining Delegation;
- The Antioquia Department Mining Delegation administers mining concessions in Antioquia, which is a Department with significant mining activity;
- Mining Energy Planning Unit (Unidad de Planeación Minero Energética), which provides support to the MME and maintains the System of Colombian Mining Information (Sistema de Información Minero Colombiano) as well as information regarding government royalties; and
- Servicio Geológico Colombiano, a technical government entity in charge of scientific investigation of non-renewable natural resources.

All Mineral Resources are the property of the state and under the 2001 Code, there is only one type of concession that allows exploration, construction and exploitation. This type of concession is valid for 30 years and can be extended for an additional 30 years. The 2001 Code allows for the continued existence of mining titles acquired under previous legislation. These licenses and permits have been grandfathered in, and are still governed by the terms and conditions of the previous legislation.

The location of a concession is given by a reference point with distances and bearing, or by map coordinates.

A surface tax (canon superficial) is due annually upon contract registration with the Mining Registry during the exploration and construction phases of the concession. It is calculated per hectare as multiples of the minimum daily wage (MDW), which is adjusted annually (for 2016, COP22,982 or approximately US$6.96).

For mining concession contracts executed and registered before the enactment of the National Development Plan Law 1753, 2015 (Plan Nacional de Desarrollo (PND)), the tax is equivalent to the MDW per hectare per year for areas up to 2,000 ha, two times the MDW per hectare per year for areas of 2,000 to 5,000 ha, and three times the MDW per hectare per year for areas of 5,000 to 10,000 ha.

For mining concession contracts executed and registered after the enactment of the National Development Plan, the tax is paid as shown in Table 4.4.

Table 4.4: Tax Payment Scheme for Concession Contracts Registered after the PND

Number of Hectares	0 to 5 years	More Than 5 Years to 8 Years	More Than 8 Years to 11 Years
	MDW/ha	MDW/ha	MDW/ha
0-150	0.5	0.75	1
151-5.000	0.75	1.25	2
5,001-10,000	1	1.75	3

Source: CGI from Law 1753/2015

The concession contract has three phases and commences upon its inscription in the National Mining Registry. The three phases are described in Table 4.5.

In special cases, some of the tenements are governed by regulations prior to the enactment of the 2001 Code. Under Decree 2655, 1988, tenements were issued in the form of exploration licenses, exploitation licenses, concession agreements and public entity granting agreements. Under said Decree, CGI´s tenements are in the exploration or exploitation phase.

Under an exploration license, the title holder is allowed to explore the area for the purpose of determining the existence of Mineral Reserves for a term of one to five years, depending on the area to be explored. Upon expiry of the exploration license, the title holder has a right to file for an exploitation license. In the case of precious minerals, exploitation licenses were granted for the following open pit mining operations: small-scale (not exceeding 250,000 m^3), medium-scale (between 250,000 to 1,500,000 m^3), and large scale (over 1,500,000 m^3) of extraction per year per

exploitation license. Exploitation licenses were also granted for the following underground mining operations: small-scale (not exceeding 8,000 tonnes (t)), medium-scale (between 8,000 to 200,000 t) and large scale (over 200,000 t) of extraction per year per license.

Under an exploitation license, a title holder is allowed to exploit minerals for an initial term of ten years, which can be extended for additional 10-year periods, or converted into a concession agreement under the 2001 Code.

The primary obligations to be complied with to maintain a tenement in good standing are outlined in Table 4.5.

In 2014, CGI executed formalization agreements with small-scale miners in several areas of CGI's integrated License 7495 (see section 4.5.1). CGI advises that none of these areas will impede its current and future operations.

Surface rights are not considered a part of the mining titles or rights and are not governed by mining laws even though the mining regime provides for expropriation of real property and the imposition of easements and rights-of-way. Surface rights must be acquired directly from the owners of such rights, but it is possible to request that judicial authorities facilitate expropriation and/or grant easements or rights-of-way necessary for a mining operation.

Table 4.5: Phases of Concession Contracts

Phase	Valid	Surface Tax?	Plan of Work Required?	Environmental Requirements	Environmental Mining Insurance Policy?	Royalty	Reports and Other Filings
Exploration	3 + (4 x 2) years	Yes	Yes	Environmental Management Plan and renewable resources permits if needed (i.e. Superficial Water Concession)	Yes. 5% of planned annual expenditure	No	Basic Mining Formats (FBM)
Construction	3 + 1 years	Yes	Yes	Requires environmental license (issued upon approval of Environmental Impact Assessment)	Yes. 5% of planned investment as per Plan de Trabajo y Obras ("PTO").	No, unless anticipated exploitation occurs	Basic Mining Formats (FBM) Royalty Declaration (in case of anticipated exploitation)
Exploitation	30 years subtracting the years under exploration and construction + 30 years	No (Exception made on areas kept by the concessionaire to undertake exploration activities during a 2-year period).	Yes	Yes. Requires Environmental License (issued upon approval of Environmental Impact Assessment)	Yes. 10 % of the result of multiplying the estimated annual production of the mineral of the concession for the price at the mine mouth for the referred mineral as annually determined by the Government	Yes Based on regulations at time of commencement	Basic Mining Formats (FBM) Royalty Declaration

Source: CGI, 2016

4.5 Royalties, Agreements and Encumbrances

Once the concession has entered the exploitation phase, the concession fees are replaced by a royalty. In the event that the concessionaire keeps an area under exploration after entering into the exploitation phase, concession fees and royalties must be paid simultaneously by the concessionaire. It is possible to initiate anticipated exploitation during the construction phase, in which case, surface tax and royalties will be payable.

Royalties are payable to the state (Royalties National Fund) at 4% of the gross value at the mine mouth for gold and silver, and 5% for copper. Gold and silver royalties are based on 80% of the London Metal Exchange afternoon fixed price for the previous month.

To the extent known by JDS, there are no option agreements or joint venture terms in place for the property.

To the extent known by JDS, there are no known obligations on land covered by claims comprising the property.

4.5.1 Formalization Agreements with Small-Scale Miners

In December 2014, formalization sub-contracts between CGI and eight small-scale mining associations were registered on the *Registro Minero Nacional*, paving the way for the implementation of legal and responsible small-scale mining operations at the Buriticá Project. These sub-contracts are the first to be executed under Law 1658 of July 15, 2013 and are be regulated by Decree 480 of March 6, 2014, which places certain legal responsibilities on the individual small-scale mining associations (i.e. environmental and technical compliance).

At the beginning of the small-scale mine operations, approximately 400 families benefited directly from the formalization, which in turn indirectly benefits local economies, communities and the government's tax base.

None of the areas jointly selected by the company and the small-scale mining associations will impede CGI's current and future operations. Additionally, the sub-contracts include a 50-metre depth restriction from surface and a 41.8 ha surface area, limiting the operations above the planned crown pillar at the top of the deposit.

Of the original formalized associations, four continued in operation at the end of 2015: Sociedad Minera San Roman S.A.S., Sociedad Minera Gualanday S.A.S., Sociedad Minera El Progreso Gold Mine S.A.S and San Antonio 2 S.A.S. Operations discontinued at the other four operations during 2015.

4.6 Environmental Liabilities

4.6.1 General

The principal environmental authority in Colombia is the Ministry of Environment and Sustainable Development (MADS), with national jurisdiction, responsible for formulating environmental and renewable natural resources policies and defining regulations focused on reclamation, conservation, management and use of natural resources, and surveillance of all activities that may have an environmental impact. All activities associated with environmental permitting and control have been delegated to the National Environmental Licensing Authority (Autoridad Nacional de Licencias Ambientales or "ANLA"). At a regional level, the MADS and ANLA functions are executed by

Regional Autonomous Corporations (CAR). Together they constitute the principal environmental authorities. In CGI´s area of operation, there are several Regional Autonomous Corporations in charge of environmental surveillance, such as Corantioquia and CORPOURABA. The MADS is entitled to take control over Regional Autonomous Corporations at its discretion, on a case by case basis, when circumstances require it to do so. Both authorities have the following functions: (i) prevent and/or suspend any activity it deems contrary to environmental standards; (ii) reserve and define areas excluded from mining activities (i.e. forest Reserves and páramo ecosystem); and (iii) approve environmental instruments, such as environmental management plans (Planes de Manejo Ambiental or "PMAs"), mining and environmental guides (Guías Minero Ambientales or "GMAs" and Environmental Impact Assessments (Estudios de Impacto Ambiental or "EIAs")), environmental licenses and permits.

Exploration activities may be performed without the need for approval of an EIA. A company will only need to file for a GMA. If renewable natural resources are needed during the exploration activities, a company must file for the corresponding permits (i.e. water concessions and discharge permits).

An EIA needs to be prepared, presented and approved at the end of the exploration phase if the concession is to proceed to the construction and exploitation phases. The EIA comprises the use of renewable natural resources under the form of a global environmental license. An environmental license is issued and an update of the EIA should be prepared following approval of the environmental license, if additional activities are to be undertaken. The EIA must include details of the baseline study, an assessment of the overall environmental impacts of the Project and plans in order to prevent mitigate or compensate for them.

A concession holder is liable for environmental remediation and other liabilities based on concession holders actions and/or omissions occurring after the date of the concession contract. The owner is not liable for environmental liabilities which occurred before the concession contract, from historical activity, or from those resulting from illegal mining activity.

4.6.2 Buriticá Environmental Liabilities

In December 2007, CGI appointed a Colombian environmental consultancy, Servicios Ambientales y Geográficos/SRK Consulting SA (SG/SRK), to carry out a baseline environmental audit at the Buriticá Project. The initial environmental assessment report was completed in 2008 and the main recommendations of the SG/SRK report were addressed in CGI's March 2008 environmental permit application to Corantioquia. The permit for the Buriticá Project was approved, with additional recommendations in 2008.

The report by SG/SRK highlighted some environmental liabilities that have since been addressed by CGI. The main recommendations included renewing lapsed water usage permits, rehabilitation of the La Mina creek, rehabilitation of the waste, sand and cyanidation tailing dumps, design of a new tailing disposal system, and the implementation of a chemical management program.

CGI began addressing these concerns in 2008, starting with the rehabilitation of the contaminated sand tailing. To mitigate this concern, CGI dry-filled and sealed designated stopes, stabilized the slope of the gravity tailing, and constructed a cyanidation tailing impoundment dam adjacent to the plant site.

SGS Consultants (SGS) were retained to assess the environmental liabilities present at the Yaraguá mine and to estimate the costs associated with the closure of these existing facilities. According to

the recommendations made by SGS, the implementation of environmental management plans was initiated in order to address these environmental liabilities.

An audit of the milling plant's processing circuit was carried out in 2012 in order to improve the industrial wastewater treatment system (Knight Piésold SA, 2012). The improvements made in response to the recommendations of Knight Piésold included:

- The installation of more settling tanks;
- The addition of flocculants to accelerate the sedimentation process;
- In Higabra, a 20,000 m^3 tailing deposit was built in order to improve storage capacity due to the limited space at the Yaraguá mine; and
- Water decantation and filtration was improved for better water treatment, particularly for handling stormwater runoff.

SG/SRK performed the first hydrogeological study for the Buriticá Project, which will be updated annually based on the information gathered. This study was presented in the EIA in December 2013.

4.6.3 Additional Permits

As indicated, additional permits required to work on the facilities during exploration activities include permits for water usage and discharge, atmospheric emissions, forestry clearance and land access. For the Buriticá Project, permits for water usage and discharge were granted in February 2009 and were valid for 10 years. In August 2012, water and discharge permits were incorporated by the Antioquia Environmental Authority for the entire term of the Buriticá Project's Environmental License (see section 20) into a global environmental license.

In August 2012, the environmental authority granted a modification to the environmental license for tunnel and road construction at the Buriticá Project, enabling construction to begin in Q4 2012.

On December 23, 2013, the company submitted an EIA to Corantioquia, representing the final modification to the environmental license for the entire surface infrastructure required to build a mine in the Higabra Valley.

On September 15, 2015, the company announced that it had requested the National Government of Colombia to assume responsibility for reviewing the application for modification to the EIA for the Buriticá Project as a PINES Project (Proyecto Estratégicos de Interés Nacional), as contemplated under Colombian law. Consequently, the company withdrew its application for the modification of the EIA from Corantioquia.

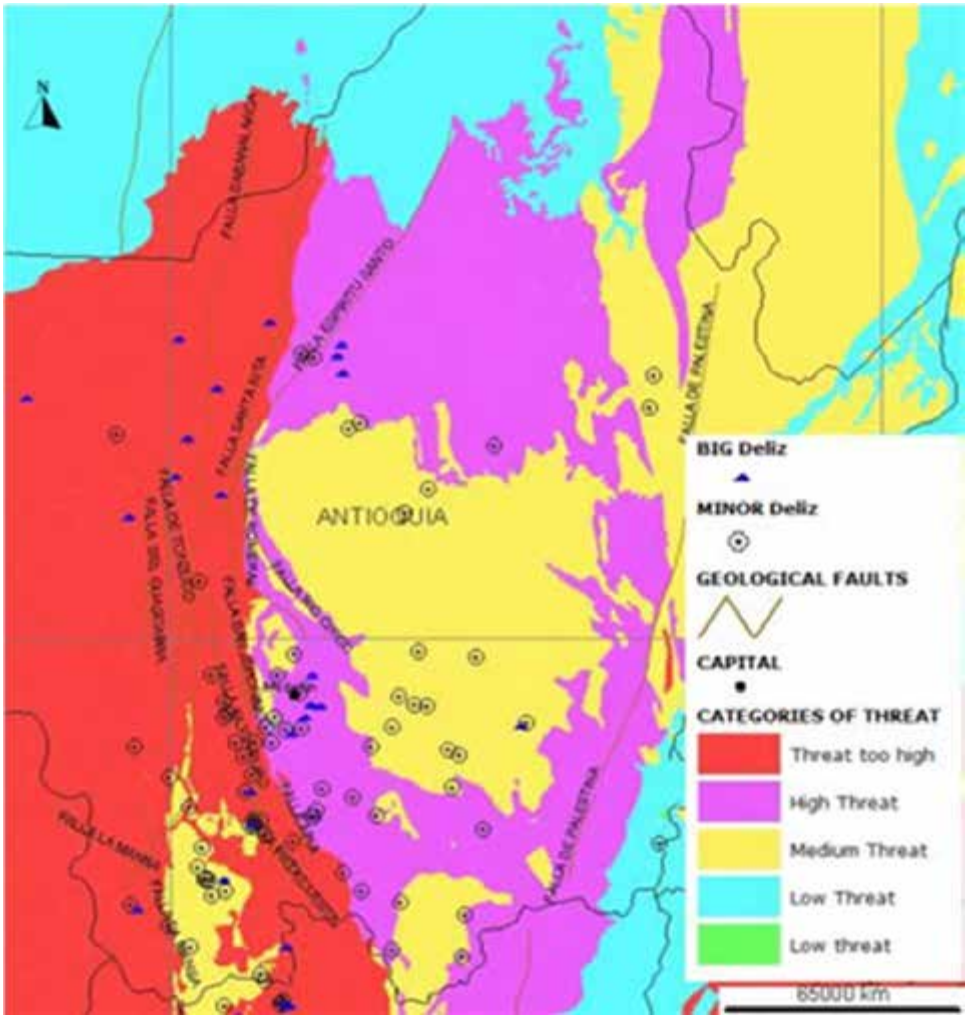
On November 20, 2015, the National Government of Colombia published Decree 2220, 2015 regulating in detail Article 51 of Law 1753, which was passed by Congress on June 9, 2015. Decree 2220 specifically applies to the Buriticá Project, which is classified as a PINES Project. Law 1753 defines the National Development Plan for Colombia from 2014-2018, and Article 51 established that the National Environment Authority (ANLA) is the competent authority to review and approve environmental applications for PINES projects., Subsequently, on January 20, 2016, the company filed with ANLA a new EIA under the provisions of Decree 2220. On February 10, 2016, the company announced that the Constitutional Court of Colombia issued a press release announcing that certain aspects of the National Development Plan (Law 1753) passed by Congress in July 2015, including Article 51, are considered unconstitutional and that they intended to issue definitive rulings

in this regard in due time. The Court indicated that ANLA would not have sole exclusivity over permitting and maintaining environmental aspects of PINES projects. Shortly thereafter, the Court published its ruling, deeming the PINES program constitutional and removing ANLA's authority with respect to having sole exclusivity over permitting by determining that environmental licensing of projects would be subject to existing regulations in force, without reference to PINES guidelines. As a result, PINES projects will be under the authority of either ANLA or regional autonomous corporations, depending on compliance with certain specifications currently in force under Decree 1076, 2015. In any event, ANLA maintains continued involvement and oversight of PINES projects whether or not the Project falls under review by a regional autonomous corporation.

4.7 Other Significant Factors and Risks

The steep topography and high rainfall in the Higabra Valley initially suggests a high risk of mass movements in the Buriticá region of Antioquia as illustrated in Figure 4-3. CGI has undertaken slope deposit mapping (based partly on ultra-high resolution LiDAR, as defined below, topography and surface morphologies) and geotechnical studies through all areas of current planned infrastructure in the Buriticá Project and has concluded that risks from major mass movements are low in these areas.

Figure 4.3: Mass Movement Threat Map, Antioquia Province



Source: Ingeominas, 2011

4.8 Seismic Hazards

The seismic hazard is defined as the probability that a parameter, such as acceleration, velocity or ground displacement produced by an earthquake, exceeds or equals a reference level.

Four hundred and seventy-five (475) municipal areas (corresponding to approximately 35% of the Colombian population) are in high seismic hazard zones, 435 municipal areas (equivalent to 51% of the Colombian population) are in intermediate seismic hazard zones, and 151 municipal areas (equivalent to 14% of the Colombian population) are in areas of low seismic hazard.

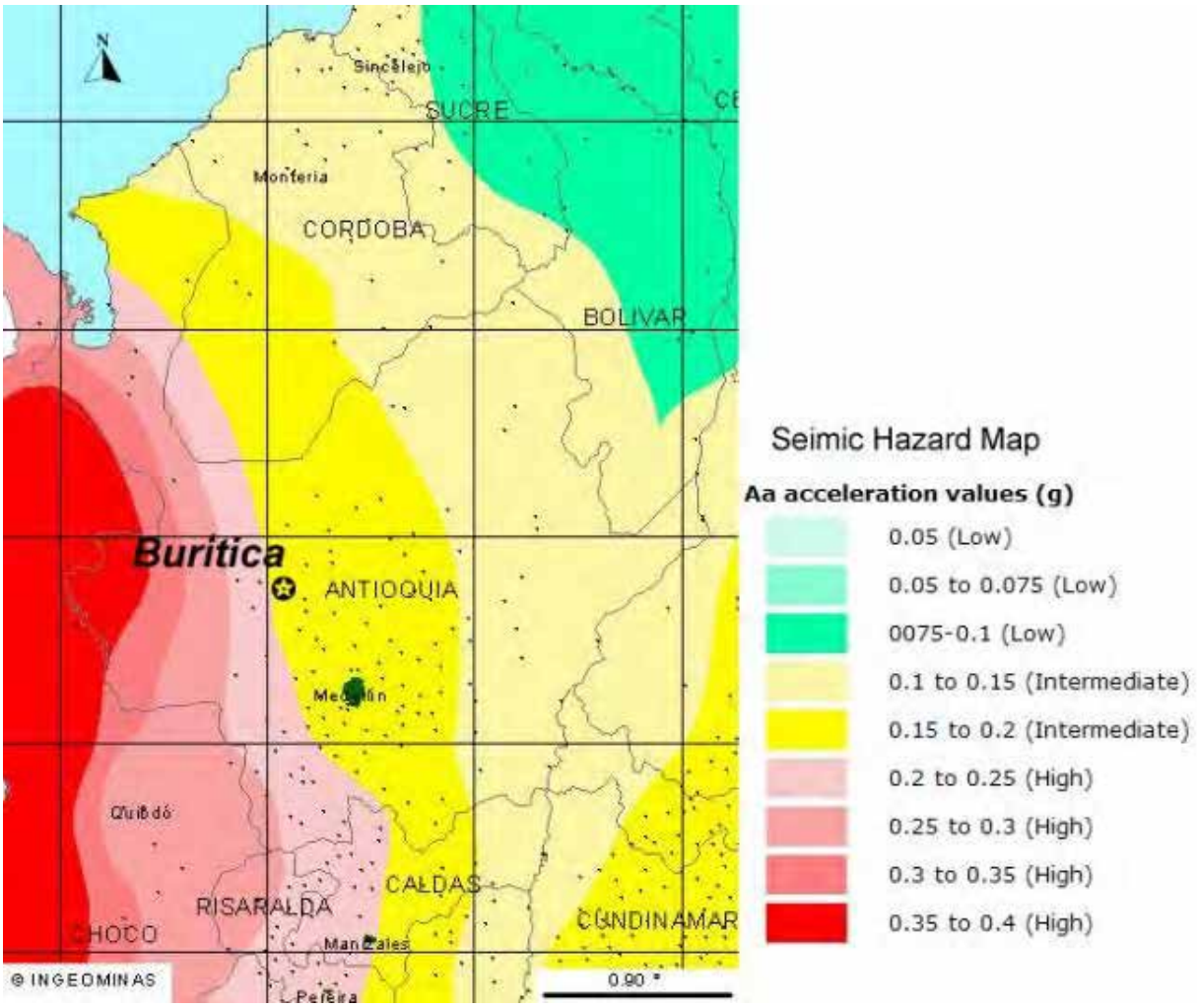
Effective peak acceleration (Aa) corresponds to horizontal accelerations of the earthquake design standards referred to in Colombian Earthquake Resistant Design and Construction (NSR-98) as a percentage of the acceleration of earth's gravity (g = 980 cm/s). These accelerations are likely to be exceeded by 10% within 50 years, corresponding to the life of a building. The parameter Aa is used to define seismic design loads required by the regulation of Earthquake Resistant Structures.

In the Lower Volcanic Zone, as defined for those regions whose design earthquake does not exceed an Aa of 0.10 g, approximately 55% of Colombian territory is included in the area of threat. In the Middle Volcanic Zone, as defined for regions where there is likely to achieve over less than or equal 0.10 g to 0.20 g, about 22% of the territory is included in this area. Seismic Hazards Zone is defined for those regions where strong earthquakes are expected with values greater than 0.20 g. Approximately 23% of Colombian territory is included in the zone of high seismic hazard.

Buriticá lies within the intermediate zone (Middle Volcanic Zone) according to the Ingeominas mapping (Figure 4.4).

CGI's topographic and geological studies of the Buriticá Project area have mapped significant faults that show evidence of displacements in Pliocene times and earlier. These fault systems show no evidence of significant activity in historical times.

Figure 4.4: Seismic Hazards Map and Values of Colombia



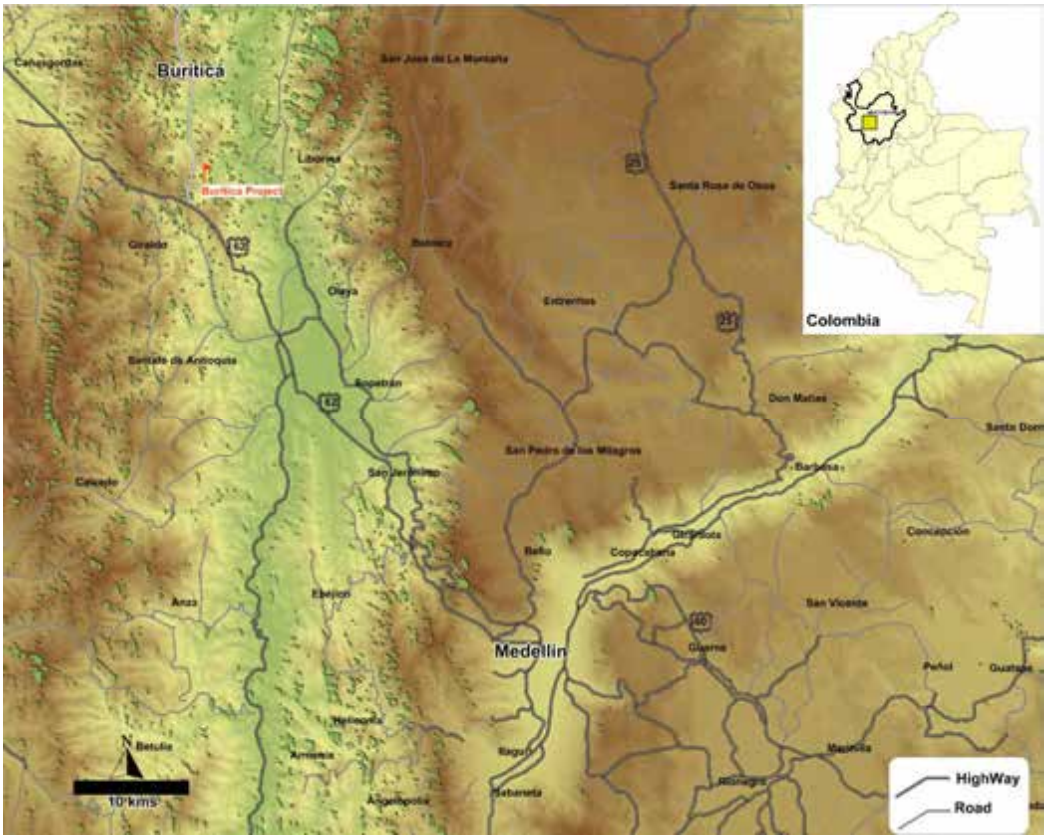
Source: Ingeominas, 2011

5 Accessibility, Climate, Local Resources, Infrastructure and Physiography

5.1 Access

The Buriticá Project is accessed via the international airport at the major city of Medellín and then north-west by driving on Highway 62 for two hours, crossing the Cauca River at the regional centre of Santa Fe de Antioquia (23 km southeast of the mine site) before turning northward towards the village of Buriticá on a minor paved road (Figure 5-1). Drill pads are accessed from the road and a network of tracks. Personnel access to the Yaraguá mine is currently by car. Access to the Higabra Valley is mainly by foot or mule.

Figure 5.1: Location of Buriticá Project NNW of Medellín, Antioquia Province, Colombia

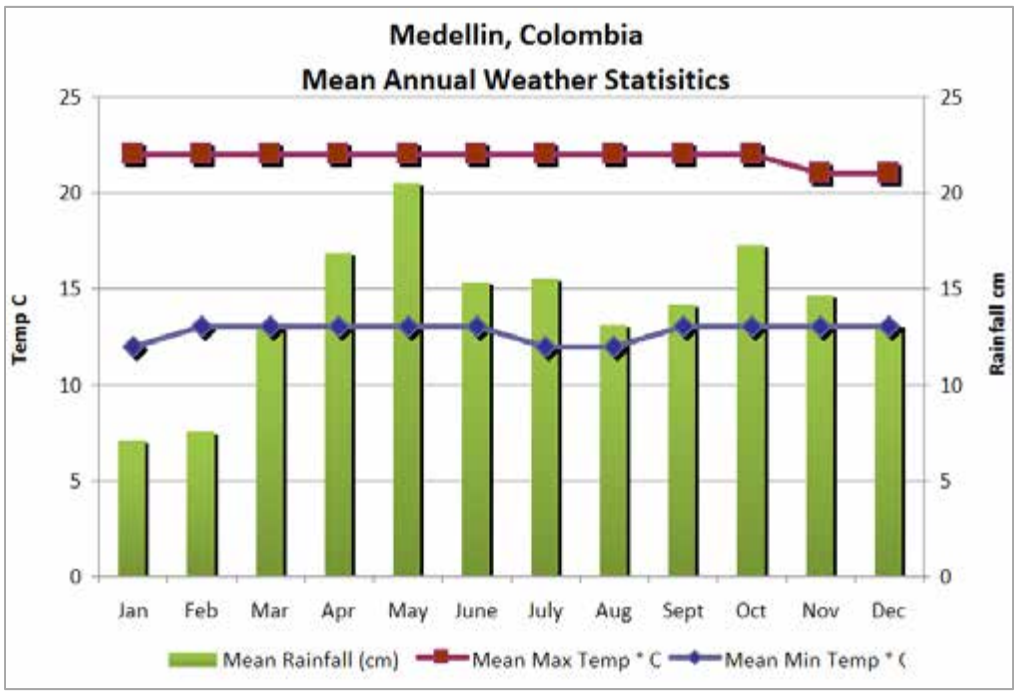


Source CGI, 2015

5.2 Climate

The terrain in the Buriticá Project area is rugged with elevations ranging from around 600 m, at the Cauca River valley to the east, to a maximum of 2,200 m. Being located in the Andean region, the mean average temperatures range from 17°C to 26°C, depending on elevation, with a mean annual rainfall of 1,690 mm (as measured at Medellín). This climate permits year round exploration and mining operations.

Figure 5.2: Rainfall and Temperature Averages for Medellín



Source: MSN Weather, 2015

5.3 Local Resources

The population of Buriticá municipality is about 7,176 inhabitants according to the Departamento Administrativo Nacional de Estadística ("DANE"), 2015. Two villages, Pinguro and Higabra are respectively located in the south and east of the Buriticá Project area, each with populations of a few hundred people.

Buriticá township is situated next to Santa Fe de Antioquia, a municipality in the Antioquia Department, Colombia. The municipality is centred approximately 45 km north of Medellín, the department capital and has a population of approximately 27,600 inhabitants according to DANE, 2015.

The economy of Santa Fe de Antioquia is based on tourism and agriculture; the main products are coffee, maize and beans.

CGI currently employs around 359 people from local communities and has active training and development programs.

5.4 Infrastructure

The Buriticá Project is well developed in terms of infrastructure, with good road access. The site is accessed via a paved road that connects the town of Buriticá to Highway 62 from the city of Medellín (driving time approximately 2 hours). It is also accessible from the major port in Cartagena, Colombia, via the highway network. This highway network accesses the upper portion of the Project site, where the Yaraguá and Veta Sur portals are located.

The Higabra Valley portion of the Project is accessed via a narrow, winding pathway, not navigable by passenger vehicles or wheeled equipment, from the Platanal area, approximately 1,500 m from the Buriticá paved road property entrance.

Colombian grid electrical power is available to the Buriticá township area. Power is supplied from the local grid via a dedicated transmission line to a transformer on site to the upper part of the Project area, for the Veta Sur and Yaraguá ramps and Yaraguá mill. Major equipment, such as the compressors, is electrically powered.

- CGI has connected the Higabra Valley to the local power grid, and is providing power to the Higabra portal from this source.
- The mine has radio and mobile phone communications.
- Mine water is sourced from the Higabra tunnel and is pumped to into a holding tank above the mine offices. The mine offices include a laboratory.

CGI currently operates a small 35 t/d maximum capacity bulk-sampling facility that has been in operation since the early 1990s. There are over 4 km of lateral underground development completed on three levels that cover 150 vertical metres as part of this small-scale mine. CGI is also currently developing three commercial-scale working fronts: the Higabra Valley Tunnel, the Veta Sur Ramp and the Yaraguá Ramp.

The construction of the Higabra Valley Tunnel began in December 2012 and was completed by June 2014 (Figure 5-3 and Figure 5-4). This horizontal development is approximately 1 km long with a 5.0 x 4.5 m section. Although this tunnel will eventually serve as the main access for all underground development by connecting with planned ramps at both the Yaraguá and Veta Sur vein deposits, it initially has been used for underground definition and vertical expansion drilling of the Yaraguá and Veta Sur vein deposits.

The construction of the Veta Sur Ramp began in December 2012 and was completed by August 2013. This development is approximately 600 m long with a 3.5 x 3.5 m section and a negative 13% gradient (Figure 5-5 and Figure 5-6). This ramp provides access to the Veta Sur vein. The ramp portal is at a slightly higher elevation than the top of the deposit (approximately 1,700 m above sea level). A 250 m horizontal cross-cut through the centre of the Veta Sur system was developed as well as a 120 m long drift partially along two veins that are part of the current mineral resource estimate.

The construction of the Yaraguá Ramp restarted in July 2014. This development is approximately 500 m long with a 3.5 x 3.5 m section. The initial objective of this ramp was for exploration purposes, however, it has consequently provided access to Level 0 (1,550 m elevation) of the existing Yaraguá mine.

CGI owns 99% of the hectares affected by existing or proposed surface infrastructure and has surface rights to the remaining areas. CGI has all necessary surface rights to conduct its current and proposed mining operations described in this technical report.

Nevertheless, if additional property and/or surface rights are needed, it is important to consider that mining is deemed a public utility and an activity of public interest; therefore, the owner of a mining title is also entitled to request from judicial authorities: (i) the imposition of easements or rights-of-way necessary for the operation, and (ii) request expropriation of lands needed for the Project, when it is not possible to have an agreement with the land owner.

Figure 5.3: Oblique 3D View of Current and Proposed Ramp and Access Development Yaraguá

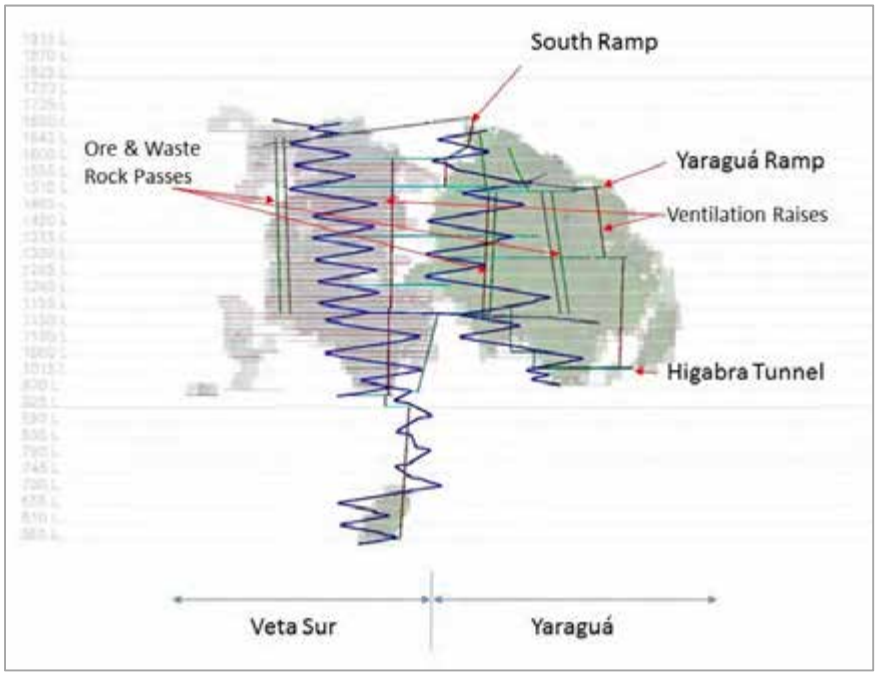
Figure 5.4: Oblique 3D View of Current and Proposed Ramp and Access Development Veta Sur

Source: CGI, 2015

Figure 5.5: Higabra Valley tunnel

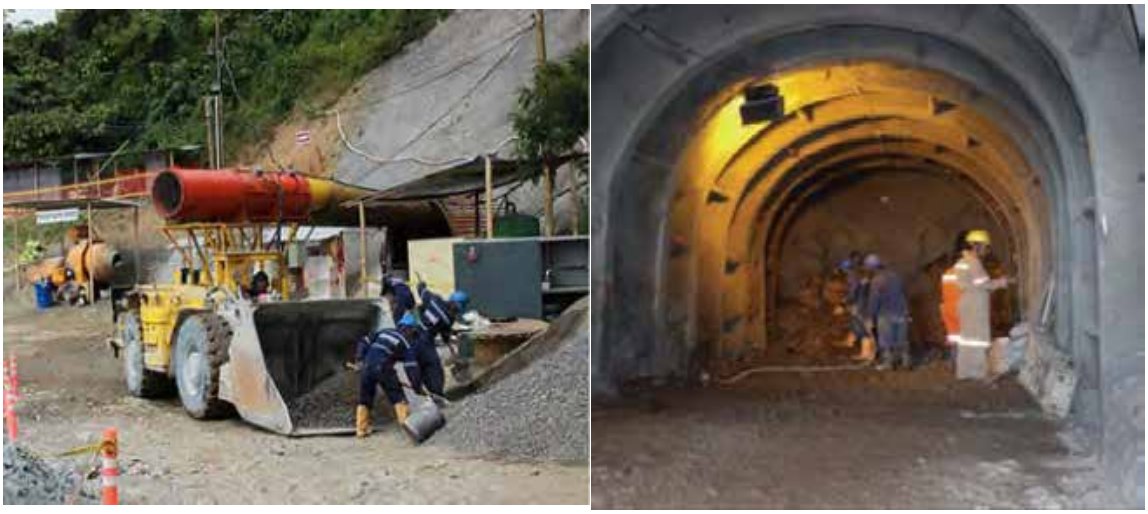
Source: CGI, 2013

Figure 5.6: Three Kilometre Ramp Accessing Veta Sur System



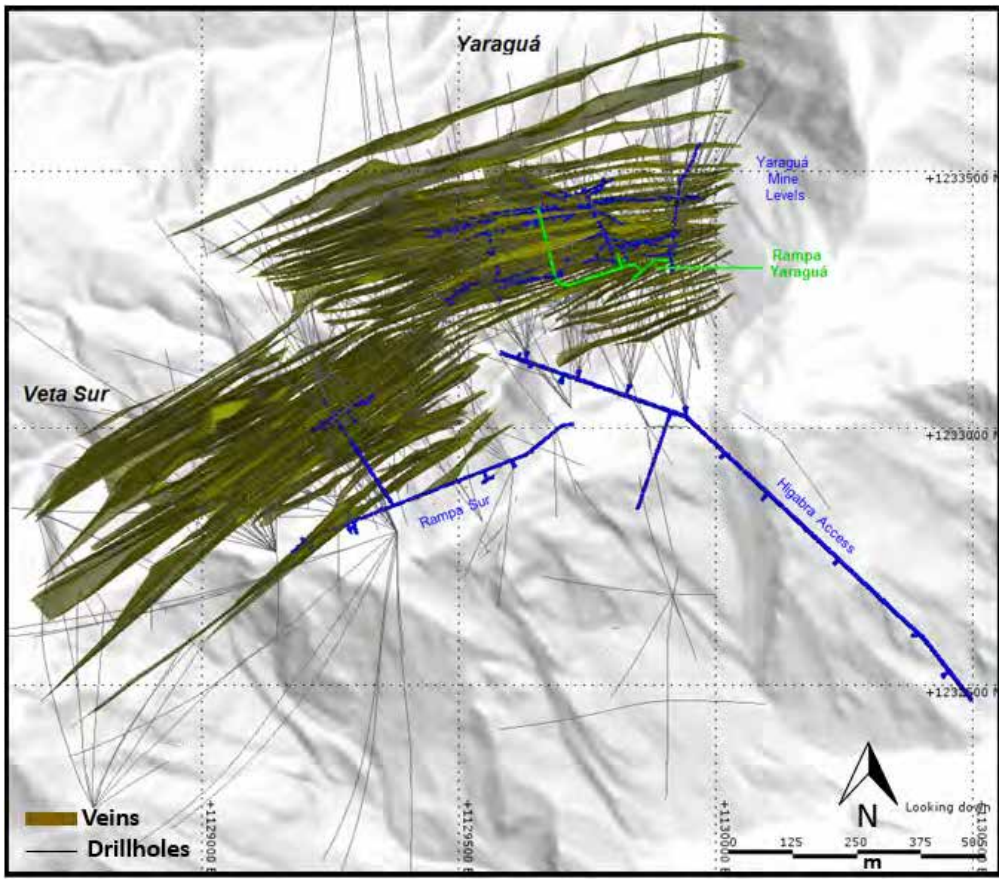
Source: CGI, 2015

Figure 5.7: Veta Sur Ramp



Source: CGI, 2013

Figure 5.8: Plan View showing Veins, Drilling and Development

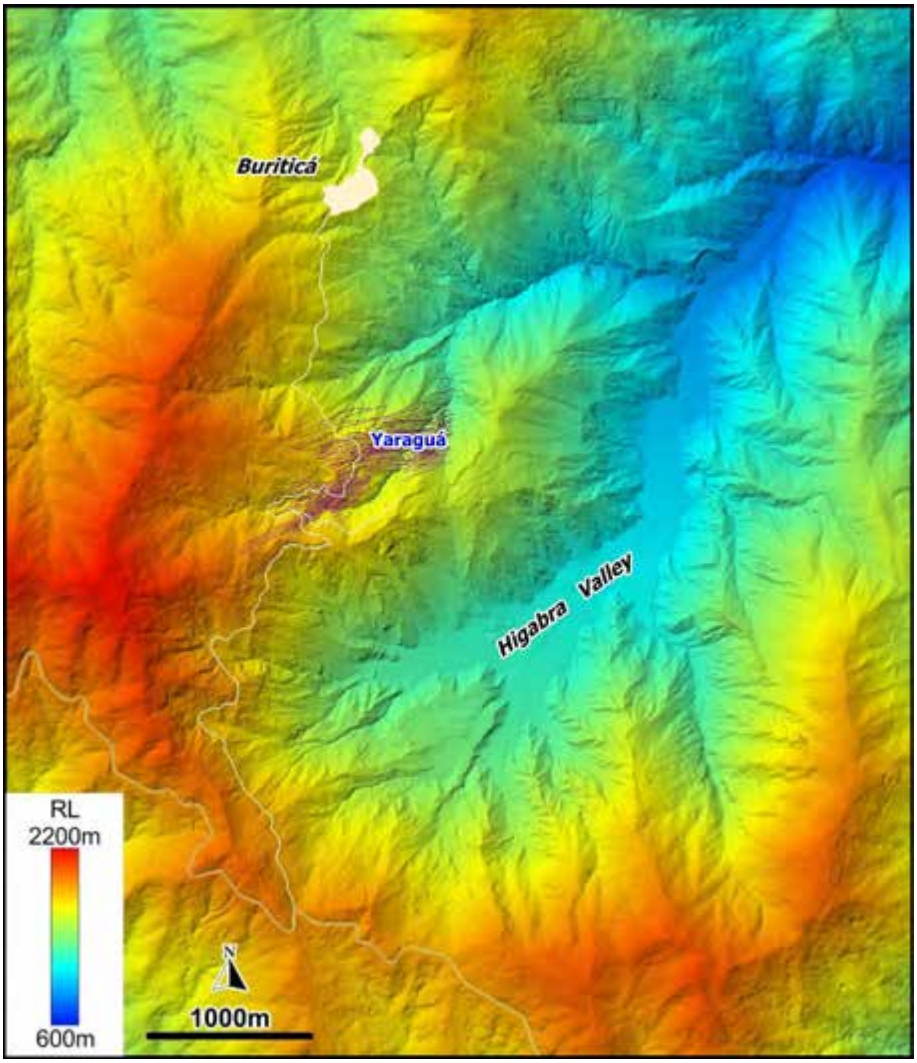


Source: CGI, 2015

5.5 Physiography

The Buriticá Project is situated in mountainous terrain of the Cordillera Central to the west of the north-draining Cauca River. Characterized by steep-sided valleys and subdued peaks, the Buriticá Project area has an elevation range from about 600 to 2,200 masl (Figure 5.8). The Yaraguá mine is located on a steep slope in thickly vegetated terrain at an approximate elevation of 1,500 masl (Figure 5.9) Steeper slopes are forested with small clearings for the cultivation of coffee, yucca, banana and other crops. The Higabra Valley, to the east of the Yaraguá mine is key topographic feature of the Buriticá Project area (Figure 5.9). This broad valley is the site for the main proposed infrastructure such as a processing plant, tailing facility and waste disposal associated with potential mining development of the Buriticá Project. The floor of the Higabra Valley has a mean elevation of 1,050 masl in this area.

Figure 5.9: Physiography at the Buriticá Project



Source: CGI, 2014

Figure 5.10: Mountainous Terrain (left) View from Diamond Drill Pad and (right) View of Yaraguá Underground Mine Portal

Source: MA site visit, 2011

6 History

6.1 Previous Ownership

Grupo de Bullet S.A. ("Bullet") held the main concessions over Buriticá prior to CGI's purchase of the concessions in 2007. On account of certain administrative proceedings, some tenements and applications are still held by other third parties related to Bullet, and awaiting assignment by the mining authority.

6.2 Previous Exploration

Gold was mined in the Buriticá area since before the arrival of Spanish colonialists. The Spanish continued mining, principally from placer and colluvial deposits. There are also several historical vein mines in the area. Several surface mapping and sampling surveys have been conducted by different companies during the 1990s including Gran Colombia Resources Ltd ("GCR"). Only the following prospects, which are considered to be within the Buriticá Project area, had material in the public domain or within CGI's files available for reporting:

- GCR delineated an area of hydrothermal alteration to the west of the Yaraguá mine measuring 700 by 400 m. Channel samples were collected from road cuts, with reported grades up to 7.9 g/t Au.
- Le Mano prospect, a massive quartz-limonite alteration zone in siliceous breccia, located 1 km south of the Yaraguá mine, was excavated with an adit. Sampling at the time reported grades of up to 5 g/t Au, 150 g/t Ag and 4.6% Zn. GCR conducted grid surface sampling in the immediate area and identified several anomalous areas near the adit. Mineralization was noted on the west side of Tonusco Fault.
- La Estera prospect, a vein prospect located 2 km south of the Yaraguá mine, was excavated in a 100 m drift which was suspended due to poor ground conditions. Average grades were reported of up to 12 g/t Au and over 1,000 g/t Ag. Other veins located in the same area, the Sulliman and Pulpito veins, had reported grades of 5 g/t over 0.5 m vein width.
- San Augustin Creek, located 1 km north of the Yaraguá mine, has a 40 m wide zone of sulphide mineralization in sedimentary rocks. Old workings were reported to the northwest of this occurrence. The mineralization was reported as being associated with a zone of sediments within igneous rocks. Samples from the contact zone were reported to contain an average grade of 1.45 g/t Au and 24.3 g/t Ag.
- La Guacamaya prospect, located just north of the Clara Creek in the Northeast of the Buriticá Project area, was identified as a contact breccia between sediments and a diorite intrusive. Sampling reportedly returned an average grade of 2.7 g/t Au in talus.

6.3 Historic Resource and Reserve Estimates

There is no known historical resource or reserve estimates for the Buriticá Project area. Initial resource estimates for the Yaraguá and Veta Sur prospects by Mining Associates of Brisbane were released by CGI in October 2011, and updates released in September 2012, May 2014, and June 2015 and are not considered historical in the context of this report.

6.4 Historic Production

In the Buriticá area, gold has been mined since before the arrival of the Spanish colonialists in the seventeenth century. Some areas of alluvium and colluvium are believed to have been worked by hydraulic methods. In several areas of the Buriticá Project, high-grade veins were worked in shallow underground artisanal operations for gold and silver and such operations continue to the present day.

Bullet acquired its Buriticá concessions over the last 20 years and undertook some development in the Yaraguá prospect. The Yaraguá mine has been producing gold semi-continuously since 1992, mainly from the Murcielagos vein family which has been partially worked over a strike length of about 470 m and a vertical extent of 160 m.

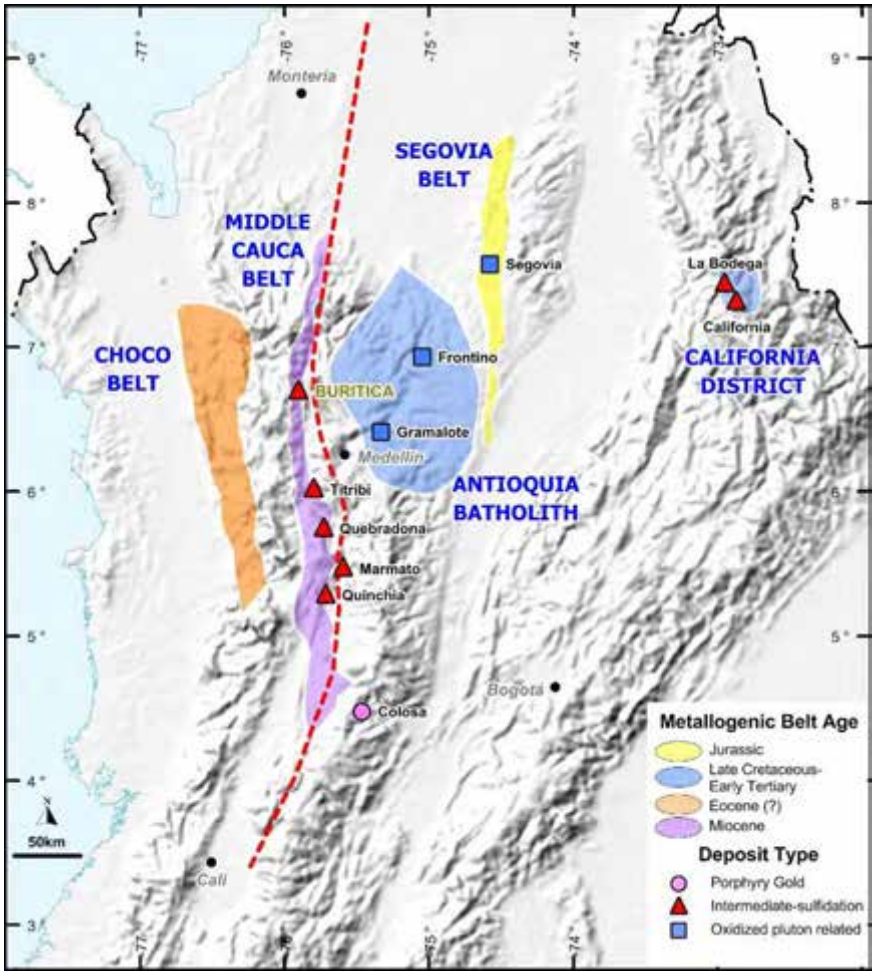
Between 2001 and 2007, the Yaraguá mine produced 11,694 oz of gold (no tonnage or grade data available).

7 Geological Setting and Mineralization

7.1 Regional Geology

The northern Andean cordillera in Colombia has been uplifted by the subduction of the Nazca oceanic plate beneath the Guiana Shield along with interaction with the Caribbean plate to the north. The region has been tectonically active from the Mesozoic through to the present. Subduction has created island magmatic arcs that have since accreted to the continental margin in generally north-south oriented belts. These magmatic arc terrains host the majority of the precious metal mineralization in Colombia (Figure 7-1).

Figure 7.1: Principal Gold Belts and Districts, Northern Colombia



Source: CGI, 2014

Oblique subduction beneath the continental margin, with northern influence of the Caribbean Plate and consequent magmatic arc development resulted in the formation of the Miocene-aged central magmatic arc that hosts the Buriticá Project. The 80 to 120 km wide belt of Pliocence to recent volcanoes extending from south-central Ecuador to central Colombia is due to the subduction of Miocene-aged Nazca oceanic crust beneath north-western South America along the Ecuador-

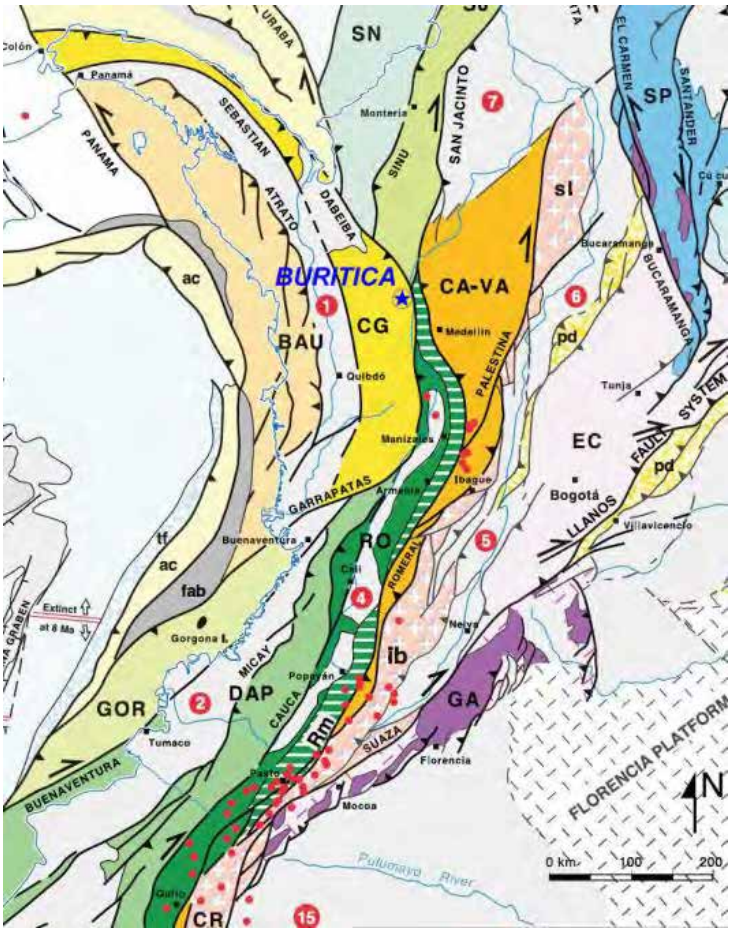
Colombia trench. Volcanism is dominated by lavas and pyroclastic rocks of andesitic, dacitic, and lesser basaltic composition. This belt is roughly coincident with the central magmatic arc.

Using the lithotectonic terminology and descriptions from Cediel et al (2003), the basement rocks of Buriticá Project are related to the emplacement of the Cañas Gordas (CG) and Baudó (BAU) terranes (Figure 7-2). The Buriticá Project is located within the CG terrane. The mafic-ultramafic, tholeiitic basaltic and overlying sedimentary assemblages of the CG terrane are allochthonous with respect to continental South America. The ultramafic-mafic-sedimentary assemblage likely originated in an oceanic volcanic-arc setting of middle Cretaceous age.

Unconstrained Northeast migration of the CG terrane was accompanied by the development of a new magmatic arc, giving rise to the emplacement of the Mande-Acandi Batholiths in the early Eocene. The Mande-Acandi Arc intrudes the western margin of the CG terrane. It ranges from tonalitic to granodioritic in composition and was constructed on oceanic crust.

The collision of the composite CG assemblage with the continent began in the Miocene and final accretion was coincident with the arrival of the BAU terrane. Magmatism associated with BAU arrival shifts eastwards at approximately 8 Ma, as recorded by a series of stocks emplaced along the margins of the Cauca-Romeral fault zone of central-north Colombia. These include the Buriticá- and other intrusive complexes, of broadly intermediate compositions (all in the 6–8 Ma range; Figure 7.3).

Figure 7.2: Lithotectonic and Morphostructural Map of North-western South America. Buriticá lies in the Miocene Arc on the Eastern Edge of the CG Terrane



Legend

EC = Eastern Cordillera
CA-VA = Cajamarca-Valdivia terrane
CR = Cordillera Real
RO = Romeral terrane
DAP = Dagua-Pinon terrane
GA = Garzón massif
GOR = Gorgona terrane
CG = Canas Gordas terrane
BAU = Baudo terrane
Rm = Romeral melange
SN = Sinú terrane
SP = Santander massif – Serranía de Perijá
fab = fore arc basin
ac = accretionary prism
ib = Ibagué block
tf = trench fill
pd = piedmonte
1 = Atrato (Choco) basin
2 = Tumaco basin
4 = Cauca-Patıa basin
5 = Upper Magdalena basin
6 = Middle Magdalena basis
7 = Lower Magdalena basin
15 = Putumayo-Napo basin
red dot = Pliocene-Pleistocene volcano

Source: Cediel et al, 2003

7.2 Geological Setting and Mineralization- Local Geology

The geology of the Buriticá region is dominated by Cretaceous basalts (unit K2-Vm) and gabbroic to ultramafic bodies (K2-Pm), and stratigraphic ally overlying Cretaceous turbiditic meta-sediments (unit k2k6-Mds). The sedimentary assemblage includes carbonaceous, variably calcareous pelitic and psammo-pelitic units. In the east and west of the Buriticá Project area, Late Cretaceous tonalitic plutonic suites (unit K2-Pf), approximately 2.5 km (E-W) by 10 km (N-S), intrude the Cretaceous basement (Figure 7-3). Basement lithologies are commonly moderately to steeply-dipping and north-northwest striking deformed into shallowly plunging broad open folds, locally with a weak axial surface cleavage and are mainly of lower greenschist facies metamorphic grade.

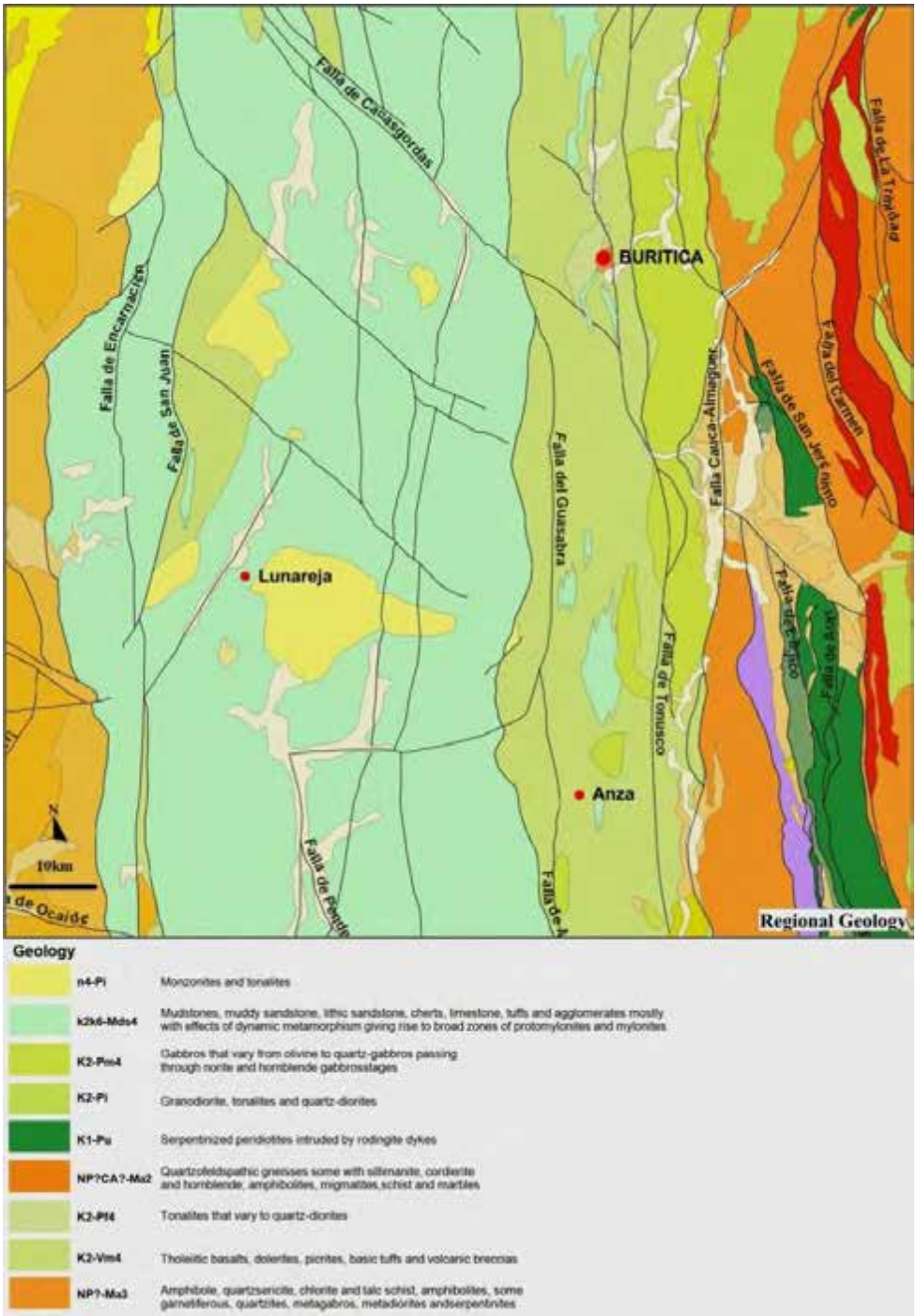
The basement assemblages are intruded by hypabyssal Miocene bodies of broadly intermediate compositions ranging from basaltic andesites to relatively mafic dacites. To the west of Buriticá these dioritic intrusions (n1n4-Pi, Figure 7-3) form relatively large bodies. In the Buriticá Project area there are several clusters of Miocene intrusions comprising steep walled stocks and dyke-like bodies exhibiting fine- to medium grain sizes and variably porphyritic textures and commonly with intrusive breccia margins. The largest of these intrusion complexes, in the Yaraguá-Veta Sur area (Figure 7-4) crops out at over 6 km^2. Other intrusion complexes have been mapped by CGI in the Guarco area to the northwest and the Pinguro area to the south (Figure 7-4). Andesitic dykes occur throughout the Buriticá Project area and these along with the exposed intrusion complexes are thought by CGI to represent offshoots of larger Miocene plutons at depth. CGI's lithogeochemical data on the Yaraguá and Veta Sur intrusions is consistent with a fractionated and hybridised continental-arc, calc-alkaline suite. Recent radiometric dating (Lesage, 2011) has placed the age of the Buriticá intrusive complexes at 7.4+/-0.1 Ma, consistent with the 6-8 Ma ages of other intrusive complexes with which porphyry copper-gold and epithermal mineralization is associated in the Middle Cauca belt.

7.3 Project Geology

The Buriticá Project area is transected (and geologically partitioned) by a set of regionally extensive north-south to north-northwest trending faults, broadly geometrically similar to the major Cauca and Romeral faults further to the east (Figure 7.3 and Figure 7.4). To the east of the Yaraguá and Veta Sur mineralized systems, the steeply-dipping Tonusco Fault system cuts off the intrusion complex and related hydrothermal alteration envelopes, consistent with the significant dip and strike slip movements on this fault as postdating intrusion and alteration. A set of east-dipping faults to the west of and possibly related to the Tonusco Fault cut across the Yaraguá and Veta Sur mineralization but appear to exhibit relatively small (and dominantly dip-slip) displacements.

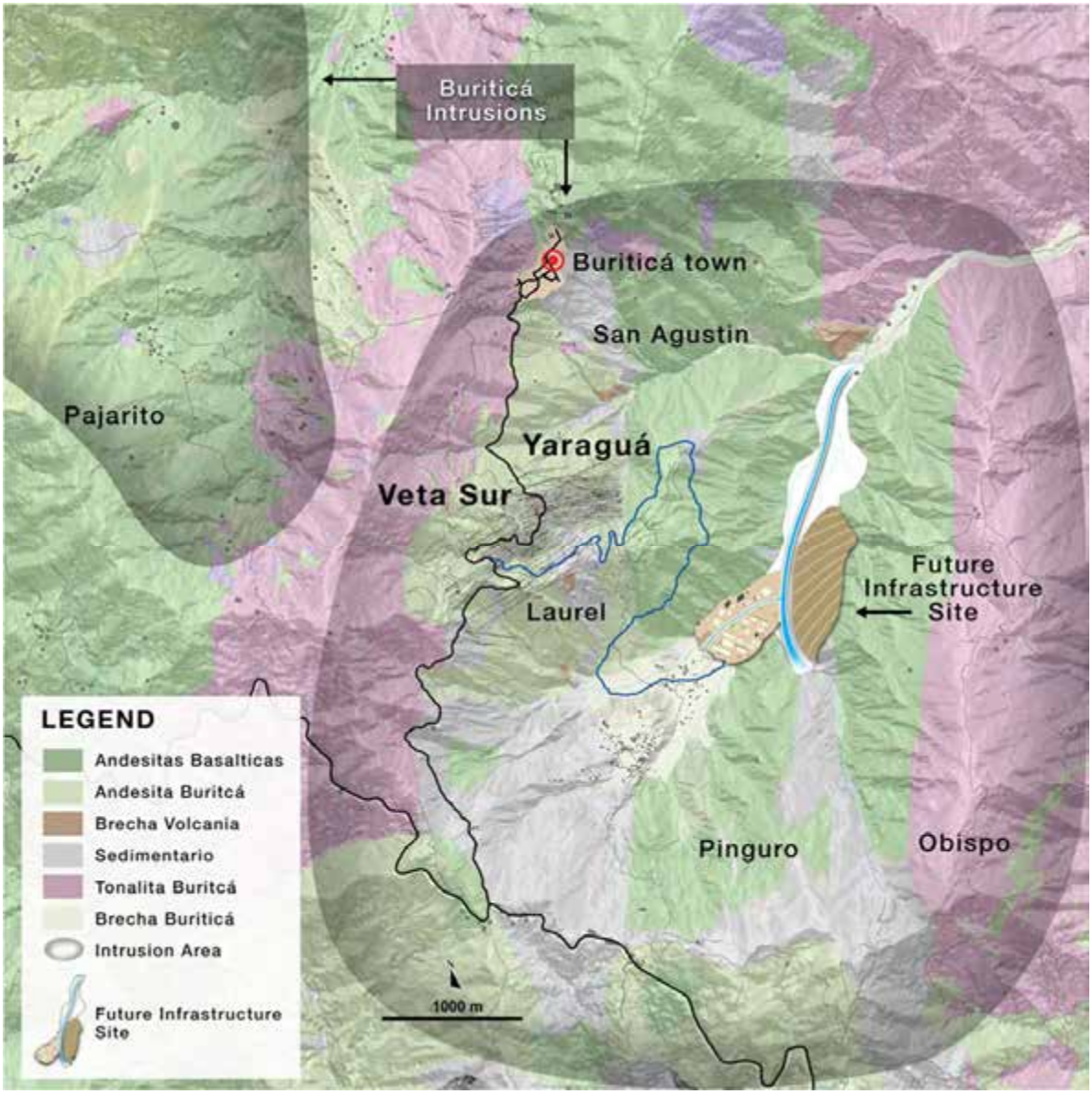
Other district-scale fault-fracture zones in the Buriticá Project are broadly spaced and of east-Northeast and west-northwest orientations, evident in drainage alignments (Figure 7.4) and other topographic expressions. These fault-fracture zones do not appear to have involved large displacements, but the geometries and distribution of vein systems and alteration are compatible with the fault-fractures having being active during formation of mineralization.

Figure 7.3: Regional Geology, Buriticá Area



Source: Ministry of Mines and Energy, Colombia, 2007

Figure 7.4: Local Geology Showing Known Deposits and Prospects



Source: CGI, 2015

7.4 Mineralization

Buriticá is the northernmost significant precious metal deposit known to date in the upper Miocene Middle Cauca belt, one of the three major gold belts identified in Colombia (Figure 7.1). The belt contains porphyry- and epithermal- to mesothermal vein styles as well as carbonate base metal gold systems broadly similar to Buriticá. Numerous copper-gold and precious metal-rich systems are developed along the 300 km belt and all appear to be spatially related to relatively small, high level Miocene intrusions of intermediate composition.

Mineralization at Buriticá is a porphyry related, carbonate base metal (CBM) gold vein/breccia system. High-grade precious metal mineralization in CBM systems may occur over substantial vertical intervals, to well in excess of a kilometre, from the porphyry level to below the shallow epithermal range (Figure.-6). Compared to low sulphidation epithermal styles CBM mineralization is sulphide-rich, with abundant pyrite ± pyrrhotite + sphalerite + galena along with minor sulfosalts, chalcopyrite and quartz-carbonate gangue mineralogy (Figure 7.5). Gold in CBM systems may be free-milling or refractory. Mineralization in CBM systems typically comprises sheeted veins, stockworks and mineralogically similar breccias with some fracture-related dissemination in wall-rocks.

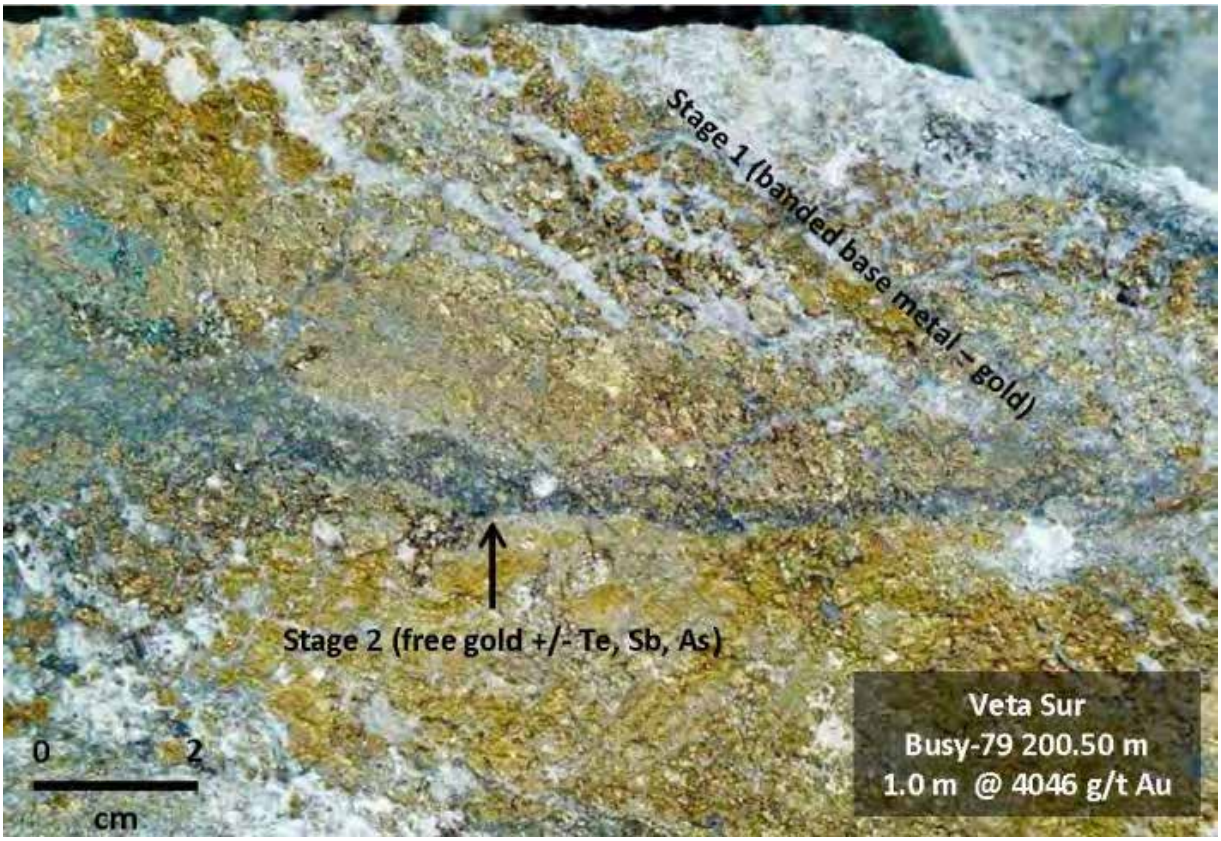
CBM systems are widespread in circum-Pacific magmatic arcs, and include the supergiant Porgera deposits in Papua New Guinea (25 million ounces of gold, previous production plus remaining reserves and resources) and the Kelian deposits (7 million ounces (Moz) of gold produced + resources) in Indonesia. Gold mineralization at Porgera has been mined over a vertical range of 500 m but potentially economic mineralization extends over more than 1,000 m depth range.

CGI recognized the strong similarities of the Porgera and Buriticá mineralized systems (Figure 7.5) and this gave CGI the confidence to drill extensively and deeply at Yaraguá and Veta Sur. An outcome of which has been the substantial resource examined in this report as well as the realization that intrusion-centred mineralization may occur over a broad elevation range elsewhere in the Buriticá Project.

Precious metal mineralization in Yaraguá and Veta Sur appears to be related to two main depositional stages. Stage I is represented by banded base metal (iron, zinc and lead) sulphide-rich mineralization with variable amounts of quartz-carbonate gangue and bands. As well as in sub-parallel narrow vein arrays, Stage I mineralization also occurs in veined (dilational) breccias in places occupying substantial areas of both Yaraguá and Veta Sur, but at grades typically lower than those of the high-grade veins. CGI's experience at the Yaraguá mine indicates that Stage I mineralization is non-refractory and recoverable by simple gravity and flotation circuits, flotation concentrates being cyanided, gold and silver then recovered by the Merrill Crowe process. Wall-rock alteration around Stage I veins comprises narrow phyllic assemblages ± K-feldspar. Stage I mineralization evidently overprints earlier potassic, phyllic and propylitic alteration.

Stage II mineralization is a texturally and chemically distinctive high-grade gold mineralization that locally cross-cuts and overprints Stage I mineralization as veins and breccia veins. Stage II mineralization is characterized by abundant free (and commonly visible) gold in siliceous and carbonate gangue, associated with arsenical pyrite and with low zinc and lead contents, relatively high arsenic and antimony but low bismuth contents. Stage II mineralization is, to date, largely known from the Veta Sur system in which this style of mineralization contributes to some very high-grade precious metal subzones, but has also been encountered in the Yaraguá system.

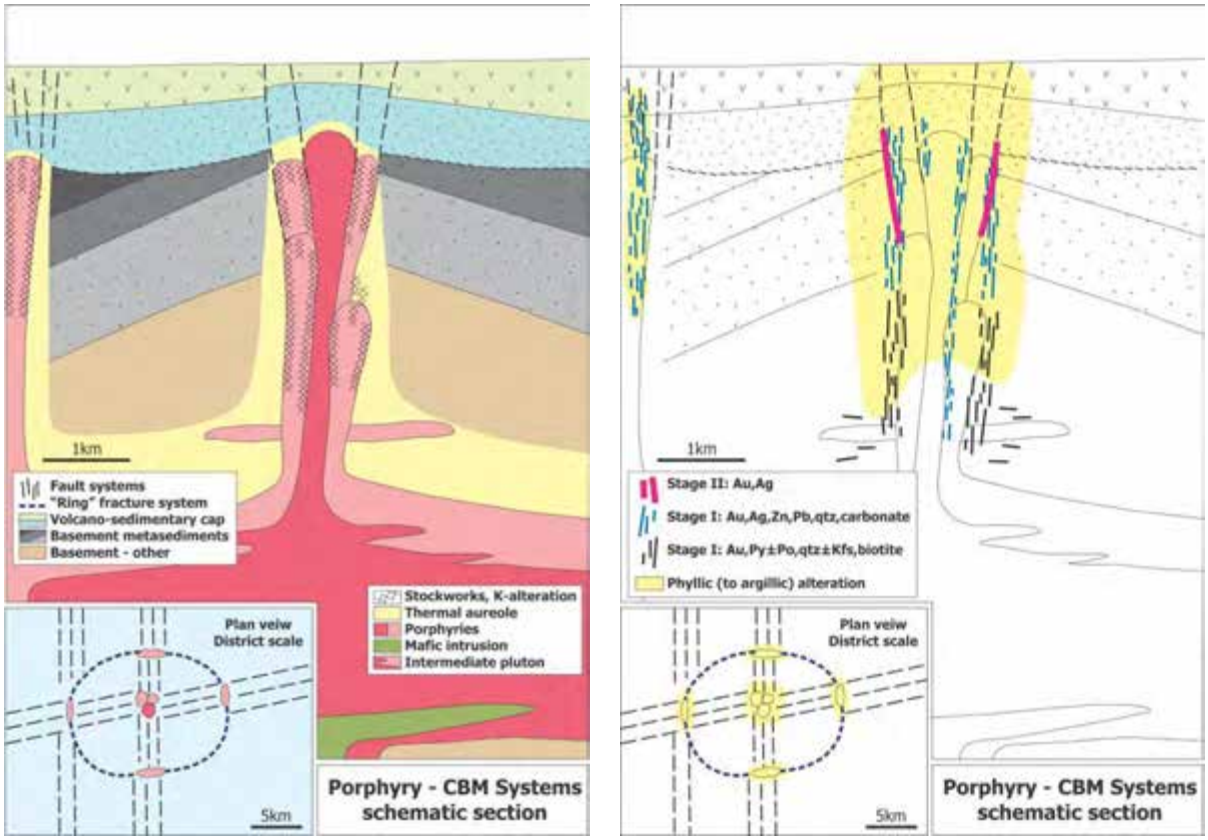
Figure 7.5: Veta Sur Vein Mineralization



Source: CGI, 2012

The distribution of precious mineralization at Buriticá in relation to host rocks and alteration is outlined schematically in Figure 7.6.

Figure 7.6: General Model for Distribution of Mineralization in Carbonate Base Metal Systems with Specific Reference to Buriticá



Source: CGI, 2014

7.4.1 Host Rocks

Alteration and mineralization in the Buriticá Project are spatially associated with the Miocene intrusion complexes. Three generations of alteration have been mapped (Figure 7.6):

- Hornfelsing and some silicification around the margins of the intrusions,
- Potassic- (biotite, +/- K-feldspar, associated with quartz magnetite–rich stockworking) and propylitic alteration mainly but variably affecting the intrusions. The alteration is associated with weak base metal anomalism in the Yaraguá-Veta Sur areas but with copper-gold mineralization in the Guarco area,
- A widespread, fracture-controlled phyllic alteration (sericite-pyrite-quartz) which overprints and may obliterate the other alterations, but is of similar age to the intrusions (Lesage, 2011). In the Yaraguá-Veta Sur area the extents of the strong phyllic alteration correspond to the broad gold, silver, zinc and lead soil geochemical anomalies.

7.4.2 Controls

Mineralization at the Buriticá Project is structurally controlled. Most vein domains in the Yaraguá system strike around 010º, but another vein set (including the Centana vein) strikes west-northwest (Figure 7.7). The main Yaraguá vein families occur in several packages over about 150 m across-strike distance, with interleaved wall-rocks (mostly andesitic intrusions and intrusion breccias) containing numerous minor veins and also breccia-style mineralization. Extensive underground development has been undertaken on several of the Yaraguá vein systems including the San Antonio and Centana veins.

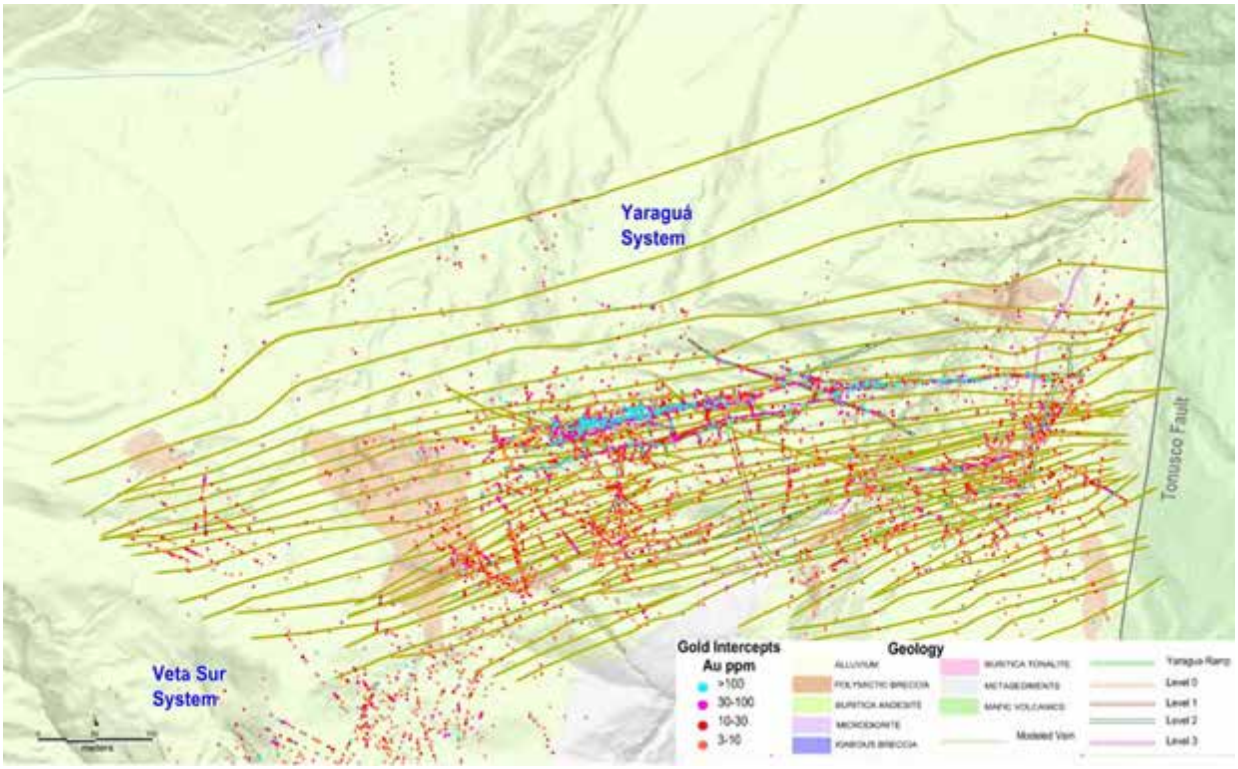
The Veta Sur system (Figure 7.8) strikes around 055º and in its north-east extents overlaps with and intersects the Yaraguá system. Host rocks include andesitic intrusions and breccias as well as basement meta-sediments and meta-basic rocks.

Precious metal–bearing vein and breccia mineralization has been located elsewhere in the Buriticá Project, principally in the Guarco, Parjarito, San Augustin, La Estera, La Mano and Pinguro areas (Figure 9.1). Porphyry copper-gold mineralization has been observed in the Guarco area. These areas are subject to ongoing exploration by CGI.

7.4.3 Dimensions and Continuity

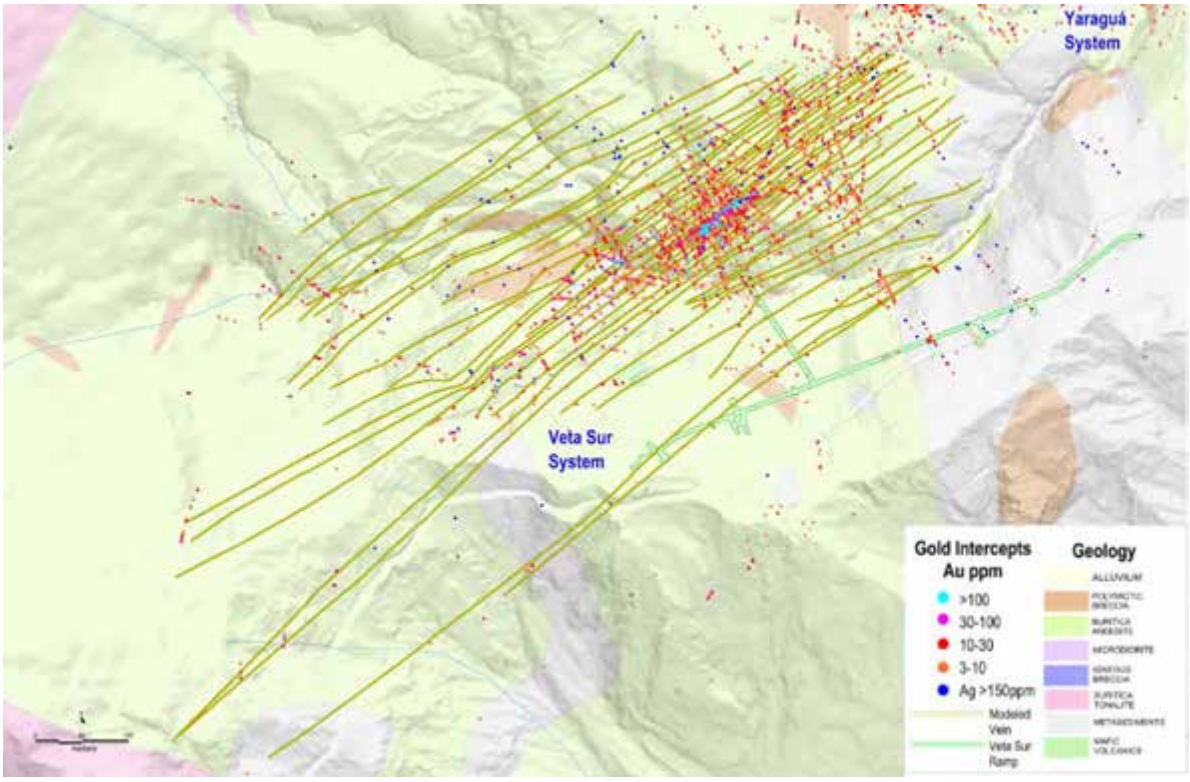
The Yaraguá system (Figure 7.7 and Figure 7.9) has been drilled along 1,125 m of strike and 1,540 vertical metres and partially sampled in underground developments. The Veta Sur system (Figure 7.8 and Figure 7.9) has been drill intersected along 1,140 m of strike and 1,600 vertical metres. Both systems are characterized by multiple, steeply-dipping veins and broader, more disseminated (breccia-style) mineralization. The Yaraguá and Veta Sur systems both remain open laterally and at depth at high grades.

Figure 7.7: Yaraguá Vein System Dimensions and Veins showing Assays Intercepts > 3ppm



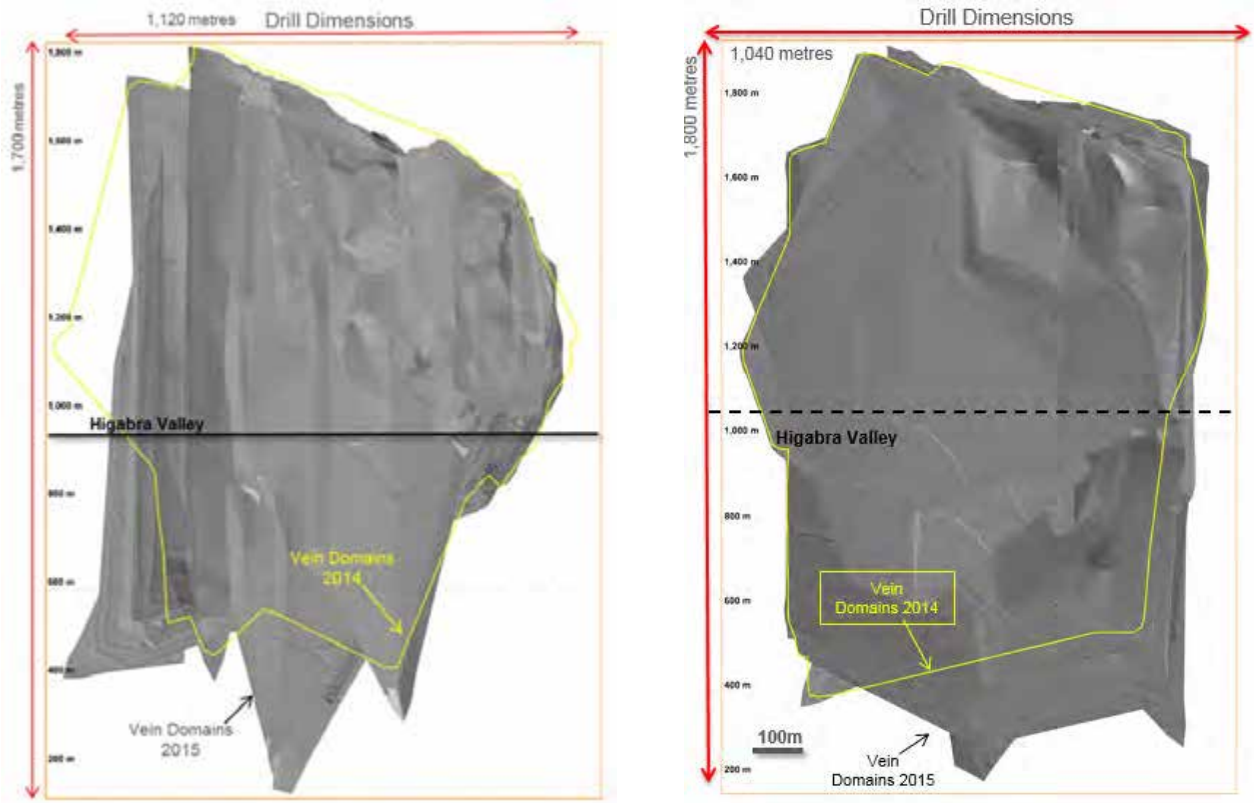
Source: CGI, 2015

Figure 7.8: Veta Sur Vein System Dimensions and Veins Showing Some Assays Intercepts



Source: CGI, 2015

Figure 7.9: Yaraguá Zone – Dimensions Yaraguá (Left) and Veta Sur (Right)



Source: CGI, 2015

8 Deposit Types

8.1 Geological Model

Buriticá is the northernmost significant precious metal deposit known to date in the upper Miocene Middle Cauca belt, one of the three major gold belts identified in Colombia (Figure 7.1).

This belt contains gold-rich porphyry copper-gold deposits (the largest of which is La Colosa, with a resource of more than 20 million ounces of gold) and also vein-style precious metal mineralization in settings described as mesothermal to epithermal and including the Buriticá deposits. Numerous gold systems are developed along the 300 km belt and many appear to be spatially related to relatively small, high level intrusions of intermediate composition. Some highly pyritic copper-gold-(zinc) mineralization in the Middle Cauca belt has also been referred to the “volcanogenic massive sulphide” association.

The Buriticá deposits are classified as intermediate sulphidation epithermal using the terminology of Sillitoe and of Hedenquist (in Simmons et al, 2005). Corbett and Leach (1997) recognized a specific sub-class of low sulphidation epithermal deposits with high base metal contents and carbonate-bearing gangue, which they named CBM association. Although such mineralization is perhaps better described as “base metal carbonate sulphide-rich”, and may occur over depth extents much larger than many low sulphidation epithermals, the Buriticá systems are referred to as the CBM association (Figure 7.6).

9 Exploration

Exploration activities at the Buriticá Project conducted prior to and during 2012 are described in full in the 2012 Technical Report. In summary, these activities consisted of topographic and geological mapping, aerial magnetic and radiometric surveys, geochemical soil surveys and other surface sampling, underground mapping and channel sampling in Yaraguá, underground drilling at Yaraguá and drilling from surface in both Yaraguá and Veta Sur.

During late 2012 and 2013, exploration activities other than diamond drilling included extensions and infill of geochemical soil surveys as well as underground mapping and channel sampling of new developments in Yaraguá. A significant boost to surface mapping has resulted from the long-awaited capture of 1 m resolution satellite imagery over the entire Buriticá Project and also a 0.5 to 1 m resolution LiDAR topographic survey over the central part of the Buriticá Project area. The detailed analysis of these products (and also ultra-high resolution aerial photography accompanying the LiDAR capture) has resulted in the mapping of the significant fault-fracture systems in the district. The surface mapping identified various vein systems and also the location of numerous artisanal workings on these vein systems in heavily vegetated and hard-to-access areas.

9.1 Soil Survey 2012-2013

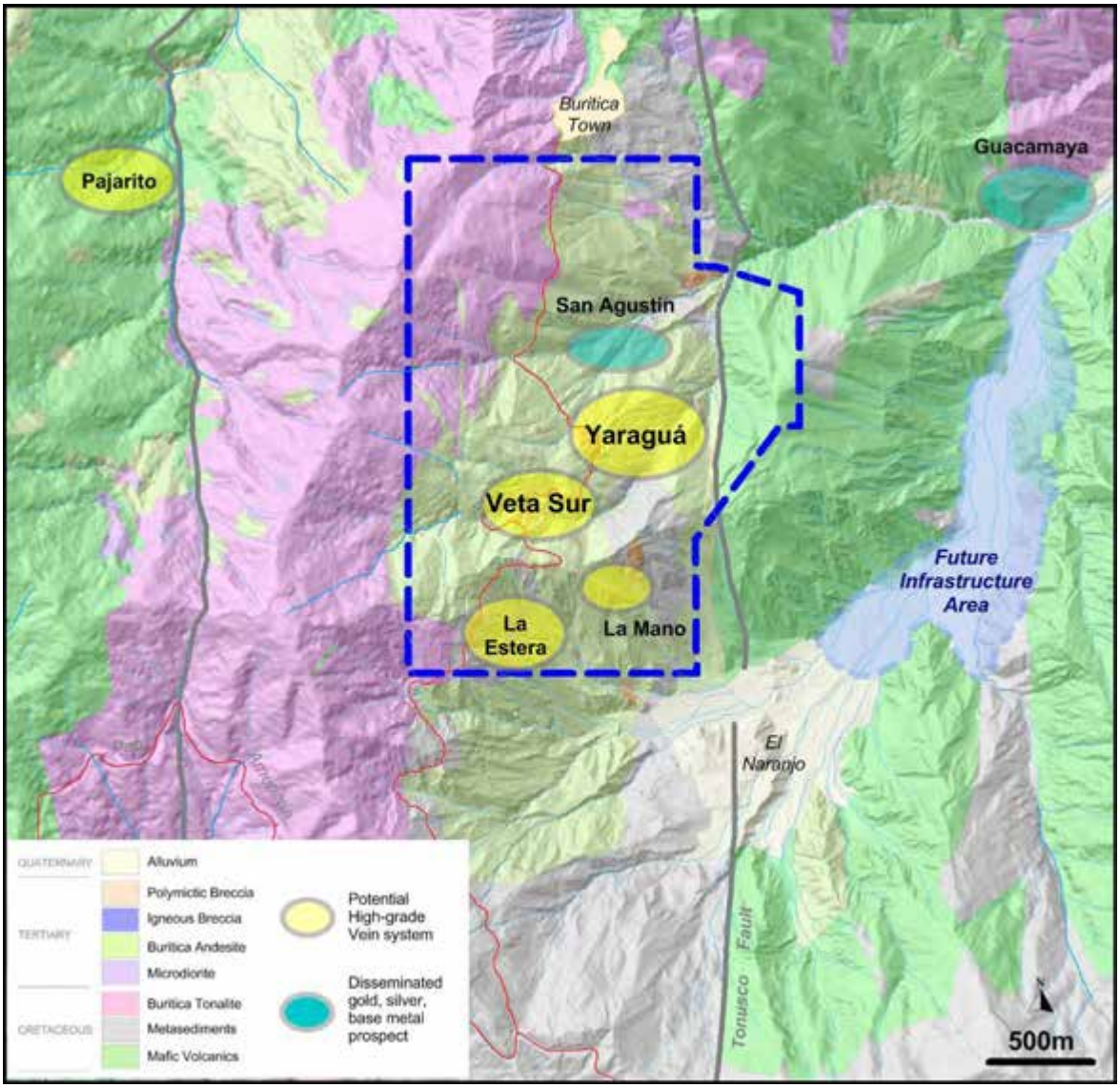
To clarify the distribution of near surface mineralization, CGI undertook a detailed soil geochemical program at 100 m line x 20-25 m sample spacing and assayed for gold, silver, lead and zinc plus a broad suite of additional elements using the base metals as pathfinders. Additional samples were taken as infill and ridge-line sampling. The survey covered an area of approximately 2.5 x 2.0 km of the Buriticá property as illustrated in Figure 9.1.

