MINERAL RESOURCE ESTIMATE AND PRELIMINARY ECONOMIC ASSESSMENT FOR UNDERGROUND MINING. ANGOSTURA GOLD-SILVER PROJECT, SANTANDER, COLOMBIA

Prepared by:

Rodrigo Mello, MAusImm

Carlos Guzman, MAusImm, (NCL Ingeniería y Construcción S.A.)

John Wells, FSAIMM

Giovanny Ortiz , MAusImm (Greystar Resources Ltd.)

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1.0 SUMMARY

Rodrigo Mello, NCL Ingenieria y Construccion Limitada (NCL) and Alquimia Conceptos S.A. (Alquimia) were commissioned by Greystar Resources Limited. (Greystar) to prepare an independent Qualified Person's Review and NI 43-101 Technical Report (the Report) for the wholly-owned Angostura gold–silver project (the Project) located in Colombia.

Greystar will be using the Report in support of the press release published on March 18, 2011.

1.1 Location and Access

The Project is located approximately 400 km north–northeast of the Colombian capital city of Santa Fé de Bogotá, and approximately 67 km northeast of the city of Bucaramanga in the Department of Santander.

Current Project access from Bucaramanga is via the partially-paved Matanza–Surata–California road, a distance of 67 km and travel time of two to three hours, depending on weather conditions. Within the Project area, access is by a network of unpaved roads, tracks and horse and foot trails.

1.2 Mineral Tenure, Surface and Water Rights, and Royalties

Greystar holds 14 mining titles covering over 31,000 ha which are held in its Branch in Colombia. The "Angostura Block", is found within mining title 3452. Along with mining licenses 101-68 and 127-68 they host the Angostura deposit. In april 2010, Greystar submitted an extension application for the exploration phase of mining title 3452 for an additional two years. This application was approved by Ingeominas in December 2010 therefore the exploration phase will expire on August 8 2012. Mining Licenses 101-68 and 127-68 are under request for extension.

Currently Greystar has outright ownership of approximately 3,700 ha of surface rights subject to certain deferred payments being made. Greystar has sufficient surface rights to support mine design, as the area of the plant and mine footprint cover an approximate area of 1,050 ha of this land.

The Colombian Mining Code grants the owner of a mining title rights to establish easements or rights of way for access and infrastructure, as well as to request expropriation of lands needed for the project. It is a reasonable expectation that the Project will be granted such easements and expropriations should they be required.

Greystar currently holds three water rights to carry out exploration works in the Angostura Block. Currently, La Plata area has a water license currently held by Sociedad Minera La Plata which will be requested for assignment to Greystar. Greystar has 8 water rights under request before the environmental authority.

On account of the acquisition of part of the Angostura Block, two royalties are payable to vendors. The first is a 5% net profits royalty for an area covering 150 ha of permit 3452 that hosts the Angostura deposit, and a 10% net profits royalty for an area of approximately 100. The underlying vendors of License 47-68 hold a 10% net profits royalty.

In addition, a royalty will be payable to the Colombian Government. According to Colombian Royalty Law, exploitation for gold production is subject to a 4% royalty on 80% of the London price fixing for the gold and silver production at pithead.

1.3 Permits

The Project requires that a work and investment plan (PTO) be prepared and approved prior to any exploitation activities being permitted.

Greystar submitted an application before Instituto Colombiano de Geología y Minería (Ingeominas) under the 2001 Mining Code for the PTO based on the 2009 pre-feasibility study for an Open Pit operation (2009 PFS) on October 23, 2009. The PTO covered the Angostura Block. As part of the normal evaluation of Greystar's PTO application, Ingeominas requested supplementary information which was timely submitted. In March 23, 2011 Greystar withdrew the PTO application. The company considered it necessary to reformulate the project addressing the government and the community´s concerns. Therefore, Greystar will study the viability of alternative options for the project, including the underground exploitation option, considered in the scoping study presented in this report.

A significant number of additional permits are required to support production activities and are required from a combination of local, Departmental and National authorities.

1.4 Environment

Field and exploration activities do not require filing of an EIA and are generally permitted under application of Mining and Environmental Guidelines, or environmental management plan or “plan de manejo ambiental” (PMA), depending on the mining regulation under which the mining title has been granted.

Exploration activities require the approval of an EIA. Greystar filed an EIA on December 22, 2009 for an open pit operation. Public consultation was conducted on the 21st of November 2010. Afterwards The Ministerio de Ambiente, Vivienda y Desarrollo Territorial (MAVDT) requested a second public consultation be held in Bucaramanga on 4 March, 2011. The second public hearing in Bucaramanga was terminated prematurely due to disorders presented during the event. As with the PTO, Greystar withdrew the EIA application from the MAVDT.

Baseline environmental studies were performed by Ingetec in 2008–2009 in support of the EIA and Project design (Open pit).

Part of the planned pit and associated mine infrastructure proposed were located within the "paramo" ecosystem, based on cartographic co-ordinates defined by the Alexander Von Humbolt Investigation Institute. However, the competent environmental authority for Colombia has not legally defined a "paramo" for the Project area; as such a declaration is contingent upon technical, social and environmental studies as prescribed in the Mining Code.

1.5 Geology and Mineralization

The Angostura deposit is considered a typical example of a high-sulfidation epithermal deposit. The deposit is hosted in the amphibolite facies Bucaramanga Gneiss, a series of meta-sediments of Proterozoic age, in a zone where a suite of porphyritic diorite to quartz monzonite bodies and dyke swarms of Triassic to Jurassic age are intruded. These rocks have been intersected by a swarm of east-west, east–northeast and southeast trending, steeply north-dipping structures (Veins). Mineralization occurs in bands, veinlets, stringers and disseminations and silicified hydrothermal breccias within the structures. Mineralized structures vary from less than 2 m for individual veins to over 40 m for composite structures and strike lengths range from less than 100 m to over 1 km. Mineralization is partly refractory. Gold occurs as occlusions in pyrite in the form of very fine-grained electrum and gold–silver tellurides (possibly calaverite and petzite). Particle sizes range from 5 µm to 180 µm.

A number of alteration styles have been noted, including argiillic, phyllic, silicification and local advanced argillic. Surface oxidation has affected the rocks at Angostura to a depth of 10–30 m at the edge of the deposit and attains depths that vary locally from 40 m to as much as 400 m in the upper and central parts especially along major faults.

The Angostura deposit is sub-divided geographically into a number of areas or sections that from south to north are referred to as El Vivito, El Silencio, Nueva Alta, La Perezosa, El Diamante, Central, La Alta and its eastern neighbour La Alta Este, El Pozo, Veta de Barro, Veta de Barro Este and Cristo Rey named after previous mines.

1.6 History and Exploration

Work completed on the Project includes geological mapping, underground mapping surface rock sampling, adit and tunnel excavation and sampling, core drilling, metallurgical testwork, ground water studies, mineral resource and mineral reserve estimations and engineering and design studies. Greystar completed a Scoping Study in 2008, a pre-feasibility study in 2009 and a feasibility study for the open pit operation that was completed in 2011. Parallel to the feasibility study a preliminary economic assessment for underground mining was developed by NCL that is part of this report.

1.7 Drilling

Drilling completed between 1994 and March 2011 in Angostura and surrounding areas including Mongora, Animas, Violetal and La Plata comprises 365,459 metres, including geotechnical-hydrogeological-condemnation drill holes. All drilling to date has been by core methods. Core was logged for geological and geotechnical parameters, and photographed. Drill collar locations have been surveyed and Greystar contracted a professional surveyor to perform the surveys.

Initial drill holes, until 1997, were measured using a Tropari instrument. From 1997 to 2003, a Sperry Sun instrument was used. From 2003, downhole surveys were typically taken at surface and 25 m intervals down hole, using a Reflex EZ-Shot instrument.

The average core recovery for the entire drill-hole database is approximately 93%, with 80% of the intervals above a 90% recovery.

Core sample lengths are variable, depending on lithology and can range from 0.5 m in highly silicified zones and visible sulfides to 3 m in areas of unaltered gneiss and dykes. The average sampled core length was 1.3 m in the 1990s drilling and has increased to nearer 1.7 m since 2003. In general, longer samples were taken in areas believed to be of below economic cut-off grade or where sample recovery was poor. Few samples are less than 0.5 m long. Sampling respects obvious lithological, alteration and mineralization breaks.

The Project database includes 9,700 specific gravity measurements on drill core samples selected according the lithology, alteration and mineralization using a wax immersion (ASTM C914-98) methodology.

1.8 Sample Preparation and Analyses

Prior to March 2004, sample preparation was performed by independent laboratories. Thereafter, preparation has taken place at the Greystar onsite laboratory under the supervision of Greystar personnel.

Since 2004, the sample preparation method consists of samples being single-stage crushed to nominally 80% passing 1.7 mm (10 mesh). Material is blended, and then a sub-sample of about 250 g is obtained by riffle splitting.

Analyses were performed by accredited independent laboratories which have included Rossbacher Laboratories Ltd. (Rossbacher) of Vancouver (1995 – 1999); Assayers Canada Limited (2004 – 2007); ALS Chemex Laboratories (ALS Chemex) (2003 – 2011); and ACME Analytical Laboratories Ltd. in Vancouver (ACME) (2003 – 2011).

Gold is assayed by fire assay with an atomic absorption spectrometer (FA/AAS) finish using a one assay-ton (29.2 g) aliquot (ALS Chemex Laboratories, Vancouver, Canada) or by a 15 g aliquot and a 30 element geochemical inductively-coupled plasma (ICP) gold method after aqua regia digestion (Acme Analytical Laboratories Ltd, Vancouver, Canada). Gold assays above 10 g/t Au and silver assays above 100 g/t Ag are re-assayed by one assay-ton FA with a gravimetric finish. At ALS Chemex, separate splits of the samples are subjected to a multi-element ICP assay, including silver and sulfur, following a four-acid digestion. All samples with ICP results that show a sulfur grade of >10% were re-assayed using the Leco method with an upper limit of 50% S.

1.9 Quality Assurance and Quality Control

There was no Greystar-sponsored quality assurance/quality control (QA/QC) program in place for the drilling campaigns from 1995 to 1999. However, a substantial program of check assaying of pulp duplicates was undertaken at Bondar Clegg Laboratories during those years, and in 2003–2004 a number of high-grade core intervals were re-sampled and rejects submitted for check assaying at ALS Chemex.

In June 2003, a QA/QC program external to the assay laboratory was instituted, consisting of submission of blanks and standard reference materials (SRMs).

Assayers Canada Limited performed secondary assays on pulp duplicate materials from April 2004 (with a three-month interruption at the end of 2005) to March 2007 at the rate of one in 20 to 30. Acme performed secondary assays on pulp duplicate materials from late 2007 to March 2011.

1.10 Data Verification

A number of data verification programs and audits have been performed over the Project history, primarily in support of compilation of technical reports on the Project. Data checks were also performed in support of the pre-feasibility and feasibility studies on the Project. A reasonable level of verification has been completed, and no material issues would have been left unidentified from the programs undertaken.

Barry Smee (Smee and Associates Consulting Ltd) is an independent auditor of the preparation laboratory as well as QC QA practices review and has made three visits to the project site since 2004 with the most recent review carried out in September of 2010.

1.11 Metallurgical Summary

The metallurgical testwork carried out by the project, has shown that Angostura ore is amenable to treatment by means of flotation and heap or agitated cyanidation.