The soil survey results are plotted in Figure 9.2. Results show anomalous values for all elements in the areas of known mineralization. They also show a strong response in La Mano and La Estera areas and lesser anomalous responses to the northwest of the Tonusco Fault (San Augustine area) and further to the west.

Gold, silver, lead and zinc are generally strongly correlated, as in the Yaraguá mineralization, with lead and zinc exhibiting broader more continuous patterns than the precious metals. Overall the soil geochemical anomalies indicate the potentially larger footprints of gold mineralized systems within the Buriticá tenements area, suggesting potential for the discovery of new veins with breccia-hosted and disseminated gold mineralization.

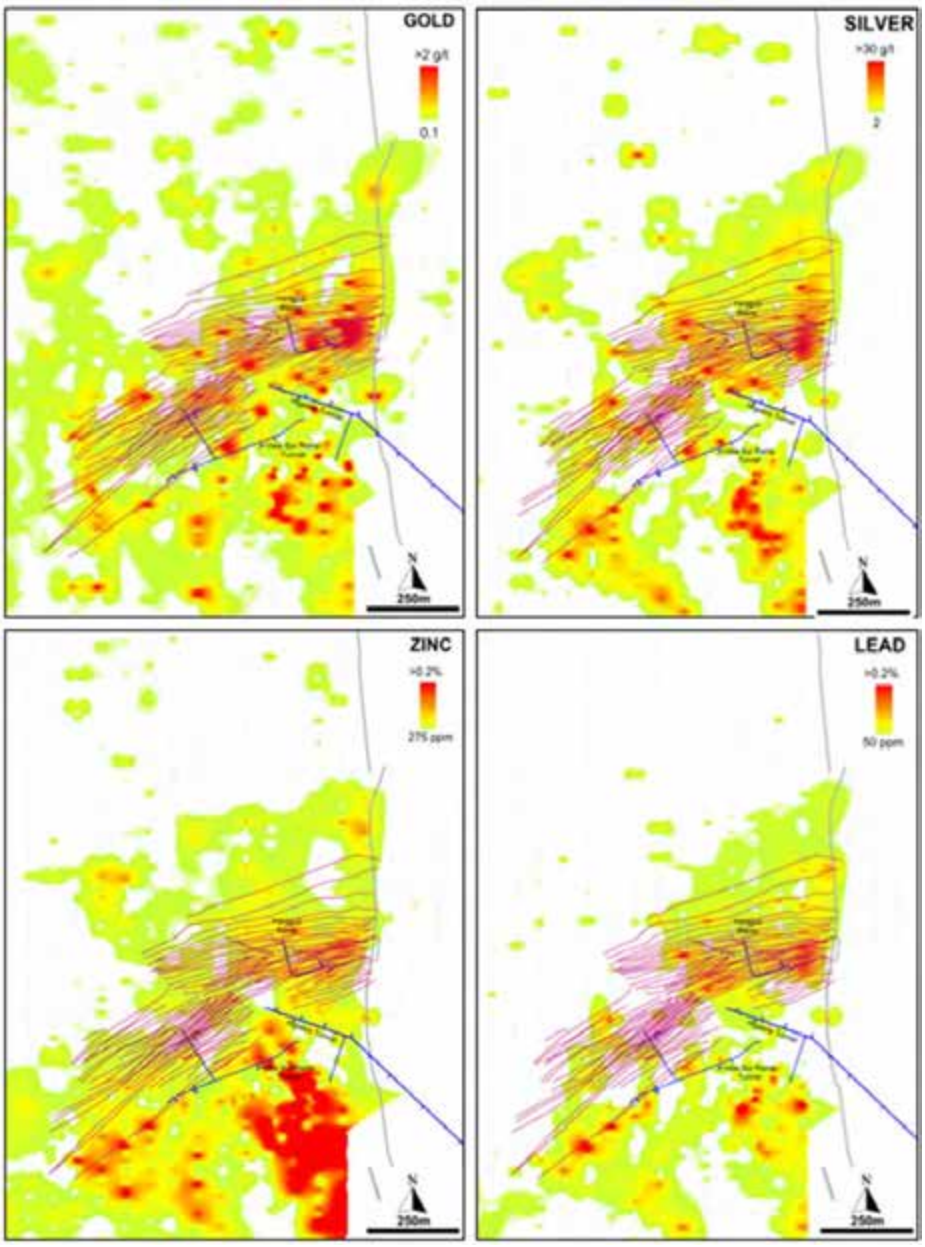
All the anomalies require follow-up investigation with more closely spaced soil sampling and trenching followed by drilling of priority targets if warranted. The strongly anomalous response in the southeast quarter of the Buriticá grid (La Mano) is partly due to downslope creep, but also due to extensive historical colluvial workings and some historical underground developments. The La Mano area is the subject of a current drilling program, commencing with drill hole BUSY228.

Figure 9.1: Soil Survey Location and Exploration Targets



Source: CGI, 2014

Figure 9.2: Soil Survey Gold, Silver, Zinc and Lead Maps



Source: CGI, 2015

9.2 Underground Channel Sampling 2012-2015

The upper levels of the Yaraguá system have some 4,500 m of historical and more recent development comprising drifts (largely along veins), cross-cuts, raises and other underground access, including historical stopes on the Murcielagos veins. These underground openings provide platforms for exploration and delineation drilling and for detailed geological and structural mapping as well as channel sampling. CGI continued the program of systematic sampling of the underground openings, including the Veta Sur and Higabra tunnels, sampling 1906 additional channels for 7,305 m. The channel samples were collected along an average 3 m spacing and 1.5 m across the drifts and raises. The (strike) length of sampled segments now totals 11,032 m (as of May 2015).

Table 9.1 lists significant sampling results since those reported in the 2012 Technical Report. CGI reported that channel sampling of San Antonio, Hangingwall and Centena veins averaged 3 m spacing and 1.9 m across the back of the drifts on Level 2. The Level 2 sampling is about 50 vertical metres below Level 1 sampling and demonstrates the strong vertical continuity of high grades in the vein sets, as well as continuity of grade along strike in central Yaraguá (Figure 9-3). The average widths are true widths and represent the assay intervals at a zero cut-off grade. In areas of vein splits or where development is “off-vein”, mineralization may extend into the walls of the underground openings hence the stated widths are minimal. Figure 9-4 illustrates the channel sampling at Veta Sur on the 51-Vein and the 62-Vein.

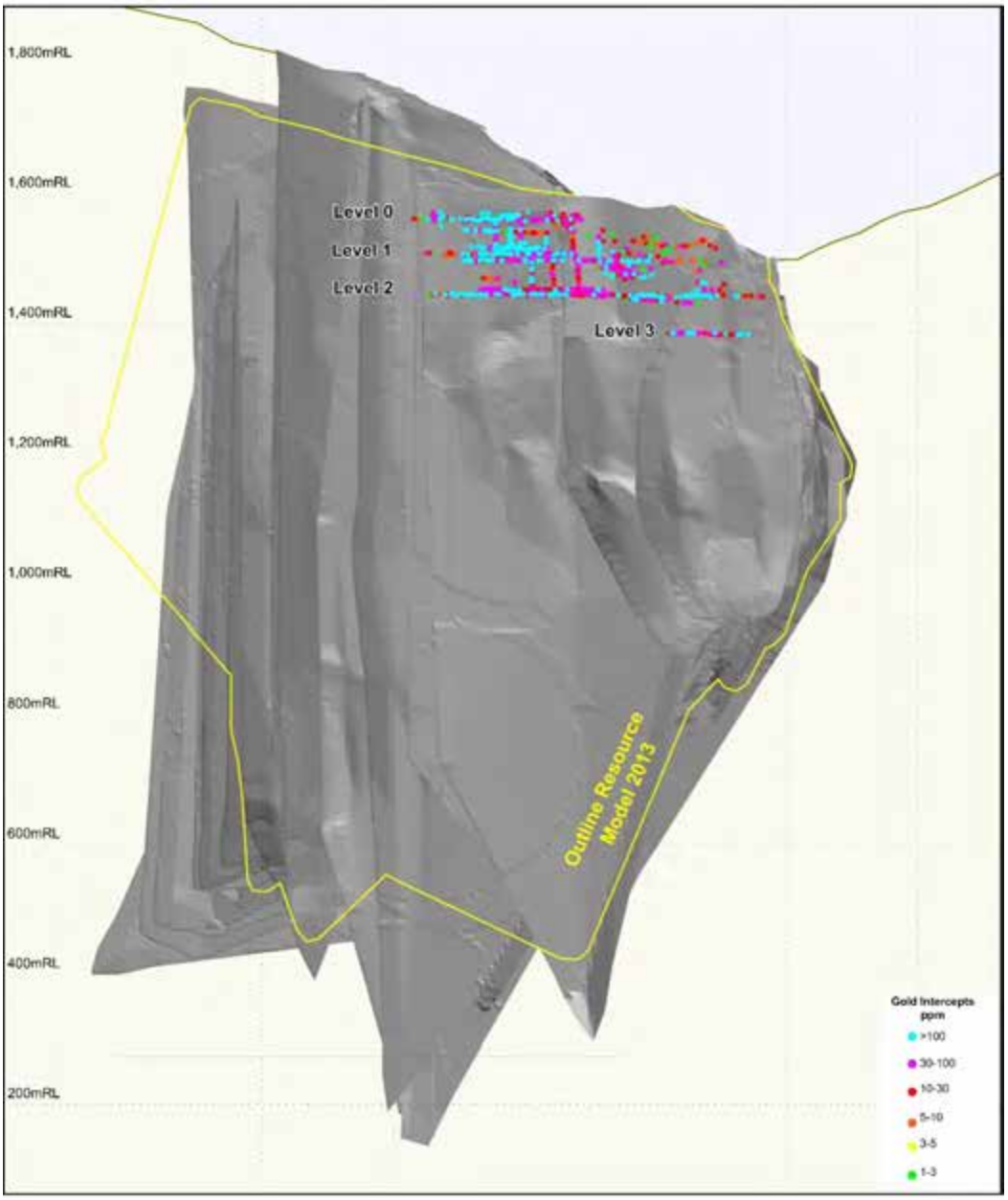
The underground sampling indicates that the high gold grades are continuous along strike and within the vertical range sampled for several of the Yaraguá vein sets. The continuity of the channel sampling data allows a higher confidence level in resource modeling. The channel samples are considered representative and have been incorporated into the data set for the mineral resource estimate (see Section 14).

Table 9.1: Channel Sampling – Significant Results

Vein Segment	RL (m)	Length (m)	Average width (m)	Au (g/t)	Ag (g/t)	Zn (ppm)
San Antonio Vein						
Level 0	1,559.97	278	0.65	18.09	32.36	11,262.83
Level0A	1,566.12	77.5	0.54	37.95	50.13	11,887.24
Level1I	1,496.44	275	0.40	27.02	43.03	9,094.57
Level1S	1,506.78	160	0.55	48.51	16.53	4,872.18
Level1A	1,515.99	83.5	0.41	75.85	93.54	19,976.96
Level2	1,442.48	495	0.43	14.35	75.02	2,677.80
Level2A	1,452.29	158	0.40	13.08	55.02	5,758.19
Level3A	1,383.00	37	0.48	6.36	0.00	0.00
Centeana Vein						
Level1I	1,496.44	70	0.46	12.37	155.32	6,350.45
Level2	1,442.48	221	0.62	10.25	72.36	8,104.33
HW Vein						
Level2	1,442.48	238	0.52	15.03	31.72	6,246.57
Level2A	1,452.29	47	0.40	17.02	22.13	8,795.51

Source: CGI, 2015

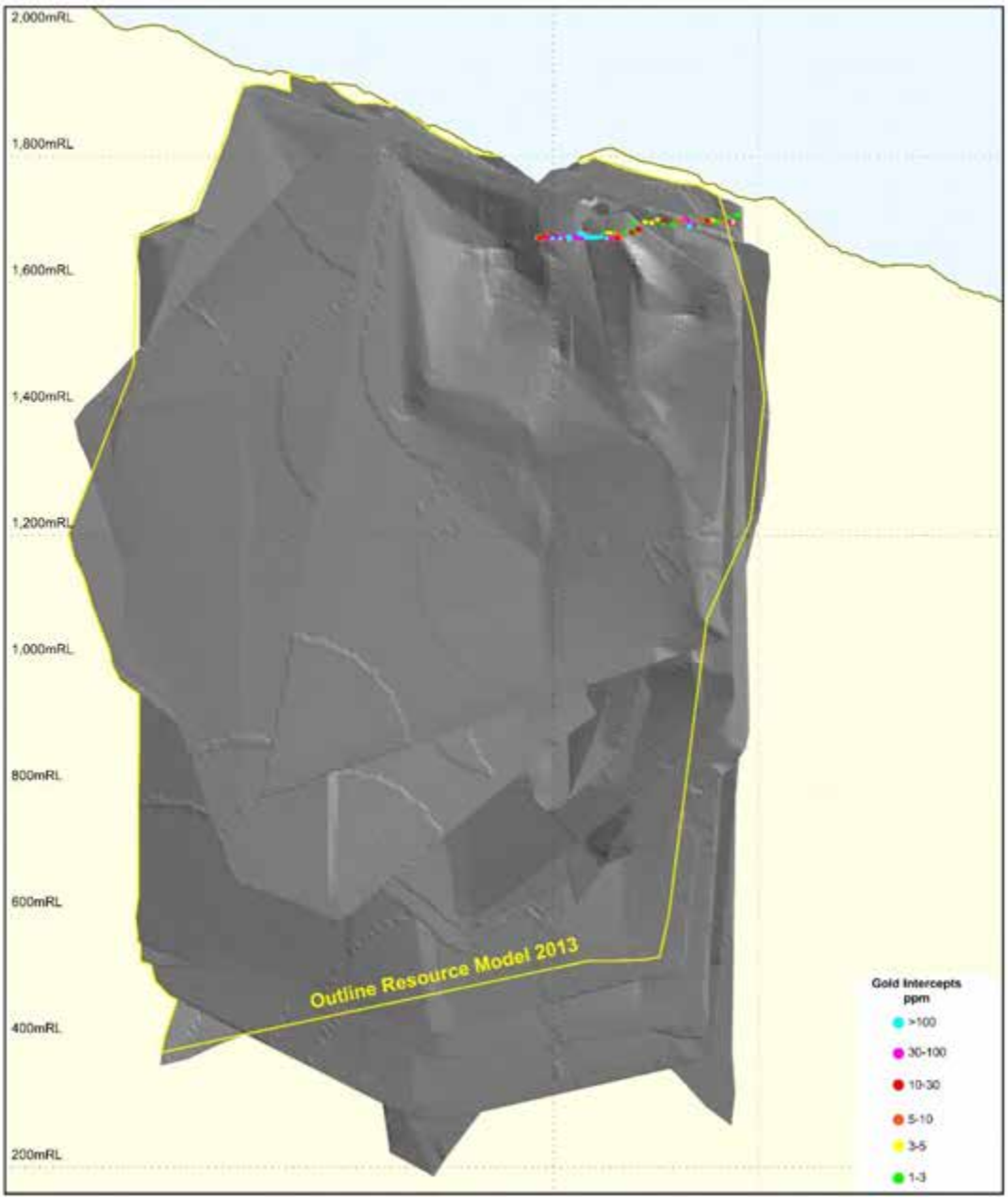
Figure 9.3 : Location of Yaraguá Channel Sampling and 2013 Resource Outline



Grey Shading is the 2015 Resource Envelope.

Source: CGI, May 2015

Figure 9.4: Location of Veta Sur Channel Sampling and 2013 Resource Outline



Grey Shading is the 2015 Resource Envelope

Source: CGI, July 2015

10 Drilling

In December 2007, CGI commenced diamond drilling at the Buriticá Project. Table 10.1 lists the drilling and sampling statistics used in the 2011 mineral resource estimate which includes drilling by CGI since commencement of drilling in 2007. All surface and underground drilling was conducted by diamond core equipment, with mainly HQ size core, some holes of which were reduced to BQ. All drill holes collars were located by surveying. Downhole surveys utilized Reflex EZ-Trac instruments.

Since June 2011, drilling has continued with ten drills at the Buriticá Project at a rate of 5,000 m per month. Table 10.2 lists the drilling and sampling statistics conducted from July 2011 to June 30, 2012. Table 10.3 lists the drilling and sampling statistics used in the 2012 mineral resource estimate which includes drilling by CGI since commencement of drilling in 2007.

Since June 2012, drilling continued with eight drills at the Buriticá Project throughout the rest of 2012 at a pace of 6,000 m per month. Table 10-4 lists the drilling and sampling statistics conducted from June 2012 to December 31, 2013. From late 2012, all drill hole collars were located by surveying and downhole surveys utilized Reflex GYRO and ACTIII instruments.

Drilling in 2014 continued with seven drills at a pace of 5,500 m per month. Directional drilling was a key element of the drilling program. In this drilling method, several “daughter” holes were deviated (in inclination and azimuth) from different positions/elevations in a “mother” hole.

The particular advantages of the directional drilling were:

- Substantial time and cost savings due to reduced drill meterage and improved drilling productivity compared with conventional drilling; and
- The ability to better target key areas, control drill deviations and achieve better angles of attack on mineralized zones.

Drilling in 2015 focused on converting Inferred resource into the Measured and Indicated categories, as well as growing the overall mineral resource (See Figure 10.1). Three drills were used at a pace of 4,100 m per month.

Tables 10.1 to Table 10.5 list the drilling and sampling statistics used in the 2014 Technical Report which include drilling by CGI since commencement of drilling in 2007. Recent drilling is shown in Figure 10.1.

Table 10.5 lists the drilling and sampling statistics used in the 2014 mineral resource estimate which include drilling by CGI since commencement of drilling in 2007. Table 10.6 lists total drilling and samples used in the current resource up to May 11, 2015.

Table 10.1: Database as of June 2011

Drilling type	Area	Drill holes/ Channels	Samples	Metres
Surface DH (BUSY-, BUSM-)	Yaraguá and Veta Sur	178	30,205	40,421
Underground DH (BUUY-)	Yaraguá	57	8,990	11,979
Channel samples (CH-)	Yaraguá	682	2,051	1,605
Total		**917**	**41,246**	**54,005**

Source: CGI, 2015

Table 10.2: Database July 2011 to June 30, 2012

Drilling type	Area	Drill holes/ Channels	Samples	Metres
Surface DH (BUSY-)	Yaraguá and Veta Sur	88	25,218	32,386
Underground DH (BUUY-)	Yaraguá	26	6,401	7,891
Channel samples (CH-)	Yaraguá	219	475	294
Total		**333**	**32,094**	**40,571**

Source: CGI, 2015

Table 10.3: Database as of June 30, 2012

Drilling type	Area	Drill holes/ Channels	Samples	Metres
Surface DH (BUSY-, BUSM-)	Yaraguá and Veta Sur	276	81,105.04	92,366.01
Underground DH (BUUY-)	Yaraguá	87	16,943.12	17,900.80
Channel samples (CH-)	Yaraguá	995	2,331.56	2,334.92
Total		**1358**	**100,296.14**	**112,601.73**

Source: CGI, 2015

Table 10.4: Database July 2012 to December 31, 2013

Drilling Type	Area	Drill Holes/ Channels	Samples	Metres Drilled
Surface DH (BUSY-)	Yaraguá and Veta Sur	73	59,758.96	54,489.55
Underground DH (BUUY-)	Yaraguá	142	38,984.88	38,225.95
Channel samples (CH-)	Yaraguá	1183	1,395.44	1,399.88
Total		**1398**	**100,139.28**	**94,115.38**

Source: CGI, 2015

Table 10.5: Database as of December 31, 2013

Drilling Type	Area	Drill Holes/ Channels	Metres Sampled	Metres Drilled
Surface DH (BUSY-, BUSM-)	Yaraguá, Veta Sur and Laurel	349	140,864	146,855.56
Underground DH (BUUY, BUUS)	Yaraguá	229	55,928	56,126.75
Channel samples (CH-)	Yaraguá	2,178	3,727	3,734.80
Total		**2,756**	**200,519**	**206,717.11**

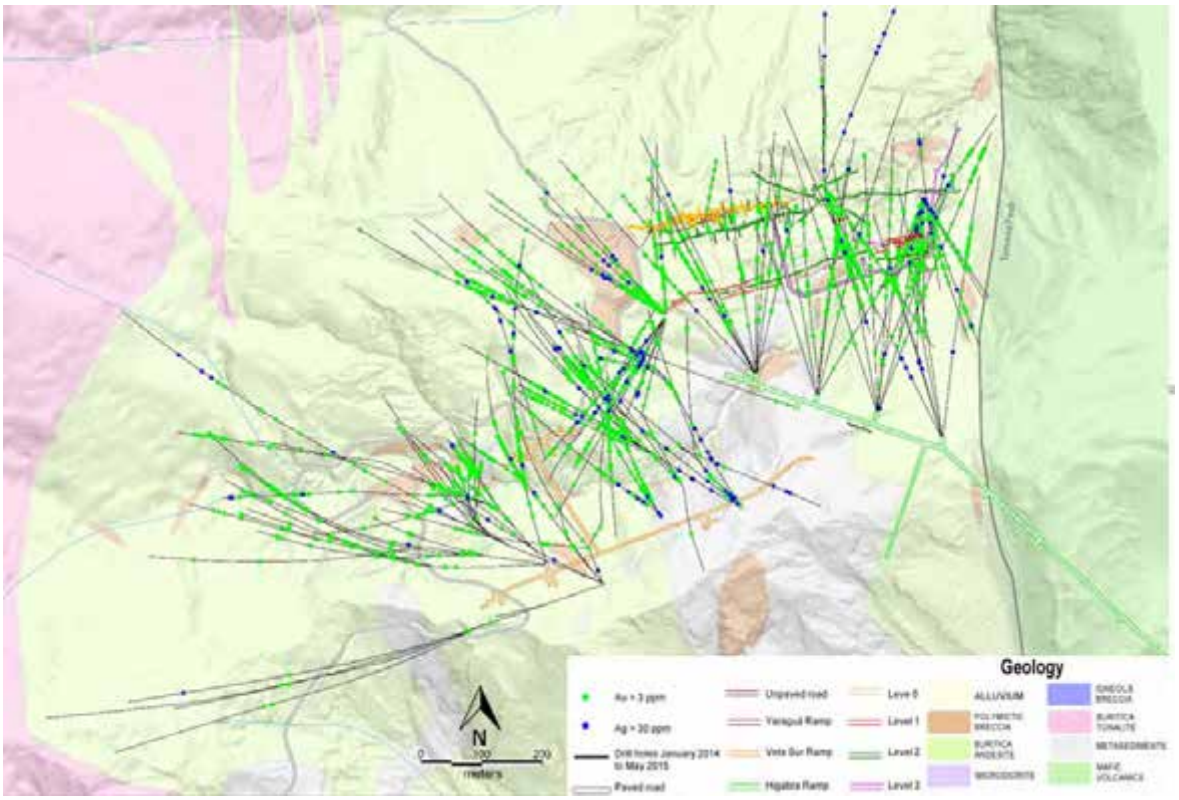
Source: CGI, 2015

Table 10.6: Verified Buriticá Database as of May 11, 2015

Drilling Type	Area	Drill Holes/ Channels	Number Samples	Metres Sampled
Surface DH (BUSY-, BUSM-)	Yaraguá, Veta Sur and Laurel	391	141,775	162,664
Underground DH (BUUY-)	Yaraguá and Veta Sur	345	115,011	108,339
Channel samples, surface samples CH-)	Yaraguá and Veta Sur	4084	11,032	7,215
Total		**4,820**	**267,818**	**278,218**

Source: CGI, 2015

Figure 10.1: Buriticá Geology and Recent Drill Locations



Source: CGI, 2015

10.1 Type, Extent, Procedures and Results

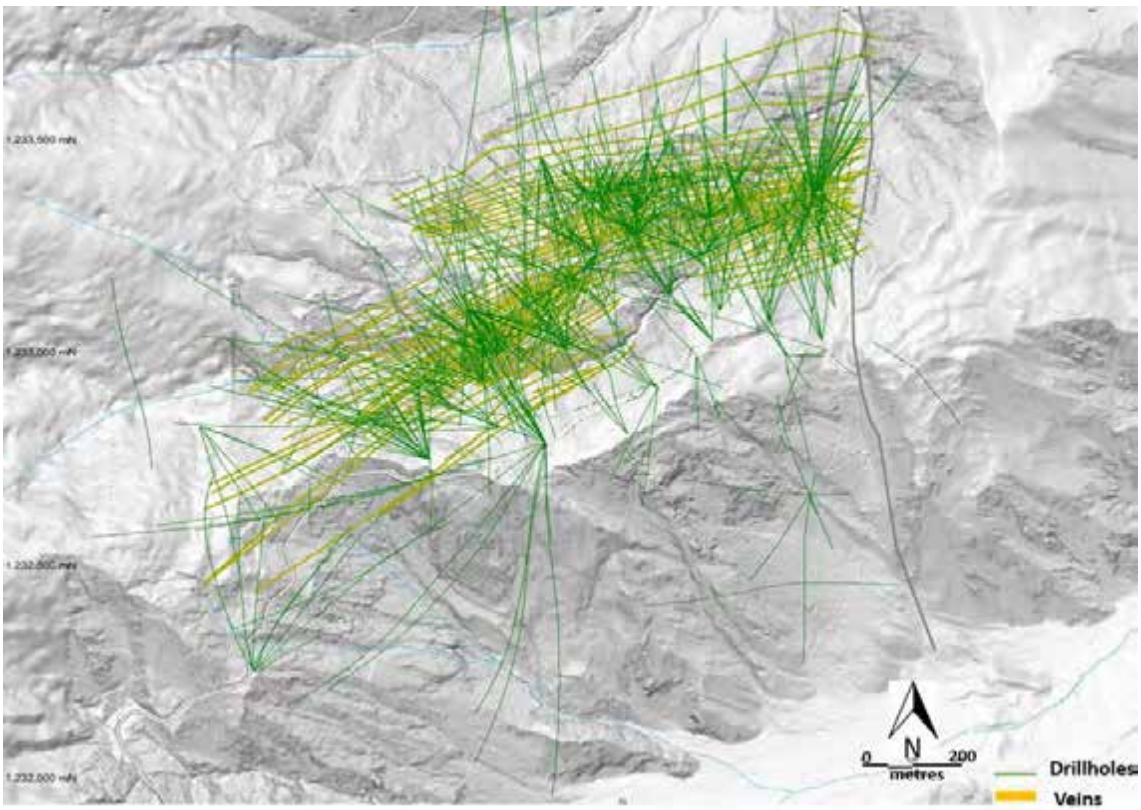
The complete significant drilling results for the Yaraguá and Veta Sur area drilling are not shown in this report as they are considered too voluminous to include for a Project at this stage.

Drill hole orientation and collar location are given in Figure 10.1 and Figure 10.2. The location, gold grades and true horizontal thickness/width of all drill and channel sample composites used in vein domain modeling are portrayed in long section views (by vein) in Figure 10.3 and Figure 10.4.

10.1.1 Drill Collar Plan and Representative Section

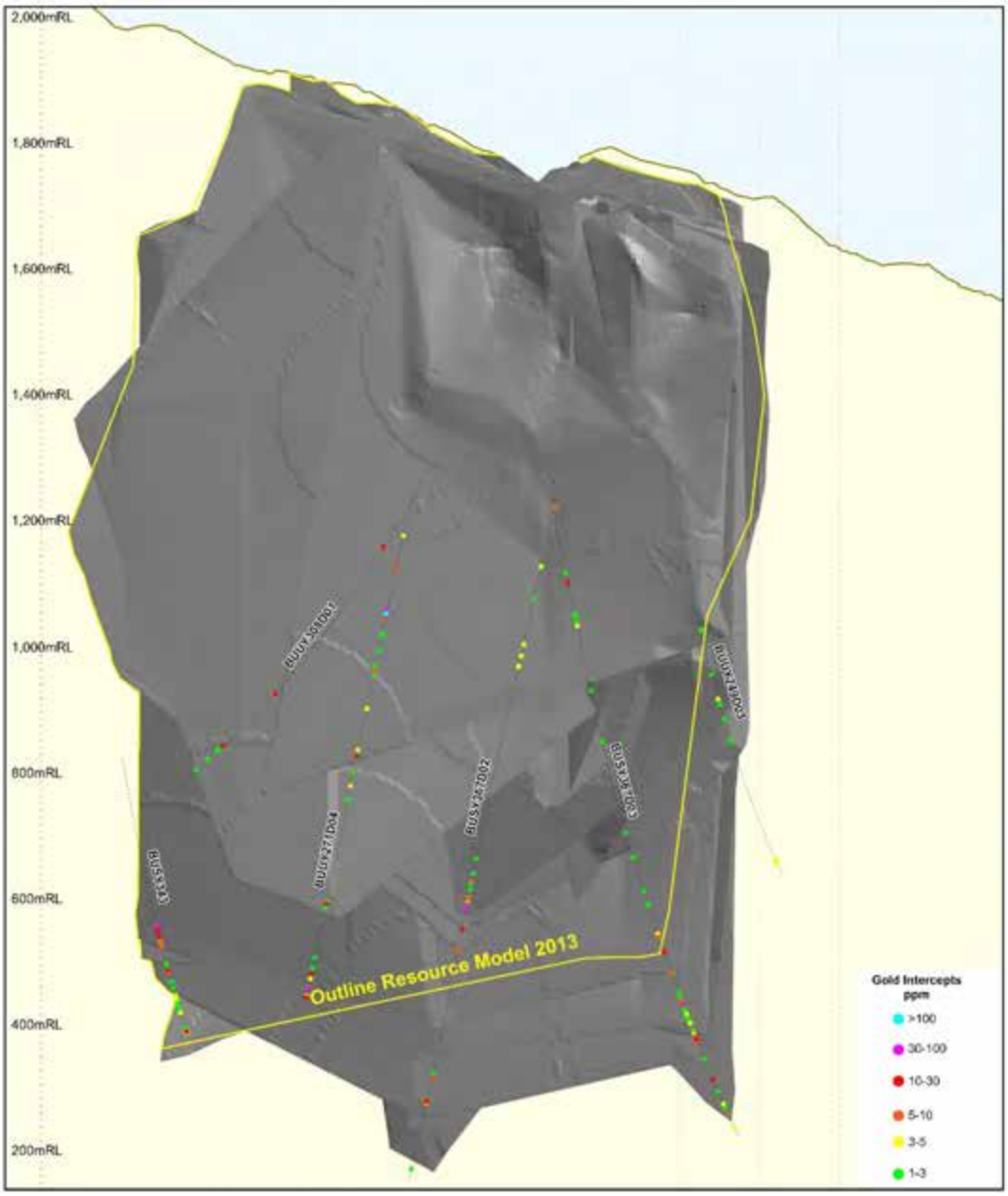
Figure 10.2 shows, in plan view, the distribution of drill holes utilized in vein domain modeling, along with traces of modeled vein domains at the 1,200 m elevation. Figure 10.3 and Figure 10.4 show representative deep drilling intercepts in long sectional view.

Figure 10.2: Project Drill Collar Plan and Vein Traces at 1,200 m RL



Source: CGI, 2015

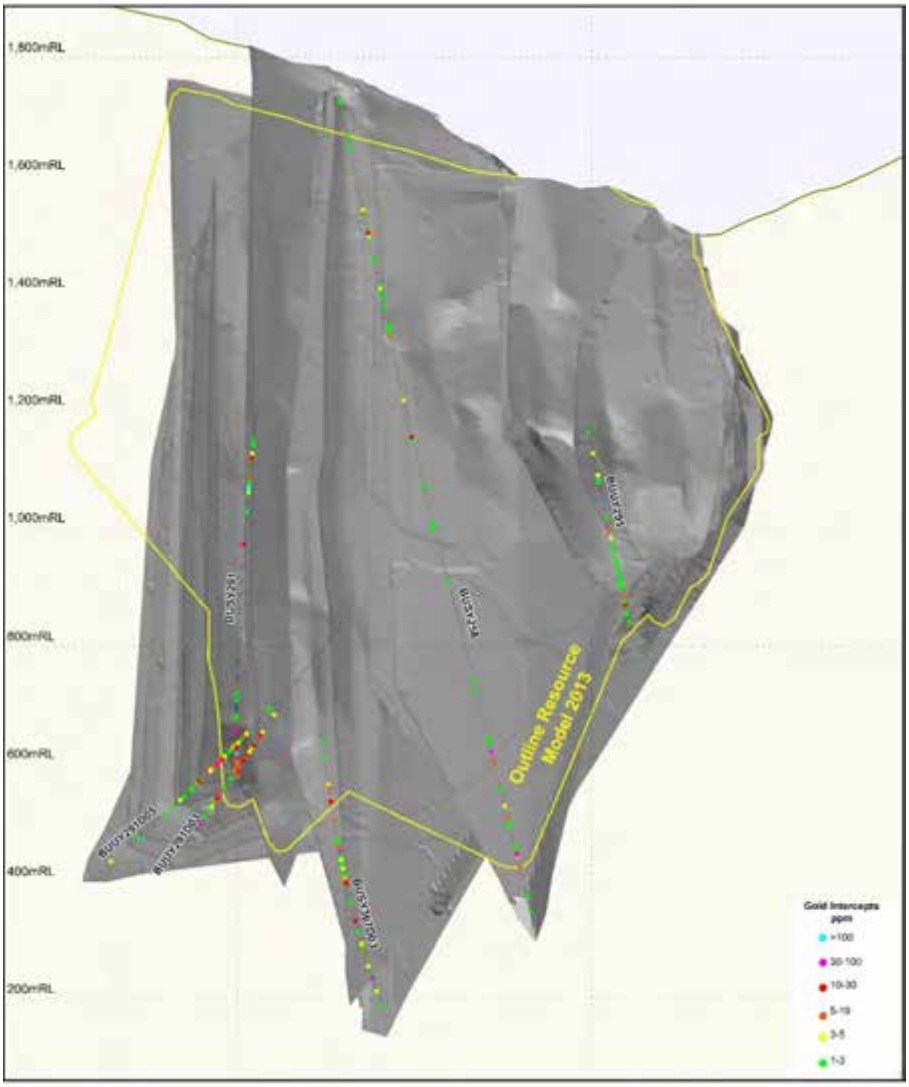
Figure 10.3: Location of Deep Drill Holes, Veta Sur



Yellow outline 2013 resource envelope, grey shading is the 2015 Resource envelope

Source: CGI, 2015

Figure 10.4: Location of Deep Drill Holes, Yaraguá 2013 Resource Envelope



Grey shading is the 2015 Yellow Outline 2013 Resource Envelope, Grey Shading is the 2015 Resource Envelope.

Source: CGI, July 2015

10.2 Accuracy and Reliability

MA concludes that core recovery results have no material impact on the mineral resource estimation. All collars were total-station surveyed and a Reflex EZ-Trac and GYRO survey tool were used for downhole surveys.

Table 10.7 lists core recovery statistics. MA concludes that core recovery results have no material impact on the mineral resource estimation. All collars were total-station surveyed and a Reflex EZ-Trac and GYRO survey tool were used for downhole surveys.

Table 10.7: Core Recovery Statistics

Recovery	% of Total
<80% recovery	9%
80-99% recovery	20%
>99% recovery	71%
Total	**100%**

Source: CGI, 2015

11 Sample Preparation, Analyses and Security

11.1 Drill Core Sample Preparation

Drill core transport, logging, sampling and storage are carried out at CGI's facility on site. After core is retrieved from the core barrel and placed in core trays (Figure 11.1), the core boxes are taken from the drill site to the logging shed where initial photos are taken before the core has been washed. The lengths are marked in core trays and wooden spacers inserted with downhole lengths added by a technician. The core trays are laid out in order of depth and the core loss is calculated while correct depths are marked in the tray.

Figure 11.1: Photographs of Core Sample Handling and Logging

Logging Area

Core Arrival

Core logging and Storage

Core logging and Storage

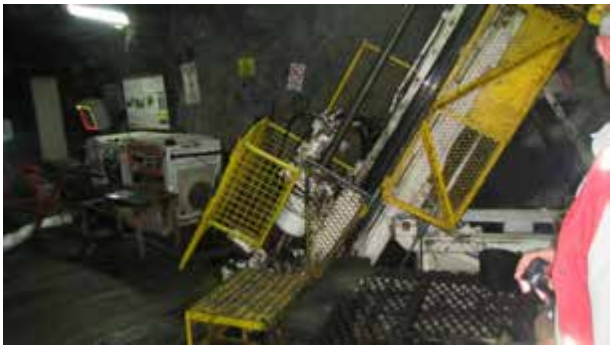
Emphasis on Safety

Source: MA Site Visit, 2015

Figure 11.2: Photographs of Buriticá Underground Drilling and Sample Handling

Small underground drill rig – Yaraguá - 2011

Drill tray for drill core samples – Yaraguá - 2011



Large directional diamond drill – Veta Sur - 2015

Source: MA site visit, 2011 and 2015

Logging is then entered into the LOGCHIEF system directly for lithological, alteration, mineralogy and geotechnical data. Sample intervals are then chosen based on lithology, alteration and mineralogy, with lithology changes, veins, alteration changes or anything else of note used to choose boundaries of sampling.

Generally the smallest interval chosen is 40 cm while unmineralized country rock will be sampled at 1.5 m intervals. Once sampling intervals have been selected, the core trays are photographed with sample intervals shown by stickers indicating sample boundaries, downhole lengths, and drill hole names. Drill core is then marked for cutting by a geologist. Cutting of the core is carried out on site at the core logging shed.

Once logging is complete, the core trays are transported to the site sample prep area. The half cut (or quarter cut for duplicates) samples are put into a bag with a barcode sticker which records the sample number, and then these are placed into a second bag, also with a sticker marking the sample number (Figure 11.3).

Samples are then wrapped and sealed with packing tape and the geologist or technician signs off the sample sheet. A geologist will handle the sampling if there is mineralization, otherwise a technician may prepare an apparently unmineralized sample.

For the channel samples, there are two forms of sampling which depend on the rock´s hardness, if the rock is fragile, the sample is extracted by sledgehammer and chisel, but if the sample is hard and out of fracture surface, it is necessary to take it out by a cutting disc, this tool permits clean cuts and continuity of the channel. Each wrapped sample is then placed into a polyweave bag ("costal"). Empty bags with the sample barcode sticker are put into the bag for laboratory pulps and standards. These fabric bags are filled according to the weight of the samples; there are typically around seven samples per costal. The costal is sealed with cable ties with the corresponding batch number and costal number written on it. The batch and costal numbers are also written on each polyweave bag.

The batch numbers must match the numbers on the sample sheet, and in addition a barcode sticker matching each sample that is in the bag must also be placed on the sample sheet. Each sample sheet records the hole depths, costal number, batch number, sample number, and the type of sample it is (standard, duplicate, half core, etc.).

Figure 11.3: Photographs of Core Dispatch and Storage

Original Assay Sheets

Sample Ledgers

Multi-level Core Storage

Core Boxes in Racks

CRM Standards and blanks

Samples bagged for dispatch

Source: MA site visit, 2011 and 2015

11.2 Channel Sample Preparation

The channel sampling is performed by technicians under a geologist's supervision Figure 11.4 and Figure 11.5).

Figure 11.4: Photographs of Buriticá Underground Channel Sampling

Vein zone, level 3

Level 2 access

Marking sample location and measuring distance from nearest survey

Marking sample zones.

Sample collection

Sample number marked at sample location

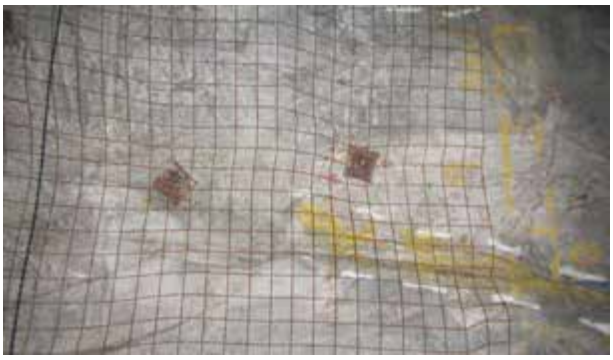
Source: MA site visit, 2011 and 2015

The samples are collected in a bucket covered by a plastic sheet that is cleaned between every sample. Sample positions are chosen by the geologist and surveyed, typically every three metres along strike in underground drifts or every three vertical metres in raises. Three samples are generally taken across the full width of the underground opening. All relevant data such as dip, structure, lithology mineralogy, size of sample etc. is logged into a book similar to the drill hole logging forms. Each sample is double rapped with barcodes inserted into the bags for recording purposes. Once the sample is taken, it is sent to the sample preparation shed to await transport and the logging data is entered into the LOGCHIEF system. Underground sampling data are stored in the database as pseudo drill holes to facilitate 3D modeling.

Figure 11.5: Photographs of Veta Sur Underground Channel Sampling

Vein wall samples, Veta Sur

Vein wall samples, Veta Sur



Samples of backs, Veta Sur

Marking sample location

Source: MA site visit, 2015

11.3 Sample Security

Each day the drill core samples are transported from the site in metal boxes properly marked with the drill hole and box number to the "PEÑITAS" industrial zone, guarded by CGI personnel. Each box is carefully tied with plastic straps for transportation. Once the core trays are laid out on tables with the straps and lids are removed in preparation for logging and core photographs.

The core samples are measured (marked–up), logged, and labelled following the internal procedures that have been endorsed by outside consultants. These samples are then cut and packed into size 8 double plastic bags, which were previously marked with stickers showing a sample number assigned by the geologist. Before the batches are sent, geologists and technicians prepare a batch checklist to track the movement of the material, identify the number of samples, batches, and Quality Control (QC) samples, with its type, and the costal number. At this stage, the checklist must be signed by the geologists, security guard, and the driver of the vehicle. When this process is completed, the batches are then sent to Medellín. Here, the warehouse foreman receives the batches from the driver, and must check against the batch checklist, and sign to verify the contents of the batches. The foreman is the individual responsible to hand deliver each of the samples to the ALS Medellín laboratory.

Upon delivery, the ALS shift supervisor verifies that all samples as specified in the laboratory request sheet are the same as delivered, then signs for their receipt. These samples are logged in the internal system called "Webtrieve" (used globally by ALS clients) and assigned a work order number known as the internal way lot. Every time a sample goes through this process, it is followed by the system indicating its stage.

The samples go through the initial preparation process at ALS Colombia (crushed, split, and pulverized) and the pulp is sent to ALS Peru (as defined below) in Lima. This pulp is packed in a paper bag and coated with plastic, then sent in heavy gauge cardboard boxes with ALS tape and coded security straps, which identifies those boxes if any that have been opened during transit between ALS Medellín and ALS Peru by customs. The leftover pulp and coarse rejects are sent to the CGI warehouse in Itagui within 45 days of the date of issue of the certificate.

11.4 Sample Analyses

11.4.1 Preparation

Sample preparation is conducted in the ALS Laboratory in Colombia ("ALS Colombia") located at Bodegas San Bartolome Bodega 3, Carrera 48B No 99 Sur - 59, La Estrella Medellín. ALS Colombia is independent from CGI.

CGI utilizes an industry standard QA/QC program. HQ and NQ diamond drill core is sawn in half with one half shipped to a sample preparation lab in Medellín run by ALS Colombia and the other half stored in the CGI warehouse. 100% of BQ diameter drill samples are shipped.

11.4.2 Analysis

After preparation at ALS Colombia, the samples are then shipped for analysis to the ALS certified assay laboratory in Lima, Peru (ALS Peru).

The remainder of the core is stored in a secured storage facility for future assay verification. Blanks, duplicates and certified reference standards are inserted into the sample stream to monitor laboratory performance. A portion of the samples are biannually check assayed at SGS Colombia, which is certified ISO 9001:2008, including sample preparation and laboratory analysis for fire assay and multi-element. Blanks used were assayed by Actlabs Colombia which has ISO 17025 with CAN-P-1579 accreditation.

11.4.3 Laboratory Independence and Certification

ALS Peru is independent from CGI and has the following accreditation:

ISO 9001:2008 certification by IQNET, The International Certification Network, for chemical analysis of geological samples and products of its industrial processing chemical analysis of environmental samples from the mining and energy industries.

ISO/IEC 17025:2005 Accreditation by the Standards Council of Canada as a Testing Laboratory.

11.5 Quality Control

Quality Assurance (QA) concerns the establishment of measurement systems and procedures to provide adequate confidence that quality is adhered to. Quality Control (QC) is one aspect of QA and refers to the use of control checks of the measurements to ensure the systems are working as planned. The QC terms commonly used to discuss geochemical data are:

- Precision: how close the assay result is to that of a repeat or duplicate of the same sample, i.e. the reproducibility of assay results;
- Accuracy: how close the assay result is to the expected result (of a certified standard); and
- Bias: the amount by which the analysis varies from the correct result.

11.5.1 QC Program

According to CGI's QA/QC procedures, the following samples are taken or inserted into the sample stream (both drill core and channel).

- Certified Reference Materials (CRM) supplied by Geostats: low, medium and high-grade Au, low-grade Ag and low-grade Zn are inserted at a planned rate of one every 159 samples. (Table 11.1 and Table 11.2);
- Field Duplicates Samples: every 19th sample is cut twice into ¼ core. Both samples are inserted into the sampling stream and prepared and assayed like any other sample;
- The result can be examined as a duplicate sample. This sample is used to monitor samples batches for poor sample management, contamination and tampering and laboratory precision;
- Coarse and pulp duplicates (DUG and "DUP"): every 19 samples the geologist chooses the sample for reanalysis at ALS. A summary the insertion rates for all duplicate samples submitted is shown in Table 11.3;

- Blanks: CGI used Coarse Blanks (BKG) and Fine Blank (BKF) assayed at Actlabs Colombia laboratory with low level of Au, Ag and Zn. These blanks are inserted as one BKF and one BKG every 19 samples. A summary the insertion rates for all Blank samples submitted is shown in Table 11.4 and Table 11.5.

At six monthly intervals CGI sends 5% of the pulps (Au>1ppm or Ag>50ppm) to the SGS Colombia laboratory for check analysis using the same analytical method as ALS.

A portion of the samples are periodically check assayed at the ACME laboratory in Vancouver and the Inspectorate laboratory in Reno, Nevada.

In addition, the independent laboratory also conducts its own internal QA/QC consisting of CRM testing, duplicates assaying and repeats.

Table 11.1: Certified Reference Material for Gold

Standard Category	Standard ID	Count	Totals
Low-Grade Au	ST_G912-5	521	2,222
	ST_G310-4	166	
	ST_G310-5	1520	
	ST_G307-8	15	
Moderate Grade Au	ST_GBMS304-6	896	2,303
	ST_G905-7	782	
	ST_G906-8	401	
	ST_SN74	224	
High-Grade Au	ST_G901-8	204	277
	ST_G310-10	55	
	ST_SP37	5	
	ST_SQ70	13	
Total Au standards inserted			**4,802**
Total drill and channel samples			**95,666**
Ratio of Au CRM to original samples			1:19
Insertion rate			5%

Source: CGI, 2015

CGI has adopted QA/QC protocols for drill core and channel sampling that meet the mineral industry standard and have acceptable insertion rates. The protocols are considered adequate for the determination of accuracy and precision.

Table 11.2: Certified Reference Material for Silver and Zinc

Standard ID	# of Inserts
GBMS304-6	896
GMS399-6	215
ST_SN74	224
ST_SQ70	13
Total	1348
Total drill and channel samples	95666
Ratio of Ag-Zn CRM to original samples	0.0902778
Insertion rate	1.50%

Source: CGI, 2015

Table 11.3: Duplicate Samples

QC_Category	DH -CH Sample Count	QC Sample Count	Ratio of QC Samples to DH Samples	Insertion Rate (%)
Field Duplicate	95,666	5,008	1:19	5
Crush Duplicate Coarse	95,666	5,025	1:19	5
Lab Pulp Split (DUP)	95,666	5,030	1:19	5

Source: CGI, 2015

Table 11.4: Blank Samples

Blank	Drill and Channel Sample Count	Blank Sample Count	Ratio of Blank Samples to Original Samples	Insertion Rate (%)
BKF	95,666	5,082	1:19	5
BKG	95,666	5,082	1:19	5

Source: CGI, 2015

Table 11.5: Blanks and CRM Labs

Standard Type	DH Sample Count	Standard Type Count	Standard Sample Count	Ratio of QC Standard to DH Samples
LAB	95666	36	29403	1:03
BK	95666	2	22748	1:04

Source: CGI, 2015

11.5.2 QC Program Results

The complete set of control charts for drill core and channel sampling QA/QC results are maintained and have been sighted by the QP.

11.5.2.1 Standards Results – Accuracy

Accuracy is identifying the true grade of a sample, often achieved by submitting CRM commonly referred to as standards.

CGI used CRMs provided by independent laboratories Geostats Pty Ltd. CGI selected 12 gold CRMs to approximate cut off, low grade, moderate grade and high-grade gold material (Table 11.6) and four silver and two zinc CRMs (Table 11.7).

CRM results are analyzed by examining returned assays relative to the CRMs expected, or certified value on a graph known as a control chart. Results within ± two standard deviations (2SD) of the expected value are deemed acceptable. Individual results between ± 2SD and ± 3 standard deviations (3SD) of the expected value are acceptable, but monitoring is required. Multiple results between ± 2SD and ± 3SD of the expected value should be confirmed, and indicate potential problems with bias. Any results outside ± 3SD are considered as 'fails' and should be examined for problems with sample number allocation and followed up with re-assaying of the entire batch if necessary.

CRM performance gates listed in Table 11.6 and Table 11.7 are based on ± 2SD and ± 3SD. the performance of the CRM.

Table 11.6 shows a summary of the CRM results. In the case of ST_G310-4 standard low grade, the highest group bias is observed. This failure is likely due to the type of assay used, as this is much more stable in higher values.

Table 11.6: CRM used by CGI for Gold Analysis

Standard ID	Expected Value (Fire Assay)	Lower Limit 2SD	Upper Limit 2SD	Lower Limit 3SD	Upper Limit 3SD
ST_G912-5	0.38	0.34	0.42	0.32	0.44
ST_G310-4	0.43	0.4	0.37	0.46	0.49
ST_G310-5	1.01	0.96	0.91	1.06	1.11
ST_G307-8	1.99	1.83	2.15	1.75	2.23
ST_G905-7	3.92	3.77	3.62	4.07	4.22
GBMS304-6	4.58	4.39	4.2	4.77	4.96
ST_G906-8	7.24	6.97	6.7	7.51	7.78
ST_SN74	8.98	8.54	9.42	8.31	9.65
ST_SP37	18.14	17.38	18.9	17	19.28
ST_SQ70	39.62	37.92	41.32	37.07	42.17
ST_G901-8	47.24	44.14	50.34	42.59	51.89
ST_G310-10	48.53	40.18	51.87	43.52	53.54

Source: CGI, 2015

Table 11.7: CRM used by CGI for Silver and Zinc Analysis QC

Standard ID	Element	Expected Value	Lower Limit 2SD	Upper Limit 2SD	Lower Limit 3SD	Upper Limit 3SD
GBMS304-6	Ag	6.1	4.5	7.7	3.7	8.5
GBMS304-6	Zn	1265	1085	1445	995	1535
GMS399-6	Ag	15.5	13.3	17.7	12.2	18.8
GMS399-6	Zn	2488	2160	2816	1996	2980
ST_SN74	Ag	51.5	48.5	54.5	47	56
ST_SQ70	Ag	159.5	146.1	172.9	139.4	179.6

Source: CGI, 2015

Table 11.8: Outcome for Gold, Silver and Zinc CRM Results 01-2014 to 05-2015

Standards Type	Standard ID	Element	Count	Fails >3DS	Percent of CRM (%)	Percent Bias (%)
Low-Grade Au	ST_G912-5	Au	521	3	0.58	-4.84
	ST_G310-4	Au	166	0	0.00	-5.77
	ST_G310-5	Au	1520	0	0.00	-1.66
	ST_G307-8	Au	15	0	0.00	-0.37
Moderate Grade Au	ST_G905-7	Au	782	0	0.00	0.15
	GBMS304-6	Au	896	1	0.11	0.67
	ST_G906-8	Au	401	0	0.00	-0.48
	ST_SN74	Au	224	3	1.34	-1.13
High-Grade Au	ST_SP37	Au	5	0	0.00	-2.15
	ST_SQ70	Au	13	0	0.00	0.10
	ST_G901-8	Au	204	1	0.49	-0.32
	ST_G310-10	Au	55	5	9.09	-6.13
Multi-Element	GBMS304-6	Ag	896	0	0.00	0.69
	GBMS304-6	Zn	896	0	0.00	2.92
	GMS399-6	Ag	215	0	0.00	-0.60
	GMS399-6	Zn	215	0	0.00	2.55
	ST_SN74	Ag	224	0	0.00	0.81
	ST_SQ70	Ag	13	0	0.00	-0.46

Source: CGI, 2015

11.5.2.2 Field Blanks - Contamination

Field blanks are obtained (in batches of 500 kg) from a road aggregate quarry within the vicinity of the Buriticá Project area. Table 11.9 shows the expected value and upper limit of the two blanks (fine blank BKF and coarse blank BKG) used by CGI. For BKF the upper limit is defined as three times the detection limit and for BKG the upper limit is defined as five times the detection limit.

BKF and BKG show good performance for gold, silver and zinc. Table 11.10 summarizes the results for the blanks. All results where the value exceeds the upper limit are considered to have been caused by contamination from the previous sample material which had a very high grade. In MA's opinion the extent of contamination does not exceed values that could affect the resource estimate.

Table 11.9: Expected Values of Blanks used by CGI

Blanks	Element	Expected Value (ppm)	Upper Limit (ppm)
BKF	Au	0.02	0.06
BKF	Ag	0.01	0.3
BKF	Zn	2	75
BKG	Au	0.02	0.1
BKG	Ag	0.01	0.5
BKG	Zn	2	125

Source: CGI, 2015

Table 11.10: Blank Results

Blanks	Element	Count	Fail	Percent Contamination (%)
BKF	Au	5082	3	0.06
BKF	Ag	5082	31	0.61
BKF	Zn	5082	1	0.02
BKG	Au	5082	7	0.14
BKG	Ag	5082	15	0.30
BKG	Zn	5082	1	0.02

Source: CGI, 2015

11.5.2.3 Field Duplicates – Precision and Bias

Duplicate results are analyzed by comparing the values obtained for a pair of samples and determining the average difference or error. Duplicate results have been evaluated using the formula:

Relative Error = 100* (Repeat Value –Original Value)/ (Repeat Value +Original Value)/2

A duplicate is considered to fail if they deviate by ±30% (Field Duplicate), by ±20% (Coarse Duplicate) by ±10% (Pulp Duplicate). Pairs with results below five times the LDL are not considered to have failed. The performance of the duplicate samples and Table 11.11 summarizes the results. MA notes that most failures found in pulp duplicates for Au and Ag are within the values below the cut-off grade considered in the resource estimate. For Ag, 99.9% of failed pairs is less than 50 ppm and for Au, 85% of failed pairs are less than 1 ppm.

Table 11.11: Duplicate Results

Duplicate Types	Element	Count Duplicates	Pairs Fail	Fail (%)
DU	Au	5008	391	7.81
DU	Ag	5008	737	14.72
DU	Zn	5008	545	10.88
DUG	Au	5025	753	14.99
DUG	Ag	5025	468	9.31
DUG	Zn	5025	47	0.94
DUP	Au	5030	1529	30.40
DUP	Ag	5030	1426	28.35
DUP	Zn	5030	163	3.24

Source: CGI, 2015

In MA's experience, compromised duplicate performance results are not uncommon for vein gold/silver deposits of this mineralization style and are a reflection of the variability of the mineralization.

11.5.2.4 Laboratory QA/QC

Laboratory checks are carried out every six months using pulps. Approximately 5% of the total samples containing >1 ppm Au or >50 ppm Ag are selected randomly and sent to SGS Colombia. Fire Assay and gravimetric finish methods are used to compare similar analytical procedures between ALS and SGS.

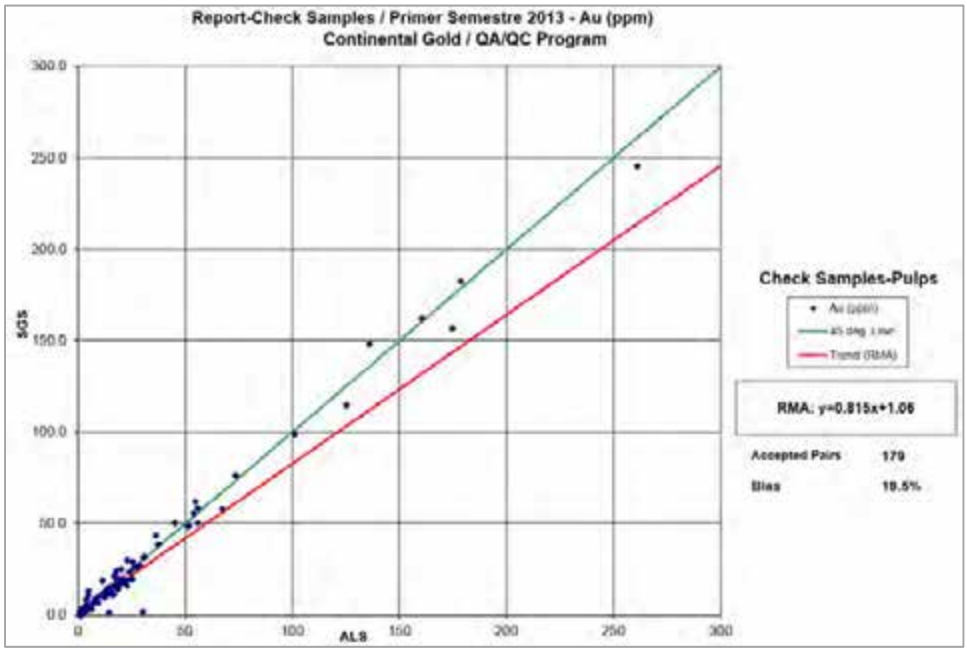
A total of 368 (179 samples for the first half of 2013 and 189 samples for the second half of 2013) were sent to the SGS laboratory for reanalysis of gold, silver and zinc.

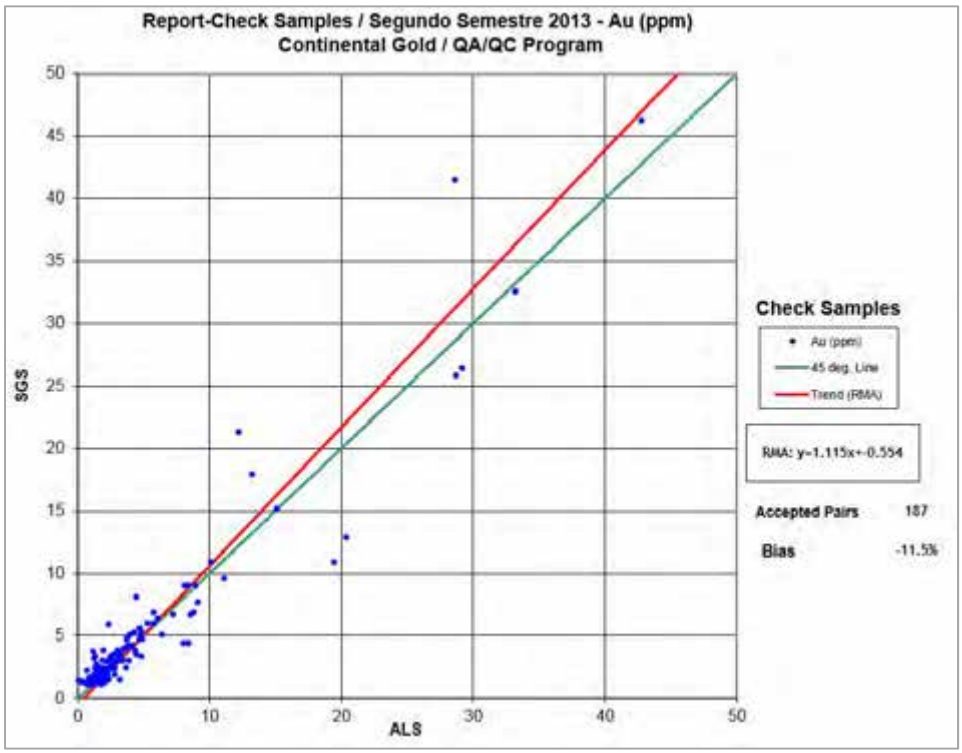
ALS internal laboratory QA/QC cross-check analysis is shown in Figure 11.6. Results are considered acceptable for gold, silver and zinc assays.

11.5.2.5 Internal control QA/QC

Laboratory Blanks and CRM shows figures of blanks and standards used by the Laboratory.

Figure 11.6: Comparison External Lab





Source: MA, 2015

11.6 Adequacy Opinion

Sample protocols, including sample methodology, preparation, security, analysis and data verification have been conducted in accordance with industry standards using appropriate Quality Assurance/quality control procedures since the inception of CGI work in 2010 under the direct supervision of CGI's Vice-President of Exploration, Mauricio Castañeda.

Generally, the results of the QA/QC program implemented by CGI are considered satisfactory and the assay data is suitable for resource estimation purposes. It is MA's opinion that the sample preparation, security and analytical procedures were adequate and follow accepted industry standards for a mining and exploration property.

Two internal audits of CGI's QA/QC protocols have recently been carried out by independent consultants AMEC and REI -RMI.

AMEC (2013) recommended:

- The samples that were ¼ drill core were recommended to be half core by AMEC. Each half core should then be sent to the laboratory for independent analysis to ensure repeatability in the evaluation. This should be complemented with appropriate photographic record of the core before cut. Additionally the bag with the coarse reject could be used as the initial core evidence.
- The low limit of detection should be replaced by half of this limit for the data processing quality control.
- Insertion current rates control samples should be adjusted so that the total insertion rate is not more than 20%, including external control samples. Planning insertions in batches of 50 samples would be more appropriate in this case, rather than in batches of 25 samples.

REI-RMI (2013) recommended:

- CGI investigate using core known to be barren for coarse blank samples (QA/QC), so that laboratory cannot detect the coarse blank and clean crushers/pulverizers immediately before preparing the coarse blank. Certification can be done by splitting and assaying core in barren lithologies.
- Assign new sample numbers to coarse duplicates and duplicate pulps (keeping careful track of sample numbering) in order to prevent laboratory from cross-checking with the original assays.

These recommendations have been implemented and are included in the descriptions above.

12 Data Verification

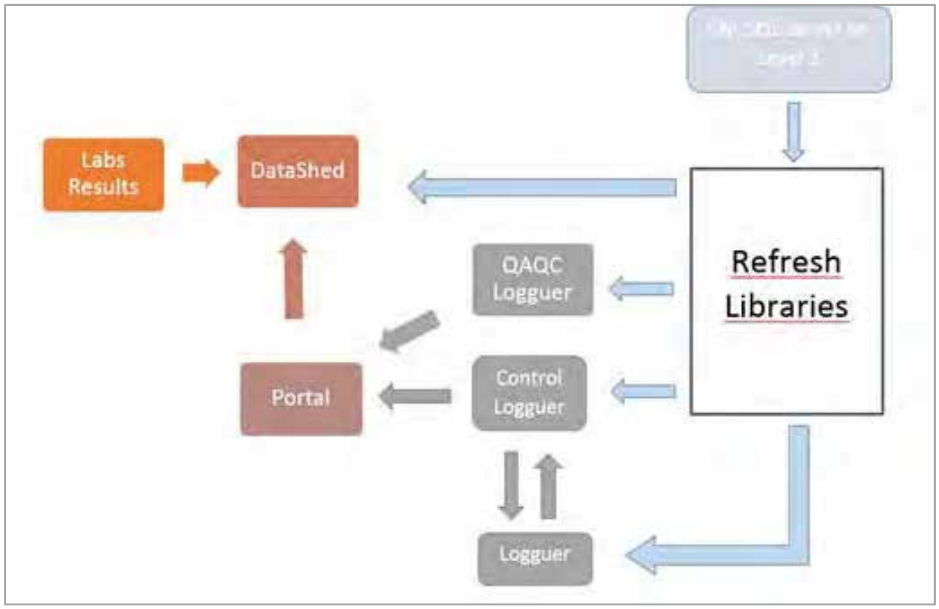
12.1 Data Verification Procedures

The data verification involved database integrity checking, independent laboratory visit, site visit, and independent sample collection.

12.1.1 Drill Hole Database

Resource modeling was based on Continental's database for Buriticá as of May 11, 2015, which Resource & Exploration Mapping Pty Ltd. ("REM") audited and provided to MA.

Figure 12.1: Buriticá Assay Data Handling Procedure



Source: CGI, 2015

MA used a Microsoft Access database that was output directly from CGI's SQL database. MA requested and received data tables and structure descriptions listed in Table 12.1. The database was in good condition, with no cleaning up of the database required.

Table 12.1: Database Tables and Description

Table	Description	Records
Collar table	Contains northing, easting, elevation, and other data for each hole and channel sample start position	4,831
Survey table	Contains vectors of drilling from the collar point	78,576
Assay table	Contains assays for Au, U, Ag, As, Ba, Bi, Cd, Ce, Co, Cr, Cs, Cu, Ga, Ge, Hf, La, Li, Lu, Mo, Ni, Nb, P, Pb, Rb, Sb, Se, Sn, Sr, Ta, Tb, Te, Th, Tl, W, V, Y, Yb, Zn, Zr, Sc, and Mn in ppm, and Al, Fe, Ca, Ti, S, Na, Mg, and K in percent.	271,473
Lithology table	Contains lithological data downhole	27,108
Structures table	Contains structural data such as faults and fractures, directions, downhole	223,601
Minerals table	Contains up to six minerals and their state in order of quantity downhole.	34,535
Alteration table	Contains up to five alteration types downhole.	34,703
Geotechnical table	Recovery and RQD measurements downhole	159,811
Specific gravity table	Specific gravity measurements downhole	11,075

Source: MA, 2015

The drill hole database contains 278,218 m of sampling as of May 11, 2015, both drill and channel samples. This can be broken down as 162,664 m of surface drilling, 115,011 m of underground drilling and 11,032 m of dominantly underground channel sampling, with very few surface samples (Table 12.2).

Table 12.2: Verified Buriticá Database as of May 11, 2015

Drilling Type	Area	Drill Holes/Channels	Metres Sampled	Metres Drilled
Surface DH (BUSY-, BUSM-)	Yaraguá, Veta Sur and Laurel	391	141,775	162,664
Underground DH (BUUY-)	Yaraguá and Veta Sur	345	115,011	108,339
Channel samples, surface samples CH-)	Yaraguá and Veta Sur	4084	11,032	7,215
Total		**4,820**	**267,818**	**278,218**

Source: MA, 2015

Many of the drill holes have lithology, alteration and structure logs and these were used along with the assay results in MA's modeling, in particular specific "vein" lithological code which was used alone to define the vein domains where no assay data had as yet been received.

Underground workings and mapping were also used to help in definition of the vein domains, in particular in the historically mined areas with extensive workings but no sample data, e.g. the Murcielagos domain family.

12.1.2 Site Visit

Mr. Vigar observed that the core storage area was monitored with a security system featuring alarms and cameras. A private security company monitors the exploration office after hours. Only managers, administration and maintenance staff have access to the exploration office.

Mr. Vigar collected relevant sampling procedures and protocols from the site exploration office. Several photos were also taken of the facility and an individual core tray was pulled out of the racks and photographed (see Section 11).

Figure 12.2: Photographs of Buriticá Site and Veta Sur Portal

Looking south from access road, plant site in valley in centre of photo, Veta Sur under ridge on the left

Local workings on Yaraguá, viewed from the graded site access road

Access to the Veta Sur portal

Source: MA site visit, 2011 and 2015

During the 2011 site visit, mineralized drill core was examined and one sample from drill hole BUSY200 (298.9 m to 299 m) was collected from the core trays on site and sent to ALS Laboratory Group S.L, Brisbane for assay. Mr. Recklies examined the underground operations at Yaraguá mine site. Two underground channel samples were collected under the geologist's observation and sent to ALS Minerals, Brisbane for assay. Three additional independent samples were conducted during the 2015 site visit by Mr. Vigar (see section 12.1.3).

Access to the mine site is via a short graded road from the sealed road that leads into Buriticá township. Several drill pads were seen on the way down to the mine site. The graded road gives direct access to the Veta Sur and Buriticá declines, as well as the plant site via a short flight of stairs.

Access to the mine site is controlled by armed security guards and all employees are required to sign in and out when coming and going. There is a gate to the mine site with a security guard present.

Mr. Vigar was taken to Levels 2 and 3 where parts of the Murcielagos and San Antonio veins were observed, as well as the main 51 vein drive at Veta Sur.

Old drill collars were noted along the access trail to the mine and photos of these were taken (Figure 12.3).

Figure 12.3: Old Drill Hole Collars Found Along Access Trail To Mine Site

Source: MA Site Visit, 2011

12.1.3 Independent Samples

Three independent samples were collected from drill core and underground exposures during the site visit and sent to ALS Laboratory in Brisbane, Queensland for analysis. All three returned significant gold and silver grade values and verified the existence of significant mineralization. These samples are selective and do not reflect expected average grades.

Table 12.3: Independent Sample Assay Results

Sample Description	Au (ppm)	Zn (%)	Ag (ppm)	As (%)	Pb (%)	Cu (%)	Ag (ppm)
Sample 1 (Drill Core) BUSY200 (298.9 m to 299 m)	41.7	0.033	22	0.091	0.009	0.125	
Sample 2 (Rock Chips) San Antonio Level 3	24.4	1.87	>1500	0.063	1.51	1.2	4,150
Sample 3 (Vein) San Antonio Level 2	52.8	2.47	146	0.148	1.385	0.038	
2 m W of survey tag 2039							

Source: MA, 2015

12.2 Limitations

The field program at CGI's Buriticá Project is being carried out at the highest professional standard and security, with a specific set of protocols and procedures that were demonstrated to be followed rigorously for the duration of Mr. Recklies' visit. The layout and maintenance of the camp facility, the layout and conduct of the drill site, the attention to health and safety protocols, and the sample collection, logging and preparation at the site match industry standards. The QA/QC procedures adopted for the submission of drill samples are industry standard.

12.3 Verification Opinion

For the purposes used in this technical report, it is MA's opinion that the data is sufficiently verified within the scope of the resource modeling and site visit.

13 Mineral Processing and Metallurgical Testing

13.1 Introduction

The metallurgical test work carried out to support development of the Buriticá process facility is described in this section of the report. The metallurgical test work creating the basis of this section was supervised by Stacy Freudigmann, P.Eng., working in conjunction with Allen Anderson from CGI and BaseMet Labs in Kamloops, BC, Canada. JDS has used the results of the test work to provide metallurgical inputs to M3, who in turn used these to develop the design criteria, and to design the process facility described in Section 17 of this report.

As permitted by Item 3 of Form 43-101F1 – Technical Report, published by the Canadian Securities Administrators (Form 43-101F1), the Qualified Person (QP) responsible for the preparation of this Section has relied upon certain reports, opinions and statements of certain experts who are not Qualified Persons. These reports, opinions and statements, the makers of each such report, opinion or statement and the extent of reliance are described.

13.2 Testing History

In previous project development phases, testing was undertaken on both variability samples and composites blended from each of the deposits on the Buriticá property. More recent test work was carried out on a “Year 1-5” composite, constructed to represent the first five years of the mine life, and also on variability composites to determine variability of individual samples selected to represent the lithology and spatial aspects of the resource. Testing on the Year 1-5 Composite was designed to optimize the Pre-feasibility Study (PFS) flowsheet. The resulting Feasibility Study (FS) flowsheet, and more closely defined process variables, were then used to determine the metallurgical performance of a number of variability composites, evaluating both gold and silver recoveries.

A considerable quantity of testing has been undertaken on the Buriticá Project in a relatively short period. Only the recent feasibility testing is directly applicable to the derivation of the current flowsheet, however, portions of the metallurgical information previously summarized by M3 in the 2014 PEA (December 22, 2014) have been included for completeness. Table 13.1 is a summary of the test work to date.

Table 13.1: Summary of Test work Completed

Year	Laboratory/ Consultant	Report No.	Mineralogy	Comminution	Gravity	Flotation	Cyanidation	Solid/Liquid Separation	Cyanide Oxidation	Other
2015	Kemetco	I2410							X	
2015	BaseMet	BL0047	X	X	X	X	X			Oxygen Uptake, Preg-robbing
2015	Terra	15SEP-002	X					X		Tailing
2014	Pocock	-						X		Rheology
2014	Montana Tech	-	X							
2014	Transmin/SGS	TM 627	X	X			X	X		
2014	SGS	Cz MET 0113/2013 MIN	X	X	X	X	X	X		
2014	Gekko	T1098-Rev5							X	
2013	McClelland	MLI Job No. 3679			X	X	X			
2013	Pocock	-						X		
2013	JKTech	-		X						
2012	Hazen	-		X						
2012	EGC	-	X							
2012	FLSmidth - Knelson	-			X					
2011	Metcon	Q770-03-028.01	X		X	X	X			

Source: JDS 2016

The results of the test programs are available in the following reports:

- Kemetco Research Inc. (Kemetco), Vancouver, Canada, November 2015, Buriticá Tails Detoxification;
- Base Metallurgical Laboratories Ltd. (BaseMet), Kamloops, Canada, January 2016, Metallurgical Testing to Support a Feasibility Study of the Buriticá Project BL0047;
- Terra Mineralogical Services Inc. (Terra), Ontario, Canada, September 2015, Determination of Gold Deportment in One Metallurgical Test Leach Residue Sample from Buriticá Gravity Tail;
- METCON Research Inc. (Metcon), Tucson, Arizona, August, 2011 Buriticá Project, Metallurgical Study;
- Hazen Research, Inc. (Hazen), Golden, Colorado, June 2012, Comminution Testing;
- Economic Geology Consulting (EGC), Reno, Nevada, July 2012, Mineralogy of Metallurgical Samples BUMM-001 through BUMM-004;
- McClelland Laboratories Inc. (McClelland), Sparks, Nevada, May, 2013, Report on Metallurgical Testing, Scoping Laboratory Cyanide Leach, Flotation & Gravity Test work Results;
- JKTech Pty Ltd. (JKTech), Santiago, Chile, September 2013, JKDW & SMC Test Report.
- SGS Mineral Services (SGS), Lima, Peru, January 2014, Report on Metallurgical Testing, Comminution, Cyanide Leach Optimization and Variability, Flotation & Gravity Test work Results.
- Transmin Metallurgical Consultants (Transmin), March 2014, TM 627 Buriticá Reorte Metalurgico, Estudio De Pre-Factibilidad PFS.
- Pocock Industrial Inc. (Pocock), Salt Lake City, Utah, March–April 2013, Flocculant Screening, Gravity Sedimentation, Pulp Rheology, Vacuum Filtration and Pressure Filtration Study for Buriticá Project.
- Pocock Industrial Inc. Salt Lake City, Utah, January 2014, Sample Characterization, Flocculant Screening, Gravity Sedimentation, Pulp Rheology, Vacuum Filtration with Wash and Pressure Filtration with Wash Study for Continental Gold, Buriticá Project Leach Residue.
- Pocock Industrial Inc. Salt Lake City, Utah, March 2014, Sample Characterization, Flocculant Screening, Gravity Sedimentation, Pulp Rheology, Vacuum Filtration with Wash and Pressure Filtration with Wash Study for Continental Gold, Buriticá Project Leach Fines.
- The Center for Advanced Mineral & Metallurgical Processing, Montana Tech of the University of Montana (Montana Tech), Butte, Montana, April 2014, Mineral Liberation Analysis (MLA) Characterization of Ore Samples from the Buriticá Project.
- FLSmidth Knelson, Langley, Canada, October 2012, Gravity Modeling Report.
- Gekko Global Cyanide Detox Group (Gekko), Ballarat, Victoria, Australia, April 2014, Continental Gold – Buriticá Project Detox Test work Report.

Test work programs completed by independent reputable metallurgical laboratories using primarily drill core samples from exploration drilling, include but are not limited to characterization and mineralogical studies, comminution studies, extended gravity recoverable gold and gravity concentration tests, flotation, leach and settling tests. Historical test work results indicate that the mineralization responded well to flotation and to cyanide leaching for precious metal extraction.

13.3 Mineralogical Evaluations

13.3.1 Mineralogy

Historically, mineralogical analysis were conducted by Economic Geology Consulting (EGC) on four composites prepared by McClelland, and also at the Center for Advanced Mineral and Metallurgical Processing on eight mineralized material samples, for the assessment of gold/silver and overall mineralogy by Mineral Liberation Analysis (MLA). The main observations are:

- Electrum and native gold were the primary gold-bearing phase in all of the samples, with trace amounts of the gold-silver telluride (petzite), in about half of the samples. The silver minerals encountered in the samples were acanthite, the tellurides, hessite and petzite, electrum, and silver-bearing tetrahedrite.
- Electrum and gold appeared to be more abundant in the head samples and typically occurred as relatively large liberated particles with grain size distributions P50 ranging from 50 to 100 µm. Gold content of the electrum/gold grains was found to be relatively high, commonly found in the range from about 82 to 92% Au.
- Gangue mineralogy was observed in two major gangue phases; one where potassium feldspar was higher at 26 to 27%, plagioclase at 22 to 26%, quartz at 13 to 16%, and pyrite at 10 to 14%; and the other where quartz was increased to 33 to 38%, 18 to 23% pyrite, 14 to 15% mica (K-mica), and 11 to 12% potassium feldspar. Carbonates, primarily ankerite/dolomite, were more common and galena was observed up to approximately 4%.

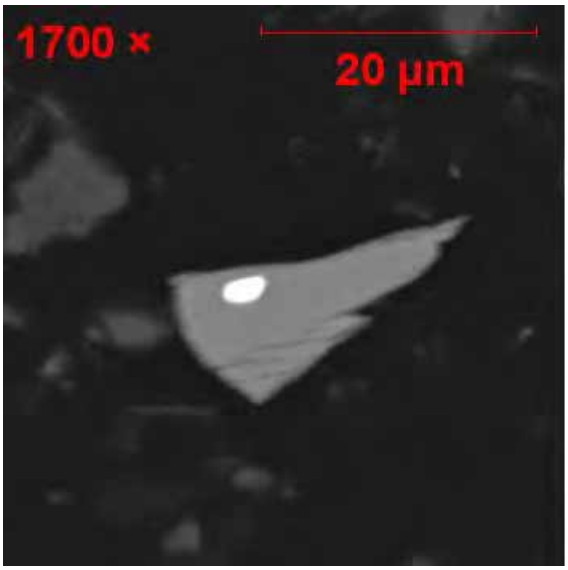
In 2015, both BaseMet Laboratories and Giovanni Di Prisco of Terra Mineralogical Services undertook characterizations of the lithologically and spatially representative variability composites and of a leach tailing residue respectively.

The modal abundance work indicated that all of the samples had substantial amounts of pyrite, ranging from 5% to almost 40%, while the Veta Sur composites had increased levels of pyrrhotite over the Yaraguá composites, which themselves show relatively higher levels of base metals, predominantly sphalerite. Some of the Veta Sur samples exhibited minor levels of base metals.

Historically both deposits have returned assays for copper. The predominant copper mineral in the FS samples is chalcopyrite, while it is clear that Yaraguá has slightly more Tennantite (~5%) and Tetrahedrite (~3%). It is interesting to note that there is more “Other Copper” in the Yaraguá samples than in the Veta Sur samples. “Other copper” here is defined as a combination of chalcanthite, cupro-tungstite, brochantite, malachite, native copper, delafossite, chrysocolla, atacamite, cuprite and petrukite.

The tail leach residue sample submitted to Terra Mineralogical Services Inc. for mineral examination was generated from a cyanide leach test performed on a gravity tail sample from the Year 1-5 Composite ground to approximately 80% passing 50 µm. The main points of interest are summarized

Figure 13.1: Sub-hedral Native Gold Grain Locked in Pyrite



Source: Terra 2015

- Native gold, electrum and petzite (gold-silver telluride) grains were the gold-bearing species identified in this cyanide leach residue;
- Native gold and electrum were found to be the predominant gold carrier (combined ~ 86% of gold), whereas petzite carried the remaining (~ 14 %);
- The entire group of gold particles identified occurred entirely locked, as minute inclusions in pyrite; and
- The average grain size of the gold particles identified in the tail sample is fine, at approximately 1.4 µm.

Although the total population of gold particles identified was limited and the data cannot be considered statistically representative, the observations indicate a clear trend on how discrete gold grains are deported to the tail sample. In addition, experience indicates that with gold grades ranging from 0.4 to 0.5 g/t a larger population of discrete gold-bearing particles should have been identified. The fact that only a limited number of gold particles were found in the leach residue sample could indicate that a fraction of gold in the Buriticá mineralization is contained in solid solution in sulphide minerals, and particularly in pyrite. This hypothesis and observations hold together well with the previous mineralogy undertaken by EGC and Montana Tech.

13.4 Test Work

For each of the test work programs, the number of drill holes sampled have increased, as illustrated in Table 13.2, with the feasibility test program taking samples from 100 drill holes. Longer intervals were sampled for the pre-feasibility, which included increased amounts of dilution. Although a representative metallurgical response was obtained, the comminution composites contained increased amounts of harder host rock, which would potentially increase those results. As the feasibility samples were selected to represent more of the mineralized vein material from Veta Sur and Yaraguá, and prepared in such a way to limit the inclusion of the low-grade host waste rock, they would be more representative of feed to the process facility than the previous programs.

Table 13.2: Test Sample Summary

Test Program	Preliminary Economic Assessment (McClelland)	Pre-Feasibility (SGS)	Feasibility (BaseMet)
Number of Drill Holes	49 drill holes	89 drill holes	100 drill holes
Interval Length Tested	240.8 m	568.6 m	444.4 m
Number of Composites	4 Composites	50 Variability Samples	Year 1-5 Optimization composite
		20 Comminution Samples	45 Variability Composites
		4 Deposit Composites	

Source: JDS 2016

13.4.1 Historical Metallurgical Testing

13.4.1.1 Metcon Research Inc.

Metcon Research carried out preliminary scoping level test work in August 2011 (Q770-03-028.01). Gravity concentration, cyanidation, and flotation tests were performed on eight composite samples with a combined average grade of 54g/t Au and 174g/t Ag, identified as: East San Antonio Vein, Murcielagos Vein, West San Antonio Vein, Breccia BX1, South Vein, composite 1, Breccia BX1 & BX2 and composite 2. It should be noted that no mercury was detected in any of the samples.

Each composite sample was subjected to gravity concentration at a grind size of approximately 80% passing 74µm using a laboratory concentrator. The gravity concentrate was cleaned by panning and the combined tails were subjected to agitated cyanide leaching. Gold extraction in the cyanidation stage ranged from 76.1% to 93.7% (average 89.1%). The best result was from the Murcielagos Vein composite sample, which indicated 93.7% gold extraction. The combined gravity-cyanidation gold extraction ranged from 95.8% to 98.8% (average 97.8%). Silver recovery averaged 57%.

13.4.1.2 McClelland Laboratories Inc.

McClelland Laboratories Inc. carried out metallurgical test work in April 2013 on composites developed from a total of 234 drill core intervals. The intervals were combined to produce four drill core composites, (approximately 60 m of drill core per composite), BUMM-001 (Yaraguá, 30.7g/t Au), BUMM-002 (Veta Sur, 10.2g/t Au), BUMM-003 (70% Yaraguá, 30% Veta Sur, 13.1g/t Au), and BUMM-004 (Non – Typical, 17.5g/t Au) on which cyanidation, flotation and gravity concentration tests were carried out.

A combined gravity concentration and cyanidation of the gravity concentration tailing test at a size distribution of approximately 80% passing 75 µm was conducted.

The gravity concentration gold recovery averaged 45% on these tests, however, the subsequent gravity recoverable gold work undertaken at FLSmidth Knelson, ranged between 68.3% and 90.9% depending on the size distribution tested. It was subsequently recommended that a gravity concentration circuit be included in the process flow sheet moving forward through the process design. The gravity concentration followed by gravity tailing cyanidation combined gold recovery ranged from 92.8 to 97.9%, while the gravity concentration followed by gravity tailing carbon-in-leach (CIL) combined gold recovery ranged from 93.6 to 98.4%, The Yaraguá composite (BUMM-001) had the highest gold recovery in both cases. Although the gravity tailing is amenable to cyanidation and the test results did not indicate occurrence of preg-robbing, the extraction kinetics are faster for the cyanidation test without carbon. Based on these results, CIL was not recommended for the mineral represented by the tested composites.

A gravity concentration test followed by gravity concentrate intensive cyanidation was conducted on each composite. A dose of 5,000 ppm NaCN was used at a grind size of approximately 80% passing 75 µm. The results of the tests indicate that although the gravity concentrate is amenable to intensive cyanidation the total gold recovered to solution in this un-optimized test was relatively low, ranging between 46.3% and 71.1%.

Based on the metallurgical tests described above, the process flowsheet selected for the PEA at this level included gravity concentration of the ball mill discharge, intensive cyanidation of the gravity concentrate, cyanidation of the gravity tailing followed by CCD (Counter-Current Decantation) and Merrill Crowe. This was the basis of the process flowsheet utilized moving forward.

13.4.1.3 SGS Mineral Services

SGS Mineral Services (SGS), under supervision by Transmin Metallurgical Consultants, undertook initial optimization and variability test work in January 2014, using 50 samples from the Yaraguá and Veta Sur deposits. The samples were combined to produce four composites with BC-001 and BC-002 representing Yaraguá, BC-003 and BC-004 representing Veta Sur. Cyanidation, flotation and gravity concentration tests were carried out on the four composite samples.

A summary of the composite samples makeup and the head assays of each are presented in Table 13.3.

Table 13.3. Composite Sample Head Assays

Composite ID	Zone	Head Grades	
		Au, g/t	Ag, g/t
BC-001	Yaraguá	11.6	28.42
BC-002	Yaraguá	14.67	35.67
BC-003	Veta Sur	11.08	35.21
BC-004	Veta Sur	13.99	28.71

Source: SGS 2016

Initial optimization testwork was undertaken, however, there was no clear relationship between grind size and recovery in this program due to un-optimized test conditions. The test conditions with the highest gold extractions were at a grind size of approximately 80% passing 53 µm, 33% solids, 48 hours, lead nitrate and oxygen addition. However, there was only a slight decrease between the gold extraction at 80% passing 74 µm and 80% passing 53 µm for these test series, and based on the coarser grinding benefits, a grind size of 80% passing 74 µm was recommended by SGS for the Buriticá mineralization. The cyanide concentrations tested did not have an impact on the gold and silver recoveries, however, an increase in the cyanide consumption was observed when the cyanide concentration increased from 1 g/L of NaCN to 1.5 g/L of NaCN and 2 g/L of NaCN. The effect of the lead nitrate on the gold and silver recovery was investigated at 25 g/t, 50 g/t and 100 g/t, showing little impact on the gold and silver recoveries. It was noted however, that the sodium cyanide consumption reduced between 15 to 29% and based on these results, a lead nitrate dosage of 100 g/t was recommended. Oxygen optimization tests were undertaken on Composite BC-10 and the results indicated that 20 mg/L of dissolved oxygen produced the highest gold and silver extractions of 95.29% and 72.72% respectively, with the final recovery being achieved after 48 hours. Based on these leach kinetics, a dissolved oxygen concentration of 20 mg/L was recommended, and a leach retention time of 48 hours utilized moving forward.

Variability testing was also undertaken in this program on 50 variability samples, 36 samples from the Yaraguá zone and 14 samples from the Veta Sur zone. The tested conditions used were as per the previous tests on the additional composites using 33% solids, 80% passing 74 µm, 1 g/L NaCN,100 g/t Pb(NO3)2, 25 mg/L DO and a 72 hour leach time, with the cyanidation test results illustrated in Figure 13.2 and Figure 13.3 respectively.

Figure 13.2: Yaraguá Zone – Variability Test Results

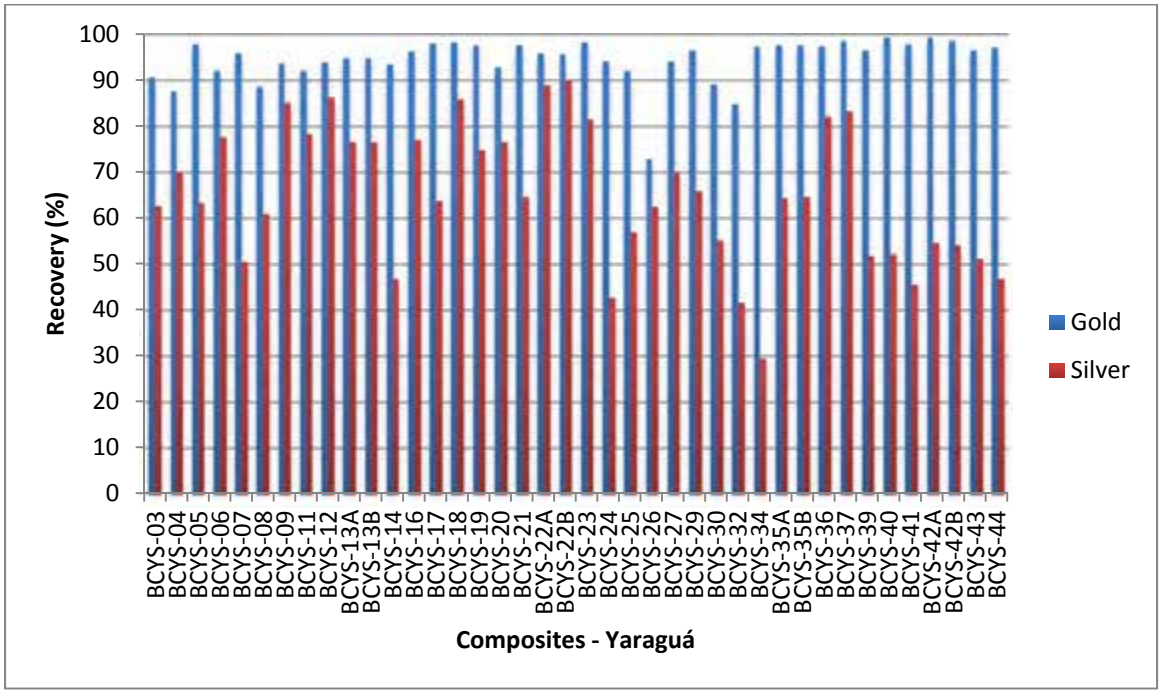
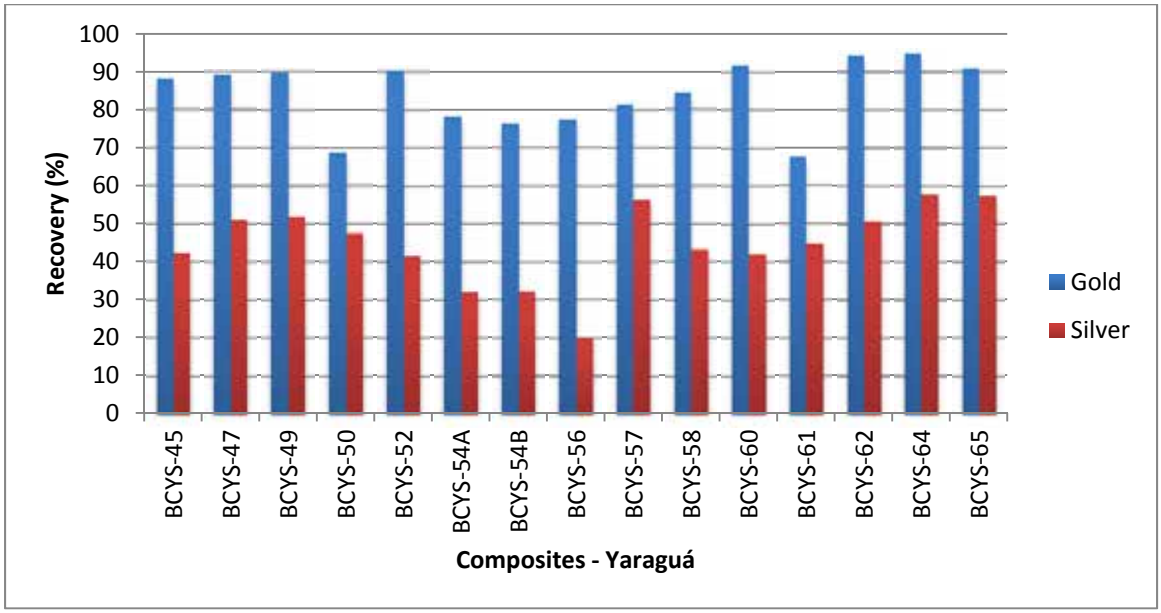


Figure 13.3: Veta Sur Zone – Variability Test Results



Source: JDS 2016

The Yaraguá zone gold extraction average was 94.7% with the majority of the samples recovering above 90%, and the silver extraction being highly variable, averaged 65.40%. The Veta Sur zone gold extraction again was relatively lower than Yaraguá, averaging 84.21% and the silver extraction averaged 44.91%.