Preliminary testwork showed that sulfide and transition ores respond well to a flotation stage performed on the whole mineral, followed by flotation tails cyanidation; oxide ore respond well to agitated cyanidation.

Table 1.11-1: Gold and Silver Metallurgical Recoveries Summary

Ore	Flotation recoveries			Au distribution flotation products		Recovery tails cyanidation	Flotation & tails cyanidation recovery
	Rougher	Cleaner	Overall	Cl concentate	Flotation tails	Global Tails	
	P80=106 µm	P80=106 µm		106 µm	106 µm	106 µm	106 µm
Sulfide	**93.0**	**93.0**	**86.5**	**86.5**	**13.5**	**58.5**	**94.4**
Transition	**75.0**	**65.7**	**49.3**	**49.3**	**50.7**	**92.3**	**96.1**
Oxide					**100.0**	**95.0**	**95.0**

Ore	Flotation recoveries			Ag distribution flotation products		Recovery tails cyanidation	Flotation & tails cyanidation recovery
	Rougher	Cleaner	Overall	Cl concentate	Flotation tails	Global Tails	
	P80=106 µm	P80=106 µm		106 µm	106 µm	106 µm	106 µm
Sulfide	**87.0**	**96.3**	**83.8**	**83.8**	**16.2**	**47.8**	**91.5**
Transition	**74.8**	**84.0**	**62.8**	**62.8**	**37.2**	**59.8**	**85.0**
Oxide					**100.0**	**84.5**	**84.5**

Flotation and tails cyanidation allow recoveries of 94.4% Au and 91.5% Ag for sulfide sample; and 96.1% Au and 85% Ag for transition sample, Table 1.11-1.

- Flotation recoveries are 86.5% gold and 83.8% silver for sulfide samples, and 49% gold and 63% silver for transition samples.
    - Rougher recoveries are 93% gold and 87% silver for sulfide samples, and 75% gold and silver for transition samples.
    - Cleaner recoveries are 93% gold and 96% silver for sulfide samples, and 66% gold and 84% silver for transition samples.
- Recoveries for rougher tails cyanidation are 59% gold and 48% silver for sulfide samples, and 92% gold and 60% silver for transition samples.

Agitated cyanidation allows recoveries of 95% Au and 84.5% Ag for oxide samples.

Alquimia have reviewed and evaluated the metallurgical testwork performed by various laboratories. Alquimia's assessment is that regrinding of the rougher concentrate to a relatively fine size, for example 37 microns, may be of benefit to the project. Our study is based upon this. This should allow the production of a reduced quantity of cleaner concentrate. This would reduce the capacity, size and cost, of the expensive refractory process unit operation. At the same time it would probably liberate more gold from the pyrite concentrate. Alquimia have assumed that a cyanidation recovery of 90% Au and 80% Ag might be achieved on the cleaner-scavenger tailings. However, all of this would require confirmatory testwork. Should a more conventional flotation concentrate be produced, without fine regrinding, it is Alquimia's view that the overall recovery would be very similar, in that more gold bearing material would be treated by the refractory process.

Flotation and cyanidation testwork has also showed that Angostura sulphide ore is refractory, and hence, sulfide oxidation tests were performed on flotation concentrate. The mineral showed good response to three oxidation techniques: roasting, pressure oxidation (POX) and biooxidation (BIOX). When a sulfide oxidation stage is considered in the circuit, gold dissolution increases from about 50% to an average value of 91% for roasting, 96% for POX and 92% for BIOX. Silver dissolution increases from 50% to approximately 60% for all alternatives

Table 1.11-2: Metallurgical Recoveries Summary

Ore	Flotation recoveries			Au distribution flotation products			Cyanidation Recovery			Overall Au recovery
	Rougher P80=106 µm	Cleaner P80=37 µm	Overall	Cl concentate P80=37 µm	Ro tails P80=106 µm	Scav tails P80=37 µm	Sulfur oxidated Conc. Roasting	Rougher Tails P80=106 µm	Scavenger Tails P80=37 µm	Roasting
Sulfide	93.0	65.1	60.5	60.5	7.0	32.5	91.0	58.5	90.0	88.4
Transition	75.0	45.0	33.8	33.8	25.0	41.3	91.0	92.3	95.0	93.0
Oxide					100.0			95.0	95.0	95.0

Ore	Flotation recoveries			Au distribution flotation products			Cyanidation Recovery			Overall Au recovery
	Rougher P80=106 µm	Cleaner P80=37 µm	Overall	Cl concentate P80=37 µm	Ro tails P80=106 µm	Scav tails P80=37 µm	Sulfur oxidated Conc. POX	Rougher Tails P80=106 µm	Scavenger Tails P80=37 µm	POX
Sulfide	93.0	65.1	60.5	60.5	7.0	32.5	96.0	58.5	90.0	91.4
Transition	75.0	45.0	33.8	33.8	25.0	41.3	96.0	92.3	95.0	94.7
Oxide					100.0			95.0	95.0	95.0

Ore	Flotation recoveries			Au distribution flotation products			Cyanidation Recovery			Overall Au recovery
	Rougher P80=106 µm	Cleaner P80=37 µm	Overall	Cl concentate P80=37 µm	Ro tails P80=106 µm	Scav tails P80=37 µm	Sulfur oxidated Conc. BIOX	Rougher Tails P80=106 µm	Scavenger Tails P80=37 µm	BIOX
Sulfide	93.0	65.1	60.5	60.5	7.0	32.5	92.0	58.5	90.0	89.0
Transition	75.0	45.0	33.8	33.8	25.0	41.3	92.0	92.3	95.0	93.3
Oxide					100.0			95.0	95.0	95.0

Ore	Flotation recoveries			Ag distribution flotation products			Cyanidation Recovery			Overall Ag recovery
	Rougher P80=106 µm	Cleaner P80=37µm	Overall	Cl concentate P80=37 µm	Ro tails P80=106 µm	Scav tails P80=37 µm	Sulfur oxidated Conentrate	Rougher Tails P80=106 µm	Scavenger Tails P80=37 µm	All
Sulfide	87.0	86.7	75.4	75.4	13.0	11.6	60.0	47.8	80.0	60.7
Transition	74.8	75.0	56.1	56.1	25.3	18.7	60.0	59.8	85.0	64.6
Oxide					100.0			84.5	85.0	84.5

Overall recoveries for flotation, pre-treated concentrate cyanidation and tails cyanidation are estimated to be (Table 1.11-2):

- Gold recoveries
    - In roasting alternative: 88.4% for sulfide, 93% for transition and 95% for oxide.
    - In POX alternative: 91.4% for sulfide, 94.7% for transition and 95% for oxide.
    - In BIOX alternative: 89% for sulfide, 93.3% for transition and 95% for oxide.
- Silver recoveries
    - In all alternatives: 60.7% for sulfide, 64.6% for transition and 84.5% for oxide

The amount of metallurgical testwork that has been carried out is extensive and is much more than would normally be produced to support a PEA. It is the result of many years of effort that were carried out to support a PFS and then a Detailed Feasibility Study.

1.12 Mineral Resources

Greystar constructed the geological model of veins from plan and sectional view interpretations. In addition, the deposit was divided into three vein families, according to the preferential directions and the structural domains. Weathering codes were assigned to each block on the basis of oxide, transitional or sulfide material. A single density value was assigned for each of the weathered zones.

Data inside the veins were composited to a standard 1 m length, except for Laches area where 1.5 m length where applied. Grade distribution was evaluated using a probability plot. Grade caps were applied to gold, silver, copper and sulfur grades. Variograms were constructed to provide the appropriate distances for search ellipsoid radii for each vein family.

Ordinary kriging was used for interpolating gold, silver, copper and sulfur. Each vein was interpolated with its own data and using an ellipse that follows its own aptitude (Strike and Dip).

The model was validated using visual methods, tabulations, and comparison between the floating window average grade of composites and interpolated values to ensure no biases were present.

Mineral resource blocks were classified as Indicated or Inferred using a combination of distance to the nearest sample, and drill hole numbers. Reasonable prospects of economic underground extraction were applied for resources reporting.

The mineral resources for the veins of Angostura Project outside of the stopes defined in the PEA are tabulated in Tables 1.12-1 to 1.12-4, using different cut off grades, 1.5, 2.0, 2.5 and 3.0 g/t Au. A crown pillar of 15 metres was used to limit the mineral resources in veins close to surface. Mineral resources have an effective date of 18 March 2011. The resources outside of the veins (Disseminated) were not evaluated and are not reported. Veins considered isolated and with poor content of gold ounces are not reported. The Qualified Person for the estimate is Rodrigo Mello, M.AusIMM, senior geologist, an independent consultant.

Table 1.12-1: Mineral Resources, outside the stopes @ 1.5 g/t Au COG

	Ore (t)	Au (g/t)	Au Oz	Ag (g/t)	Cu (%)
			INDICATED		
Oxides	1,233,974	3.48	138,058	13	0.023
Transition	4,548,357	3.57	521,421	21	0.032
Sulfides	14,614,648	3.47	1,629,434	20	0.082
Sub-total	**20,396,979**	**3.49**	**2,288,913**	**20**	**0.067**
			INFERRED		
Oxides	761,366	3.62	88,572	15	0.027
Transition	1,411,480	4.18	189,471	17	0.050
Sulfides	10,224,700	3.69	1,212,061	23	0.093
Sub-total	**12,397,546**	**3.74**	**1,490,104**	**22**	**0.084**

Table 1.12-2: Mineral Resources, outside the stopes @ 2.0 g/t Au COG

	Ore (t)	Au (g/t)	Au Oz	Ag (g/t)	Cu (%)
			INDICATED		
Oxides	925,381	4.06	120,805	14	0.024
Transition	3,357,309	4.21	454,857	22	0.034
Sulfides	10,484,414	4.15	1,398,448	23	0.090
Sub-total	**14,767,105**	**4.16**	**1,974,111**	**22**	**0.073**
			INFERRED		
Oxides	581,949	4.20	78,548	16	0.028
Transition	1,103,422	4.86	172,324	18	0.052
Sulfides	7,378,145	4.43	1,051,960	28	0.099
Sub-total	**9,063,515**	**4.47**	**1,302,832**	**26**	**0.089**

Table 1.12-3: Mineral Resources, outside the stopes @ 2.5 g/t Au COG

	Ore (t)	Au (g/t)	Au Oz	Ag (g/t)	Cu (%)
			INDICATED		
Oxides	681,946	4.70	103,152	14	0.024
Transition	2,455,658	4.94	389,967	23	0.035
Sulfides	7,616,340	4.87	1,192,721	27	0.096
Sub-total	**10,753,944**	**4.88**	**1,685,840**	**25**	**0.078**
			INFERRED		
Oxides	403,684	5.06	65,624	14	0.028
Transition	866,067	5.57	155,216	18	0.051
Sulfides	5,498,970	5.19	917,334	32	0.104
Sub-total	**6,768,721**	**5.23**	**1,138,174**	**29**	**0.092**

Table 1.12-4: Mineral Resources, outside the stopes @ 3.0 g/t Au COG

	Ore (t)	Au (g/t)	Au Oz	Ag (g/t)	Cu (%)
			INDICATED		
Oxides	499,214	5.43	87,198	15	0.025
Transition	1,783,624	5.77	330,880	24	0.037
Sulfides	5,642,124	5.62	1,019,271	31	0.102
Sub-total	**7,924,963**	**5.64**	**1,437,349**	**28**	**0.083**
			INFERRED		
Oxides	308,467	5.78	57,328	14	0.028
Transition	666,322	6.42	137,632	18	0.050
Sulfides	4,207,439	5.94	803,700	35	0.107
Sub-total	**5,182,227**	**5.99**	**998,661**	**32**	**0.095**

The mineable resources inside the stopes defined in the Preliminary Economic Assessment (PEA), are considered Inferred and are tabulated in the Table 1.13.2.