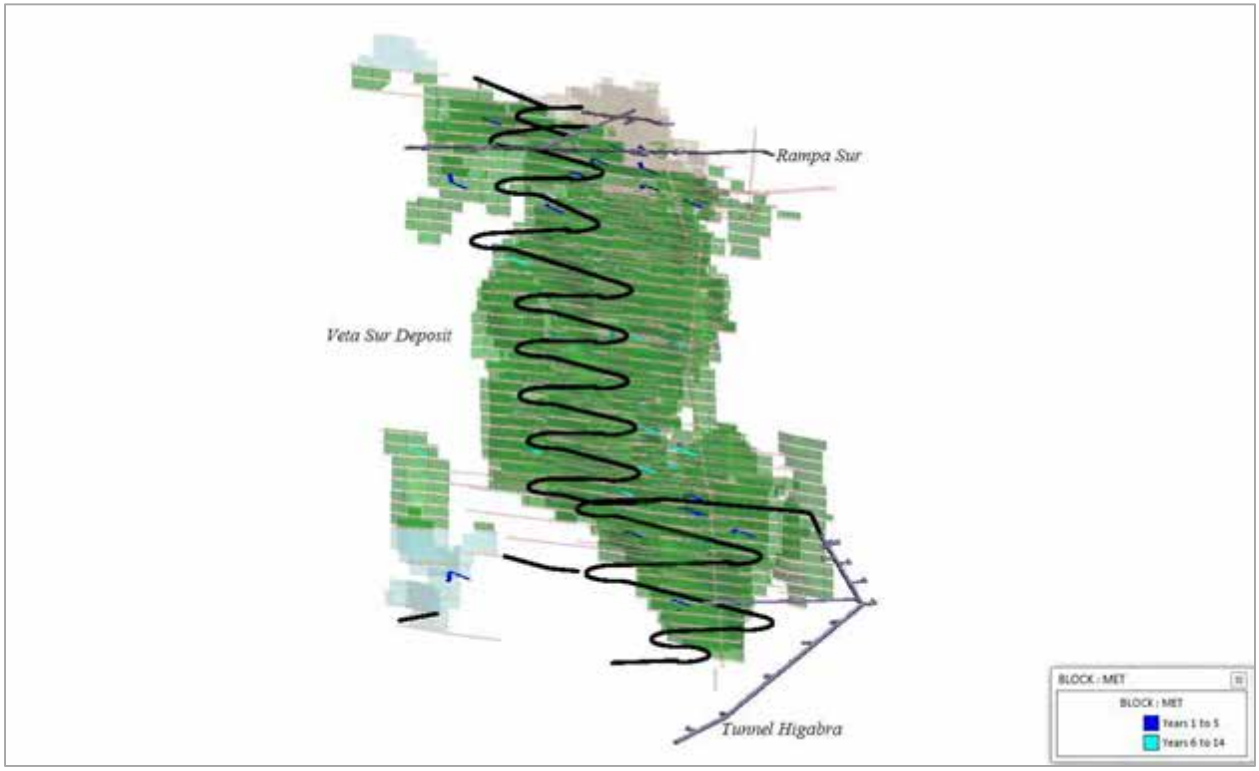
13.4.2 Feasibility Sampling and Composites

The feasibility metallurgical samples, a list of these being available in the test work report provided by BaseMet were selected to cover different lithologies and from different depths, drill holes and areas of the mineralized zones in an effort to be spatially representative of the deposits, and that the corresponding recoveries would be representative of the mineralization types in those deposits. The samples were selected by JDS and CGI site staff packaged these for shipment to BaseMet in Canada.

13.4.3 Veta Sur

Figure 13.4 indicates the locations of the metallurgical samples for Veta Sur. Veta Sur is processed throughout the mine life and represents on average approximately 38% of the process plant feed. Consequently, 229 individual interval samples were selected and combined into 46 metallurgical samples. These metallurgical samples were then combined at Base Met Labs to form 21 metallurgical variability composites, identified as “Comp A” through “Comp U”, being representative of the lithologies and spatial areas of the deposit.

Figure 13.4: Veta Sur Feasibility Metallurgical Sample Locations



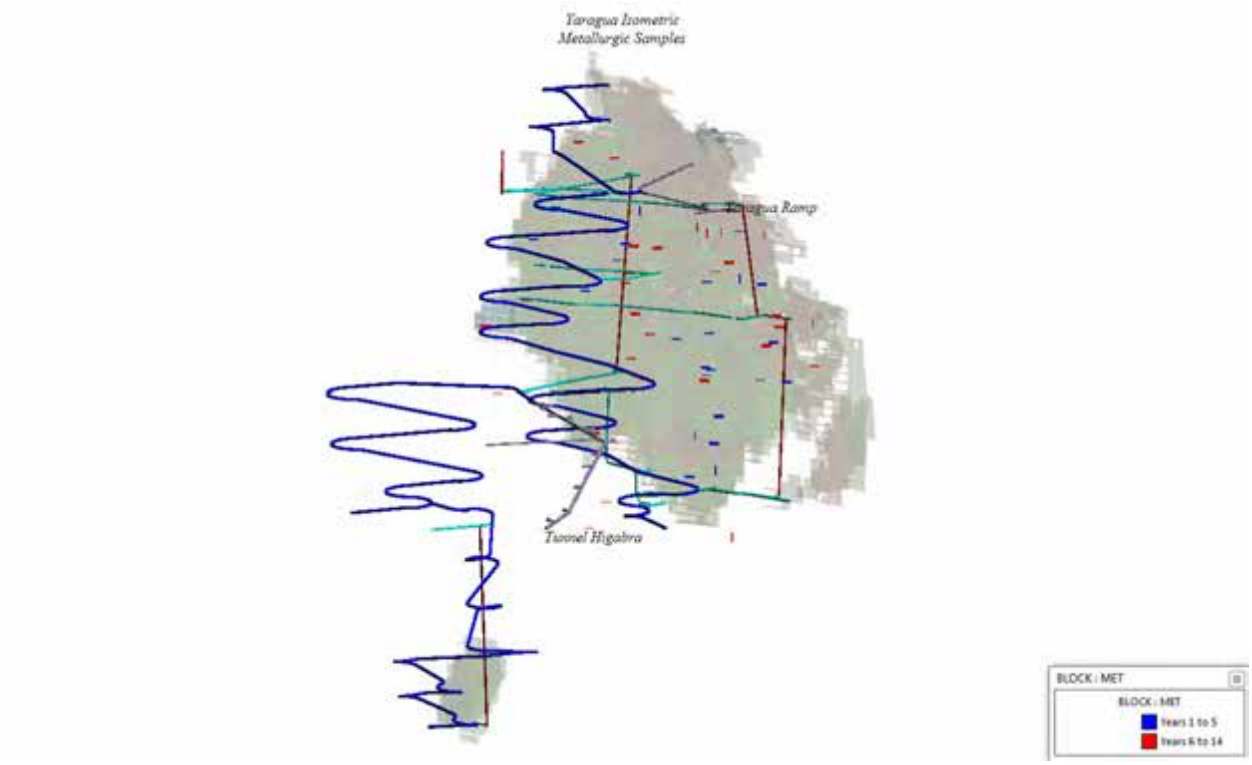
Source: JDS 2016

The darker blue shaded samples in Figure 13.5 were to be mined in the first five years of the mine life, and metallurgical composites from these areas were selected to form part of the composite identified as “Year 1-5 Composite” used for optimization test work.

13.4.4 Yaraguá

Figure 13.5 indicates the locations of the metallurgical samples for Yaraguá. Yaraguá is also processed throughout the mine plan and represents the majority of the process plant feed over the life of mine at approximately 62%. Two hundred seventy-nine individual interval samples were selected and combined into 50 metallurgical samples. These metallurgical samples were then combined at Base Met Labs to form 24 metallurgical variability composites, identified as “Comp AA” through “Comp XX”, produced to represent lithology and spatial areas of the deposit.

Figure 13.5: Yaraguá Feasibility Metallurgical Sample Locations



Source: JDS 2016

The blue shaded blocks in Figure 13.6 were to be mined in the first two years of the mine life, and metallurgical composites from these areas were selected to form part of the composite identified as "Year 1-5 Composite" used for optimization test work.

13.4.5 Year 1-5 Composite

The Year 1-5 Composite was created to represent the first five years of the process plant feed and to provide sufficient mass for feasibility process variable optimization test work. Based on the high-level mine plan available in July 2015, at the commencement of the feasibility metallurgical test program, it was determined that the process plant feed over the first five years would be made up from approximately 60% Veta Sur and 40% Yaraguá; which was the target composite makeup.

An effort was made to represent the LOM gold and silver grade during this period and also maintain the levels of detrimental metallurgical elements such as copper, lead, zinc, sulphur and arsenic. The composite was later compared with the feasibility mine plan and its representativity confirmed.

13.4.6 Comminution Test Work

13.4.6.1 Historical Comminution Test Work

Comminution testing was carried out by Hazen Research, Inc. in June 2012 (Hazen Project 11555). One composite sample prepared by McClelland, was subjected to JKTech full drop-weight, Bond crusher impact work index (CWi), Bond rod mill work index (RWi), Bond ball mill work index (BWi), and Bond Abrasion Index (Ai) testing. The Buriticá Composite 1 was identified with the number 53154-1. A second comminution testing program was carried out by SGS in September 2013 as part of the larger test program described in Section 13.4.1.3 and summarized in Table 13.1 and Table 13.6. The comminution program was conducted on the 20 variability samples and four composites and included: RWi, BWi, Ai, JKTech full drop-weight and SMC Test. Table 13.4 summarizes the CWi, RWi, BWi, Axb and Ai for this Buriticá Project comminution scoping sample, 53154-1 and the average results from the test work undertaken by SGS.

Table 13.4 : Historical CWi, RWi, BWi, Axb and Ai Results

Laboratory	CWi (kWh/t)	RWi (kWh/t)	BWi (kWh/t)	A x b	Ai (g)
Hazen/McClelland	13.62	16.8	16.3	36.2	0.349
SGS	-	18.8	19.0 (150) **22.6 (106)**	36.5	0.142

Source: Hazen/SGS Note: Average BWi at SGS at a closed screen size of 150µm and 106µm.

The drop-weight test result classifies the Buriticá Composite 1 and the average results from the variability comminution tests at SGS as hard for resistance to impact breakage or “moderately hard” to “very hard”, with the 75th percentile A*b of the SGS variability tests of 35.4 being similar to the average and the previous McClelland result.

Composites tested during the 2013 test work were harder than the composite tested in the previous 2012 test work at Hazen. Upon review of the composite makeup, it was hypothesized that due to the nature of the vein deposit and the compositing procedure, a larger than typical proportion of host rock was included in the comminution composite structure, which may have contributed to the elevated comminution results.

The 75th percentile of the Bond Abrasion Index of the 20 variability samples and of the four composites samples undertaken at SGS was 0.19 g with an average of 0.14 g.

13.4.6.2 Feasibility Comminution Test Work

Each of the variability composites for the feasibility test work program undertaken at BaseMet Labs was subjected to both SMC and Bond Ball Mill Work Index testing. Six (6) of the variability composites representing the main lithologies were selected and submitted to Bond Abrasion Index testing. Table 13.5 presents a summary of the feasibility variability composite SMC test results.

Table 13.5: Feasibility Comminution Test Results Summary

Deposit	Sample ID	DWi kWh/m3	A x b	Bwi (kWh/mt)	Ai (g)
Veta Sur	Average	7.19	43.02	16.26	0.05
	Median	6.93	41.6	15.85	0.06
	75th	8.47	33.2	17.2	0.07
Yaraguá	Average	8.08	37.04	15.99	0.05
	Median	8.22	34.2	16.2	0.04
	75th	9.65	28.6	17.25	0.06
Total	**Average**	**7.67**	**39.83**	**16.11**	**0.05**
	Median	**8.2**	**35.3**	**16**	**0.05**
	75th	**9.3**	**30.5**	**17.25**	**0.07**

Source: JDS 2016

Twenty-seven (27) of the 45 variability composites returned an SMC A*b result less than 40, while the other 40% of the results are “softer” and produce an overall 75th percentile of 30.5.

In the BWi tests, the 80% passing size of the product averaged 78 µm with a closed screen size of 106 µm. Due to the 75th percentile returning a 17.3kWh/mt result and thus not as hard as the previous test work, composites were selected for repeat tests and shipped to SGS Lakefield. The results from these tests checked well with the BaseMet Lab results, which leant validity to the results.

As the feasibility samples were selected to represent more of the mineralized vein material from Veta Sur and Yaraguá, and prepared in such a way to limit the inclusion of the low-grade host waste rock, which would be more representative of feed to the process facility, the feasibility results from the variability program were used for design inputs.

The Bond Abrasion Index of six Feasibility Study variability composites would be considered much “less abrasive” than the previous test work results. Based on the difference between the Ai results from the different programs, which may be due to the industry-wide issue of different laboratories using different types of metals to undertake the test, all of the Ai data was collated and a Bond Abrasion Index (Ai) of 0.184 g was recommended for the design input.

13.4.7 Feasibility Optimization Test Work

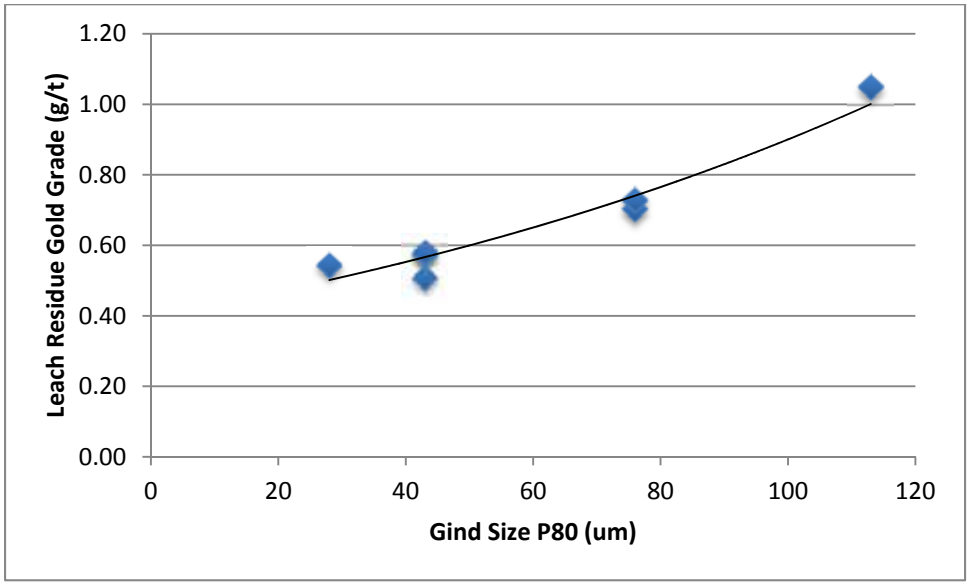
As mentioned previously, the Year 1-5 Composite was created to represent the first five years of the process plant feed and was formed to undertake testing in order to refine the process variables for the leach process. For the initial seven tests, each test charge was submitted to an individual gravity separation step. A larger "bulk" gravity test, Test 8, was undertaken to generate a sufficient mass for subsequent gravity concentrate and tailing for subsequent leach optimization work. Note that the recoveries in the optimization test work do not include the gravity recoverable portion, so overall recovery would be higher.

13.4.7.1 Gravity Tailing Test work

Effect of Grind

The first seven tests were undertaken as individual tests, such that each charge was submitted to gravity concentration to determine the effect of grind size distribution on the gravity tail leach performance. Gravity tail leach conditions developed in previous test work programs were used to undertake this test series.

Figure 13.6: Year 1-5 Composite Gravity Tailing Leach Residue Grade vs. Grind Size



Source: JDS 2015

Due to the presence of coarse gold in the mineralization and the affect the coarse gold had on the gold recovery calculation, the leach tail grade was monitored to indicate gold extraction performance. Figure 13.7 illustrates the effect of grind size on tailing grade for the initial tests, and review indicates that the gold tailing grade decreases with decreasing grind size. Preliminary analysis of this relationship indicated that a finer grind to approximately 80% passing 50 µm would be justified, however, additional test work to evaluate this opportunity is recommended.

Even though 80% passing 74µm was selected for the primary grind size for the process and the variability testing, 80% passing 50µm was used for the grind size for the majority of the optimization test work on the Year 1-5 Composite.

Effect of Cyanide Concentration

The cyanide consumption and tail grades in the initial 12 optimization tests indicated that the cyanide consumption had a strong relationship with cyanide concentration on the grind size tested. Cyanide consumption increases with decreasing grind size, if the cyanide concentration is held the same, however it was observed that the cyanide consumption decreases at the same grind size once the concentration is decreased with no change in tail grade. Although the gold tail grade is not affected for a decrease in cyanide concentration from 1,000ppm to 500ppm, the cyanide consumption decreases by approximately 1.45kg/t. Silver tailing grade did increase slightly for this decrease in cyanide concentration, however, preliminary analysis indicated the savings in cyanide and subsequent detoxification costs outweighed the value of the increased silver recovery at higher cyanide concentration. Decreasing the cyanide concentration below 500ppm for a grind size of approximately 80% passing 70 µm, causes an increase in the gold tail grade. Additional work however would be required to fully optimize the cyanide concentration in the leach, but it is suggested that there is little value added to the Project at this level of study and further optimization should be pursued once in production.

Based on these test results, 500 ppm NaCN was selected as the target NaCN concentration for additional work and for the variability tests.

Effect of Lead Nitrate

Lead nitrate did not have a marked effect on gold tailing grade or cyanide consumption, but it did impact the silver leach tailing grade. Preliminary analysis indicated that the improved silver tailing grade justified the cost of the lead nitrate addition. Addition of 0.1 kg per tonne Pb (NO3)2 was selected for the remaining optimization tests and for the variability test work, but higher dosages may be justified, pending additional study. BMA mineralogy indicates that there is pyrrhotite present, predominantly in the Veta Sur ores. Lead Nitrate addition may assist with reducing the effect of decreased gold dissolution when pyrrhotite is present in the process plant feed.

Effect of Other Process Variables

Other optimization tests were undertaken to evaluate the effect of various schemes to reduce cyanide consumption or improve leach test performance including pre-leach aeration, low leach pH, high leach pH, cement for pH control, CIL processing and air versus oxygen sparging. Conclusions from this test group are summarized below:

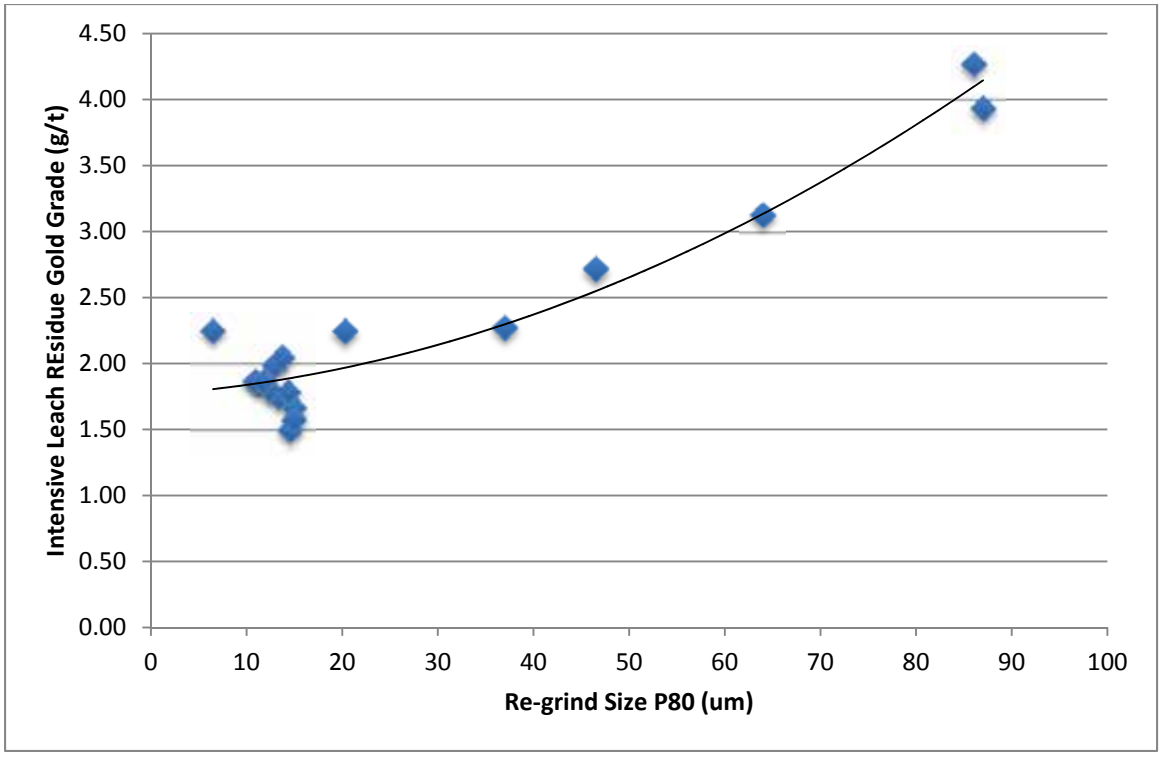
- Cyanide consumption at pH 10 was higher than at pH 11 or pH 12 when using lime for pH control;
- Cement for pH control at pH 11.5 was effective at reducing cyanide consumption and cement consumption rates were similar or only slightly higher to lime consumption rates;
- A pre-leach aeration step was not justified;

- Leaching at higher density (45% solids) did not affect leach performance compared to the standard leach at 33% solids;
- Air for leaching did not adversely affect leach tail grade compared to oxygen, however it did negatively impact the leach kinetics over the first 24 hours of leaching;
- CIL processing was not beneficial; and
- Leach time of 48 hours is adequate.

13.4.7.2 Gravity Concentrate Test work

A series of tests were also undertaken on the bulk gravity concentrate in an effort to more fully develop the variables around intensive concentrate leaching. Test results indicate generally a decreasing gold intensive leach tailing grade as the regrind size was reduced from approximately 80% passing 80 µm with no regrind, to less than 80% passing 20 µm as illustrated in Figure 13.7.

Figure 13.7: Year 1-5 Composite Gravity Concentrate Intensive Leach Residue Grade vs. Grind Size



Source: JDS 2016

No lead nitrate was added to the gravity concentrate leach as the recoveries were in line with previous experience for intensive leaching and it was not deemed to be required. The tailing grade did increase on the test result where a NaCN concentration of 1,000 ppm was used, and 2,000 ppm was recommended for the plant design should an intensive leach be included in the flowsheet.

It is noted that the leach tailing grade on sample that was not reground was just over 4.0 grams gold per tonne of concentrate. After regrinding, the leached gravity tailing grade decreased to 1.5 to 2.4 grams gold per tonne, depending on the test. Additional work is required to further optimize the gravity concentrate process with respect to increased mass pull to the gravity concentrate and leach conditions with respect to cyanide concentration should gravity concentrate leaching be included in the process flowsheet in the future.

13.4.8 Feasibility Variability Test work

Based on the process variable development, the variability tests were then undertaken on the 45 variability composites from both mineralized zones. Of the 45 composites, only eight tests were repeated, and all repeats responded as expected except composite R from Veta Sur, which had relatively elevated levels of arsenic compared with the other composites. Test results are summarized in Table 13.6 and included in the BaseMet report.

Table 13.6: Variability Test Result Summary

		Calc Head	Calc Head	Au Recovery (%)		Ag Recovery
		Au (g/t)	Ag (g/t)	Gravity	48hr	48hr (%)
Veta Sur	min	3.4	5.8	33.7	66.3	80.5
	average	9.8	44.4	61.9	91.6	51.2
	median	9.8	30.9	58.3	91.7	50.3
	max	20	384	85.8	98.8	48.8
Yaraguá	min	0.5	3.9	34.6	89.5	88.9
	average	12.8	68.2	60.3	95.5	57.4
	median	6.8	33.7	62.9	95.9	55
	max	61.5	348	90.9	99.2	55.7
Overall	min	0.5	3.9	33.7	66.3	6.8
	average	11.4	57.1	61.1	93.7	54.5
	median	8.2	31	60.7	93.7	52.4
	max	61.5	384	90.9	99.2	88.9

Note: Averages are based on the most appropriate result, excluding 1000ppm NaCN repeats and including tests with the desired grind size.

Source: JDS 2016

Review of Table 13.8 indicates that the Veta Sur gold recovery averaged 91.67%. Analysis of the repeat tests indicates that grind size indeed plays a role in the gold recovery. The median Veta Sur gold recovery was 91.7%. Yaraguá responded well to the applied flowsheet and leach conditions, with the average recovery at 95.5% Au and the median at 95.9% Au.

Using the appropriate tests for the average, (excluding 1000ppm NaCN and including repeat tests with the required grind size), the average gold recovery was 93.7%.

Analysis of the variability results showed an excellent gold recovery correlation with head arsenic concentration for the Veta Sur composites, while the Yaraguá composite gold recoveries correlated well with iron, (or pyrite), levels. Both of these recovery responses compare well with the mineralogy observed through the deposits and on the tailing samples.

Geometallurgical techniques may yield gold recovery relationships that might improve gold recovery modeling and further study would be required to more fully understand these.

13.4.9 Solid-Liquid Separation Test work

Three solid-liquid separation programs were conducted on the Buriticá mineralization. The first in April 2013 was conducted at Pocock Industrial Inc. for McClelland on leached flotation tailing and leached flotation concentrate samples from the BUMM-001 – BUMM-004 composites. The average 80% passing particle size of the flotation tailing was approximately 113 µm, while the average 80% passing particle size of the flotation concentrate was 42 µm.

The second program was conducted on one leach residue sample from the SGS test work program, and also tested by Pockock Industrial Inc. in January 2014. This leach residue sample was generated from the composite BC-09, which was historically prepared from 196 m of drill core from 31 holes. Composite BC-09 contained 14 intervals from 12 drill holes from the Veta Sur zone and 19 intervals from 19 drill holes from the Yaraguá zone. The particle size distribution of this leach residue was approximately 80% passing 78 µm.

The third program was also conducted on another leach residue sample from the SGS test work program, and also tested by Pocock Industrial Inc. in March 2014. This leach residue sample was generated from the composite BC-10, which was historically prepared from 265 m of drill core from 40 drill holes. Composite BC-10 contained ten intervals from nine drill holes from the Veta Sur zone and 36 intervals from 31 drill holes from the Yaraguá zone. This leach residue was received and then separated into a coarse and fine fraction using a hydro-separator. The coarse fraction was not tested; however, the particle size distribution of fine fraction from this leach residue was approximately 80% passing 41 µm.

A brief summary of some of the recommendations from the testing programs, concentrating on the BC-09 leach residue sample being more representative of process variables, is as follows below.

The flocculant selected for the best overall performance; and thus used in subsequent thickening testing was Hychem AF 304, a medium to high molecular weight, 15% charge density, and anionic polyacrylamide. Minimum flocculant dose requirements for thickening on the BC-09 leach residue were observed in the range of 15 - 25 g/t, and should be delivered at a maximum solution concentration of 0.1- 0.2 grams per litre (g/L)..

Results of static or conventional thickening tests indicated that the BC-09 leach residue sample thickened very well at feed solids concentrations in the range of 15% - 20%. At feed solids concentrations higher than 25%, flocculation efficiency may be difficult to achieve in a plant situation, resulting in lower settling/rise rates and higher unit area requirements. Hence, a maximum thickener feed solids concentration of 15% - 20% was recommended for this material.

The maximum predicted operating density range for standard thickener on leach residue with a grind size of approximately 80% passing 78 µm was from 57 to 61%. Recommended maximum operating densities for leached concentrate material was from 54 to 67% and for leached flotation tail is 56 to 63%.

The minimum unit area for conventional thickener sizing on BC-09 leach residue with a grind size of 80% passing 78 µm is between 0.195 and 0.22 m^2/t/d and the net feed loading rate range is 4.7 to 5.2 m^3/m^2h, with an average of 4.95 m^3/m^2h.

13.4.9.1 Pulp Rheology

A comparison of apparent viscosity at reference shear rates and varying percent solids for the thickened BC-09 leach residue sample for CCD Stage: 1 and CCD Stage: 3 – n is as follows:

Table 13.7: Pulp Rheology Test Results on BC-09 Leach Residue Sample

Material	Solids Conc. (%)	Coefficient of Rigidity (Pa·s)	Yield Value (Pa or N/m²)	Apparent Viscosity, Pa·s @ Shear Rates								
				5 Sec⁻¹	25 Sec⁻¹	50 Sec⁻¹	100 Sec⁻¹	200 Sec⁻¹	400 Sec⁻¹	600 Sec⁻¹	800 Sec⁻¹	1000 Sec⁻¹
Leach Residue (CCD Stage: 1)	64.8	0.070	69.0	6.443	2.493	1.656	1.100	0.731	0.486	0.382	0.323	0.283
	63.7	0.054	55.2	4.812	1.837	1.213	0.801	0.529	0.350	0.274	0.231	0.202
	59.9	0.033	25.0	2.490	0.898	0.578	0.373	0.240	0.155	0.120	0.100	0.087
	54.8	0.017	8.6	1.163	0.369	0.225	0.137	0.084	0.051	0.038	0.031	0.026
Leach Residue (CCD Stage: 3 - n)	65.1	0.108	72.7	7.747	2.774	1.782	1.145	0.736	0.473	0.365	0.304	0.264
	63.2	0.078	54.4	5.771	2.014	1.279	0.813	0.517	0.328	0.252	0.209	0.180
	59.9	0.048	27.0	3.249	1.082	0.674	0.420	0.261	0.163	0.123	0.101	0.087
	55.2	0.029	10.4	1.337	0.480	0.309	0.199	0.128	0.082	0.063	0.053	0.046

Source: Pocock 2016

The decreasing apparent viscosity, with increasing shear rate or "shear thinning" behavior of the underflow pulps examined is characteristic of the pseudoplastic class of non-Newtonian fluids (in the solids concentration range tested). It demonstrates the need to achieve and maintain a specific velocity gradient or shear rate in order to initiate and maintain flow. Underflow pulps with yield values in excess of 30 N/m^2 (Pascals) measured on pre-sheared pulps are normally beyond the capabilities of conventional thickening and pumping systems.

Specialized equipment and design engineering are generally required if high underflow densities with yield values greater than 30 N/m^2 are to be considered.

13.4.9.2 Filtration

Based on the test data obtained, the recommended type of filter press for the leach residue material is a recess plate with air blow and membrane squeeze.

Recovery results from washing analysis in these pressure filtration tests indicated that a wash ratio of N = 2.0 – 3.0 for the air blow and membrane squeeze tests, would require two 2,000 mm units with a P19 frame size and require 297 – 379 filter chambers; in this wash range, removal efficiencies were between 96.76% and 98.42%. With higher wash ratios the recovery efficiency only slightly increased but would require more pressure filter units.

With filter feed solids of 58.5% the estimated moisture of the filter cake for the air blow and membrane squeeze case 14.3%. At these moistures the filter cakes produced from pressure filtration testing of the leach residue material were easily dischargeable from the testing apparatus and generated a stackable and conveyable cake.

13.4.10 Cyanide Oxidation Test work

The historical cyanide oxidation test work was undertaken by Gekko Systems Pty. Ltd in Australia in April 2014. The SO2/O2 cyanide destruction process was successful in reducing the levels of CNWAD to the target of less than 1.0 mg/L. A detox feed slurry containing 559.5 mg/L CNWAD was effectively treated and resulted in a stable final CNWAD of less than 0.2 mg/L under reaction conditions of 34% solids, 4.5 gSO2 / g CNWAD, pH 8.5 and 115 minutes retention time. No copper catalyst addition was required in this test work due to sufficient level of soluble copper in the leach tail. The SO2/O2 cyanide destruction process could not obtain the target CNWAD level if the pH was decreased from 8.5 to 8.0, or the retention time was decreased from 115 to 80 minutes.

Based on the results from the test work program the SO2/O2 cyanide destruction process was the preferred process, as the Caro's acid method was unable to reduce the CNWAD to the target level.

Kemetco Research Inc. in Vancouver undertook a program of cyanide leaching and detoxification testing using Year 1-5 Optimization composite gravity tailing samples, in October 2015. The purpose of the work was to verify plant design criteria, and to deliver representative treated effluent samples for environmental testing. Leach conditions of 34% solids, pH to 11.5 (with cement), 100 g/t lead nitrate, 48-hours of leaching with 500 mg/L NaCN at a dissolved oxygen level >15 ppm.

In total, six extended bench-scale tests were conducted to verify properly-optimized detoxification conditions. An average reduction of 37 mg/L Cu and 545 mg/L of CNp was obtained with NaCN-spiked DT feed, resulting in average discharge levels of 0.2 mg/L CNp, 0.6 mg/L Cu and 490 mg/L CNO-, using a 4.5:1 SO_2:CN_{WAD} ratio, and 90 minutes of retention at pH 8.5, without the need for additional Cu-catalyst. The picric acid method (CNp) was used to estimate the CNWAD contents of weak acid dissociable cyanide.

Table 13.8: Kemetco Detoxification Test Summary on Year 1-5 Composite Gravity Tailing

Test No	SO2/ CNp Ratio	%Solids PD	CNp ppm	As ppm	Cu ppm	Fe ppm	Avg, mV
batch	5.0:1	32.25	0.09	0.43	0.13	0.15	-150.8
DT1	5.0:1	32.25	0.09	0.16	0.19	0.20	8.0
DT1A	4.0:1	32.25	0.05	0.67	1.15	0.19	-33.9
DT2	4.5:1	28.62	0.32	0.89	0.51	0.37	-56.1
DT3	4.5:1	48.17	0.13	1.05	1.21	0.38	-101.2
DT4	4.5:1	36.88	0.05	0.66	0.27	0.16	-52.9
DT5B'	4.5:1	36.33	0.74	0.86	0.76	<0.2	-237.4
AVG	4.5:1	35.25	0.20	0.67	0.60	0.21	-47.2

Source: Kemetco

The changes in the key variables for these tests produced consistent results, with CNp results all less than 1ppm, including the high pulp density of 48%. This would indicate that the oxidation circuit might be able to be designed after tailing thickening without dilution of the slurry and further work would be required to determine the validity of this approach.

It was concluded that the oxidation test parameters were valid in achieving the target discharge cyanide concentrations, and that excursions to higher pulp densities and slightly shorter retention times could be tolerated. Substitution of cement for lime, and air for O2 was found to be possible, but further test work would be required to optimize these conditions in conjunction with optimization of the cyanide leach circuit and precious metals recovery steps.

13.5 Other Design Considerations

13.5.1 Mercury

Only traces of mercury have been observed in the composites sampled. Typically, if the mercury level is below 50 ppm in the process plant feed, in gold districts where mercury is present, it is not expected to be an issue downstream, either as a competitor for gold in the extraction process or for health reasons. 50ppm is an experience-based guideline as it is dependent on the extraction potential of the mercury and its geological form. Above 50 ppm, mercury mitigation actions may be required, however, it is understood that the Buriticá process facility has included these abatement processes in the design.

13.5.2 Reagent Consumption

During the feasibility optimization test series, the addition of cement for alkalinity control was tested against lime for both gravity tail and concentrate. Table 13.9 reports the average comparable data from the feasibility test work for the gravity tail leach.

Table 13.9: Lime versus Cement Consumption on Gravity Tail Leach

Dose [NaCN] (ppm)	Grind Size	NaCN Consumption (kg/t)	Cement Consumption(kg/t)
500	75	0.8	2.6
500	75* Regrind	1	2.4
	Average:	0.9	2.5
Dose [NaCN] (ppm)	Grind Size	NaCN Consumption (kg/t)	Lime Consumption (kg/t)
500	75 + 50	1.2	2
500	75* Regrind	1.7	2.6
	Average:	1.5	2.3

Source: JDS 2016

Based on these results, use of cement alkalinity control in the Buriticá process should be considered as it decreases cyanide consumption in the gravity tailing leach process by approximately 17% over using lime.

13.6 Recovery Correlations

At the conclusion of the test program at Base Met Labs, and upon analysis of the available data from this program, the overall recovery relationships were developed.

It was observed that recovery relationships were more highly correlated when the data from two mineralized zones, Veta Sur and Yaraguá, were treated separately. Consequently, two recovery relationships were developed. As discussed in the following paragraphs, the best correlations were found to be with arsenic for Veta Sur and with iron for Yaraguá, even though sulphur did produce a correlation, it was not as accurate a prediction of recovery when compared to arsenic and iron. CGI and JDS completed an effort for this Feasibility Study to incorporate the iron and arsenic assays into the block model and mine plan and the recovery relationships are as follows;

Recovery Equations:

For Veta Sur, the recovery relationships proposed for gold grades between 3.0 – 24.0 g/t Au and silver grades between 5.0 – 105.0 g/t Ag, based on the feasibility data would be:

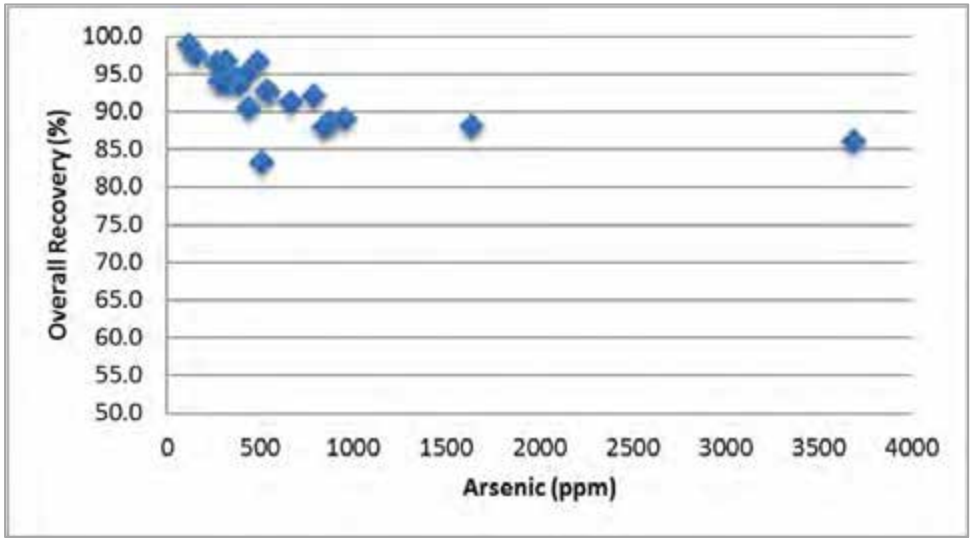
- Gold Recovery (%) = 95.627 – 0.006861 x Arsenic(ppm);
- Silver Recovery (%) = 64.408 – 0.4317 x (silver grade (g/t));
- For Yaraguá the recovery relationships proposed for gold grades tested between 1.0 – 93.0 g/t Au and silver grades between 3.0 – 190.0 g/t Ag, based on the feasibility data would be:
- Gold Recovery (%) = 102.4 – 1.0672 x Fe (%);
- Silver Recovery (%) = 72.864 – 0.2787 x (silver grade (g/t)).

These equations include 0.75% deduction for solution losses and other plant inefficiencies, as experienced in typical operations.

13.6.1 Veta Sur

The data in Figure 13.10 indicates a correlation between arsenic levels and recovery for the Veta Sur mineralization, with recovery beginning to decrease at arsenic levels in the feed above approximately 400-500ppm As.

Table 13.10: Veta Sur Overall Gold Recovery versus Arsenic

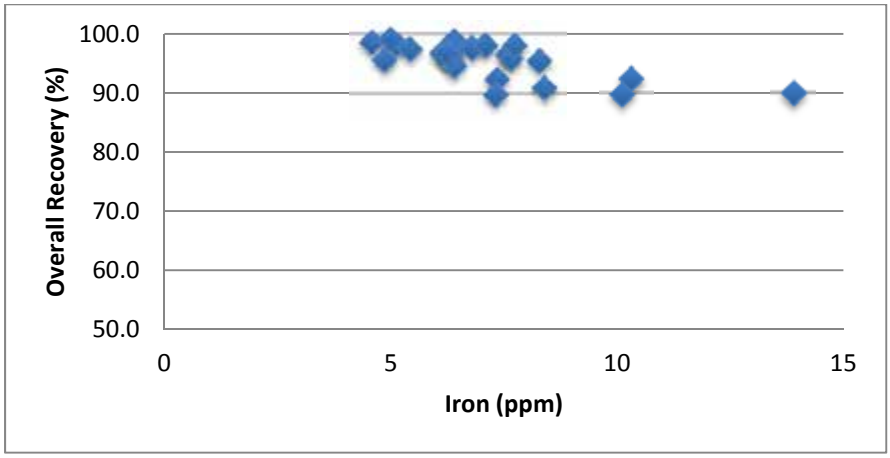


Source: JDS

13.6.2 Yaraguá

Similarly with Yaraguá as for Veta Sur, the data indicates a correlation between arsenic levels and recovery for the Yaraguá mineralization, with gold recovery also beginning to decrease at arsenic levels in the feed above approximately 500ppm, however, the correlation was found to be more variable in Yaraguá than the relationship with iron as illustrated in Figure 13.8, which provided a better correlation with the gold recovery.

Figure 13.8: Yaraguá Overall Gold Recovery versus Iron



Source: JDS

14 Mineral Resource Estimate

A maiden mineral resource estimate was prepared in accordance with CIM guidelines and NI 43-101 disclosure standards for CGI by Mr. Andrew J. Vigar of MA and reported in the 2011 Technical Report. In November 2012, the maiden resource was updated by MA, accompanied by the 2012 Technical Report. A further update was prepared by MA, accompanied by the 2014 Technical Report.

This mineral resource estimate is an update to the 2014 estimate and utilized a similar but more advanced estimation methodology on the larger and more extensive database. The cut-off date for information used in preparation of this report, and thus the effective date, is 11 May, 2015.

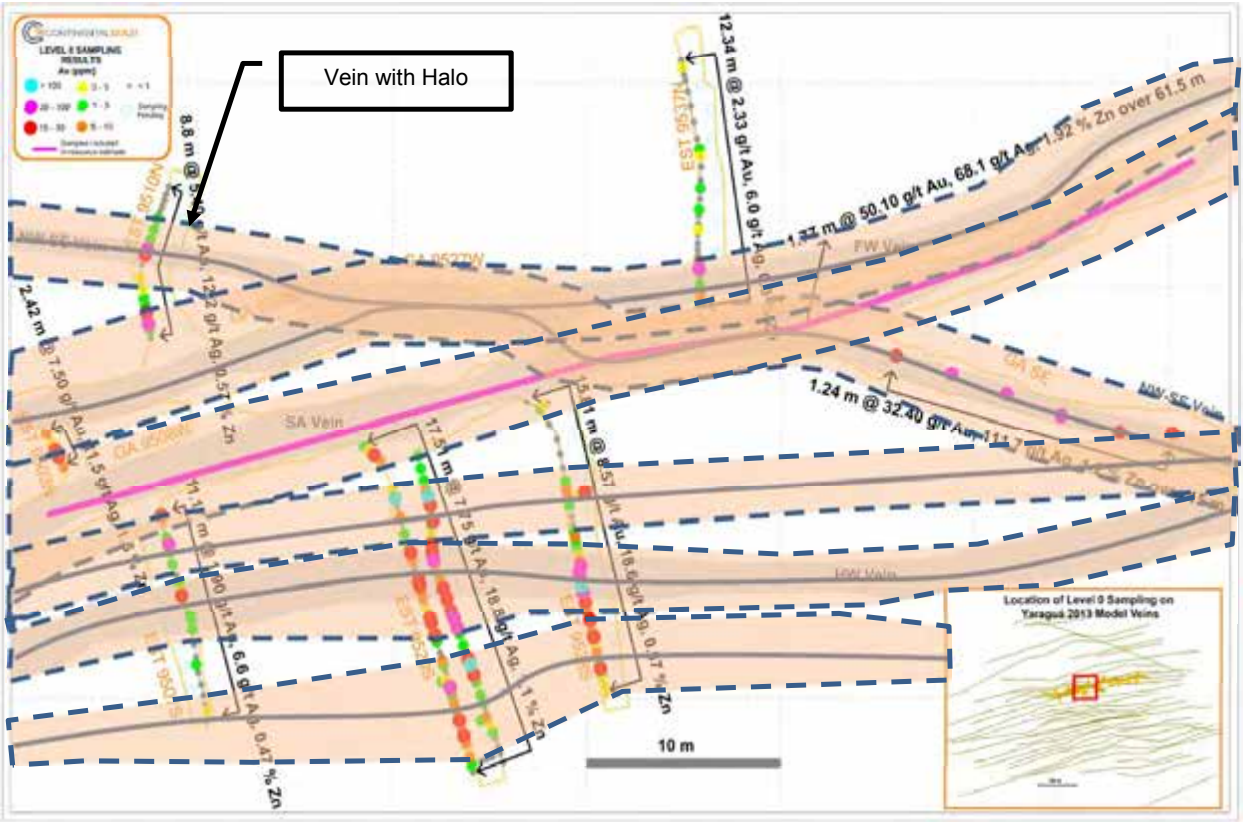
14.1 Approach

High-grade precious metal mineralization at Buriticá is largely confined to vein domains which comprise a complex array of one or more veins, veinlets and fracture-related wall-rock disseminations. The vein domains are referred to by the shorthand name “veins” in this report. Such vein domains have very large strike and dip extents compared with their horizontal widths. Due to the relatively coarse nature of the sample density in relation to the proposed narrow vein mining strategy at Buriticá, it is necessary to be able to model both the thickness and grade of vein domains. Where available, underground channel samples have been incorporated into the data set and provide detailed grade information through the extents of the underground openings sampled. The issue of mixed sample supports, (channel vs. drill hole) and number and variability of informing samples is addressed locally before mineral resource categories are identified.

During 2014, a 650 m long program of numerous short cross-cuts located on Levels 0 and 2 of the pilot scale Yaraguá mine was developed, along with a 250 m strike drive and cross-cuts at Veta Sur. The aim of the program was to confirm vein and grade continuity and to examine precious metal grades for calculating dilution in the “halo” around the main estimated veins that will be sequenced early in future mine production. This program identified that certain areas of both systems contain wider zones of mineralization with “stringers” and minor vein splits carrying up to several grams per tonne gold that will impact on the design and dilution factors of the final mine design (Figure 14-1).

Based on this new information, the approach to the resource estimation was expanded to include the modeling of grades in the “Halo” within 2 m on each side of outer edge of each individual vein. Note that the resource estimation for the veins is unchanged from the Independent Technical Report and Resource Estimate on the Buriticá Project 2015, effective date May 11, 2015. Veins less than 1 m in width were diluted out to 1 m using block grades from within the dilution halo. The final resource reporting grade is at the 1 m minimum width with the remainder of the “Halo” also stored in the final 3D block model. Where veins are closer together, as in Figure 14-1, the “Halo” zones can overlap to form an almost continuous envelope but as each block in the 3D model is unique, “double counting” cannot occur. It is possible in rare cases for the “Halo” zone to have gold grades above final resource reporting cut-off, but these have not been included in the resource estimates as reported, which are for the vein domains only.

Figure 14.1: Plan View of Drift and Cross-cut Sampling on Level 0, Yaraguá



Source: MA, 2015

14.1.1 Mining Associates Methodology of Modeling Vein System

MA used Geovia Surpac software to model the vein thickness and grades for Au, Ag, Zn and Pb. MA's methodology facilitates highly constrained resource estimates within the narrow hard boundaries of vein domains in unfolded space. This methodology has been extended to the estimation of the "Halo" zone around the veins.

MA's methodology has been developed over the last 20 years to assist in the interpretation, modeling and resource estimation of complex vein systems. The methodology employed by MA allows for the interpretation of several overlapping and cross-cutting vein arrays simultaneously. Drill intercepts or channel samples can be updated or re-allocated from one vein to another with ease but are only used once. Changes are recorded in a relational database.

MA's methodology has been utilized on numerous narrow vein gold systems including Beaconsfield Gold Mines in Australia, Tuvatu Gold Project in Fiji, Tolukuma Gold Mine and Kainantu Gold Mine in PNG as well as having been utilized in the 2011, 2012 and 2013 resource estimations for the Buriticá Project.

The current resource estimation methodology has been modified from that used in previous resource estimates, prior to the Independent Technical Report and Resource Estimate on the Buriticá Project 2015, effective date May 11, 2015.

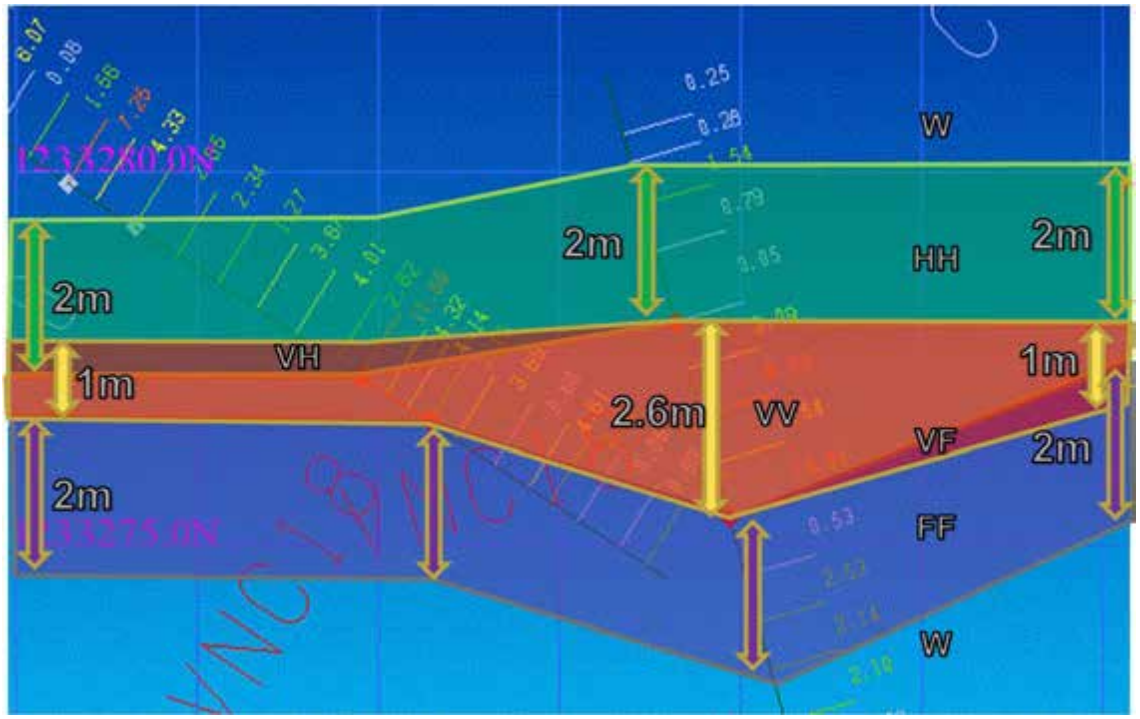
To more accurately account for dilution and the application of a minimum mining width for resource statements, in this case of 1 m. To achieve this, grades in dilution “halos” around each vein have been estimated.

Following tagging of vein intervals into the database, vein grade and thickness values are estimated into each panel of a model in ‘unfolded’ 2D space. Grade in a 2 m halo either side of each vein boundary is also estimated, using footwall and hangingwall informing samples separately. Each vein composite with a true thickness value less than 1 m is then ‘diluted’ to 1 m using material from whichever halo material has, from the hangingwall or the footwall, the higher grade (Figure 14-2).

The resulting material type codes are:

- W – Waste, no grade estimated;
- HH – Halo Hangingwall. Grade estimated from samples within the original 2 m from the original vein boundary. Width reduced by amount taken by vein dilution;
- VH – Portion of the Hangingwall Halo taken to give the minimum vein width of 1 m. Grade is the weighted average of the original vein and halo grades. HH grade was higher than the HF grade;
- VV – original vein. Grade is the original estimate if undiluted, or weighted average if Halo was added to reach the minimum width;
- VF - Portion of the Footwall Halo taken to give the minimum vein width of 1 m. Grade is the weighted average of the original vein and halo grades. HF grade was higher than the HH grade; and
- FF – Halo Footwall. Grade estimated from samples within the original 2 m from the original vein boundary. Width reduced by amount taken by vein dilution.

Figure 14.2: Vein Modeling System and Final Type Codes



Source: MA, 2015

14.2 Supplied Data

Database validation is managed by CGI using Datashed connected to an SQL database. All data was provided to MA as per MA request. MA closed further entries to the database for resource modeling as of May 11, 2015.

MA used a Microsoft Access database that was output directly from CGI's SQL database. MA requested and received data tables and structure descriptions listed in Table 14.1. The database was in good condition, with no cleaning up of the database required.

Table 14.1: Database Tables and Description as of May 11, 2015

Table	Description	Records
Collar table	Contains northing, easting, elevation, and other data for each hole and channel sample start position	4,831
Survey table	Contains vectors of drilling from the collar point	78,576
Assay table	Contains assays for Au, U, Ag, As, Ba, Bi, Cd, Ce, Co, Cr, Cs, Cu, Ga, Ge, Hf, La, Li, Lu, Mo, Ni, Nb, P, Pb, Rb, Sb, Se, Sn, Sr, Ta, Tb, Te, Th, Tl, W, V, Y, Yb, Zn, Zr, Sc, and Mn in ppm, and Al, Fe, Ca, Ti, S, Na, Mg, and K in percent.	271,473
Lithology table	Contains lithological data downhole	27,108
Structures table	Contains structural data such as faults and fractures, directions, downhole	223,601
Minerals table	Contains up to six minerals and their state in order of quantity downhole.	34,535
Alteration table	Contains up to five alteration types downhole.	34,703
Geotechnical table	Recovery and RQD measurements downhole	159,811
Specific gravity table	Specific gravity measurements downhole	11,075

Source: MA, 2015

The drill hole database contains 278,218 m of sampling as of May 11, 2015, both drill and channel samples. This can be broken down as 162,664 m of surface drilling, 115,011 m of underground drilling and 11,032 m of underground channel sampling with very few surface samples

Many of the drill holes have lithology, alteration and structure logs and these were used along with the assay results in MA's modeling, in particular specific lithological code "vein" was used to define the vein domains where no assay data had as yet been received.

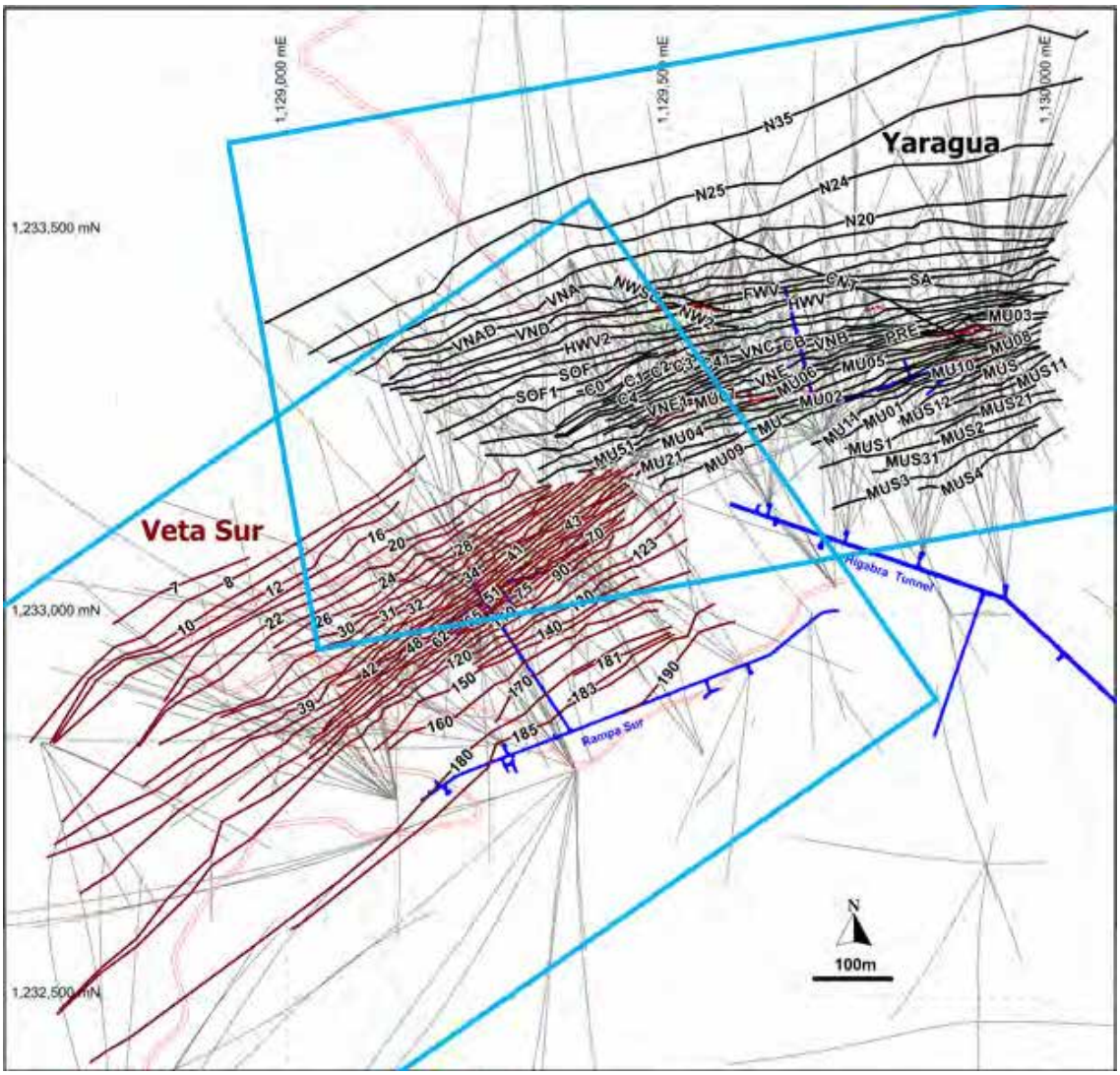
In the case of Yaraguá, underground workings and mapping were also used to help in definition of the vein domains, in particular in the historically mined areas with extensive workings but no sample data, e.g. Murcielagos vein domains.

14.3 Dimensions

Since intensive exploration began on the Buriticá property in 2008, CGI has outlined a 2 x 2 km carbonate base metal-type gold vein/breccia system related to a centre of porphyry-style intrusions and alteration. To date, the most extensively explored vein systems have been Yaraguá, to the Northeast, and Veta Sur, to the Southwest (Figure 14-3). The Yaraguá vein system has been drill-outlined along 1,125 m of strike and 1,540 vertical metres and partially sampled in underground developments. The Veta Sur system has been drill intersected over 1,140 m along strike and 1,600 vertical metres and partially sampled in underground developments. There are 89 veins in the current resource envelope for Buriticá, 51 veins at Yaraguá (Figure 14-3) and 38 at Veta Sur (Figure 14-4). Both systems exhibit veins with steep, sub-vertical dips, and both remain for the most part open at depth and along strike.

A couple of small, poorly sampled veins were interpreted from drilling to the north of Yaraguá; however these were not included in the resource estimate.

Figure 14.3: Plan View of Buriticá Deposit with Block Model Outlines



Boxes represent unconstrained block model extents

Source: MA, 2015

At Yaraguá, exploration drilling and also underground development on several of the vein domains have defined the extents of the veins as modeled. Most veins are open to depth as well as to the east and west.

Of the 51 veins at Yaraguá, 49 strike east (010º) and two strike north-west, intersecting the east-striking veins. The 49 east-striking veins are arranged in coherent packages over cross strike extents of around 500 m and vein domains modeled are listed in Tables 14-4 and 14-5.

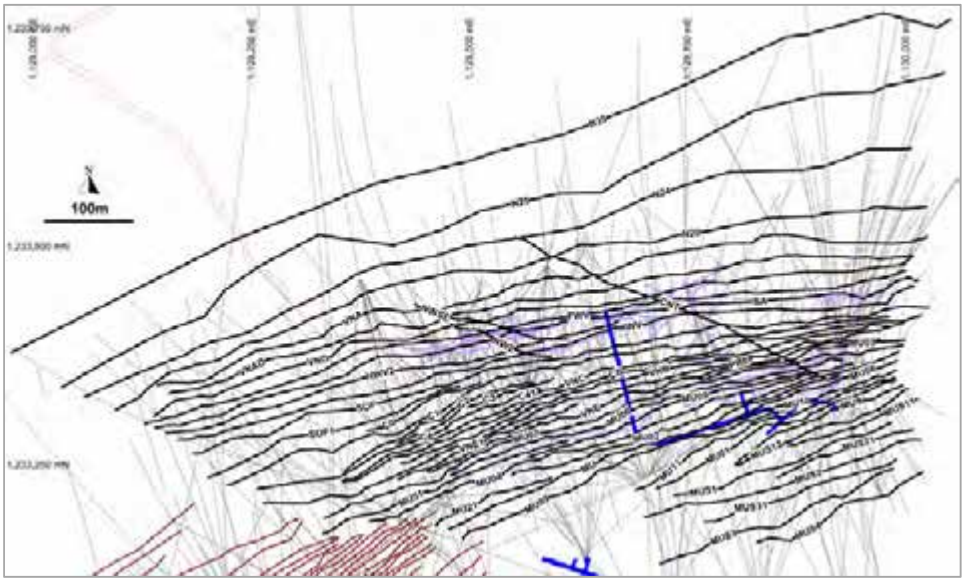
The vein strike extents, as modeled, vary from a minimum of 50 m along strike and a maximum of 1,100 m. Modeled vein extents vary from a minimum of 50 m to a maximum of 1,300 m vertically.

Of the 38 veins at Veta Sur, all strike Northeast with sub-vertical dips. Veta Sur veins arranged in coherent packages over cross-strike extents of around 400 m and vein domains modeled are listed in Figure 14-1. Vein strike extents, as modeled, vary from a minimum of 70 m to a maximum of approximately 1,000 m. Vein vertical extents, as modeled, vary from a minimum of 150 m to a maximum of approximately 1,350 m.

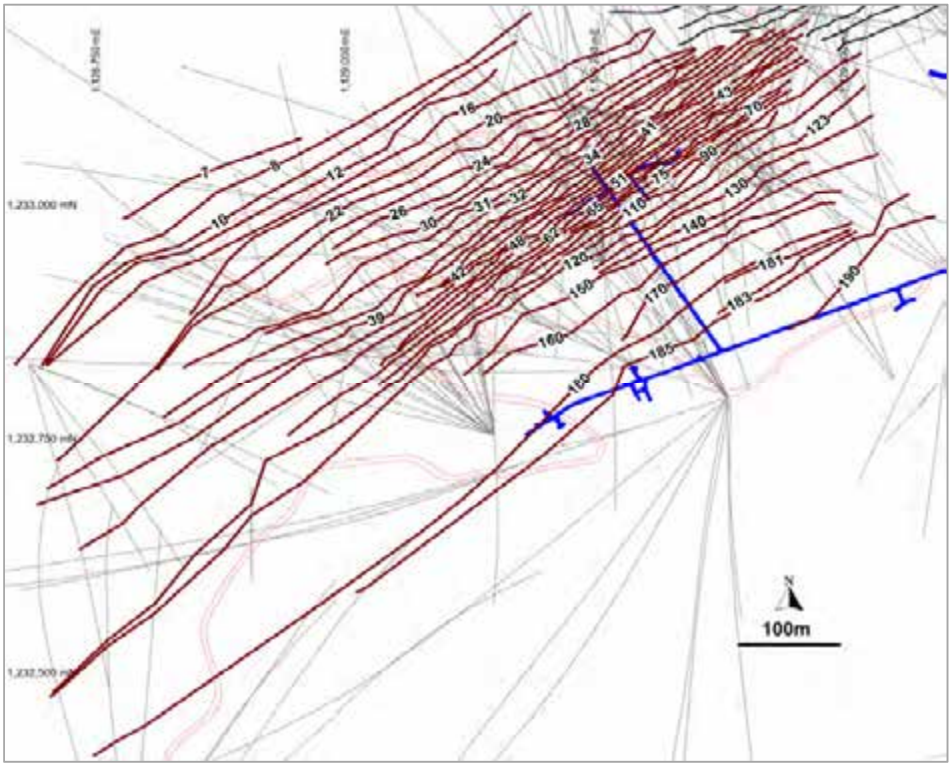
The above dimensions are the projected dimensions of the known vein domains based on drilling and underground development where available. Some veins are closed off along strike, but others continue into the last drill hole along any particular direction. Given the consistent nature of these continuous veins it is likely with more drilling these veins will be extended further at depth and along strike.

Figure 14.4: Boxes Represent Unconstrained Block Model Extents

Yaraguá labelled veins



Veta Sur labelled veins



Source: MA, 2015

When veins were tagged, veins were interpreted at distances up to and over 500 m, as long as no information to the contrary, which is typically drilling intercepts with low grades, was found between the respective tags. Interpolation between vein composites is only extended up to 200 m; attributes are not interpolated past this. Effectively any block further than 200 m from a vein composite will not be estimated, although it can still be considered part of the vein. These volumes are obvious targets for drill hole targeting.

14.4 Geologic Interpretation

A geological model is the spatial representation of a set of data that helps miners to understand what is in the ground, from the interpretation of a specialist in earth sciences (generally a geologist). This interpretation is an approximation of reality from the data obtained as a result of geological exploration. A model never perfectly predicts reality. Only if and when the mine is developed will we know if the model represented a good interpretation.

A geological model is constructed from many inputs as described below:

- Drill holes in surface and underground in 3D space;
- Sampling channels;
- Any structural features e.g. faults, joints;
- Geomorphology and topography from LiDAR surface;
- Mapping (surface and underground) and sections; and
- Mining developments.

The assumptions applied to the interpretations utilized in this model was checked and validated to ensure consistency. The drill holes and channel samples were checked for any obvious errors, such as a drill hole location above the topographic surface.

Table 14.2: Generation of Boundary Surfaces

Triangulation	Commonly a surface (a digital terrain model) is visualized by creating triangular facets produced by joining points on its surface using a semi-automatic Delaunay triangulation technique. A surface produced by triangulation is sometimes modified by superimposing a regular grid on it.
Wire Frame Modeling	This is created by using triangulation to produce an isometric projection of, for example, a rock type, mineralization envelope or an underground stope. Volumes can be determined directly of each closed wireframe (solid).
Block Models	Geological representation (and grade evaluation) of a deposit can be undertaken by constraining an existing 3D block model by intersecting with a wire frame model so that only the blocks within the limits are defined and evaluated. This involves interpolation of attribute values from spare drilling data using geostatistical techniques.
Boundary Models	Where the mineralization envelope is not well defined, a boundary can be formed around groups of blocks that meet certain criteria. By allowing blocks cut by this boundary to be sub-blocked and the volume re-determined for these sub-blocks, the boundary can be refined and redrawn in an iterative way. Combining blocks from sections in a wire frame model allows global estimates of grade and tonnage of each rock/alteration type. Other boundaries may be an outcrop or sub-crop, the lease boundary, a predetermined mining depth or a structural feature such as a fault.

Source: MA, 2015

MA has constructed a 3D wireframe model and block model using the industry standard Geovia Surpac software (Version 6.7).

A detailed step-by-step description of the general methodology can be found in the estimation section of this report. Drill hole assay intervals (or underground channel samples) with appropriate gold grades are selected (tagged) and these intercepts (composites) are converted into points in space, which are then used to create each vein 3D model.

The database was updated with the domain codes in the Yaraguá and Veta Sur regions, representing mineralized veins. For an intercept to qualify as a domain code, the following basic selection criteria were followed, in decreasing hierarchy:

- Gold grade greater than 2 g/t Au and/or 100 g/t Ag over a true width of 1 m; or
- No assays but a lithology code “vein” in the expected location; or
- Sub-grade areas where the interpreted vein domain passed through the drill hole but was not already coded (i.e. “brought through”).

For sub-grade intercepts that were “brought through”, the composites were generally made as close to a metre in length as possible using the highest sub-grade intercept available.

Vein domains thus defined on gold assays and geology were also used for the estimation of silver and zinc grades. A single downhole (or along channel) composite sample was extracted from the drill hole database for each intercept. Separate grade composites were made for silver and zinc within these vein domains such that samples with no results for silver and/or zinc were not used.

14.5 Data Preparation and Statistical Analysis

The Buriticá database was connected directly to Surpac (geological and mining software) for data display, downhole compositing, wireframing of homogeneous grade domains and block model estimation. Statistical analysis (univariate statistics) of the grade data and variography was carried out using Surpac's geostatistical utilities.

Prior to a statistical analysis, grade domaining is normally required to delineate homogeneous areas of grade data. Statistical analysis does not take into account the spatial relationships of the data. In the case of the Buriticá Project's mineral resource estimate, each vein was considered to be an individual domain.

14.5.1 Drill Hole Spacing

Drill hole data spacing is variable within each domain but the company aimed for an intercept pattern of between 20 and 50 m.

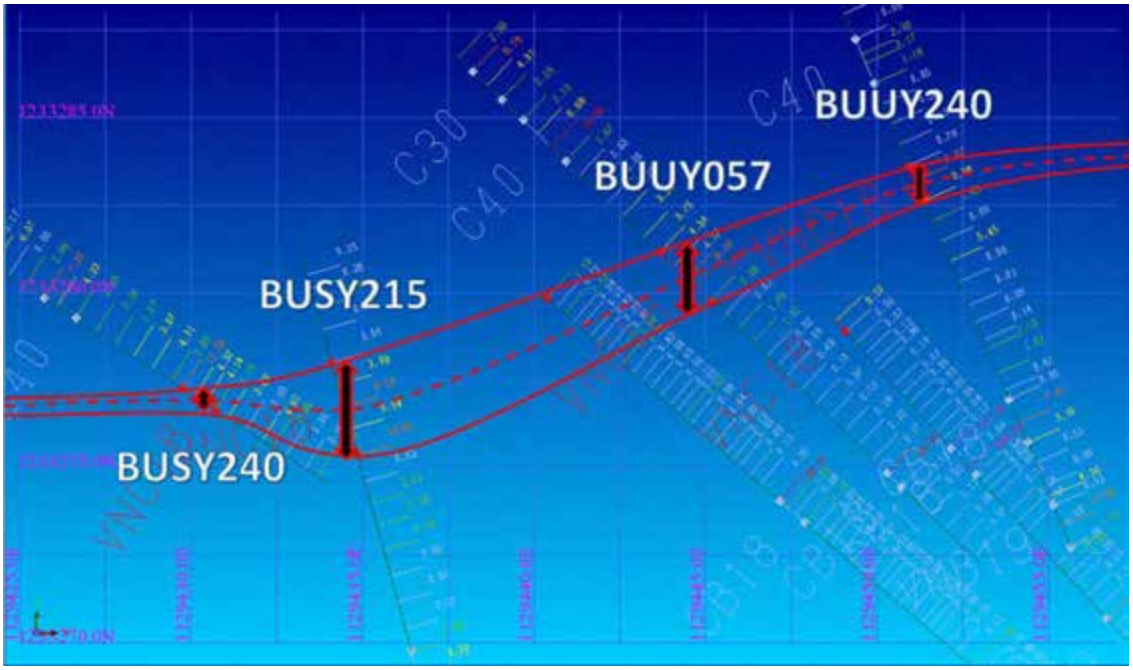
14.5.2 Compositing

Many of the drill hole samples were assayed at 1.5 m intervals, with some intervals being as short as 10 cm. Channel samples are generally shorter than the drilling samples. Mineralized veins can be as thin as 0.1 m, and channel sample intervals reflect the true thickness of the veins' variability across-strike. The grades are highly variable with the sample length; some very short samples were selected on geological grounds from highly mineralized material only, with longer samples adjacent, from the same vein, of lower grade material. The following approach was taken to remove this variability.

The objective of compositing data is to obtain an even representation of sample grades and to eliminate any bias due to different sample lengths. One complete composite for each vein intercept was used, representing a full sample across the full width of the vein no matter the intercept angle. This is done so that informing sample is of equal value in the estimation process, being a full width sample of the vein. There is no subdivision within the veins of grade.

The method used is similar to the approach of True Width reporting, as recommended by the CIM, of calculation and application of the True Width (as defined by the CIM guidelines) estimations is shown in the Figure 14-5, with estimated grades similar to the original sample grades.

Figure 14.5: Conversion of oblique intercepts to True Width – VNC18



VNC				Sample Data		1.0m Composites, 50% min, within vein tags					Tue Width Vein Composites			
hole_id	depth_from	depth_to	Length	Au_ppm	Ag_ppm	depth_from	depth_to	Length	Au_ppm	Ag_ppm	Length	T width	Au_ppm	Ag_ppm
BUSY229	310.25	311.00	0.75	7.02	3.29	310.25	311.25	1.00	6.11	2.93	1.75	1.09	4.94	2.46
BUSY229	311.00	312.00	1.00	3.38	1.84	311.25	312.00	0.75	3.38	1.38				
BUUY057	83.85	84.85	1.00	22.23	9.60	83.85	84.85	1.00	22.23	9.60	3.55	1.76	10.01	5.84
BUUY057	84.85	85.50	0.65	8.04	4.35	84.85	85.85	1.00	6.69	4.48				
BUUY057	85.50	86.40	0.90	4.17	4.73	85.85	86.85	1.00	4.25	4.43				
BUUY057	86.40	87.40	1.00	4.34	4.06	86.85	87.40	0.55	4.34	2.23				
BUUY240	81.00	81.40	0.40	4.14	6.00	81.00	82.00	1.00	4.97	5.95	1.60	0.85	7.16	6.91
BUUY240	81.50	82.00	0.50	4.32	5.32	82.00	82.60	0.60	11.55	8.85				
BUUY240	82.00	82.60	0.60	11.55	8.85									
BUSY215	301.50	302.50	1.00	3.09	5.91	301.50	302.50	1.00	3.09	5.91	4.20	2.74	7.11	9.13
BUSY215	302.50	303.50	1.00	8.50	7.24	302.50	303.50	1.00	8.50	7.24				
BUSY215	303.50	304.50	1.00	5.54	12.00	303.50	304.50	1.00	5.54	12.00				
BUSY215	304.50	305.70	1.20	10.61	11.00	304.50	305.50	1.00	10.61	11.00				
						305.50	305.70	0.20	too short	too short				
Average Sample Grade				7.46	6.48			0.91	7.60	6.33	2.78	1.61	7.31	6.09
Length Weighted Average Grade				7.71	6.71				**7.69**	**6.52**			**7.54**	**6.81**

Source: MA, 2015

14.5.3 Univariate Statistics

Univariate statistics were generated on each of the veins at Buriticá for gold, silver and zinc raw grades. Raw grade statistics allow direct comparisons of current vein selections with vein selections used in previous resource estimates.

Coded domain intercepts were extracted from the database and grouped into informing sample files. Basic statistics were generated for each vein and checked for discrepancies such as zeroes, missing samples, or grades below cut-off. Any tagging errors found were corrected in the database and the composites were re-extracted.

Uncut sample statistics for gold, silver and zinc, classified by vein are shown below in Table 14.3 (Veta Sur) and Table 14.4 (Yaraguá). At Veta Sur, mean composite gold grades vary from a maximum of 30.81 g/t Au for vein 123 to a minimum of 1.33 g/t Au for vein 12. Coefficients of variations for gold for these veins vary between about 0.7 and 8.7, which are typical for veined gold deposits. These values are before application of the 2 mgms sub-domains – mgms is metres* gold grams/tonne.

Mean silver grades were generally higher for veins to the northwest part of Veta Sur, with the maximum value of 180.58 g/t Ag for vein 190, while the lowest mean silver grade was vein 160 to the southeast, with a grade of 7.98 g/t Ag.

MA does not consider the zinc to be economically material to the resource, however zinc commonly effects recovery and reagent consumption, thus is included in the resource estimate.

Table 14.3: Uncapped Statistics of Gold, Silver and Zinc Composites by Veins for Veta Sur

Vein	No. Samples	Mean Grades			Max Grades			CoV		
		Au (g/t)	Ag (g/t)	Zn (%)	Au (g/t)	Ag (g/t)	Zn (%)	Au	Ag	Zn
7	7	3.34	12.83	0.10	7.05	32.83	0.51	0.95	0.80	1.93
8	22	2.01	68.03	0.20	14.10	673.00	2.13	1.48	2.43	2.41
10	22	2.89	71.36	0.31	15.96	436.00	1.53	1.32	1.79	1.24
12	21	1.33	63.12	0.54	6.91	652.00	6.03	1.34	2.41	2.51
16	38	6.18	99.10	0.54	109.07	1460.00	4.90	2.91	2.56	2.27
20	54	2.53	74.02	0.54	14.56	526.44	3.82	1.49	1.74	1.60
22	50	1.89	53.40	0.51	17.77	365.00	5.60	1.73	1.70	2.22
24	74	4.50	53.28	0.37	73.74	866.26	3.12	2.75	2.45	1.88
26	76	3.43	62.89	0.46	43.13	1260.00	5.52	2.39	2.93	2.23
28	52	4.19	29.95	0.35	57.25	250.57	3.73	2.22	1.72	1.80
30	97	3.05	36.27	0.41	74.11	337.00	7.00	2.87	1.78	2.32
31	116	8.52	31.42	0.30	352.22	721.50	9.70	4.16	2.94	3.68
32	123	4.52	34.78	0.27	97.40	641.13	4.22	2.65	3.01	2.11
34	156	3.13	39.13	0.29	38.76	1914.80	4.91	1.76	4.12	2.33
39	175	4.70	24.50	0.33	164.51	517.00	12.86	2.88	2.31	3.32
41	141	3.37	21.74	0.21	28.25	327.17	1.83	1.67	2.06	1.43
42	157	6.54	30.51	0.28	104.98	365.99	3.67	2.29	1.91	1.73
43	179	4.98	19.08	0.43	70.49	216.00	11.85	2.19	1.64	2.71
48	194	3.93	27.30	0.28	45.77	471.79	6.98	1.71	2.02	2.27
51	188	25.16	81.54	0.42	1729.78	723.83	5.47	5.16	1.69	2.00
62	194	19.06	87.15	0.36	924.30	1775.00	8.30	3.92	2.45	2.55
65	194	6.59	53.99	0.43	111.93	1490.00	12.52	2.09	2.69	2.64
70	117	6.17	24.46	0.18	155.02	159.39	1.69	2.78	1.41	1.36
75	121	3.37	22.83	0.21	39.24	282.91	1.97	1.89	1.85	1.44
90	181	9.36	25.71	0.32	545.56	757.00	10.77	5.43	2.79	2.84
110	156	4.35	20.61	0.28	82.70	466.00	4.77	2.50	2.45	1.91
123	96	30.81	63.86	0.17	2615.41	2906.00	2.59	8.66	4.94	2.41
130	67	6.56	27.89	0.47	77.30	416.00	17.25	2.28	2.48	4.53
140	56	6.51	15.36	0.37	231.00	80.60	5.49	4.71	1.17	2.53
150	67	3.79	13.51	0.40	38.80	80.37	11.40	1.76	1.16	3.51
160	50	2.98	7.98	0.18	26.20	57.00	2.32	1.76	1.29	1.99
170	31	2.50	11.79	0.32	24.38	63.50	3.51	1.78	1.24	1.98
180	45	3.45	29.44	0.27	20.00	483.00	2.62	1.35	2.59	1.70
181	10	1.79	20.89	0.09	5.04	87.09	0.29	0.96	1.30	1.08
183	10	2.99	25.22	0.18	6.11	145.02	0.65	0.72	1.73	1.12
185	47	2.69	27.21	0.38	26.28	391.60	4.31	1.86	2.50	2.28
190	19	2.15	180.58	0.62	11.29	1915.00	2.96	1.41	2.54	1.23

Table 14.4: Uncapped Statistics of Gold, Silver and Zinc Composites by Veins for Yaraguá

Vein	No. Samples	Mean Grades			Max Grades			CoV		
		Au(g/t)	Ag (g/t)	Zn (%)	Au (g/t)	Ag (g/t)	Zn (%)	Au	Ag	Zn
N35	28	6.82	236.36	0.42	41.58	2330.99	3.94	1.58	2.32	2.01
N25	41	5.32	15.43	0.41	73.21	89.56	8.02	2.82	1.57	3.16
N24	68	5.12	45.77	0.37	75.17	749.86	9.83	2.38	2.28	3.58
N20	101	3.37	28.84	0.31	43.50	511.00	6.84	1.98	2.49	2.56
VNA8	119	3.78	26.54	0.43	84.74	333.63	7.29	2.32	2.01	2.61
VNAD9	141	10.36	44.38	0.45	396.00	786.14	13.70	4.49	2.67	3.28
VND10	172	5.81	22.83	0.36	151.62	702.00	10.45	2.87	3.03	2.83
FWV11	299	16.51	45.39	0.98	231.24	631.85	25.64	1.83	1.78	2.18
SA12	792	42.34	125.50	1.34	1337.56	9520.00	29.99	2.40	3.47	1.81
HWV2	117	6.81	37.92	0.40	129.00	1124.36	3.51	2.07	3.21	1.48
HWV1	424	16.45	29.12	0.91	634.47	634.67	11.01	3.00	1.91	1.53
SOF10	258	11.01	29.43	0.48	830.57	1428.81	7.92	4.99	4.37	2.11
SOF11	59	5.30	15.77	0.16	46.70	262.39	1.23	1.79	2.47	1.40
C10	43	13.03	21.73	0.19	109.07	216.41	0.89	1.82	1.90	0.97
C11	243	3.49	11.97	0.37	111.99	403.00	8.34	2.65	2.83	2.51
C20	86	10.91	8.96	0.42	508.47	123.09	1.90	5.05	1.85	1.11
C30	78	4.58	6.07	0.38	48.77	55.00	2.27	1.49	1.22	1.29
C40	90	5.09	6.71	0.36	66.73	39.80	2.35	1.67	1.03	1.30
C41	12	3.71	8.67	0.61	12.25	22.09	1.78	0.96	0.74	0.82
VNC18	224	4.71	12.79	0.37	66.01	350.16	7.38	1.83	2.78	1.86
CB18	220	5.33	13.97	0.43	161.18	839.48	6.71	2.89	4.27	1.84
VNB19	234	5.26	18.14	0.57	133.70	920.42	10.02	2.69	4.02	1.98
PRE20	173	3.32	10.08	0.39	30.30	231.82	10.29	1.57	2.40	2.40
VNE30	211	3.99	10.36	0.39	107.58	247.85	4.31	2.54	2.35	1.77
VNE31	80	5.59	13.30	0.77	48.80	145.48	7.00	1.71	1.68	1.48
MU3	93	2.87	11.09	0.46	23.58	113.66	13.18	1.47	1.76	3.39
MU7	211	4.53	19.23	0.50	162.71	1765.00	22.80	3.02	6.35	3.31
MU6	135	8.24	35.14	0.84	127.34	486.00	9.16	2.24	2.21	1.91
MU51	72	3.51	30.72	0.45	54.85	397.90	3.98	2.33	2.46	1.71
MU5	87	5.32	10.24	0.27	78.69	142.27	3.55	2.25	2.17	2.12
MU4	164	3.09	48.32	0.56	37.81	2170.00	14.80	1.74	4.89	2.88
MU2	113	6.05	26.05	0.53	232.34	423.50	12.60	3.83	2.32	2.78
MU21	25	3.84	18.14	0.25	28.58	77.81	1.91	1.70	1.19	1.72
MU71	102	4.61	20.73	0.28	62.10	379.85	4.14	2.05	2.63	1.93
MU8	60	3.99	18.87	0.66	44.32	216.20	8.23	1.70	2.11	2.02
MU10	64	6.06	34.46	1.07	71.39	405.54	24.70	1.89	2.22	3.13
MU9	85	2.33	23.50	0.24	32.13	568.67	2.30	1.92	3.23	1.86
MU11	83	5.81	20.47	0.56	168.83	211.00	5.00	3.24	1.82	1.72
MU1	71	9.03	21.15	0.76	231.49	215.00	8.42	3.18	1.58	1.87
MUS1	59	5.03	28.46	0.60	42.70	913.00	3.79	1.63	4.15	1.29
MUS12	52	8.29	37.89	0.85	83.48	508.00	6.58	1.75	2.37	1.73
MUS11	40	3.45	12.73	0.39	28.70	114.00	1.93	1.70	1.83	1.22
MUS10	51	6.02	12.66	0.39	190.50	119.00	4.54	4.42	1.87	1.79
MUS21	35	1.86	10.43	0.52	6.73	41.80	3.66	0.96	1.05	1.39
MUS20	63	4.07	10.20	0.35	142.93	129.00	1.94	4.42	1.76	1.40
MUS31	41	1.72	8.21	0.31	10.40	74.20	3.96	1.34	1.72	2.08
MUS30	51	2.55	15.60	0.34	20.44	206.00	2.12	1.39	2.11	1.47
MUS40	26	4.33	4.90	0.29	84.03	34.90	2.67	3.77	1.48	2.29
NWSE3	103	26.67	113.58	0.90	271.99	1320.00	13.35	1.84	1.94	2.23

Source: MA, 2015

The maximum mean composite gold grade for a vein at Yaraguá was the flagship San Antonio (SA) vein, with a mean composite gold grade of 42 g/t. The Murcielagos set of veins contained the lowest mean gold grade veins, with veins Murcielagos South 31 (MUS31) and Murcielagos South 21 (MUS21) having means of 1.72 g/t and 1.86 g/t respectively. MA included these veins, which have grades less than half the gold cut-off grade used for selection of vein intercepts, and have no economic levels of silver, but do contain some sub-domains above cut-off grade. These values are of course before application of the 2 mgms sub-domains.

Highest mean silver grade for a vein at Yaraguá was 236.36 g/t for vein N35, while the lowest mean silver grade of 4.9 g/t was found in MUS40.

Raw composite grades for gold, silver and zinc from each vein were compared against the raw composite grades from MA's 2013 resource where possible. Substantial reinterpretation has been carried out, and the amount of drilling substantially increased, especially in Veta Sur, making a direct comparison of little real value.

14.5.4 Grade Capping

Capping is the process of reducing the grade of outlier samples to values that are representative of the surrounding grade distribution. Reducing the value of an outlier sample grade minimizes the overestimation of adjacent blocks in the vicinity of an outlier grade value. At no stage are sample grades removed from the database if grade capping is applied. The risks associated with the treatment of the high grades are to potentially over (or under) estimate the contained metal of the deposit.

Gold and silver are naturally nugget (Poisson distribution) in nature and prone to outliers. Statistical parameters such as coefficient of variation and mean plots, metal loss, histograms and log probability plots were used as guides to determine the appropriate grade cap. The effect of this capping can be seen by comparing the uncapped results to the capped results (see Table 14.5 and Table 14.6).

Composite caps were applied to gold, silver, and zinc grades in each vein before estimation. Capped vs. uncapped statistics were generated for gold-silver and zinc, and are shown in full in Table 14.7 and Table 14.8. Gold cap values varied from 5 g/t to 332 g/t, silver cap values from 20 g/t to 1905 g/t and zinc cap values from 0.3% to 7.5%. The number of samples capped in any one vein generally did not exceed 3% of the total, except where there were a small number of samples.

Table 14.5: Veta Sur Gold Capped and Uncapped Data Statistics

Au	Uncapped Composite Data				Capped Composite Data				Grade	
Domain	Count	Mean	Maximum	CV	# Capped	Mean	Cap	CV	% Cap	% Δ
7	7	3.34	7.05	0.95	1	3.34	7	0.95	14%	0%
8	22	2.01	14.1	1.48	1	2	13.9	1.47	5%	0%
10	22	2.89	15.96	1.32	1	2.85	15.1	1.29	5%	-1%
12	21	1.33	6.91	1.34	1	1.32	6.5	1.31	5%	-1%
16	38	6.18	109.07	2.91	1	4.87	59.5	2.17	3%	-21%
20	54	2.53	14.56	1.49	1	2.53	14.3	1.49	2%	0%
22	50	1.89	17.77	1.73	1	1.83	14.6	1.63	2%	-3%
24	74	4.5	73.74	2.75	1	4.3	59.3	2.63	1%	-4%
26	76	3.43	43.13	2.39	2	3.25	32.7	2.28	3%	-5%
28	52	4.19	57.25	2.22	2	3.68	31.1	1.82	4%	-12%
30	97	3.05	74.11	2.87	1	2.6	30.6	2.17	1%	0%
31	116	8.52	352.22	4.16	3	5.78	68.9	2.48	3%	0%
32	123	4.52	97.4	2.65	2	3.73	37.5	1.89	2%	0%
34	156	3.13	38.76	1.76	2	3.06	26.8	1.66	1%	0%
39	175	4.7	164.51	2.88	2	3.89	34.3	1.55	1%	0%
41	141	3.37	28.25	1.67	2	3.34	26.3	1.65	1%	0%
42	157	6.54	104.98	2.29	4	5.93	57.2	1.98	3%	0%
43	179	4.98	70.49	2.19	2	4.91	60.6	2.14	1%	0%
48	194	3.93	45.77	1.71	4	3.74	29.3	1.55	2%	0%
51	188	25.16	1729.78	5.16	2	16.54	202.7	2.08	1%	-34%
62	194	19.06	924.3	3.92	2	14.5	198.1	2.24	1%	-24%
65	194	6.59	111.93	2.09	3	5.84	49	1.6	2%	-11%
70	117	6.17	155.02	2.78	2	5.46	78.5	2.16	2%	-12%
75	121	3.37	39.24	1.89	2	3.35	37.3	1.86	2%	-1%
90	181	9.36	545.56	5.43	3	4.68	74.5	2.49	2%	-50%
110	156	4.35	82.7	2.5	2	4.2	60.3	2.36	1%	-3%
123	96	30.81	2615.41	8.66	1	5.85	219.4	4.12	1%	-81%
130	67	6.56	77.3	2.28	2	6.36	70.2	2.2	3%	-3%
140	56	6.51	231	4.71	1	3.36	54.2	2.22	2%	-48%
150	67	3.79	38.8	1.76	1	3.66	30.4	1.65	1%	0%
160	50	2.98	26.2	1.76	1	2.96	24.9	1.73	2%	0%
170	31	2.5	24.38	1.78	1	2.36	19.9	1.58	3%	0%
180	45	3.45	20	1.35	1	3.45	19.8	1.35	2%	0%
181	10	1.79	5.04	0.96	1	1.79	5	0.96	10%	0%
183	10	2.99	6.11	0.72	1	2.98	6	0.71	10%	0%
185	47	2.69	26.28	1.86	1	2.59	21.6	1.76	2%	0%
190	19	2.15	11.29	1.41	1	2.08	10.1	1.36	5%	0%

Table 14.6: Yaraguá Capped and Uncapped Gold Statistics

Au	Uncapped Composite Data				Capped Composite Data				Grade	
Domain	Count	Mean	Max	CV	# Cap	Mean	Cap	CV	% Cap	% Δ
N35	28	6.82	41.58	1.58	1	6.74	39.3	1.56	4%	0%
N25	41	5.32	73.21	2.82	1	5.11	64.8	2.75	2%	0%
N24	68	5.12	75.17	2.38	2	4.66	48	2.11	3%	0%
N20	101	3.37	43.5	1.98	2	3.13	29.9	1.72	2%	0%
VNA8	119	3.78	84.74	2.32	2	3.27	24.3	1.53	2%	0%
VNAD9	141	10.36	396	4.49	3	6.14	103.1	2.89	2%	0%
VND10	172	5.81	151.62	2.87	5	4.59	45.6	0	3%	0%
FWV11	299	16.51	231.24	1.83	8	14.83	91.8	1.51	3%	-10%
SA12	792	42.34	1337.56	2.4	12	37.21	331.7	1.69	2%	0%
HWV2	117	6.81	129	2.07	2	6.11	49.1	1.5	2%	0%
HWV1	424	16.45	634.47	3	11	13	132.3	1.95	3%	-21%
SOF10	258	11.01	830.57	4.99	3	7.84	116.9	2.26	1%	0%
SOF11	59	5.3	46.7	1.79	2	5.22	44	1.75	3%	0%
C10	43	13.03	109.07	1.82	2	12.62	93.5	1.75	5%	-3%
C11	243	3.49	111.99	2.65	4	3.04	35.8	1.82	2%	-13%
C20	86	10.91	508.47	5.05	1	6.55	133.1	2.55	1%	-40%
C30	78	4.58	48.77	1.49	2	4.15	19.8	1.13	3%	-9%
C40	90	5.09	66.73	1.67	2	4.51	22.6	1.19	2%	-12%
C41	12	3.71	12.25	0.96	1	3.62	11.1	0.92	8%	-3%
VNC18	224	4.71	66.01	1.83	5	4.39	35.6	0	2%	0%
CB18	220	5.33	161.18	2.89	11	4.07	25.3	1.45	5%	-24%
VNB19	234	5.26	133.7	2.69	6	4.14	34.6	1.76	3%	0%
PRE20	173	3.32	30.3	1.57	5	3.19	20.8	1.46	3%	0%
VNE30	211	3.99	107.58	2.54	4	3.59	41.2	2	2%	0%
VNE31	80	5.59	48.8	1.71	2	5.44	36.6	1.65	3%	0%
MU3	93	2.87	23.58	1.47	3	2.74	14.4	1.36	3%	0%
MU7	211	4.53	162.71	3.02	4	3.91	50.6	2.12	2%	0%
MU6	135	8.24	127.34	2.24	4	7.49	73.2	1.94	3%	0%
MU51	72	3.51	54.85	2.33	2	3.15	29	1.98	3%	0%
MU5	87	5.32	78.69	2.25	2	4.73	43.7	1.89	2%	0%
MU4	164	3.09	37.81	1.74	4	2.93	20.4	1.57	2%	0%
MU2	113	6.05	232.34	3.83	2	4.57	68	2.29	2%	0%
MU21	25	3.84	28.58	1.7	1	3.51	20.3	1.51	4%	0%
MU71	102	4.61	62.1	2.05	3	4.2	33.5	0	3%	0%
MU8	60	3.99	44.32	1.7	1	3.57	18.9	1.31	2%	-11%
MU10	64	6.06	71.39	1.89	2	5.54	40.8	1.6	3%	-9%
MU9	85	2.33	32.13	1.92	2	2.15	18.1	1.6	2%	-8%
MU11	83	5.81	168.83	3.24	2	4.08	27.7	1.36	2%	-30%
MU1	71	9.03	231.49	3.18	2	6.56	56	1.83	3%	0%
MUS1	59	5.03	42.7	1.63	1	4.9	35	1.55	2%	-3%
MUS12	52	8.29	83.48	1.75	1	7.66	50.9	1.51	2%	-8%
MUS11	40	3.45	28.7	1.7	1	3.37	25.7	1.64	3%	-2%
MUS10	51	6.02	190.5	4.42	1	3.57	65.8	2.7	2%	-41%
MUS21	35	1.86	6.73	0.96	1	1.86	6.7	0.96	3%	0%
MUS20	63	4.07	142.93	4.42	1	2.84	65.8	2.99	2%	-30%
MUS31	41	1.72	10.4	1.34	1	1.71	10.1	1.33	2%	0%
MUS30	51	2.55	20.44	1.39	1	2.38	12.1	1.18	2%	-6%
MUS40	26	4.33	84.03	3.77	1	3.57	64.3	3.49	4%	-17%
NWSE3	103	26.67	271.99	1.84	3	25.95	206.8	1.76	3%	0%
NW2	76	7.23	117.77	2.23	1	6.53	65.1	1.81	1%	-10%
CNT1	185	22.35	306	2.04	5	20.55	167.9	1.78	3%	-8%

Source: MA, 2015

14.5.5 Hybrid Method

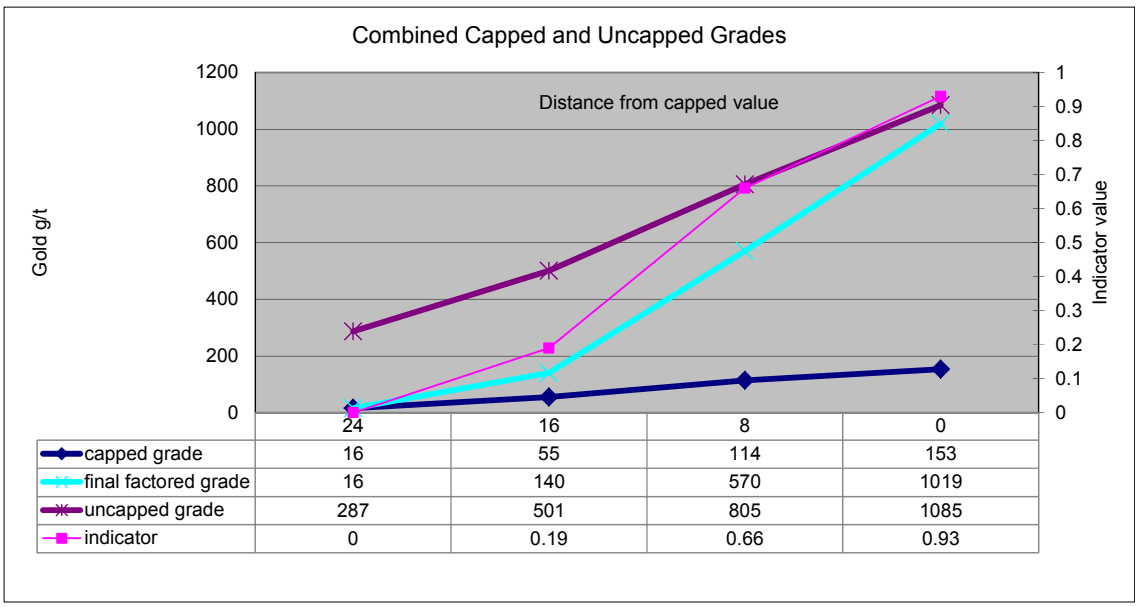
In previous estimates of Buriticá resources, as in this one, composite grades were capped for each vein utilized to create a "capped estimate". This was done because the high-grade outlier composites will have an overwhelming influence on any blocks for which they are used to estimate. Capping the grade reduces the amount of metal that will be estimated into blocks informed by these outlier samples, hopefully preventing overestimation. The outlier composites have passed QA/QC, and so are considered a real value that represents the grade of the vein at the composite location. The problem with these outlier composites is not that the grades are too high or are not considered real or reliable, but that the effect on the blocks within the range of these high-grade outlier composites will be higher than other composites.