1.13 Preliminary Mining Study

Considering the geometry and geotechnical conditions of the orebody, different mining methods were analysed for the underground exploitation of the Angostura deposit.

According to the rock conditions presented, a geotechnical assessment was provided by the specialist consultants AKL S.A, whose recommendations for mining methods were:

- Veins with less than 5 m width = Bench and Fill Stoping
- Veins within 5 m and 20 m width = VCR (Vertical Crater Retreat)
- Veins within 20 m and 40 m width = Open Stoping

Given the distribution of widths and the geotechnical conditions of the rock, bench and fill has been assumed as the common method for the determination of mine plans and costs in this study.

The economic underground mineable resources were estimated using an ore resources block model based on wireframes for the definition of the high grade zones (this model was provided by Greystar and is different to that used for the open pit FS), including indicated and inferred mineral resources.

The reader is cautioned that the underground mining study is a preliminary assessment and it includes inferred mineral resources that are considered too *speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves. There is no certainty that the preliminary assessment will be realized. No Mineral Reserves have been estimated.*

Mineable resources were determined from selected veins by generating a contour at 3.0 g/t Au cut-off grade. These contours were created from plan views at 20m. The cut-off grade value was calculated from the following set of parameters, Table 1.13-1.

Table 1.13-1: Parameters for Mineable Resources Calculation

Total Mine Cost (Production & Maintenance)	40	US$/t
Process Cost	20	US$/t
G&A	10	US$/t
Selling	10	US$/oz Au
Recovery Au	85	%
Au Price	850	US$/oz

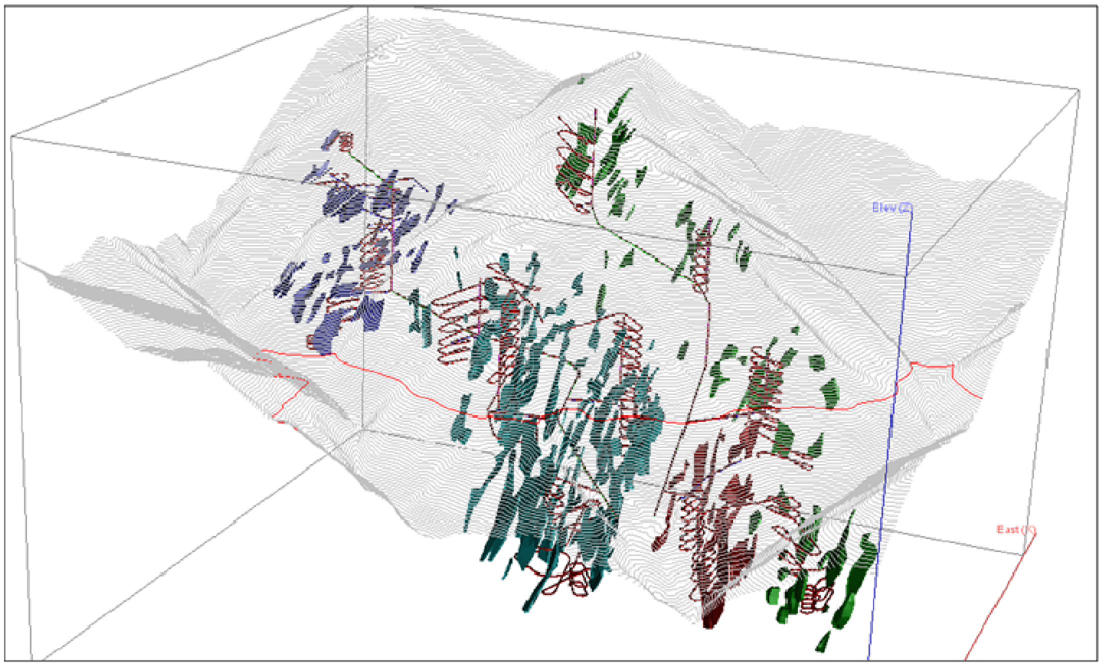
The total mineable resources are 13.98 Mt at grades of 5.35 g/t Au and 29.6 g/t Ag, as shown in Table 1.13-2.

Table 1.13-2: Mineable resources @ 3.0 g/t Au COG (Diluted)

	Ore (Mtonnes)	Au (g/t)	Au Koz	Ag (g/t)	Cu (%)
INFERRED					
Oxides	0.62	5.75	114	18.5	0.03
Transition	2.29	5.68	418	22.0	0.04
Sulfides	11.08	5.26	1,873	31.8	0.11
TOTAL	**13.98**	**5.35**	**2,405**	**29.6**	**0.09**

A general view is presented in Figure 1.13-1 which includes topography, stopes and mine design.

Figure 1.13-1: General Mine 3D view.