To more effectively deal with high-grade outliers, MA considered grades for gold, silver and zinc and assessed appropriate caps. Additionally MA restricted the spatial influence of high-grade outliers by using an inverse distance squared estimated binary indicator value within a 25 m search radius of a grade cap. A hybrid grade was calculated then by combining capped and uncapped estimates weighted by the indicator value (Figure 14.6). This was only within areas of capped values, being 151 of 6,460 informing composites at Yaraguá (2%) and 62 of 3,403 at Veta Sur (again, 2%).

This approach allows the real composite grade to still have an effect within a small area of influence, without causing overestimation of blocks further away from these composites. This methodology allowed MA to use harsher grade caps for some of these composites than would have otherwise been used for fear of removing too much metal from the estimate. These harsher grade caps make for a more stable population which should result in a better estimate overall.

The degree of influence varies from 93% at the capped sample to 19% at 16 m in Figure 14.6.

Figure 14.6: Final Combination of Capped and Uncapped Values



	24	16	8	0
capped grade	16	55	114	153
final factored grade	16	140	570	1019
uncapped grade	287	501	805	1085
indicator	0	0.19	0.66	0.93

Source: MA, 2015

14.6 Variography

The most important bivariate statistic used in geostatistics is the semi-variogram (variogram). The experimental variogram is estimated as half the average of squared differences between data separated exactly by a distance vector 'h'. Variogram models used in grade estimation should incorporate the main spatial characteristics of the underlying grade distribution at the scale at which mining is likely to occur.

Variogram analysis was undertaken for gold, silver and zinc grade within each major vein domain that contain sufficient data to allow a variogram to be generated. Omnidirectional variograms were generated per vein using Surpac.

14.6.1 Methodology

Variograms were constructed for vein domain true widths, as well as Au, Ag and Zn values in all vein domains. This was performed using all the vein composites collectively as most veins had too little data to generate reliable variograms. The ranges used were the same for each element but the nugget and sills were varied.

Isotropic (omnidirectional) variograms were created because there were not enough sample data within each vein domain to give directional variogram maps. The nugget of the width was set to zero as the variability of a width measurement should be zero, i.e. if the width is measured twice at the same place in the vein, the measurement should be exactly the same, therefore nugget=0.

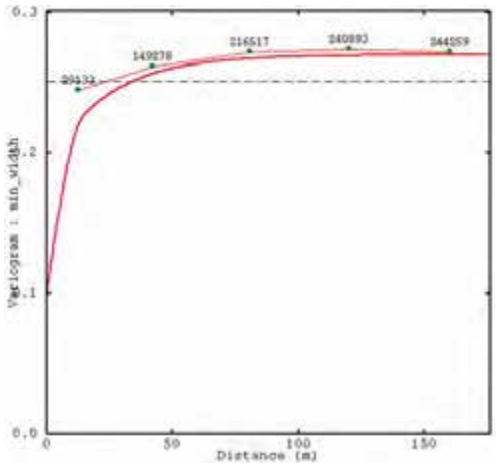
After extensive testing of changing variogram and other estimation parameters for each variable at Veta Sur, the estimation results were found to be sensible and consistent. The same parameters were used at Yaraguá and validated to ensure they were suitable.

14.6.2 Variogram Models and Parameters

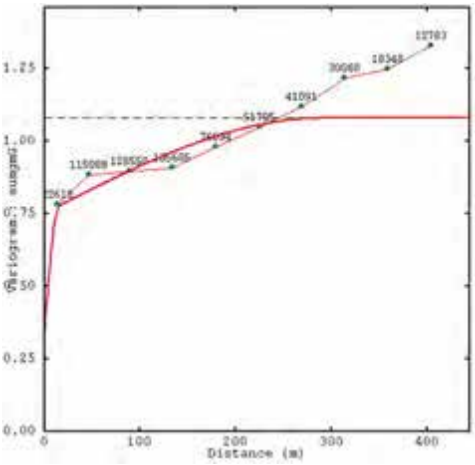
The nature of the compositing used at Buriticá (one composite across the vein) meant that typically only 100 or less composites per vein domain were available. Veta Sur veins domains 48, 43, 51, 62, and 65 each had more than 100 composites and were initially selected for variogram modeling. These domains showed some anisotropy but were inconsistent between different veins and different attributes. Without a structural model to support these trends, a single, omnidirectional variogram model for each grade attribute (gold, silver, zinc and true width) was used throughout the entire vein set, based on all the veins combined at Veta Sur.

Variograms indicate reasonable continuity between development cuts underground. CGI development mapping confirms veins have good geological continuity along strike. However vein grades can vary significantly and it is not uncommon for the grade to go from below cut-off to several hundred g/t Au in one cut. Kriging estimation parameters derived from variogram models are shown in Figure 14.7.

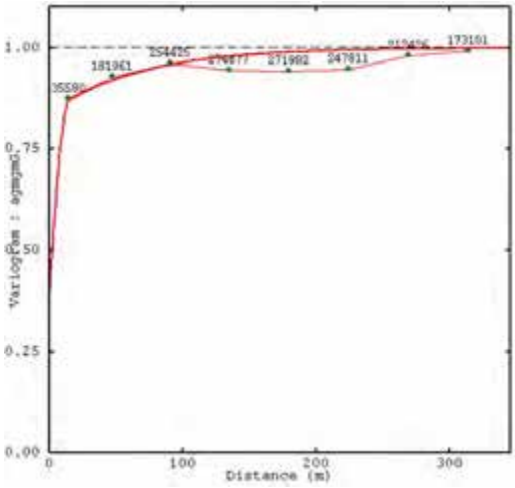
Figure 14.7: Variograms (using all vein domains) for Yaraguá and Veta Sur



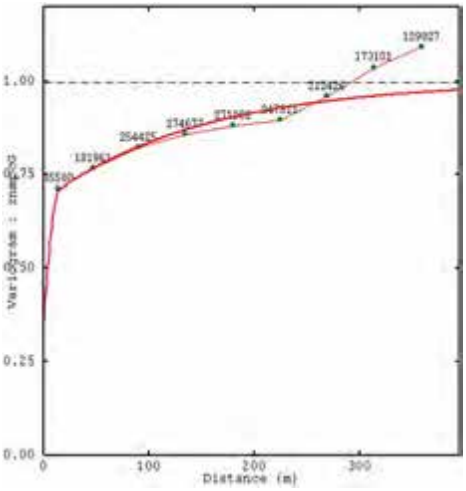
Minimum width variogram with parameters



Au accumulation variogram and parameters



Ag accumulation variogram and parameters



Zn accumulation variogram and parameters

Source: MA, 2015

High and low-grade gold accumulation variograms were constructed for use inside and outside the high and low-grade domains of each vein. The high-grade variogram, which had a range of less than 50 m, was too short and was causing smoothing of the grades or not estimating blocks at all, so the low-grade variogram was used for estimation of all gold accumulation values in both high and low-grade domains. The results of the block estimation were validated against the informing composites for every vein.

Table 14.7: Kriging parameters, for Yaraguá and Veta Sur

	Gold	Silver	Zinc	Vein Width
Major/semi	1	1	1	1
Nugget	0.3	0.4	0.4	0.3
Sill 1	0.5	0.44	0.44	0.38
Range 1	30	30	30	88
Sill 2	0.2	0.16	0.16	0.18
Range 2	200	200	200	256
Major axis direction	Omnidirectional	Omnidirectional	Omnidirectional	Omnidirectional

Source: MA, 2015

Grade Estimation

MA examined the drilling and channel data using Surpac geology and mining software package.

The methodology utilized by MA estimates the grades and true widths of veins. This is done in unfolded space using 8 m X and Y grid spacing. The estimation area is extended beyond the outer data points by expansion of a fixed distance to create a boundary perimeter; the boundary is then smoothed with the result that the expansion is reduced to less than the target at the extremities. The expansion distance is therefore a maximum, rather than a fixed value. The expansion for Buriticá is a maximum of 30 metres. Thickness at the extension boundary is set to zero.

Grade estimations are made using three different methods so that the results can be compared; these are Nearest Neighbour (capped), Inverse Distance Squared (capped) and Ordinary Krige (capped). The true widths are estimated directly using Ordinary Kriging (no capping).

14.6.3 Block Model

Two 3D block models were created, one for Yaraguá and one for Veta Sur. These models contain the same attributes and use the same estimation techniques, but were created separately to follow the strike of each region, Veta Sur veins strike differently to Yaraguá veins.

14.6.3.1 Methodology

MA's methodology using the Surpac software consists of the following steps.

- Database – validation of the drill hole database. Selection of downhole composites, if required. This was done by CGI and tags were checked by MA for Yaraguá and Veta Sur;
- Intercept Selection. The drill hole data is displayed in section and elevation slices showing assays. Intercepts are selected and coded for each vein based on the following selection criteria, in priority order:
- Grade – intervals greater than 2 g/t Au and/or 100g/t Ag;
- No assays received but a lithology code "vein" in the expected location;
- Sub-grade areas where the interpreted vein domain passed through the drill hole, but was not already coded (brought through);

- Horizontal Width Calculation - At each intercept centroid, the strike and dip of the vein was determined and the horizontal vein width calculated;
- Basic Statistics and Upper Grade Caps -Univariate statistics of grades, accumulation and horizontal width for vein composites for each vein were examined. Mean, median, standard deviation and variance were calculated for both normal and log-transformed data. A cumulative probability plot was prepared for each data set in both normal and log-transformed formats. For all veins, a break occurs at around 2 g.m. accumulation – g.m. is gold grams * metres, separating 'high grade' from 'low-grade' mineralization. Grade caps were determined from examination of univariate statistics and fields for capped gold, silver and zinc grades were added;
- Unfolding and Variography-Vein composites were unfolded into a single plane, such that the y axis became vertical depth, the x axis became parallel to the strike of the vein (in this case the easting), and the z axis (in this case the northing) was set at a dummy value of 0.The original coordinates were stored in the model for later be refolding. Variography was undertaken in this 2D space. Values for anisotropy and a two structure spherical model were recorded for grades and thickness. Where no directional variograms were clearly determined (as commonly happens with less than 100 data points, or where the data is unevenly distributed) then an isotropic variogram (all ratios = 1) is used;
- Estimation Constraint – For each vein domain an estimation boundary was manually digitized using a 2 g.m. cut-off to separate high grade and low-grade sub-domains in 2D unfolded space;
- Unfolded Grid Model and Extension– a model of the vein centre was generated using coded intercepts. Estimation by four methods for each element and vein true widths. This was done in unfolded space using selectable x and y grid spacing. The estimation area was extended beyond the outer data points by expansion of 30 m to create a boundary perimeter. The boundary was smoothed with the result that the expansion was reduced to less than the target at the extremities. Expansion distance is therefore a maximum, rather than a fixed value. In extreme cases, say where the extension is based on a single drill hole, no extension will occur at all. Estimates for grade and metal accumulation were made for capped and uncapped data using Nearest Neighbour, Inverse Distance Squared and Ordinary Kriging. Thickness was estimated directly using Ordinary Kriging. The >2g.m Au boundaries (from step 6) were used at this stage to constrain estimation to inside and outside these boundaries;
- Capped Values Influence – Inverse distance squared method within a 25 m search radius was used to estimate a binary (between 0 and 1) indicator value where the indicator cut-off was the capping value. A "hybrid" estimate for all blocks was calculated by combining capped and uncapped estimated weighted by the indicator value. This effectively preserves very high grades within the final model, but limits their spatial influence. Hybrid grades were named "final" OK grades in the 2D and 3D block models (see full description in section 14.5.5);
- Refolding and True Width Correction – The grid was re-folded to its original 3D position. This was done by replacing the dummy northing with the stored real northing. Some smoothing of the surface using surface modeling algorithms (not geostatistics) was undertaken; this removed local spikes and steps due to clustering of data. Changes were small, generally less than half the grid spacing. The "slope" of the surface in 3D space relative to the 2D surface was measured as a percentage gradient; this value was recorded as it is similar to that used in "Connolly Diagrams". The True Width value was then corrected using this factor. Note that

"slope" value is measured at each node of the grid and is a function of the surface geometry; the more the surface moves from the normal the greater the correction – in effect an "auto-correction". This is much better than using an average strike and dip for the surface (too general), a drill core measurement (too local) or geostatistics (too smoothed);

- Vein Halo Definition and Halo Grade Estimation – a number of steps were performed to generate a halo surrounding veins into which grades are estimated: (see full description in section 14.1);
- Extrapolation – at each vein intercept a halo was generated by extrapolating ± 2 m horizontally from the vein boundaries in the plane of the vein;
- Tag generation – halo tags were generated in the database for this material, designated as the vein code with a suffix "_F" for footwall and "_H" for hangingwall. Since all veins are sub-vertical, the north side was consistently nominated as footwall;
- If any sample lengths within a vein halo were less than 1 m (most likely for channel sampling), then the sample was diluted by zero grade to a 2 metre sample;
- Halo grade composites were unfolded and estimated into the 2D model within designated footwall and hangingwall blocks;
- Mineralized halos were estimated using the same parameters as the host vein;
- Dilution – veins less than the minimum mining width of 1 m were diluted to a full 1 m length with either footwall or hangingwall halo material, depending on which is the higher grade. The vein contact was pushed the appropriate distance into the footwall or hangingwall as required;
- Mineral Resource categories – Resource categories were defined by manual digitizing in long section view for each vein, based on a combination of the number of informing samples, sample distances and kriging variance. Mineral resource categories were stored in the block model;
- Blocks that occur in the footwall and hangingwall halos were also categorized by pushing out the categorizations from the vein into both the hangingwall and the footwall for the purpose of converting Mineral Resources to Mineral Reserves (Section 15). No measured category was assigned to the footwall or hanging wall mineralization. The mineral resource reported herein does not contain blocks from outside of the 1 m minimum width vein wireframes;
- Solid Creation – The 3D centre plane of the vein was converted to a closed 3D solid. Footwall and hangingwall surfaces were created by translating the 3D centre plane half the width of the vein to the north and south respectively. These were then joined at the edge, which is a common boundary, to create a vein solid;
- Block Model – Volumes from the final closed 3D solids were used to flag blocks in the final 3D block model for each vein. Variables from the solids, including grades, widths, slope, kriging variance, number of informing samples, nearest drill hole name and distances, etc., were all stored in the block model. Each vein block was given a vein name and number. If any 3D solids overlap spatially, priority for block assignment is given in the following order:
- Resource category – grades for the highest confidence resource category are used if categories are different

- Grade – if categories are the same, then priority is given to the higher grade (based on Au equivalent)
- Bulk Density – Bulk densities for each block below the topographical surface were set to a constant value, using 3.1t/m^3 for vein material and 2.8 t/m^3 for waste. For veins diluted with halo material, assigned densities were calculated using a length weighted average of vein and waste material;
- Null Blocks – blocks that effectively represent empty space were flagged as air (above the original topography), pit (mined out in an open pit), or stoped (removed by underground mining);
- Validation – Values within the block model were compared to informing drill composites. Basic statistics for block model and drill composites were compared. Distributions of grades in space (by elevation and northing) were compared. Blocks nearest to drill holes were compared with informing drill holes. Estimates using different estimation methods were compared in total and above cut-off;
- Reporting – Resource can be reported by resource category, by vein, by cut-off grades, by different methods (sensitivity to method and upper cuts), by elevation (tonnes per vertical metre), by X and Y dimensions.

14.6.4 Domaining

A domain is a volume that delineates the spatial limits of a single grade population, has a single orientation of grade continuity, is geologically homogeneous and has statistical and geostatistical parameters that are applicable throughout the volume (i.e. the principles of stationarity apply). Typical controls that can be used as the boundaries to the domains include structural features, weathering, mineralization halos, and lithology.

Due to the nature of narrow high-grade veins each vein was treated as a geological domain. It is assumed stationarity is achieved within each vein.

A preliminary domain modeling and mineral resource estimation run was made for each vein domain. Areas within some vein domains were located where a group of composites all had a significantly higher gold grade than the surrounding composites. These areas were separated into sub-domains by a 2 m x grams gold boundary and individually estimated to reduce the effect of “smearing”, or mixing of high and lower grade domains. Samples and volumes inside these sub-domains were selected and estimated separately.

The 2 m x grams boundary prevents very high grades, which in Yaraguá are commonly clustered around underground workings, from smearing out to overprint areas with low grades. Not all veins required a 2 m x grams gold boundary.

14.6.5 Block Model

The Buriticá 3D block model (Table 14.8) used regular shaped blocks measuring 4.8 x 16 x 16 m (YXZ, where Y is the across-strike direction), although the width is estimated during the unfolding process and is diluted to a minimum of 1 m. The choice of the block size was patterned with the trend and continuity of the mineralization, taking into account the dominant drill pattern and size of veins. The orientation of the block model is normal to the direction of the strike. To accurately measure the volume of the mineralized wireframe inside each block, volume sub-blocking to 0.3 x 1 x 1 m was used. Blocks above the topography were tagged and excluded from the model estimation, as were mined out areas. Two block models were used, one for Yaraguá and one for Veta Sur. Both models were rotated so that the X axis was parallel to the dominant strike direction of the different vein sets.

Table 14.8: 3D Block Model Parameters

Coordinate System	Corner	Yaraguá			Veta Sur*		
		Northing	Easting	RL	Northing	Easting	RL
NAT	Bottom left	1,232,950	1,129,000	200	1,232,100	1,128,725	350
	Top right	1,233,650	1,130,200	2,008	1,232,901.60	1,130,101	2,014
Estimation Block Size		NA	16	16	NA	16	16
Volume/ Block Size		0.3	1	1	0.3	1	1
Rotation		-10°			-35°		

*Note: Veta Sur block model NAT corner coordinates have been calculated before rotation is applied

Source: MA, 2015

14.6.6 Panel Size

Panel size selection for 2D estimation was based on the current drill pattern, sample density and mining method: overly small blocks were avoided to minimize smoothing and bias. Grade estimation used Ordinary Kriging with 2D estimation blocks sized 8 x 8 m within the plane of the vein to reflect the density of drilling and hence number of informing samples.

The 3D Buriticá block model used regular shaped blocks measuring 4.8 x 16 x 16 m (YXZ). A sub-block size of 0.3 (N) x 1 (E) x 1 m (RL) was used against all wireframes for volumes.

14.6.7 Block Model Attributes

Block model attributes are shown in Table 14.9.

Table 14.9: Block Model Attributes

Attribute Name	Description
a01_au_ok_fn	Au ok estimate final diluted grade
a02_ag_ok_fn	Ag ok estimate final diluted grade
a03_zn_ok_fn	Zn ok estimate final diluted grade
a04_pb_ok_fn	Pb ok estimate final diluted grade
a05_material	HH =Halo Hangingwall, VH = Vein diluted HH, VV = vein, VF = Vein diluted HF, FF = Halo Footwall, W = Waste
a13_bd	bulk density
a18_ctw	corrected true thickness
a19_dw	diluted thickness
a20_rescat	1 measured, 2 indicated, 3 inferred, 4 undf, 5 mined out, 6 waste
aueq	gold-silver at 1:60
vein_name	Vein Name
Diluted_Au	Diluted Au ok estimate
Diluted Ag	Diluted Ag ok estimate
Diluted Zn	Diluted Zn ok estimate
Diluted Pb	Diluted Pb ok estimate

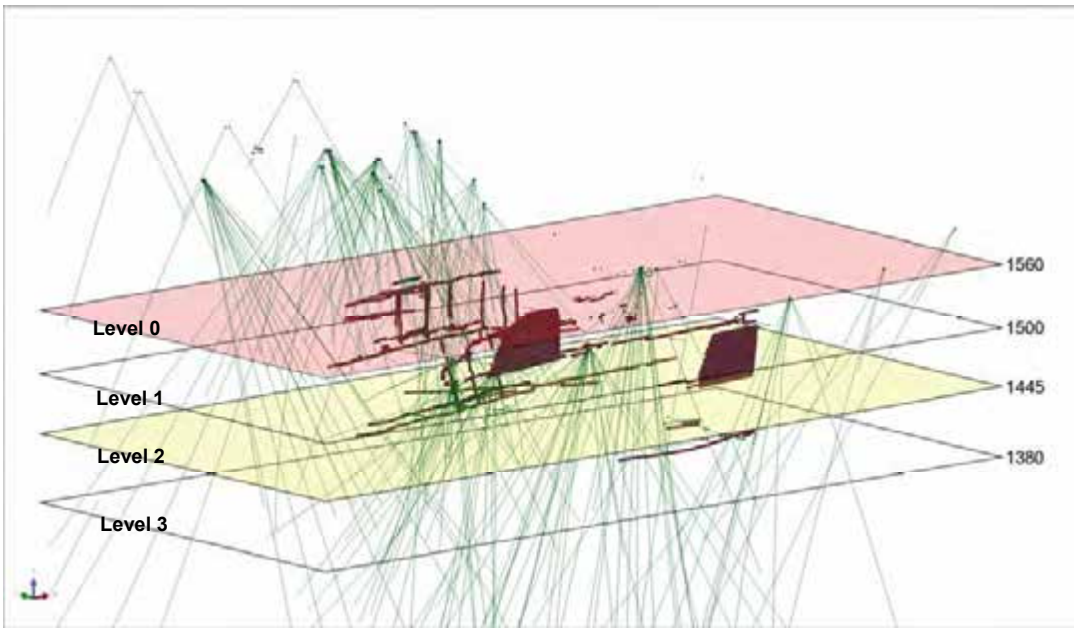
Source: MA, 2015

14.6.8 Previous Mining

Mining has only been carried out in the Yaraguá area of Buriticá, in parts of the San Antonio (SA) vein and in some parts of the Murcielagos vein domains. Areas of known small-scale mining at Yaraguá that have been surveyed have been excluded from the current resource estimate. Digitized cross sections of the development tunnels and stopes were used to remove the blocks that were mined from the model prior to resource reporting. The blocks removed from the resource model are illustrated in Figure 14-8. Mined out areas at Veta Sur were also allowed for.

3D sub-blocks of 1 x 0.3 x 1 m (XYZ) are permitted to represent the volume and mined out tunnels. Each block is then selected as being inside or outside the stope boundary.

Figure 14.8: All underground working blocks within the Yaraguá mine area viewed from the East. Numbers indicate RL



Source: MA, 2015

14.6.9 Search Parameters

Search radii were dictated by both the sampling grid and variogram features, i.e. anisotropy and ranges of variogram dictate the basic shape and size of the search ellipse which can be reduced or increased depending on the sampling grid and in such a way that, ideally, between eight and 12 samples would be captured in the ellipsoid around each block.

Drill densities at this stage of exploration in some areas necessitate search radii greater than indicated by variography. All veins and halos were estimated using an isotropic search of 200 m for grade and 256 m for vein width.

14.6.10 Informing Samples

Informing samples were vein composites within domains, one composite sample across each vein or halo intercept. Considerable care was taken to ensure no bias was introduced, and the new method tested against previous estimates. The informing sample grades were capped on a domain or sub-domain basis as determined by geostatistical analysis. A maximum of eight informing samples were used to estimate a block. As well as the tightly-constrained vein domains, high- grade and low-grade sub-domains were used within the vein domains, using a 2 m x gram gold boundary with at least one sample within that boundary.

14.6.11 Estimates by Vein

The May 2015 estimates by vein, above a 3 g/t Au krig cut-off, are shown in Table 14-10 and Table 14-11 clipped for topography and mined out areas (see “Notes to Accompany Resource Statement” at subsection 14.15.1).

Table 14.10: Veta Sur May 2015 Combined Measured, Indicated, and Inferred Estimates by Vein

Vein name	Tonnes	Au (g/t)	Ag (g/t)	Zn (%)
VS7_V	40,039	4.2	7.8	0
VS8_V	104,036	5.7	4.6	0
VS10_V	158,178	6.3	6.1	0
VS12_V	56,358	5.4	7.6	0
VS16_V	561,189	11.7	24.7	0.1
VS20_V	304,466	6.1	40.6	0.2
VS22_V	359,146	4.9	37.2	0.1
VS24_V	334,468	14.4	22.9	0.4
VS26_V	262,318	12.8	34.6	0.1
VS28_V	217,823	10.1	24	0.2
VS30_V	191,937	9.9	21.5	0.1
VS31_V	523,272	20.5	33.6	0.1
VS32_V	505,682	11.1	76.3	0.1
VS34_V	471,424	6.2	25.9	0.1
VS41_V	261,064	10	32.2	0.2
VS42_V	436,393	13.4	39.4	0.2
VS43_V	562,400	10.9	28.9	0.2
VS39_V	852,444	7.8	22.3	0.1
VS48_V	760,283	8.4	26.9	0.1
VS65_V	402,115	9.2	59.7	0.6
VS62_V	618,645	16.8	63.5	0.2
VS51_V	329,351	14	76.2	0.3
VS70_V	191,652	9.8	27.1	0.1
VS75_V	148,016	6.1	34.6	0.2
VS90_V	290,601	11.3	29.2	0.3
VS110_V	344,761	9.6	27	0.3
VS120_V	347,322	11.3	16.9	0.1
VS123_V	202,362	22.8	27.4	0.2
VS130_V	413,946	8.3	10.8	0.1
VS140_V	188,929	6.9	14.3	0.2
VS150_V	341,147	6.1	16.9	0.2
VS160_V	181,080	4.7	7.3	0.1
VS170_V	46,922	4.4	17.6	0.2
VS180_V	140,576	5.3	58.5	0.2
VS181_V	11,437	3.2	20.7	0.1
VS183_V	43,338	4.1	33.1	0.3
VS185_V	85,820	6.9	28.8	0.1
VS190_V	9,040	3.6	20.5	0.5
Total	**11,299,980**	**10.4**	**33**	**0.2**

Source: MA, 2015

Table 14.11: Yaraguá May 2015 Combined Measured, Indicated, and Inferred Estimates by Vein

Vein name	Tonnes	Au g/t	Ag g/t	Zn%
N35_V	336,844	5.2	69.2	0.4
N25_V	441,194	6.8	34.6	0.2
N24_V	597,597	11.3	50.7	0.2
N20_V	428,494	6.5	35.4	0.4
VNA8_V	753,386	7.5	33.7	0.5
VNAD9_V	638,332	9.3	38.3	0.4
VND10_V	612,150	8.6	30.5	0.3
FWV11_V	563,714	10.9	37.8	0.9
SA12_V	976,289	16.5	48.8	0.6
HWV2_V	509,258	8.4	52.9	0.4
HWV1_V	852,788	12.5	41.2	0.7
SOF10_V	884,492	12.1	17.6	0.4
SOF11_V	203,831	11	16.9	0.1
C10_V	174,253	23.7	35.7	0.2
C11_V	307,852	6.1	14.4	0.4
C20_V	177,943	15	12.6	0.3
C30_V	217,859	5.6	7	0.4
C40_V	412,372	7.7	9.2	0.3
C41_V	25,414	3.4	8.3	0.7
VNC18_V	577,583	7	13.6	0.4
CB18_V	524,880	7.4	12.4	0.4
VNB19_V	477,816	7.3	18.1	0.5
PRE20_V	683,270	6.7	8.9	0.3
VNE30_V	328,867	7.5	15.1	0.4
VNE31_V	220,913	11.8	19.2	0.5
MU3_V	139,571	5.7	13.7	0.9
MU7_V	676,097	8.7	15.1	0.3
MU6_V	411,538	9.5	36.6	0.6
MU51_V	355,725	10.2	19.8	0.3
MU5_V	204,183	5.7	10.1	0.3
MU4_V	479,044	5.4	23.8	0.3
MU2_V	232,413	9.4	37.7	0.4
MU21_V	229,931	8.4	22.3	0.3
MU71_V	366,806	8.9	31.5	0.4
MU8_V	72,336	6	19.1	0.4
MU10_V	111,581	9.5	52.7	0.9
MU9_V	206,147	6.1	74.6	0.5
MU11_V	227,370	7.6	32.7	0.6
MU1_V	189,117	9.9	21.4	0.6
MUS1_V	206,961	6.5	19.9	0.7
MUS12_V	157,354	10.6	40.2	0.8
MUS11_V	66,175	6.4	17.9	0.5
MUS10_V	136,837	12.7	22.4	0.7
MUS21_V	44,210	3.8	13.9	0.6
MUS20_V	99,308	15.5	13.4	0.4
MUS31_V	49,941	3.6	18.4	0.2
MUS30_V	113,724	6.1	17.1	0.5
MUS40_V	47,000	18.8	6.4	0.3
NWSE3_V	159,041	16.2	50.6	0.6
NW2_V	64,071	10	24.5	0.6
CNT1_V	253,723	16.6	92.5	0.8
Total	**17,227,595**	**9.5**	**30.2**	**0.5**

Source: MA, 2015

14.7 Validation and Comparison with Previous Estimates

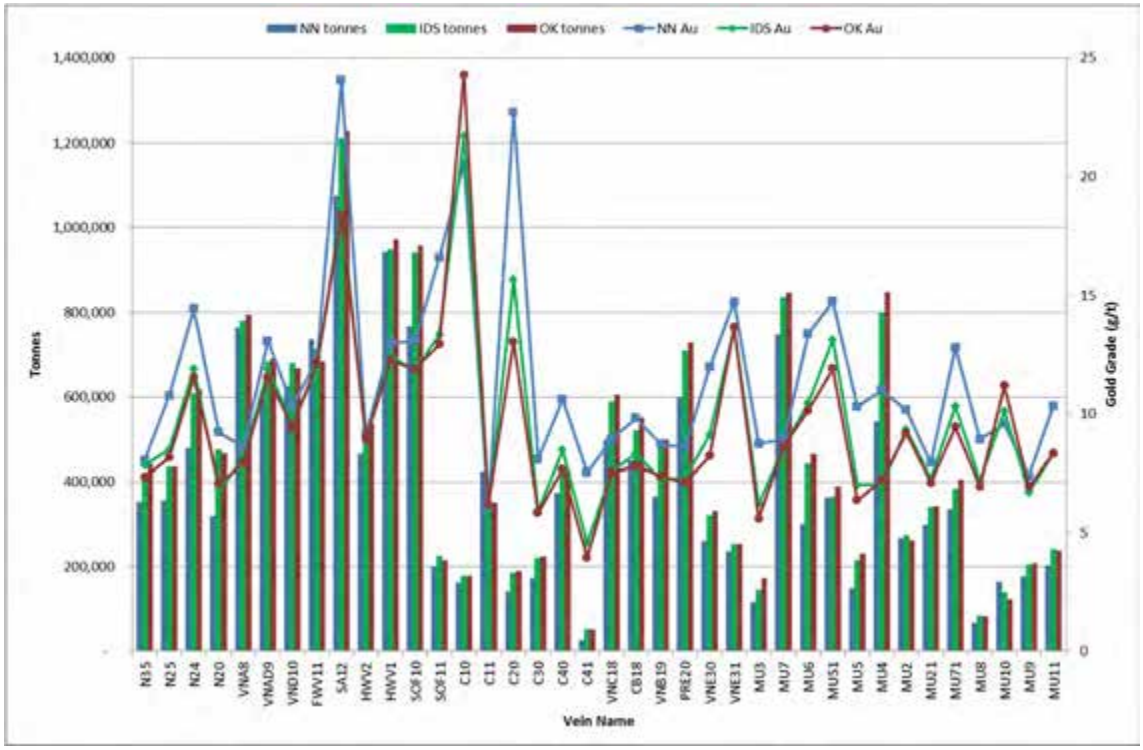
14.7.1 Validation

Gold, silver and zinc grades were estimated using three different estimation techniques: Ordinary Kriging (OK), Inverse Distance Squared (IDS) and Nearest Neighbour (NN). Results of the three estimation methods by vein for gold in Yaraguá and Veta Sur are shown in the tables below. Total ounces of gold for each estimation technique are all within 2% of each other but NN gives lower tonnes and higher grades. MA considers this an acceptable difference that confirms estimations are being done correctly.

Table 14.12: Comparison of 2D Estimation Methods for Veta Sur (not cut for topo or mined out)

Vein	NN Values			IDS Values			OK Values		
	Tonnes	Au	Oz	Tonnes	Au	oz	Tonnes	Au	Oz
VS7	78,380	6.6	16,632	92,606	6	17,864	96,857	5.92	18,435
VS8	119,105	7.39	28,299	115,982	6.67	24,872	107,194	6.61	22,780
VS10	245,542	7.5	59,208	201,922	7.76	50,377	172,625	8.53	47,342
VS12	83,158	5.32	14,224	83,158	5.18	13,849	83,158	5.14	13,742
VS16	557,575	14.72	263,877	612,254	12.79	251,764	621,491	12.65	252,765
VS20	634,822	6.81	138,992	636,346	5.77	118,048	572,829	5.73	105,529
VS22	434,615	5.41	75,595	448,745	5.01	72,282	467,433	5.01	75,292
VS24	330,226	15.27	162,122	341,308	15.39	168,879	356,280	15.64	179,151
VS26	261,404	15.28	128,418	265,413	14.11	120,404	266,210	13.43	114,945
VS28	199,862	11.96	76,852	226,155	10.76	78,237	224,982	10.54	76,239
VS30	161,856	12.55	65,308	201,101	10.62	68,664	197,332	10.8	68,519
VS31	473,382	23.83	362,683	693,818	17.07	380,776	747,011	16.43	394,599
VS32	450,318	14.44	209,063	510,465	13.12	215,323	527,683	12.37	209,862
VS34	550,889	7.35	130,179	613,417	6.65	131,150	649,304	6.39	133,395
VS39	992,250	9.38	299,237	1,057,743	8.89	302,324	1,060,576	8.88	302,793
VS41	268,737	10.71	92,535	263,612	10.69	90,601	266,779	10.79	92,547
VS42	656,393	11.24	237,204	707,630	10.22	232,513	733,157	10.03	236,423
VS43	701,196	11.14	251,140	783,991	10.29	259,369	779,000	10.53	263,728
VS48	749,188	9.67	232,921	807,029	8.75	227,033	832,317	8.71	233,076
VS51	321,804	15.36	158,918	342,662	15.27	168,227	350,496	14.6	164,523
VS62	554,057	17.57	312,980	653,828	18.13	381,112	677,392	17.84	388,531
VS65	437,442	10.06	141,485	451,790	9.3	135,086	461,057	9.05	134,151
VS70	108,203	12.68	44,111	197,848	9.55	60,747	203,174	10.13	66,171
VS75	132,640	7.49	31,941	160,451	6.58	33,944	157,573	6.55	33,183
VS90	292,564	11.92	112,121	312,257	11.04	110,834	306,952	10.85	107,076
VS110	419,254	12.06	162,561	405,112	11.54	150,304	373,924	11.59	139,334
VS120	386,027	12.71	157,745	383,613	11.97	147,631	373,606	11.67	140,177
VS123	277,960	10.51	93,924	255,822	11.76	96,724	257,709	13.33	110,446
VS130	277,646	18.45	164,694	435,822	13.31	186,500	460,987	12.18	180,521
VS140	212,869	9.62	65,838	202,291	8.85	57,559	208,669	7.93	53,201
VS150	420,107	7.07	95,493	453,744	6.11	89,134	442,381	6.46	91,880
VS160	280,725	10.57	95,400	272,949	9.71	85,210	274,688	8.91	78,688
VS170	34,373	7.86	8,686	47,048	6.21	9,393	61,279	5.54	10,915
VS180	562,098	9.9	178,911	868,560	6.31	176,206	882,888	5.91	167,758
VS181	12,313	3.37	1,334	12,313	3.92	1,552	12,313	4.01	1,587
VS183	36,137	4.84	5,623	45,115	4.2	6,092	45,115	4.25	6,165
VS185	328,913	10.95	115,794	460,197	7.37	109,044	496,324	7.03	112,179
VS190	28,544	7.32	6,718	22,084	5.83	4,139	9,436	8.93	2,709
VetaSur	13,072,574	11.42	4,798,765	14,646,201	10.27	4,833,769	14,818,181	10.14	4,830,358

Figure 14.9: Comparison of 2D Estimation Methods for Yaraguá (not cut for topo or mined out)



Source: MA, 2015

Table 14.13: Comparison of Buriticá Total Resources by Different Estimation Methods, above 3g/t Au Cut-offs

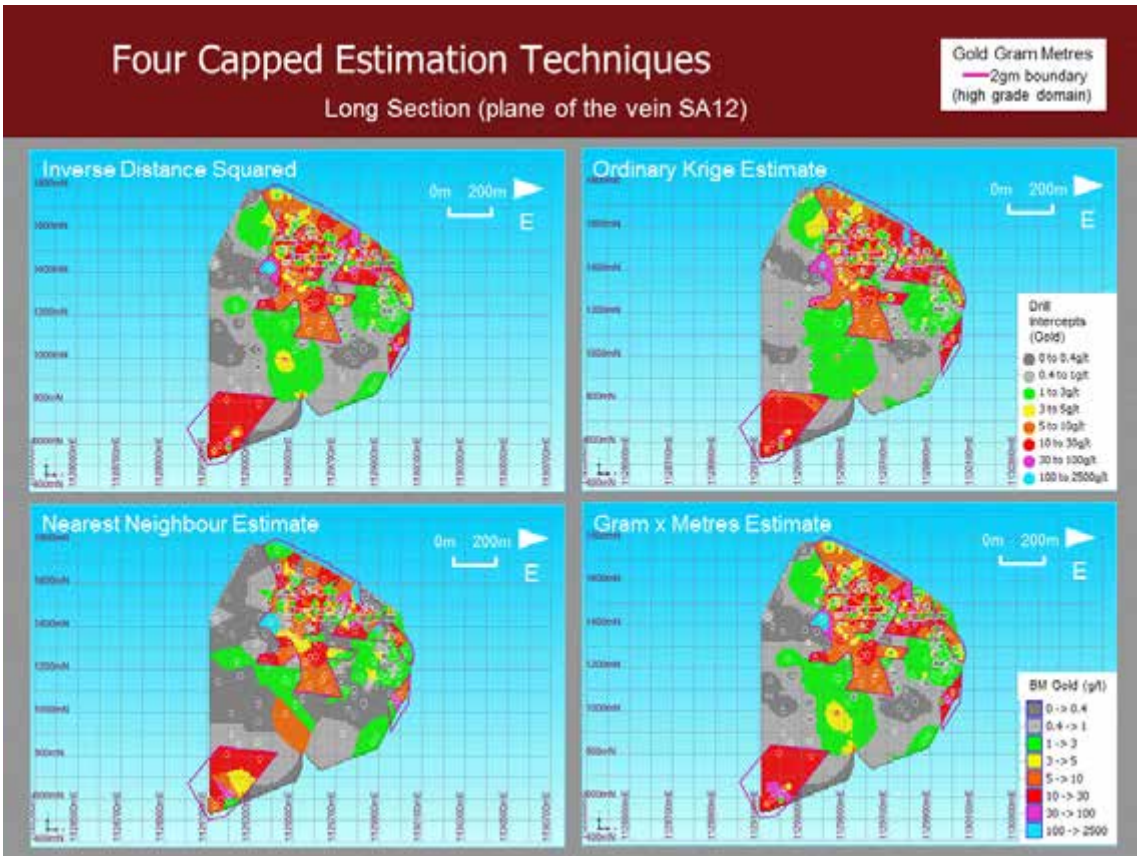
Vein System	NN Values*			IDS Values *			OK Values *		
	tonnes	Au	oz	tonnes	Au	oz	tonnes	Au	oz
Veta Sur	13,073,000	11.42	4,799,000	14,646,000	10.27	4,834,000	14,818,000	10.14	4,830,000
Yaraguá	16,656,000	12.36	6,618,000	19,078,000	10.58	6,490,000	19,503,000	10.31	6,462,000
Totals	29,729,000	11.95	11,417,000	33,724,000	10.44	11,324,000	34,321,000	10.23	11,292,000

*"Raw" 2D estimates: undiluted and not depleted by mining and topography

Source: MA, 2015

The results of the four estimation methods for San Antonia Vein ("Vein SA") from Yaraguá are shown in Figure 14.10. The ounces of gold for each estimation technique are all less than 1% of each other (Table 14.13). MA considers this an acceptable difference that confirms that the estimations are being done correctly.

Figure 14.10: Vein SA showing Informing Samples for Gold and Estimated by the Four Different Techniques



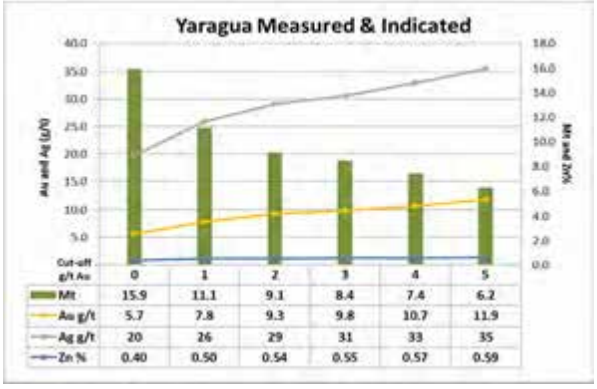
Source: MA, 2015

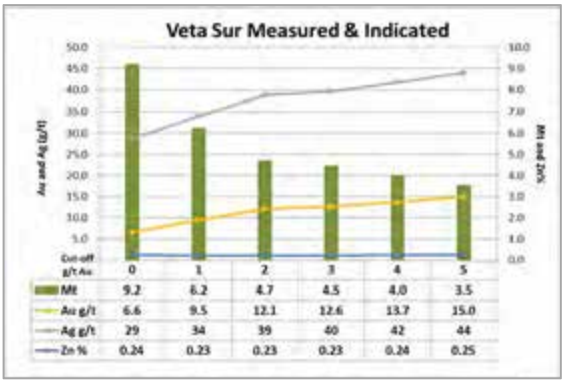
Results as tabulated show an acceptable comparison in grade and tonnes. Alternative estimation methods performed as expected due to sparse data compared to the unbiased estimation abilities of Ordinary Kriging. The Ordinary Kriging method was selected as best representing the Buriticá resource. Grade capping is shown to reduce the grade with a slight impact on tonnes.

14.8 Economic Cut-off Parameters

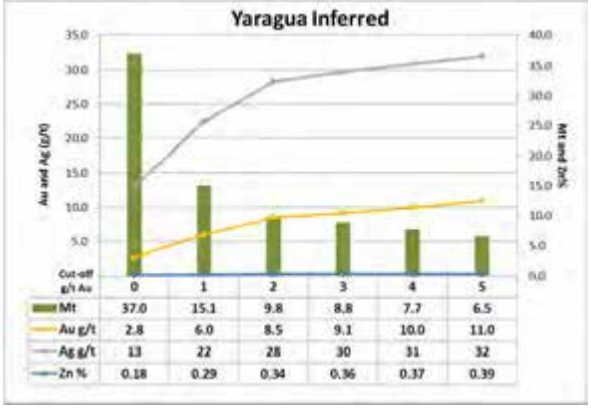
Grade-Tonnage charts for 0 to 5 g/t gold cut-off grades are presented in Figure 14-11. By mineral resource category for Yaraguá and Veta Sur vein systems. These are the final OK estimates for vein domains, diluted to one metre minimum horizontal widths using estimated halo grades.

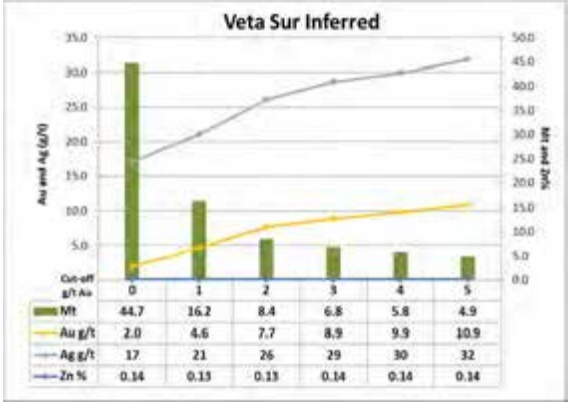
Figure 14.11: Grade-tonnage Charts by Resource Category and Vein System



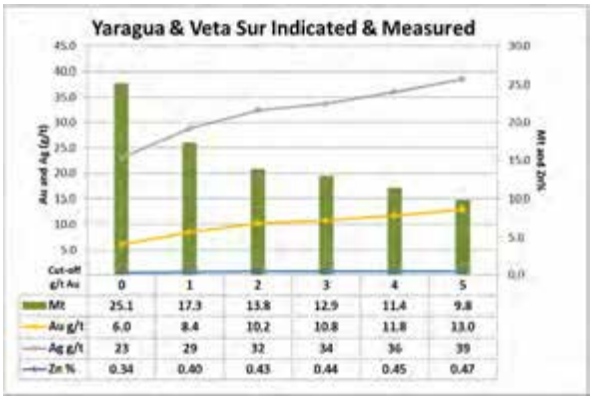


Grade-tonnage charts for Measured and Indicated resource categories by vein system





Grade-tonnage charts for the Inferred resource category by vein system



See "Notes to Accompany Resource Statement",14.15
Source: MA, 2015

14.9 Bulk Density

A constant value of 3.1 t/m^3 as supplied by CGI was used as the dry bulk density of vein domains. This value was chosen as the mean of several hundred measurements of specific gravity of sulphide-rich core samples and more than five hundred densities for individual vein composites computed from modal data calculated from complete multi-element assays. Halo material around veins were assigned a value of 2.8 t/m^3 based on specific gravity measurements. Veins diluted with halo material were assigned densities calculated using a length weighted average of vein and halo material. It is noted no moisture measurements are recorded; moisture content can affect the bulk density of the rocks, potentially introducing a bias in the tonnage estimate.

14.10 Mining and Metallurgical Factors

No mining factors have been applied to the in-situ grade estimates for mining dilution or loss as a result of the grade control or mining process. No metallurgical factors have been applied to the in-situ grade estimates.

MA notes that variations in silver, lead and zinc grades would suggest some variability in metallurgy is likely to exist and suggests that this be examined in future selective test work.

14.11 Assumptions for Reasonable Prospects for Eventual Economic Extraction

Final resource numbers in this report were reported above a gold grade of 3 g/t, and a gold-silver equivalent grade (AuAgEQ=Au grade + (Ag grade/60)) of 3 g/t. These gold cut-off grades reflect the conceptual costs for underground development, mining and treatment.

There have been no assumptions made as to metal prices or recoveries in this mineral resource estimate other than gold equivalents that are calculated for AuEq = Au + Ag/60.

14.12 Resource Classification

Yaraguá and Veta Sur veins were classified into three resource categories, and the results were reported above a 3 g/t Au cut-off. In addition, a resource was reported below a 3 g/t Au cut-off and above a 3 g/t AuEq cut-off. AuEq is calculated as Au + Ag/60. It is MA's view that these gold cut-off grades reflect the range of conceptual costs for underground development, mining and treatment.

A mineral resource is a concentration or occurrence of diamonds, natural solid inorganic material, or natural solid fossilized organic material including base and precious metals, coal, and industrial minerals in or on the earth's crust in such form and quantity and of such a grade or quality that it has reasonable prospects for economic extraction. The location, quantity, grade, geological characteristics and continuity of a mineral resource are known, estimated or interpreted from specific geological evidence and knowledge. (CIM 2014)

Dilution envelopes or halos have also been categorized for the purpose of converting Mineral Resources to Mineral Reserves (Section 15). Blocks that occur in the footwall and hangingwall halos of the main veins are categorized by pushing out the inferred or indicated categorizations in the vein into both the hangingwall and the footwall. The mineral resource reported does not contain blocks from the dilution halos that occur outside of the 1 m minimum width vein wireframes.

A breakdown of the Buriticá Project resource estimate by resource category is provided below (Tables 14.14 and 14.15).

Table 14.14: Combined Yaraguá and Veta Sur Mineral Resources above a 3 g/t Au Cut-off, Effective May 11, 2015

Category	M tonnes	Au g/t	Ag g/t	AuEq g/t	Zn %	Au Moz	Ag Moz	AuEq	Zn Mlb
Measured	0.89	19.0	55	19.9	0.7%	0.54	1.58	0.57	13.4
Indicated	12.00	10.2	32	10.7	0.4%	3.94	12.40	4.14	112.6
M and I	12.89	10.8	34	11.4	0.4%	4.48	13.98	4.71	126.0
Inferred	15.6	9.0	29	9.5	0.3%	4.5	14.7	4.8	91

Note – Reported tonnage and grade figures have been rounded from raw estimates to reflect the order of accuracy of the estimate. Minor variations may occur during the addition of rounded numbers. There have been no assumptions made as to metal prices or recoveries in this mineral resource estimate other than gold equivalents that are calculated for AuEq = Au + Ag/60. M in Figures and Tables is millions.

Source: MA, 2015

Table 14.15: Combined Yaraguá and Veta Sur Additional Resources below a 3 g/t Au cut-off and above a 3 g/t AuEq cut-off, Effective May 11, 2015

Category	M tonnes	Au g/t	Ag g/t	AuEq g/t	Zn %	Au Moz	Ag Moz	AuEq	Zn Mlb
Measured	0.03	2.5	53	3.4		0.00	0.06	0.00	
Indicated	0.28	2.5	53	3.4		0.02	0.48	0.03	
M and I	0.31	2.5	53	3.4		0.03	0.54	0.03	
Inferred	1.2	1.9	96	3.5		0.07	3.6	0.1	

Note – Reported tonnage and grade figures have been rounded from raw estimates to reflect the order of accuracy of the estimate. Minor variations may occur during the addition of rounded numbers. There have been no assumptions made as to metal prices or recoveries in this mineral resource estimate other than gold equivalents that are calculated for AuEq = Au + Ag/60. M in Figures and Tables is millions

Source: MA, 2015

Results of the 2D estimation for each vein domain were combined to a normal 3D block model with sub-block size of 1 m (E) by 0.3 m (N) by 1 m (RL). The model was screened for topography and existing workings. The number of informing samples, presence of underground development or mining, clustering of data, and distance to nearest informing sample were used by the QP to digitize contiguous areas defining resource categories in longitudinal section using the following guidelines below.

14.12.1 Measured Mineral Resource

- A contiguous zone with at least eight informing vein composites;
- Maximum of eight informing vein composites used for each block estimate;
- Distance to the nearest informing sample is generally less than 10 m but not more than 15 m; and
- Conditional Bias Slope is generally greater than 0.8 but not less than 0.6.

14.12.2 Indicated Mineral Resource

- A contiguous zone with at least four informing vein composites;
- Between four and eight informing vein composites used for each block estimate;
- Distance to the nearest informing sample is generally less than 25 m but not more than 40 m internally; and
- Conditional Bias Slope generally greater than 0.5.

14.12.3 Inferred Mineral Resource

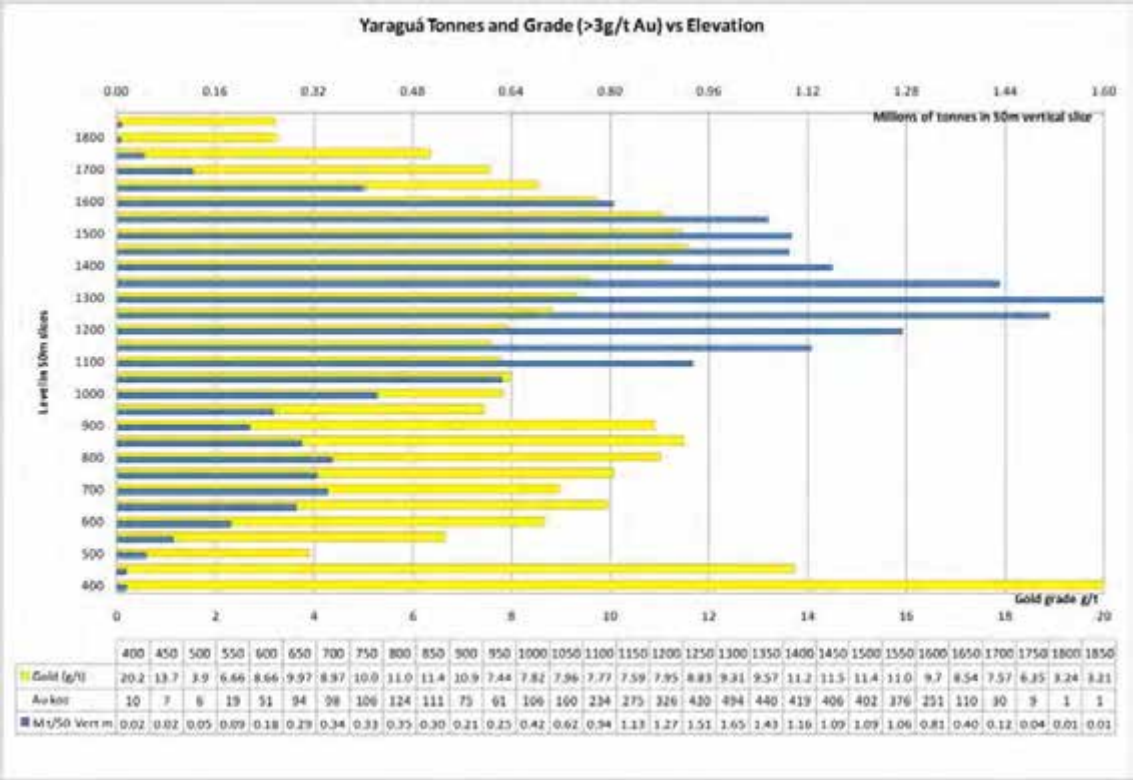
- A contiguous zone with at least three informing vein composites;
- Generally between three and eight informing vein composites used for each block estimate;
- Distance to the nearest informing sample is generally less than 100 m;
- Some minor additional areas with at least one informing sample within 150 m are included at depth where geological continuity is good but drilling is sparse; and
- Areas within a vein domain already taken by development and/or historical mining have been removed and screened for topography. Volumetric estimates were converted to tonnage estimates utilizing an average specific gravity of 3.1, based on core measurements and computed from multi-element assay data.

14.13 Vertical Distribution of Resources

A chart of tonnage and gold grade (calculated at 50 m vertical intervals) versus elevation (1,500 m range) for Yaraguá (above 3 g/t gold cut-off) is presented in Figure 14.12 and illustrates the character of the Mineral Resources at Yaraguá over a 1,000 m vertical range.

Level slice tonnages decrease above elevations of 1,600 m due to a lack of shallow drilling and due to various veins intersecting with the topographic surface. Tonnages decrease below elevations of 1,000 metres due to sparser drilling data, resulting in limited modeling of vein domains at depth. Gold grades otherwise show limited variation over a 1,400 m elevation range, but are highest at the lowest elevations drilled to date.

Figure 14.12: Yaraguá Tonnes and Grade per Vertical Metre



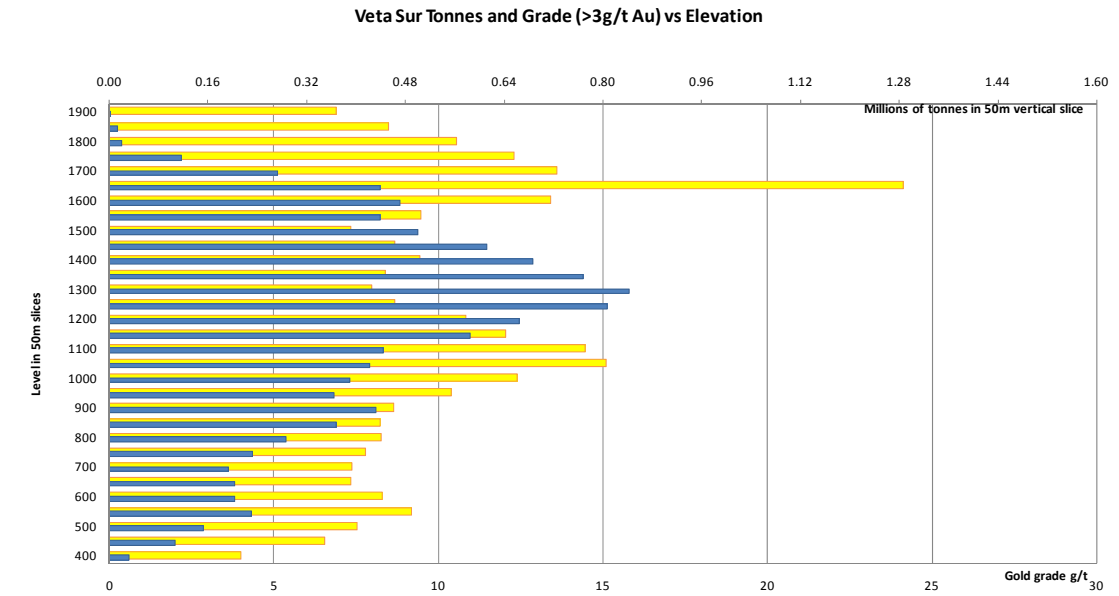
	400	450	500	550	600	650	700	750	800	850	900	950	1000	1050	1100	1150	1200	1250	1300	1350	1400	1450	1500	1550	1600	1650	1700	1750	1800	1850
Gold (g/t)	20.2	13.7	3.9	6.66	8.66	9.97	8.97	10.0	11.0	11.4	10.9	7.44	7.82	7.96	7.77	7.59	7.95	8.83	9.31	9.57	11.2	11.5	11.4	11.0	9.7	8.54	7.57	6.35	3.24	3.21
Au koz	10	7	6	19	51	94	98	106	124	111	75	61	106	160	234	275	326	420	494	440	419	406	402	376	251	110	30	9	1	1
Mt/50 Vert m	0.02	0.02	0.05	0.09	0.18	0.29	0.34	0.33	0.35	0.30	0.21	0.25	0.42	0.62	0.94	1.13	1.27	1.51	1.65	1.43	1.16	1.09	1.09	1.06	0.81	0.40	0.12	0.04	0.01	0.01

*Note: Mt in Figures and Tables is millions of metric tonnes
See "Notes to Accompany Resource Statement" at subsection 14.15.1.

Source: MA, 2015

Chart of tonnage and gold grade (calculated at 50 vertical metre intervals) versus elevation (1,500 m range) for the bulk of Veta Sur resource (above 3 g/t gold) is presented in Figure 14.13. Highest grade resources occur between elevations of 1,600 to 1,800 m. Otherwise, the Veta Sur system demonstrates consistent gold grades over more than 1,000 m of vertical extent. Tonnages for level slices decrease at higher elevations, due to the intersection of certain veins with the topographic surface and also limited by drilling at shallow depths. Lower tonnages at lower elevations are a reflection of limited deep drilling conducted to date. However, gold grades continue to be high (averaging 9 g/t gold) in these areas and suggest further potential at depth in Veta Sur.

Figure 14.13: Veta Sur Tonnes and Grade per Vertical Metre



	400	450	500	550	600	650	700	750	800	850	900	950	1000	1050	1100	1150	1200	1250	1300	1350	1400	1450	1500	1550	1600	1650	1700	1750	1800	1850	1900
Gold (g/t)	4	6.56	7.54	9.18	8.31	7.33	7.37	7.78	8.26	8.23	8.66	10.4	12.4	15.1	14.4	12.0	10.8	8.67	7.99	8.38	9.43	8.67	7.34	9.48	13.4	24.1	13.6	12.3	10.5	8.5	6.91
Au koz	4	23	37	68	54	48	46	58	76	97	120	122	155	205	206	226	232	225	216	207	208	171	118	134	203	341	119	46	7	4	0
M t/50 Vert m	0.03	0.11	0.15	0.23	0.20	0.20	0.19	0.23	0.29	0.37	0.43	0.36	0.39	0.42	0.44	0.58	0.67	0.81	0.84	0.77	0.69	0.61	0.50	0.44	0.47	0.44	0.27	0.12	0.02	0.01	0.00

*Note: Mt in Figures and Tables is millions of metric tonnes

See “Notes to Accompany Resource Statement” at subsection 14.15.1.

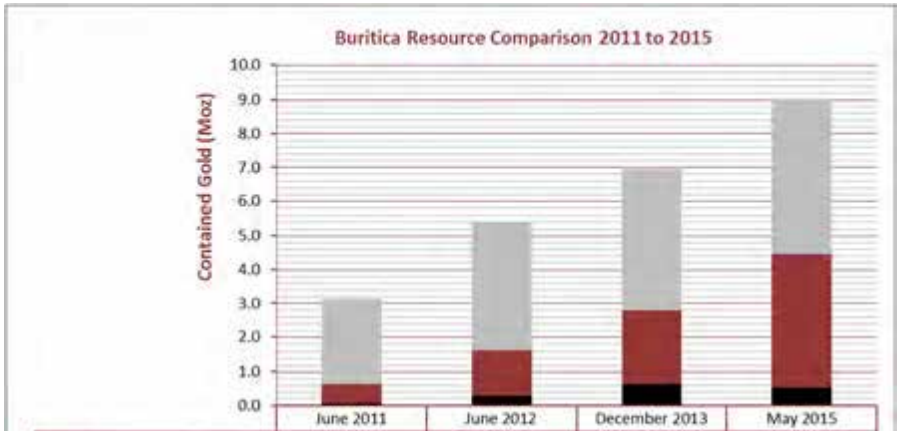
Source: MA, 2015

The majority of both Yaraguá and Veta Sur resource estimates in the tonnage-grade versus elevation tables are at RLs above the current tunnel development.

14.14 Comparison with Previous Estimates

Figure 14.14 shows the substantial increase in Buriticá gold resources from 2011 (the first mineral resource estimate) to the current estimate. This increase in total resources is coupled with significant increases in Measured and Indicated (essentially Indicated) resources. Despite the successful conversion of large tonnages of previously Inferred gold resources to the Indicated category, precious metal grades in the latter category have increased in the 2015 resource estimate relative to the 2014 Estimate. Inferred gold resources in this updated estimate also exhibit increased grade over the two previous estimates. The resource herein is the same as in Independent Technical Report and Resource Estimate on the Buriticá Gold Deposit 2015 effective May 11, 2015.

Figure 14.14: Buriticá Gold Resource Comparison 2011 to 2015



	June 2011	June 2012	December 2013	May 2015
Inferred Contained Au (Moz)	2.5	3.8	4.2	4.5
Indicated Contained Au (Moz)	0.6	1.4	2.1	3.94
Measured Contained Au (Moz)	0.1	0.3	0.6	0.54
Measured Grade Au (g/t)	55.1	27.8	20.4	19.0
Indicated Grade Au (g/t)	16.4	12.3	9.0	10.2
Inferred Grade Au (g/t)	11.4	8.8	7.8	9.0
Measured Tonnage (Mt)	0.04	0.32	0.99	0.89
Indicated Tonnage (Mt)	1.07	3.42	7.41	12.00
Inferred Tonnage (Mt)	6.9	13.3	16.7	15.6

Mineral resource estimates in have been rounded to the nearest 100,000 t. MA has more confidence in grades than tonnage as grade was used as a selection criteria (or cut-off).

Source: MA, 2015

14.15 Mineral Resource Estimate Statement

The updated mineral resource estimate for the Buriticá Project has been prepared in accordance with CIM guidelines, and under the guidance of NI 43-101 Disclosure Standards for Reporting Mineral Projects. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

Mineral Resources are reported separately for Yaraguá and Veta Sur and combined. They are capped, Kriged estimates at 3 g/t Au cut-off grades and diluted to one metre minimum horizontal vein thicknesses. No halo dilution from beyond the 1 m minimum width is included in these Mineral Resources. Additional resources below 3 g/t Au cut-off and above 3 g/t AuEq cut-off are also reported for one metre minimum horizontal vein thicknesses. AuEq is calculated as Au + Ag/60. Gold cut-off grades reflect conceptual costs for underground development, mining and treatment.

Table 14.16: Estimated Resources, May 11, 2015

Yaraguá Mineral Resources above 3 g/t gold cut-off grade, May 11, 2015									
Category	**M tonnes**	**Au g/t**	**Ag g/t**	**AuEq g/t**	**Zn %**	**Au Moz**	**Ag Moz**	**AuEq Moz**	**Zn Mlb**
Measured	0.59	18.9	45	19.6	0.90%	0.36	0.84	0.37	11.4
Indicated	7.82	9.2	30	9.7	0.50%	2.3	7.44	2.43	91.4
M and I	8.41	9.8	31	10.3	0.60%	2.66	8.28	2.8	102.8
Inferred	8.8	9.1	30	9.6	0.40%	2.6	8.4	2.7	70
Additional Yaraguá Resources below 3 g/t Au cut-off and above 3 g/t AuEq* cut-off, May 11, 2015									
Category	**M tonnes**	**Au g/t**	**Ag g/t**	**AuEq g/t**	**Zn %**	**Au Moz**	**Ag Moz**	**AuEq Moz**	**Zn Mlb**
Measured	0.01	2.6	49	3.4	0	0	0.02	0	0
Indicated	0.15	2.7	33	3.3	0	0.01	0.16	0.02	0
M and I	0.16	2.7	34	3.3	0	0.01	0.17	0.02	0
Inferred	0.3	2.7	32	3.2	0	0	0.3	0	0
Veta Sur Mineral Resources classified above 3 g/t gold cut-off grade, May 11, 2015									
Category	**M tonnes**	**Au g/t**	**Ag g/t**	**AuEq g/t**	**Zn %**	**Au Moz**	**Ag Moz**	**AuEq Moz**	**Zn Mlb**
Measured	0.3	19.2	76	20.5	0.30%	0.19	0.7	0.2	2
Indicated	4.17	12.2	37	12.8	0.20%	1.63	5	1.71	21.2
M and I	4.48	12.6	40	13.3	0.20%	1.82	5.7	1.91	23.2
Inferred	6.8	8.9	29	9.4	0.10%	2	6.3	2.1	21
Additional Veta Sur resources classified below a 3g/t Au cut-off and above 3 g/t AuEq* cut-off, May 11, 2015									
Category	**M tonnes**	**Au g/t**	**Ag g/t**	**AuEq g/t**	**Zn %**	**Au Moz**	**Ag Moz**	**AuEq Moz**	**Zn Mlb**
Measured	0.02	2.4	56	3.4	0	0	0	0	0
Indicated	0.13	2.3	75	3.5	0	0.01	0.3	0.02	0
M and I	0.15	2.3	73	3.5	0	0.01	0.4	0.02	0
Inferred	0.9	1.7	118	3.6	0	0	3.3	0.1	0
Combined Yaraguá and Veta Sur Mineral Resources above 3 g/t Au cut-off, May 11, 2015									
Category	**M tonnes**	**Au g/t**	**Ag g/t**	**AuEq g/t**	**Zn %**	**Au Moz**	**Ag Moz**	**AuEq Moz**	**Zn Mlb**
Measured	0.89	19	55	19.9	0.70%	0.54	1.58	0.57	13.4
Indicated	12	10.2	32	10.7	0.40%	3.94	12.4	4.14	112.6
M and I	12.89	10.8	34	11.4	0.40%	4.48	13.98	4.71	126
Inferred	15.6	9	29	9.5	0.30%	4.5	14.7	4.8	91
Combined Yaraguá and Veta Sur Mineral Resources below 3 g/t Au cut-off and above 3 g/t AuEq cut-off, May 11, 2015									
Category	**M tonnes**	**Au g/t**	**Ag g/t**	**AuEq g/t**	**Zn %**	**Au Moz**	**Ag Moz**	**AuEq Moz**	**Zn Mlb**
Measured	0.03	2.5	53	3.4	0	0	0.06	0	0
Indicated	0.28	2.5	53	3.4	0	0.02	0.48	0.03	0
M and I	0.31	2.5	53	3.4	0	0.03	0.54	0.03	0
Inferred	1.2	1.9	96	3.5	0	0.07	3.6	0.1	0

Note – Reported tonnage and grade figures have been rounded from raw estimates to reflect the order of accuracy of the estimate. Minor variations may occur during the addition of rounded numbers. There have been no assumptions made as to metal prices or recoveries in this mineral resource estimate other than gold equivalents that are calculated for AuEq = Au + Ag/60. M in Figures and Tables is millions

Source: MA, 2015

MA is not aware of any environmental, permitting, legal, title, taxation, socio-economic, marketing, political or other relevant issues that may materially affect the estimates of Mineral Resources. The site is currently operational and no impediments of this nature were evident during MA's site visit or raised during discussion with management. Subsequent to the completion of the mineral resource estimate for the Buriticá Project, CGI announced the formalization of small-scale, near surface mining in the region of the resource envelopes. These mining activities are to be restricted to within a 50 m depth range from surface and may have a small but currently non-quantifiable effect on-situ Mineral Resources. MA would note that all areas of known small-scale mining at the Yaraguá have been surveyed and have been excluded from the current mineral resource estimates.

14.15.1 Notes to Accompany Resource Statement

- The Buriticá Project is owned by Continental Gold Inc. (TSX: CGI; OTCQX: CGOOF);
- This mineral resource estimate is based on 278,218 m of drilling and channel sampling from 4,820 holes and 267,818 m of assays;
- Mined out portions were removed from the final model;
- MA has reviewed the company procedures and visited the site on one occasion during the course of the 2014 drill program;
- The resource model is constrained by domains consisting of 3D wireframe models. The drill hole data was displayed in section and elevation slices showing assays and geology. Intercepts were selected and coded for each vein domain based on the following selection criteria, in decreasing hierarchy:
    - Gold grade greater than 2 g/t Au and/or 100 g/t Ag;
    - Minimum mining width of 1.0 m;
    - Sub-grade areas where the interpreted vein domain passed through the drill hole but was not already coded (i.e. "brought through");
- Sub-domains were created to separate >2 gram metre and <2 gram metre volumes inside some veins. These volumes and associated informing composites were treated independently;
- Drill intercepts within each lode were flagged in a database table and composited for each assay element separately to give informing sample downhole composites, one for each vein intercept;
- MA applied top caps to the composites for each vein. Grade caps were selected to restrict the influence of outliers, and varied by vein;
- A minimum of one sample and maximum of eight samples were used for each block;
- The estimation block size for the geological model was 8 m in X and 8 m in Z with width and grade estimated in unfolded 2D space as a variables. Grade and thickness were interpolated by domain using OK estimation with parameters based on directional variography by domain. Estimates were validated against informing samples and with Nearest Neighbour, Inverse Distance Squared, and Kriged uncapped estimates. Thickness of the vein was also estimated by OK estimation;
- Where blocks were within 25 m of a capped value, a hybrid estimate was used – being a combination of the capped and uncapped estimate weighted by a binary indicator;

- Every 8 m * 8 m (X * Z) block with a vein width (in the north direction) less than 1.0 m is set to a width of 1.0 m. Blocks with a width greater than 1.0 m had no change;
- The volume for each vein was defined by a wireframe in 3D space and is used to constrain the resource blocks;
- Results were re-folded and stored in a 3D block model with sub-block size of 1 m (E) x 0.3 m (N) by 1 m (RL) used against all wireframes for volumes. The model was screened for topography and mined out areas by block;
- Lower cut-off grade of 3 g/t Au was used for the reporting of the mineral resource estimate. In MA's view such cut-off grades are likely to be relevant to the potential development of the Yaraguá and Veta Sur deposits by underground mining methods; and
- Reported tonnage and grade figures have been rounded off to the appropriate number of significant figures to reflect the order of accuracy of the estimate. Minor variations may occur during the addition of rounded numbers.

15 Mineral Reserve Estimates

15.1 Introduction

The mineral reserve estimate documented in this section was estimated based on Canadian Institute of Mining (CIM) guidelines that define Mineral Reserves as "the economically mineable part of a Measured or Indicated mineral resource demonstrated by at least a Preliminary Feasibility Study. This Study must include adequate information on mining, processing, metallurgical, economic and other relevant factors that demonstrate, at the time of reporting, that economic extraction can be justified. A Mineral Reserve includes diluting materials and allowances for losses that may occur when the material is mined."

Furthermore, CIM guidelines stipulate that "Mineral Reserves are those parts of Mineral Resources which, after the application of all mining factors, result in an estimated tonnage and grade which, in the opinion of the Qualified Person(s) making the estimates, is the basis of an economically viable Project after taking account of all relevant processing, metallurgical, economic, marketing, legal, environmental, socio-economic and government factors. Mineral Reserves are inclusive of diluting material that will be mined in conjunction with the mineralized material and delivered to the treatment plant or equivalent facility. The term 'Mineral Reserve' need not necessarily signify that extraction facilities are in place or operative or that all government approvals have been received. It does signify that there are reasonable expectations of such approvals."

For the Buriticá Project, Mineral Resources were converted to Mineral Reserves using estimated cut-off grades (COGs) by mining method for each deposit based on estimated gold price, mining dilution and losses, mining costs, processing costs & recoveries, general and administration costs, transport & refining costs, and royalties. In-situ mining shapes were selected to prepare a diluted stope inventory, and then an iterative process was used to prepare a mine design and production schedule in order to estimate mining costs subsequently used to re-estimate COGs. Sub-economic diluted mining shapes were then excluded, and iterations were performed until all material met COG criteria. During the process, these diluted mining shapes were reviewed to verify there was sufficient margin to pay for stope access development costs. The final production plan iteration established the Mineral Reserves estimate.

Although both deposits outcrop at surface or sub-outcrop close to surface, only underground mining methods were considered for Buriticá. Choice of mining method was driven principally by geomechanical criteria as well as vein geometry, orientation, and in some cases degree of isolation from the main Yaraguá and Veta Sur stoping areas.

A potential unknown identified by the QP is the extent of unauthorized and illegal mining activities in the Yaraguá and Veta Sur areas. CGI has entered into formalized sub-contracts with several small-scale mining associations. These sub-contracts established mining limits for the associations to work within, but unauthorized mining outside of those limits has occurred in some areas. In addition, illegal mining activities have been discovered, on occasion, adjacent to Continental's underground workings in the Yaraguá and Veta Sur deposits. In late July 2015, CGI surveyed the extent of unauthorized mine workings in areas that were accessible. These areas are excluded from the Mineral Reserves; however, the extent of impact in the upper areas of the reserve may not be fully quantified.

The QP has not identified any risk including legal, political, or environmental that would materially affect potential Mineral Reserves development except for the following: 1) Continued unauthorized mining activities; and, 2) successfully securing from the Columbian government the required permits for Project development and operation. The QP is not aware of unique characteristics related to this Project that would prevent the granting of such permits.

Gregory A. Blaylock, P.Eng. is the Qualified Person who prepared this disclosure and who participated in and supervised all aspects of converting Mineral Resources to Mineral Reserves for the Project. He is independent of the issuer in accordance with National Instrument 43-101 Standards of Disclosure for Mineral Projects.

The Mineral Reserve estimate for the Project is shown in Table 15.1. The Mineral Reserve estimate is based on Measured and Indicated Resources only and was determined by applying cost estimates, a gold price of $950 US$/oz and an exchange rate of 2,850 Colombian Pesos to one US$. The effective date for the mineral reserve estimate contained in this report is July 31, 2015.

Mineral Reserves are included in Total Mineral Resources (Section 14) except for the dilution component of Mineral Reserves that was not reported in Mineral Resources. Mineral Resources include vein domains only, while Mineral Reserves include metal contained in dilution tonnes comprised of “halo” resource blocks classified as an indicated resource outside of the vein domains. See sections 14.1 and 14.1.1 for an explanation of how the “halo” model was prepared and used for the mineral resource estimate. The “halo” model provides: 1) A way to estimate Mineral Resources using a 1.0 m minimum mining width, and 2) Dilution tonnes and grades for Mineral Reserves.

Table 15.1: Summary of Mineral Reserve Estimate

Deposit and Classification	Tonnes (kt)	Diluted Grade	
		Au (g/t)	Ag (g/t)
Proven Reserves			
Yaraguá	450.6	20.49	47.6
Veta Sur	226.8	22.19	84.66
Total Proven	677.4	21.06	60.01
Probable Reserves			
Yaraguá	8,378.80	7.01	20.82
Veta Sur	4,660.60	9.09	25.38
Total Probable	**13,039.40**	**7.76**	**22.45**
Total Proven and Probable	**13,716.80**	**8.41**	**24.31**

Source: JDS, 2016

Contained metal in the Table 15.1 totals 3,710,000 oz gold and 10,719,000 oz silver, including dilution material and mining losses. Slight differences in contained metal ounces may result from using tonnes and grades rounded to significant figures.

15.2 Basis of Mineral Reserve Estimate

All vein interpretations from the resource block model were reviewed and verified using Surpac™ software to ensure that only Measured and Indicated Resources were included and to verify the reliability of the classifications. A total of 72 veins were identified to include in the Mineral Reserve estimates.

Vulcan™ software was used to calculate Mineral Reserve estimates. The resource block model was imported into Vulcan™ and all resource blocks with gold values of 3.0 g/t or greater for Yaraguá and Veta Sur were delineated. In areas with formalization sub-contracts, the legal boundary descriptions were modeled as 3D solids including a 5 m boundary pillar to exclude from the Mineral Reserves. Also, any unauthorized mining activity identified from Continental's July 31, 2015 survey was excluded, and in the adjacent areas, criteria included cut and fill (C&F) extraction using 80% recovery. The remaining resource block was then initially divided into 15 m high sublevels. Geomechanical evaluation supported this level spacing as well as a 30 m strike length for typical stopes. These dimensions were used to establish an initial pool of in-situ undiluted mining shapes. In some areas, these shapes included material grading less than 3.0 g/t Au and some inferred resource blocks. All inferred material included in tonnage calculations was assigned a 0 g/t Au value.

Planned external dilution based on mining and geomechanical criteria was then added to the in-situ mining shapes using Vulcan™ software. COGs developed for Yaraguá and Veta Sur were then applied to the diluted shapes to establish the preliminary economic stope inventory used as a basis for mine design and Mineral Reserve estimation.

15.3 Mining Methods and Mining Costs

Three underground mining methods will be used at Buriticá: Longhole open stoping (LHOS), C&F, and shrinkage stoping. Mining method selected was driven primarily by geomechanical criteria. Longhole mining was the preferred mining method due to higher productivities and lower mining costs compared to either C&F or shrinkage. Preliminary mining cost estimates were used to calculate an initial COG to establish the potentially economic stope inventory, which was then used to estimate mining costs on a first principles basis for all three mining methods from the resulting mine design and production schedule. Using these mining costs, the COG was again calculated, applied to the stope inventory, and the mine design and production schedule were revised accordingly. Final direct and indirect costs, and COGs were estimated after several iterations, and final checks ensured that no uneconomic stopes were included in the final mine plan and Mineral Reserves.

15.4 Cut-off Grades

Due to differences in metallurgical recovery of gold, COGs were estimated separately for the Yaraguá and Veta Sur deposits. Final cut-off grades are based on an exchange rate of 2,850 COP:US$, a $950/oz gold price, and feasibility level cost and recovery estimates as shown in Table 15.2.

Table 15.2: Cut-Off Grade Calculation Basis

Item	Unit	Value
Revenue & Cost of Sales		
Gold Price	US$/troy ounce	950
Payable Metal	%	99.93
Assay, Transport, Refining & Insurance Costs	US$/troy ounce	1.09
Royalties, Effective % basis of Mine Mouth Gross Metal Value	%	3.2
Production Unit Costs		
Mining - Longhole	US$/ore tonne mined	54.51
Mining - Cut & Fill	US$/ ore tonne mined	70.11
Mining - Shrinkage	US$/ ore tonne mined	57.12
Mining – Development Ore	US$/ore tonne mined	54.85
Processing	US$/tonne milled	26.16
General & Administration	US$/tonne milled	14.13
Management Fee	US$/tonne milled	3.01
Metallurgical Recoveries		
Yaraguá	%	95.4
Veta Sur	%	92.3
Mining Recoveries		
Longhole Mining	%	95
Cut & Fill Mining	%	90
Shrinkage Mining	%	95

Source: JDS, 2016

Silver value was not included in COG calculations due to its relatively small contribution to total value. However, revenue for silver produced is included in the project economic model, and silver is included in reserves.

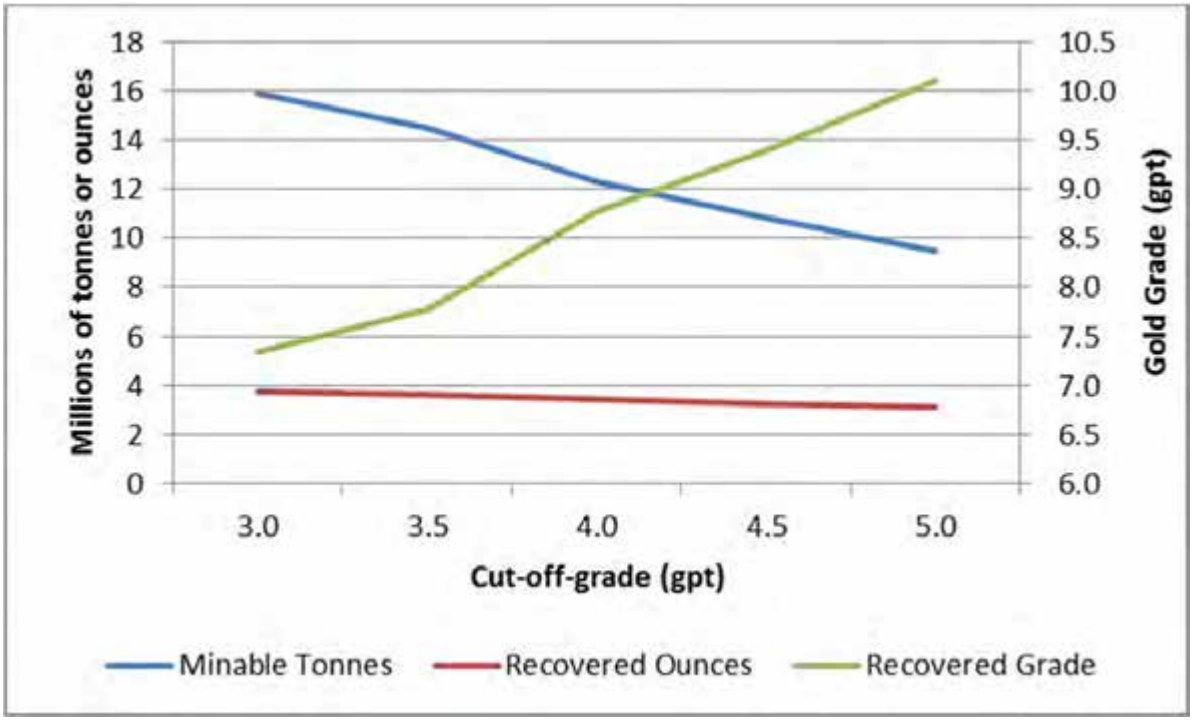
A separate COG was estimated for development, including stope sills and drifting on vein. The COG calculation excludes the development drifting cost, but includes all other production costs. This COG was used to determine if planned development in mineralized material would be included as mill feed in the mine plan or considered waste tonnage.

Using the calculated COGs, a mine plan was completed for average stope COG's of 3.8 g/t Au for Yaraguá and 4.0 g/t Au for Veta Sur. However, during the Feasibility Study, market conditions indicated that a gold price above $950/oz should be used. At $1,200/oz gold, the average break-even COG is approximately 3.2 g/t. Concurrently with the mine design and production schedule iterations and COG determination, trade-off evaluations, performed to assess the gold production profile and return on investment, indicated that using the cut-off grades established for $950/oz gold resulted in several benefits: Less overall development and lower metres developed per produced gold ounce; more uniform metal production profile; and improved return on investment potential. For these reasons, the final selected mine design and production plan was based on COGs of 3.8 g/t for Yaraguá and 4.0 g/t for Veta Sur. As a consequence, use of the higher COGs provides a mine plan potentially viable to $950/oz gold price.

Other observations supporting these findings and decision to use the selected COGs:

- Resource characteristics;
- Figure 15.1 shows that increasing the COG from 3 g/t to 4g/t results in a small decrease to total recovered gold ounces compared to a significant drop in mineable tonnes and increase to recovered gold grade. These relationships indicate minimal loss to total recovered metal by not mining marginal material in the 3 to 4 g/t grade range;
- It should be noted that the material between 3 g/t and 4 g/t is generally located in separate discrete stopes which would bear the full cost of development and production, rather than material that could be incrementally extracted from the wall rock in a minable stope. For this reason, the material is not incremental to the diluted minable stope shape, and an analysis including development and stope operating costs would be required to determine marginal contribution and benefits to mine economics. Lower margin material not included in the production plan, but located along strike and towards the vein fringes is not sterilized, and if economic, could be recovered later in the mine life.

Figure 15.1: COG Resource Grade – Tonnage Curves



Inferred resources not included
Source: JDS, 2016

- Production and economic throughput considerations;
- Feasibility Study evaluations established that 3,000 t/d mining rate is a practical limit which achieves operational balance for development and stope production. Given this production constraint, the opportunity existed to improve project financial metrics using a mining schedule with higher-margin ore, and economic assessments verified that such a mine plan improved IRR and NPV. Also production plan provides increased upfront revenue reducing payback time, and as a consequence, project capital risk is reduced;
- It should be noted that high level economic assessments indicated that mine plans using planning cut-offs of 4.5 g/t and 5.0 g/t could potentially produce equal or improved IRR and NPV results. Separate mine designs and production plans would be required in order to fully evaluate these alternatives.

15.5 Dilution

15.5.1 Internal Stope Dilution

Veins in the resource model with less than 0.8 m true width were diluted to a minimum 1.0 m mining width using “halo” estimated grades as described in Sections 14.1 and 14.1.1 of this report. This internal dilution already included in the resource model vein domains is included in the mineral reserve estimate.

15.5.1.1 Discrete Vein Stopes

There are 31 veins in Veta Sur and 41 veins in Yaraguá included in the mine plan and Mineral Reserves. The majority of mining shapes are stopes comprised of single, discrete veins. For veins with an undiluted true width of less than 80 cm, dilution is limited to material added to bring the vein to the minimum mining width of 1.0 m.

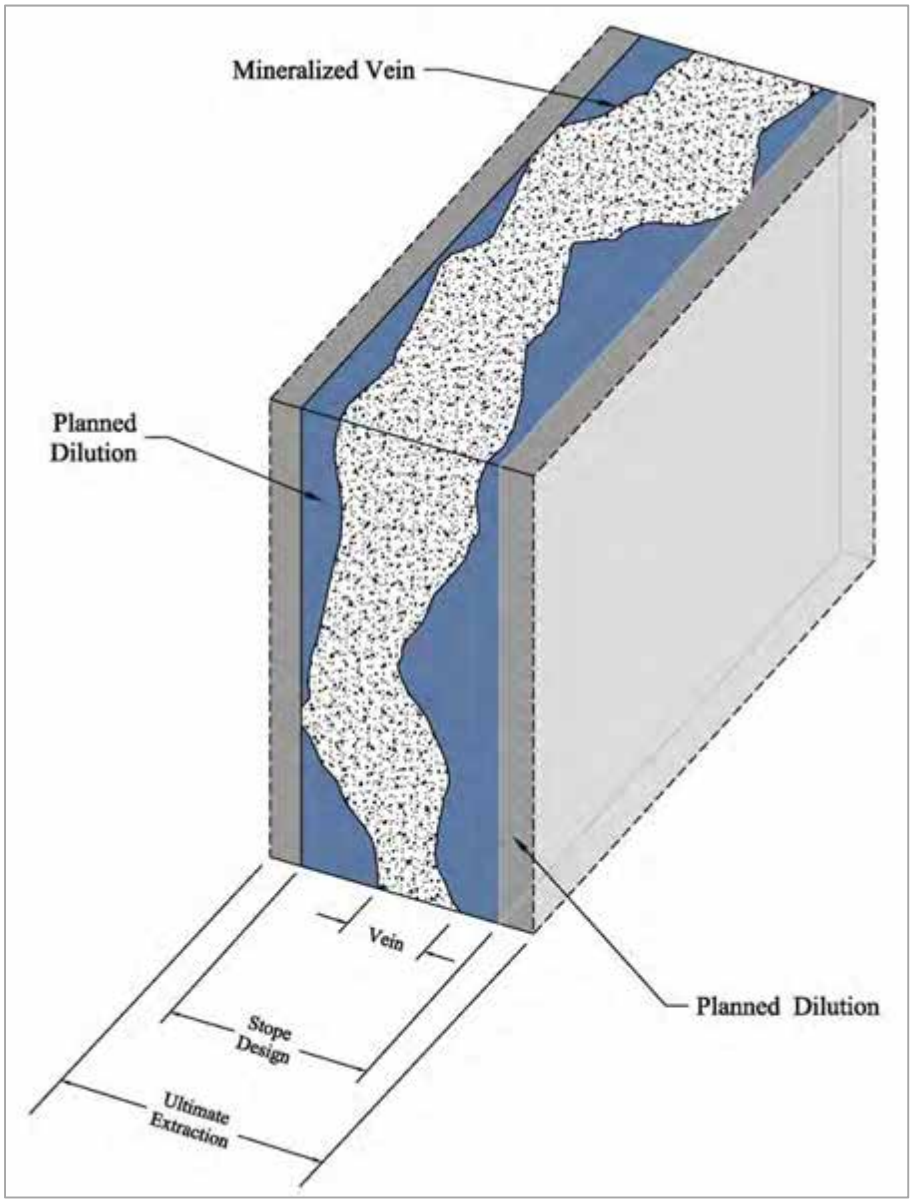
Other potential sources of internal dilution are resource blocks within the stope shapes with Au grades less than 3.0 g/t, or inferred category material which is assigned a value of zero. The uneconomic material or inferred blocks included within the mining shapes is a result of realistic stope design geometries for each mining method. Figure 15.2 shows an example of designed internal dilution in a typical planned discrete vein stope.

15.5.1.2 Combined Vein Stopes

In numerous situations, vein proximity provided the opportunity for improved economic extraction by combining two or more veins in a stope. Some veins were close enough to combine them into economic mineable shapes that would include the uneconomic or inferred material between and within veins. Identifying the combined vein stopes was largely a visual, manual exercise based on cross sections showing vein geometry and block grades.

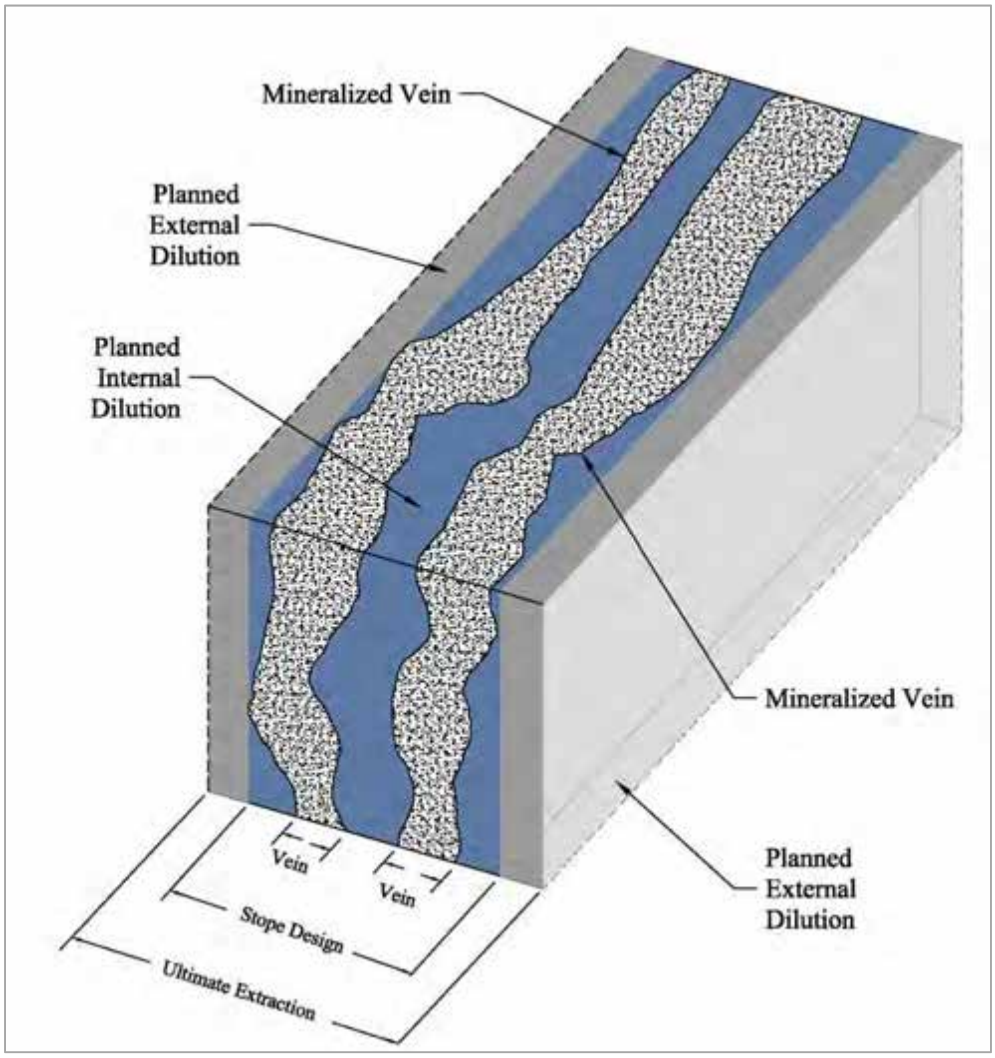
The metal content of the uneconomic material between veins that was classified as measured or indicated is included in the Mineral Reserves, and any inferred blocks that may have been included between veins were assigned a gold grade value of zero. Figure 15.3 shows an example of two veins combined into a larger stope block with the internal dilution between veins.