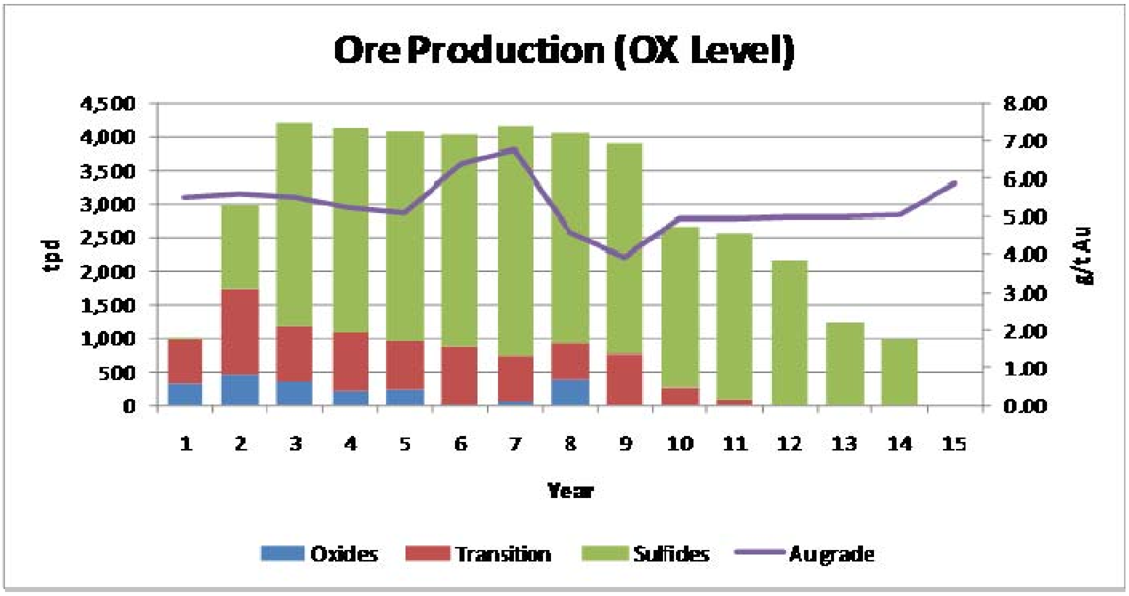


A total of 69 km of horizontal and 5 km of vertical development will be required for accesses, transport, preparation, ore passes and ventilation for the life of the mine.

The production plan was prepared for each stope independently and then integrated into a global plan to establish the maximum production capacity for the underground mine. Stopes were divided vertically to allow maximum productivity. The production plan was prepared estimating productivities per area involved in a sector. Productivity estimation is a function of the stope width involved and the mining method applied to the area.

The following graph (Figure 1.13-2) presents the resulting production plan for the mine. The maximum mine production rate is 4,000 tonnes per day (tpd) maintained for a period of 7 years.

Figure 1.13-2: Summary of Mine Production Plan



1.14 Equipment

The mine equipment estimate has been carried out based on the mine production and development plans. Equipment performances were estimated considering average distances. Estimation was made based on 8 hours/shift (5 effective operation hours), 3 shifts/day and 360 days/year.

Loading will be carried out by 7 cubic yards Load Haul Dump units (LHD's). LHD's will load into low profile trucks. Hauling will be performed by 20 t trucks. Hauling activities will comprise ore hauling from the mine to the crushing station and backfill material hauling from the dump to the stopes.

Fleet estimates indicate a maximum of 11 LHD units, 11 jumbos for development plus 3 bolting units, 3 DTH drilling rigs for bench drilling and 48 trucks.

1.15 Process Description

The proposed process flowsheet incorporates the following major process operations:

- Primary, Secondary and Tertiary Crushing
- Grinding
- Rougher Flotation – Regrinding and Cleaner and Scavenger Flotation

- Sulfur oxidation
  - Alternative A – Roasting/Cu acid leaching/Counter Current Decanter "CCD" circuit concentrate
  - Alternative B – Pressure oxidation (POX)/CCD circuit concentrate
  - Alternative C – Bio oxidation (Biox)/CCD circuit concentrate
- Intensive Cyanidation and Dewatering
- Conventional Cyanidation
- CCD circuit tailings – Cyanide destruction
- SART - CIC – Elution - EW – Smelting
- Tailings Disposal

1.16 Capital Costs

The total mine capital cost is US$ 220 M for the life of the mine with US$ 108 M for equipment and US$ 49 M for development. The initial capital is US$ 20.6 M. These numbers include a 35% contingency given the preliminary nature of the analysis.

The capital costs for the three scenarios of the processing plant and infrastructure were developed by Alquimia, varying from US$ 259 M to US$ 286 M and shown in Table 1.16.-1.

Table 1.16-1: Process & Infrastructure Capital Expenditure

			Nominal	Year 0	Year 4	Year 8
Alternative A	Roasting	KUS$	286,081	280,963	2,559	2,559
Alternative B	POX	KUS$	283,898	278,780	2,559	2,559
Alternative C	BIOX	KUS$	258,872	253,754	2,559	2,559

1.17 Operating Costs

Mine operating cost has been estimated at an average of 40.4 US$/t. Mine operating costs were calculated using unit prices and consumption factors.

The processing operating costs were also estimated by Alquimia and are presented in Section 16. The average cost varies between 26.0 US$/t (Roasting) to 27.1 US$/t (BIOX).

A general and administrative cost (G&A) was estimated as US$ 5.0/t.

Table 1.17-1 summarizes the operating costs, considering the three processing options.

Table 1.17-1: Operating Costs Summary

		Roasting	POX	BIOX
Mining Cost	US$/t	40.4	40.4	40.4
Processing Cost	US$/t	26.02	26.25	27.09
G&A	US$/t	5.0	5.0	5.0
Selling Costs	US$/oz	5.00	4.89	4.97
Royalty (3.2%)	US$/oz	35.0	34.9	35.0
Cathodes Transport	US$/t Cu	70.0	70.0	
Total Cost	**US$/oz**	**509.0**	**496.9**	**512.9**

1.18 Financial Analysis

A preliminary evaluation has been carried out by NCL upon the basis of the presented mine schedule and capital and operating costs for three different process scenarios, Table 1.18-1. Pre-tax NPV at 5% discount rate and IRR of the cash flows have been calculated for a gold price of 1,015 US$/oz and a silver price of 15.85 US$/oz. Higher prices were applied to the two initial years of the plan (1,170 US$/oz Au and 18.25 US$/0z Ag).

Table 1.18-1: Summary of Economic Evaluation

		Roasting	POX	BIOX
Dore Produced	Oz	12,983,907	13,040,538	12,995,233
Gold in dore	Oz	1,928,577	1,985,209	1,939,904
Silver in dore	Oz	7,725,719	7,725,719	7,725,719
Copper in dore	lb	228,316	228,316	228,316
Copper in cathodes	lb x 1000	17,758	17,758	
Sulfuric Acid	kt	881		
Mine Cost	US$/t	40.4	40.4	40.4
Process Cost	US$/t	26.02	26.25	27.09
G&A	US$/t	5.0	5.0	5.0
Selling Costs	US$/oz	5.00	4.89	4.97
Royalty	US$/oz	35.0	34.9	35.0
Cathodes Transport	US$/t Cu	70.0	70.0	
Total Cost	US$/oz	509.0	496.9	512.9
Initial Capital	KUS$	301,630	299,447	274,421
Mine	KUS$	20,667	20,667	20,667
Process & Infrastructure	KUS$	280,963	278,780	253,754
Total Capital	KUS$	506,462	504,279	479,253
Mine	KUS$	220,381	220,381	220,381
Process & Infrastructure	KUS$	286,081	283,898	258,872
NPV (5%)	KUS$	400,193	397,040	355,823
IRR	%	21.4%	21.5%	21.3%

1.19 Exploration Potential

The Angostura deposit remains open at depth and to the south. Additional potential remains in the greater Project area. Three major regional targets have been identified, including Móngora, Violetal and La Plata to the south of the Angostura deposit.

1.20 Conclusions

- In the opinion of the QPs, the Project that is outlined in this Report has met its objectives in that mineralization has been identified that can support estimation of Mineral Resources and there is sufficient additional scientific and technical information to have supported a preliminary economic assessment which, based on the assumptions made, returns positive economics. Additional metallurgical test work and geotechnical investigations are recommended to fine-tune the pre-feasibility process design.

- The project shows positive economic indices for all the scenarios evaluated and for the different sensitivity analyses performed

- Three alternatives can be used for sulfur oxidation: roasting, pressure oxidation and biooxidation.

- Roasting tests showed that a 91% of gold recovery can be reached.

- Pressure oxidation tests showed that a 96% of gold recovery can be reached.

- Biooxidation tests showed that a 92% of gold recovery can be reached.

- The production plan was prepared estimating productivities per area involved in a sector. Productivity estimation is a function of the stopes width involved and the mining method applied to the area. The mine production rate is 4,000 tonnes per day (tpd), maintained during 7 years.

- Loading will be made with 7 cubic yards Load Haul Dump units (LHD`s). LHD`s will load into low profile trucks. Hauling will be performed by 20 t trucks. Hauling activities will comprise ore hauling from the mine to the crushing station and backfill material hauling from the dump to the stopes.

- Fleet estimates indicate a maximum of 11 LHD units, 11 jumbos for development plus 3 bolting units, 3 DTH drilling rigs for bench drilling and 48 trucks.

- The total mine capital cost is 220 MUS$ for the life of the mine, with MUS$ 108 for equipment and 49 MUS$ for development. The initial capital is 20.6 MUS$. These numbers include a 35% contingency given the preliminary nature of the analysis

- Mine operating cost has been estimated at an average of 40.4 US$/t. Mine operating costs were calculated using unit prices and consumption factors.

1.21 Recommendations

- Criteria for construction of vein wireframes should be reviewed and adjusted according to the requirements of an underground operation.

- To develop population analysis for different elements in the different areas of the deposit to better reflect the variations of the deposit, especially silver, copper and sulfur during the resource estimate.

- To re-evaluate the bulk density for high grade veins using the specific gravity measurements of the high grade population.

- To re-evaluate the oxidation level model for the high grade veins population.

- Improve resource estimate classification by further drilling.

- Improve the accuracy of the definition of the high grade veins and incorporate into the model the relevant metallurgical variables (ore type, etc.).

- Develop a more complete geotechnical analysis of the different areas selected to validate the recommendations for stopes dimensions and ground support.

- Develop a more detailed mine layout and include the design of the ventilation and dewatering systems, considering options for the treatment and use of the water extracted from the mine.

- Develop a more detailed analysis of the surface layout, including location of the mine portals and processing plant. The location and design of ore stockpiles needs to be considered.

- It is recommended to establish a geometallurgical model for the first five years of exploitation, focusing the exploration in the "Perezosa" and "Silencio – Los Laches" sulfide zones, which corresponds to the higher resources percentages. The metallurgical model can be established from representative samples, hopefully equidistant with each other, obtained from existing drilling campaigns, as well as programmed new ones.

- Further metallurgical testwork should be carried out, in order to study different parameters, such as the impact of grind size in flotation and cyanidation and the effect of the scheme of reagents addition and/or pulp density in gold and silver recoveries. Also it is recommended to study the effect of regrinding size in cleaner flotation and in cleaner tails cyanidation

2.0 INTRODUCTION

Rodrigo Mello, NCL Ingeniería y Construcción Limitada (NCL) and Alquimia Conceptos S.A. (Alquimia) were commissioned by Greystar Resources Limited. (Greystar) to prepare an independent Qualified Person's Review and NI 43-101 Technical Report (the Report) for the wholly-owned Angostura gold–silver project (the Project) located in Colombia (Figure 2-1).

The Report discloses the results of resource estimation of high grade veins and the preliminary economic assessment for underground mining operation completed on the Angostura Project in February 2011. Greystar will be using the Report in support of the press release published on March 18, 2011.

All measurement units used in this Report are metric, and currency is expressed in US dollars unless stated otherwise.