Figure 15.2: Discrete Stope Dilution



Source: JDS, 2016

Figure 15.3: Combined Stope Dilution



Source: JDS, 2016

15.5.2 External Stope Dilution

Planned external dilution was included by increasing the designed stope width using estimated overbreak based on in-situ stope width and proximity to other stopes. For both the discrete and the combined vein stopes, the dilution skin overbreak from Table 15.3 was used to estimate the ultimate extraction widths.

Table 15.3: External Dilution Parameters

Dilution Basis	Dilution "skin" (cm)
Vein true width < 80 cm, discrete vein mining, dilution grades from halo model	0
Vein true width > 80 cm < 2.0 m, discrete vein mining, dilution grades from halo model	20
Vein true width >2.0 m < 6.0 m, discrete vein mining, dilution grades from halo model	30
Vein true width > 6.0 m, discrete vein mining, dilution grades from halo model	50
Combined vein mining, dilution grades from halo model	50

Source: JDS, 2016

Figures 15.2 and 15.3 show how planned external dilution is defined and accounted for in all stope designs by increasing stope widths to model overbreak for the fully diluted mining shapes.

After all stope designs were complete, the resultant diluted stope shape solids were validated and used to interrogate the resource block models. Results of these interrogations provide diluted tonnes and metal grades for each mining shape using only measured or indicated resource blocks from the vein and halo models.

15.5.3 Backfill Dilution

15.5.3.1 Longhole Stopes

Dilution in longhole stopes from paste backfill is estimated to average 2.9%. Dilution from paste backfill is applied to all longhole stopes in addition to internal and external dilution estimates. No metal values are assumed for the paste backfill.

Dilution from paste backfill in longhole stopes will come from the floor when working on top of paste backfill and from ends where mining along strike, adjacent to a previously filled stope. The intent of working on top of cemented paste fill is to provide a durable visible marker horizon to maximize recovery. Floor dilution from paste backfill will be minimal as the paste will contain cement and be designed for a minimum 7-day strength of 350 kPa. Where stopes are two or three sublevels high there will be no floor dilution between sublevels since the stacked stopes are drawn down from the bottom sublevel. Where draw points are not utilized, as with many 15 m high discrete vein stopes, remote longitudinal mucking on vein is estimated to average 10 cm of floor dilution.

The amount of paste backfill dilution from adjacent stope ends is a function of stope width and height as summarized in Table 15.4.

Table 15.4: Paste Backfill Dilution Parameters

Stope Width	Stacked Stope Height (m)	Paste Strength Required (UCS @ 7 days, kPa)	Paste Backfill Dilution (cm)
Less than 2 m	15	350	50
	30	350	100
	45	350	100
2 – 6 m	15	700	50
	30	700	100
	45	700	100
6 – 15 m	15	1000	50
	30	1400	100
	45	1400	100

Source: JDS, 2016

15.5.3.2 Cut & Fill Stopes

The high-grade nature of the deposit and expected disproportionate percentage of value in fines support estimating dilution due to mining loss. A 10% mining loss (see Section 15.6) is used to estimate gold losses in the backfill as well as the lower grades due to over-mucking the bottom. Additional dilution from waste rock backfill used in C&F stopes is not included since losses are accounted for in the 90% mining recovery.

15.5.3.3 Shrinkage Stopes

An estimated 10% dilution from backfill is estimated to come from shrinkage stopes where waste rock has either been top-loaded into the stope during drawdown or additional waste rock comes in from backfilled adjacent stopes. No metal values are assumed in dilution from waste rock.

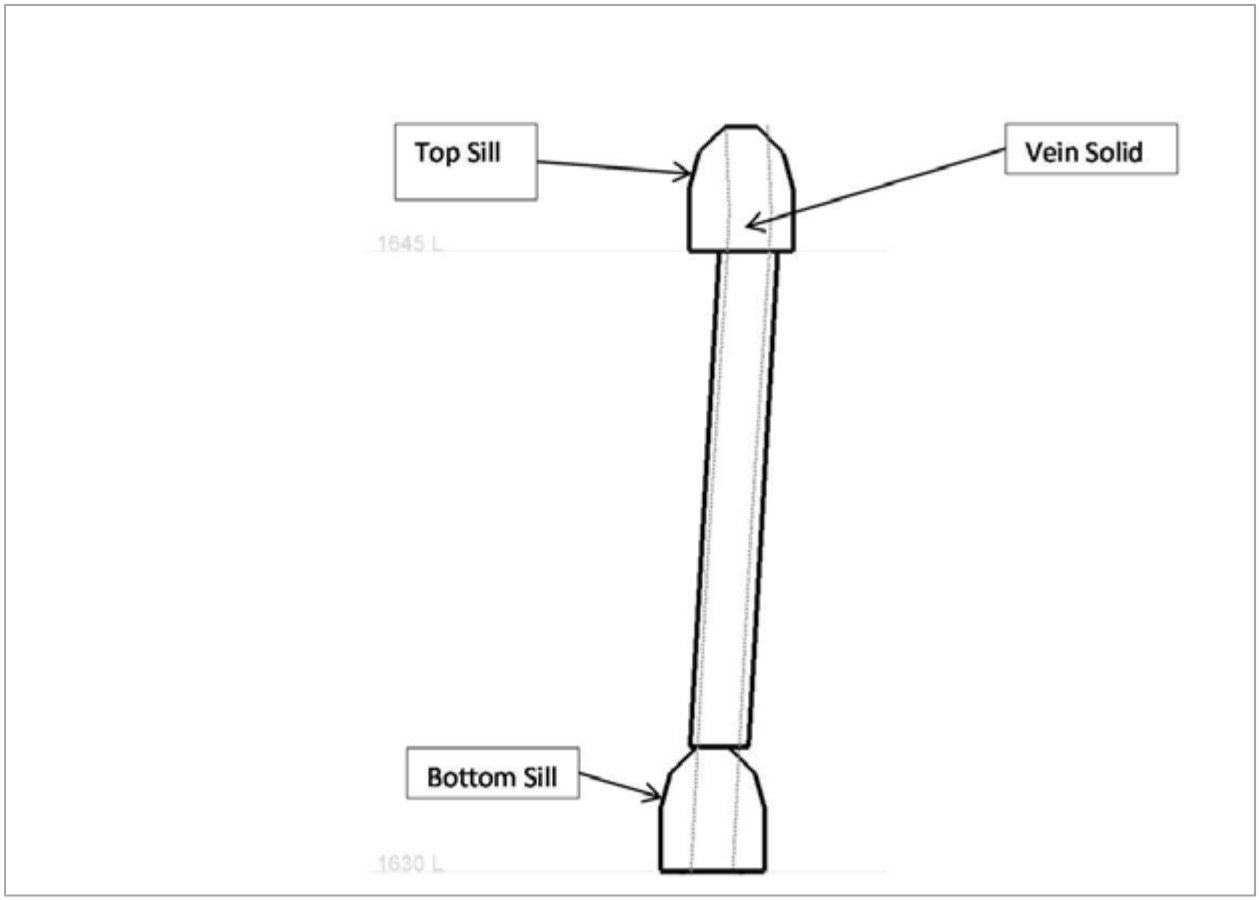
15.5.4 Development Ore Dilution

Stope sills developed to prepare stopes for mining are designed to a width of 2.5 m and incur varying degrees of dilution depending on vein width. The 2.5 m width is considered the minimum development heading width. For stopes containing veins wider than 2.5 m the sill development heading widths will vary to accommodate full vein exposure, and no external dilution is added to these development ore tonnes.

Stope access development that occurs on vein will have larger mining dimensions of either 3.5 m x 3.5 m or 4.5 m x 4.5 m. The objective is to minimize development in waste while generating development ore tonnes that pass the development ore COG.

To illustrate for typical narrow vein extraction, Figure 15.4 shows greater dilution the in top and bottom sills than in the longhole void.

Figure 15.4: Ore Development Dilution Example



Source: JDS, 2016

15.6 Mining Recovery

Mining recoveries were estimated for the three mining methods planned for Buriticá; longhole stoping, cut & fill mining and shrinkage stoping. The following table summarizes mining recoveries by mining method.

Table 15.5: Summarized Mining Method Recoveries

Mining Method	Mining Recovery (%)
Longhole	95
Cut & Fill	90
Shrinkage	95
Development Ore	95

Source: JDS, 2016

Mining recoveries were applied to each individual stope or development shape depending on mining method. In areas where mining has already occurred, the mining method selected is C&F with a mining recovery of 80% applied to stopes impacted by historical mining.

15.7 Mineral Reserve Estimate

The Buriticá underground Mineral Reserves estimate is shown Table 15.6.

Table 15.6: Buriticá Underground Mineral Reserves

Proven & Probable Reserves			
		Diluted Grade	
Deposit/Mining Method	**Tonnes**	**Au**	**Ag**
	(kt)	**(g/t)**	**(g/t)**
Yaraguá Longhole	4,578.60	8.02	21.67
Yaraguá Cut & Fill	1,665.80	9.33	28.48
Yaraguá Shrinkage	119.4	8.13	54.48
Yaraguá Development Ore	2,465.60	5.99	17.34
Total Yaraguá	8,829.40	7.7	22.19
Veta Sur Longhole	3,373.60	10.03	28.93
Veta Sur Cut & Fill	223.6	16.68	54.22
Veta Sur Shrinkage	95.5	7.61	24.97
Veta Sur Development Ore	1,194.70	7.62	21.23
Total Veta Sur	4,887.40	9.7	28.13
Total Buriticá	13,716.80	8.41	24.31

Source: JDS, 2016

The contained metal in the summary of Mineral Reserves shown in Table 15.5 totals 3,710,000 oz gold and 10,719,000 oz silver. Mineral reserves include dilution material and mining losses. Slight differences in contained metal ounces may result from using tonnes and grades rounded to significant figures in Tables 15.1 and 15.5.

16 Mining Methods

16.1 Introduction

Three underground mining methods were selected for Buriticá; longhole open stoping (LHOS), cut & fill (C&F), and shrinkage stoping. Mining method selection was driven primarily by geomechanical rock quality, vein geometry, depth, proximity to old workings, stope margin and vein continuity. Unless geomechanical and geometry characteristics required either C&F or shrinkage, longhole mining was the preferred mining method due to higher productivities and lower mining costs compared to either C&F or shrinkage.

16.2 Deposit Characteristics

There are two vein systems that will be exploited in the Buriticá mine: Yaraguá and the Veta Sur. The Yaraguá system generally strikes to the east, and has been drill intersected over 1,125 m along strike and 1,540 m vertically. There are several veins within the system that strike to the northwest, intersecting the east-striking veins. Vein strike lengths vary from 50 to 1,100 m and vein dip distances vary from 50 to 1,300 m. The Veta Sur system strikes to the Northeast, and has been drill intersected over 1,140 m along strike and 1,600 m vertically. Vein strike lengths vary from 70 to 1,000 m, and vein dip distances vary from 150 to 1,350 m.

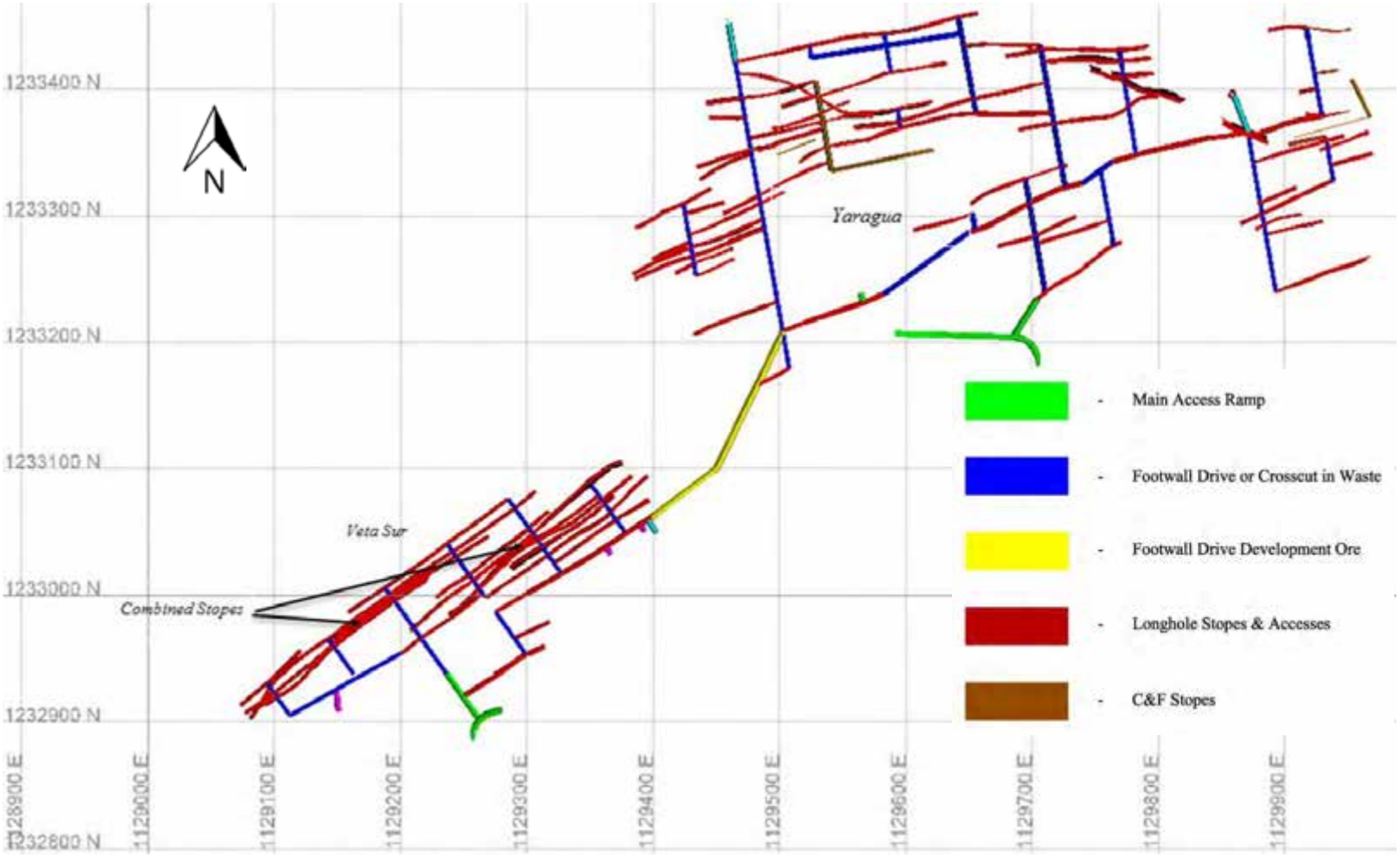
The mine production plan and Mineral Reserves include 72 veins; 41 in Yaraguá and 31 in Veta Sur. Horizontal distances between veins typically vary between 2 to 25 m, and in some cases veins converge or cross. Although most veins occur as discrete structures which can be mined alone, where practical and economic, certain closely spaced veins were combined during stope design. Yaraguá average diluted stope width is 1.9 m, and Veta Sur average diluted stope width is 3.2 m. Overall average diluted stope width is approximately 2.4 m, although combined vein stopes can reach 12 to 15 m width.

Metallurgical testwork determined that gold recovery differs for the two deposits. In general, ore from Veta Sur exhibits on average, 3% lower recovery than that from Yaraguá. Another important factor is Veta Sur on average has higher arsenic concentrations. Due to the metallurgical recovery difference and higher average Veta Sur arsenic content, the mine production plan balanced extraction to limit total mill feed to no greater than 50 percent from Veta Sur.

The mineralization continuity along strike and down dip at both Veta Sur and Yaraguá deposits has been well documented from extensive drilling and underground exploration as well as through CGI's Yaraguá mine operation experience including trial mining initiatives to date. As expected for high-grade precious metal mineralization, gold and silver distributions can be highly variable; however, the veins demonstrate excellent continuity above the COG values along strike and down dip. This important characteristic facilitates mining method selection because the veins can be sub-divided into economic mining shapes that are contiguous for extraction. See Figure 16.1 for an example of stope continuity on multiple veins between 1390 masl and 1405 masl.

Figure 16.1: Plan View Veta Sur and Yaraguá Showing Stope Outlines at Elevation 1390

Source JDS 2015



16.3 Geotechnical Analysis and Recommendations

16.3.1 Geomechanics

16.3.1.1 Geomechanical Characterization

The vein systems and surrounding host rock at Buriticá were characterized geomechanically from drill core; logging for rock mass quality and discontinuity orientation, and laboratory strength testing of core samples. This information was supplemented with mapping of existing underground workings. Two separate characterization campaigns have been completed for the Project. In 2014, a pre-feasibility level characterization program was initially performed by Ingenieria de Rocas Ltda. (Ingeroc). In 2015, a comprehensive supplemental characterization program was then performed by SRK to provide geomechanical characterization at a feasibility level (SRK 2016).

Overall, the two characterization programs consisted of the following components:

- Geomechanical logging of 3,950 m of core from ten diamond drill holes across the deposits;
- Orientation of discontinuities in core from four of the ten drill holes;
- Laboratory core sample strength testing including 81 uniaxial compressive strength (UCS), 14 triaxial compressive strength (TCS), 22 Brazilian tensile strength (BTS) and three saw-cut direct shear strength tests;
- Geomechanical mapping of existing underground workings and documentation of excavation stability; and,
- Rock mass classification of core logging data according to the Barton (1974) Q' and Bieniawski (1989) RMR systems.

16.3.2 Geomechanical Domains and Rock Mass Properties

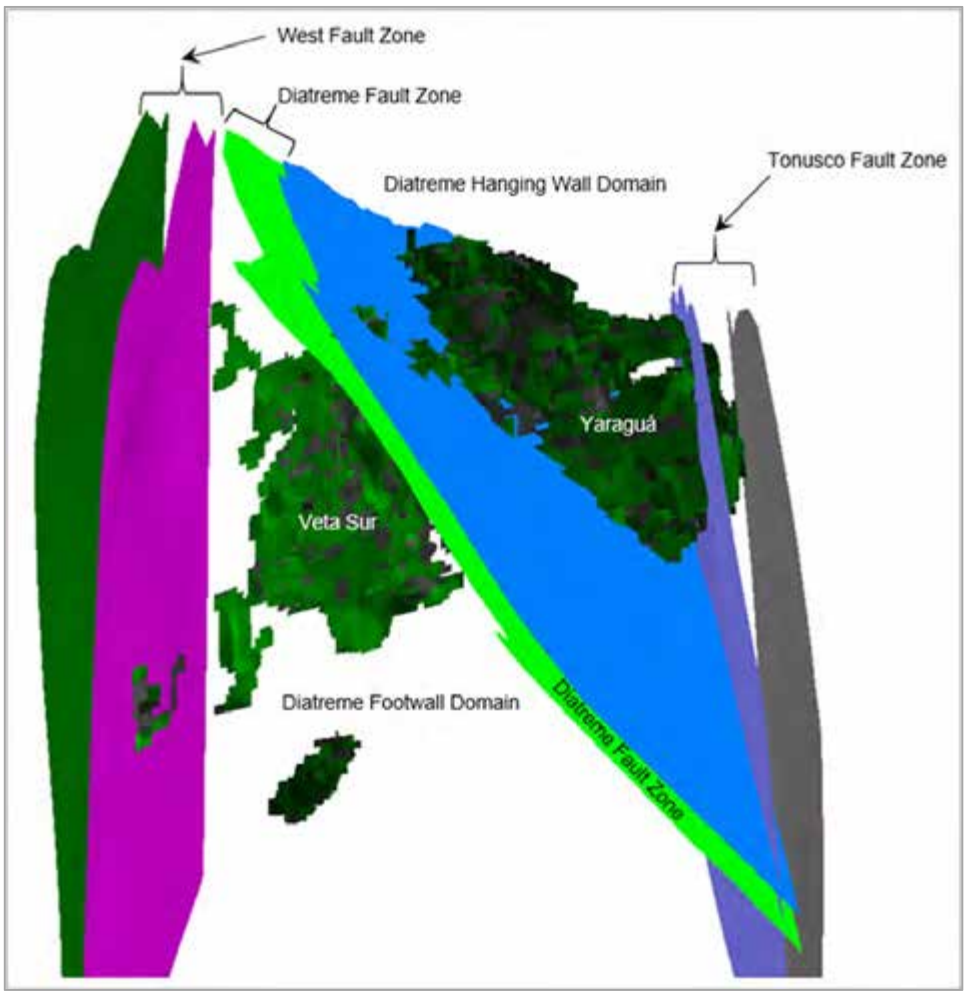
Based on the geologic structural model, surface topography and the geomechanical characterization described above, the deposit was divided into 3D structural-geomechanical domains. The individual domains grouped areas of similar rock mass quality, which were used for developing geomechanical design parameters. Geologic structure was identified as the dominant factor controlling rock quality domains compared to lithology. Five structural-geomechanical domains were delineated (see Figure 16.2).

- **Diatreme Fault Zone Domain**: Delineated by two sub-parallel thrusts approximately 50 m apart containing many thrusts or thrust zones. The rock within this domain is typically poor geomechanical quality, and intensely sheared and damaged. The average rock strength from three valid UCS tests conducted on samples obtained is 50 MPa; however, this value is anticipated to be unrepresentatively high as most of the rock core from this zone was too heavily fractured to obtain laboratory test samples;
- **Diatreme Hanging Wall Domain**: Defined as the area above the Diatreme Fault Zone and generally consists of fair quality rock with frequent Diatreme-parallel structures. The average intact rock strength from 17 valid UCS tests conducted on samples obtained within the domain is 98 MPa;
- **Diatreme Footwall Domain**: Defined as the rock below the Diatreme Fault Zone. The footwall domain is typically good geomechanical rock quality with significantly fewer structures. The

average intact rock strength from 24 valid UCS tests conducted on samples obtained from the domain is 90 MPa;

- **West Fault Zone Domain**: Damaged zone associated with the regional West fault. Rock mass quality is anticipated to be poor, similar to the Diatreme Fault Zone. No laboratory test samples were obtained from the West Fault Zone Domain; however, the UCS strength is anticipated to be similar to that of the Diatreme Fault Zone; and,
- **Tonusco Fault Zone Domain**: Damaged zone associated with the regional Tonusco Fault. Rock mass quality is anticipated to be poor, similar to the Diatreme Fault Zone. No laboratory test samples were obtained from the Tonusco Fault Zone Domain; however, the UCS strength is anticipated to be similar to the Diatreme Fault Zone.

Figure 16.2: 3D view of geomechanical domain boundaries (looking north)



Source SRK 2015

16.3.3 Stope Design Recommendations

The LHOS layout is designed using 15 m sublevel spacings. Where rock quality is sufficiently high to remain stable for stacked longhole stopes, sequential 15 m sublevels will be blasted up to a maximum height of 45 m, which will then be mucked from the lowest of the levels, and drawn down to create sufficient void space for continued longhole blasting. Stope lengths are typically 30 m. For the few 10-15 m wide stopes, lengths will be reduced to 15 m to reduce the hydraulic radius and improve ground stability. Potvin's (2001) empirical stope design method was used to estimate stable stope dimensions for the various ground conditions.

Design of parallel longhole stopes included minimum pillar width criteria based on stope height, rock mass quality, useful life and confinement conditions:

- 45 m height: 4 m pillar;
- 30 m height: 3 m pillar;
- 15 m height: 2 m pillar;

During drawdown, stopes will only be open to full height at the end of extraction. Blasted muck will provide confinement support to stope walls prior to final drawdown. Empty stopes will then be backfilled with cemented paste or waste rock depending on sequencing requirements.

For localized areas where poor ground conditions are anticipated such as within the Diatreme, West and Tonusco Fault Zone Domains, C&F mining will be used. Table 16.1 summarizes mining methods used for each geotechnical domain. Ultimate stope height will be determined from definition drilling results prior to mining. Shrinkage stoping is utilized based more on economic considerations than geomechanical. Shrinkage makes up less than 2% of the total reserves and is utilized in isolated areas of the deposit.

Table 16.1: Mining Methods and Stope Dimensions by Domain

Geomechanical Domain	Cut & Fill	15-m High LHOS	30-m High LHOS	45-m High LHOS
Diatreme Fault Zone	70%	30%	-	-
Tonusco Fault Zone	70%	30%	-	-
Diatreme Hanging Wall Zone	-	15%	60%	25%
Diatreme Footwall Zone	-	10%	20%	70%
West Fault Zone	100%	-	-	-

Source SRK 2016

16.3.4 Stoping Sequence and Sill Pillars

The magnitude and direction of in-situ stresses in the Buriticá Project area have not been directly measured. For this reason, stress conditions were estimated based on regional geologic information and site topographic features. Most of the mining will occur above the Higabra Valley floor in what is anticipated to be a relatively low stress environment with horizontal stresses approximately equal to or less than vertical stresses. However, high stress gradients can exist beneath deep valley floors such as the Higabra due to geometric effects thus horizontal stresses are anticipated to be higher below the valley floor.

To reduce potential stress impacts on mining, stopes have been sequenced to direct stresses away from the central mining areas and toward outer abutments. The mine plan follows an overall geomechanical sequence to mine from north to south. Northernmost veins are scheduled to be mined early as levels are developed in an attempt to divert stresses away from remaining stopes on the level.

Where three or more closely spaced parallel stopes need to be mined discretely, the outer stopes will be taken first leaving the middle stope(s) to create a wide temporary pillar between the outer stopes. After the outer stopes are backfilled, the middle stope(s) will then be mined in 15 m lifts to minimize potential latent effects caused by mining the adjacent stopes.

16.3.5 Stope Backfill Specifications

Cemented backfill will be used to fill stope voids to provide confinement on the waste rock pillars in the hanging wall and footwall between parallel stopes spaced close together, and as the fill develops strength, to reduce lateral loads on these pillars. This is an important aspect when mining closely spaced sub-vertical veins so adjacent veins planned for mining are not sterilized due to potentially unstable waste rock pillars between sub-parallel stopes. The use of cemented backfill also enables secondary stopes along strike to be mined directly adjacent to the primary backfilled stopes without requiring a rib-dip pillar.

Requirements for cemented paste backfill UCS strength were developed analytically using Marston's (1930) two dimensional arch solution to estimate the maximum vertical stress within the stope backfill, taking into consideration the horizontal pressure or confinement provided by the stope hanging wall and footwall. The required backfill strength was then calculated to resist the maximum vertical stress predicted by Marston's equations assigning a factor of safety of approximately 2.

Based on the analyses, the following was concluded:

- Stopes up to 2 m wide - 7-day UCS of 350 kPa should be used. Once cured, the maximum vertical backfill stress will be reached within the first 15 m of backfill height and as such, the required backfill strength is the same for the 15-m, 30-m and 45-m high cases;
- Stopes between 2 and 6 m wide - 7-day UCS of 700 kPa should be used;
- Stopes between 6 and 15 m wide - 7-day UCS of 1 MPa for 15 m high stopes, and 1.4 MPa for the 30-m and 45-m high stopes should be used.

The specifications for backfill UCS requirements are provided for 7-day curing strengths in alignment with the planned approximate two week mining cycle (including mining, backfilling and curing).

16.3.6 Mine Infrastructure and Offset Distances

Mine infrastructure will include two ramp systems (one for Yaraguá and one for Veta Sur), a light maintenance shop area on the 1525 m level, electrical substations, explosives storage facilities, compressor stations, ore and waste rock passes, ventilation raises and small diameter boreholes for backfill and utilities.

The light maintenance shop is located in the Diatreme Hanging Wall domain where good rock quality is anticipated. Pillars between the individual bays within the shop areas are designed with a minimum 2:1 (W:H) ratio for stability purposes.

Ore passes and ventilation raises are designed within the Diatreme Footwall and Hanging Wall domains to the extent possible; however, a small percentage will have to be constructed through the Diatreme Fault Zone. Pilot holes will be drilled and geotechnical investigations conducted in advance of the raises to verify rock quality. If necessary, fault or fracture zones may be pressure grouted from the pilot hole to improve ground conditions for the larger diameter raises.

Primary ramps are located in the good quality domains to the greatest extent possible, and where ramps and other access development headings must pass through the Diatreme, Tonusco, or West Fault Zones, appropriate ground support will be used.

Offset distances were confirmed with 3D numerical stress modeling. Minimum offset distances are shown in Table 16.2.

Table 16.2: Minimum Offset Distances

Geotechnical Guidance	Minimum Value (M)
Minimum offset from nearest stope, primary access and haulage ramp	30
Minimum offset from nearest stope, internal access ramps	20
Minimum offset from nearest stope, footwall drives	20
Minimum offset from nearest stope, drawpoint scram drifts	8
Minimum offset from nearest stope, ore and waste passes	25
Minimum offset from nearest stope, ventilation raises	25
Minimum offset from nearest stope, conventional raise	15

Source JDS 2015

16.3.7 Ground Support

Based on the range of Q' values as well as the size and expected life of the various mine openings, ground support recommendations were developed according to the Barton (1974) criteria. Different criteria were used for primary (permanent access) and secondary (temporary) stopes and development openings.

16.4 Hydrogeology Analysis and Recommendations

16.4.1 Conceptual Hydrogeologic Model

The development of the conceptual hydrogeological model of the Buriticá area was based on recently obtained field information, previous investigation results, and experience with similar hydrogeologic environments.

Available information verified that the hydrogeologic system is a fractured rock aquifer, where groundwater flow and movement is controlled predominantly by the quantity and permeability of fractures associated with faulting and mineralization, and not by primary rock permeability, which overall is low. It is believed that the aquifer system in the Buriticá area is dominantly unconfined, although some areas may locally be confined due to lack of fracturing.

16.4.2 Hydrogeologic Modeling

To estimate groundwater inflows to underground workings during mining, Montgomery & Associates constructed a numerical groundwater flow model calibrated with available hydrogeologic data.

16.4.2.1 Groundwater Flow Direction

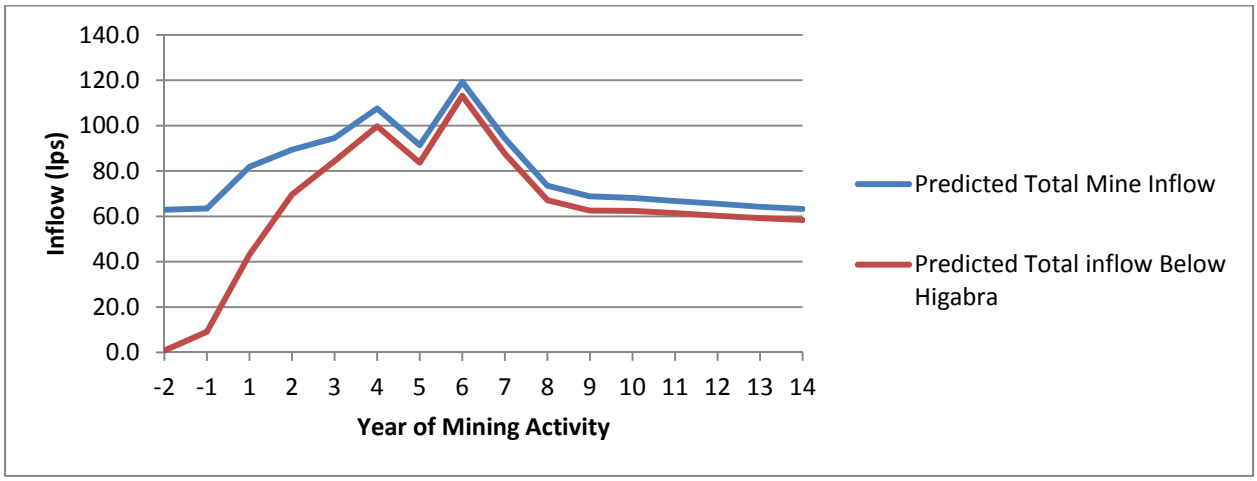
Groundwater flow direction is controlled by hydraulic gradient and also the orientation of the fractures. Regionally, the groundwater system flows are generally to the north and east following topographic gradient. Locally, the groundwater gradient indicates that flow in the Buriticá area is toward the east/Northeast from Yaraguá and Veta Sur areas.

16.4.3 Hydrogeologic Modeling Results

Figure 16.3 shows modeled groundwater inflow rates using the mine design and schedule. The main focus of the hydrologic modeling is to estimate the groundwater inflow volumes to the mine workings during development and operations. Using hydraulic conductivity, porosity, and permeability parameters in conjunction with infiltration coefficients, Montgomery & Associates estimated groundwater inflows.

Inflows were estimated for areas both above the Higabra level where water will flow by grafity from the mine and below the Higabra level, where pumping will be required. As mining advances below the Higabra into the deeper levels of Yaraguá, groundwater inflows increase over time to reach a maximum in years 4 to 6 when mining occurs between elevation 555 and 730 masl. These modeled inflows were used to design and estimate costs of pumping stations with sufficient capacity for peak estimated inflows. By the end of year 10, ore between elevations 555 and 730 masl will be mined out, and total inflows stabilize at about 60 l/s. Seventy-nine percent of LOM ore originates from above the Higabra level, and is not impacted by these deeper flows and pumping requirements.

Figure 16.3: Modeled Groundwater Inflow Rates

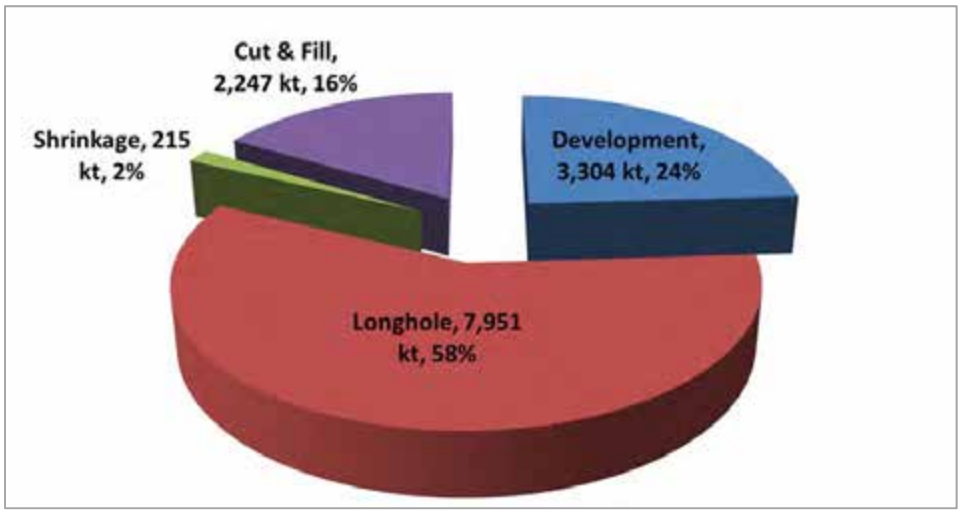


Source JDS 2015

16.5 Mining Methods

Three underground mining methods were selected for Buriticá; LHOS, C&F and shrinkage stoping. The bulk of the mining is with LHOS. C&F stopes are prevalent closer to surface in the Yaraguá deposit and along the Tonusco and Diatreme faults zones where ground is weaker. Shrinkage stoping is reserved for discrete, narrow veins in isolated areas that cannot support the sublevel development to mine them with more productive methods. Figure 16.4 illustrates the ore tonnage breakdown by mining method. Substantial mill feed comes from development ore.

Figure 16.4: Ore Tonnes by Mining Method



Source JDS 2015

16.5.1 Longhole Mining

LHOS will be used where rock quality and ground conditions allow and where vein thickness is relatively uniform. It is the highest productivity method selected for the production phase, but given many veins are relatively narrow, considerable preparation is required to maintain stope inventory.

LHOS is the least selective of the three methods when applied over long vertical distances due to drill hole deviation and vein geometry variations. To mitigate these effects, longhole drilling distances at Buriticá will be limited to 12 m.

During top and bottom sill development, mine geologists will inspect and map headings to identify structural controls and collect channel samples as part of the mine grade control protocol. Information from channel sampling and mapping in the top and bottom sills will be used, in conjunction with exploration and definition drilling results, to identify the economic limits and complete final stope designs. Mine experience to date verifies that structural boundaries can be used to define vein and stope limits. The mining engineers will use all collected grade control information to prepare drill hole and blasting sequences to minimize extraction dilution and maximize ore recovery. Mine surveyors will then locate and mark drill hole locations and provide drill hole lengths and orientations for drilling.

According to final stope design, the longhole mining cycle will begin with blasting the drop raise to provide a free face for the first longhole round and initial empty volume for blasted swell. Production blasting will begin at the stope ends and retreat to the access.

16.5.2 Cut and Fill

C&F mining methods were selected for areas that have lower quality rock. C&F was selected for all stopes within 70 m of surface due to observed weathering profiles, which are associated with weaker ground conditions. In areas of documented historic mining, C&F was also selected because these areas are typically near surface and the selectivity allows safe and high recovery extraction. This method limits vertical stope wall exposure during extraction.

The C&F mining cycle begins with the pivot ramp driven downward from the footwall to access the bottom of the stope block at its midpoint along strike. Once the bottom sill is mined, production uppers (2 m) are drilled and blasted for the first cut, and then the ore is mucked and the stope backfilled with mine waste rock using a remote LHD. The back is then taken down in the pivot ramp to access the next lift. The stope is then supported with the drill jumbo or miners using jacklegs working from the waste fill. Once ground support is installed, mine geologists will map and sample the stope back as part of the grade control protocol. The next cut won't be drilled until the limits of economic mineralization are marked on the back.

An alternative to the uphole technique (uppers) would be breast mining in which the face is advanced in horizontal 2-3 m rounds, depending on stope width, and then ground support installed. This C&F technique allows ground support to be installed soon after blasting and is suitable for weaker ground shorter standup times.

16.5.3 Shrinkage Stoping

Shrinkage stoping accounts for less than 2% of planned mill feed but is nonetheless an important option for mining methods selected for Buriticá and offers additional production flexibility. Shrinkage stopes are designed and planned in isolated areas of the mine where extensive level development is not justified or required. It will be the least productive of the three mining methods and will be used only where economically justified.

Stope development begins with sill cuts driven on vein at the top and bottom of the stope block along with conventional raises driven at both ends. Raises provide personnel and service access, as well as flow through ventilation. Extraction drifts and drawpoints are driven on the bottom level.

Production mining commences with drilling the full stope with uppers using conventional jackleg drills and stopers. After blasting, approximately 30% to 35% of ore blasted within the stope must be drawn down in a controlled fashion so enough broken ore remains in the stope as a work platform. The process of drawing down ore can result in an uneven working surface within the stope so slushers and shovels will be used to level the stope floor as ground support is installed on re-entry.

Before subsequent rounds are drilled and blasted, mine geologists will inspect, channel sample, and mark the boundaries of economic mineralization. Keeping blasted ore within the stope provides a working surface for miners as well as maintaining stope sidewall stability.

The last step of the shrinkage stope mining cycle will be to completely draw down all remaining blasted ore within the stope. Access into the stope by miners will no longer be required so the stope is completely emptied. Once emptied, the stope can be left empty, or filled with waste rock or paste backfill.

16.6 Mining Dilution and Recoveries

See Sections 15.5 and 15.6 for a discussion on how mining dilution and recovery parameters were developed and used for stope design.

16.7 Mine Design

Existing access ramps at Yaraguá and Veta Sur will be slashed to 5 m x 5 m, and portals reconstructed as necessary prior to continuing development. The Higabra portal and adit will not require slashing or portal reconstruction. All main ramps will be mined to full dimensions of 5 m x 5 m with an arched back. Ramps will be located as close to the deposits as possible yet far enough away to ensure life of mine (LOM) stability.

Mine design criteria were developed as shown in Table 16.3.

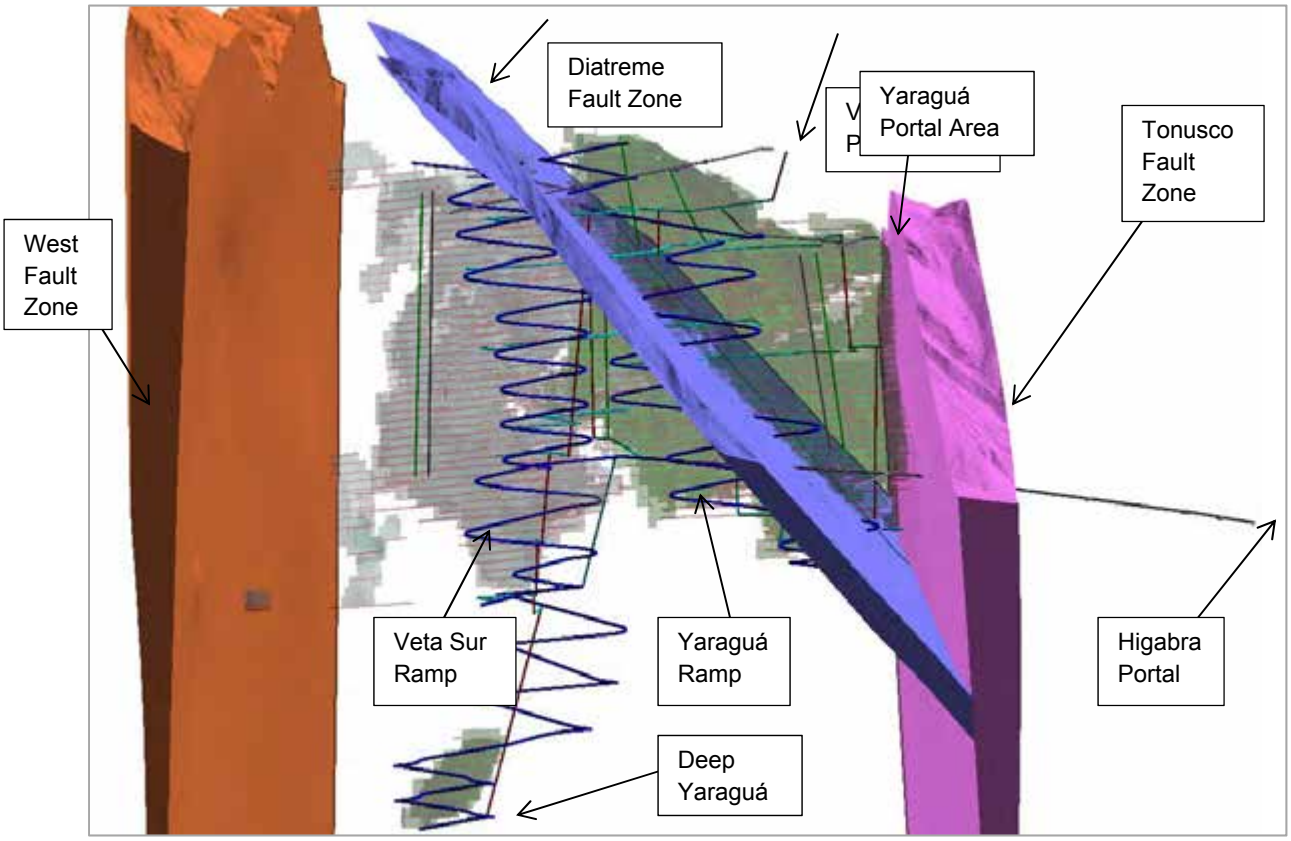
Table 16.3: Mine Design Criteria

Development Heading Parameters - Horizontal/Incline/Decline	Width (m)	Height (m)	Length (m)	Maximum Gradient (%)	Minimum Gradient (%)
Main access and haulage ramps	5	5	Varies	15	15
Centerline radius of curvature - main access and haulage ramps			20		
Internal access ramps for longhole, C & F, and shrinkage stopes	4	4	Varies	15	15
Centerline radius of curvature - internal ramps			16		
Attack ramps for cut and fill stopes	3.5	3.5	Varies	18	-18
Footwall drives above Higabra elevation (scooptram ore and waste to ore/waste passes)	3.5	3.5	Varies	2	2
Footwall drives below Higabra elevation (truck loadout required)	4.5	4.5	Varies	2	2
Longhole stope sills (minimum dimensions, maximum width depends on vein width)	2	3	30	2	2
Stope drawpoints	4	4	8	2	2
Exploration/definition drilling bays	4.5	6	15	2	2
Muck bays (main access and haulage ramps)	4.5	4.5	12	-20	-2
Sumps	4.5	4.5	8	-20	-2

Source: JDS 2015

Additional design criteria include avoidance of fault zones where possible, and when unavoidable, use of additional ground support and reduced advance rates in mine scheduling. Figure 16.5 shows an illustration of the primary ramps, portals, ventilation raises, ore and waste passes in relation to identified fault zones.

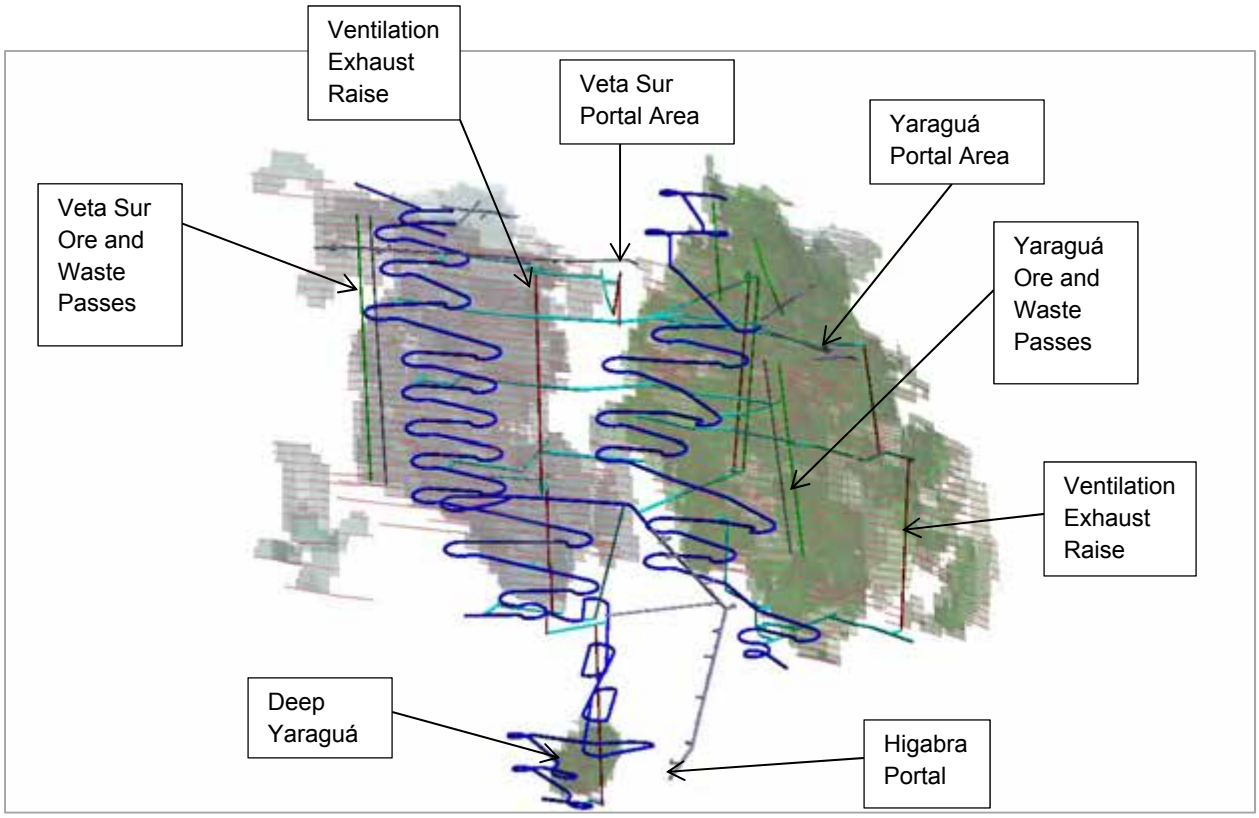
Figure 16.5: Primary Mine Infrastructure Design (Looking North)



Source: JDS 2015

Figure 16.6 shows the primary mine infrastructure in a 3D rendering looking North-East. The deep Yaraguá orebody will be accessed by extending the Veta Sur ramp to elevation 555 masl. The opportunity exists to complete infill drilling in the zone between the bottom of the main Yaraguá resource and deep Yaraguá. If the intervening Yaraguá inferred resource can be upgraded, this material could be included in future mine plans.

Figure 16.6: Primary Mine Infrastructure Design (3D Rendering Looking NE)



Source: JDS 2015

All primary ramps shown are 5 m x 5 m, ore and waste passes are 2.0 to 3.0 m diameter raise bores, and the majority of ventilation raises are raise bores 4.0 to 5.0 m in diameter. The ventilation raise servicing the deep Yaraguá is a 3.0 m x 5.0 m raise climber excavation.

16.7.1 Level Access Design and Layouts

Sublevel access will be developed at 15 m vertical intervals from the Veta Sur and Yaraguá ramps. Cross-cuts will be driven to access all veins on the level, and access drifts will be driven on vein as much as possible. Mine geologists will map and channel sample development headings to identify structural boundaries, and to verify location and intervals of economic veins for near-term mine design and planning. Concurrently, definition drilling will be performed to obtain additional information for mine planning and stope design. An extensive definition drilling program has been included in the operating plan. The information collected during development and operations will be used to reconcile with the resource model and use experience gained to improve resource estimation methodologies.

16.7.2 Access

Surface access to the Veta Sur and Yaraguá portals is by existing road. Higabra portal will be accessed from the main valley access road scheduled for construction prior to pre-production mining activities. These three portals will serve all mine access requirements for the mine life.

16.7.3 Development Types

Lateral development types include conventional mechanized jumbo drill & blast for primary and access ramps, footwall drives, cross-cuts, sumps, maintenance facility, and other excavations for mine services. This development is scheduled using CGI equipment and personnel.

Vertical development types include raise bores for ventilation, waste passes and ore passes. Raise bore diameters range from 2.0 m for rock passes to 5 m for ventilation, and will be contracted. Rectangular and square raises are also planned to be developed using a raise climber, which will also be contracted. A small amount of conventional raising may be required and will be accomplished using CGI equipment and personnel.

Boreholes of varying diameters are also planned to deliver paste backfill, accommodate pipes for clean and mine drainage water, and carry other services. These boreholes are scheduled using contractor raise bore machines that are able to drill pilot holes up to 24” diameter.

16.8 Mine Services

16.8.1 Mine Ventilation

The ventilation network and fresh air supply quantities were designed to comply with U.S. and Canadian ventilation standards. Required airflows were determined at multiple stages during the mine life, using equipment numbers and utilization, specific engine types and exhaust output, and personnel working underground. For Buriticá the following were determined:

- Equipment being purchased and used underground will meet the new Tier 3 or Tier 4 U.S. EPA diesel emission standards;
- During peak production, 16,680 m^3/min will be required to remove diesel emissions;
- Total 400 employees at 3.0 m^3/min/person underground will require 1,200 m^3/min; and
- The additional 1,320 m^3/min is for worker comfort and air quality.

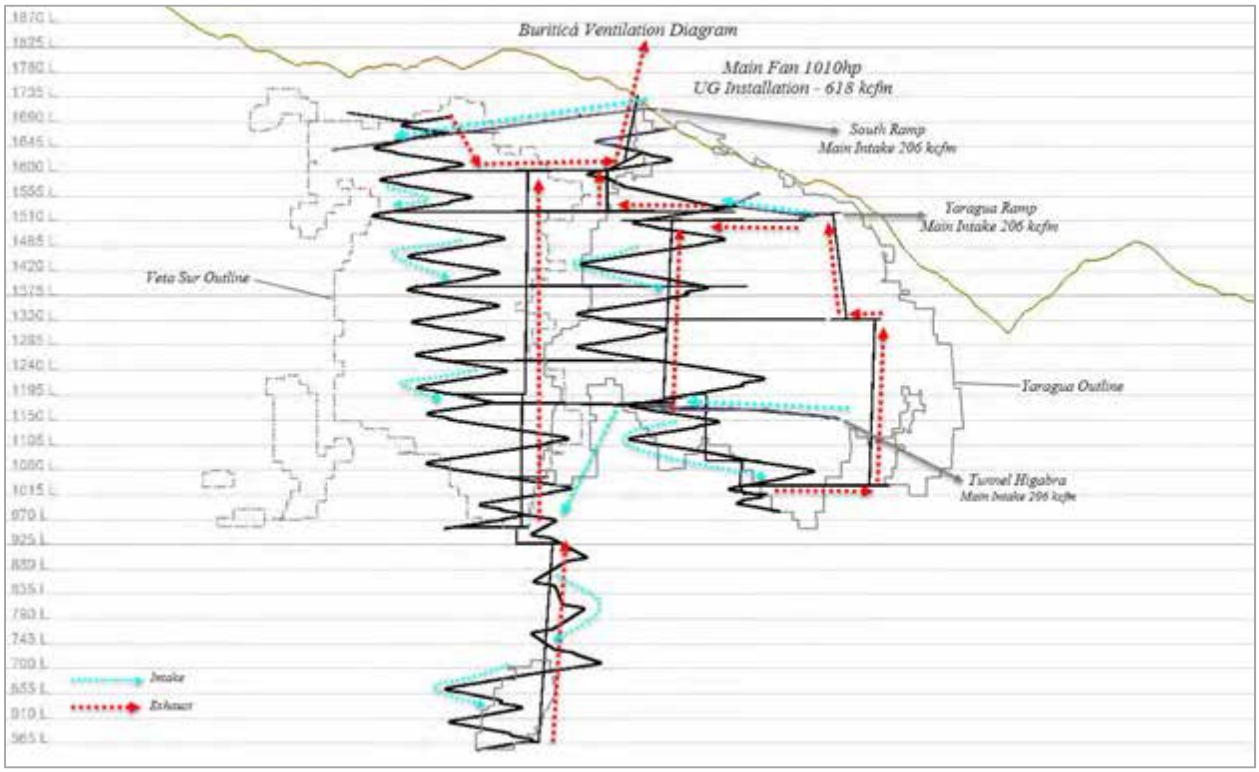
Total designed capacity is 19,200 m^3/min. Initially, surface fans, vent ducting, and booster fans will be used to advance underground development from the Higabra adit, Yaraguá and Veta Sur ramps. CGI has sufficient ventilation fans currently on-site to use for development start-up.

During pre-production, the peak ventilation quantity required will occur in month 16 (147.7 m^3/s or 312,700 cfm). After the connection is made between the Yaraguá ramp and upper Veta Sur ramp, exhaust fans will be installed at the Yaraguá portal, and fresh air will intake from the Veta Sur portal and the Higabra Portal. This connector drift will minimize pre-production ventilation fan requirements and power costs.

The permanent ventilation network will consist of three components: Primary ventilation fan, secondary fans and auxiliary fans. The main ventilation fan will be a single 19,200 m^3/min fan installed underground in a ventilation gallery. Fresh air will intake via the three portals and be directed to the active mining levels through the main ramps. Level ventilation will be controlled primarily by regulators. A series of 4.0 m and 5.0 m diameter exhaust raises will collect and direct exhaust air to the main fans and out of the mine. Air from the exhaust airways will not be re-used again in work areas.

Figure 16.7 shows the primary ventilation circuit to support mine production. All primary ramps will be fresh air throughout the mine life. The ventilation system is design to balance without using mine doors.

Figure 16.7: Primary Ventilation Network



Source: JDS 2016

16.8.2 Water Supply

During pre-production mine development, the current Yaraguá and Veta Sur mine service water systems will be used, and supplied from a stream source in the Higabra Valley. Once the Veta Sur ramp and boreholes for water pipes are complete the permanent mine water is supplied from dewatering flows.

Production water requirements are estimated to average 6 to 8 litres per second (l/s) with a peak demand of 12.1 l/s. Water from the Higabra level will be pumped to an intermediate holding/transfer tank located on the 1390 m level at a design rate of 12 l/s. From this location, water will distributed

on levels beneath the 1390 level, or pumped to the Veta Sur portal holding tank at the same design rate of 12 l/s. The surface tank will supply water for the upper levels, and fire water and water to the pastefill plant. Level switches installed in the intermediate and surface tanks will be connected to pump controls to maintain supply.

16.8.3 Dewatering

Drain holes from the Higabra adit will dewater the upper parts of the mine where it will flow by gravity to surface. For levels above Higabra, there are several service boreholes designed for each deposit and located between them which extend from the uppermost workings down to the Higabra level. Level accesses, cross-cuts, and most other mine development will be driven at an average +2% grade to direct mine water flows towards these main drainage boreholes, or intermediate sumps.

Dewatering below Higabra will include drainage galleries and collection sumps developed with the main decline. Pumps, designed to handle up to 10% solids, will discharge water to the Higabra adit where it will flow by gravity to surface. The cost for sumps, pumps and excavations are included as sustaining capital. All Higabra adit discharge will be piped to settling ponds or directly to the water treatment facility.

16.8.4 Electrical Distribution

Electrical power is currently supplied to Continental's small-scale operation from the 13.2 kV local power grid. During the pre-production period, peak power demand may require supplemental diesel gensets.

A new 110 kVA power line as described in Section 18, Project Infrastructure and Services, will be connected with the main substation located in Higabra Valley. From the Higabra substation, the main mine power cable will enter the Higabra portal, and overhead transmission lines will supply power to the Veta Sur and Yaraguá portals. All main mine power will be distributed at 13.8 kV; the primary ventilation fans will be 4,160 volts with other mine equipment specified at 600 volts. Mobile load centers and transformers at compressor locations will step down main mine power to required voltages. This main power distribution will follow ramp systems with level take-offs as required.

In the production phase, the mine power distribution system is designed with redundancy to ensure the mine maintains power in case one power line should be disrupted for any reason.

RDalflyen Consulting Inc. conducted a preliminary load flow study for the mine using the estimated mine power demand provided by JDS. The information provided was used to develop a model of the mine power system using ETAP™ software version 12.6.5. A Motor Acceleration Analysis was performed to identify any issues with motor starting when all the utilized loads are online. Soft starts are recommended for motors greater than 100 HP in order to improve overall system reliability. Furthermore, soft starts may be required to simultaneously start multiple motors.

Table 16.4 summarizes estimated mine power requirements by period.

Table 16.4: Summary LOM Power Requirements

Year	Total Connected Load (kW)	Total Utilized Load (kW)
-2 (pre-production)	1,314	425
-1 (pre-production)	1,812	725
1	5,657	2,921
2	6,575	3,502
3	6,951	3,877
4	8,770	4,597
5	9,825	4,658
6	9,523	5,166
7	9,549	4,474
8	9,479	4,082
9	9,344	2,964
10	9,094	2,781
11	8,959	2,899
12	9,364	3,029
13	8,192	2,065
14	7,546	1,016

Source JDS, 2016

16.8.5 Compressed Air

Compressed air will be supplied using 185 kW portable electric compressors that will service mining areas spanning 3 to 4 levels; these compressors will be relocated as mining progresses.

16.8.6 Mine Communications

A leaky feeder system with RFID tracking is included. Mobile equipment operators, light vehicles, and supervisors will be equipped with hand-held radios to communicate with personnel on surface. RFID tags on trucks will be used for tracking equipment movements and equipment dispatching.

16.8.7 Portable Refuge Chambers

Portable refuge chambers are planned for use where no secondary egress is available during mine operations.

16.8.8 Maintenance Facilities

An underground maintenance facility with an equipment wash bay, lube & oil change bays, jackleg repair, electrical shop, and warehouse will be constructed for routine services and small repairs. All equipment requiring a major service, and component change-out or repairs will be taken to the surface facility.

16.9 Unit Operations

16.9.1 Drilling

There are five principal drilling machines selected for Buriticá. Each has their own primary use:

- Electric-hydraulic development drill jumbos. Twin boom jumbos are planned for large dimension development rounds. A single boom jumbo will be used for smaller development headings;
- Bolting Machines – Ground support installation;
- Stopemate – Longhole drilling;
- Long Tom jackleg drill jumbos – C&F production drilling, Longhole stope sill development; and
- Jackleg drills – Shrinkage mining and general purpose drilling.

Drilling productivities (m/percussion hour) were built up from first principles and these vary by drilling machine type and heading dimensions. Jumbo drilling rates vary from 60 m/hr in small headings to 80 m/hr in large headings. Long Tom drill penetration rates average 35 m/hr, longhole drill machines average 38 m/hr and jackleg drilling averages 17 m/hr.

16.9.2 Blasting

Blasting crews will be trained and certified in Colombia for explosives use. ANFO will be used for most production blasting and some development rounds. Emulsion cartridges will be used in wet conditions. Also required will be boosters, primers, detonators, shock tube, detonation cord and other ancillary blasting supplies.

Bulk explosives will be stored in a secure surface powder magazine in accordance with current Colombian explosives regulations. The blasting crews will pick up the estimated quantities of explosives required for each shift using explosives transport vehicles and deliver those explosives to working faces and explosives-loading equipment underground. Excess explosives and accessories will be returned to the secure powder magazine every shift.

Blasting will occur at designated times using a centralized blasting system, and where ventilation allows, multi-blasting of isolated high priority development headings is anticipated.

16.9.3 Ground Support

Ground support will be installed in accordance with specifications based on geomechanical analysis provided by SRK for the various rock qualities expected (see Section 16.3.5). Electric-hydraulic bolters with screen handlers and shotcrete spraying machines will be used. Some ground support will be installed using Long Tom and jackleg drills.

There will be a shotcrete batch facility located at the Veta Sur portal area and transmixers have been included in the mining equipment list to deliver a wet mix to the shotcrete machine underground.

16.9.4 Mucking

The largest LHDs with a nominal 7.5 yd^3 bucket capacity will be used for primary ramp development and excavation of other larger development headings or stopes. The mid-size LHD has a 6 yd^3 bucket capacity and will be used for footwall drives, cross-cuts, and internal ramps, excavation of stopes and construction of backfill barricades. The smallest LHD has a 2.6 yd^3 bucket capacity and will be used to muck out longhole sill drifts and for working in C&F stopes.

Development LHD's will typically muck a blasted round to a nearby remuck bay in order to clear the working face prior to ground support installation. Rock temporarily stored in the remuck is then either trammed to a rock pass or loaded into a haul truck. Where development headings are proximal to Cut & Fill stopes, development waste rock will get trammed to the C&F stope for backfill.

Stope ore will be mucked with LHD and either direct trammed or trucked to the rock pass system. In-mine haulage distances will be limited to approximately 400 m.

16.9.5 Hauling

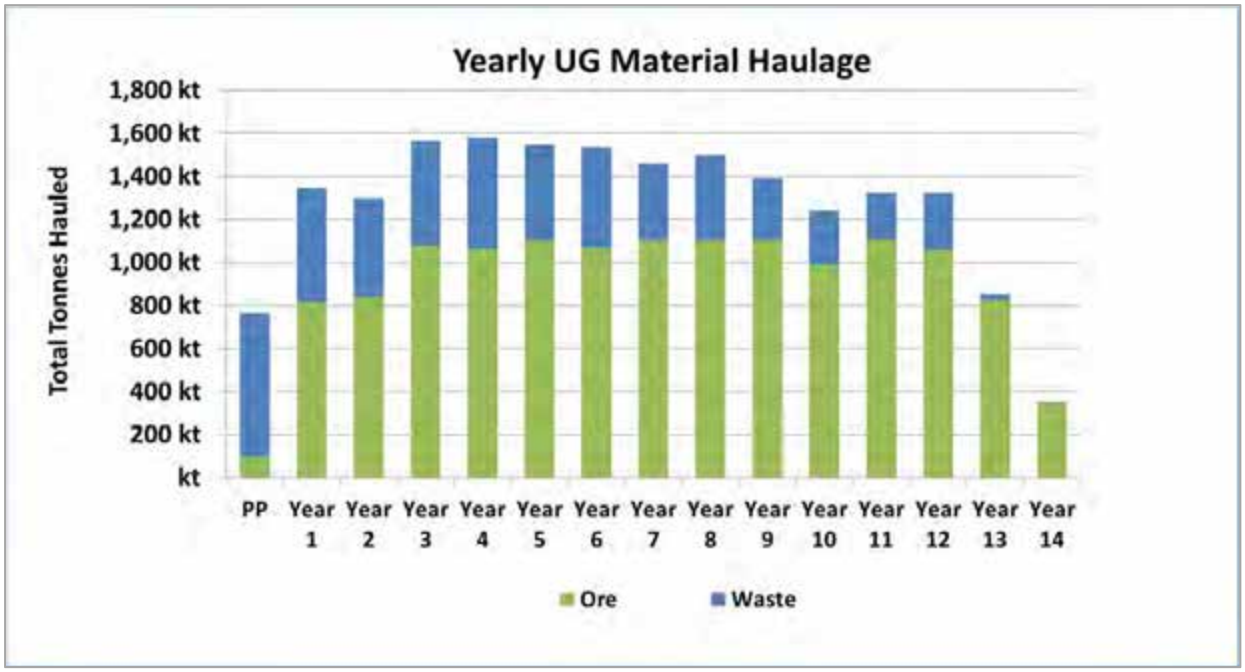
Haulage will be by conventional low-profile underground mining trucks of 30 or 40 t capacity. For mining areas located no greater than approximately 200 vertical metres above the Higabra level, trucks loaded underground will transport the ore/waste directly to surface. For areas located greater than approximately 200 vertical metres above the Higabra level, loaded trucks will haul to an ore or waste pass. Short rock passes are designed between sublevels and main ore & waste passes to cover the full vertical extent of the Yaraguá and Veta Sur deposits down to the Higabra level. Access to the main rock passes will be on primary sublevels approximately every 45-60 m vertically.

Ore and waste delivered to the main rock passes will be truck loaded on the Higabra level for truck haulage to surface. Truck chutes for loading ore and waste at the Higabra level are included in the fixed mining equipment list.

Ore hauled to surface during pre-production will be placed in a lined stockpile area to be moved later to the crusher. During operations, ore will be hauled directly to the crusher stockpile. Waste rock hauled to surface will be deposited at the south end of the TSF where it will be rehandled by the tailing facility surface fleet for placement as structural fill for TSF construction.

See Figure 16.8 for a summary of ore and waste tonnes hauled during the LOM production period.

Figure 16.8: Yearly Underground Haulage Summary



16.9.6 Backfill

The principle method of backfilling for Buriticá will be with paste backfill comprised of filtered tailing, water, and cement at varying proportions. Cemented Rock Fill (CRF) will be used as required during early mine development until the mill is running and the paste plant is commissioned. Loose Rock Fill (LRF) will be used in C&F stoping operations and to fill stope voids that do not require engineered backfill.

16.9.6.1 Paste Backfill

MineFill Services Inc. (MFS) verified that paste fill can be produced with site materials with the strength requirements which meet geomechanical design criteria. The paste backfill plant is designed as a continuous mix plant with a capacity capable of processing 100% of the tailing solids produced by the mill (3,000 t/d), and an operating capacity which will be typically 60 to 70% of the design capacity. The lower operating rate allows the paste plant to catch up in the event that backfilling falls behind due to bottlenecks in the mine. The Veta Sur portal location (elevation 1,716 masl) was selected for the paste plant due to proximity to existing and future mine workings and the use of gravity to deliver the paste. Other advantages of this location included proximity to the highway for cement deliveries and to the existing power infrastructure. Locating the paste plant close to the source of tailing at a lower elevation was investigated but was determined not viable due primarily to requirements for pumping paste backfill long vertical distances.

The paste delivery network was designed with two independent borehole paste distribution systems, one each for Yaraguá and Veta Sur. Automated paste diverter valves were designed into the delivery system. Piping on levels will be connected to paste delivery boreholes for distribution to stopes on or below that level.

16.9.6.2 Cemented Rock Fill

Prior to having the complete paste plant and tram transport system commissioned, CRF will be used. CRF is a mixture of mine development rock, cement and water that will be produced underground using a LHD to mix the components in a muck bay then tramming the mixed CRF to the mined stope.

16.9.6.3 Loose Rock Fill

Development waste will be used as Loose Rock Fill (LRF), especially in C&F mining areas. This development waste rock can also be used for filling longhole stopes that do not require engineered backfill.

16.10 Mine Equipment

All underground mine equipment required to meet the life of mine plan is summarized in Table 16.5.

Table 16.5: Underground Mobile Equipment Fleet (average number of units)

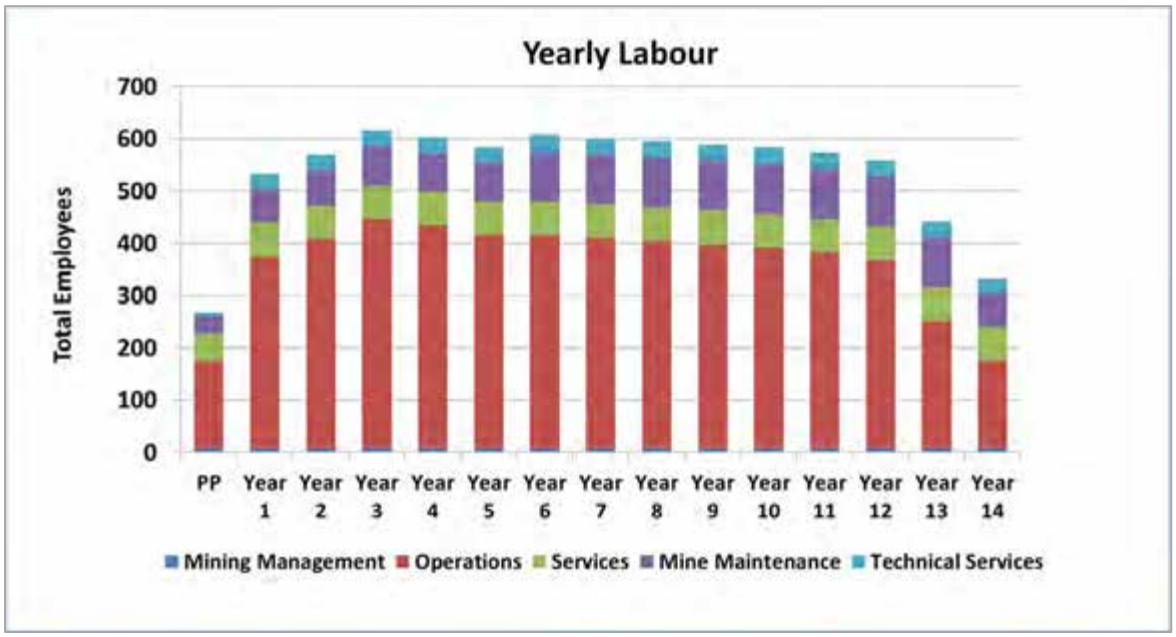
	YEAR															
EQUIPMENT TYPE	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14
Jumbo 2 Boom	3	3	3	3	4	4	4	4	4	4	3	3	2	2	1	1
Jumbo 1 Boom	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Long Tom Twin Boom Drill	0	5	8	8	8	8	8	8	8	8	8	8	8	7	3	1
Longhole Drill	0	2	3	4	4	4	4	4	4	4	4	3	3	3	3	2
LHD (4t/2.6yd)	1	4	8	10	11	13	13	13	12	12	12	12	12	12	9	6
LHD (10t/6yd)	1	2	5	5	6	6	6	6	5	5	5	5	5	5	3	2
LHD (14t/7.5yd)	2	2	3	3	3	2	2	2	2	2	2	2	2	2	1	1
40 Tonne Haul Truck	3	4	4	4	3	3	2	2	2	2	2	2	2	1	1	1
30 Tonne Haul Truck	1	4	10	10	12	12	12	12	12	12	12	12	12	10	8	5
Bolter	2	2	3	3	3	3	3	3	3	3	3	3	3	3	2	2
Motor Grader	1	1	2	2	3	3	3	3	3	3	3	3	3	3	3	3
Scissor Lift	1	4	5	6	6	6	6	6	6	6	6	6	5	5	2	2
ANFO Loader	2	4	6	6	6	6	6	6	6	6	6	6	6	6	5	3
Boom Truck	1	1	2	3	3	3	3	3	3	3	3	3	3	3	3	3
Fuel/Lube Truck	2	2	2	3	3	3	3	3	3	3	3	3	3	3	3	3
Mechanic Truck	1	1	2	3	3	3	3	3	3	3	3	3	3	3	3	3
Boom Truck	1	1	2	3	3	3	3	3	3	3	3	3	3	3	3	3
Personnel Carrier	3	4	6	7	7	7	7	7	6	6	6	6	6	6	4	3
Utility Vehicle	1	1	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Shotcrete Machine	1	2	2	2	2	2	2	2	2	2	2	2	2	2	1	1
Transmixer	1	1	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Backhoe with Rock Breaker	0	0	0	2	2	2	4	4	4	4	4	4	4	4	4	4
Telehandler	2	2	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Light Vehicle - Various Users	17	17	17	18	18	18	18	18	18	18	18	18	18	18	18	18

Source: JDS 2016

16.11 Mine Personnel

Life of mine personnel requirements are summarized in Figure 16.8.

Figure 16.9: Mine Labour by Year



Source: JDS 2016

Tables 16.6 to 16.11 show minimum and maximum numbers of personnel planned for the life of mine for Mine Management, Operations, Mine Services, Mine Maintenance, and Technical Services, respectively.

Table 16.6: Mine Management

Position	Minimum	Maximum
Mining Manager	1	1
Mine Superintendent	1	1
Maintenance Manager	1	1
Technical Services Manager	1	1
Yaraguá Mine Captain	2	2
Veta Sur Mine Captain	2	2
Mine Clerk	2	2
Sub-Total	10	10

Source JDS 2016

Table 16.7: Mine Operations

Position	Minimum	Maximum
Yaraguá Shift Supervisor	3	3
Veta Sur Shift Supervisor	3	3
Blasting/Powder Crew Supervisor	3	3
Blasting/Powder Crew - Miner II	6	18
Blasting/Powder Crew Helper - Auxiliary Miner	6	18
Development Lead Miner - Miner I	6	24
Ore Production Lead Miner - Miner I	3	6
Jumbo Operator - Contractor Trainer	0	6
Jumbo Operator - Miner I	9	12
Jumbo Operator - Miner II	9	12
Longhole Drill & Blast Operator - Contractor	0	12
Longhole Drill Operator - Miner I	9	12
Longhole Drill Operator - Miner II	9	12
Long Tom Drill Operator - Miner I	21	24
Long Tom Drill Operator - Miner II	21	24
Scooptram Operator - Miner I	18	27
Scooptram Operator - Miner II	24	39
Haul Truck Operator - Miner II	39	45
Bolter Operator - Miner I	3	9
Bolter Operator - Miner II	3	9
Jackleg Operator - Miner II	9	48
Jackleg Operator - Miner III	9	48
Grader Operator - Miner II	6	9
Backhoe Operator - Miner II	6	12
Nipper - Miner III	6	9
Grizzly Attendant - Auxiliary Miner	6	12
Sub-Total	**237**	**456**

Source JDS 2016

Table 16.8: Mine Services

Position	Minimum	Maximum
Mine Services Supervisor	3	3
Backfill Superintendent	1	1
Backfill Supervisor	3	3
Backfill Crew - Miner II (LHD)	6	6
Backfill Crew - Miner III	6	6
Backfill Crew - Auxiliary Miner	6	6
Ventilation - Auxiliary Miner	3	3
Compressed Air - Auxiliary Miner	3	3
Mine Water - Auxiliary Miner	3	3
Services Ground Support - Miner I	3	3
Services Ground Support - Miner II	3	3
Pump Station Attendant	3	3
Mine Electrician	3	3
Telehandler Operator - Miner II	12	12
Transmixer Operator - Miner II	3	3
Sub-Total	**61**	**61**

Source JDS 2016

Table 16.9: Mine Maintenance

Position	Minimum	Maximum
Maintenance Supervisor	3	3
Lead Mechanic	6	13
Heavy Equipment Mechanic I	3	9
Heavy Equipment Mechanic II	3	9
Heavy Equipment Mechanic III	3	9
Mechanic Helper - Auxiliary	12	18
Electric/Hydraulic Mechanic I	6	9
Electric/Hydraulic Mechanic II	6	9
Pneumatic Machine Mechanic II	12	18
Sub-Total	54	97

Source JDS 2016

Table 16.10: Technical Services

Position	Minimum	Maximum
Chief Engineer	1	1
Geotechnical Engineer	1	1
Chief Geologist	1	1
Chief Surveyor	1	1
Ventilation Engineer	1	1
Mine Surveyor	2	2
Surveyor Helper	4	4
Sr Geologist	1	1
Jr Geologist	1	1
Sampler	9	12
Sr. Mine Engineer	1	1
Jr. Mine engineer	1	1
Projects Engineer - surface & U/G	1	1
Short Term Mine Planner	1	1
Medium Term Mine Planner	1	1
Long-Term Mine Planner	1	1
Draftsman	1	1
Resource Modeler	1	1
Sub-Total	**30**	**33**

Source JDS 2016

Table 16.11: Total Mine Workforce

Department	Minimum	Maximum
Management	10	10
Mine Operations	237	456
Mine Services	61	61
Mine Maintenance	54	97
Technical Services	30	33
Total	**392**	**657**

Source JDS 2016

Modern, large scale underground hard rock mining is new to Colombia, and for this reason, several key positions will be filled with expatriate professionals. Skilled expatriate miners will also be included in early years of mine life as trainers and operators.

16.12 Mine Production Schedule

The Buriticá underground mine schedule was optimized using Minemax™ iGantt schedule optimizer software. To produce a balanced schedule, inputs and constraints that represent the design, mining productivities and unit operations were included in the optimization process. See Table 16.12 for maximum daily development rates used.

Table 16.12: Development Productivities

Single Headings	Advance Rate (m/d)
Portal Rehab/construction	0.65
Critical Infrastructure – Ramp slashing and rehab, Ramps, Footwalls, connections, vent access	6
Level Access, connection (unless already at 6 m/d)	5
All other development and ramps below Higabra (expected water inflow)	3.5
Multiple Headings where Development is Grouped into Levels	
More than 1,400 m required on level	20
1,000 to 1,400 m required on level	15
500 to 1,000 m required on level	10
Less than 500 m required on level	5

Source JDS 2016

Development and production ramp-up were modeled using initial periods of reduced productivities. Pre-production scheduling guidelines were established to:

- Ensure sufficient development to sustain ore production when plant construction and commissioning is complete;
- Include only development that was required for pre-production; and
- Adequate longhole stope preparation and initiation of longhole mining in selected locations, and
- Provide a development ore stockpile for plant commissioning.

The maximum ore mining rate was capped at 3,000 t/d and the minimum ounces mined was set to achieve early high ounce production for the constraints set. A number of schedule iterations and manual adjustments to the sequence were made in order to produce a robust, sensible, and realistic schedule.

Tables 16.13 and 16.14 show single stope productivity constraints and production tonnage per sublevel constraints included in the plan.

Table 16.13: Single Stope Productivities

Mining Method	Ore Tonnes/Day
Longhole	170
C&F	65
Shrinkage	55

Source JDS 2016

Table 16.14: Stoping Productivities by Tonnes per Level

Mining Method	Tonnes per Level	# of Working Stopes	Ore Tonnes/Day
Longhole	Less than 6,000	1	170
	6,000 – 15,000	2	340
	15,000 – 30,000	3	510
	30,000 – 110,000	5	850
	More than 110,000	6	1,020
C&F	Less than 5,000	2	130
	5,000 – 7,000	3	195
	7,000 – 9,000	5	325
	9,000 – 13,000	6	390
	More than 13,000	7	455
Shrinkage		1	55
		2	110

Source JDS 2016

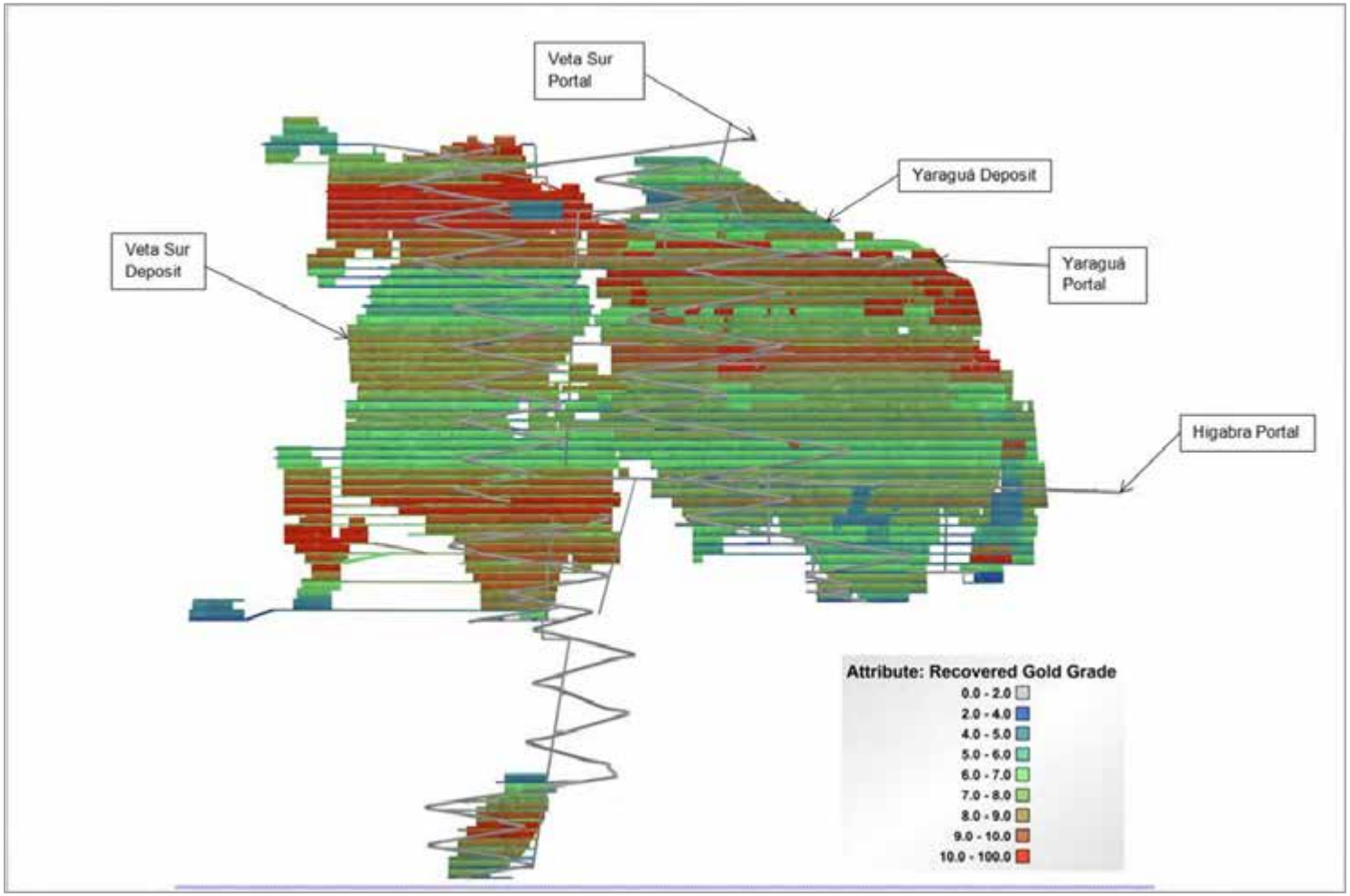
The schedule targets the highest grade areas of Veta Sur and Yaraguá early in the mine life, with a maximum of four and minimum of two mining areas active at any one time. Mining areas will be comprised of 3 to 4 levels each to focus available resources and share mine infrastructure as much as possible. Production mining will begin at the lowest level of each mining area and follows a logical ascending sequence.

Final results of the iGantt schedule were organized such that physical metres, tonnes and ounces were broken down into different categories for direct use in the cost model. From the final schedule, cost model requirements including items such as the mining fleet, manpower, consumables, ventilation, pumping, and power were determined and used to develop costs from first principals. Reports were generated by month from the start of development (pre-production) through to the first two years of production, quarterly for years 3 and 4, and annually thereafter. See Table 16.15 for a summary of mine schedule results, and Figures 16.9 to 16.11 for mine development and ore extraction sequence.

Table 16.15: Summary Mine Schedule Results

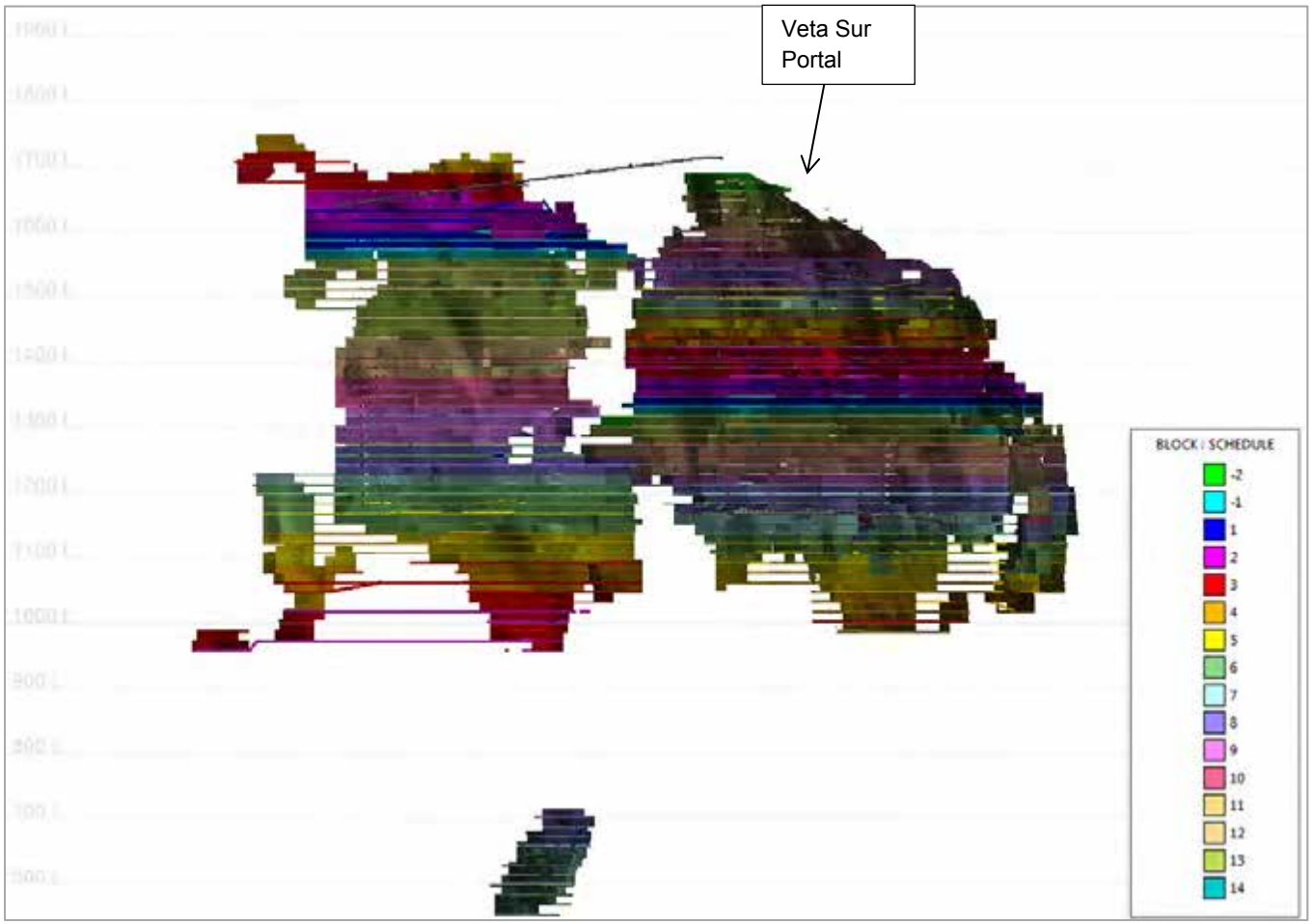
		Pre-Prod	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Total
Mine Plan Quantities	**Unit**																
Total Stoping Ore	ktonnes	20.6	527.5	568.8	815	804.4	813.7	781.1	826.2	886.3	819.8	686.7	867.2	823	819.7	352.9	10,413.00
Total Development Ore	ktonnes	76.6	286.7	270.8	263.7	263.2	288.9	289.3	277.3	217.5	282.9	303.4	239.2	237.9	6.3	-	3,303.80
Grand Total Ore Mined	ktonnes	97.2	814.1	839.6	1,078.70	1,067.60	1,102.50	1,070.40	1,103.60	1,103.90	1,102.70	990.2	1,106.40	1,060.90	826	352.9	13,716.80
Mined Au Grade	g/t	6.94	12.16	11.81	8.56	7.74	8.81	8.31	8.07	8.07	7.61	7.64	7.51	6.96	7.54	8.83	
Mined Ag Grade	g/t	22.48	37.14	34.72	27.95	21.36	22.2	22.64	23.01	24.23	22.02	21.5	20.79	20.33	22.86	25.51	
Veta Sur Waste	ktonnes	292.4	361.8	119.1	140	120.7	104.6	105.7	58.6	164.3	76.9	75.9	90.9	133.8	10.5	1.8	1,857.00
Yaraguá Waste	ktonnes	377	171.8	339.3	346.5	394.3	340.3	358.7	298.6	233.1	211.6	174	124.4	129.6	16.5	2.4	3,518.20
Total Waste	ktonnes	669.4	533.6	458.4	486.4	515.1	444.9	464.5	357.2	397.4	288.5	249.9	215.3	263.4	27	4.1	5,375.20
Lateral Waste Development	metres	14,735	12,037	11,635	13,690	13,480	11,511	12,402	10,456	9,954	8,884	7,981	6,601	8,126	259	-	141,752
Lateral Ore Development	metres	4,289	12,796	13,443	12,411	13,520	15,489	14,598	13,304	11,586	12,791	13,709	11,691	12,124	294	-	162,043
Total Lateral Development	metres	19,024	24,834	25,078	26,100	27,000	27,000	27,000	23,760	21,540	21,675	21,690	18,291	20,250	554	-	303,796

Figure 16.10: Mined Grade Distribution



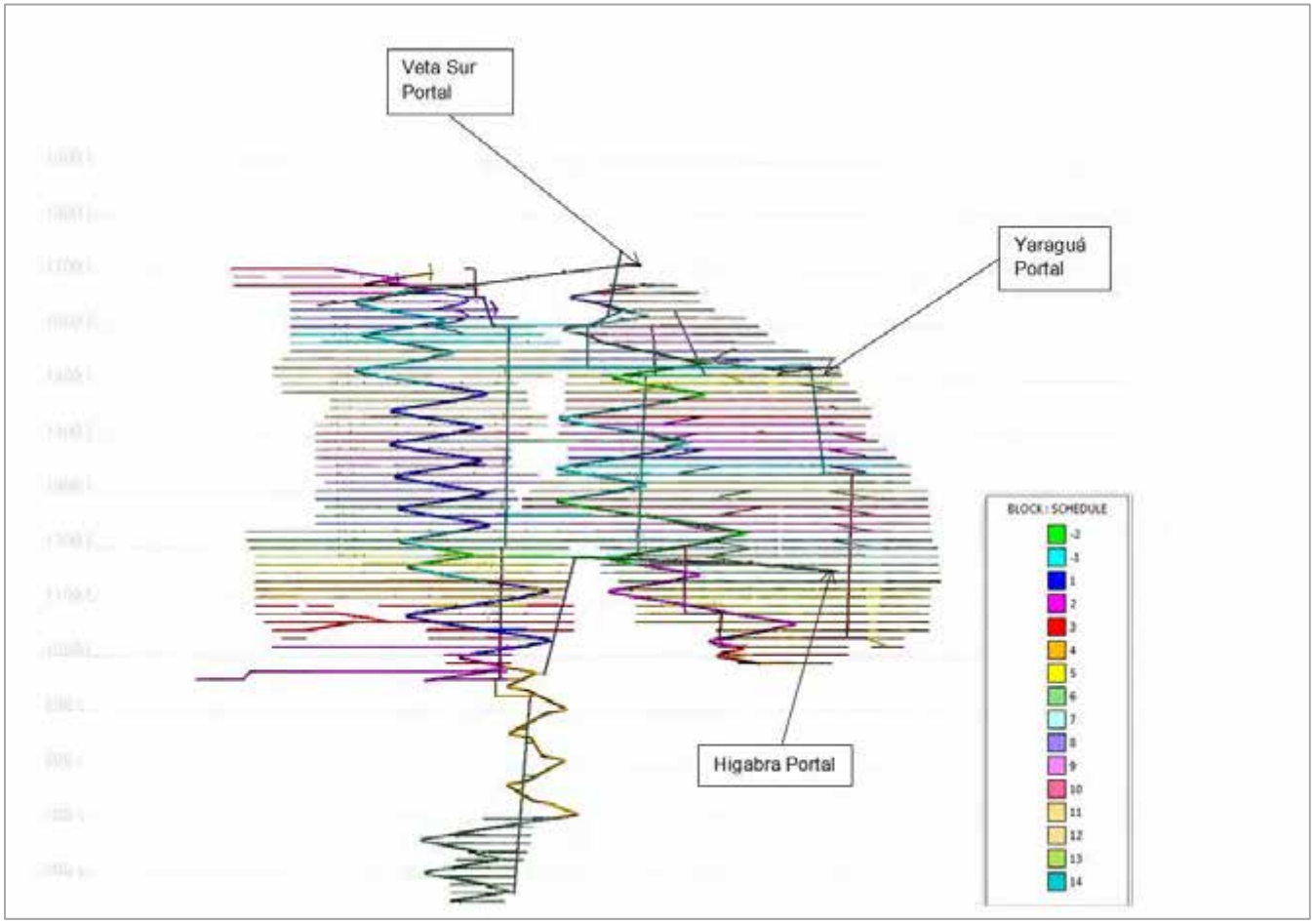
Source: JDS 2016

Figure 16.11: Life of Mine Ore Extraction Sequence - Composite Long Section Looking North



Source: JDS 2016

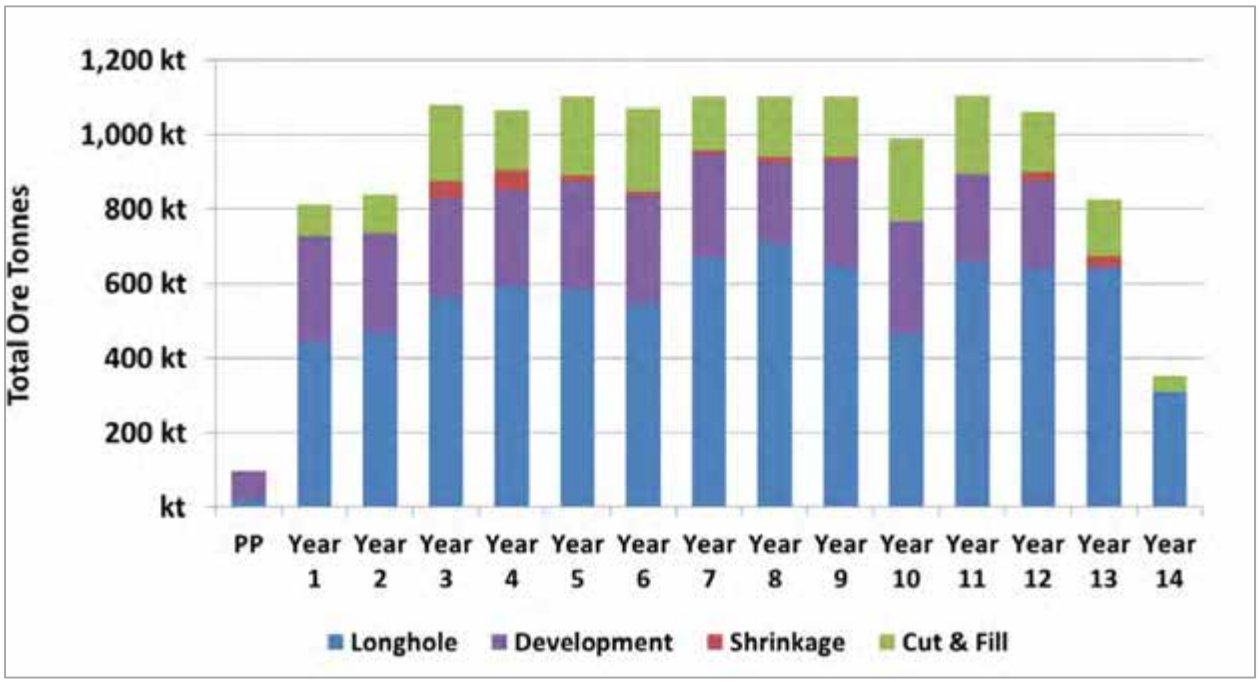
Figure 16.12: Life of Mine Development Sequence - Composite Long Section Looking North



Source: JDS 2016

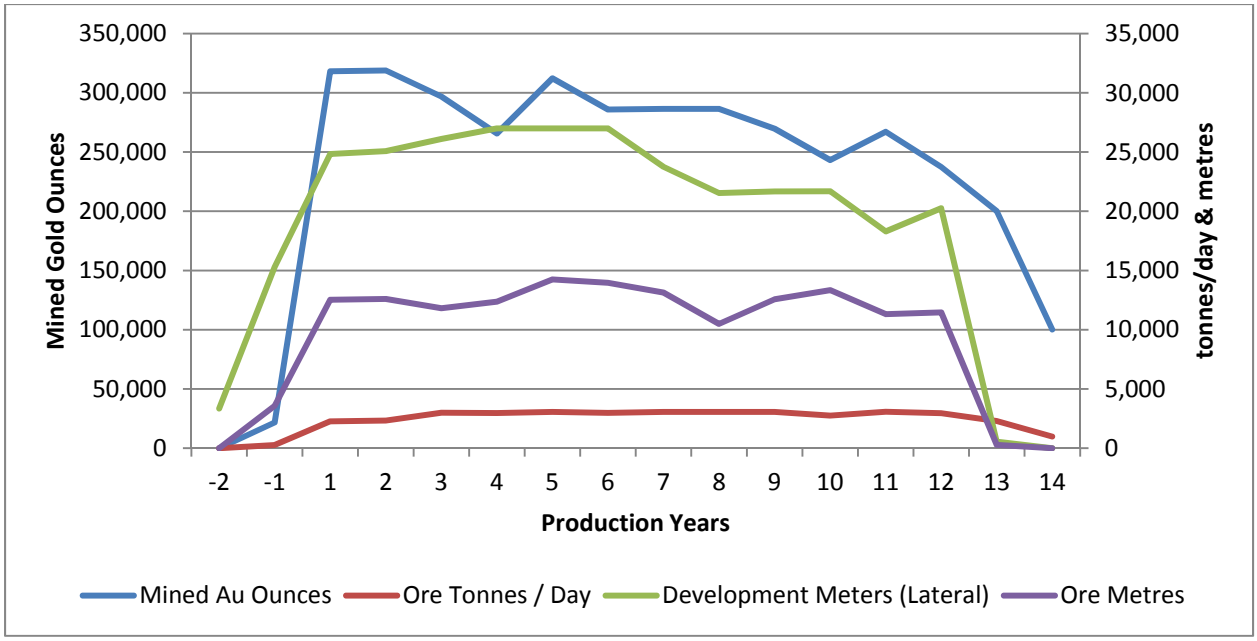
Figure 16.12 shows yearly ore production by mining method, and Figure 16.13 shows LOM primary production metrics.

Figure 16.13: Yearly Ore Production by Mining Method



Source: JDS 2016

Figure 16.14: Life of Mine Production Metrics



Source: JDS 2016

Mined gold ounces will average slightly over 300,000 oz/year for years 1 to 5, and then decrease gradually for the remaining mine life.

Total lateral development advance rates will average 65 metres/day during years 1 to13. This average rate is achievable due to multiple mining areas and headings simultaneously under development.

A total of 303,400 m lateral development and 7,360 m vertical development are planned. Raise bores of differing diameters account for 4,520 m vertical development, 800 m of raise will be developed using a raise climber, and the balance of 2,050 m is estimated for internal drop raises for ore and waste handling between levels.

Approximately 1,920 meters of service boreholes covering the vertical extents of Yaraguá and Veta Sur will be drilled using large diameter raise bore pilot hole drilling equipment.

17 Process Description/Recovery Methods

The key points of this section are:

- The proposed flowsheet is based on interpretation of test work completed to date by metallurgical consultant as described in Section 13;
- The process design follows these steps: Crushing > Grinding > Gravity Concentration > Cyanide Leach > Counter-Current Decantation (CCD) > Merrill Crowe > On-site refining to doré bars;
- Tailing will be dewatered by filtering prior to disposal;
- The process plant utilizes conventional technology and equipment which are standard to the industry;
- The process plant is designed to process 3,000 tonnes per day, or 1,095,000 tonnes per year at 91% availability, operating for 365 days per year; and
- Process water is recycled and thus minimizes water consumed by process.

17.1 Plant Design Criteria

Table 17.1 is a summary of the main components of the process design criteria used for the study. More detailed design criteria have been produced for internal use.

Table 17.1: Process Design Criteria

Description	Unit	Design
Mineral Characteristics		
Maximum mine-run mineral size, mm	mm	357
Mineral specific gravity (process design)		2.79
Mineral bulk density, (weight) t/m^3, design	t/m^3	1.6
Mineral Moisture Content		
Design	%	7
Range	%	3 to 7
Mineral Assay (Average)		
Gold assay, g/t	g/t	8.5
Silver assay, g/t	g/t	24.2
Mineral abrasion index, Bond, (Ai)	g	0.185
Mineral Work Index, kWh/t		
Crushing (Cwi)	Whr/t	13.6
Rod mill (Bwi) 75th percentile	Whr/t	19.54
Ball mill (Bwi) 75th percentile	Whr/t	17.2
Production Schedule		
Mineral Crushing and Milling Rate, average, dry t/y	t/y	1,095,000
Mineral Crushing and Milling Rate, average, dry t/d	t/d	3,000
Production Schedule		
Crushing		

Description	Unit	Design
Days per year		365
Hours per day, @3000 t/d	t/d	12
Shifts per day, @3000 t/d	t/d	1
Hours per shift		12
Shifts per week		7
% Availability (excluding start up)	%	75
Mineral crushing rate, design, dry t/h	t/h	333
Grinding, Gravity, Leaching		
Days per year		365
Hours per day		24
Shifts per day		2
Hours per shift		12
Shifts per week		14
% Availability (excluding start-up)	%	91
Milling rate, design, dry t/h	t/h	137
Merrill Crowe		
Days per year		365
Hours per day		24
Shifts per day		2
Hours per shift		12
Shifts per week		14
% Availability (excluding start-up)	%	99
Refinery		
Days per year		260
Hours per day		12
Shifts per day		1
Hours per shift		12
Shifts per week		5
% Availability (excluding start-up)	%	99
Tailing Filter Plant		
Days per year		365
Hours per day		24
Shifts per day		2
Hours per shift		12
Shifts per week		14
% Availability (excluding start-up)	%	80

Source: M3, 2016

The process mass balance was developed for the Buriticá process using MetSim. The process simulation assumed the following recoveries and grades based on completed test work (see Table 17.2).

Table 17.2: Metal Production Schedule

Parameter	Unit	Au	Ag
Mine Head Grades	g/t	8.5	24.2
Gravity Recovery	%	40 to 90	9
Leach Extraction	%	90	56
CCD Wash Efficiency	%	99	99
Merrill Crowe Extraction	%	99	99
Overall Extraction	%	96	60
Soluble Loss	%	0.5	0.5
Overall Plant Recovery	%	95	59
Metal Production	kg/d	24	43
Metal Production	Troy oz/y	284,000	503,000

Note: Recovery information provided by metallurgical consultant.

Source: M3, 2016

17.2 Plant Design

This section presents the process design that will govern the design of the mineral processing facility including crushing, milling, gravity concentration, agitation leaching, counter-current decantation CCD, Merrill Crowe zinc precipitation, cyanide oxidation, and tailing. The process plant designed for the Buriticá Project utilizes processes and equipment which are standard for the industry. This includes cyanide leach followed by CCD, and utilizing filtered tailing for disposal.

The design basis for the processing facility is 3,000 dry metric tons per day (t/d) or 1,200,000 dry metric tons per year (t/a) at 91% mill availability.

The following items summarize the process operations required to extract gold and silver from the Buriticá mineralized material.

- Size reduction of the mineral by a primary jaw crusher to reduce the ore size from run-of-mine (ROM) to P_{80} minus 150 mm;
- Storing primary crushed material in a covered stock pile and then reclaiming by feeder and conveyor belt;
- The crushed material will be conveyed to a grinding circuit to liberate gold and silver minerals. The grinding circuit consists of a wet semi-autogenous (SAG) grinding mill and a ball mill. The SAG mill will be operated in closed circuit with a screen and a pebble crushing circuit. The ball mill will be operated in closed circuit with hydro-cyclones to produce a cyclone overflow product with the desired grinding product size of 80% passing 74 µ. The SAG and ball mill discharge will be treated in a gravity circuit;
- The gravity circuit will consist of two high capacity, continuous centrifugal concentrators followed by a magnetic separator and a concentrating table. The gold concentrate will be upgraded to a

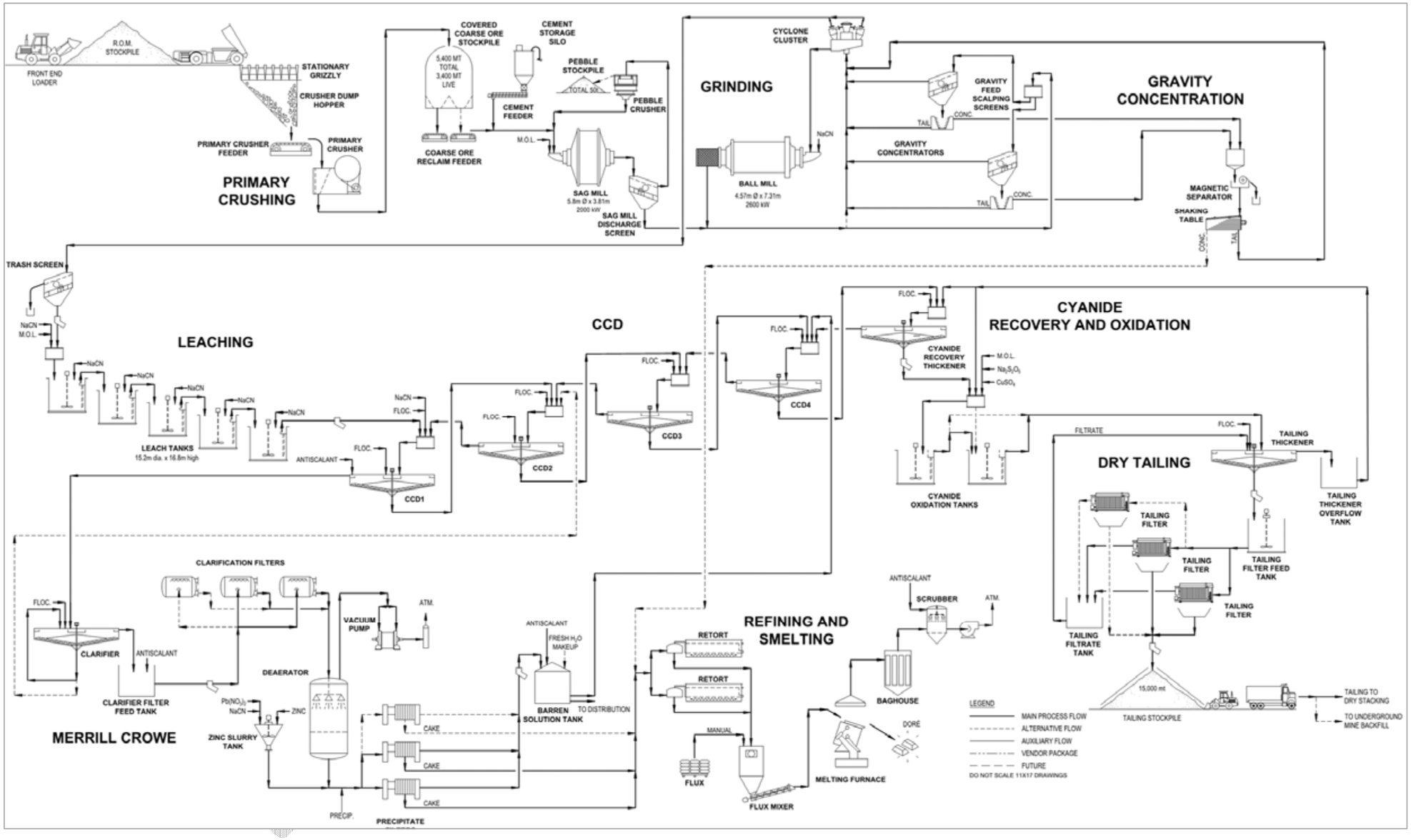
concentrate that may be refined directly. Concentrate table tailing will be returned to the grinding circuit;

- The cyclone overflow will pass over a trash screen ahead of the leach tanks;
- Cyanide leaching of the slurry in agitated tanks with oxygen addition;
- Liquid/solid separation using a four stage CCD circuit;
- Recovery of gold and silver from the pregnant leach solution in a Merrill Crowe plant;
- Melting the zinc precipitate and gravity concentrate with fluxes to produce a gold-silver doré bar which is the final product of the mineral processing facility;
- Partial recovery of the cyanide in a thickener;
- Oxidation of residual cyanide in the leach tailing stream using a sulphur compound and oxygen. Copper catalyst addition is not expected to be required due to sufficient level of soluble copper in the leach tails;
- Thickening and filtering of oxidized slurry to recycle water to the process;
- Filtered tailing will be placed in a lined TSF; and
- Water from tailing dewatering is recycled for reuse in process. Plant water stream types include: process water, fresh water, and domestic water.

17.2.1 Storage, Preparation and Distribution of Reagents to be used in the Process

Reagents which require handling, mixing storage and distribution include: sodium cyanide, caustic soda, cement, calcium hydroxide, lead nitrate, flocculant, antiscalant, sodium metabisulfite, copper sulphate pentahydrate, diatomaceous earth, metallic zinc dust, and refinery fluxes. A simplified schematic of the overall process of the proposed processing plant is presented in Figure 17.1.

Figure 17.1: Overall Process Flow Diagram of the Proposed Processing Plant



Source: M3, 2016

17.3 Process Plant Description

17.3.1 Crushing

ROM mineralized material will be trucked from the underground mines by 30 and 40-tonne rear dump haul trucks to stockpiles to be blended as needed prior to feeding primary crusher. The ROM mineral stockpile will be designed to contain 14,000 t. A front-end loader (FEL) will feed a stationary grizzly over the crusher dump hopper. Alternatively the trucks may dump directly into the dump hopper. A mobile hydraulic rock breaker will be provided to break oversize ROM material at the stockpile, the stationary grizzly, and at the crusher dump pocket.

An apron feeder will discharge the scalped ROM material directly into a jaw crusher, where it will be reduced from a size of F_{100} = 357 mm to a product size of P_{80} = 150 mm. The jaw crusher product will discharge onto the crusher discharge conveyor. The crusher discharge conveyor will discharge to the covered coarse ore stockpile. A belt weigh scale mounted on the crusher discharge conveyor will measure the feed to the coarse mineralized material storage.

The primary crushing facility will be equipped with an air compressor for maintenance equipment such as air tools. A self-cleaning electric magnet will be installed over the discharge of the stockpile feed conveyor to remove tramp metal.

A “wet spray” system will be installed to suppress dust in the mineral feed steams, transfer points and dump pockets.

17.3.2 Coarse Mineralized Material Stockpile and Reclaim

The covered coarse ore stockpile will have a 3,400 t live capacity and a 5,400 t total capacity. A bulldozer and/or a front-end loader will recover dead storage. There will be a concrete tunnel containing two draw points below the crushed material stockpile that will provide material to two reclaim feeders, one operating and one standby, which will discharge onto a mill feed conveyor. Each feeder will have a capacity of 137 t/h nominal (250 t/h maximum). Reclaim feeders will be variable speed and controlled to maintain a set point mineral feed rate to optimize the grinding circuit. The mill feed conveyor will transport the crushed mineral from the stockpile to the SAG mill.

A belt weigh scale mounted on the mill feed conveyor will measure the new feed to the SAG mill providing a signal for adjusting the reclaim feeders speed, makeup water addition, and cement addition. Cement will be fed dry to the SAG feed conveyor to provide alkalinity for the leach reaction.

A dust collector will be installed to collect dust around the discharge of the reclaim feeders. Collected dust will discharge onto the mill feed conveyor.

SAG grinding balls will be loaded using a ball loading system onto the mill feed conveyor.

17.3.3 Grinding and Gravity Concentration

The grinding circuit will process an average of 3000 t/d, at 91% availability, operating 24 hours per day, 365 days per year. A SAG mill will reduce crushed mineralized material from F_{80} = 150,000 µm to T_{80} 2000 µm and a ball mill will further reduce the mineral to P_{80} = 74 µm.

A SAG mill measuring 5.8 m diameter and 4.2 m long (3.8 m effective grinding length), powered by a 2000 kW variable speed motor will perform primary grinding of the material in closed circuit with a vibrating screen. The undersize material will pass through the vibrating screen into a gravity feed box, where it will combine with ball mill discharge.

Oversize material will be transferred to a SAG mill oversize conveyor and will be crushed in pebble crusher and returned to the SAG mill. Ball chips will be removed prior to the crusher using a self-cleaning electric magnet. Combined SAG and ball mill discharge slurry will be pumped using variable speed horizontal centrifugal slurry pumps to a splitter box ahead of the gravity circuit. Slurry will be split to feed two bowl concentrators each preceded by a scalping screen to remove oversize from the bowl concentrator feed. Scalping screen oversize will flow by gravity to the cyclone feed box. Gravity tailing will flow by gravity to the cyclone feed box.

Slurry will be pumped from the cyclone feed box to hydro-cyclones. The underflow of the hydro-cyclones will flow by gravity to a ball mill measuring 4.6 m diameter and 7.5 m long (7.2 m effective grinding length), powered by a 2600 kW motor. The ball mill will discharge over a trommel screen. The trommel product will be washed by process solution sprays and ball chips will be rejected out the end of the trommel into a tote bin. Trommel undersize will be combined with SAG mill discharge screen undersize. Cyclone overflow (final grinding circuit product) is directed to the leach circuit.

Cyanide solution may be added to the ball mill feed and/or to the gravity feed box. Barren solution will be used as dilution water in both the SAG mill and ball mill circuits.

Concentrate from the centrifugal gravity concentrators will feed a magnetic separator to remove grinding media and then to a cleaner gravity table. Concentrate from the cleaner table will be recycled and upgraded to produce a gravity concentrate that can be refined directly. Cleaner gravity tailing will be returned to the cyclone feed sump.

17.3.4 Leaching and CCD Plant

The hydro-cyclone overflow will flow by gravity to a trash screen for removal of tramp material. Trash screen oversize will discharge into a tote bin to be periodically removed for disposal. The trash screen undersize will report to the leach feed pump box.

The leach circuit will consist of five agitated tanks. Each tank will have a working volume of approximately 2,758 m^2 providing approximately 48 hours of retention time at 38% solids for 3,000 t/d. Cyanide solution may be added to the tanks as required to maintain desired free cyanide levels. Oxygen from the pressure swing adsorption (PSA) oxygen plant will be piped to all tanks and sparged under the agitator impeller to maintain the desired dissolved oxygen level in each tank. Milk of lime will be added to maintain pH.

Slurry will advance by gravity from leach tank to leach tank, exiting the last leach tank and reporting by gravity flow to the CCD feed sampler. The CCD feed sampler discharge will gravity flow to a series of four high rate 17 m diameter CCD thickeners for washing and solid-liquid separation. Flocculant will be added as needed to the thickeners feed to aid in settling.

The leach residue is washed in CCD to remove soluble gold and silver. Slurry, at 60% solids, will be advanced by pumping from thickener to thickener, exiting the last tank and reporting to the cyanide recovery thickener ahead of cyanide oxidation. Barren solution, used as wash water, is introduced into the final CCD thickener (CCD No. 4).

Solution is advanced by gravity counter-current to the solids. Overflow from CCD Thickener No. 1 (pregnant solution) will be pumped to a clarifier ahead of Merrill Crowe.

17.3.5 Cyanide Recovery Thickener and Cyanide Destruction

Underflow from the last stage of CCD will report to a high rate 17 m diameter cyanide recovery thickener. Flocculant will be added as needed to the thickener feed to aid in settling.

The withdrawal rate of settled solids will be controlled by a variable speed, thickener underflow pump to maintain either thickener underflow density or thickener solids loading. The cyanide recovery thickener underflow will be pumped to the oxidation circuit.

Overflow from the cyanide recovery thickener will gravity flow to the CCD Thickener No. 4 dilution box.

The cyanide recovery thickener underflow is sampled in the cyanide recovery tailing sampler. Underflow from the cyanide recovery thickener will be diluted to approximately 35% solids using tailing thickener overflow in the cyanide oxidation feed box. There are two cyanide oxidation tanks that may be operated in parallel or in series. In the cyanide oxidation tanks, Weak Acid Dissociable (WAD) residual cyanide will be oxidized to the relatively non-toxic form of cyanate by a process using sodium metabisulfite and oxygen, with copper sulphate as a catalyst when needed. Milk of lime will also be added to maintain a slurry pH of 8.0 to 9.0. The more stable iron cyanides are precipitated from solution as an insoluble ferrocyanide compound. The cyanide levels in the tailing slurry are thereby reduced. Copper catalyst addition is not expected to be required due to sufficient level of soluble copper in the leach tails. Each cyanide oxidation tank will provide a residence time of approximately two hours based on the test work.

Slurry discharged from the oxidation circuit will be pumped to the tailing thickener feedbox.

A concrete containment slab on grade and containment walls will contain rain runoff and process spills. A sump pump will transfer the spills back to the process.

17.3.6 Reagents

Reagent storage, mixing, and pumping facilities will be provided for all of the reagents used in the processing circuits. Table 17.3 below is a summary of reagents used in the process plant.

Table 17.3: Process Reagents and Consumption Rates

Reagent	Consumption (kg/t)
Sodium cyanide (NaCN)	1
Sodium Hydroxide (NaOH)	0.1
Cement	2.5
Calcium Hydroxide	0.6
Lead Nitrate ($PbNO_3$)	0.1
Sodium Metabisulfite ($Na_2S_2O_5$)	1.4
Zinc Dust	0.04
Diatomaceous Earth (DE)	0.15
Flocculant	0.12
Antiscalant	0.07
Copper Sulphate Pentahydrate ($CuSO_4$*$5H_2O$)	
Refinery Fluxes	0.05
Grinding Balls – 125 mm	0.8
Grinding Balls – 50 mm	1.1
Crusher Liners	0.007
Mill Liners	1.9

Source: M3, 2016

17.3.7 Merrill Crowe Precipitation

Gold, silver and mercury will be recovered from the pregnant solution by adding fine metallic zinc dust to precipitate the metal ions. The precipitate will be filtered out in filter presses and then retorted to remove trace levels of mercury. The dried precipitate is blended with fluxes and smelted for production of doré bars containing silver and gold.

The process of recovering silver and gold by the Merrill Crowe process includes:

- Clarification and filtering of pregnant solution to remove suspended solids;
- De-aeration of pregnant solution to reduce dissolved oxygen;
- Precipitating gold and silver metal by addition of metallic zinc dust;
- Filtering and air drying of precipitate;
- Heating the precipitate in a vacuum chamber to remove mercury; and
- Melting the precious metal precipitate in a crucible furnace to produce doré bars.

Pregnant solution from the CCD thickener No. 1 overflow tank will be pumped to a clarifier. Solution will be processed through the clarifier to remove solids prior to being pumped to the Merrill Crowe area. Pregnant solution from the clarified filter feed tank will be pumped using a horizontal centrifugal pump, to three self-cleaning pressure leaf clarifier filters. The clarifier filter will be leaf type with an automated leaf wash sequence that will activate as needed based on pressure or time. The operating filters will be pre-coated using DE as a filter aid, and in addition, has a continuous body feed addition of DE to assist filtering as needed.

For these purposes, a pre-coat tank and a body feed tank will be provided with the filter units. The clarifier pre-coat pump will be a horizontal centrifugal pump. The clarifier body feed pump will be a peristaltic type pump which will pump filter aid in to the filter feed stream. Pressure for filter operations will be provided by the clarifier filter feed pump. Filtrate, clarified solution, will discharge directly into the deaerator tower via a spray nozzle within the tower.

Clarified solution will be passed through the deaerator tower to remove dissolved oxygen to less than 0.2 ppm prior to zinc dust addition. The deaerator will be maintained at close to an absolute vacuum by a vacuum pump. The deaerator internals will include a packed bed that serves to distribute the clarified solution into a series of thin films. The deaerator level control valve will be submerged in barren solution to ensure that no air leaks into the lines via the valve seals or flanges. Dissolved oxygen level will be monitored using an installed sensor.

The clarified and deaerated pregnant solution will be withdrawn from the bottom of the deaerator tower by a single-stage, vertical, in-line, centrifugal pump, submerged in barren solution to prevent re-entry of air through the pump gland. The pump will discharge through the precipitation filter presses. Just prior to the filter press feed pump an emulsion of zinc dust and barren solution will be added to the deaerated solution to precipitate the silver and gold.

Zinc dust will be hand loaded into a zinc feeder hopper and will discharge via a feeder into a zinc dust mixing system which will emulsify the zinc dust with barren solution. DE may be pumped using a peristaltic type pump to the zinc slurry tank as body feed as required, to extend the filter cycle. The slurry tank will be continuously supplied with barren solution via a constant head tank to prevent air from entering the process. The slurry is then pumped to three plate and frame filter presses, where the precipitated precious metals are collected. The operating filters are pre-coated using DE as a filter aid. The precipitate pre-coat pump will be a horizontal centrifugal pump. The plate and frame press is air blown to remove solution, and manually opened and cleaned with precipitate being collected in carts.

Barren solution (filtrate) exiting the Merrill Crowe circuit will be pumped to the barren solution tank using vertical, in-line, centrifugal pumps. A recirculation loop will be provided to allow high barrens occurring after press changes to be recirculated and retreated. Barren solution will be distributed to the grinding area, gravity concentration circuit, concentration leach, cyclone overflow trash screen as spray water, CCD dilution boxes, pre-coat and body feed tanks, zinc slurry tank, cyanide mix tank, caustic mix tank, lead nitrate mix tank, as gland water, and to the filter press feed pump water box. Excess barren solution will be sent to the oxidation circuit as needed.

Pregnant solution is sampled ahead of the unclarified solution tank. Barren solution is sampled ahead of the barren solution tank.

17.3.8 Refinery

The refinery unit operations will be batch operations. The frequency of the batch operation will be varied to accommodate anticipated grade variations. Zinc precipitate will be loaded into retort boats, or trays, for treatment in a retort. The retort will heat precipitate/concentrate to 650°C to vaporize mercury, which may be present in low concentrations. Retort vapor will be withdrawn from the retort by a vacuum pump, which will pull the vapor through a condenser where the mercury will condense and flow into a mercury collection tank. Mercury will be recovered from the tank periodically. Exhaust from the retort vacuum pump will pass through a sulphur impregnated carbon (SIC) filter before venting to atmosphere.

The retort, mercury condenser system, mercury collection tank, vacuum pump, and carbon filter will be supplied as a complete packaged system ready for utility hook-up and operation.

Following the retort cool down cycle, the dry precipitate and/or the concentrate will be mixed with fluxing materials and will be charged to a diesel fired, indirect fired crucible melting furnace and brought to the melting temperature. When it is fully molten, the charge separates into two distinct layers: slag (on the top) and metal (on the bottom). The slag layer, containing most of the impurities, is poured off first into a conical slag pot. The metal layer, containing the gold and silver and minor impurities, is poured off next into bar molds. Doré will be sampled using vacuum tubes during pouring.

Doré bars will be the final product of the operation. Slag will be collected and returned to the mill circuit. Fumes from the melting furnace will be collected through ductwork and cleaned in a high temperature material baghouse and then with a wet scrubber with caustic addition before discharging a clean off-gas to the atmosphere.

The refining building will be constructed as a secure building.

17.3.9 Tailing Dewatering and Disposal

Oxidized slurry will be pumped to a 17 m diameter tailing thickener. Flocculant will be added as needed to aid in settling. The withdrawal rate of settled solids from the tail thickener will be controlled by two variable speed thickener underflow pumps to maintain either thickener underflow density or thickener solids loading. Tailing thickener underflow will be pumped to an agitated tailing filter feed tank. Tailing thickener overflow will be pumped using two horizontal variable speed centrifugal pumps back to the cyanide recovery circuit and to the oxidation circuit for dilution water.

Slurry from the filter feed tank will be pumped using variable speed horizontal centrifugal pumps to three tailing filters. The filter cake will fall onto a discharge conveyor. Filtered tailing at approximately 14.3% moisture (by weight) will be transferred to a covered tailing stockpile with a 15,000 t tailing capacity using a transfer conveyor. A belt scale will be used to maintain an accurate metallurgical balance. A sweep type sampler will be installed over the tailing transfer conveyor to collect samples for moisture analysis. Filtrate will discharge into a tailing filtrate tank and be pumped using fixed speed horizontal centrifugal pumps back to the tailing thickener feedbox.

17.3.10 Water Management and Requirement

A water balance for the process plant at a production rate of 3,000 t/d was developed for the Buriticá Project using MetSim modeling. The Buriticá process plant is projected to require approximately 24.6 m^3/h of fresh water makeup to sustain its operation. This is equivalent to 0.2 m^3 of water per tonne of mineral processed, which is within typical operating ranges for plants with filtered tailing.

Fresh water for the Buriticá Project will be supplied from wells and surface water. Water will be pumped from three operating wells to the fresh/fire water tank.

The fresh/fire water tank will supply the requirements for domestic water following treatment, various plant requirements, and fire water reserves in the event of an emergency. Fresh water system will be designed to prevent contamination with cyanide containing solutions.

Fresh water will be treated by a filter/chlorinator process, the treated water will be discharged into the domestic water tank. The domestic water will flow by gravity to the personnel and mill facilities.

17.3.11 Mill Power Consumption

The power consumption in the process plant for a typical year is shown in Table 17.4, with a total consumption of Megawatt hours per year (MWhr/a).

Table 17.4: Summary of Process Area Power Consumption

Code and Area Description	Average* Annual Power (MWh/y)	Average* Power / Tonne Processed (kWh/tonne)
AREA 100 — PRIMARY CRUSHING	583	0.59
AREA 200 — STOCKPILE AND RECLAIM	415	0.42
AREA 300 — GRINDING	29,079	29.68
AREA 310 — GRAVITY CONCENTRATION AND INTENSIVE LEACHING	972	0.99
AREA 315 — GRAVITY TABLE	245	0.25
AREA 400 — CONCENTRATE LEACH	2,995	3.06
AREA 450 — CCD	959	0.98
AREA 500 — MERRILL CROWE	4,135	4.22
AREA 510 — REFINERY	538	0.55
AREA 600 — WATER AND TAILING	5,220	5.33
AREA 630 — CYANIDE OXIDATION	507	0.52
AREA 650 — FRESH WATER SYSTEM	890	0.91
AREA 800 — REAGENTS	526	0.54
AREA 900 & 911 — ANCILLARIES AND BUILDINGS, MILL MAINTENANCE BLDG	3,188	3.25
AREA 906 — TRUCK SHOP BUILDING	197	0.20
AREA 909 — LABORATORY	132	0.13
Total	**50,580**	**51.62**

*Average based on 14.0 years.

Source: M3, 2016

18 Project Infrastructure and Services

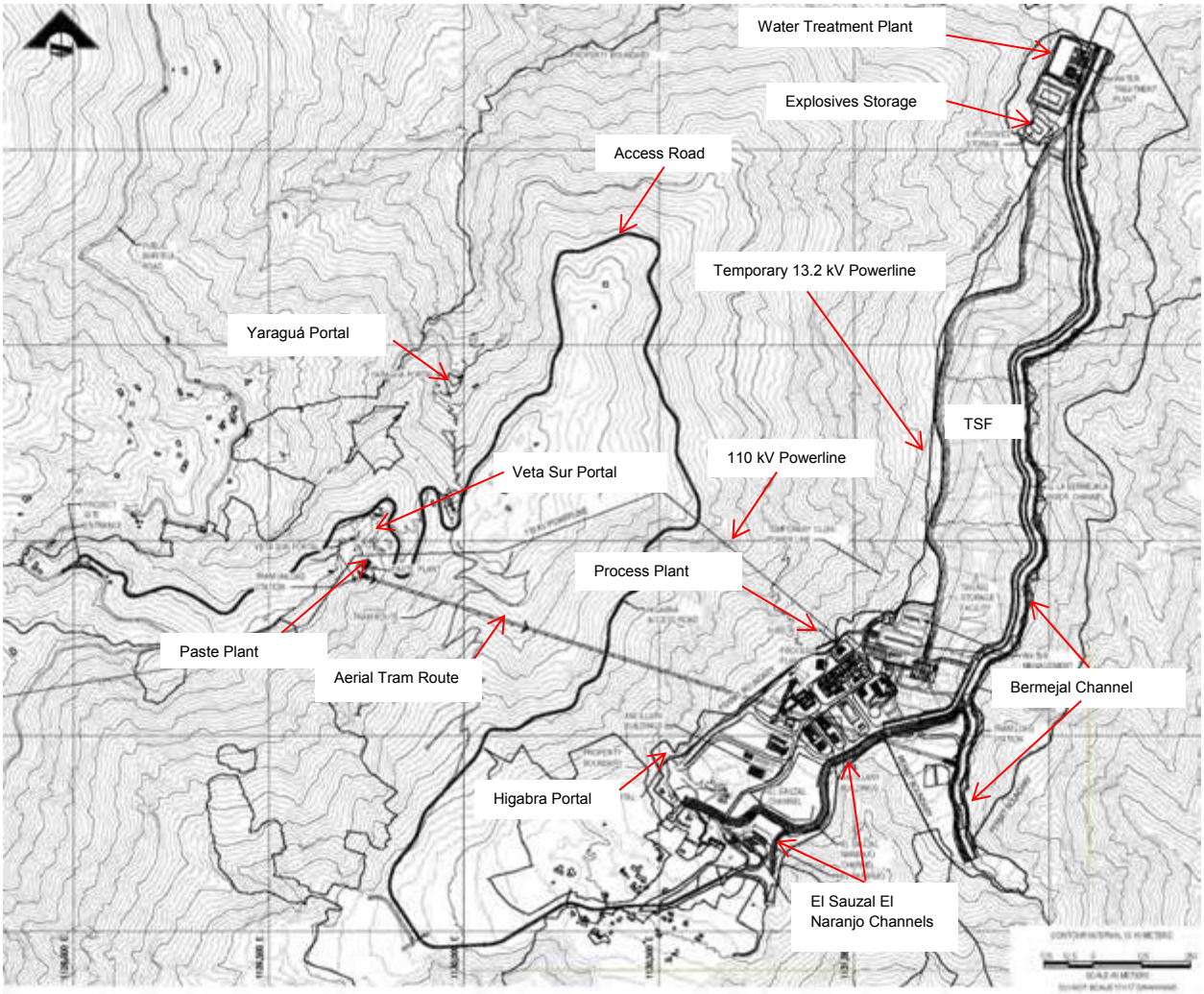
The Buriticá Project infrastructure and services are designed to support the operation of a 3,000 t/d mine and processing plant, operating on a 24 hour per day, 7 day per week basis. It is designed for the local conditions and rugged topography.

The main infrastructure for the Project consists of the following facilities:

- A 5.9 km access road between the existing property entrance and the Higabra Valley, originating at the paved road leading to Buriticá;
- Gold and silver processing plant with security, administration, and personnel facilities;
- Paste backfill plant for providing cemented paste to the underground workings;
- Aerial Tram for transporting tailing from the process plant to the paste backfill plant;
- Mine support facilities including mobile equipment maintenance, mine personnel facilities, and shotcrete mixing plant;
- Explosive storage facility;
- Utility infrastructure for the site: water, sewer, fire protection and communications;
- 13.2 kV grid power supply for the pre-production phase;
- 110 kV power transmission line connected to the EPM national electricity grid;
- Mine water sediment settling ponds and water treatment plant;
- Surface water handling infrastructure to manage local streams and runoff from the facilities; and
- Tailing Storage Facility (TSF).

The overall site layout is shown in Figure 18.1.

Figure 18.1: Overall Site Layout



Source: M3, 2015

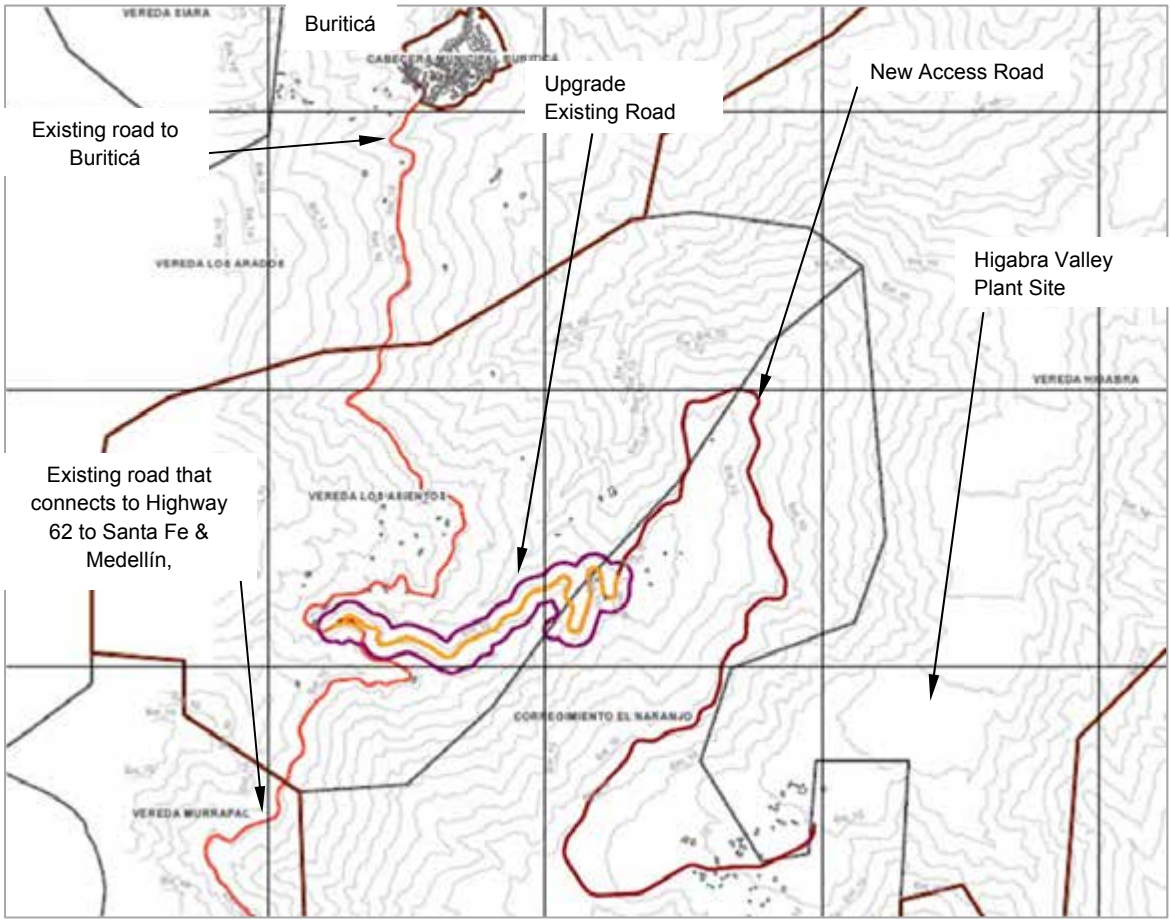
18.1 Access Road

18.1.1 Background and Description

The Project site is accessible via the highway network linking Medellín to Buriticá after turning northward towards Buriticá on a minor paved road. The project access road, partially constructed to 1.63 km, will connect the existing main entrance (along the minor paved road leading to Buriticá) to the Higabra Valley plant and TSF site. The road, which is designed for deliveries of equipment and supplies during construction and operation, has been developed to a detail engineering level by INTEGRAL Ingenieros Consultores S.A. (Colombia). The EIA for the road was approved by Corantioquia on August 30, 2012, and permits to construct the road have been granted.

The access road alignment is shown in Figure 18.2.

Figure 18.2: Access Road Map



Source: Integral, 2012

18.1.2 Design and Construction

The road will be built in three phases to provide the level of access needed as the Project develops while minimizing capital expenditures.

- Phase 1: Initial pioneering road to allow passage of construction machinery and semi-loads of equipment, using construction equipment assistance where necessary. Phase 1, which has already been completed to 1.63 km, will provide access to the valley in order to start construction and is scheduled to start in Q4 2016 and be completed in Q1 2017. This stage has already been bid for construction.
- Phase 2: Widening corners and reducing grades to allow highway tractor trailer traffic access for equipment deliveries; scheduled completion August 2017. Phase 2 includes gravel surfacing, slope stabilization and hydro-seeding.
- Phase 3: Completion to final specification and paving of this road, remaining slope stabilization work and hydro-seeding; scheduled completion in year two of operations.

The designs are based on:

- Colombian Norms for Road Design from INVIAS (National Road Institute); and
- A Policy on Geometric Design of Highways and Streets, 2011, AASHTO.

18.2 Site Geotechnical Conditions

The plant and TSF site geotechnical conditions have been investigated to a level sufficient for Feasibility Study. The existing site investigations comprise reports by Integral, 2014, Integral, 2015, geophysical seismic refraction data, site geologic mapping and reconnaissance investigations by SVS 2012. The materials of the upper 20 to 35 m consist of a heterogeneous mix of gravels and cobbles (to boulders) in predominantly clayey silt to silty sand matrix down to more distinct gravel, cobbles and boulder layers (consistent with slope wash, colluvium and alluvial deposition). The natural fill material is underlain by bedrock.

18.2.1 Infrastructure Foundation Requirements

Prior to detailed design, additional valley floor site characterization will be conducted in the plant site area to provide information for major structures and building foundation designs.

Facilities and building site pads will be built on cut surfaces, compacted and topped with coarse gravel. Areas of fill will be used primarily for storage and laydown areas. Side slopes of pads will be limited at 1.5:1, and where greater slope is required, gabion baskets or concrete retaining walls will be used. The concrete foundations are designed as raft slabs due to poor basement soils susceptible to differential setting; the area is a relatively high seismic zone.

18.3 On-Site Infrastructure

On-site infrastructure will be located as close to the process plant as possible to make efficient use of the space.

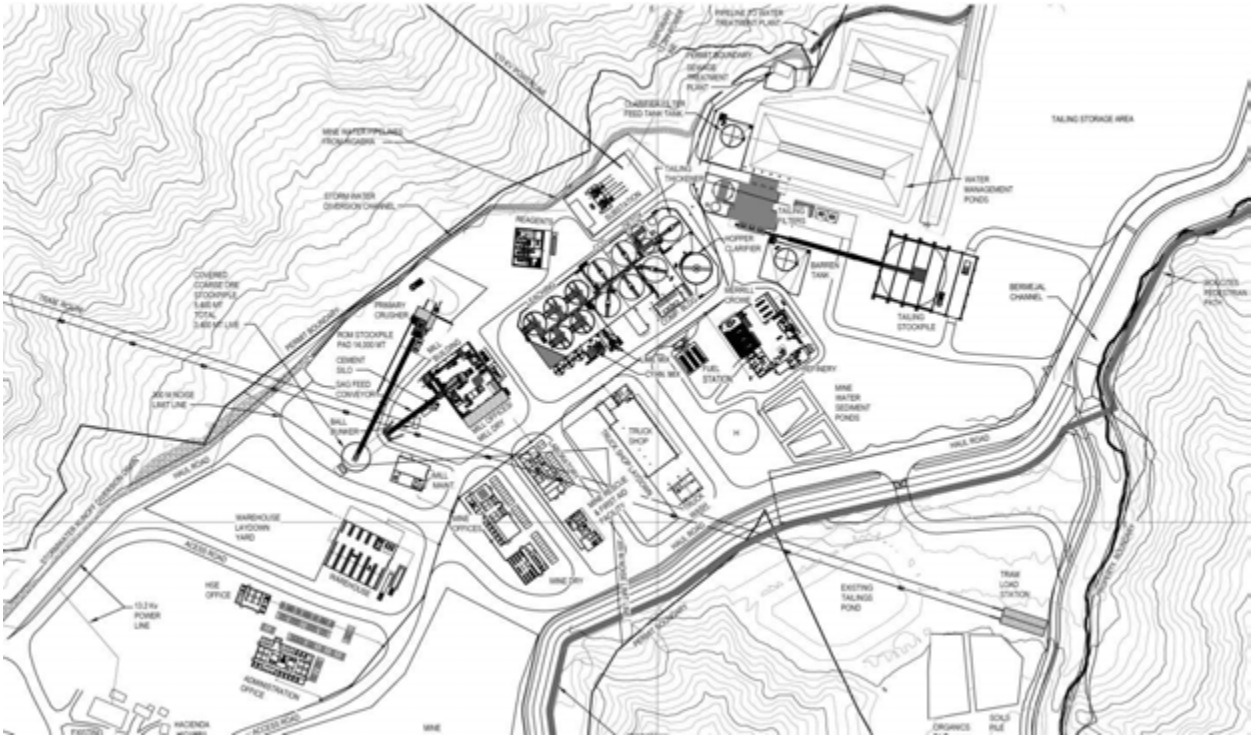
18.3.1 Process Plant

The primary process facilities include:

- ROM Stockpile pad of 14,000 Mt capacity;
- Primary crusher installation housed in structural steel, metal clad building;
- Coarse ore stockpile and reclaim, covered with concrete dome;
- Grinding and gravity area in a structural steel, metal clad building;
- Leaching and CCD tanks and thickeners on concrete pad with concrete walled containment;
- Merrill Crowe plant and gold refinery in a structural steel, metal clad building;
- Tailing filters in structural steel, metal clad building; and
- Tailing stockpile of 15,000 Mt capacity in a covered structure.

The process plant layout and facilities are shown in Figure 18.3.

Figure 18.3: Site Plan



Source: M3, 2015

18.3.2 Site Security

Site security facilities will include a main entrance security building, site fencing and guard posts at entrances to the property. The main entrance at the Buriticá road, and at the plant entrance in the valley will be controlled as checkpoints for vehicles and pedestrians entering the site. The entire site will be fenced with chain-link fencing, and any potential access points from local community trails will have provisions for access control.

18.3.3 Administration and Office Buildings

There are two single story buildings, one for general administration, and a second for health, safety and environmental departments. The office space allocated is sufficient for the planned staff and will be optimized in the detail design phase.

18.3.4 Warehouse and Laydown Areas

A steel building will warehouse parts and consumables that need protection from the elements. The building will include light vehicle access doors at each end, and a concrete slab floor for forklifts and pallet rack shelving; two offices with cataloging/receiving area will be included. A concrete receiving pad will be included for offloading highway tractor trailers.

The warehouse will also have a laydown yard for mine and process plant equipment and spare parts storage. This yard is approximately 75 x 100 m, and enclosed with a security fence.

A separate laydown yard is provide for materials, consumables, and other supplies used in the mine, process plant and surface operations (approximately 16,500 ^{m2}).

18.3.5 Process Plant Maintenance

The process plant maintenance will be in a structural steel building (approximately 220 m^2 working area) including a 10 Mt bridge crane. The building will include a tool crib, supervision and planning offices, and personnel facilities.

18.3.6 Assay Lab

The assay laboratory is equipped with the necessary analytical equipment to perform all routine assays for the mine, the process facility, and the environmental departments, as well as metallurgical testing and sample preparation equipment for core and rock samples. The building will be a single story structure equipped with ventilation, dust collectors, temperature and climate controls.

18.3.7 Mine Office and Mine Dry Facilities

The mine office building will house the mine planning and engineering staff, and include a lunch area for staff. It will also be equipped with a large training and orientation room.

A mine dry building has provisions for men's and women's change room showers.

18.3.8 Truck Shop – Vehicle Maintenance

The truck shop complex is designed for repair and maintenance of mobile equipment and light vehicles. A drive-through wash station will be near the truck shop on a separate concrete slab. An upper office area will house the maintenance staff.

The truck shop layout area breakdown is provided in Table 18.1.

Table 18.1: Truck Shop/Wash Bay Floor Areas

Description	Area (m^2)	Comments
Service Bays	900	five equipment bays, one oil change bay, each 6 m wide x 25 m deep
Work Bays	300	one weld bay and one workshop bay
Tool Storage/Consumables	400	Tool crib, tool storage, consumables and lube storage
Wash Bay	136	8 m wide x 17 m deep

Source: JDS, 2016

18.3.9 Mine Explosives Storage

The explosive storage facility will be located at the north end of the Project site on a 50 x 50 m pad. The explosive storage building will be a reinforced concrete constructed facility with a roof that is steel framed and cladded assembly; capacity is 200 t explosive storage. A separate detonator building will have capacity for approximately 60,000 nonels. There will be a concrete wall or earthen berm separating the two buildings.

18.3.10 Paste Backfill Plant

A paste plant for controlled mixing and distribution of backfill to the underground is located near the Veta Sur portal. The plant is sized to process up to 3,000 t/d tailing and will typically operate at 60 to 70% design capacity. The paste plant will include three main components: tailing feeder, paste preparation building, and binder silo. The paste preparation building will have two floors. In the main building, the lower floor will house the paste pumps and hydraulic power packs, while the second floor will house the paste mixer, the control room, and electrical room. The layout of these facilities has been arranged to make best use of the existing topography with a minimum of earthworks.

18.3.11 Shotcrete Mixing Facility

A shotcrete mixing facility will be adjacent to the paste plant at the Veta Sur portal. It consists of a binder silo and cement blower, cement unloading station, aggregate and sand storage, and shotcrete mixer. A single storage silo will supply both the paste plant and shotcrete mixer. A blower will transfer the cement from this silo to a dedicated paste plant silo. Shotcrete will be mixed and delivered to the mine via truck.

18.3.12 Medical Clinic and Mine Rescue Facility

The medical clinic and mine rescue facility will be housed in two separate buildings with emergency vehicle parking in a sheltered breezeway in between. The medical clinic includes provisions for an emergency first aid station, consultation offices, and pharmaceutical storage. The mine rescue section includes provisions for mine rescue equipment storage, a mine rescue training facility, and offices for mine rescue staff and records.

18.3.13 Fuel Storage

Diesel fuel storage capacity will consist of three 50,000 L double walled horizontal fuel tanks inside containment units, mounted on a single concrete pad. A fuel dispensing station will provide for vehicle fueling, and the entire installation will be protected with concrete bollards.

18.3.14 Utilities

18.3.14.1 Sewage Treatment

Sewage will be treated in a standard sanitary waste water treatment plant. The treatment plant includes primary separation basin, two aeration basins, settling basin with skimmer, aerobic sludge digester with aeration, chlorine contact chamber, gas chlorinator and blowers.

18.3.14.2 Fresh and Fire Water

Fresh water is drawn from three shallow wells located at the upper end of the Project site in the Higabra Valley. The fresh water is pumped to the fresh/Fire Water Tank, which will include the fire water reserve. Fresh water is pumped from this tank to the potable water treatment plant and miscellaneous points such as hose stations. The fire water is pumped into the ring main via the fire water pump skid located by the tank.

18.3.14.3 Potable water

Water will be pumped from the fresh and fire water tank to a potable water filter/chlorinator system. The treated water will be stored in potable water storage tank and distributed for use on site.

18.3.14.4 Fire Protection Systems

The Buriticá site facilities will include fire protection in accordance with applicable codes and standards. The fire alarm system will consist of manual pull stations at building exits and audible and visual notification devices throughout the work areas. The firewater distribution will feed from a fire water tank and modularized pump unit. The fire water pump system will include a main pump and jockey pump which are electrically powered, and a diesel-driven standby pump. A fire truck will provide supplemental protection.

All surface mobile equipment will be fitted with fire extinguishers. The fleet of underground mining equipment will also contain fire suppression systems.

All buildings and conveyors will have fire extinguishers and some will have standpipe systems and fire truck connections.

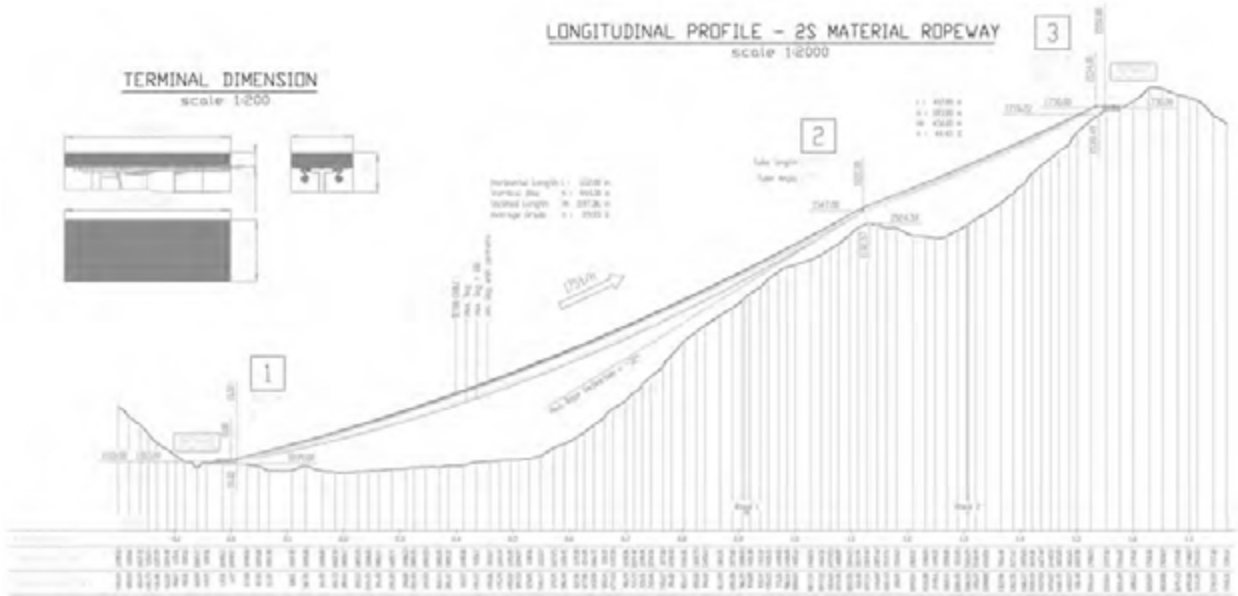
18.3.14.5 Communication systems

Site-wide communications design will incorporate reliable communications systems to ensure that personnel at the mine site have adequate voice, data, and other communication channels available. A number of integrated systems will be provided for on- and off-site communication at Buriticá, including at the processing plant. On-site communications will be facilitated by Voice Over Internet Protocol (VoIP) phones, optical fibre cable network, a wireless network, and a leaky feeder for the underground mines. A trunked radio system consisting of hand-held, mobile and base digital radios will provide wide-area communications coverage VHF radio.

18.3.15 Aerial Tram

An aerial tram system will be used to deliver tailing from the Higabra Valley plant site to the paste plant located at the Veta Sur portal area. Tailing will be delivered to the load station by truck from the tailing stockpile, and will be carried to the unload station in material transport buckets designed for tailing. The buckets will transport tailing at a rate of 175 t/hr at a tram speed of 6 m/s to the paste plant, where they are unloaded directly into the feeder. The tram profile is shown in Figure 18.4, and the plan view alignment is shown in Figure 18.1.

Figure 18.4: Tram Profile



Source: Doppelmayr, 2015

18.4 Power

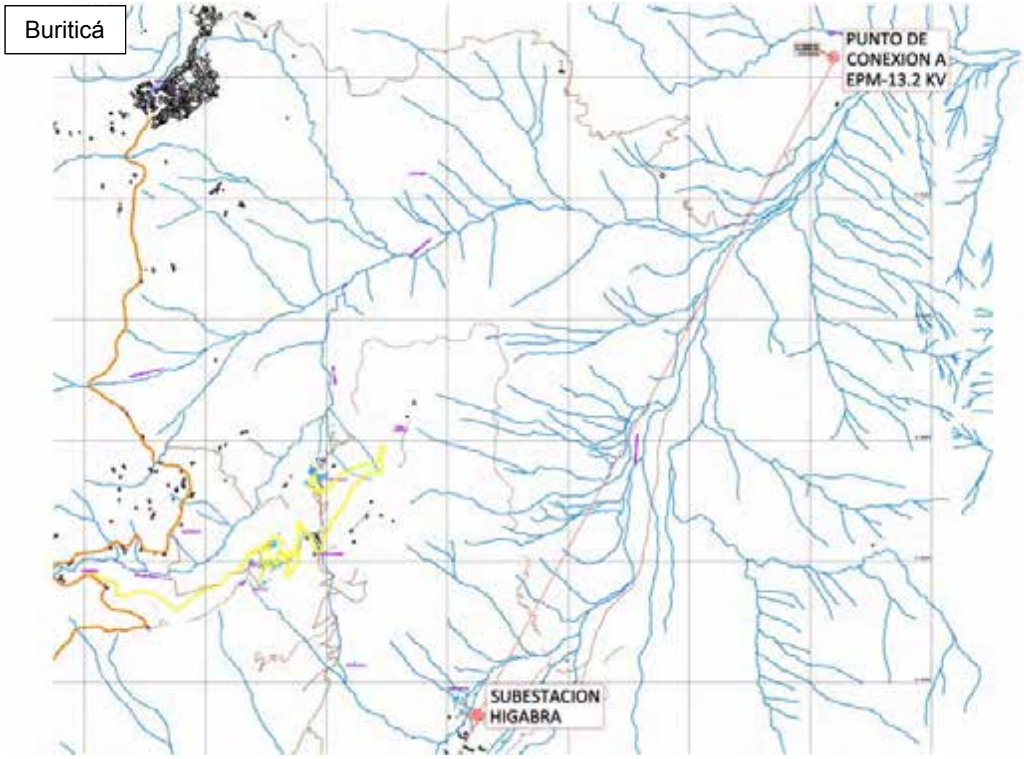
18.4.1 Existing 13.2 kV Power from EPM Grid

The Project is currently supplied power from the EPM 13.2 kV grid, which is connected to the power line from Chorodó and services the Buriticá village. The power available in 2015 was 4 MVA. There is dedicated power of 1.2 MVA for the existing operations which currently has on-site distribution to the Yaraguá portal and mill, Veta Sur and a line running to the Higabra Valley feeding an existing substation near the Higabra Valley portal. This substation steps down the power from 13.5 kV to 480 V for the Higabra portal supply.

18.4.2 Temporary 13.2 kV Power Line for Pre-Production

Another 13.2 kV power line will be built to supply 1 MVA of additional power for pre-production mining, and will connect to the Higabra Valley substation from the EPM grid at Liborina. The preliminary design has been complete for a 3.5 km route up the Higabra Valley. The main connection and termination points are shown on the map in Figure 18.5.

Figure 18.5: Map of Connection Points for 13.2 kV Temporary Powerline



Source: CGI, 2015

18.4.3 110 kV Power Supply

Power will be supplied for the life of mine through a 33 km 110 kV overhead transmission line connected to the EPM grid. It will start at the existing EPM Chorodó electrical substation and end in the Higabra Valley connecting to a new Project substation.

Engineering has been developed by HMV Ingenieros Ltda. (Colombia), leaders in project development of overhead transmission lines in Colombia, Peru, Chile and Brazil. HMV has completed a conceptual design of the connection to the EPM substation and the transmission line to site. Approval has been received from EPM for this design.

The EIA for the 110 kV overhead transmission line has been developed and submitted for review and approval to the Corporation for the Sustainable Development of Urabá (“CORPOURABA”).

Single line diagrams of the main site substation and internal power distribution have been generated by M3 for estimate purposes for the FS.

18.4.3.1 Connection Study

A connection study by HMV has been conducted to supply power to the Project. They analyzed connection alternatives taking into account the EPM owned electrical power grid and substations. The factors considered for the study were as follows:

- Generation and transmission expansion plans of the National Interconnected System (SIN);
- Projected domestic demand;
- Electrical analysis for normal steady state operating conditions and contingency;
- Short-circuit analysis;
- Economic evaluation of the selected alternatives, including the benefit/cost of each relationship;
- Quality indicators, evaluation and operational capacity of the system which provides the network performance with the new associated facilities to the Buriticá Project; and
- The connection to the Chorodó Substation utilizing 110 kV, at a distance of approximately 33 km northwest of the Project, was approved by EPM as the best technical-economical alternative.

18.4.3.2 Chorodó Substation Design

The existing Chorodó substation requires construction of three additional bays to facilitate the connection to the busbar. The design requires the installation of the following equipment:

- Voltage transformer;
- Lightning arrestor;
- Disconnect switch with ground connection;
- Current transformer;
- Three phase circuit breaker;
- Disconnect switch; and
- Control and protection cabinet.

18.4.3.3 110 kV Overhead Transmission Line Design

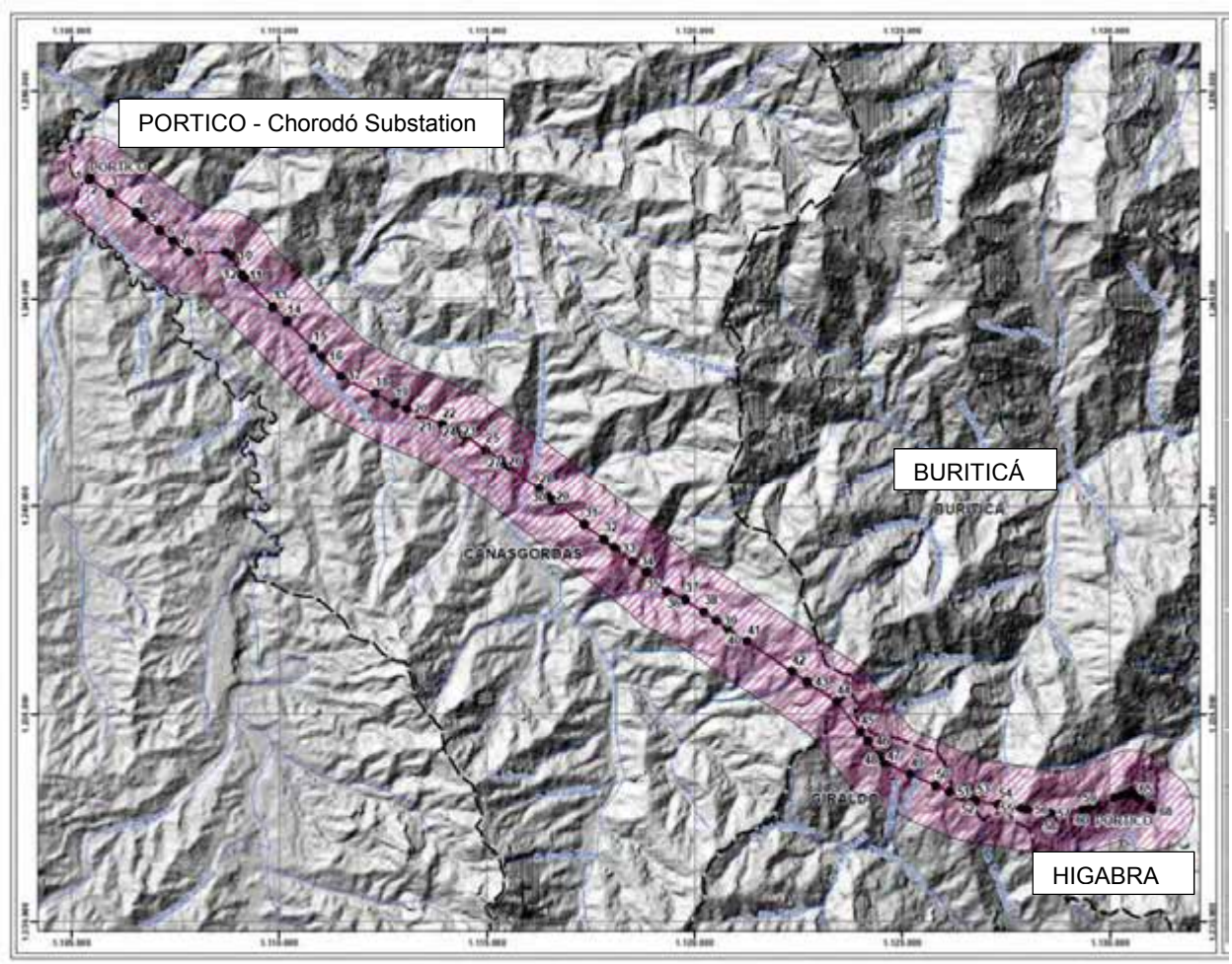
HMV completed a preliminary design of the 110 kV transmission route to site in March 2015. The design followed a route previously chosen based on a study of three route alternatives taking into account environmental considerations, property taxes, technical, and economic aspects. The following criteria were used to determine the Buriticá Project route and design:

- Low environmental impact, to facilitate environmental authority license approval;
- Low environmental cost, to reduce execution time and cost;
- Avoid population areas;
- Tower site accessibility;
- Minimized number of vertices, to facilitate the construction and reduce costs;
- Constructability, to reduce execution time and cost; and

- Transmission line length.

The selected route is shown in Figure 18.6.

Figure 18.6: 110 kV Powerline Route



Source: HMV Ingenieros, 2015

The route from the connection point to the mine site goes through the municipalities of Cañasgordas, Buriticá, and Giraldo in the Department of Antioquia. The total line length is approximately 31km.

The following engineering activities have been conducted for the design:

- Topographic surveys through an airborne LiDAR with a Global Positioning System (GPS) and Inertial Measurement Unit (IMU);
- Land surface mapping;
- Conductor wire size and type selection;
- Tower location selection at suitable topographic sites;

- Geotechnical Study for Tower Foundation Design;
- Shielded design using local lightning density criteria; and
- Technical specifications for supplies, construction and erection, work quantities and project budget.

The design itself consists of 68 towers along the route:

- Twenty-seven (27) suspension towers lattice type A, galvanized;
- Thirty-six (36) suspension towers lattice type B, galvanized;
- Five (5) retention towers lattice type C, galvanized; and
- Two (2) gantries.

18.4.3.4 110 kV Line Permitting Status

Due to it being a 110 kV overhead transmission line, the EIA was submitted to the regional autonomous corporation. The transmission line route crosses two regional autonomous corporations. Because 87% of the overhead line area of influence is in the Corporation for the Sustainable Development of Urabá ("CORPOURABA"), and 13% is in the Regional Autonomous Corporation of Central Antioquia ("CORANTOQUIA"), ANLA (National Authority for Environmental Licenses – Ministry of Environment) has determined that CORPOURABA will be one responsible for Continental's licensing process. The EIA for the 110 kV overhead transmission line is being developed and is intended to be presented to CORPOURABA in 2016.

Since 2013, other approvals (such as no impacts to archaeological sites or ethnic communities), and approval to investigate impacts on wildlife biodiversity, have been received. There are no identified concerns with the EIA approval.

18.4.4 On-Site Power Distribution

On-site power will be distributed from a 56 MW rated substation, connected to the 110kV transmission line. The substation will have two 26.6 MVA transformers, one operating and one on standby, stepping down the 110 kV to 13.8 kV. Power from the substation will be distributed on site primarily at 13.8 kV on overhead lines to the site facilities and mine portals. Smaller secondary substation transformers will be located at the distribution endpoints to provide power at 4160, 600, or 480 V, depending on the service requirements.

18.4.5 Emergency Back-Up Power

A back-up 3 MW generator will be connected to the on-site power distribution to provide emergency power during line power outages. The emergency power is designed to be sufficient to run mine ventilation, provide emergency lighting, and operate slurry agitators to maintain agitation in process plant tanks.

18.5 Water Management Plan

18.5.1 Water Management Overview and Strategy

This section describes the infrastructure planned for the water conveyance, storage and treatment. The water management plan is described in Section 20.4.2.

18.5.1.1 Water and Load Balance

A water and load balance model was developed using a GoldSim model to optimize the water management strategy and evaluate water treatment needs during pre-production, operations, closure and post-closure in order to meet water quality guidelines. The water and load balance model is based on mass balance principles, available hydrology inputs, water management plans, the mine schedule, and the best available water chemistry inputs. The GoldSim model was developed by Montgomery and Associates, and peer reviewed by Tierra Group International.

The water management infrastructure designs prepared for this study are used in the water balance, and therefore, have been validated as suitable.

18.5.2 Water Management Infrastructure

The water management infrastructure consists of the structures designed to handle contact water for delivery to the water treatment plant (WTP), and to non-contact water diverted around the facilities. All water from the mine goes to the WTP surge pond; however, there are two different types of mine water (see Section 16). The sediment laden mine water goes to sediment settling ponds prior to the WTP surge pond, and the clear mine water, piped directly from underground drain hole sources, goes to the WTP surge pond.

18.5.2.1 Contact Water Management Infrastructure

The contact water handling structures are designed to prevent mine and contact water from discharge to the natural environment, and to delivery for treatment to meet Colombian discharge standards prior to discharge. The contact water infrastructure includes:

- Pipeline from Higabra portal to sediment settling ponds;
- Mine water sediment settling ponds, one operating and one standby;
- Pipeline from mine water settling ponds to WTP surge pond;
- Pipeline from Higabra portal to the WTP surge pond for clear mine water;
- WTP surge pond; and
- Water Treatment Plant.

The contact water ponds are lined with an HDPE layer in contact with the water, and secondary HDPE and geotextile liners underneath the top layer. All ponds include a concrete ramp for clean out using surface equipment.

18.5.2.2 Non-Contact Water Management Infrastructure

The non-contact water structures are designed to prevent natural water, including runoff and valley stream flows, from coming into contact with the facilities (e.g., ore, waste and mineral processing plant).

The non-contact water infrastructure includes:

- Plant site surface ditches and diversion structures to direct water into the solids settling pond;
- Site rainwater settling pond;
- Environmental control pond;
- Diversion channel #1 to divert water from the North side of the valley around the plant site;
- El Sauzal and El Naranjo diversions channels around the plant site;
- West side channel; and
- Bermejal channel.

The natural streams in the valley are all diverted in channels around the facilities. Each channel is designed for a 100-year event. The channels are shown in Figure 18.1, and are listed in Table 18.2.

Table 18.2: Diversion Channels

Name	Length (m)	Construction
Diversion Channel #1	400	Concrete ditch
El Sauzal	400	Hydrotex articulated block lining – concrete filled bags
El Naranjo	170	Hydrotex articulated block lining – concrete filled bags
El Sauzal / El Naranjo	610	Hydrotex articulated block lining – concrete filled bags
Bermejal Channel	2,310	Natural alluvial – with topsoil stripped off
West Side Channel	1,620	Natural alluvial – with topsoil stripped off

Source: JDS, 2016

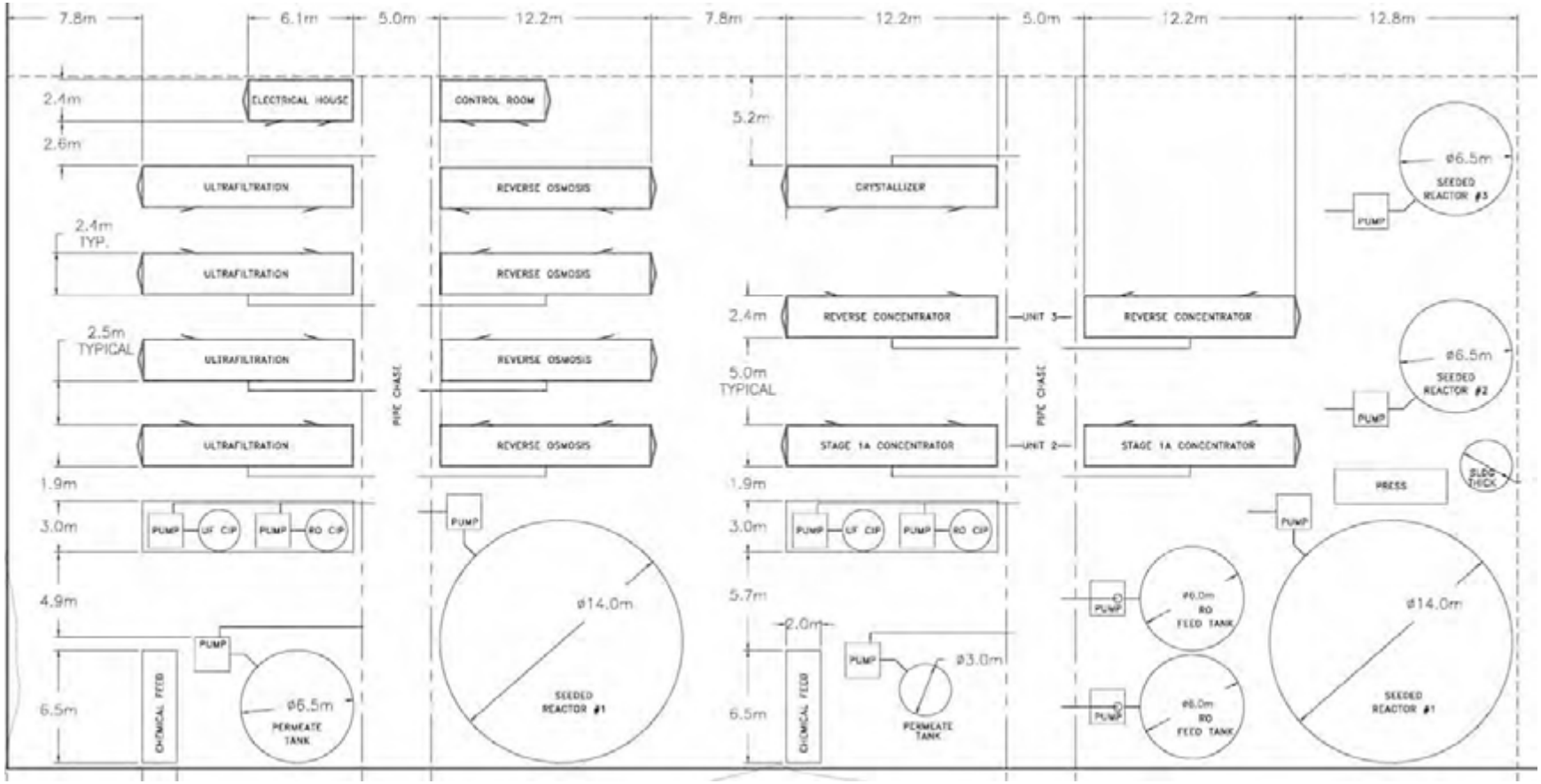
The non-contact rainwater is collected in ditches and directed to the sediment settling pond, which overflows into the environment control pond, before releasing into the diversion channel #1. The environmental control pond will have a sampling location for testing prior to release into the local stream. Depending on quality, water from this pond will be designed to go to the WTP if necessary.

18.5.2.3 Water Treatment Plant

The water treatment plant is designed using membrane technology. The majority of the equipment will be assembled and housed into 40-ft shipping containers. The initial plant installation includes (See Figure 18.7 for layout).

- 3 +1 Reserve Ultra-filtration (UF) systems, with Back-wash, pre-filtration and clean-in-place (CIP) skids;
- 3 + 1 Reserve Reverse Osmosis (RO) systems, CIP, Pre-filter and Transfer Pump Skids;
- 1 HDS Clarifier Package System with Filter Press Wesco or equivalent;
- 1 UF post Clarifier;
- 1 +1 Reserve High Pressure Concentrator RO;
- Membrane skids individually pre-piped and pre-wired; and
- 1 Brine Concentrator.

Figure 18.7: Water Treatment Plant Layout



Source: MDS 2016

Membrane technology is an absolute filter for most multivalent heavy metals, and removal rates > 99% can be achieved. For this reason, and based on treatment requirements, this method was selected rather than conventional lime precipitation.

The system will separate the water into two streams; 75% of the flow is clean water suitable for discharge, and 25% of the flow is a concentrate with the heavy metals. This concentrate will be further processed and filtered down to a brine concentrate. The plant includes a crystallizer system that will then evaporate the brine concentrate to produce a solid for disposal.

Because the membrane trains are containerized and utilize a modular design, water plant capacity can be expanded as the mine discharge volume changes.

18.6 Tailing Management Facility

The TSF will be constructed, operated and reclaimed in phases as mining proceeds. Initially, before the start of tailing production, part of the ultimate TSF footprint will be used for construction laydown and ore stockpile (lined as required). Pre-mining waste rock will be used to construct the truck loading area and first TSF operating cell. While the first cell is being filled, a second cell will be constructed. This sequential phasing of cell construction and tailing deposition will continue throughout the mine life.

Once the first cell is filled with tailing, it will be reclaimed and a cover constructed to provide progressive TSF reclamation throughout the mine life. The progressive reclamation will reduce the contact water volumes even as the TSF is enlarged.

The geochemical characterization of the tailing material and waste rock are covered in Section 20.4.1.2.

18.6.1 Geotechnical Characterization

18.6.1.1 TSF Site Geotechnical

The site geotechnical conditions are described in section 18.2. Existing conditions will provide a suitable TSF foundation with some acceptable static settlement during and concurrent with tailing placement.

18.6.1.2 Tailing Material Geotechnical

The TSF is designed to receive a range of tailing composition, gradations, and moisture content. The tailing will generally be slightly clayey to silty, fine sand to silt with 80 percent finer than 75 microns. The tailing moisture content delivered from the filter press to the TSF is designed to be 14 percent. The actual moisture content will vary depending on the tailing gradation and the filter press operation.

18.6.1.3 Mine Waste Rock Geotechnical

For the design, it has been assumed that the rock has zero cohesion and a friction angle of 37º. The compacted density is anticipated to be between 18 and 20 kN/m^3.

18.6.1.4 Site Clearing Materials

During site clearing and TSF foundation preparation, materials will be removed, processed, and stockpiled. The very thin layer of roots and topsoil will be separated and stockpiled for use as the cover material, to the extent possible. Excavated foundation will be separated and screened into sizes, including boulders for rip rap, gravels for drains, sand and silt for filters, and clay for bedding. No gradation curves for the in-situ material are available, but visual observations indicate that sufficient material quantities will be available from screening.

Generally, the materials are good quality and suitable for TSF construction.

18.6.2 TSF Layout and Cross Section

18.6.2.1 Compliance with Standards and Good Practice

The TSF is designed and will be constructed, operated, and closed to comply with Colombian, and meet or exceed international standards.

Good practice for tailing management includes:

- Eliminate surface water from the impoundment;
- Promote unsaturated condition in the tailing with drainage provisions; and
- Achieve dilatant (density) conditions throughout the tailing deposit by adequate moisture control, placement and compaction.

The TSF design meets these good practice principles. The TSF includes compacted rockfill embankments, compacted filter-pressed tailing, and strong foundation soils that are not liquefiable. To promote chemical stability, the TSF includes basal liners, drains, and a compacted cover that will minimize infiltration.

18.6.2.2 Location

The TSF is located in the Higabra Valley downstream of the mine's primary underground access portal and the process plant (see Figure 18.1).

18.6.2.3 Production and Mine Life

In the period before full production, approximately 315,000 m^3 waste rock from the mine development will be hauled to surface. This rock will be used to construct the working decks and the initial perimeter berms of Cell 1. In subsequent years the annual volume of waste rock will decline from about 200,000 m^3 in year one to less than about 100,000 m^3 by year seven. This waste rock will be used for perimeter berm construction of subsequent cells. The estimated life of mine waste rock volume is 1.4 Mm^3. Any waste rock volume deficit will be supplemented with alluvial material excavated within the TSF footprint.

The volume of tailing deposited at the TSF will increase from about 300,000 m^3 per year to a maximum of about 420,000 m^3 per year. The estimated life of mine tailing volume for surface storage is 4.9 Mm^3. The remainder of the tailing will be placed underground as paste fill.

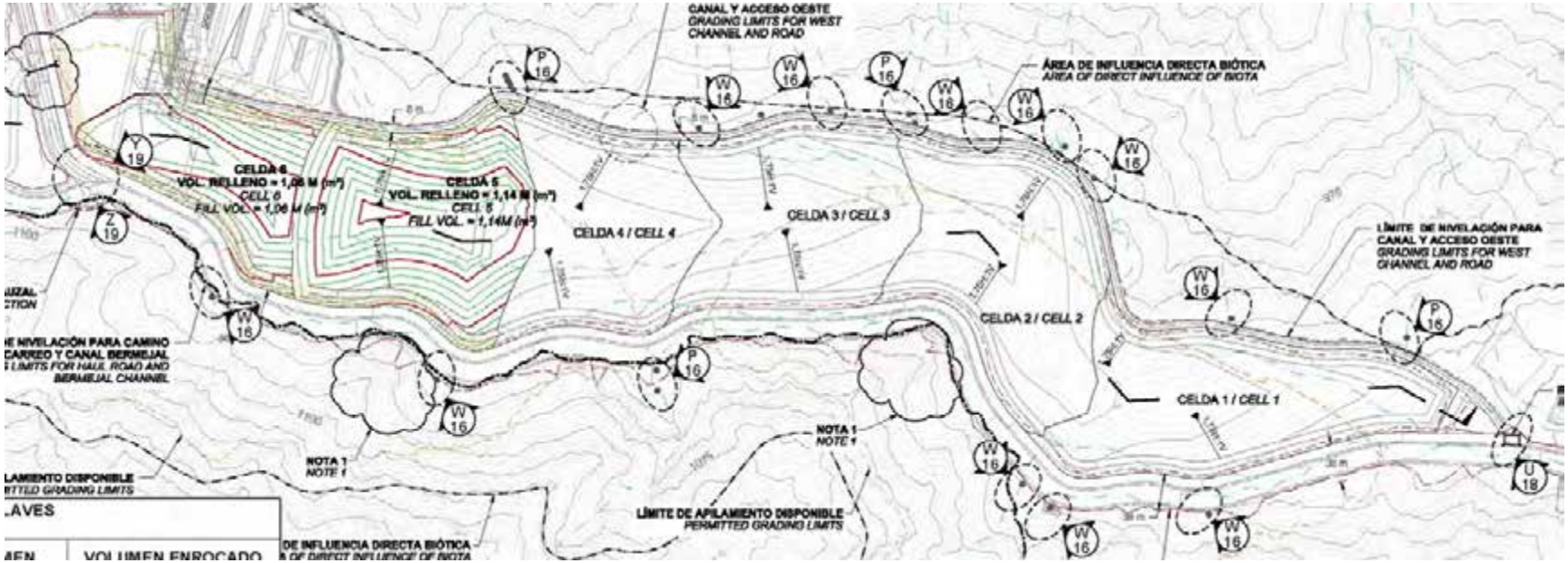
On the basis of these estimated volumes, the TSF required capacity is approximately 6.5 Mm^3.

18.6.2.4 Facility Layout and Design

The TSF layout is shown in Figure 18.8. Key design considerations:

- Maximum use of the central valley and maintaining the existing Eastern Bermejal natural drainage course;
- Avoids tailing placement against or immediately below the steep native hillside slopes;
- Least contributory area for surface water run-on to the tailing - only direct precipitation falling on an active cell becomes contact water;
- No requirement to excavate or blast up-slope diversion channels into the steep hillsides above the TSF fills;
- Optimum surface water runoff management from the valley and the tributary drainages; in essence, such waters continue to flow un-intercepted in natural channels between the hillsides and the facility.

Figure 18.8: Detailed Layout of the TSF



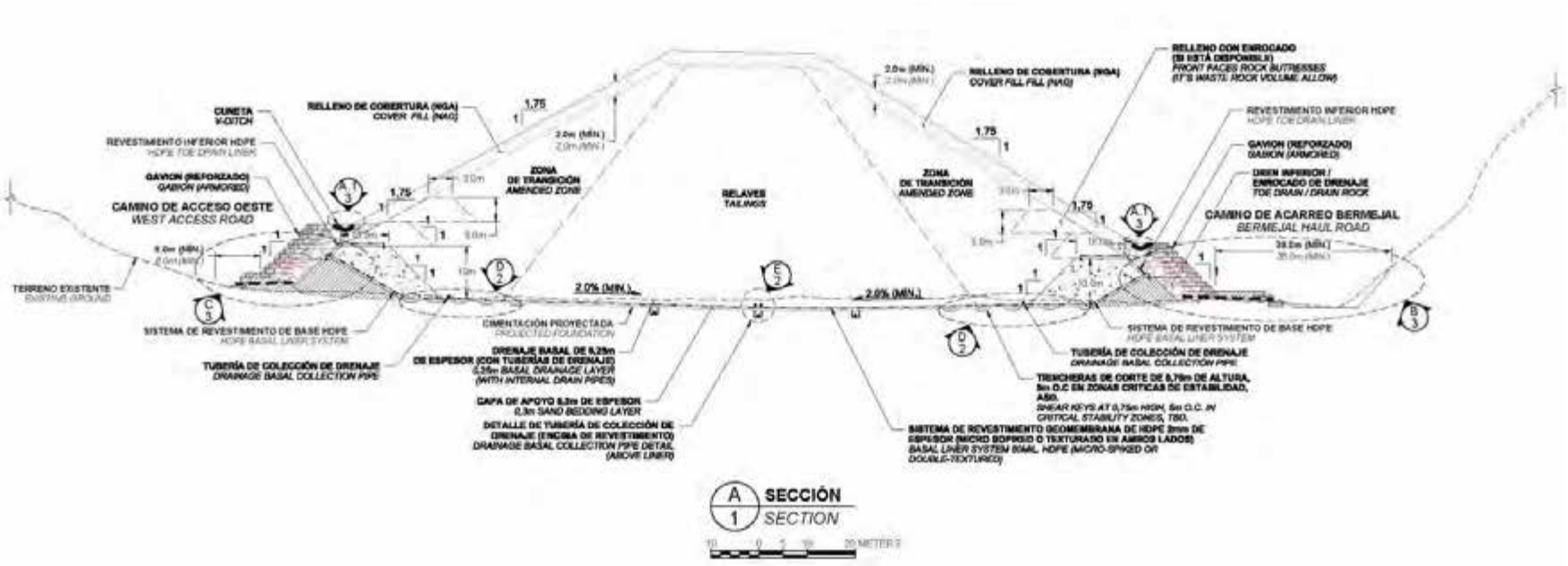
Source: RGC, 2015

18.6.2.5 TSF Cross Section

The main TSF components include (see Figure 18.9):

- Foundations;
- Subsurface drains – to prevent groundwater from affecting the foundations;
- Basal liner – HDPE liner to isolate the tailing material from the natural ground;
- Basal drain – to water treatment plant;
- Perimeter buttresses – rock fill berms and gabion baskets;
- Subsequent perimeter berms – waste rock;
- Tailing – inner zone is tailing with no binder; outer zones amended with binder as required;
- Rockfill interim layers; and
- Inclined drainage layer.

Figure 18.9: Typical Cross Section of the TSF Looking North



Source: RGC, 2015

18.6.3 Phased Construction and Operation

18.6.3.1 Cell Numbers

The construction, operation, and closure of the TSF involves sequential construction from North to South, such that reclamation of each cell can follow closely behind active deposition.

18.6.4 TSF Support Facilities

Preparation and delivery of tailing to the TSF is done via the following:

- Dewatered tailing conveyed from the filter press plant to the filtered tailing stockpile;
- Tailing loaded at the stockpile using a front-end loader into trucks for transport to the active placement cell deck area;
- The east access road used to transport tailing from the stockpile to the placement area; and
- A west maintenance access road will provide access along the west perimeter channel/western TSF toe.

18.6.5 Surface Water Management

The layout includes the features for surface water management for the TSF:

- On the east side of the TSF, a wide corridor is used for the main (east) Bermejal channel and access road;
- On the west side of the TSF, a narrower corridor is used for the west drainage channel; and
- Engineered (rock check dam) inlets and energy dissipation structures to accommodate tributary stream confluences with the main channels.

18.6.6 TSF Contact Water Management

The TSF design includes the following features for contact water management:

- A basal liner consisting of a composite geosynthetic clay layer (GCL) sandwiched between two textured (double-textured or micro-spiked) HDPE geomembranes to prevent seepage of drainage from the tailing and waste rock to groundwater;
- Over the basal liner there will be a drainage layer that collects seepage from the tailing and waste rock and directs it to collection points at the perimeter toe;
- A sediment pond will be included in the detailed design phase to collect rainwater contacting exposed tailing and waste rock during operations; and
- Piping from the collection points and the sediment pond will direct the contact water to the WTP surge pond.

19 Market Studies and Contracts

19.1 Market Studies

Market studies for gold doré sales included obtaining indicative refining and payable terms from a leading-industry entity. CGI currently has contractual arrangements for shipping and refining doré produced at the Yaraguá pilot facility.

Table 19.1 shows the terms used in the economic analysis.

Table 19.1: NSR Assumptions Used in the Economic Analysis

Assumptions	Unit	Value Au	Value Ag
Payable	%	99.9	99.7
Refining Charge	US$/oz	$0.83	
Insurance	% of payable value	0.006	
Transport Cost	US$/oz	$0.62	

Source: JDS, 2016

19.2 Royalties

Royalty payments are calculated at 3.20% of the recovered metal value to doré; the life of mine royalty payments are estimated to be US$137M.

19.3 Metal Prices

The precious metal markets are highly liquid and benefit from terminal markets around the world (e.g., London, New York, Tokyo, and Hong Kong). Historical gold prices are shown in Figure 19.1 and indicate the change in metal prices from 2000 to 2016. Historical silver prices and average COP:US$ exchange rates are shown in Figures 19.2, and 19.3, respectively.

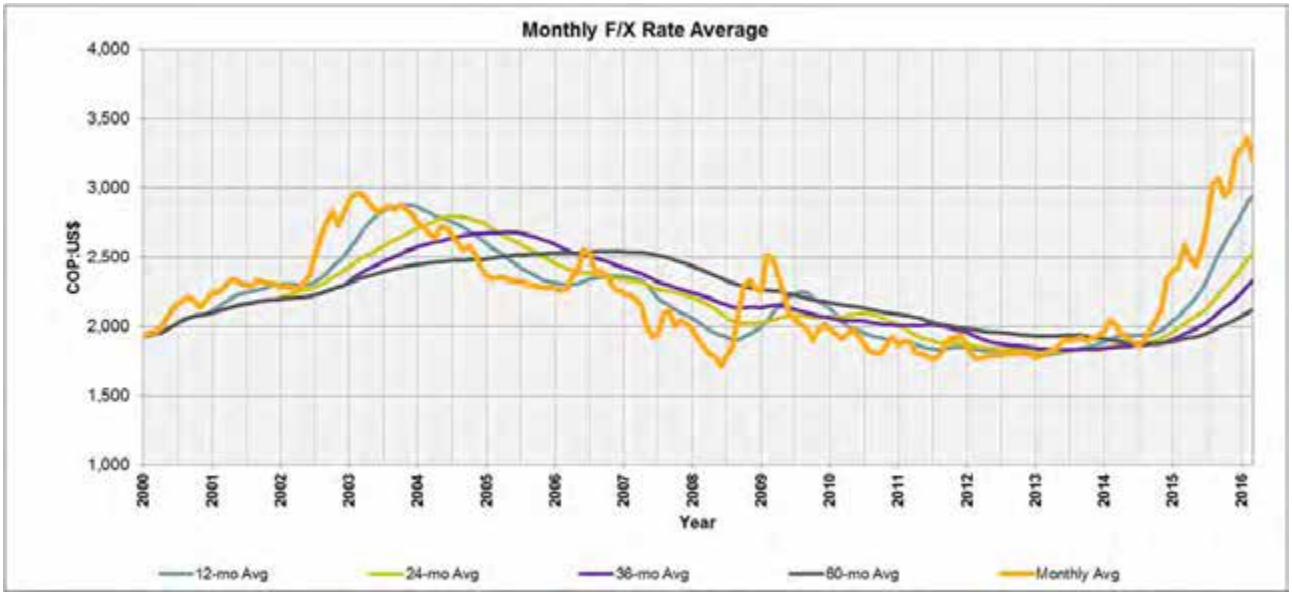
Figure 19.1: Gold Price History

Source: JDS, 2016

Figure 19.2: Silver Price History

Source: JDS, 2016

Figure 19.3: Monthly Average COP:US$ Exchange Rate



Source: JDS, 2016

The silver and gold prices used in the economic analysis are based on the lesser of spot price for January 2016 and the 6-month average sourced from Kitco Metals Inc. The COP:US$ exchange rate estimate used in the economic analysis is based on the January 2016 6-month average price sourced from the International Monetary Fund and recognizing the current sustained lows being experienced against the US$. A sensitivity analysis to metal prices and exchange rate was completed as part of the overall economic analysis; the results of this are discussed in Section 23. Table 19.2 shows the metal price and exchange rate used in the economic analysis.

Table 19.2: Metal Price and Exchange Rate used in the Economic Analysis

Assumptions	Unit	Value
Au Price	US$/oz	1,200
Ag Price	US$/oz	15.00
Exchange Rate	COP:US$	2,850

Source: JDS, 2016

20 Environmental Studies, Permitting and Social or Community Impact

20.1 Environmental Studies and Background Information

CGI operates the Yaraguá mine, located in the municipality of Buriticá and operating in the same area as the proposed Buriticá Project. The Yaraguá mine includes a small underground mine and 35 t/d mill. The initial environmental licensing of the operation was modified into a global environmental license granted by Regional Autonomous Corporation of Central Antioquia (Corantioquia) Resolution No. 130 HX-1208-5963 dated August 28, 2012, for the Comprehensive Mining Concession Contract No. 7495 dated September 14, 2011, registered with the National Mining Agency on March 20, 2013. The environmental license has been partially amended by means of Administrative Resolutions No. 130 HX-1311-6885 and 130 HX-1311-6886 dated November 13, 2013. Global environmental licenses involve the use of all renewable natural resources, such as water concessions and discharge permits, without need for separate filing.

Environmental baseline studies began in 2010 and hydrogeological studies in 2011and 2012 (phases I, II, and III). In August 2012, Corantioquia granted a modification to the original environmental license for underground development and road construction at the Buriticá Project, and work began in Q4 2012. Between October 2012 and December 2013, CGI completed an Environmental Impact Assessment (Estudio de Impacto Ambiental - EIA) in accordance with Terms of Reference issued by Corantioquia. In the EIA, CGI requested Corantioquia to modify the existing environmental license to include the new mining development and construction of a 2,000 t/d mill, along with remediation, reclamation, and closure.

The EIA document was submitted to Corantioquia on December 23, 2013 by means of a legal registration filing No. 130 HX-1312-1902. As part of the permitting process being carried out by Corantioquia, the environmental authority issued Administrative Act No 160 HX-1501-7551 dated January 27, 2015 in which additional study information was requested. CGI provided the requested information on March 27, 2015.

On September 15, 2015, CGI announced that it had requested the National Government of Colombia to assume the responsibility of reviewing the application for modification to the EIA for the Buriticá Project as a PINES Project, as contemplated under Colombian law, for which the company withdrew its application for the modification of the EIA from Corantioquia.

On November 20, 2015, the National Government of Colombia published Decree 2220, regulating in detail Article 51 of Law 1753, which was passed by Congress on June 9, 2015. Decree 2220 specifically applies to the Buriticá Project, which is classified as a Project of National Strategic Interest (PINES) Project. Law 1753 defines the National Development Plan for Colombia from 2014 to 2018, and Article 51 established that the National Environment Authority (ANLA) was the competent authority to review and approve environmental applications for PINES projects. Subsequently, and under provisions of Decree 2220, CGI filed a new EIA with ANLA on January 20, 2016.

On February 10, 2016, CGI announced that the Constitutional Court of Colombia issued a press release announcing that certain aspects of the National Development Plan (Law 1753) passed by Congress in July 2015, including Article 51, are considered unconstitutional and that they intended to issue definitive rulings in this regard in due time. The Court indicated that ANLA (the National Environmental Agency) would not have sole exclusivity over permitting and maintaining environmental aspects of PINES projects. Shortly thereafter, the Court published its ruling, deeming the PINES program constitutional and removing ANLA's authority with respect to having sole exclusivity over permitting by determining that environmental licensing of projects would be subject to existing regulations in force, without reference to PINES guidelines. As a result, PINES projects will be under the authority of either ANLA regional autonomous corporations, depending on compliance with certain specifications currently in force in Decree 1076-2015. In any event, ANLA maintains continued involvement and oversight of PINES projects whether or not the Project falls under review by a regional autonomous corporation.

20.2 Main Project Area Environmental Characteristics

20.2.1 Climate and Meteorology

Annual precipitation in the Buriticá Project area ranges from 1,050 mm/year to 1,450 mm/year. Precipitation in the project area shows bimodal seasonality, with maximums in May (173 mm) and October (197 mm). January and February show the least amount of rainfall, and months with no rainfall are infrequent. Maximum 24-hour precipitation is highly variable ranging from 80 mm/day to 150 mm/day for a 100-year return period. Annual average daily temperatures range from 21°C to 18°C.

20.2.2 Hydrology

There are two principle drainage basins: 1) Quebrada La Mina (approximately 5.8 km^2); and 2) Quebrada Bermejal (approximately 12.3 km^2). These streams combine to form Quebrada La Tesorero (area 18.3 km^2), which discharges to the Cauca River.

20.2.3 Water Quality

20.2.3.1 Surface Water

Data have been collected for baseline water quality in both the dry and wet seasons. In areas where other third party mining activities take place, which is in the upper section of Quebrada La Mina and more recently Quebrada El Sauzal, the majority of the evaluated water quality parameters exceed Colombian surface water discharge limits presumably because said third party mining activities may not be employing adequate environmental control measure.

Unfiltered water samples taken by the company in two locations along Quebrada El Sauzal show elevated concentrations of mercury, likely from third party gold processing activities upstream of the Project. Water samples for dissolved metal concentrations were taken in October 2015 at the same two locations and both results returned mercury concentrations below US EPA drinking water standards for mercury. These results indicate that the elevated levels are from metallic mercury in the sediments. Enhanced baseline sampling of Higabra soils and water quality is warranted prior to construction so the extent of contamination can be quantified and managed if required.

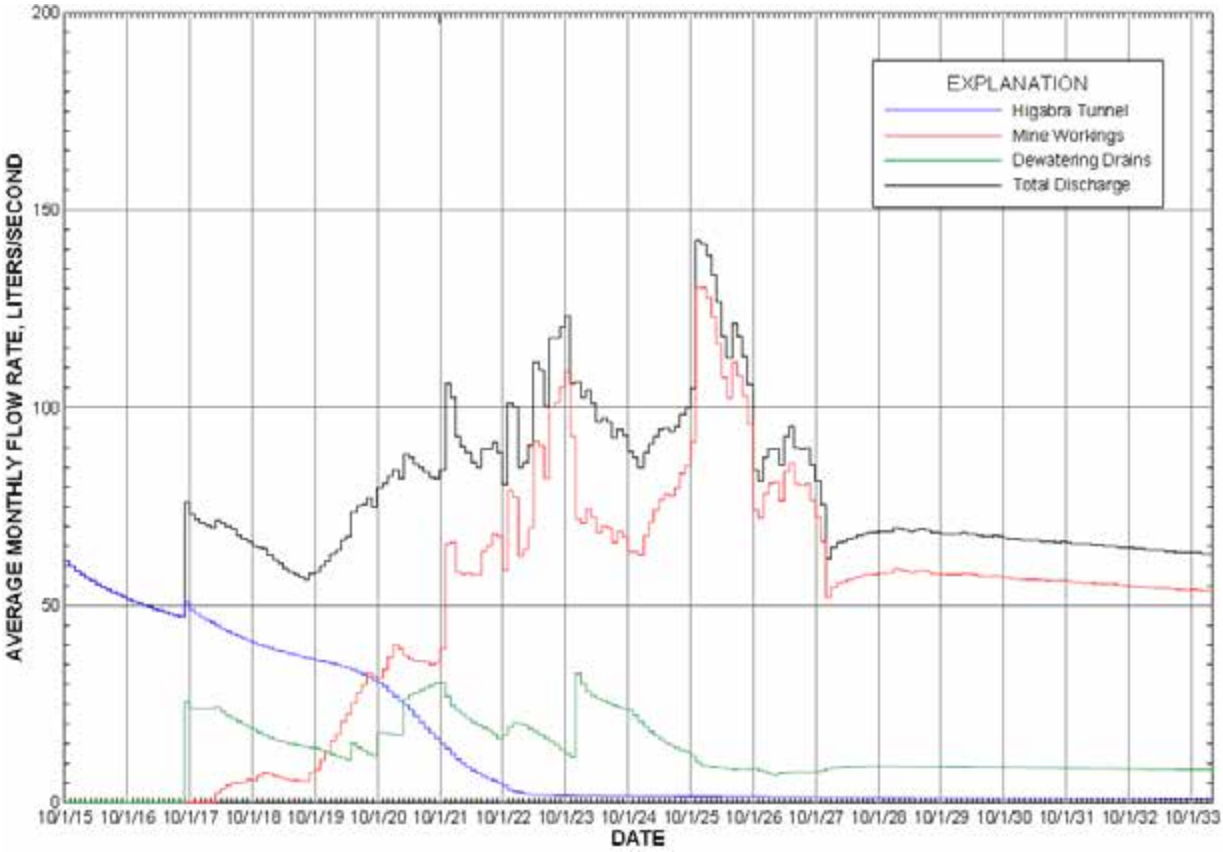
In the Quebrada Bermejal and Quebrada Tesorero, the majority of water quality test results meet surface water discharge standards. Surface water quality limits became more stringent with implementation of Colombia Resolution 631, 2015 which will be in effect as the Project moves into production. These new standards were used for water management system designs.

20.2.3.2 Underground Mine Water

Montgomery and Associates estimated mine water inflows with a numerical model including regulated flows using valved drains. Monthly flows ranged from a low of 56 l/s to a high of 141 l/s and averaged 88 l/s over the project life as shown in Figure 20.1.

Increased mine flow beginning in October 2021 corresponds to mining at elevations below the Higabra tunnel level (1,150 m elevation).

Figure 20.1: Predicted Average Monthly Mine Inflow (with dewatering)



Source: Montgomery Associates, 2015

Contact water quality for the underground mine was estimated using static and kinetic Humidity Cell Test (HCT) results from Schlumberger Water Services. Geochemical modeling of the chemistry of water passing through the mine openings has been undertaken using estimated rates of groundwater ingress derived from the numerical groundwater model by Montgomery and Associates (December 2015). The modeling work has been used to quantify both the flow rates and potential solute ranges over the mine life. These results were used for water treatment design criteria.

Geochemical water chemistry modeling under active and passive dewatering scenarios was performed using (a) stope wall rock (notably alteration) classification and (b) the early stage of kinetic test results. The modeling results indicate that mine water will remain near neutral pH throughout LOM, with only arsenic and occasionally sulphate potentially exceeding Colombian "Limites Máximos Permisibles" (LMP) thresholds. Kinetic tests results will continue to be monitored in order to verify any other parameters increasing in concentration. It is also reasonable to expect that both NO3 and NH3-N will be present in mine contact water from explosives use. These parameters are not currently subject to regulation within the Colombian LMP framework.

Groundwater currently draining from the mine has naturally elevated levels of chloride and sulphate. Water that is currently collected from drain holes is kept separate from other mine water and does not require treatment. Once mine development begins, all mine water discharges will be collected for treatment to meet Colombian discharge limits.

20.2.4 Hydrogeology

There are four identified hydro-stratigraphic units:

- Relatively high permeability alluvial aquifer unit;
- Highly fractured hard rock with medium to high permeability, associated with intensely fractured and mineralized intrusive and volcanic rocks;
- Fractured rock aquifer with low to medium permeability; and
- Cretaceous meta-sediments with very low permeability.

The Tonusco Fault, immediately east of the deposit, is considered a low permeability feature based on monitoring data and field observations. Aquifer recharge appears directly related to rainwater infiltration, and is controlled by the vertical permeability of the geologic structures.

Alluvium deposits occur primarily as valley fill with limited vertical extent. Based on field observations, alluvial deposits seem to be hydraulically connected to the overlying stream channels. Rock fragments in Quebrada La Bermejal alluvium reach meter sizes; this, in addition to the channel dimensions (25 to 30 m), indicates frequent repeated flooding and deposition of alluvial material. Drill results have established variable thickness; reaching 60 m. Permeabilities range from moderate to high mainly due to the inter-granular primary porosity of the unconsolidated, poorly sorted alluvial materials and their corresponding thicknesses.

The rock units consist of andesitic rocks, basalts (Barroso volcanic complex), tonalites, and meta-sediments. Groundwater flow and storage in these units is predominantly within fractures. As such, these are fracture-controlled systems that are discrete to each rock unit.

20.2.5 Vegetation

Forestry surveys to identify species of concern have been conducted. No anticipated impacts from construction or operations beyond those mitigated by conventional measures, nor sensitive ecosystems have been identified. The Project area vegetation cover is mainly comprised of a mosaic of crops, pastures, and natural spaces (14.91%), riparian forest (40.38%), and other grasslands (44.71%). Special permits for intervention of species classified as vulnerable have been obtained.

20.2.6 Land Use and Social Aspects

The CGI industrial site is located in the Higabra Valley, which gave its name to a rural district (vereda) comprised of a settlement of approximately 275 inhabitants; 65 families of close kinship among themselves and the neighbouring communities. There are certain traditional access rights that must be maintained which do not impact development of the Project. Nevertheless, in the event access rights may overlap with the project development, the company is entitled with legal measures to adequately structure rights-of-way.

The communities in the area where the Buriticá Project is located have traditionally been mestizo farmers devoted to agricultural activities based on coffee and subsistence farming. The Mogotes community, located in the Higabra Valley about 2.7 km downstream of the northern project boundary, is the exception, as it is a community of artisanal miners who have made a living mainly from small-scale mining (barequeo) along the Cauca River beaches and banks.

The situation of this rural district in 2010 was not very promising from an economic and demographic perspective. It was facing demographic deceleration due to young people migrating to Medellín. The families that remained made their living from temporary employment or working in subsistence agriculture on small properties not larger than 5 ha. Farmed produce was mostly shared or exchanged with their neighbouring relatives. Currently the local economy has regained dynamism as a result of initiatives to provide services to project contractors as well as the hiring of residents for work activities being carried out by Continental.

A social concern has been the development of artisanal mining activities, which has impacted the community at all levels because there has been no planning to accommodate these activities. Ongoing problems include environmental contamination as well as associated social problems and criminal activity.

The main areas where artisanal mining has been developed are as follows:

- Mogotes rural district: artisanal miners conducting placer gold extraction on the banks of the Cauca River;
- Higabra rural district: Hardrock artisanal mining in this rural district not characterized as having an artisanal mining tradition; and
- Los Asientos rural district: A primary centre of hardrock artisanal mining growth with a considerable influx of miners.

20.3 Permitting

20.3.1 Environmental License

The permitting process for obtaining environmental licenses for mining projects in Colombia is set out in Law 99, 1993, and is detailed and governed by Decree 1076 of 2015, Section 2, Article 2.2.2.3.2.2, which establishes that the Ministry of Environment and Sustainable Development has the authority for granting or denying an environmental license for the mining of metallic minerals and precious and semi-precious stones for projects involving removal of useful and waste material of 2,000,000 t/y or more. According to said provisions, regional autonomous corporations (CAR) are the authority for granting environmental licenses for projects involving removal of useful and waste material under the aforementioned threshold.

The exception created by Decree 2220 specifically applied to the Buriticá Project, due to it being designated a Project of National Strategic Interest (PINES). On September 15, 2015, CGI announced that it had requested the National Government of Colombia to assume the responsibility of reviewing the application for modification to the EIA for the Buriticá Project as a PINES Project, as contemplated under Colombian law. Consequently, the company withdrew its application for the modification of the EIA from Corantioquia.

On November 20, 2015, the National Government of Colombia published Decree 2220, regulating in detail Article 51 of Law 1753, which was passed by Congress on June 9, 2015. Decree 2220 specifically applies to the Buriticá Project as a PINES Project. Law 1753 defines the National Development Plan for Colombia from 2014 to 2018, and Article 51 established that the National Environment Authority (ANLA) was the competent authority to review and approve environmental applications for PINES projects. Subsequently, and under provisions of the Decree 2220, CGI refiled a new EIA with ANLA on January 20, 2016.

On February 10, 2016, CGI announced that the Constitutional Court of Colombia issued a press release announcing that certain aspects of the National Development Plan (Law 1753) passed by Congress in July 2015 including Article 51, are considered unconstitutional and that they intended to issue definitive rulings in due time. The Court indicated that ANLA (the National Environmental Agency) will not have sole exclusivity over permitting and maintaining environmental aspects of PINES projects. Shortly thereafter, the Court published its ruling, deeming the PINES program constitutional and removing ANLA's authority with respect to having sole exclusivity over permitting by determining that environmental licensing of projects would be subject to existing regulations in force, without reference to PINES guidelines. As a result, PINES Project will be under the authority of either ANLA or regional autonomous corporations, depending on compliance with certain specifications currently in force in Decree 1076-2015. In any event, ANLA maintains continued involvement and oversight of PINES projects whether or not the Project falls under review by a regional autonomous corporation.

20.3.2 Other Environmental Permits

Unless a global environmental license is issued, in addition to the environmental license to operate a mining project, a number of other environmental permits are required. Additional permits required to work on the facilities include permits for water usage and discharge, atmospheric emissions, forestry clearance and land access. For the Buriticá Project, permits for water usage and discharge were initially granted in February 2009 and valid for 10 years. Nevertheless, in August 2012, water permits were validated by Corantioquia for the entire term of the currently approved environmental license, as a global environmental license.

In Colombia, the global environmental license encompasses all the permits, authorizations, and/or concessions for the use and/or exploitation of the renewable natural resources that are necessary over the life of the Buriticá Project, work or activity. The other permits applied for in the EIA are summarized in Table 20.1 below; therefore, they will be granted when the EIA is obtained.

Ordinary solid waste, hazardous waste, and recyclable waste will be managed by third party companies having the necessary environmental permits and licenses for such purposes.

Table 20.1: Other Permits Required

Type	Permit
Water Abstraction Concession*	Surface Water for Domestic and Industrial Use
	Groundwater for Industrial Use
Discharge Authorization*	Domestic Waste Water
	Water from Workshops and Fuel Station
	Mine Water
	Seepage water from filtered tailing storage and waste disposal facilities
Construction of Hydraulic Works	Intervention of the Quebrada Bermejal channel
	Quebrada El Sauzal water intake
	Quebrada El Sauzal diversion channel and drainage bridge
	El Naranjo diversion channel and intermittent drainage bridge
	Intervention of intermittent natural drainage channels
Vegetation	Single Forest Use (Aprovechamiento Forestal Único)
Atmospheric Emissions	Emissions from Refinery-Smelter
	Emissions from Refinery-Gas Purging
	Atmospheric Emissions – construction and assembly phase
	Atmospheric Emissions – operational and beneficiation phase

Note: * = Permits for water abstraction and discharge were granted in February 2009 and are valid for 10 years. In August 2012, water permits were validated by the Antioquia Environmental Authority for the entire term of the Buriticá Project's Environmental License

Source: M3, 2014; JDS 2016

20.4 Environmental Management

The Environmental Management Plan considerations are summarized in Table 20.2.

Table 20.2: Environmental Management Plan

Component	Impact Considerations
Water	Domestic waste water
	Civil works waste water
	Water from mining activities
	Surface water from the process plant and TSF
	Domestic waste water treatment sludge
	Surface runoff water, water from stream channel diversion, and rainwater
	Groundwater from small diameter exploratory drill holes monitoring and sealing
	Groundwater recharge protection
Air and Noise Quality	Air and noise impacts
Domestic and Industrial Waste	Solid waste handling
	Hazardous and special waste handling
Mine Waste	Tailing and waste rock handling
Chemical Substances and Supplies	Fuel, chemical substance and explosive handling
	Cyanide management
Soil	Erosion control and slope stabilization measures
	Surface soil management
Vegetation	Vegetal cover management and removal
	Remnant forest conservation
	Protected areas management
Fauna	Endangered species identification and mitigation
	Terrestrial fauna management (birds, mammals, reptiles, and amphibians)
	Strategies for the conservation of the Leopardus tigrinus species (Oncillia, or little spotted cat)
Social and Cultural	Community information and participation program
	Environmental education program - workers and community
	Labour hiring practices
	Cultural memory and patrimony program
	Third party impacts program
	Social compensation and mitigation program
	Preventive archaeology program
	Controlled Access around the mine site

Source: M3, 2014; JDS modified 2016

20.4.1 Tailing and Waste Rock Disposal Environmental Management

The Tailing Storage Facility (TSF) is planned to store the filtered tail material produced by the process plant and for the waste rock removed from the mine.

To establish the optimum TSF location, a risk assessment including environmental considerations was completed for several alternatives to identify a suitable location, to avoid the most sensitive areas, and to minimize environmental impacts. Locations were constrained due to topography and land ownership.

The TSF has been divided into multiple cells to contain a total of 6.5 Mm^3 of material over the project life; 4.9 Mm^3 of tailing and 1.6 Mm^3 of waste rock supplemented with local alluvial materials. The TSF will be constructed, operated, and reclaimed in phases as mining proceeds. Once the first cell is filled, it will be covered and reclaimed. In this way reclamation will be ongoing throughout the mine life.

Mine waste rock and locally excavated alluvial material are integral to supporting mine production in that they will be used to construct perimeter berms, embankments, and buttresses for the cells. The TSF will be a co-disposal facility for all materials produced and not used for mine backfill. Approximately 34% of the tailing produced in the mill will be used as mine backfill (approximately 4.7 million tonnes).

The TSF will be underlain by a composite geosynthetic clay layer (GCL) sandwiched between two textured (double-textured or micro-spiked) HDPE geomembranes. The liner system will use shear keys to increase geotechnical stability. The facility will incorporate drainage for zones both above and below the basal liner.

Over the basal liner there will be a drainage layer that collects seepage from the tailing and waste rock to the perimeter where the water will be directed to the water treatment facility. A bedding layer will be placed over the drain and liner to protect it from puncture by waste rock or construction activities.

A series of French drains will be installed beneath the facility to eliminate pore pressure buildup in the alluvium beneath the facility. These drains are anticipated to be trenches approximately 300 by 300 mm, lined with geotextile, and gravel filled.

20.4.1.1 Geotechnical

The geotechnical design for the TSF is included in Section 18.6.

20.4.1.2 Geochemical Summary

Based on the results of geochemical modeling performed for the Buriticá TSF, the following key considerations are of relevance to the process of contact water management system design. GoldSim model results provided by Montgomery and Associates indicate that under all modeled scenarios at least 50% of water generated from active TSF cells is tailing contact runoff. The modeling results showed:

- Average volumes of tailing seepage are expected to be very low, except for short periods while tailing material is placed and being compacted. This is due to the very low unsaturated hydraulic conductivity of the tailing when compacted. Outer faces of the TSF containment berms will be lined to their final elevation and runoff will not contact any contained Potentially Acid Generating (PAG) waste rock.
- Runoff from tailing surfaces is expected to be weakly alkaline with solute concentrations compliant with Colombian surface water quality discharge standards.
- Runoff from the soil cover is expected to have near neutral pH and solute concentrations in compliance with Colombian Surface water discharge standards.
- Modeling results of PAG waste rock seepage and runoff indicate that runoff is likely to have alkaline pH of about pH 8.4 and compliant solute concentrations.

It should also be noted that at very low flow rates, It is possible that waste rock runoff and/or seepage flows could have acidic pH and/or non-compliant sulphate and iron concentrations. These flows will be sampled to determine if water treatment will be required. If the results are in compliance, the flows will be discharged. If not, the flows will be treated prior to discharge.

20.4.1.3 Compliance with Standards

The TSF is designed and will be constructed, operated, and closed to comply with Colombian regulations, and meet or exceed international standards (see Section 18.6).

20.4.2 Water Management

The water management plan will be designed to optimize water resource use, which will be minimized through the use of a filtered tailing system and best management practices.

Based on current test information, naturally occurring clear groundwater encountered in the mine will need to be treated to meet Colombian surface water discharge standards for precious metal mines as defined in Colombian Resolution 631, 2015. Once the development phase of mining begins, mine water discharges will be collected for treatment prior to discharge.

Non-contact surface water will be diverted with channels designed and constructed to minimize contact with the facilities. Runoff that will contact ore materials will be collected in a sediment settling pond and outflow water directed to the water treatment plant (WTP).

Domestic water will be supplied from shallow groundwater wells located upstream of the process facility in the El Sauzal drainage. A reverse osmosis water treatment plant will be used to treat the water to make it suitable for domestic potable water usage.

Sewage wastewater will be treated in a plant of modified activated sludge design. A small amount of sludge will be generated, and by virtue of the treatment-method, can be used as fertilizer for TSF revegetation.

20.4.3 Water Treatment Facility

Water for treatment will come from two main sources; groundwater from the mine, and meteoric contact and process water. Over 90% of the water treated will be from the mine with the remainder from the TSF, ore stockpile and process areas.

The WTP design, consisting of containerized modules, allows flexibility for capacity changes. It will be designed to treat dissolved metals, sulphates and chlorides, and expected TDS levels. A membrane treatment system will be used to remove greater than 99% of multivalent heavy metals. Constituents removed from the water will be recovered in a brine solution that will be processed through a crystallizer to produce a solid product. Calcium sulphate will also be recovered from the high density sludge system.

20.4.4 Monitoring Plan

Monitoring plan considerations are summarized below.

Table 20.3: Monitoring Plan

Component	Area Monitored
Water	Waste Water
	Receiving Stream Quality
	Sludge Generated at the Domestic Waste Water Treatment Plant
	Sediment and Fluvial Dynamics
	Groundwater Level and Quality
	Diversion channel Monitoring for Hydrogeology
	Hydrological Surface Water Follow-Up.
Atmospheric Emissions and Noise	Noise and Vibration
	Atmospheric Emission
Waste	Solid Waste
Flora and Vegetation	Vegetal Cover
Fauna	Terrestrial Fauna Management
Social	Programs for Environmental Management Plans for Socio-economics
Facilities	Stabilization of Slopes, Facilities and Mining Areas

Source: M3, 2014; JDS updated 2016

20.5 Social or Community Impact

The Buriticá Project is located in the Higabra Valley, in which resides a settlement of approximately 275 inhabitants and 65 families of close kinship among both themselves and the communities in the neighbouring rural districts. These are farming communities that had been facing demographic deceleration as described in Section 20.2.6.

Informational meetings were held with communities in both the direct and indirect influence areas during EIA preparation. Topics addressed with the direct-influence area communities of Mogotes, El Naranjo, Higabra, Asientos and Buriticá Village included EIA socialization and the Environmental Management Plan.

Residents generally realized the potential of formal job creation in the area as well as the potential improvements for transportation resulting from the Project. Also residents saw the operation as a welcome alternative to certain third party mining, which has been creating environmental and social concerns including criminal activity. Residents had many questions concerning interruption to established community paths and mitigation measures being implemented for the Project.

Ongoing dialog with both residents and Environmental and Municipal representatives will continue, and be part of the social management plan. Key plan components include:

- Community information and participation program;
- Environmental education program for workers and the community;
- Institutional strengthening program;
- Labour hiring program;
- Cultural memory and patrimony program;
- Third party damage payment program;
- Social compensation and mitigation program;
- Preventive archaeology program;
- Physical separation program around the mine site; and
- Mitigation program by mobility interruption.

A concern for residents is illegal mining, which involves primarily people from outside the Buriticá area. As well, residents display environmental and social concerns and unease due to criminal activity and challenges to the respect for the rule-of-law. Another serious concern is lack of safety and a noticeable increase in illegal mining accidents. In 2013, the local government authority (Gobernación) implemented a mine closure plan by first controlling access into the area for those that weren't long-term residents and further closing of illegal operations. Subsequently, the Government of Antioquia, the municipality of Buriticá, the Ministry of Mines and Energy through the National Mining Agency – Mining Formalization Area – and CGI, met and implemented a pilot program, unprecedented at a national level, to legitimize several credible associates in a mining formalization program. A benefit to this initiative was implementation of environmental management plans for each formalized mining area.

In January, 2016, national and regional authorities in Antioquia began another initiative to close illegal mining operations in the area. The National Mining Agency has ordered local authorities, with the participation of CGI as the legal title holder in the area, to formally close down those illegal operations as a preventive measure due to danger from potential internal collapsing of mine openings in areas of illegal activity. A Security Council, led by the Buriticá Mayor and the regional police commander, has issued the corresponding closure orders to remove the miners from several illegal mines in the area.

20.6 Closure

Mine closure requirements are regulated by Decree 1076 of 2015. Article 2.2.2.3.9.2 describes the steps to be taken. In summary they are:

Three months prior to the finalization of exploitation, the company is to provide a study to the environmental authority which addresses the following:

- Identify site environmental impacts at the time of closure;
- Demolition plan;
- Drawings and plans showing the location of infrastructure for closure;
- All obligations to be fulfilled and work to be completed; and
- The closure plan costs including pending compliance items.

The environmental authority has one month to comment on the closure plan.

When the closure plan is initiated, the company must post an insurance bond to cover the closure costs for the closure period, and for three years following closure completion.

20.6.1 Objectives

The following objectives have been considered for closure planning:

- Compliance with current environmental legislation in the country, adopting environmental protection standards;
- Focus on protecting affected areas after closure, restoring them to a condition similar to pre-mining conditions;
- Environmental protection using techniques and technologies designed for risk control, land stabilization, and physical and chemical discharge containment, with a focus on degradation prevention;
- Public health and safety protection as well as the environment from physical and chemical impacts in the area of influence;
- Closure incorporating new technologies that improve environmental reclamation and closure performance; and

- Social management standards compliance for the social, economic, and institutional development of the Buriticá Project area.

20.6.2 Design Standards

The key closure design standards include the following:

- Safety: Dismantling or removing infrastructure and installations that create risk for personal safety. All remaining supplies will be removed from the site, and hazardous waste disposed in accordance with applicable regulations;
- Physical Stability: Topography reconfigured to integrate the terrain and surface drainage with the area, and ensure physical stability of remaining facilities;
- Geochemical Stability: Covers reducing infiltration will be used to minimize seepage from reclaimed facilities and to protect the receiving waters. Ongoing water quality monitoring will ensure chemical parameters meet the water quality requirements; and
- Future Land Use: Facilities will be reclaimed and left in a condition to facilitate future planned use for the area.

At the end of the Buriticá Project's useful life, morphological reconfiguration of the affected land will be carried out as well as installation of the necessary infrastructure ensuring land stability and landscape reclamation.

Based on the social and demographic dynamics verified during the course of the Buriticá Project's useful life, the potential uses of the intervened area will be jointly defined with the community.

20.6.3 Closure Components

Final and permanent closure costs by major facility are summarized below:

20.6.3.1 Underground Mine

- Mine surface facilities salvage and demolition except for the water treatment plant and related facilities;
- Underground mine equipment salvage;
- Sealing mine entrances to prevent unauthorized access; and
- Hydraulic plug installation in the Higabra tunnel to allow mine flooding and reduce water discharge to minimal seepage flow.

20.6.3.2 Filtered Tailing and Waste Rock Storage Facility:

The TSF will be constructed in sequential cells as a series of expansions, and reclamation will occur concurrently with operations. Therefore, most of reclamation activities costs will be realized during the operating period.

20.6.3.3 Water Treatment Plant

Continued operation of a reduced-capacity WTP is planned to treat remaining mine flows and TSF waters post site closure.

20.6.3.4 Other

- Retention of roads, bridges, fences, and paths being used by local communities; and
- Disturbed surface area regrading and revegetation.

Geochemical studies have been carried out and include kinetic testing and seepage studies for the TSF estimated flows and water chemistry. It has been determined that both tailing and waste material could have long-term acid drainage generation potential, and therefore seepage will be collected and monitored during closure and post-closure. The closure plan calls for total cover of the TSF to prevent water infiltration and seepage. Seepage not meeting discharge standards would be routed to the WTP prior to discharge. It is expected that the volume of water would be minimal, if any.

Main closure activities considered for the Project are summarized below.

Table 20.4: Summary of Closure Activities

Facility	Activity
PROGRESSIVE CLOSURE	
Underground Mine	The longhole stoping and cut & fill methods will be utilized; therefore, the majority of stopes will be backfilled during operations.
	Gradual access closure to prevent access, or for ventilation control as mining progress.
Tailing and Waste Rock Storage Facility	Runoff water ditch maintenance and monitoring.
	Progressive covering with soil material and revegetation.
	Maintenance of drainage structures for collection, and treatment.
	Maintaining diversion structures.
	Facility physical stability monitoring.
FINAL CLOSURE	
Underground Mine	Facility and infrastructure dismantling.
	Mine entrance closure with concrete plugs with drainage pipes. Higabra tunnel closure includes non-draining hydraulic plug.
	Surface opening closure including raises as required using concrete plugs.
	Warning and cautionary sign posting.
Beneficiation Plant	Removal of material to waste disposal facility or TSF.
	Facility wash down and water treatment.
	Empty and neutralize tanks and equipment that may have contained industrial material such as cyanide or acid solutions.
	Processing facility dismantling.
	Concrete foundation and structure demolition.
	Land reclamation and revegetation.
	WTP to be maintained during post-reclamation period until acceptable water quality is achieved.
Tailing Storage Facility	Diversion and contour channels will be maintained for storm water diversion.
	Cover: Soil cover and revegetation.
	Maintain facility drainage mechanisms until seepage stops.
	Surge and Collection Ponds: Maintain contact water collection ponds
	Slope Movement Protection: Closure design to protect against potential slope instability.
	Access route closure without impeding established travel routes used by local communities.
	Warning and cautionary sign posting.
Water Management Facilities	Mine Water Management: Maintain mine water drainage facilities until discharge flows subside or water standards can be met.
	Process Water Management: Remove all beneficiation plant structures.
	Contact Water Management: Maintain collecting facilities and handle TSF contact water until there are no drainage flows or water standards can be met.
	Rainwater and Runoff Water Management: Diversion and collection ditches for rainwater and surface runoff remain during closure and are maintained during post-closure.
	Ditches collecting storm water runoff from facilities may also be removed if not required during closure or pot-closure.
Electrical Supply	Dismantling by an authorized company.
	Substations, electrical power lines, and fittings will be dismantled using procedures and specific electrical industry regulations.
Ancillary	Cleaning and decontamination of facilities and equipment; prevent chemical, fuel or oil spillage.

Facility	Activity
Facilities	Dismantling: Concrete foundation and infrastructure demolition.
	Land reclamation: Land grading and revegetation.
POST-CLOSURE	
Underground Mine	Preventive Sign Maintenance
	Mine Entrance and Drift Closures Inspection and Maintenance
	Drainage water management and monitoring: Treatment as required, or discharged to a receiving body.
Tailing Storage Facility	Channel maintenance.
	Preventive Sign Maintenance.
	Water Treatment: Water quality must be monitored and treated as required prior to discharge. The pond must be monitored until discharge stops or water standards can be met.
Water Management Facilities	Facility must be maintained.
Maintenance Programs	Physical Maintenance: Maintain physical slope stability at TSF, and all other active facilities.
	Geochemical Maintenance: The condition of covers in TSF will be maintained and performing according to design.
	Hydrological Maintenance: Periodical maintenance and cleaning of runoff water drainage system, including runoff ditches around the TSF, to prevent erosion and sediment accumulation.
Monitoring Programs	Physical Stability Monitoring: Evaluation of the geotechnical stability of the rehabilitated facilities.
	Geochemical Stability Monitoring: Inspection of covers and condition of closed facilities; surface and groundwater monitoring according to post-closure plan.
	Hydrological Stability Monitoring: Drainage works inspection and maintenance during post-closure, including diversion channels, contour channels, drainages, control ponds, and collection ponds.
	Biological Monitoring: Ongoing inspection and monitoring of wild flora and fauna species in reclaimed and natural areas within the mine operational site.
	Groundwater monitoring: Downstream of the TSF during post-closure.

Source: M3 2014; JDS updated 2016

Cost estimates for closure are summarized below.

Table 20.5: Closure Cost Estimation

Item	Closure plan costs (US$ M)
Surface Demolition & Ground Reshaping	2.2
Waste Removal	0.5
TSF Closure Cover	0.7
TSF	0.2
Mine Surface Infrastructure Demolition, Raise Covers	0.4
Mine Portal Plugs (includes hydraulic plug)	3.1
Revegetation/Reseeding	0.1
Post-Closure Monitoring & Maintenance	1.7
Water Treatment (post-closure)	5
General & Powerline	0.5
Indirects, Owner's Costs & Taxes	1.4
Contingency	1.6
Sub-total	17.5
Salvage Value	-7.5
Total Cost US$	**10**

Numbers may not add due to rounding.

Source JDS, 2016

21 Capital Cost Estimate

21.1 Introduction & Estimate Results

Project capital costs total US$662M, consisting of the following distinct phases:

- Initial Capital Costs – includes all costs to develop the project to a 2,100 tonnes per day production. Initial capital costs total $389M and are expended over a 35-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 $273M and are expended in operating years 1 through 13.

The capital cost estimate was compiled utilizing input from engineers, contractors, and suppliers with experience delivering projects in Latin America. Wherever possible, bottom-up first principle estimates were developed and benchmarked against other projects of similar size and site conditions.

Table 21.1 presents the level 1[1] capital estimate details for both initial and sustaining capital costs in Q1 2016 dollars with no escalation.

Table 21.1: Level 1 Capital Estimate Detail

WBS	Description	Initial Capital (US$ M)	Sustaining Capital (US$ M)	Total Capital (US$ M)
1000	Site Development	10.9	-	10.9
2000	Underground Mining	86.5	178.3	264.8
3000	Processing Facilities	97.6	11.6	109.2
4000	Tailing & Waste Rock Management	7.7	16.1	23.8
5000	Off-Site Infrastructure	10.0	12.7	22.7
6000	On-Site Infrastructure	45.3	18.7	64.0
7000	Project Indirect Costs	28.5	9.1	37.5
8000	Engineering & EPCM	27.8	0.6	28.4
9000	Owners Costs	21.8	4.3	26.1
9800	Taxes	17.7	14.0	31.7
9900	Contingency	35.4	7.0	42.4
	Grand Total	389.2	272.5	661.7

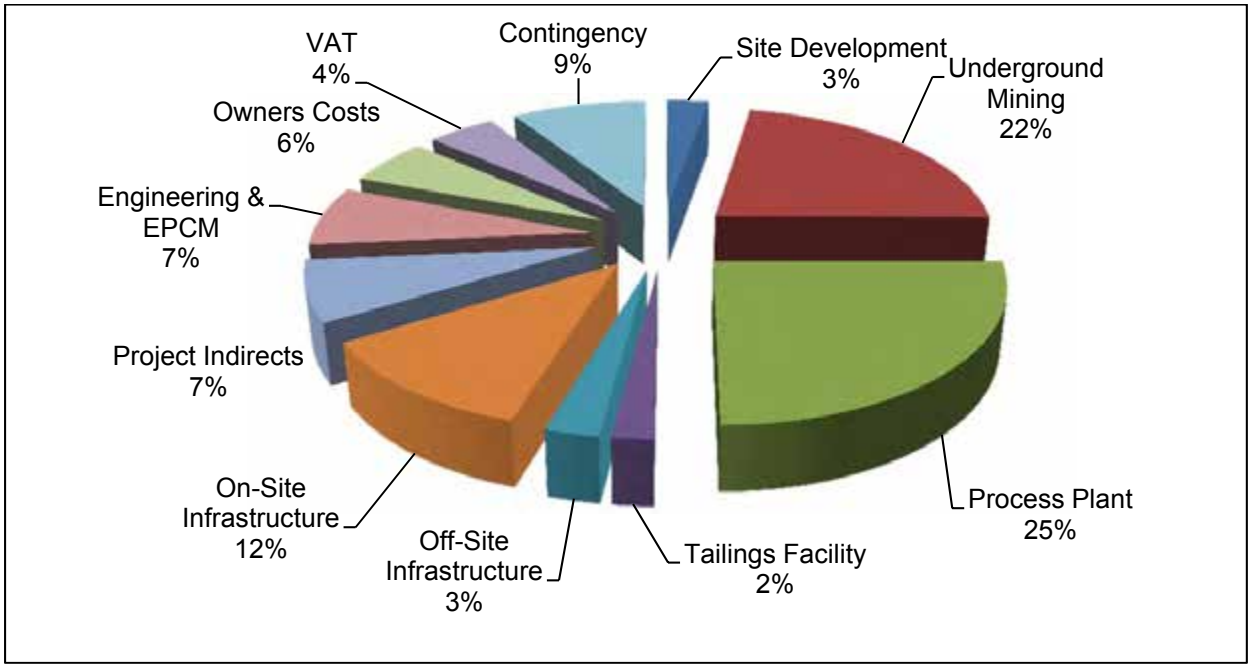
Note: Subtotals/totals may not match due to rounding

Source: JDS, 2016

It should be noted that $28.1M of the $31.7M Value-Added Tax ("VAT") cost is recovered as tax credits within the income tax buildup of the economic model. Refer to Section 23 for additional details.

[1] Level 1 refers to the level of detail within the Project Work Breakdown Structure ("WBS"). For example, area 2000 is the first level of breakdown or "Level 1".

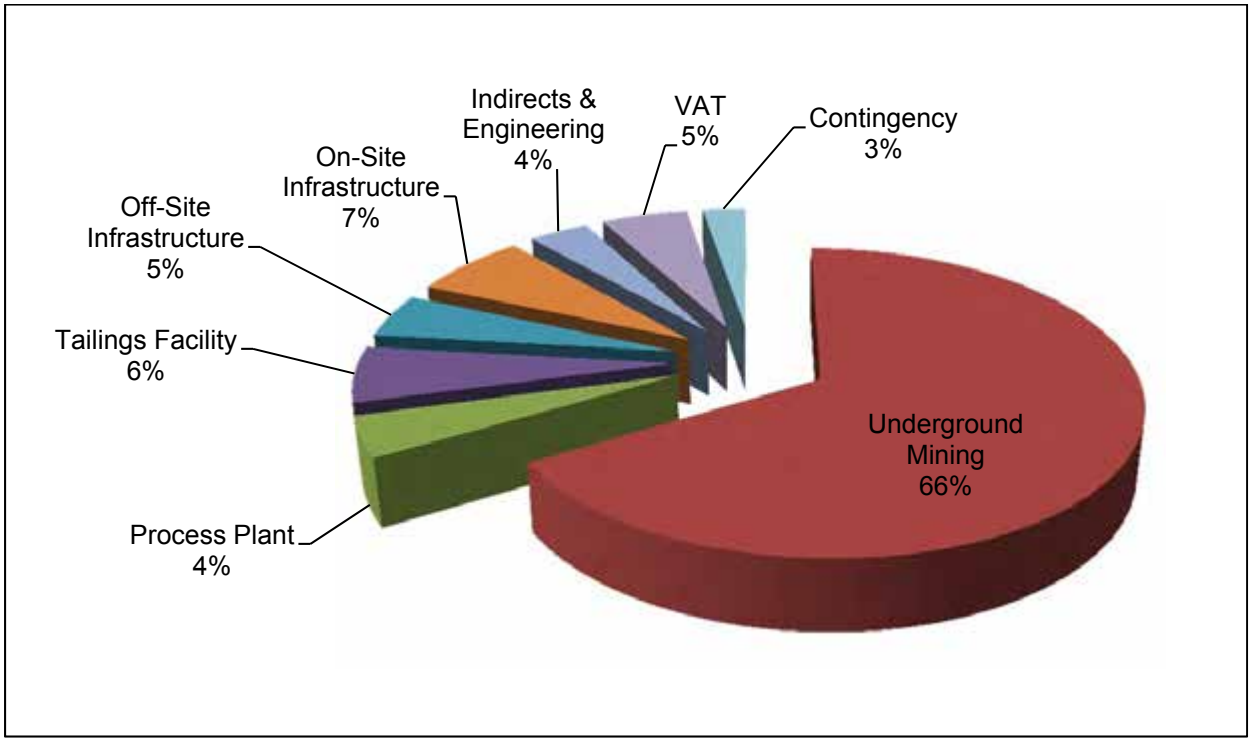
Figure 21.1: Distribution of Initial Capital Costs



Source: JDS, 2016

The majority of sustaining capital costs relate to underground lateral and vertical development.

Figure 21.2: Distribution of Sustaining Capital Costs

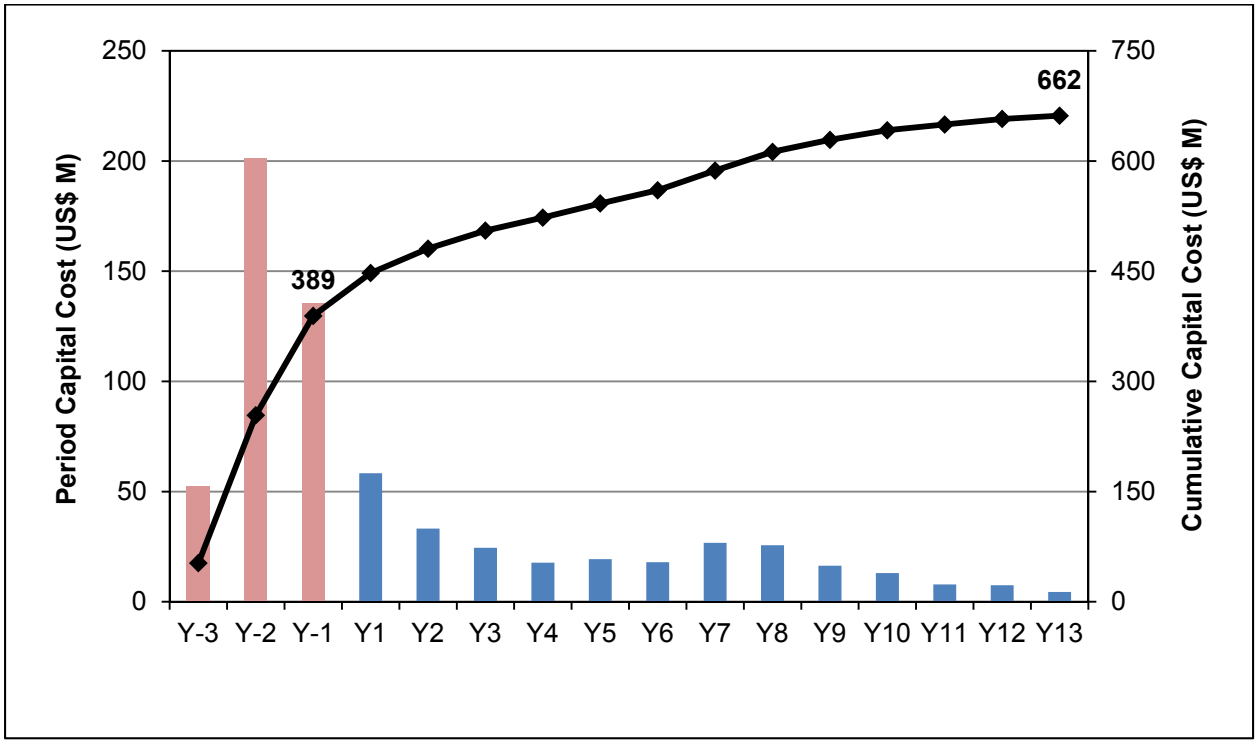


Source: JDS, 2016

21.2 Capital Cost Profile

All capital cost expenditures have been included in the economic cash flow model according to the development schedule. Figure 21.3 presents an annual life of mine capital cost profile.

Figure 21.3: Life of Mine Capital Cost Profile



Source: JDS, 2016

21.3 Basis of Capital Estimate

21.3.1 Scope of Estimate

The Buriticá capital estimates include all costs to develop and sustain the Project at a commercially operable status.

Initial capital costs include all construction costs incurred until the declaration of commercial production (2,100 tonnes per day). Costs related to the operation of the mine and process plant between first ore introduction and commercial production (a period of 60 days) are included as operating costs occurring in the pre-production phase, and as such, are not included in the capital cost buildups. Refer to Section 22 for details of the project operating costs.

The capital costs do not include the costs related to operating consumables inventory purchased before commercial production; these costs are considered within the working capital estimate described in Section 23.

Sunk Costs and Owners Reserve accounts are not considered in the Feasibility Study estimates or economic cash flows.

21.3.2 Key Estimate Assumptions

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

- The capital estimate is based on the contracting strategy, execution strategy, and key dates described within the Project Execution Plan in Section 25 of this report;
- Underground mine development activities will be performed by the owner's forces; and
- All surface construction (including earthworks) will be performed by contractors.

21.3.3 Key Estimate Parameters

The following key parameters apply to the capital estimates:

- **Estimate Class:** The capital cost estimates are considered Class 3[2] feasibility cost estimates (-15%/+20%). The overall project definition is estimated at 30%.
- **Estimate Base Date:** The base date of the capital estimate is January 1st, 2016. No escalation has been applied to the capital estimate for costs occurring in the future. Proposals and quotations supporting the Feasibility Study Estimate were received in Q3 2015 and are generally valid for a period of sixty (60) to ninety (90) days.
- **Units of Measure:** The International System of units (SI) is used throughout the capital estimate.
- **Currency:** All capital costs are expressed in United States Dollars (US$). Table 21.2 presents the exchange rates used for costs estimated in foreign currencies.

Table 21.2: Estimate Exchange Rates

Currency	Code	X : US$
Colombian Peso	COP	2,850
Canadian Dollar	CAD	0.75
Euro	EUR	1.10

Source: JDS, 2016

21.3.4 Estimate Responsibility Matrix

A responsibility matrix has been developed as a part of the basis of estimate. JDS is responsible for the overall management, development, assembly, and accuracy of the overall capital cost estimate with input from other companies.

[2] ACEE defines a Class 3 estimate as a budget authorization estimate based on 10% to 40% project definition, semi-detailed unit costs with assembly level line items, and an accuracy of between -20%/+30% and -10%/+10%.

21.3.5 Labour Rates

21.3.5.1 Contract Labour Rates

Contractor labour rates were built up by applying appropriate burdens to base labour rates provided by Colombian labour providers to determine all-in commodity unit labour rates.

Table 21.3: Contract Labour Rates

Category	Civil	Conc.	Struct.	Arch.	Mech.	Pipe	Elec.	Control
Direct Rate (1)	7.63	8.90	9.41	9.28	9.41	9.28	9.28	9.41
Transportation & Facilities	1.06	1.06	1.06	1.06	1.06	1.06	1.06	1.06
Warehouse & Tool Crib	-	0.20	0.20	0.20	0.20	0.20	0.20	0.20
Tools/PPE/Consumables	1.50	3.50	3.50	3.50	3.50	3.50	3.50	3.50
Training & Awards	0.60	0.60	0.60	0.60	1.10	1.10	1.10	1.10
Supervision	1.08	1.43	1.48	1.46	1.53	1.51	1.51	1.53
Non-Productive Time	1.08	1.43	1.48	1.46	1.53	1.51	1.51	1.53
Overhead & Profit	3.24	4.28	4.43	4.39	4.58	4.54	4.54	4.58
All-in Rate	**16.18**	**21.39**	**22.15**	**21.96**	**22.90**	**22.71**	**22.71**	**22.90**

Note 1: Direct rate includes payroll and social burdens, and shift premiums

Source: JDS, 2016

21.3.5.2 Operational (Owner) Labour Rates

Operational labour rates were built up from first principles, in consultation with CGI human resources and legal resources. Base rates are based on current CGI salaries for similar positions, and legal and union premiums and benefits were built up to create all-in rates. Operational labour rates and staffing levels are described further within Section 22.

21.3.6 Fuel & Energy Supply

21.3.6.1 Fuel

Fuel consumption is based on the engineered estimates within the various direct WBS areas of the estimate. Fuel pricing of $0.67/L, inclusive of delivery to site, is based on firm quotations received by CGI supply chain personnel.

21.3.6.2 Electricity Supply

Permanent power costs are based on the estimated demands for underground and surface facilities, and a budgetary utility cost of US$0.11/kWh.

21.3.7 Site Development

Site development includes all costs to develop the plant site area, including:

- Bulk cut/fill, rough, and final grading for all roads & pads;
- Water management structures, including surface drainage features and water storage ponds;
- Earthen liners (ROM stockpile & ponds);
- Mechanically stabilized (“MSE”) retaining walls and other slope stabilization;

- Concrete water diversion channels, including all bridge structures;
- Underground & surface utilities, including HDPE piping to water treatment plant;
- Water supply well drilling;
- Site fencing; and
- Process area lighting.

Concrete retaining walls are integral to foundation works of the various plant areas, and are thus included the concrete estimates of the process sub-areas.

Earthworks quantities were developed from a 3D civil model. Other earthworks and site development quantities were developed based on site layouts and requirements typical of similar projects. Engineering quantity allowances were added for overbuild, spillage, and wastage. Database unit pricing was applied to the engineered quantities.

It is assumed that all site earthworks will be performed by a contractor, with no Owner supplied construction or support equipment provided.

21.3.8 Underground Mining

21.3.8.1 Underground Mobile Equipment

Underground mining equipment quantities and costs were determined through buildup of mine plan quantities and associated equipment utilization requirements. Budgetary quotes were received and applied to the required quantities.

An allowance for a major overhaul at 60% of the initial equipment purchase price is allowed half-way through the usable life of the equipment.

21.3.8.2 Underground Infrastructure

Design requirements for underground infrastructure were determined from calculations by engineers with specialized knowledge of ventilation, dewatering, paste backfill, and material handling design.

Budgetary quotations were received 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.

21.3.8.3 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.

- It is assumed that all lateral development and conventional raises will be performed by CGI staff, and that Alimak raises will be performed by a contractor.
- Lateral development fuel, equipment usage, power, and consumables requirements were developed based on the mine plan requirements. Manufacturer database equipment usage rates were applied to the required operating hours. Quotations were obtained by CGI for consumables and the rates described in Section 21.3.6 were applied for fuel and power.

- Lateral development labour requirements were determined by the required equipment fleet in operation. Supervision and support services were estimated, based on requirements for similar operations. Operational labour rates were applied as described in 21.3.5.2.
- Budgetary quotations for Alimak raises were obtained from a contractor and applied to the mine plan quantities.

21.3.8.4 Capitalized Development Costs

Capitalized development costs are defined as mine operating expenses (operating development, ore extraction, mine maintenance, and mine general costs) incurred prior to the introduction of ore 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 development costs are included in the asset value of the mine development and are depreciated over the mine life within the financial model.

21.3.9 Surface Construction (Plant & Site Infrastructure)

Table 21.4 presents a summary of the basis on which the processing facilities and surface infrastructure elements for the Project have been estimated.

Table 21.4: Surface Construction Basis of Estimate

Commodity	Estimate Basis
Equipment	
Major Equipment	Budget quotations were solicited from qualified suppliers for the major equipment identified in the flowsheets and equipment register. A proposal accuracy of +/- 15% was requested from suppliers.
Minor Equipment	In-house data (firm and budgetary quotations from recent projects) was used for minor or low value equipment.
Installation (Labour & Materials)	
Site Preparation	Quantities have been developed from 3D models. Database unit pricing from similar projects used, with the exception of the Main Access Road, where averaged competitively bid unit rates were utilized. All surface civil works is assumed to be performed by contractors.
Concrete	Quantities have been developed from 3D model layouts and design calculations. Allowances were made for lean concrete. Database unit pricing from similar projects used for material costs and installation man-hours.
Structural Steelwork	Quantities have been determined from 3D models, design calculations and general arrangement drawings, with factored considerations for minor steel and connections. Database unit pricing from similar projects used for detailing, supply, fabrication, and erection costs.
Mechanical Bins and Chutes	Quantities for plate work, abrasion resistant ducting, and other mechanical plate work were developed based on general arrangement ("GA") drawings and expertise with similar operations. Database unit pricing from similar projects used for supply, fabrication, and installation costs.
Process Piping and Valves	General piping quantities were developed, based on line lists developed from process and instrumentation diagrams ("P&IDs"), and rough lengths determined from GA drawings. Database unit pricing from similar projects used for spool detailing, supply, fabrication, and installation costs.
Electrical	Electrical quantity take-offs are based on initial equipment lists and single line diagrams. Database unit pricing for materials and installation man-hours.
Electrical	Quantities have been based on preliminary P&IDs Database unit pricing for materials and installation man-hours.
Construction Equipment	
Capital Equipment Overhauls	Construction equipment costs are included according to the tasks performed and the crew hours involved. Construction equipment is included as a direct cost at $8.50/hour. This account is used for rentals and any purchase of commonly shared equipment (such as cranes and forklifts that are eventually turned over to the owner), scaffolding, and subcontractor equipment charges.
Permanent Equipment Upgrades & Overhauls	
Capital Equipment Overhauls	A sustaining capital allowance of US$1M per year (inclusive of VAT & contingency) has been allowed in years 1 through 13 for the overhaul or improvements to processing facilities. This capital allowance is pro-rated among the process areas by initial capital cost.

Source: JDS, 2016

21.3.9.1 Water Treatment Plant

The water treatment plant ("WTP") is constructed in four phases throughout the life of the Project, coinciding with treatment capacity requirements. The water treatment plant design and equipment list was developed by a specialty contractor.

A budgetary design/supply quotation was obtained from a specialized WTP vendor. Installation costs were estimated as described in Section 18. Table 21.5 presents the time phasing and total direct costs of the water treatment plant.

Table 21.5: Water Treatment Plant Installation Phases

Phase	Description	Year	Direct Cost (US$ M)
1	Initial build, including pads & ponds	-3	11.5
2	One additional UF/RO train	3	1.8
3	One additional HDS, 2nd stage UF/RO, and crystalizer	5	3.6
4	One additional UF/RO train	7	1.8
	Total		**18.8**

Note: Direct costs only – indirect costs, VAT, and contingency are included elsewhere in the estimate.

Source: JDS, 2016

21.3.9.2 Aerial Tailing Tram

The tailing tram is purchased in Year 1 of operations and installed in Year 2 (after a 12 month equipment lead time). A preliminary tram design was performed by a specialist consultant to determine the tram layout and tower requirements.

A budgetary design/supply quotation was obtained from an aerial tram supplier. Mechanical installation costs were estimated leveraging experience from tram specialists that have worked on similar installations. Civil and concrete costs were estimated as described in Section 21.3.7. Allowances were made for the following inter-discipline interfaces:

- Access road to tram tower
- Electrical supply and tie-ins
- Instrumentation/automation and tie-in to plant control system

Costs for the haulage of tailing material on the Main Access Road to the paste plant during the transition from construction to production are included as an operating expense for mining until the tailing tram is constructed.

21.3.10 Tailing Storage Facility

21.3.10.1 Tailing Storage Facility Construction

Earthworks quantities were developed from engineering drawings by the design engineer. Database unit pricing was applied to the engineered quantities.

It is assumed that TSF construction will be performed by a contractor, during both the construction and operations phase.

Progressive closure of the TSF (a rock and soil cover) is included within the closure costs, described in Section 23.

21.3.10.2 Pre-Production Surface Waste Rock Handling

Costs to handle underground waste rock, on surface, from the waste rock staging area (the south lobe of the TSF) or the Yaraguá portal to the TSF containment cell (north lobe) are included in the initial capital costs. Costs for surface haulage of waste rock and dried tailing during operations are included as operating costs for processing.

The basis of estimate assumes an owner procured and operated equipment fleet. Material quantities, schedule, and corresponding equipment utilizations have been defined by the mine plan. Equipment acquisition costs are described in Section 21.3.12. Database equipment usage rates have been applied for parts, fuel, and consumables. Operational labour rates for technical, supervision, and operations personnel have been applied, as described in Section 21.3.5.2.

21.3.11 Off-Site Infrastructure

21.3.11.1 Site Access Road

Main access road quantities are based on “issued-for-bid” designs performed by engineering.

Main access road construction is performed in three phases:

- Phase 1: Initial pioneering road to allow construction machinery and semi-loads of equipment, using construction equipment assistance, access to the valley in order to start earthworks and facilities installations.
- Phase 2: Widening of corners and reduction of grades to allow highway tractor trailer traffic access for equipment deliveries. Phase 2 includes gravel surfacing, and slope stabilization and hydro-seeding.
- Phase 3: Completion & paving, performed in Year 2, includes costs for gravel sub-base, asphalt paving, and the remaining slope stabilization and hydro-seeding.

Unit costs and contractor indirect costs for the access road are based on the average of three competitive bids received by CGI in Q2 2015 for Phase 1 of the road. Database unit rates were used for activities not included in the tender documents (such as paving).

Table 21.6: Site Access Road Construction Phases

Phase	Description	Year	Direct Cost (US$ M)
1/2	Gravel surfaced access and 80% of ground support	-3	9.9
3	Paving and completion of ground support	2	1.2

Note: Direct costs only – indirect costs, VAT, and contingency are included elsewhere in the estimate.

Source: JDS, 2016

21.3.11.2 Low-Voltage (13.8kV) Power Transmission Line

A budgetary quotation from a local contractor was received for constructing the 13.8kV power transmission line.

21.3.11.3 High-Voltage (110kV) Power Transmission Line

The 110kV power line design, supply, and installation requirements and costs were developed by a Colombian engineering group specializing in power line designs.

Although the 110kV power line is constructed in Year -2, the power line costs are amortized over ten years (Year 1 through 10) as a facility fee to the project. An annual interest rate of 15.9% is applied against the design/build cost of the line and paid in equal installments (classified as sustaining capital) in the first 10 years of the mine life. Table 21.7 presents the financing cost buildup.

Table 21.7: 110kV Power Line Costs

Parameter	Unit	Value
Design/build cost of 110kV line	US$ M	7.3
Amortization Period	Years	10
Annual Interest Rate	%	15.9
Annual Payments	US$ M	1.9
Total Payments Made	**US$ M**	**18.9**
Cost of Borrowing	US$ M	11.6

Note: Financed costs include all indirect costs, engineering, Owners costs, VAT, and contingency associated with the power line installation

Source: JDS, 2016

21.3.12 Surface Equipment Fleet

Surface equipment fleet requirements are determined based on material movement requirements and experience at similar operations, and considering site conditions specific to the Project. Waste rock/tailing handling equipment requirements are based on equipment utilization requirements for the haulage operations. No equipment replacements are anticipated for the surface equipment fleet, with the exception of light vehicles, which are all replaced in Year 7. Used equipment acquisition has been considered for low utilization equipment.

Database unit pricing has been applied to the surface equipment fleet quantities.

21.3.13 Indirect Costs

21.3.13.1 Construction Support Contracts

During the construction phase, capital costs have been included for the following support contracts:

- General construction services: Crew of ten personnel for 24 months during the bulk of construction activities to provide miscellaneous support beyond the capacity of CGI's Site Services group.
- First aid and medical services: Full time medical coverage, on both day and night shift.
- Solid waste management: Allowance for supply of solid waste bins, and costs for waste pickup.

- Janitorial services: Labour for cleaning of temporary and permanent offices and public areas.

These contracts (with the exception of general construction services) will continue into the operations phase, and are included as an operating expense under the general and administration sector.

21.3.13.2 Temporary Facilities

Capital cost provisions have been made for the following general site indirect costs related to temporary facilities:

- Temporary Offices: Allowances for the provision (rental or purchase) of office and lunchroom trailers for EPCM and contractor personnel.
- Temporary communications: Allowances for temporary communications systems and networking until permanent systems are established.
- Miscellaneous field procurement: A total of US$1M ($40k x 27 months) has been allowed for field procurement for general site consumables not related to specific contractor scopes, such as bulk lumber, maintenance parts for temporary facilities, safety program supplies, site services supplies, and short term equipment rentals.

21.3.13.3 Construction Power

Although permanent (grid) power will be available to the Project early in the development schedule, It is anticipated that several diesel power generators will be required during the plant site and ancillary area construction activities; larger, central generators for construction facilities and construction power at the main plant work faces, and smaller generators to support ancillary area construction (such as the water treatment plant).

The acquisition costs for diesel generators are included in surface equipment fleet, and fuel and maintenance costs are included in the project indirect costs. Allowances have been made in the indirect costs for temporary power distribution, including power distribution panels surrounding the various work faces.

Small generators (less than 50kW) will be provided by contractors and are included in the construction equipment costs.

21.3.13.4 Contractor Indirect Costs

The following contractor indirect costs are burdened within the labour crew rates described in Section 21.3.5.1:

- Transportation & Facilities;
- Warehouse & Tool Crib;
- Tools/PPE/Consumables;
- Training & Awards;
- Supervision;
- Non-Productive Time; and

- Overhead & Profit.

Above the contractor indirect rates identified above, deterministic estimates have been included in the estimate for contractor mobilization and scaffolding materials. Scaffolding erection and maintenance is included in the direct hours.

21.3.13.5 Passenger Transportation (Bussing)

Costs are included within the estimate for the provision of passenger bussing services between Santa Fe and the Project site. Bussing requirements (quantities) were developed based on the following parameters:

- The total site man-days required for the Project, and distribution of personnel working day and night shift. Since the majority of personnel work dayshift only during construction, 70% of passenger busses will travel empty one-way;
- A passenger bus capacity of 25 persons; and
- An assumed 25% of personnel will reside in Buriticá, and therefore will not require bussing services, during the construction phase.

21.3.13.6 Freight

Freight costs during the construction phase have been developed based on the estimated number of loads. A total of 1,200 cargo loads is estimated (exclusive of fuel and explosives), at an average cost of approximately $6,600/load. Freight costs include customs fees, brokerage, and temporary warehousing.

Freight costs for the operations phase (sustaining capital) have been estimated in the following manner:

- 10% allowance of purchase price for underground and surface mobile equipment acquisitions and replacements;
- 5% allowance of purchase price for underground mining infrastructure and equipment rebuild materials;
- First principles estimate for the aerial tram and mill maintenance building components; and
- 10% allowance of purchase price for water treatment plant and mill maintenance facility equipment.

Fuel and cement commodity pricing includes delivery to site, and is thus excluded from the freight costs.

21.3.13.7 Commissioning & Spare Parts

Commissioning activities have been estimated on the basis that the EPCM and construction contractors will complete all pre-commissioning activities up to the introduction of first ore, at which time, CGI's processing staff will assume care, custody, and control of the plant and begin process commissioning and ramp-up.

Spare parts, first fills, and vendor assistance costs have been factored based on similar projects.

21.3.14 Engineering & EPCM

21.3.14.1 Detailed Engineering

Engineering costs within the estimate are based on budgetary quotations received from entities involved in the development of the Feasibility Study.

21.3.14.2 Project & Construction Management

Project and construction management costs are built up based on experience with similar sized projects. A detailed schedule of rates was applied against a staffing plan aligned with the Project schedule.

- Expatriate wages were used for contract project management (PM) personnel within the Project and construction management cost buildup; and
- Mine management and operations, field procurement, environmental, and some safety and administrative positions integral to the overall construction management effort will be provided by CGI. The costs for these positions are included in the mining costs for mining positions and the Owners costs for G&A positions.

21.3.15 Owner's Costs

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.

21.3.15.1 Water Treatment Plant Operation

The following cost elements are included in the initial capital costs for operation of the water treatment plant for twenty-one months:

- Technical and operations labour;
- Power supply; and
- Reagents, parts, and third party services.

21.3.15.2 Process Plant Operations

The following processing related costs are included in the initial capital:

- Management, technical, operations, and maintenance labour employed during the construction phase;
- Energy costs for power consumed during process commissioning and ramp-up activities; and
- Equipment usage costs related to the process plant (feed loader, dozer, etc.).

21.3.15.3 Pre-Production G&A – Labour

Costs for general and administrative labour are included for the following sectors:

- Management & Administration;
- Accounting;
- Human Resources;
- Community Relations;
- It & Communications;
- Procurement & Logistics;
- Health & Safety;
- Environmental;
- Security; and
- Surface Infrastructure & Maintenance.

21.3.15.4 Pre-Production G&A – Equipment

Costs for CGI owned site support equipment usage are included for the following sectors:

- Site Services;
- Warehouse/Material Management;
- Security;
- Health, Safety, & Environment; and
- Admin/Management.

21.3.15.5 Pre-Production G&A – Expenses & Services

Costs for general and administrative expenses and fees are included for the following sectors:

- Power line maintenance;
- Business travel;
- Safety and medical items;
- Environmental costs;
- Security & military support cost;
- Engineering support;
- Information management & communications;
- Office expenses;
- Human resources;

- Titles and associations;
- Insurance; and
- Socio-economic costs.

21.3.15.6 Owner's Project Management Team

Costs for the CGI Project Team are built up from a staffing plan aligned with the project schedule, and application of CGI provided rates. Costs are based on utilizing 80% expatriate labour for the Project Team.

21.3.16 Taxes

Value-added tax ("VAT" or "IVA") applies to goods and services provided in Colombia.

VAT is applied to the initial capital estimate as follows:

- 16% applied to permanent and mobile equipment acquisitions; and
- 16% of an estimated 10% contractor profit (net 1.6%) applied to labour, materials, equipment usage, and all other misc. costs.

VAT is not applied to the following items

- Owner labour;
- Capital mine development;
- Capitalized mine production costs; and
- Electricity supply.

21.3.17 Contingency

Contingency is a provision for project costs that will likely occur, but cannot be accurately defined or estimated. Including project contingency in a capital cost estimate is necessary to determine the most likely project cost.

A quantitative risk analysis (QRA) was completed for the Project to determine the capital risk profiles. A blended contingency was applied to the estimate through constructing and executing a probability analysis model. Costs were logically grouped by type and the P_{10} and P_{90}[3] variations were defined for both quantity and unit price risk. The model utilized Monte-Carlo sampling (5,000 iterations) to determine the contingency amounts. The analysis excluded any risks related to escalation and foreign exchange.

The selected contingency for pre-production is US$35.4 million or 10% which represents a P_{75} level of confidence.

The project contingency currently included in the estimate does not include provisions for management reserve.

[3] P_{10} and P_{90} refer to that confidence level for a given estimate. By definition, a P_{90} refers to a 90% level of confidence that the value (estimate) *will not* be exceeded, and P10 refers to a 90% level of confidence that the value (estimate) *will* be exceeded. The P_{75} confidence chosen for the capital contingency refers to a 75% confidence that the contingency provisions will not be exceeded.

21.3.18 Exclusions

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

- Finance and interest charges;
- Operating spare parts;
- Working or deferred capital (included in the financial model);
- Financing costs;
- Currency fluctuations;
- Lost time due to severe weather conditions;
- 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.); and
- Escalation cost.

22 Operating Cost Estimate

22.1 Introduction & Estimate Results

Life of mine (LOM) operating costs for the Project average $100.50/tonne processed. This includes the following sectors:

- Underground mining;
- Mineral processing;
- General & administration; and
- Corporate management fee.

The operating costs exclude off-site costs (such as shipping and refining costs), taxes, and government royalties. These cost elements are used to determine the net smelter return ("NSR") in the economic model, and are discussed in Section 23.

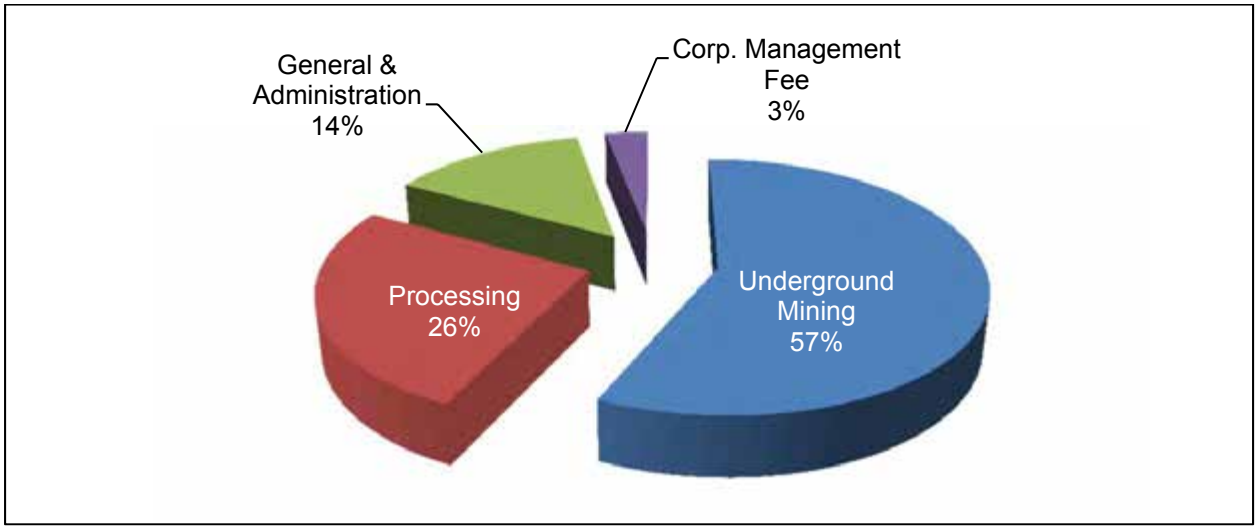
Table 22.1 presents a summary of the LOM operating costs, expressed in United States Dollars ("US$") with no escalation. Figure 22.1 illustrates the distribution of operating costs among the cost sectors.

Table 22.1: Operating Cost Summary

Sector	Average US$ M/year	Life of Mine $US M	$/t processed
Underground Mining	56.3	784.7	57.21
Processing	25.8	358.8	26.16
General & Administration	13.9	193.8	14.13
Corporate Management Fee	3.0	41.3	3.01
Total Mine Operating Costs	**99.0**	**1,378.6**	**100.50**

Source: JDS, 2016

Figure 22.1: Distribution of Operating Costs



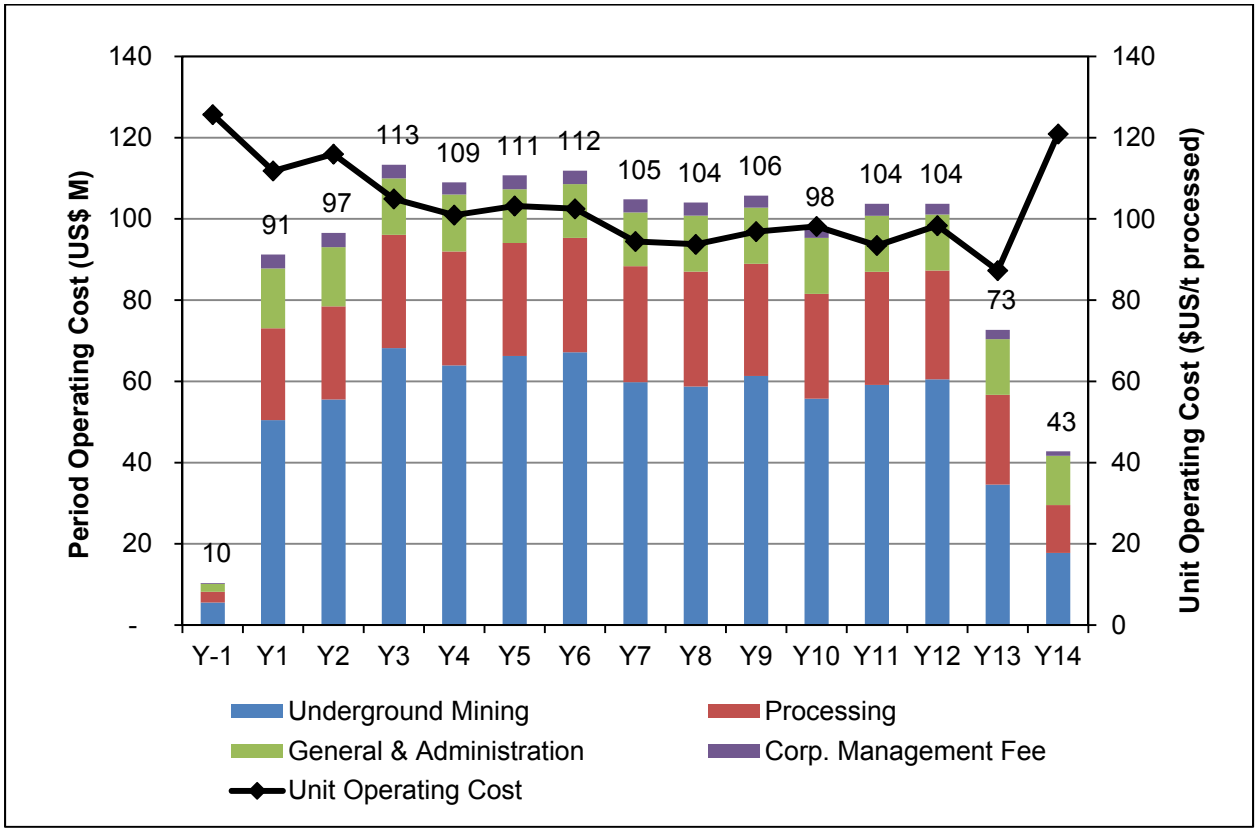
Source: JDS, 2016

The operating cost estimate was compiled utilizing input from engineers, contractors, and suppliers with experience operating projects in Latin America. Wherever possible, bottom-up first principle estimates were developed and benchmarked against other projects of similar size with similar site conditions.

22.2 Operating Cost Profile

All operating costs have been included in the economic cash flow model according to the development schedule. Figure 22.2 presents an annual life of mine operating cost profile.

Figure 22.2: Life of Mine Operating Cost Profile



Source: JDS, 2016

22.3 Operational Labour Rate Buildup

Operational staff labour rates have been built up by applying legal and discretionary burdens against base labour rates. Eleven wage scales were defined, and applied to the various operational positions based on skill level and expected salary. CGI legal and human resources personnel were involved in the buildup and verification of the operational labour rates.

22.4 Mine Operating Costs

The mine operating costs include the following functional areas:

- Underground Mining
  - Waste development – costs of main ramps, raises, drifts and attack ramps including drilling, blasting, mucking, and hauling;
  - Production – ore extraction costs including drilling, blasting, mucking, and hauling;
  - Backfill – backfill operating costs including the paste plant;
  - Mine Maintenance – maintenance labour costs that support all other sectors; and
  - Mine General – costs for mine support activities, such as technical services, shared infrastructure, support equipment, and definition drilling.
- Tailing Transportation
  - Surface Tailing Haulage – transport costs of filtered tailing from the process plant to the paste plant prior to tram commissioning;
  - Aerial Tailing Tram – aerial tailing tram operating costs.

Table 22.2: Mining Operating Cost Summary

Area	Average US$ M/year	Life of Mine $US M	US$/t processed
Underground Mining	**55.8**	**777.8**	**56.70**
Waste Development	12.9	179.7	13.10
Production	30.1	419.5	30.59
Backfill	4.6	63.4	4.62
Mine Maintenance	1.5	21.3	1.56
Mine General	6.7	93.9	6.84
Tailing Transportation	**0.5**	**6.9**	**0.50**
Surface Tailing Haulage	0.1	0.9	0.07
Aerial Tailing Tram	0.4	5.9	0.43
Total	**56.3**	**784.7**	**57.21**

Source: JDS, 2016

22.4.1 Underground Mining

22.4.1.1 Underground Mining Labour

Underground mining staffing levels related to production activities are built up based on the productivities (man-hours) required for mining activities occurring within a given time period. As such, mining manpower changes during the mine life. Mining labour rates are based on a mix of expatriate and local hire labour.

Mine labour (including supervision and support) related to development drifting is distributed between capital development (sustaining capital costs) and operating development (operating costs), based on the activities being performed within a given time period. As such, only a portion of the mine staffing is allocated within the mining operating costs.

Table 22.3: Underground Mining Labour Costs

Area	Average LOM Staff	Average US$ M/year	Life of Mine $US M	US$/t processed
Lateral Waste Development	254	3.6	50.7	3.70
Production	66	1.4	19.4	1.41
Backfill	22	0.4	5.0	0.36
Mine Maintenance	84	1.5	20.3	1.48
Mine General	30	0.9	12.3	0.90
Total	**456**	**7.7**	**107.7**	**7.85**

Note: Staffing levels and costs include only operating costs. Balance of mine staffing costs are included within the sustaining capital costs

Source: JDS, 2016

22.4.1.2 Underground Mining Fuel Consumption

Underground mining fuel consumption has been built up based on the required equipment operating hours dictated by the mine plan for development or production based equipment, and annual allowances for support or fixed infrastructure equipment, based on experience at similar operations.

The unit fuel price used in the estimate is US$0.67/litre, which includes US$0.10/litre for delivery to site.

Table 22.4: Underground Mining Fuel Costs

Area	Average US$ M/year	Life of Mine $US M	US$/t processed
Lateral Waste Development	0.6	8.5	0.62
Production	2.0	28.2	2.05
Backfill	0.6	8.5	0.62
Total Mine Consumables Costs	**3.2**	**344.4**	**3.30**

Source: JDS, 2016

22.4.1.3 Underground Mining Equipment Operations

Underground mining equipment usage costs are based on the equipment operating hours required for the life of mine plan. Equipment usage costs include unit costs ($/hr) for the following elements:

- Maintenance parts;
- Tires;
- Lubricants; and
- Boxes, buckets, and ground engaging tools.

Unit costs for the elements above have been obtained from equipment manufacturers databases. Equipment replacements and major (mid-life) overhauls are included in the sustaining capital costs.

Table 22.5: Underground Equipment Costs

Area	Average US$ M/year	Life of Mine $US M	US$/t processed
Lateral Waste Development	2.9	39.7	2.90
Production	9.7	135.0	9.84
Mine General	0.9	13.0	0.95
Total	**13.5**	**187.7**	**13.68**

Source: JDS, 2016

22.4.1.4 Underground Mining Power

Electrical power consumption has been based on the equipment connected loads, discounted for operating time and the anticipated operating load level.

Electricity unit cost is based on a budgetary rate of $0.11/kWh.

Table 22.6: Underground Mining Power Costs

Area	Average US$ M/year	Life of Mine $US M	US$/t processed
Lateral Waste Development	0.5	7.0	0.51
Production	1.8	25.7	1.87
Backfill	0.3	4.0	0.29
Mine General	0.3	3.7	0.27
Total Mine Consumables Costs	**2.9**	**40.4**	**2.95**

Source: JDS, 2016

22.4.1.5 Underground Mining Consumables

Mining consumable usage rates are built up based on the mine plan quantities for development and production activities.

Unit costs are typically based on budgetary quotations. Minor item costs are based on catalog or database values. Seven percent of the base quoted or database pricing for consumables has been added within the commodity pricing for delivery (freight) to site.

Table 22.7: Mining Consumables Costs

Area	Average US$ M/year	Life of Mine $US M	US$/t processed
Lateral Waste Development	5.3	73.7	5.37
Production	15.2	211.3	15.40
Backfill	3.9	54.4	3.97
Mine Maintenance	0.1	1.0	0.07
Mine General	0.3	4.1	0.30
Total Mine Consumables Costs	**24.7**	**344.4**	**25.11**

Source: JDS, 2016

22.4.1.6 Definition Drilling

An all-in quoted budgetary rate of US$119.75/m has been applied to the definition drilling quantities within the life of mine plan. This equates to a total LOM cost of US$52.2M or US$3.81/tonne processed.

22.4.2 Tailing Delivery to Paste Plant

The tailing delivery area of the operating cost estimate includes the costs related to transportation of dried process tailing from the processing facilities, located in the Higabra Valley to the paste plant, located at the Veta Sur portal. Material will initially be trucked on surface, and then transported by an aerial tailing tram once it is constructed. A budgetary all-in contract rate of US$4/t has been applied to the paste fill tailing quantities for the surface haulage requirements.

Aerial tailing tram operating parameters and costs have been estimated by a lift and tram specialist consultant in conjunction with a tram line manufacturer. It is assumed that the mine maintenance staff will also provide maintenance labour to perform maintenance activities on the aerial tram.

Table 22.8: Tailing Delivery Cost Summary

Sector	Life of Mine $US M	US$/t processed
Surface Tailing Haulage	**0.9**	**0.07**
Surface Tailing Haulage	0.9	0.07
Aerial Tailing Tram	**5.9**	**0.43**
Labour	1.6	0.11
Power	2.3	0.17
Maintenance Materials	2.1	0.15
Total	**6.9**	**0.50**

Source: JDS, 2016

22.5 Process Operating Costs

The process operating costs can be separated into three distinct areas: mineral processing, water treatment, and TSF. Table 22.9 presents a summary of the process operating costs by area. The subsections below provide the buildups of each area.

Table 22.9: Processing Operating Cost Summary

Area	Average US$ M/year	Life of Mine $US M	US$/t processed
Mineral Processing Plant	**21.0**	**293.1**	**21.37**
Mill General	0.6	7.7	0.56
Primary Crushing & Reclaim	0.8	11.3	0.83
Grinding & Gravity Concentration	8.5	118.8	8.66
Concentrate Leach & CCD	4.5	62.8	4.58
Merrill Crowe and Refinery	1.4	20.0	1.46
Tailing and Cyanide Oxidation	2.8	38.8	2.83
Ancillaries	1.2	17.2	1.25
Process Maintenance	1.2	16.5	1.20
Water Treatment Plant	**1.1**	**15.0**	**1.10**
Tailing Storage Facility	**3.6**	**50.7**	**3.70**
Total	**25.8**	**358.8**	**26.16**

Source: JDS, 2016

22.5.1 Mineral Processing

22.5.1.1 Mineral Processing Labour

Milling operations and maintenance staffing levels have been built up based on experience at similar operations. Labour costs are based on fully burdened staffing wage bandings, as described in Section 22.3.

22.5.1.2 Mineral Processing Power

Electrical power consumption has been based on the equipment connected loads, discounted for operating time and the anticipated operating load level.

Electricity unit cost is based on a budgetary rate of $0.11/kWh.

Table 22.10: Mineral Processing Unit Power Consumption & Cost

Area	Unit Consumption (kWh/tonne)	Power Cost ($/kWh)	US$/t processed
Primary Crushing & Reclaim	1.02	0.11	0.11
Grinding & Gravity Concentration	30.92	0.11	3.40
Concentrate Leach & CCD	4.04	0.11	0.44
Merrill Crowe and Refinery	4.77	0.11	0.52
Tailing and Cyanide Oxidation	6.75	0.11	0.74
Ancillaries	5.16	0.11	0.57
Total	**52.66**		**5.79**

Source: JDS, 2016

22.5.1.3 Mineral Processing Reagents

Milling reagent consumption rates have been determined from the metallurgical test data or experience from other operations (when test data was not available). Unit pricing is based on budgetary quotations obtained by CGI supply chain.

22.5.1.4 Other Mineral Processing Costs

Maintenance parts costs have been factored, based on the direct capital costs of the equipment within each area. An allowance of 10% of the maintenance parts costs has been included for maintenance services. Annual allowances have been included for lubricant consumption, outside services, and miscellaneous supplies/tools.

Grinding media and liners have been estimated on a kilogram/tonne basis, based on experience at similar operations.

Table 22.11: Other Processing Costs

Group / Item	Average US$ M/year	Life of Mine $US M	US$/t processed
Maintenance Parts	2.2	30.3	2.21
Maintenance Services	0.2	3.0	0.22
Lubricants	0.1	1.1	0.08
Outside Services	0.1	1.5	0.11
Supplies/Tools	0.4	5.2	0.38
Liners	0.5	6.8	0.50
Grinding Media	2.9	40.8	2.98
Mobile Equipment Consumables	0.4	4.9	0.36
Total	**6.7**	**93.7**	**6.83**

Source: JDS, 2016

22.5.2 Water Treatment Plant

The water treatment plant sector includes all costs related to the operation of the water treatment plant. It is assumed that the mineral processing management and maintenance staff will also carry responsibility for the supervision and maintenance of the water treatment plant, respectively.

Table 22.12: Water Treatment Operating Cost

Item	Average US$ M/year	Life of Mine $US M	US$/t Processed
Labour	0.1	1.4	0.10
Power	0.5	6.9	0.51
Chemicals & Consumables	0.2	3.2	0.24
Maintenance Parts & Services	0.3	3.3	0.25
Total	**1.1**	**15.0**	**1.10**

Source: JDS, 2016

22.5.3 Tailing Storage Facility

TSF operating costs include the costs to perform the following activities:

- Load, transport, place, and compact dried tailing material from the tailing stockpile building to the Tailing Storage Facility ("TSF") using articulated surface haul trucks;
- Load, transport, place, and compact waste rock (dumped by underground mine haul trucks) from a staging stockpile (near the tailing stockpile building) to the TSF using articulated surface haul trucks;
- Blend binder into the processed tailing stream, as required for strength; and
- Manage the above activities (supervision and technical staff).

Table 22.13: Tailing Storage Facility Operating Cost

Area	Average US$ M/year	Life of Mine $US M	$/t Processed
Labour	0.5	7.1	0.52
Equipment Operations	2.5	34.3	2.50
Tailing Binder	0.7	9.3	0.68
Total	**3.6**	**50.7**	**3.70**

Source: JDS, 2016

22.6 General & Administration Operating Costs

Table 22.14 outlines the G&A operating costs that are considered in the economic model. They are broken out into the following categories:

- Labour; and
- Services & Expenses.

Table 22.14: General & Administration Operating Cost Summary

Sector	Average US$ M/year	Life of Mine $US M	US$/t processed
G&A Labour	**2.9**	**41.0**	**2.99**
G&A Labour	2.9	41.0	2.99
G&A Services & Expenses	**11.0**	**152.8**	**11.14**
Off-Site Contract Services	0.2	2.1	0.15
On-Site Contract Services	0.5	6.8	0.50
Logistics & Freight	1.0	14.0	1.02
Surface Infrastructure	0.7	9.7	0.71
Mobile Equipment Operation	1.3	17.5	1.27
Safety & Medical	0.5	6.9	0.50
Environmental	0.8	10.8	0.79
Security	1.0	14.6	1.06
Engineering	1.1	14.7	1.07
IT & Communications	0.3	4.2	0.31
Office Expenses	0.2	2.1	0.16
Human Resources	0.1	1.2	0.09
Titles & Associations	0.1	0.6	0.05
Insurance	2.4	33.6	2.45
Socio-Economics	1.0	13.9	1.01
Total	**13.9**	**193.8**	**14.13**

Source: JDS, 2016

22.6.1 G&A Labour

G&A staffing levels have been built up based on experience at similar operations. Labour costs are based on fully burdened staffing wage bandings. Expatriate wage scales are included for high level managerial positions.

22.6.2 G&A Services & Expenses

G&A services and expenses have been estimated in consultation with current CGI area managers, and considering other similar operations. Major items (logistics, mobile equipment, and insurance) are built up from first principles. Minor items are factored, based on other estimate parameters (such as number of staff) or are general allowances.

22.7 Corporate Management Fee

An allowance of 1% of the annual net smelter return has been allowed for the provision of corporate services to the Project, including legal, finance, treasury, and other support services. This is equivalent to US$3.0M or US$3.01/t processed.

22.8 Taxes

Value-added tax ("VAT" or "IVA") applies to goods and services provided in Colombia; however, IVA is fully refundable for operating expenses. As such, no provisions for IVA are included in the operating cost estimates.

22.9 Contingency

No operating cost contingency provision has been included in the estimate.

23 Economic Analysis

An engineering economic model has been used to collate the study results in order to estimate and evaluate project cash flows and economic viability.

The financial evaluation presents results for Net Present Value ("NPV"), Internal Rate of Return ("IRR"), and payback. The economic model allows sensitivity evaluations for changes in metal prices, grades, exchange rates, operating costs, and capital costs to determine their relative importance for evaluating investment decisions.

Section 23.1 summarizes the economic results, Section 23.2 provides the results of an economic sensitivity analysis, and Section 23.3 provides additional details relating to the various key inputs within the economic model.

23.1 Economic Results

Based on the findings of the FS, it can be concluded that the Project is economically viable with an after-tax IRR of 31.2% and an NPV of $860.2M at a 5% discount rate. Table 23.1 presents the results of the evaluated scenario.

Table 23.1: Summary of Economic Results

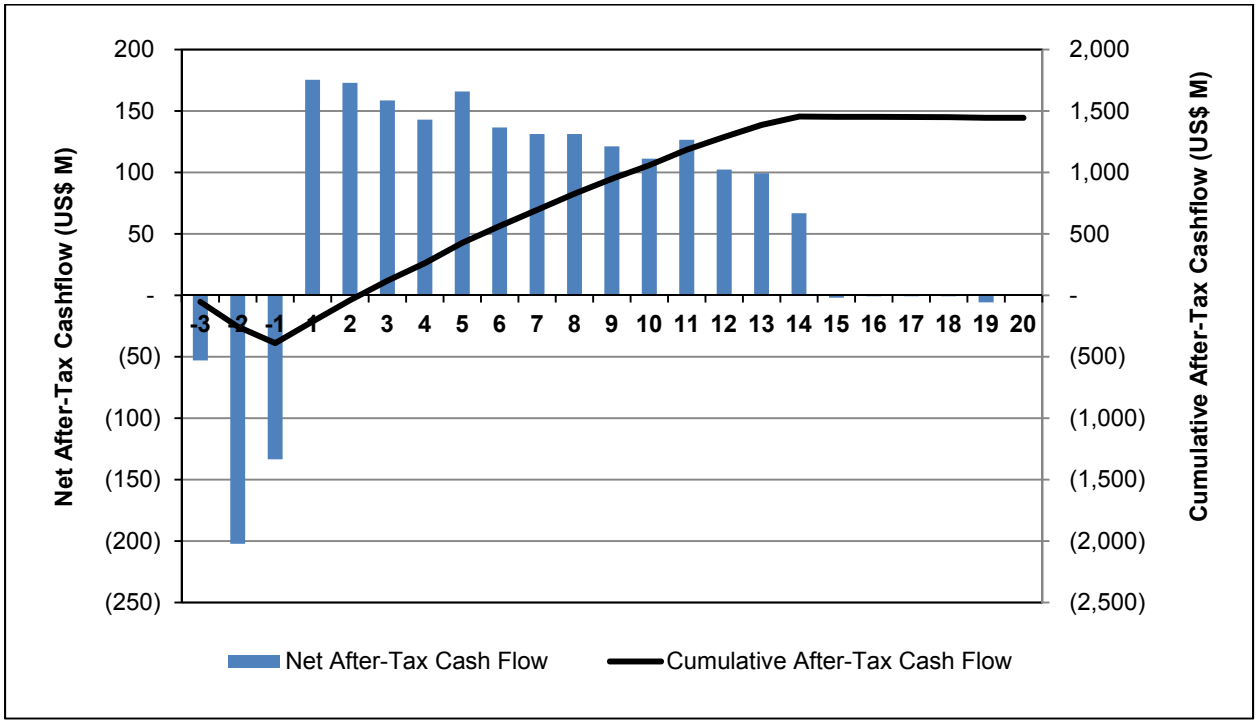
Parameter	Unit	Value
Gold Price	US$/ounce	1,200
Silver Price	US$/ounce	15.00
Exchange Rate	COP:US	2,850
Net Revenues	US$ M	4,130.9
Operating Costs	US$ M	1,378.6
Cash Flow from Operations	US$ M	2,752.3
Capital Costs	US$ M	661.7
Closure Cost (net of Salvage Value)	US$ M	10.0
Net Pre-Tax Cash flow	US$ M	2,081.0
Pre-Tax $NPV_{5\%}$	US$ M	1,263.3
Pre-Tax IRR	%	38.0
Pre-Tax Payback	years	1.9
Total Taxes	US$ M	636.2
Effective Tax Rate	%	33
After-Tax $NPV_{5\%}$	US$ M	860.2
After-Tax IRR	%	31.2
After-Tax Payback	years	2.3
Break-Even After-Tax Gold Price	US$/ounce	634

Payback is calculated on annual cash flows without considering discount rates or inflation.

Source: JDS, 2016

Figure 23.1 illustrates the projected after-tax cash flows, undiscounted.

Figure 23.1: Annual and Cumulative After-Tax Cash Flows



Source: JDS, 2016

Table 23.2 provides a LOM cost summary, and includes unit cash-costs for comparative purposes.

Table 23.2: Cash Cost Summary

Item	Parameter	LOM Cost (US$)	Unit Cost (US$/Payable Au Oz)
A	Pre-Production Capital Costs	389.2	111.6
B	Life of Mine Sustaining Capital Costs	272.5	78.1
C	Closure Cost (net of Salvage Value)	10.0	2.9
D	Operating Costs	1,378.6	395.1
E	Refining and Transportation	14.9	4.3
F	Royalties	137.2	39.3
G	Silver Credits	(96.1)	(27.5)
	Total Cash Cost, net Ag Credits *(D+E+F+G)*	**1,434.6**	**411.2**
	All-in Sustaining Cost, net Ag Credits *(B+C+D+E+F+G)*	**1,717.1**	**492.1**
	All-in Sustaining & Construction Costs, net Ag Credits *(A+B+C+D+E+F+G)*	**2,106.3**	**603.7**

Source: JDS, 2016

Table 23.3 presents a condensed annual cash flow model for the evaluated scenario.

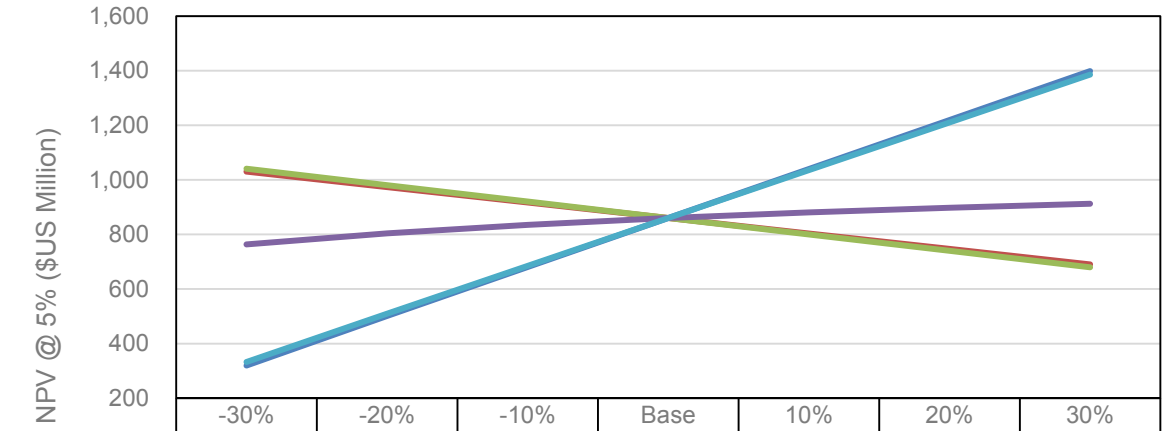
Table 23.3: Condensed Annual Cash Flow Model

Item	Unit	Pre-Production	Production	Life of Mine	Year -3	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15-20
Mine Schedule																						
Total Ore Mined	k tonne	97.2	13,619.5	13,716.8	-	-	97.2	814.1	839.6	1,078.7	1,067.6	1,102.5	1,070.4	1,103.6	1,103.9	1,102.7	990.2	1,106.4	1,061.0	826.0	352.9	-
Au Grade	g/t	6.9	8.4	8.4	-	-	6.9	12.2	11.8	8.6	7.7	8.8	8.3	8.1	8.1	7.6	7.6	7.5	7.0	7.5	8.8	-
Ag Grade	g/t	22.5	24.3	24.3	-	-	22.5	37.1	34.7	28.0	21.4	22.2	22.6	23.0	24.2	22.0	21.5	20.8	20.3	22.9	25.5	-
Processing Schedule																						
Total Ore Processed	k tonne	82.5	13,634.3	13,716.8	-	-	82.5	816.0	832.5	1,080.0	1,080.0	1,073.0	1,091.5	1,110.0	1,110.0	1,091.5	999.0	1,110.0	1,054.5	832.5	353.8	-
Au Recovery	%	94%	94%	94%	-	-	94%	92%	93%	94%	94%	95%	95%	95%	95%	94%	94%	94%	94%	94%	93%	-
Ag Recovery	%	60%	60%	60%	-	-	60%	48%	52%	59%	63%	63%	62%	64%	63%	62%	62%	63%	63%	62%	57%	-
Recovered Au	k oz	17.4	3,474.2	3,491.5	-	-	17.4	291.5	295.6	281.6	253.4	287.2	276.9	273.7	273.0	251.9	230.8	252.7	222.5	189.9	93.5	-
Recovered Ag	k oz	35.6	6,389.5	6,425.2	-	-	35.6	468.5	481.8	573.5	465.9	482.3	494.9	525.6	543.7	482.2	425.1	467.6	434.3	378.1	166.1	-
Net Smelter Return																						
Au Payable	US$ M	20.8	4,166.1	4,186.9	-	-	20.8	349.6	354.5	337.7	303.8	344.4	332.0	328.2	327.4	302.0	276.8	303.0	266.8	227.7	112.2	-
Ag Payable	US$ M	0.5	95.6	96.1	-	-	0.5	7.0	7.2	8.6	7.0	7.2	7.4	7.9	8.1	7.2	6.4	7.0	6.5	5.7	2.5	-
Refining & Assay Costs	US$ M	(0.0)	(9.0)	(9.0)	-	-	(0.0)	(0.7)	(0.7)	(0.8)	(0.7)	(0.7)	(0.7)	(0.7)	(0.7)	(0.7)	(0.6)	(0.7)	(0.6)	(0.5)	(0.2)	-
Transportation & Insurance	US$ M	(0.0)	(5.9)	(5.9)	-	-	(0.0)	(0.5)	(0.5)	(0.5)	(0.4)	(0.5)	(0.5)	(0.5)	(0.5)	(0.4)	(0.4)	(0.4)	(0.4)	(0.4)	(0.2)	-
Royalties	US$ M	(0.7)	(136.5)	(137.2)	-	-	(0.7)	(11.4)	(11.6)	(11.1)	(10.0)	(11.3)	(10.9)	(10.8)	(10.7)	(9.9)	(9.1)	(9.9)	(8.8)	(7.5)	(3.7)	-
Net Smelter Return	**US$ M**	**20.6**	**4,110.3**	**4,130.9**	**-**	**-**	**20.6**	**344.0**	**349.0**	**333.9**	**299.8**	**339.2**	**327.4**	**324.1**	**323.6**	**298.2**	**273.1**	**299.0**	**263.6**	**225.0**	**110.5**	**-**
Operating Costs																						
Underground Mining	US$/t	67.21	57.14	57.21	-	-	67.21	61.83	66.70	63.15	59.20	61.77	61.55	53.87	52.92	56.18	55.79	53.27	57.37	41.53	50.18	-
	US$ M	(5.5)	(779.1)	(784.7)	-	-	(5.5)	(50.5)	(55.5)	(68.2)	(63.9)	(66.3)	(67.2)	(59.8)	(58.7)	(61.3)	(55.7)	(59.1)	(60.5)	(34.6)	(17.8)	-
Processing	US$/t	31.99	26.12	26.16	-	-	31.99	27.71	27.54	25.82	25.97	25.90	25.79	25.72	25.46	25.30	25.82	25.08	25.41	26.57	33.27	-
	US$ M	(2.6)	(356.2)	(358.8)	-	-	(2.6)	(22.6)	(22.9)	(27.9)	(28.0)	(27.8)	(28.2)	(28.5)	(28.3)	(27.6)	(25.8)	(27.8)	(26.8)	(22.1)	(11.8)	-
General & Administration	US$/t	23.98	14.07	14.13	-	-	23.98	18.04	17.53	12.88	12.98	12.35	12.14	11.91	12.45	12.66	13.80	12.41	13.06	16.47	34.34	-
	US$ M	(2.0)	(191.8)	(193.8)	-	-	(2.0)	(14.7)	(14.6)	(13.9)	(14.0)	(13.3)	(13.3)	(13.2)	(13.8)	(13.8)	(13.8)	(13.8)	(13.8)	(13.7)	(12.2)	-
Management Fee	US$/t	2.50	3.01	3.01	-	-	2.50	4.22	4.19	3.09	2.78	3.16	3.00	2.92	2.91	2.73	2.73	2.69	2.50	2.70	3.12	-
	US$ M	(0.2)	(41.1)	(41.3)	-	-	(0.2)	(3.4)	(3.5)	(3.3)	(3.0)	(3.4)	(3.3)	(3.2)	(3.2)	(3.0)	(2.7)	(3.0)	(2.6)	(2.2)	(1.1)	-
Total Operating Costs	**US$/t**	**125.68**	**100.35**	**100.50**	**-**	**-**	**125.68**	**111.80**	**115.96**	**104.94**	**100.92**	**103.18**	**102.48**	**94.43**	**93.74**	**96.88**	**98.15**	**93.46**	**98.34**	**87.28**	**120.92**	**-**
	US$ M	**(10.4)**	**(1,368.2)**	**(1,378.6)**	**-**	**-**	**(10.4)**	**(91.2)**	**(96.5)**	**(113.3)**	**(109.0)**	**(110.7)**	**(111.9)**	**(104.8)**	**(104.1)**	**(105.7)**	**(98.0)**	**(103.7)**	**(103.7)**	**(72.7)**	**(42.8)**	**-**
Production Income																						
Operating Income	**US$/t**	**123.9**	**201.1**	**200.7**	**-**	**-**	**123.9**	**309.8**	**303.2**	**204.2**	**176.6**	**213.0**	**197.5**	**197.6**	**197.8**	**176.3**	**175.2**	**175.9**	**151.6**	**183.0**	**191.5**	**-**
	US$ M	**10.2**	**2,742.1**	**2,752.3**	**-**	**-**	**10.2**	**252.8**	**252.4**	**220.6**	**190.8**	**228.5**	**215.5**	**219.3**	**219.5**	**192.5**	**175.0**	**195.3**	**159.9**	**152.3**	**67.8**	**-**
Capital Expenditures																						
Site Development	US$ M	(10.9)	-	(10.9)	(4.5)	(6.3)	(0.1)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Underground Mining	US$ M	(86.5)	(178.3)	(264.8)	-	(35.5)	(51.0)	(42.8)	(18.2)	(15.8)	(12.5)	(9.1)	(12.5)	(15.1)	(18.4)	(9.9)	(8.9)	(6.2)	(6.0)	(2.9)	(0.0)	-
Processing Plant	US$ M	(97.6)	(11.6)	(109.2)	(3.2)	(59.4)	(35.0)	(0.9)	(0.9)	(0.9)	(0.9)	(0.9)	(0.9)	(0.9)	(0.9)	(0.9)	(0.9)	(0.9)	(0.9)	(0.9)	-	-
Tailing & Waste Rock	US$ M	(7.7)	(16.1)	(23.8)	(2.2)	(3.5)	(1.9)	(1.9)	(1.8)	(1.8)	(1.8)	(1.8)	(1.8)	(1.8)	(1.8)	(1.8)	-	-	-	-	-	-
Off-Site Infrastructure	US$ M	(10.0)	(12.7)	(22.7)	(10.0)	-	-	(1.2)	(2.4)	(1.2)	(1.2)	(1.2)	(1.2)	(1.2)	(1.2)	(1.2)	(1.2)	-	-	-	-	-
On-Site Infrastructure	US$ M	(45.3)	(18.7)	(64.0)	(12.6)	(27.2)	(5.5)	(4.2)	(5.8)	(1.8)	-	(3.6)	-	(3.3)	-	-	-	-	-	-	-	-
Project Indirect Costs	US$ M	(28.5)	(9.1)	(37.5)	(2.8)	(14.6)	(11.1)	(2.3)	(1.0)	(0.7)	(0.2)	(0.6)	(0.3)	(1.3)	(0.9)	(0.6)	(0.5)	(0.3)	(0.2)	(0.2)	(0.0)	-
Engineering & EPCM	US$ M	(27.8)	(0.6)	(28.4)	(7.5)	(15.1)	(5.3)	(0.1)	(0.1)	(0.1)	(0.1)	(0.1)	(0.1)	(0.1)	(0.1)	(0.1)	(0.1)	-	-	-	-	-
Owners Costs	US$ M	(21.8)	(4.3)	(26.1)	(2.1)	(10.0)	(9.7)	(0.4)	(0.4)	(0.4)	(0.4)	(0.4)	(0.4)	(0.4)	(0.4)	(0.4)	(0.4)	-	-	-	-	-
Taxes (VAT/IVA)	US$ M	(17.7)	(14.0)	(31.7)	(2.9)	(11.3)	(3.5)	(3.6)	(1.4)	(1.1)	(0.3)	(0.7)	(0.4)	(1.9)	(1.5)	(1.0)	(0.8)	(0.5)	(0.3)	(0.4)	(0.0)	-
Contingency	US$ M	(35.4)	(7.0)	(42.4)	(4.8)	(18.3)	(12.3)	(1.0)	(1.3)	(0.7)	(0.4)	(0.9)	(0.4)	(0.9)	(0.4)	(0.4)	(0.3)	(0.1)	(0.1)	(0.1)	-	-
Total Capital Costs	**US$ M**	**(389.2)**	**(272.5)**	**(661.7)**	**(52.6)**	**(201.3)**	**(135.3)**	**(58.3)**	**(33.2)**	**(24.5)**	**(17.7)**	**(19.3)**	**(17.9)**	**(26.8)**	**(25.6)**	**(16.3)**	**(13.0)**	**(7.8)**	**(7.5)**	**(4.5)**	**(0.0)**	**-**
Closure Costs																						
Salvage Value	US$ M	-	7.5	7.5	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	3.7	3.7
Closure & Monitoring	US$ M	-	(17.5)	(17.5)	-	-	-	-	-	-	-	(0.1)	(0.1)	(0.1)	(0.1)	(0.1)	(0.1)	(0.1)	(0.1)	(0.1)	(3.1)	(13.7)
Closure Costs Net Salvage	**US$ M**	**-**	**(10.0)**	**(10.0)**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**(0.1)**	**(0.1)**	**(0.1)**	**(0.1)**	**(0.1)**	**(0.1)**	**(0.1)**	**(0.1)**	**(0.1)**	**0.6**	**(10.0)**
Working Capital																						
Working Capital	US$ M	(7.6)	7.6	-	-	-	(7.6)	-	-	-	-	-	-	-	-	-	-	-	-	-	7.6	-
Cash Flows																						
Net Pre-Tax Cashflow	**US$ M**	**(386.6)**	**2,467.2**	**2,080.6**	**(52.6)**	**(201.3)**	**(132.7)**	**194.5**	**219.2**	**196.1**	**173.1**	**209.1**	**197.5**	**192.5**	**193.8**	**176.1**	**161.9**	**187.3**	**152.3**	**147.8**	**76.0**	**(10.0)**
Cum. Pre-Tax Cashflow	US$ M				(52.6)	(253.9)	(386.6)	(192.2)	27.0	223.1	396.2	605.3	802.9	995.3	1,189.2	1,365.3	1,527.2	1,714.5	1,866.8	2,014.6	2,090.6	2,080.6
Taxes	US$ M	(2.0)	(634.2)	(636.2)	(0.3)	(0.9)	(0.8)	(19.1)	(46.2)	(37.4)	(30.1)	(43.1)	(60.9)	(61.2)	(62.5)	(54.9)	(50.7)	(60.7)	(49.9)	(48.4)	(9.1)	-
Net After-Tax Cashflow	**US$ M**	**(388.6)**	**1,833.0**	**1,444.4**	**(52.9)**	**(202.2)**	**(133.5)**	**175.4**	**173.0**	**158.7**	**143.0**	**166.0**	**136.6**	**131.2**	**131.3**	**121.2**	**111.3**	**126.6**	**102.4**	**99.4**	**66.9**	**(10.0)**
Cum. After-Tax Cashflow	US$ M				(52.9)	(255.1)	(388.6)	(213.2)	(40.2)	118.5	261.4	427.4	564.0	695.3	826.6	947.8	1,059.1	1,185.7	1,288.1	1,387.5	1,454.4	1,444.4

23.2 Economic Sensitivities

To assess Project value drivers, sensitivity analyses were performed for the NPV and IRR. The results of this analysis are shown in Figure 23.2 and Figure 23.3. The Project proved to be most sensitive to changes in the metal pricing and gold grades, and least sensitive to changes in exchange rate.

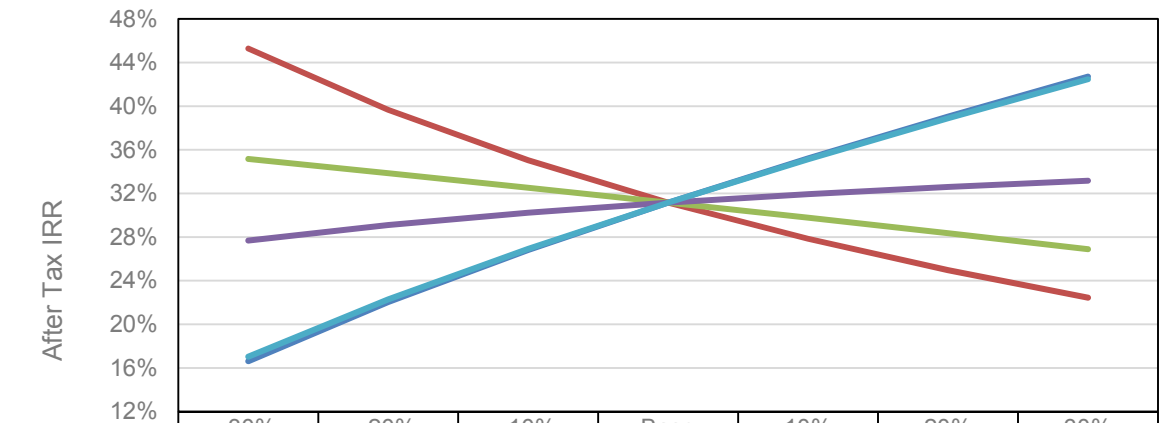
Figure 23.2: After-Tax NPV5% Sensitivities



	-30%	-20%	-10%	Base	10%	20%	30%
Price	320	501	681	860	1,040	1,219	1,398
CAPEX	1,030	974	917	860	804	747	690
OPEX	1,041	981	920	860	800	740	680
FX	764	804	835	860	881	898	912
Au Grade	333	510	685	860	1,035	1,211	1,386

Source: JDS, 2016

Figure 23.3: After-Tax IRR Sensitivities

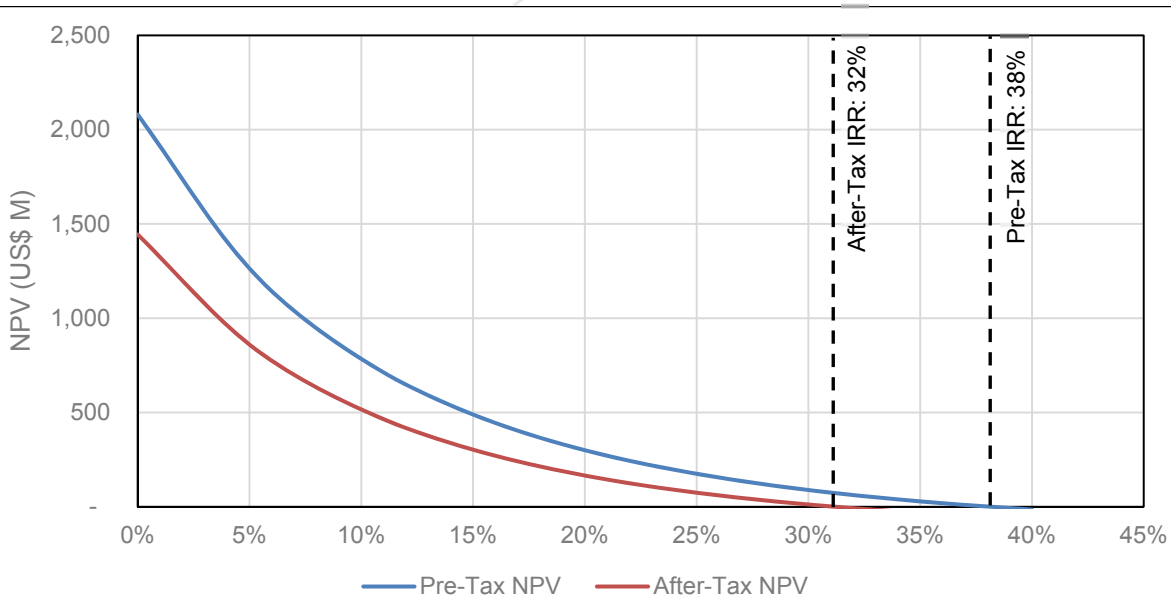


	-30%	-20%	-10%	Base	10%	20%	30%
Price	16.6%	22.0%	26.8%	31.2%	35.2%	39.1%	42.7%
CAPEX	45.3%	39.7%	35.0%	31.2%	27.8%	25.0%	22.4%
OPEX	35.2%	33.9%	32.5%	31.2%	29.8%	28.3%	26.9%
FX	27.7%	29.1%	30.2%	31.2%	31.9%	32.6%	33.2%
Au Grade	17.0%	22.3%	26.9%	31.2%	35.1%	38.9%	42.5%

Source: JDS, 2016

Figure 23.4 presents the pre-tax and after-tax NPV profile for the Project, showing the sensitivity of discount rates against the NPV.

Figure 23.4: Discount Rate Sensitivity on NPV



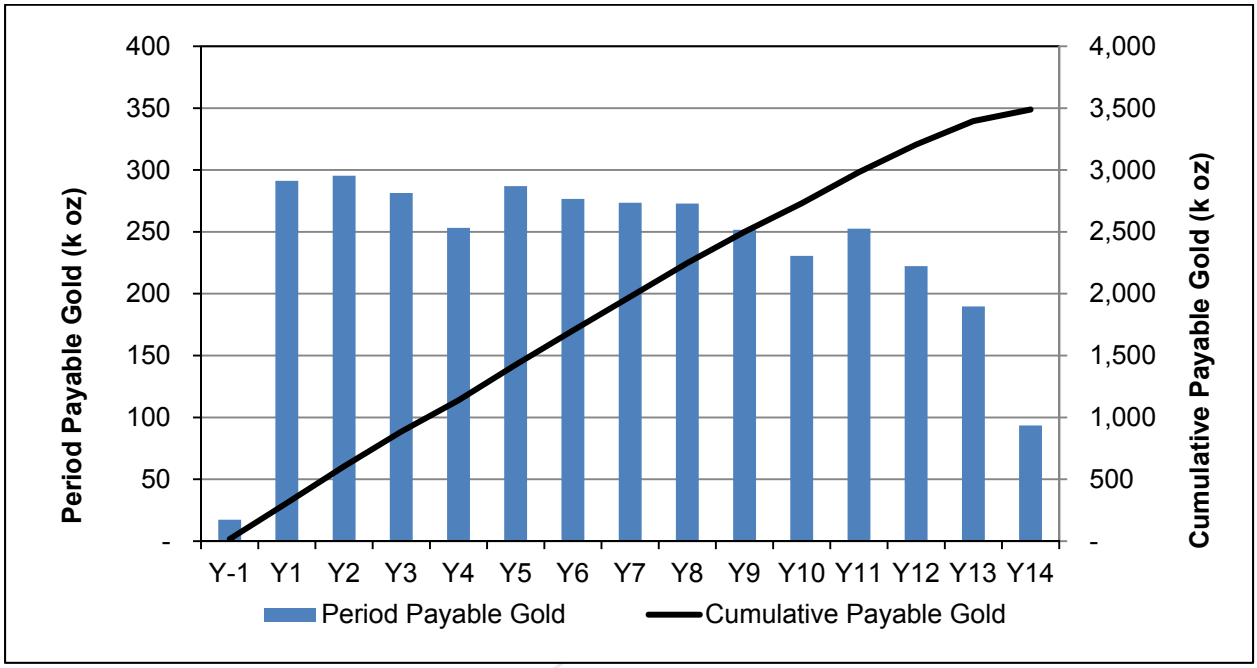
Source: JDS, 2016

23.3 Economic Model Inputs

23.3.1 Metal Production

Mining and milling production is based on the respective mine and plant schedules, and recovery parameters determined through metallurgical testing. Figure 23.5 shows the gold production schedule over the LOM.

Figure 23.5: Gold Production Schedule



Source: JDS, 2016

23.3.2 Revenues and NSR Parameters

23.3.2.1 Revenues

Annual revenue is determined by applying estimated metal prices to the annual payable metals estimated for each operating year. Sales prices have been applied to life of mine production without escalation or hedging. The revenue is the gross value of payable metals before refining charges and transportation charges. Metal sales prices used in the base case evaluation are US$1,200/ounce for gold and US$15.00/ounce for silver.

23.3.2.2 NSR Parameters

Table 23.4 presents the NSR parameters used in the model. As mentioned in Section 19, no contractual arrangements for shipping or refining exist at this time; however, the refining terms have been validated by third party subject area experts and deemed appropriate.

Table 23.4: Smelter Terms

Parameter	Unit	Value
Payable Gold	%	99.9
Payable Silver	%	99.7
Refining Charge	US$/ounce	0.83
Assay Charges	US$/shipment	604
Transportation Costs		
Assumed Shipments per Year	# shipments	24
Basic Shipment Charge	US$/shipment	1,830
Helicopter Charge	US$/shipment	2,043
Weight Charge	US$/kg	9
Customs Fees	US$/shipment	564
Insurance Costs		
Insurance & Safekeeping	US$/$1000	0.34

Source: JDS, 2016

23.3.3 Royalties

The Project is subject to royalty payments, which have been considered in the economic evaluation. Royalty payments are calculated at 3.20% of the value of the recovered metal; the life of mine royalty payments are estimated to be US$137.2M.

23.3.4 Capital & Operating Costs

Section 21 and Section 22 of this report present the details and basis of the capital and operating cost estimates, respectively. Each of the cost estimates were aligned to the construction and production schedules to produce the respective cash flows.

23.3.5 Salvage Value

Much of the capital equipment brought to site will have some resale value even at the end of mine life. Table 23.5 presents a summary of the purchase price of the equipment and the expected resale value after considering the costs of disassembly. These costs are included as a credit to the Project at the end of the mine life (Year 14 & 15).

Table 23.5: Salvage Value Estimate

Sector	Capital Costs (US$ M)	Estimated Residual Value	Cash Value (US$ M)
Underground Mining Equipment	81.4	5%	4.1
Processing Equipment (1)	10.1	10%	1.0
Paste Backfill Equipment	1.8	5%	0.1
Power Generation Station	1.2	25%	0.3
Ancillary Buildings	8.0	2%	0.2
Assay Lab Equipment	0.9	10%	0.1
Effluent Treatment Plant	17.1	5%	0.9
Aerial Tram Components	7.1	5%	0.4
Surface Equipment Fleet	10.5	5%	0.5
Total	**138.2**		**7.5**

Note 1: Includes only select equipment: crushers, grinding mills, and pressure filters

Source: JDS, 2016

23.3.6 Reclamation & Closure

Reclamation and closure activities are described in Section 20. Table 23.6 summarizes the closure costs within the model. Progressive reclamation activities (the TSF soil cover) occur between Year 5 and Year 14. Demolition and closure activities occur at the end of Year 14 and into Year 15. Ongoing monitoring and maintenance activities are largely incurred between Year 15 and Year 20, with the exception of ongoing water treatment, which has been assumed to continue in perpetuity.

Table 23.6: Reclamation & Closure Cost Summary

Category	Total Cost (US$ M)	%
Progressive Closure *(activities occurring during operations)*	0.8	5
Demolition & Closure	8.6	49
Ongoing Monitoring & Maintenance	8.0	46
Total	**17.5**	**100**

Source: JDS, 2016

23.3.7 Working Capital

A US$7.6M working capital allowance for the purchase and storage of consumables inventory is assumed incurred once the Project begins processing ore (end of Year -1). This value is equivalent to approximately one month of total operating costs. Working capital is recaptured at the end of the mine life and the final value of the account is $0.

23.3.8 Taxes

The tax calculations in the financial model are based on the current tax laws, most notably, Colombian Tax Reform Law 1739 of December 23, 2014.

Table 23.7 presents the basis of calculation for the various Colombian taxes, as well as the total tax paid by the Project within each category.

Table 23.7: Payable Taxes & Basis

Tax Category	Tax Rate & Basis Applied	Total Tax Paid (US$ M)
Corporate Income Tax (CIT)	25.0% of taxable income Equipment VAT paid in the pre-production phase is applied as a tax credit in Year 1 Equipment VAT paid in the operation phase is applied as a tax credit in the year the equipment is purchased	454.2
Equality Tax (CREE)	9.0% of all taxable income The following CREE premiums apply for incomes above US$280,000: 8.0% in Year -3 9.0% in Year -2 The CREE premium is scheduled for termination in Year -1 (2019) Tax credit carry forward (before construction) are applied as a tax credit in Year 1	173.8
Financial Transactions Tax	0.4% of all refining, OPEX, pre-production CAPEX, and sustaining CAPEX costs.	8.2
Wealth Tax	Wealth Tax is not included in the tax model, as (per Law 1739) Wealth Tax will be dissolved in 2018 (prior to the start of Operations).	-
Total Taxes Paid		**636.2**

Note 1: "Total Tax Paid" is net of all applicable tax credits

Source: JDS, 2016

24 Adjacent Properties

There are no adjacent properties to those held by CGI that are considered relevant to this technical report.

25 Other Relevant Data and Information

25.1 Project Execution & Development Plan

25.1.1 Introduction

The purpose of the Buriticá Project Execution Plan ("PEP") is to describe the project development strategies that were considered for the FS capital cost estimate and project schedule, and to provide the future framework for organizing the engineering, procurement, and construction phases.

The PEP for the FS is based on the following principles:

- Promote safety in design, construction, and operations to succeed;
- Use fit-for-purpose designs, constructions, and operations;
- Establish permanent infrastructure early, to the extent practical, to minimize costs of temporary construction facilities;
- Negotiate contracts with suppliers, contractors, and engineers with proven track records in Latin American mine developments; and,
- Eliminate surplus management overhead and project oversight.

25.1.2 Project Execution Locations

CGI currently operates a corporate office in Medellín. It is not expected that any significant volume of project work will be performed in this office; however, it will be used as a hub for project meetings as required. Detail design will be performed by firms specializing in engineering for mining and milling project. The work will largely be performed outside Colombia. An EPCM contract will be awarded to oversee engineering, procurement and construction management.

In the field, CGI will initially maintain a small office in Santa Fe during the start of project development until infrastructure is available at site. Also Santa Fe will be used to house workers during construction. The proximity of Buriticá to the Project site will make it a source of labour for mine operations and a place to source and house construction labour as well. Starting in late 2017, the centre of activity will shift to the Project Site as major construction activities kick-off and major procurement ramps down. The project strategy is to have the majority of personnel based at the Project site to avoid the need for large satellite offices.

25.1.3 Project Development Schedule Overview

A resource-loaded level 3 project schedule was developed for the Project, using the capital cost estimate as the basis for on-site man-hours to establish activity durations.

Table 25.1 presents a summary (level 1) schedule for the development of the Buriticá Project.

The critical path for the Project runs through the detailed engineering and construction activities related to the processing facilities. Other near critical activities include construction of Phase 1 (initial access) of the Main Access Road, site preparations (earthworks and temporary facilities), and the water treatment plant installation.

Table 25.1: Project Summary Schedule

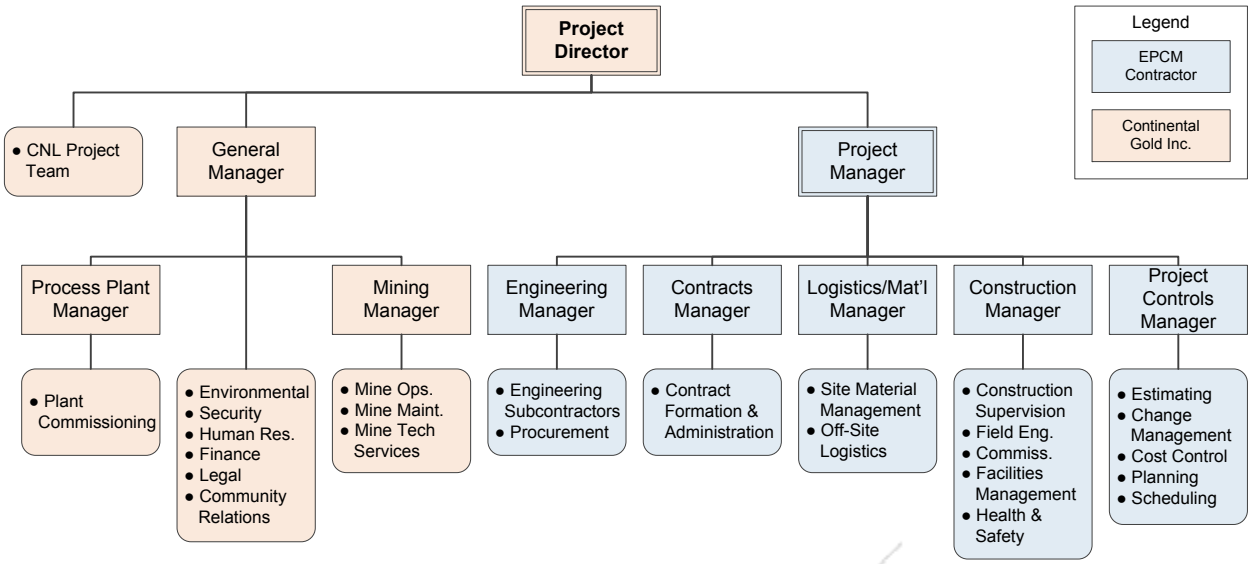
Activity	'16	2017				2018				2019			
	Q4	Q1	Q2	Q3	Q4	Q1	Q2	Q3	Q4	Q1	Q2	Q3	Q4
Detailed Engineering													
Site Earthworks & Tailing Facility													
Mine Infrastructure													
Process and Infrastructure													
Major Procurement													
Engineering Contract Formation													
Construction Contract Formation													
Major Equipment Procurement													
Major Equipment Fab/Delivery													
Construction													
13.8kV Power Line													
Site Access Road													
Tailing & Waste Rock Facility													
Contact Water Treatment Plant													
Plant Site Earthworks													
Ancillary Buildings													
110kV Power Line													
Process Facilities													
Paste Plant													
Underground Mine Development													
Commissioning													
Plant Commissioning & Ramp-Up													
Commercial Production (1-Sep-19)												◆	

25.1.4 Project Management

25.1.4.1 Organization & Responsibilities

The Project Management Team (PM Team) will be an integrated team comprised of the Owners personnel, the EPCM Contractor, and various engineering sub-contractors. The PM Team will oversee and direct all engineering, procurement, and construction activities for the Project. Figure 25.1 presents a representative preliminary organization chart.

Figure 25.1: Preliminary Project Management Team Organization Chart



Senior Project Management

Overall delivery of the Project to the defined metrics will be the responsibility of the CGI Project Director. The Project Director will provide high level direction to the project management (PM) Team, with support from the EPCM Contractor and the Owner's Pre-Operational team to manage Project activities.

The EPCM Project Manager will be responsibility for the execution of Project activities, including detailed engineering, procurement, logistics, construction, commissioning, and Project Controls.

Owners Operations Team

A portion of the Owner's Operations team will be mobilized during the Project development phase for functions required over the life of mine (i.e. not limited to construction support):

- Mining operations, including maintenance;
- Environmental;
- Security;
- Accounting;
- Community Relations;
- Human Resources;
- Site Services (EPCM Construction Manager responsible during construction); and
- Site Purchasing (EPCM Materials Manager responsible during construction).

Engineering Team

The Engineering Manager will oversee, coordinate, and integrate Engineering activities. The engineering team will consist of various engineering sub-contractors, who will develop the detailed designs and specifications for the Project, and then transition to the field to provide Quality Assurance ("QA"), field engineering, and commissioning support.

Procurement Team

The EPCM Engineering Manager will oversee and manage Procurement activities undertaken by engineering contractors (formation and administration of engineering and construction contracts will be overseen and managed by the EPCM Contracts Manager). The procurement/logistics team will use the prepared engineering design packages to obtain competitive tenders, and secure vendors and construction contractors to provide the appropriate goods and services.

Logistics Team

The Logistics/Materials Manager will oversee and coordinate all logistics activities. The logistics team will determine and coordinate the best methods for the movement of materials, equipment, and people to, from, and at the Project site.

Construction Management Team

The Construction Manager will lead construction management team ("CM Team") and be responsible for construction safety, progress, and quality. The CM Team will coordinate and manage all site activities to ensure construction progresses on schedule and within budget.

Commissioning Team

The Commissioning Manager will oversee the commissioning team, and be responsible for the timely handover of process and infrastructure systems to the Owner once construction activities have been substantially completed. The commissioning team will be supported by discipline engineering resources to complete pre-commissioning activities and obtain technical acceptance and transfer care, custody, and control of completed systems to the Owner.

Project Controls Team

The Project Controls Manager will oversee the Project Controls team, and be responsible for the development, implementation, and administration of the processes and tools for project estimating, cost control, planning, scheduling, change management, progressing, and forecasting.

25.1.4.2 Project Procedures

During the Project setup phase (immediately upon project approval), a Project Procedures Manual will be developed, which will outline standard procedures for site construction. This document will focus on the interfacing between the Owner, the EPCM, and engineering contractors, and address delegation of authority, change management, procurement workflows, quality assurance, and reporting standards.

25.1.5 Engineering

25.1.5.1 Engineering Execution Strategy

The general engineering execution strategy for the Project will be to utilize multiple engineering firms with specialized knowledge of their assigned scope. Coordination of engineering interfaces and overall management of engineering schedule and deliverables will be the responsibility of the EPCM Engineering Manager. The following major engineering contract packages have been identified for the Project:

- Detailed engineering & procurement of process facilities and select on-site infrastructure and field engineering support;
- Detailed engineering of tailing and waste rock storage facility and associated water diversion structures;
- Site access road engineering support;
- Water treatment plant design;
- Hydrological characterization;
- Geochemical analysis;
- Water balance design;
- Power transmission line detailed design; and
- Paste plant process design.

25.1.5.2 Engineering Management

Baseline Engineering Data

Engineering data from the FS, including (but not limited to) design criteria, flow sheets, material take-offs, and drawings are considered the engineering baseline data, and form the basis for the capital cost estimate and schedule. Deviations from these baseline engineering inputs, beyond clarifying and finalizing scope, and detailing of designs will be subject to the project change management processes.

Design Criteria Approval

The Project critical path includes timely completion of engineering activities. To prevent delays or late changes in engineering deliverables and to keep efforts focused, a formal engineering approval procedure will be enacted for the Project.

Engineering Progress & Performance Monitoring

Each engineering contractor will provide a deliverables list as part of their services proposal. Deliverables (and their associated budgets) will be grouped into logical Engineering Work Packages (EWPs), which will be used as the metric for tracking engineering progress for the Project.

25.1.6 Procurement & Contracting

25.1.6.1 Procurement Execution Strategy

The general procurement execution strategy for the Project will involve utilizing known suppliers, with a preference for local or regional suppliers and construction contractors. The engineering subcontractor Procurement Manager (under the direction of the EPCM Engineering Manager) will have overall responsibility for the majority of pre-purchased procurement and contract formation activities. Contract administration will be the responsibility of the EPCM Contracts Manager on site.

25.1.6.2 Construction Contracting Strategy

Table 25.2 presents a listing of the major contract packages identified for the Project. For the purpose of the Feasibility Study, all mechanical, piping, electrical, and instrumentation ("MPEI") works have been identified as performed by a single entity within the Project estimate and schedule. During project execution, a minimum of two MPEI contractors will be engaged to avoid reliance on the performance of a single entity. During contractor pre-qualifications, if multi-discipline contractors cannot be sourced, then a horizontal contracting strategy will be employed (separate contractors for each trade, i.e. mechanical, piping, electrical, and instrumentation).

The strategy for the underground mining activities on the Project is to use Owner forces, with select seconded trainers from contract labour providers.

Table 25.2: Major Construction Contracts (Capital Phase)

PBS	Contract	Estimated Man-Hours	Estimated Value ($M US)	Contract Type
CC001	Bulk Earthworks (Plant Site Area)	206,000	7.8	Unit Rate
CC002	Tailing Facility & Diversion Channel Construction	80,000	4.5	Unit Rate
CC003	Main Access Road Construction	165,000	10.0	Unit Rate
CC005	Potable Water Well Drilling	8,000	0.3	Unit Rate
CB001	Concrete Installations	544,000	20.9	Unit Rate
CE001	LV Power Transmission Line Installation	3,000	0.1	Lump Sum
CE002	HV Power Transmission Line Installation	41,000	-	Lump Sum
CE003	Underground Electrical Distribution	3,000	0.2	Lump Sum
CA001	Plant Site Architectural Works	70,000	4.3	Lump Sum
CS001	Structural Steel Supply & Erection	145,000	10.7	Unit Rate
CS002	Ancillary Buildings Supply/Install	136,000	10.4	Lump Sum
CG001	Mechanical/Piping/Electrical/Instrumentation #1	876,000	32.0	T&M
CG002	Mechanical/Piping/Electrical/Instrumentation #2			
CG003	Pre-Operational Commissioning Support	10,000	1.0	T&M

Overall Note: man-hours include direct and indirect personnel, for the construction (capital) phase only

Note 1: The HV power line is provided by the electricity provider and repaid during operations as an amortization fee

25.1.6.3 Procurement Schedule & Critical Activities

The major procurement phase of the Project will occur in 2017 and early 2018. Procurement activities will be prioritized to schedule critical items, both due to fabrication/delivery time of the equipment (such as the grinding mill package and underground mining equipment), and due to the necessity to obtain certified vendor data to complete structural and foundation designs. Table 25.3 presents the ten longest lead packages for the Project.

Table 25.3: Long Lead Purchased Equipment

Pkg	Description	Estimated Value ($M US)	Estimated Lead Time (ARO, weeks)
PM006	Grinding Mill Package	7.0	67
PM015	Thickeners Package	3.1	45
PM028	Paste Mixer	0.1	40
PM029	Paste Pump	1.3	40
PM025	Oxygen Plant	0.8	37
PM001	Primary Crusher	0.4	35
PM008	Cone Crusher	0.3	35
PM016	Deaerator Package	3.4	35
PM024	Lime Slaker (Vertimill)	0.3	35
PM002	Apron Feeders	0.9	34
PM301	Underground Equipment – Loaders & Trucks	11.5	34
PM302	Underground Equipment – Drills & Bolters	4.7	34

Tendering and award of the following packages are considered time critical:

- Main access road construction;
- Detailed engineering packages, particularly the Tailing Storage Facility, and Processing and Infrastructure packages;
- Water treatment plant;
- Site earthworks;
- Concrete installations; and
- Tailing facility construction.

25.1.6.4 Selection of Suppliers & Contractors

A competitive bidding process will be applied to achieve the best commercial and technical results from the procurement effort. During the project setup phase, any preferred vendors will be identified and sole source strategies implemented into the procurement plan. Local involvement will form part of the bid evaluation scoring criteria in order to give preference for suppliers and contractors in Colombia (specifically the Antioquia area).

The level of vendor quality surveillance/inspection (VQS) required for each package will be established during bid evaluations, and will be determined by evaluating a supplier's ability to achieve suitable quality according to specifications and project quality assurance requirements.

25.1.7 Logistics & Material Management

25.1.7.1 Logistics Execution Strategy

The general logistics strategy for the Project is as follows:

- Ensure expediting activities achieve the Project schedule requirements;
- Manage freight movement on a global basis to maximize leveraging the freight tonnage/volume to optimize cost associated with the movement of freight; and
- Identify and optimize various aspects such as logistics, customs clearance and local content.

25.1.7.2 Shipping Routes

International freight will be shipped into the Mamonal Port in Cartagena. From the port, goods will be transported 800 km via road (an estimated travel time of 18 hours). Domestic goods are expected to originate within a 700 km radius of Buriticá, based on the distance to the capital city of Bogotá (700 km). All road transport is on paved road, with the exception of the 3.5 km site access road, which will be constructed with a gravel surface until year 2 of operations.

25.1.7.3 Freight Quantities

Table 25.4 presents the estimated international and domestic freight quantities for the Project.

Table 25.4: Major Construction Milestones

Grouping	International		Domestic	
	Loads	Tonnes	Loads	Tonnes
Pre-purchased equipment	218	4,360	513	10,680
Contractor equipment & materials	-	-	630	14,200
Misc. Field Packages	-	-	65	1,300
Mining Consumables	-	-	220	5,300
Diesel Fuel	-	-	630	12,500
Total	**218**	**4,360**	**2,058**	**43,980**

25.1.7.4 Shipping Constraints

The following considerations, based on a detailed road survey between the Mamonal Port and Buriticá, will be made in the design of shipping units for the Project, or when considering any pre-assembly opportunities.

- Width: 5.90 m;
- Height: 4.85 m;
- Maximum length: 18.29 m; and
- Maximum weight: 60 t including truck & trailer.

Early in the construction phase of the Project, before the Main Access Road is widened and graded to design, small quantities of freight will need to be mobilized to the Project, including temporary facilities and the mobile aggregate crusher. Transportation of this freight on the Main Access Road may require the assistance of tracked equipment.

International freight will be containerized to the greatest extent possible to reduce port and yard handling fees, and to expedite offloading at site.

25.1.7.5 Pre-Assembled Equipment

Pre-assembly strategies reduce overall site man-hours and the associated indirect costs, but require more careful engineering and logistics planning. The following goods have been identified for pre-assembly within the FS estimate:

- Electrical houses;
- Water treatment plant;
- Fuel tanks;
- Water and process tanks (up to 5m diameter);
- Fuel loading/unloading station;
- Conveyors (shipped in pre-fabricated lengths); and
- Transfer towers, braced frames, and stair towers.

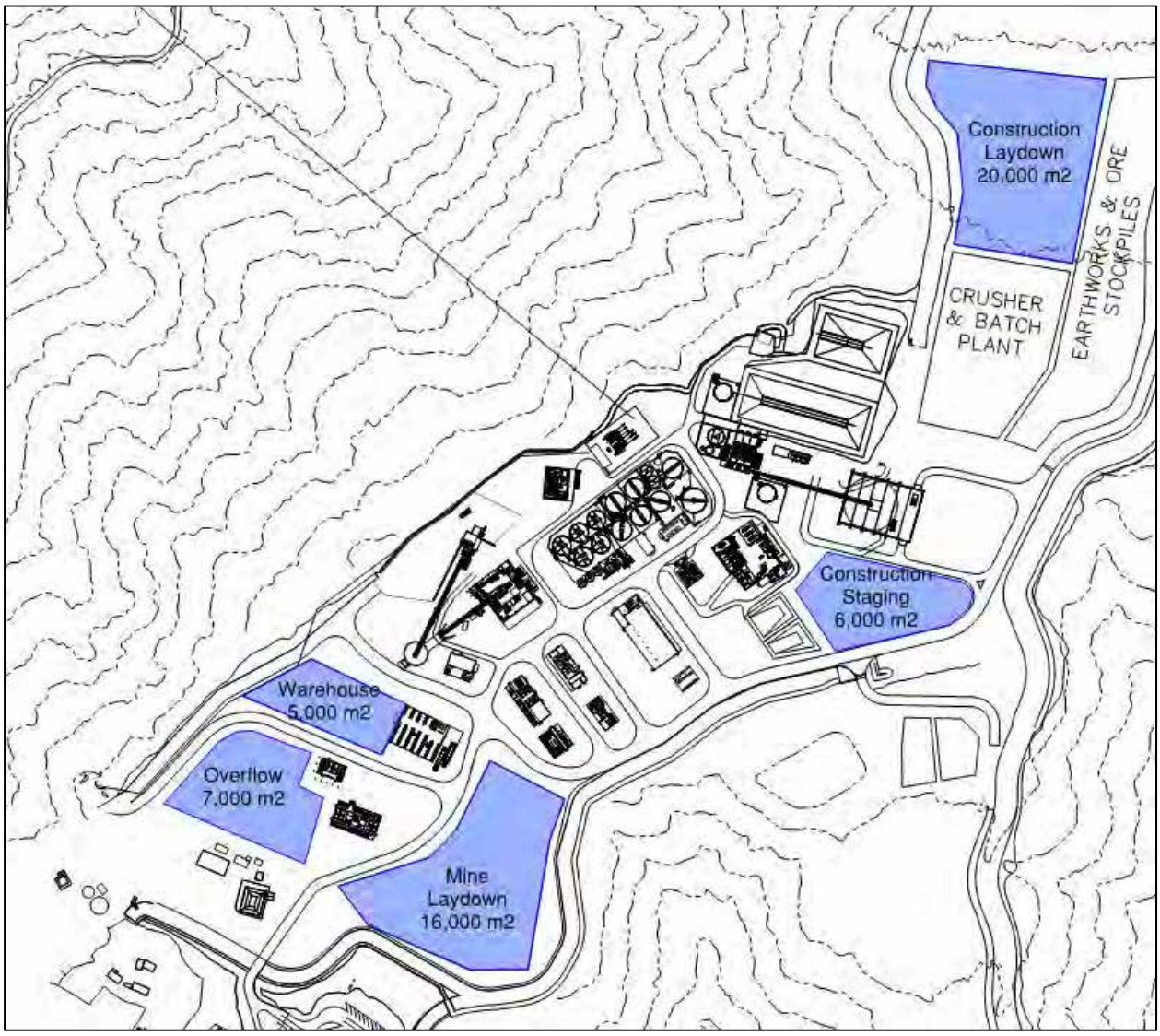
25.1.7.6 Site Materials Strategy

The general strategy for site materials control is as follows:

- Control and supervise materials movement at site through materials/inventory control from receiving, preservation, inventory and free-issue to contractor to meet the project requirements for equipment and materials procured by the EPCM or CGI (process equipment, as an example).
- Leverage contractor methods and procedures for receipt, storage, and retrieval of materials procured within their scope of work.
- Utilize a common labour pool for warehouse and laydown staff (equipment operators & labourers) for the management and movement of freight, except for items requiring special handing or rigging (such as structural steel).

- Utilize a single temporary warehouse to be used for the receipt and storage of all equipment requiring climate controlled indoor storage. Equipment and material that do not require climate controlled storage will be stored in laydown areas within the construction site. Use of sea containers and/or temporary shelters will be required to store goods that need to be protected from the environment. Figure 25.2 presents the available site laydown areas.

Figure 25.2: Site Laydown Areas



25.1.8 Construction

25.1.8.1 Construction Execution Plan Overview

The main objectives of the construction execution strategy include:

- Execute all activities with a goal of Zero harm to people, assets, the environment, or reputation;
- Strive to eliminate process, operational and maintenance safety hazards;
- Meet or exceed environmental regulatory and permit requirements to minimize impact;
- Deliver a high-quality facility that meets or exceeds the defined project goals;
- Establish and maintain a high level of motivation by providing a positive working environment for all personnel;
- Identify and remove barriers that affect project progress;
- Cultivate an atmosphere of positive social impact in the surrounding communities; and
- Identify outstanding achievements during construction and commissioning of the Project.

The path of construction for the Buriticá Project is driven by two main critical paths:

- Key infrastructure development early in the schedule to support the start of pre-production mining activities for the development of underground workings and ore production; and
- Concurrent construction of the processing and ancillary areas to allow early operations.

The overall construction duration from the start of the Main Access Road construction to the declaration of commercial production is 35 months.

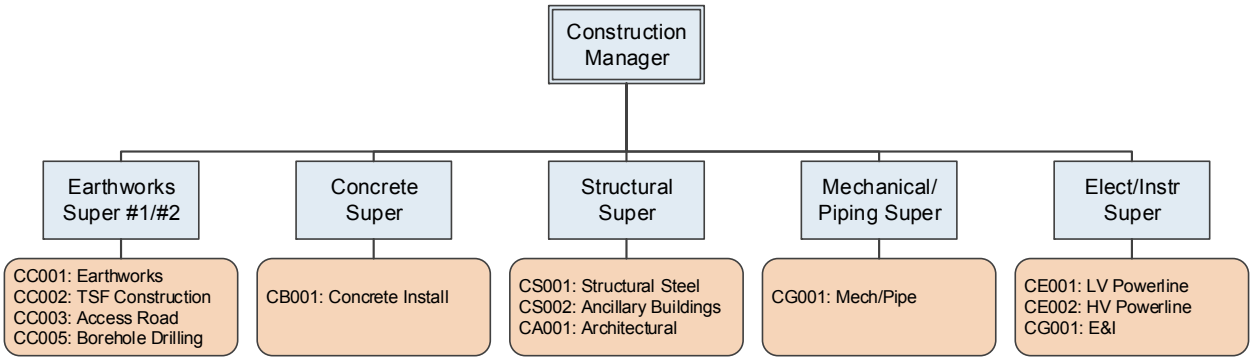
25.1.8.2 Site Management

During the construction Phase, the EPCM Project Manager (or his designate) will be responsibility for the surface construction areas. The EPCM Site Manager and CGI General Manager will closely coordinate site activities, and responsibilities will be separated for areas such as the underground mine.

25.1.8.3 Construction Management

The EPCM Construction Manager will be responsible for construction contractors oversight. Figure 25.3 shows the construction management responsibility for the EPCM trade superintendents.

Figure 25.3: Construction Management Responsibilities



25.1.8.4 Safety Management

A comprehensive Safety Management Plan ("SMP") will be developed prior to site mobilization. The SMP will address overall safety policies, procedures, and standards for the Project, including standard operating practices and emergency response plans.

25.1.8.5 Quality Management

Construction quality will be managed through the implementation of a Site Quality Management Plan ("SQMP"), which will detail the site quality management systems to be used for all construction activities. The SQMP encompasses all activities of the Project, including design, procurement and construction. Site QA is the responsibility of the EPCM Field Engineering team, and is verification that QC is being performed by the contractor, subcontractor, laboratory and third party inspection services.

25.1.8.6 Construction Quantities

Table 25.5 presents the estimated major commodity quantities for the Project. Quantities are based on the Feasibility Study engineering take-offs and capital estimate.

Table 25.5: Major Construction Commodities & Man-Hours

Discipline	Commodity Quantity	
	UOM	Quantity
Main Access Road Earthworks	km	5.7
Power Line Installations	km	3.5 (LV) + 33 (HV)
Bulk Earthworks (including TSF)	m3	631,000
Liner Installations (HDPE, Geotextile, GCL, Geonet, & Geogrid)	m^2	285,000
Site-Wide Concrete (including lean-concrete)	m3	26,400
Structural Steel (including process buildings)	Tonne	2,600
Cladding (not including ancillary buildings)	m^2	13,400
Ancillary Buildings	m^2	7,500
Mechanical	# tagged equip	466
Piping (Process & Utility)	m	27,700
Piping (Overland HDPE)	km	13.2
Cable Tray	km	105
Power & Control Cable	km	10

Note: Piping and electrical quantities above do not include the WTP or Paste Plant – only monetary allowances were made in the piping and electrical estimates for these areas.

25.1.8.7 Construction Milestones

Table 25.6 presents the major construction milestones for the Project.

Table 25.6: Major Construction Milestones

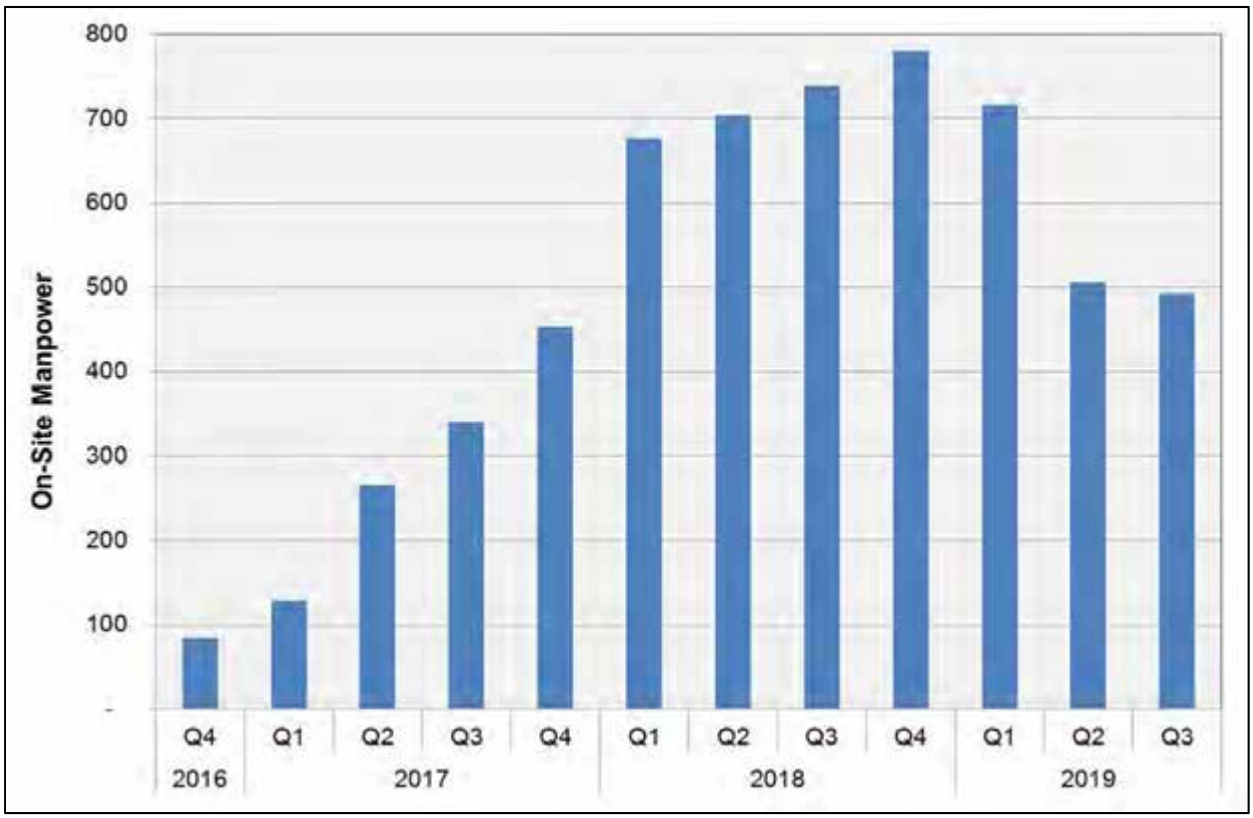
Milestone	Date
Begin Main Access Road Construction	1-Nov-16
Site Accessible by Wheeled Equipment *(begin bulk cut/fill activities at the TSF and plant area; mobilize crusher)*	21-Mar-17
Site Accessible by Tractor Trailers *(allows heavy equipment deliveries)*	8-Aug-17
Begin Process Plant Construction *(Concrete Installations)*	1-Sep-17
Complete Tailing Storage Facility	17-Dec-17
Commission Water Treatment Plant	25-Sep-17
Begin Underground Mining	1-Jan-18
Mechanically Complete Paste Plant	4-Jun-19
Mechanically Complete Process Facilities	9-Jun-19
Commercial Production Declared	1-Sep-19

25.1.8.8 Construction Resourcing

Site Manpower Loading

The construction workforce will average approximately 750 personnel through the peak (around 12 months). Figure 25.4 presents construction manpower during construction (extracted from the resource-loaded schedule).

Figure 25.4: Construction Manpower Summary



25.1.9 Commissioning

25.1.9.1 Commissioning Methodology

Progressive commissioning for the Buriticá Project will be performed by subsystem. A system will be defined as a logical grouping of equipment or systems that can be placed in service more or less by itself and that contribute to a common purpose or functionality. Wherever possible, facilities will be commissioned early in the development schedule (as in the case of ancillary buildings, water treatment plant, etc.) and be turned over to CNL for ownership and operation. A detailed Commissioning Plan will be developed during detailed engineering.

25.1.9.2 Commissioning Safety & Training

The Health, Safety, and Environmental Plan (HSE Plan) developed during execution will address specific safety procedures that will apply during the commissioning stage of the Project. The commissioning and turnover phase presents significant and unique safety risks. A comprehensive lock-out tag-out program is an effective control to manage these risks.

25.1.9.3 Commissioning Stages

- *Construction Release (Stage 1):* Construction contractor completes a system subject to agreed punch list items.
- *Pre-Operational Equipment Testing (Stage 2):* Energize and test individual equipment within subsystems to ensure functionality; includes equipment functionality tests controlled by the Plant Control System (signed off loop diagrams).
- *Pre-Operational Systems Testing (Stage 3):* Systems tested with water, air and insert materials, and capable of continuous and safe operation with all instrumentation connected, the control system operational, and all interlocks functional.
- *Ore Commissioning (Stage 4):* Plant ready to accept ore and all operating and maintenance staff are fully trained to operate and maintain the plant; individual systems operate successfully under load for a defined period of time.
- *Ramp-Up (Stage 5):* Increase ore feed design throughput rate.

26 Interpretations and Conclusions

Results of this Feasibility Study demonstrate that the Buriticá Project warrants development due to its positive, robust economics.

It is the conclusion of the QPs that the FS summarized in this technical report contains adequate detail and information to support a feasibility level analysis. Standard industry practices, equipment and design methods were used in this Feasibility Study and except for those outlined in this section, the report authors are unaware of any unusual or significant risks, or uncertainties that would affect Project reliability or confidence based on the data and information made available.

For these reasons, the path going forward must continue to focus on obtaining the environmental permit modification approval, while concurrently advancing key activities that will reduce project execution time.

Risk is present in any mineral development Project. Feasibility engineering formulates design and engineering solutions to reduce that risk common to every Project such as resource uncertainty, mining recovery and dilution control, metallurgical recoveries, political risks, schedule and cost overruns, and labour sourcing.

Potential risks associated with the Buriticá Project include:

- **Environmental Permit Modification –** The environmental permit modification, not yet approved, is of paramount importance, and further delays will increase project execution time. Without the permit modification approval, the Project cannot proceed and failure to secure the necessary permits could stop or delay the Project. In place is a project design that gives appropriate consideration to the environment and local people together with the development of close relationships with local communities and government to support a thorough Environmental and Social Impact Assessment.
- Unauthorized mining activity.
- **Reserve** - A potential unknown identified by the QP is the extent of unauthorized and illegal mining activities in the Yaraguá and Veta Sur areas. Unauthorized mining activities have been discovered, on occasion, adjacent to Continental's underground workings in the Yaraguá and Veta Sur deposits. In late July 2015, CGI surveyed the extent of unauthorized mine workings in areas that were accessible. These areas are excluded from the Mineral Reserves; however, the extent of impact in the upper areas of the reserve may not be fully quantified.
- **Management of Unauthorized Mining Activities** – The unauthorized mining activity near the Project has the potential to disrupt and delay construction activities, and to deplete the resource. The government of Colombia has the responsibility and has committed to address and manage this situation including actions to control these illegal activities.
- **Groundwater** - Groundwater below the Higabra level has been modeled, but with some uncertainty. The extent, to which the inflow estimates are realized, required that the development plan addresses mitigating variation. Increases in the actual amount of groundwater encountered would impact development costs. Drilling for drainage, and operational definition drilling included in the mine plan will help to identify specific water bearing zones with higher than expected flows and establish control and/or management procedures. As well, initiating certain development earlier in the mine life to allow more time for dewatering may prove cost effective.

- **Stability of natural slopes** –The steep topography, high rainfall, and seismic hazard level in the Higabra Valley suggests the potential for mass movements and remnants of past events are evident. The access road, power lines, and facilities located in the valley bottom would be at risk. CGI has undertaken slope deposit mapping (based partly on ultra-high resolution LiDAR, topography and surface morphologies) and geotechnical studies throughout all areas of current planned infrastructure in the Buriticá Project and has concluded that risks from major mass movements are acceptable.
- **Comminution** – The wall rock is much harder than the vein material and it appears that increased waste rock in the metallurgical samples increases the comminution parameters. Grade control and proper mining execution when implemented will maintain minimal unplanned dilution, which would minimize potential impacts on grade, throughput, and operating costs. Continued test stoping at Yaraguá mine will help to verify dilution estimates.
- **Plant Feed Blend** - Determination of the amount of feed to the plant that contains increased levels of arsenic needs to be refined, particularly from Veta Sur where it indicates a decrease in the gold recovery. It is recommended that additional study applying the variability data to date and geometallurgical models be used to optimize mine production scheduling and blending techniques to more fully understand and possibly improve gold recovery.
- **Geomechanical Conditions –** Comprehensive studies were done to accurately estimate anticipated ground conditions. There is a risk that a larger percentage of the ore must be extracted using C&F rather than the longhole method resulting in higher costs. If this situation were to occur, the sensitivity analyses show that Buriticá Project economics continue to justify development of the mine and mill.

The FS has highlighted several opportunities to increase mine profitability and project economics, and reduce identified risks.

- **Inferred Resources** – Inferred resources are not included in the production schedule; however, a plan to infill drill specific areas could significantly improve the Project economics, especially in the footwall and hanging wall, and for zones between the Yaraguá main and Yaraguá deep zones. Operational definition drilling will test inferred resources as part of the production sequence. Identification of additional resources will have a compounding positive effect in that the development per ore tonne will be decreased as well as vertical mining advance rates. Additionally, mine plans for specific areas, such as those below the Higabra level would change dramatically with the addition of resource, by allowing more methodical mining below the Higabra and reducing development and water handling costs by allocating over a larger reserve base.
- **Mine Grade Strategy** – Cut-off grade trade-off evaluations indicated that a COG higher than the 3.8 g/t for Yaraguá and 4.0 g/t for Veta Sur could provide equal or better project economics. In-depth mine design and scheduling would be required to validate any potential benefits, and to determine if an alternative plan with higher early year COG would provide increase upfront cash flow and decrease risk without diminishing the mineable resource. The mining strategy is flexible and depending on market conditions at the time of commissioning, opportunity exists to adjust the mine plan.
- **Increased Gold Recovery** - Potential exists to increase gravity gold recovery using alternative methods such as intensively leaching gravity concentrate and/or regrinding gravity concentrate.

- **Grind Size** - There exist potential economic benefits of improving recovery using a finer primary grind size.
- **Flotation of a bulk concentrate followed by regrind and cyanidation** - The opportunities with this approach trade additional complexity for a coarser primary grind and indications of initial test work for overall gold and silver recovery improvements, and potential benefits of two tailing streams – a non-sulphide un-leached coarse tailing and a reduced tonnage of finer sulphide tailing, decreased plant footprint and capital requirements.
- **Ultra-Filtration Water Treatment** – Capital and operating costs are relatively high due to the large water volumes treated, and the process produces a brine waste product, which is crystalized to a solid in the FS operations plan. Alternatives for disposing of the brine such as a liquid in the paste backfill, or investigating methods to reduce the brine volume, could significantly reduce costs.
- **Alternative TSF construction material** – Stability considerations require that a binder be used to increase TSF stability due to a limited supply of development waste rock. Alternative material for buttress and cover could be sourced from an optimized TSF foundation excavation design. Engineering to determine the best excavated material balance compared to binder addition may result in significant cost savings.
- **Construction Costs** - Civil construction capital cost may be reduced by additional engineering to better balance cut and fill of the plant site and water management infrastructure.
- **Concrete Reduction** - Opportunity exists to reduce concrete quantities with more geotechnical investigation during detailed engineering.
- **Powerline Finance Cost** - The 110 kV powerline supply and installation cost are financed in the economic model at 15%. Opportunities to reduce this cost impact to project economics need to be investigated.

27 Recommendations

Due to the positive, robust economics, it is recommended to expediently advance the Buriticá Project to construction and development, and then production. The recommended development path is to continue efforts to obtain the environmental permit modification approval while concurrently advancing key activities that will reduce project execution time. Associated project risks are manageable, and identified opportunities can provide enhanced economic value.

The Project exhibits robust economics with the assumed gold price, COP exchange rate, and consumables pricing. The risks are acceptable as well. Value engineering and recommended fieldwork should be advanced in preparation of permit approval in order to de-risk the construction schedule and minimize costs.

From project risks and opportunities, the following were identified as critical actions that have the potential to strengthen the Project and further reduce risk and should be pursued as part of the project development plan.

Regarding the Section 1.15 opportunities, recommendations are as follows:

- **Inferred Resources** – Inferred resources are not included in the production schedule; a plan should be developed to infill drill specific areas that could potentially provide significant improvement to Project economics.
- **Mine Grade Strategy** – A trade-off study evaluating project economics for various mine cut-off grades should be advanced. The study requires mine design and production schedules using different cut-off grades in order to compare results and subsequent recommendations.
- **Increased Gold Recovery** – Comparative economic trade-off evaluations should be performed to determine if there is potential to increase gravity gold recovery using alternative methods such as intensively leaching gravity concentrate and/or grinding gravity concentrate to improve recovery.
- **Grind Size** – Additional studies are warranted to evaluate the potential economic benefits of improving recovery using a finer primary grind size.
- **Flotation of a bulk concentrate followed by regrind and cyanidation** - A high level trade-off study comparing Flotation to Whole Ore Leach (WOL) was completed based on historic metallurgical test work. The results demonstrated economic, and operational advantages of WOL. It is recommended that these results be verified with current test results.
- **Ultra-Filtration Water Treatment** – Due to the relatively high capital and operating costs for water treatment, a study is warranted to determine alternative methods to dispose of the brine waste product, or reduce the brine volume.
- **Alternative TSF construction material** – The benefits to using excavated alluvial material to reduce binder requirements while maintaining required stability performance should be assessed. The assessment would need to include evaluating TSF foundation excavation design. As part of this work, modeling the deposit by alteration type to better plan production and disposal of Potentially Acid Generating/Non-Acid Generating (PAG/NAG) waste rock types is also recommended.

- **Construction Costs -** Civil construction cost may be reduced by additional engineering to better balance cut and fill of the plant site and water management infrastructure perhaps reducing the CAPEX of plant earthworks. This opportunity should be targeted in the detailed engineering phase.
- **Concrete Reduction** – Existing geotechnical site characterization needs to be assessed, and then a program planned to obtain required information to design foundations for major mill equipment and TSF.
- **Powerline Finance Cost** – A comprehensive study to develop powerline installation and power supply finance alternatives is needed to fully assess potential reductions to capital and operating costs.

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29 Units of Measure, Abbreviations and Acronyms

actual cubic feet per minute	Acfm
ampere	A
annum (year)	a
bed volumes per hour	BV/h
billion	B
billion tonnes	Bt
billion years ago	bya
billions of years	Ga
British thermal unit	BTU
centimetre	cm
centipoise	cP
cubic centimetre	cm^3
cubic feet per minute	cfm
cubic feet per second	ft^3/s
cubic foot	ft^3
cubic inch	in^3
cubic metre	m^3
cubic metres per hour	m^3/h
cubic metres per second	m^3/s
day	d
days per week	d/wk
days per year (annum)	d/a
dead weight tonnes	DWT
decibel adjusted	dBa
decibel	dB
degree	°
degrees Celsius	°C
diameter	ø
dollar (American)	US$
dollar (Canadian)	C$
dry metric tonne	dmt
environmental impact assessment	EIA
foot	ft
gallon (US)	gal
gallons per minute (US)	gpm
Gigajoule	GJ
Gigapascal	GPa
Gigawatt	GW
gram	g
grams per litre	g/L
grams per tonne	g/t
hectare (10,000 m^2)	ha
hertz	Hz
horsepower	hp
hour	h
hours per day	h/d
hours per week	h/wk
hours per year	h/a

hydraulic conductivity	K
inch	in
kilo (thousand)	k
kilogram	kg
kilograms per cubic metre	kg/m^3
kilograms per hour	kg/h
kilograms per square metre	kg/m^2
kilometre	km
kilometres per hour	km/h
kilopascal	kPa
kilotonne	kt
kilovolt	kV
kilovolt-ampere	kVA
kilowatt	kW
kilowatt hour	kWh
kilowatt hours per tonne	kWh/t
kilowatt hours per year	kWh/a
litre	L
litres per minute	L/min
litres per second	L/s
megabytes per second	Mb/s
megapascal	MPa
megavolt-ampere	MVA
megawatt	MW
metre	m
metres above mean sea level	mamsl
metres below-ground surface	mbgs
metres below sea level	mbsl
metres per minute	m/min
metres per second	m/s
microns	µm
milligram	mg
milligrams per litre	mg/L
millilitre	L
millimetre	mm
million	M
million bank cubic metres	Mbm^3
million bank cubic metres per annum	Mbm^3/a
million tonnes	Mt
minute (plane angle)	'
minute (time)	min
month	mo
Normal cubic metres per hour	Nm^3/h
parts per billion	ppb
parts per million	ppm
pascal	Pa
pounds per square inch	psi
revolutions per minute	rpm
second (plane angle)	"
second (time)	s
specific gravity	SG

square centimetre	cm^2
square foot	ft^2
square inch	in^2
square kilometre	km^2
square metre	m^2
standard cubic feet per minute	Scfm
tonne (1,000 kg) (metric ton)	t
tonnes per day	t/d
tonnes per hour	t/h
tonnes per year	t/a
tonnes seconds per hour metre cubed	ts/hm^3
Troy ounce	oz
volt	V
week	wk
weight/weight	w/w
wet metric tonne	wmt

1.1 General Abbreviations and Acronyms

abrasion index	Ai
acid rock drainage	ARD
atomic absorption spectroscopy	AAS
Bench Face Angle	BFA
Bond Ball Mill work index	BMWi
Canadian Institute of Mining, Metallurgy and Petroleum	CIM
capital cost allowance	CCA
capital expenditure	CAPEX
carbon dioxide	CO_2
carbon-in-leach	CIL
carbon-in-pulp	CIP
carbon monoxide	CO
Certified Reference Material	CRM
Coefficient of Variation	CV
Copper sulphate	$CuSO_4$
crushing work index	CWi
cumulative net cash flow	CNCF
cut-off grade	COG
dead weight tonnage	DWT
drift and fill	DF
electrowinning	EW
engineering, procurement, and construction management	EPCM
fresh air raise	FAR
field electrical centre	FEC
Footwall	FW
Geological Strength Index	GSI
Global Positioning System	GPS
gold	Au
hanging wall	HW
hydrated lime	$Ca(OH)_2$
internal rate of return	IRR
International Standards Organization	ISO

internet protocol	IP
inter-ramp angle	IRA
JDS Energy & Mining Inc.	JDS
Lerchs-Grossman	LG
life of mine	LOM
local area network	LAN
metal leaching	ML
Metal Mining Effluent Regulations	MMER
Mine Closure and Reclamation Plan	MCRP
net cash flow	NCF
net present value	NPV
net smelter return	NSR
neutralization potential/acid production	NP/AP
non-potentially acid generating	NPAG
overburden	OVB
oversize	O/S
post pillar cut and fill	PPCF
potentially acid generating	PAG
Pre-feasibility Study	PFS
Preliminary Economic Assessment	PEA
Qualified Person	QP
quality assurance/quality control	QA/QC
Rock Mass Rating (1989 version)	RMR89
Rock Quality Designation	RQD
semi-autogenous grinding	SAG
sodium cyanide	NaCN
sodium hydroxide	NaOH
sodium metabisulphite	SMBS
specific gravity	SG
sulphur dioxide	SO_2
Tailings Storage Facility	TSF
three-dimensional	3D
total dissolved solids	TDS
total suspended solids	TSS
two dimensional	2D
unconfined compressive strength	UCS
uninterruptible power supply	UPS
variable frequency drive	VFD
Voice over Internet Protocol	VoIP
Volcanic-turbidite series	VTS
waste rock storage area	WRSA
wide-area network	WAN
weak acid dissoluble	WAD
weak acid dissoluble cyanide	CN_{WAD}
work breakdown structure	WBS
Workers Compensation Board	WCB

1.2 Abbreviations and Acronyms used in this Report

Feasibility Study	(FS)
JDS Energy & Mining Inc.	(JDS)
Continental Gold Inc.	(CGI)
Mean sea level	(masl)
Carbonate base metal	(CBM)
Mining Associates	(MA)
Cut off Grade	(COG)
Loose rock fill	(LRF)
Cemented rock fill	(CRF)
Counter-Current Decantation	(CCD)
Semi-autogenous grinding	(SAG)
Tailing Storage Facility	(TSF)
Environmental Impact Assessment (Estudio de Impacto Ambiental)	(EIA)
Project of National Strategic Interest	(PINES)
Life-of-mine	(LOM)
Net smelter return	(NSR)
Colombian Peso	(COP)
Whole Ore Leach	(WOL)
Potentially Acid Generating	(PAG)
Non-Acid Generating	(NAG)
Schlumberger Water Services	(SWS)
Ministry of Mines and Energy (Ministerio de Minas y Energía),	(MME)
Plan Nacional de Desarrollo	(PND)
Regional Autonomous Corporations	(CAR)
SGS Consultants	(SGS)
Qualified Person	(QP)
Pre-feasibility Study	(PFS)
Ordinary Kriging	(OK)
Inverse Distance Squared (IDS) and Nearest Neighbour	(NN)
Canadian Institute of Mining	(CIM)
Cut-and-fill	(C&F)
Longhole open stoping	(LHOS)
Triaxial compressive strength	(TCS)
Brazilian tensile strength	(BTS)
Load-haul-Dump LHD diatomaceous earth	(DE)
Voice Over Internet Protocol	(VoIP)
Light Detection and Ranging	(LiDAR)
Global Positioning System	(GPS)
Inertial Measurement Unit	(IMU)
Water treatment plant	(WTP)
High density polyethylene	(HDPE)
Ultrafiltration	(UF)
Reverse Osmosis	(RO)
Clean-in-place	(CIP)
Geosynthetic clay layer	(GCL)
Humidity Cell Test	(HCT)
"Limites Máximos Permisibles"	(LMP)
Value Added Tax	(VAT or IVA)
Net Present Value	(NPV)

Internal Rate of Return	(IRR)
Project Management	(PM)
Project Execution Plan	(PEP)
Quality Assurance	(QA),
Construction management	(CM)
Mechanical, piping, electrical, and instrumentation	(MPEI)
Vendor quality surveillance/inspection	(VQS)
Site Quality Management Plan	(SQMP)
Safety Management Plan	(SMP)
Health, Safety, and Environmental	(HSE)

Appendix A

QP Certificates

JDS Energy & Mining Inc.
Suite 900 – 999 West Hastings Street
Vancouver, BC V6C 2W2
t 604.558.6300

jdsmining.ca

CERTIFICATE OF AUTHOR

I, Wayne Corso, P.E., do hereby certify that:

1. I am currently employed as Vice President of Engineering with JDS Energy & Mining Inc. with an office at Suite 151 – 2015 W. River Road, Tucson, AZ, 85704;

2. This certificate applies to the technical report titled "Buriticá Project NI 43-101 Technical Report Feasibility Study, Antioquia, Colombia", with an effective date of February 24, 2016 prepared for Continental Gold Inc.

3. I am a Professional Mining Engineer (P.E. #58884) registered with the Arizona Board of Technical Registration. I am a member of the Society for Mining Metallurgy and Exploration.

   I am a graduate of the Colorado School of Mines. I have been involved in mining operations and projects including technical aspects of resource estimation, mine planning, process design as well as economic analysis since 1984.

   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.

4. I visited the Buriticá Project site on April 8-9, 2015;

5. I am responsible for Sections 1; 2; 3; 4; 5; 6; 18; 19; 21; 22; 23; 24; 25; 26; 27; 28; 29 of the 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 not had prior involvement with the property that is the subject of the Independent Technical Report;

8. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

9. As of the date of this certificate, to the best of my knowledge, information and belief, this technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading;

Effective Date: February 24, 2016
Signing Date: March 28, 2016

(original signed and sealed) "Wayne Corso P.E."

Wayne Corso, P.E.

JDS Energy & Mining Inc.
Suite 900 – 999 West Hastings Street
Vancouver, BC V6C 2W2
t 604.558.6300

jdsmining.ca

CERTIFICATE OF AUTHOR

I, Michel Creek, P.E., do hereby certify that:

This certificate applies to the Technical Report entitled Buritica Project Feasibility Study (the "Technical Report") with an effective date of February 24, 2016, prepared for Continental Gold Ltd.;

1. I am currently employed as Project Manager, with JDS Energy & Mining Inc. with an office at Suite 151, 2015 West River Rd., Tucson, AZ 85704;

2. I am a graduate of Montana College of Mineral Science & Technology with a BS in Mining Engineering, 1987, and an MS in Mining Engineering, 1988. I have been involved in Mining since 1980;

3. I am a Registered member (No. 682200RM) in good standing of The Society for Mining, Metallurgy and Exploration, Inc. (SME);

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 engineering and mineral processing design, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101. I am independent of Continental Gold Ltd. as described in Section 1.5 of NI 43-101;

5. I visited the Buritica project site on October 1st and 2nd, 2015;

6. I am responsible for for Section number 20;

7. I have no prior involvement with the property that is the subject of this Technical Report;

8. As of the effective date of the Technical Report, 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;

9. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-01 and Form 43-101F1.

Effective Date: February 24, 2016
Signing Date: March 28, 2016

(original signed and sealed) "Michel Creek, P.E."

Michel Creek, P.E., SME Registered Member 682200

JDS Energy & Mining Inc.
Suite 900 – 999 West Hastings Street
Vancouver, BC V6C 2W2
t 604.558.6300

jdsmining.ca

CERTIFICATE OF AUTHOR

I, Austin Hitchins P. Geo, do hereby certify that:

1. I am currently employed as Senior Geologist with JDS Energy & Mining Inc. with an office at Suite 900 – 999 West Hastings Street, Vancouver B.C. V6C 2W2;

2. This certificate applies to the technical report titled "Buriticá Project NI 43-101 Technical Report Feasibility Study, Antioquia, Colombia", with an effective date of February 24, 2016 prepared for Continental Gold Inc.

3. I am a Professional Geoscientist registered with the Association of Professional Engineers and Geoscientists of British Columbia (license: 22869). I am also registered with the Association of Professional Geoscientists of Ontario (license: 0359).

   I am a graduate of the University of Alberta with a B.sc in geology. I have worked at underground mining operations, corporate settings, and performed technical aspects of resource estimation and project evaluation since 1983.

   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.

4. I visited the Buriticá Project site on April 8th and 9th of 2015;

5. I am responsible for Sections 7, 8, 9 and 10 of the 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 not had prior involvement with the property that is the subject of the Independent Technical Report;

8. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

9. As of the date of this certificate, to the best of my knowledge, information and belief, this technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading;

Effective Date: February 24, 2016
Signing Date: March 28, 2016

(original signed and sealed) "Austin Hitchins P. Geo"

Austin Hitchins, P. Geo

JDS Energy & Mining Inc.
Suite 900 – 999 West Hastings Street
Vancouver, BC V6C 2W2
t 604.558.6300

jdsmining.ca

CERTIFICATE OF AUTHOR

I, Gregory A. Blaylock, P.Eng. do hereby certify that:

1. I am currently employed as Project Manager (Associate) with JDS Energy & Mining Inc. with an office at Suite 151 – 2015 W. River Road, Tucson, AZ, 85704;

2. This certificate applies to the technical report titled "Buriticá Project NI 43-101 Technical Report Feasibility Study, Antioquia, Colombia", with an effective date of February 24, 2016 prepared for Continental Gold Inc.

3. I am a Professional Engineer, L1007 with the Northwest Territories and Nunavut Association of Professional Engineers and Geoscientists;

   I am a graduate of the University of Idaho with a Bachelor of Science Degree in Mining Engineering (1985) and a graduate of the University of the Witwatersrand with a Master's Degree in Engineering (1987). I have practiced my profession continuously since June 1, 1985 and have been involved in the evaluation, design and operation of numerous hard rock underground precious metal mining projects in the roles of operations engineering and management, corporate engineering and management and as an independent consultant.

   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.

4. I visited the Buriticá Project site on April 8-9, 2015, October 15-22, 2015 and October 27, 2015.

5. I am responsible for Sections 15 and 16 of the 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 not had prior involvement with the property that is the subject of the Independent Technical Report;

8. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

9. As of the date of this certificate, to the best of my knowledge, information and belief, this technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading;

Effective Date: February 24, 2016
Signing Date: March 28, 2016

(original signed and sealed) "Gregory A Blaylock P.Eng."

Gregory A Blaylock, P.Eng.

JDS Energy & Mining Inc.
Suite 900 – 999 West Hastings Street
Vancouver, BC V6C 2W2
t 604.558.6300

jdsmining.ca

CERTIFICATE OF AUTHOR

I, Stacy Freudigmann do hereby certify that:

1. I am currently contracted as a Project Manager with JDS Energy & Mining Inc. who has an office at Suite 900 – 999 W Hastings Street, Vancouver, BC, V6C 2W2;

2. This certificate applies to the technical report titled "Buriticá Project NI 43-101 Technical Report Feasibility Study, Antioquia, Colombia", with an effective date of February 24, 2016, (the "Technical Report") prepared for Continental Gold Inc. ("the Issuer");

3. I am a Professional Engineer (P.Eng. License #33972) registered with the Association of Professional Engineers, Geologists of British Columbia. I am a Member of the Canadian Institute of Mining and Metallurgy and the Australasian Institute of Mining and Metallurgy.

   I am a graduate of James Cook University with a B.Sc.(Hons) in Industrial Chemistry (1996) and Curtin University, Western Australia School of Mines with a Grad.Dip. Metallurgy (1999). I have been involved in mining since 1996 and have practiced my profession continuously since 1996. I have held senior process and metallurgical production and technical positions in mining operations in Canada and Australia. I have worked as a consultant for over five years and have performed metallurgical management, process managment, project management, cost estimation, scheduling and economic analysis work for a number of engineering studies and technical reports located in Latin America, Europe, Asia Pacific, USA and Canada.
   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.

4. I have not visited the Buriticá Project site;

5. I am responsible for Section number 13 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 NI 43-101;

7. I have not had prior involvement with the property that is the subject of the Independent Technical Report.

8. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

9. As of the effective date of the Technical Report and the date of this certificate, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading;

Effective Date: February 24, 2016
Signing Date: March 28, 2016

(original signed and sealed) "Stacy Freudigmann P.Eng."

_____

Stacy Freudigmann, P.Eng

Mining Associates Pty Ltd
Level 4, 67 St Paul's Terrace
PO Box 161
Spring Hill QLD 4004 AUSTRALIA
T 61 7 3831 9154 F 61 7 3831 6754
ABN 29 106 771 67

CERTIFICATE OF AUTHOR

I, Andrew Vigar, do hereby certify that:

1. I am an independent Consulting Geologist and Professional Geoscientist with my office at Level 4, 67 St Paul's Terrace, Brisbane, Queensland 4001, Australia.
2. This certificate applies to the technical report titled "Buriticá Project NI 43-101 Technical Report Feasibility Study, Antioquia, Colombia", with an effective date of February 24, 2016 prepared for Continental Gold Inc.
3. I was elected a Fellow of the Australasian Institute of Mining and Metallurgy ("The AusIMM") in 1993. My status as a Fellow of The AusIMM is current. I am a Member of the Society of Economic Geologists (Denver). I am recognized by the Australian Securities and Investments Commission and the Australian Stock Exchange as a Qualified Person for the submission of Independent Geologist's Reports. I am an Adjunct Fellow at the School of Earth Sciences at the University of Queensland.

   I graduated from the Queensland University of Technology, Brisbane, Australia in 1978 with a Bachelor Degree in Applied Science in the field of Geology.

   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;
4. I have visited the Buriticá Project site 24 to 27 of February 2015;
5. I am responsible for Sections 11, 12 and 14 of the report;
6. I was involved with the Mining Associates report prepared in 2015 and titled: Independent Technical Report and Resource Estimate on The Buriticá Gold Project 2015, Buriticá Gold Project, Colombia

1. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the National Instrument 43-101;
2. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.
3. As of the date of this certificate, to the best of my knowledge, information and belief, this technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading;

Effective Date: February 24, 2016
Signing Date: March 28, 2016

(original signed and sealed) "Andrew Vigar FAusIMM."

Andrew Vigar (FAusIMM).

CERTIFICATE OF AUTHOR

I, David Stone, P.E., do hereby certify that:

1. I am currently employed as President of MineFill Services, Inc., that is a Washington, USA, domiciled Corporation.
2. I am a graduate of the University of British Columbia with a B.Ap.Sc in Geological Engineering, a Ph.D. in Civil Engineering from Queen's University at Kingston, Ontario, Canada, and an MBA from Queen's University at Kingston, Ontario, Canada.
3. I have practiced my profession for over 30 years and have considerable experience in the preparation of engineering and financial studies for base metal and precious metal projects, including Preliminary Economic Assessments, Preliminary Feasibility Studies and Feasibility Studies.
4. I am a licensed Professional Engineer in Ontario (PEO #90549718) and I am licensed as a Professional Engineer in a number of other Canadian and US jurisdictions.
5. I have read the definition of 'Qualified Person' set out in National Instrument 43-101 – *Standards of Disclosure for Mineral Projects* ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a Qualified Person for the purposes of NI 43-101.
6. I visited the subject property on June 10, 2014.
7. I am responsible for the report content related to the paste backfill plant (portions of Sections 16 and 18)
8. I am independent of the Issuer applying all the tests in Section 1.5 of NI 43-101.
9. I have had no prior involvement with the property.
10. I have read NI 43-101 and NI 43-101F1 and this Technical Report has been prepared in compliance with that instrument and form.
11. As of the Effective Date of the Technical Report (February 24, 2016), 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.
12. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their website accessible by the public, of the Technical Report.

Effective Date: February 24, 2016
Signing Date: March 28, 2016

(original signed and sealed) "David Stone P.E."

David Stone, P.E.

Suite 900, 580 Hornby St., Vancouver BC, Canada V6C 3B6
Phone (778) 331-5060 Fax (604) 684-8073
www.robertsongeoconsultants.com

CERTIFICATE OF AUTHOR

I, Jack Caldwell, P.E., do hereby certify that:

1. I am currently employed as a Civil Engineer with Robertson Geoconsultants of Suite 900, 580 Hornby Street, Vancouver, BC V7J 3K3.

2. This certificate applies to the technical report titled "Buriticá Project NI 43-101 Technical Report Feasibility Study, Antioquia, Colombia", with an effective date of February 24, 2016 prepared for Continental Gold Inc.

3. I am a Professional Engineer (P.E. 58841) registered with the California Board of Professional Engineers, Land Surveyors, and Geologists. I am also a P.Eng. (12652) registered with the Association of Professional Engineers and Geoscientists of British Columbia.

   I am a graduate of the University of the Witwatersrand, Johannesburg, South Africa. I have been involved in mining operations and projects including technical aspects of tailings facility design, construction, operation, and closure since 1974.

   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.

4. I visited the Buriticá Project site in July 2015.

5. I am responsible for section of the report that pertain to the design of the tailings storage facility (Section 18.6).

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 not had prior involvement with the property that is the subject of the Independent Technical Report;

8. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

9. As of the date of this certificate, to the best of my knowledge, information and belief, this technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading;

Effective Date: February 24, 2016
Signing Date: March 28, 2016

(original signed and sealed) "Jack Caldwell P.E."

Jack Caldwell, P.E.

ARCHITECTURE
ENGINEERING
CONSTRUCTION MANAGEMENT

CERTIFICATE of QUALIFIED PERSON

I, Laurie M. Tahija, Q.P., do hereby certify that:

1. I am currently employed as Vice President by:

   M3 Engineering & Technology Corporation
   2051 W. Sunset Road, Ste. 101
   Tucson, Arizona 85704
   U.S.A.

2. I am a graduate of Montana College of Mineral Science and Technology, in Butte, Montana and received a Bachelor of Science degree in Mineral Processing Engineering in 1981.

3. I am recognized as a Qualified Professional (QP) member (#01399QP) with special expertise in Metallurgy/Processing by the Mining and Metallurgical Society of America (MMSA):

4. I have practiced mineral processing for 35 years. I have over twenty (20) years of plant operations and project management experience. I have been involved in projects from construction to startup and continuing into operation. I have worked on scoping, pre-feasibility and feasibility studies for mining projects in the United States and Latin America, as well as worked on the design and construction phases of some of these projects.

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 am responsible for section 17 of the technical report titled ""Buriticá Project NI 43-101 Technical Report Feasibility Study, Antioquia, Colombia", with an effective date of February 24, 2016 prepared for Continental Gold Limited.

7. I have not had prior involvement with the property that is the subject of the Technical Report.

8. I have visited the Buriticá project site on February 13, 2012.

9. As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information required to be disclosed to make the report not misleading.

10. I am independent of the issuer applying all of the tests in section 1.5 of National Instrument 43-101.

11. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

2051 W. Sunset Rd.
Suite 101

Tucson, Arizona
85704

t 520.293.1488
f 520.293.8349

www.m3eng.com

12. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated this 28rd day of March, 2016.

ORIGINAL SIGNED AND SEALED

Laurie Tahija, Q.P.
Print name of Qualified Person

SRK Consulting (U.S.) Inc.
1125 17th Street, Suite 600
Denver, Colorado
80202

denver@srk.com
www.srk.com

Tel: 303.985.1333
Fax: 303.985.9947

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 "Buriticá Project NI 43-101 Technical Report Feasibility Study, Antioquia, Colombia", with an effective date of February 24, 2016", prepared for Continental Gold 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 have visited the Buriticá Project site on April 8-9, 2015 and October 23-24, 2015;
5. I am responsible for preparation of section 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 not had prior involvement with the property that is the subject of the 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 24, 2016
Signing Date: March 28, 2016

"Original Signed and Sealed"

Michael E. Levy, P.E, P.G.

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