2.1 Qualified Persons

The following people served as the Qualified Persons (QPs) as defined in National Instrument 43-101, *Standards of Disclosure for Mineral Projects*, and in compliance with Form 43-101F1. The QPs responsible for the preparation of the Report are:

- Carlos Guzmán, Principal Mining Engineer (MAusIMM), was responsible for the overall preparation of the report.
- Rodrigo Mello, Independent Consulting Geologist (MAusIMM), was responsible for the preparation of the resource estimation and issues related with this discipline in Sections 13.4, 13.5, 13.6, 14 and 17.
- John Wells, Metallurgical Engineer (FSAIMM), provided an independent review and analysis of the metallurgy and process plant in Sections 16, 18.2, 18.3, 18.4.2, 18.5.2 and 18.5.3 of the report.
- Giovanny Ortiz, Exploration Manager from Greystar Resources Ltd. (MAusIMM) was responsible for the preparation of the geology, geological model, exploration and issues related with this discipline in Sections 7, 8, 9, 10, 11, 12 and 13.

Other Expert Persons:

- Americo Delgado, Superintendent of Metallurgy from Greystar Resources, was responsible for the metallurgical testwork program and the review of the process plant design and the issues related with this discipline in this report.

Figure 2-1: Project Location Plan

2.2 Site Visits

Mr. Rodrigo Mello from NCL visited the Angostura site and the Greystar office in Bucaramanga, Colombia, from June 15th to 20th, 2009 and from February 2nd to 5th, 2010. During the first visit Mr. Mello reviewed the exploration procedures carried on the Angostura project and supervised the first exercise of resource estimation with Mr. Giovanny Ortiz, Greystar's exploration manager. In the second visit in February 2010, the definitive resource estimation was completed and the procedures used for the construction of the oxidation model were defined. During the same visit, a representative portion of the database was audited, comparing the electronic records with the original logs and analytical certificates.

Mr. Carlos Guzman visited the project site on August 26, 2010 to familiarize himself with the Angostura project, including the exploration tunnel Perezosa II.

Mr. John Wells visited Angostura project between January 9 and 12, 2010 to familiarize himself with the project and the metallurgical testwork program.

2.3 Effective Dates

The Report effective date is taken to be the date of completion of the Mineral Resource Estimation and is 28 February, 2011.

2.4 Previous Technical Reports

Greystar has previously filed the following technical reports on the Project:

- Mello, R., and Felder, F., 2010: Mineral Resource Estimate, Angostura Gold-Silver Project, Santander, Colombia: NI 43-101 technical report prepared by NCL Ingeniería y Construcción S.A. for Greystar Resources Limited, effective date 25 August, 2010.
- Greig, D., Alfaro, M., Munoz, E., McPartland, J., and Miranda, D., 2009: Angostura Gold Project, Preliminary Feasibility Study Technical Report NI 43-101: NI 43-101 technical report prepared by GRD Minproc for Greystar Resources Limited, effective date 5 May 2009.
- Sironvalle, M.A., 2009: Technical Report, December 8, 2008, Mineral Resource Estimate, Angostura Gold Project, Santander, Colombia: NI 43-101 technical report prepared by Metálica Consultores S.A. for Greystar Resources Limited, effective date 21 January 2009.

- Thalenhorst, H., 2008: Technical Report December 1, 2007 Mineral Resource Estimate Angostura Gold Project, Santander Colombia: NI 43-101 technical report prepared for Greystar Resources Limited, effective date 31 January 2008
- Wells, J.A., Watson, K., Tough B., Thalenhorst, J., and McPartland, J., 2007: Angostura NI 43-101 Independent Technical Report: revised NI 43-101 technical report prepared by Hatch Engineering Limited for Greystar Resources Limited, effective date 19 July 2007
- Wells, J.A., Watson, K., Tough B., Thalenhorst, J., and McPartland, J., 2007: Angostura NI 43-101 Independent Technical Report: NI 43-101 technical report prepared by Hatch Engineering Limited for Greystar Resources Limited, effective date 13 July 2007
- Thalenhorst, H., 2006: Technical Report Updated Mineral Resource Estimate Angostura Gold Project Santander Colombia: NI 43-101 technical report prepared by Strathcona Mineral Services Limited, effective date 30 August, 2006
- Burns, N., 2005a: Resource Update, Angostura Project, Santander, Colombia, September 14, 2005: NI 43-101 technical report prepared by Snowden Mining Industry Consultants for Greystar Resources Limited, effective date 14 September, 2005
- Burns, N., 2005b: Technical Report for the Angostura Project, Santander, Colombia: NI 43-101 technical report prepared by Snowden Mining Industry Consultants for Greystar Resources Limited, effective date 11 April, 2005
- Thalenhorst, H., 2004: Technical Report Mineral Resource Estimate Angostura Gold Project Santander Colombia: NI 43-101 technical report prepared by Strathcona Mineral Services Limited, effective date 27 August, 2004
- Thalenhorst, H., and Stone, B.G., 2003: Technical Report on the 1999 Resource Estimate Prepared by Kinross Technical Services, Amended and Restated: NI 43-101 technical report prepared by Strathcona Mineral Services Limited, effective date 24 September 2003
- Thalenhorst, H., 2002: Angostura Gold-Silver Project, Colombia – Updated Review of the 1999 Mineral Resource Estate Prepared by Kinross Technical Services: NI 43-101 technical report prepared by Strathcona Mineral Services Limited, effective date 17 May 2002

2.5 References

The primary reference source for Report preparation is:

RODRIGO MELLO AND NCL, 2011: MINERAL RESOURCE ESTIMATE AND PRELIMINAR ECONOMIC ASSESMENT FOR UNDERGROUND MINING.

ANGOSTURA GOLD-SILVER PROJECT, SANTANDER, COLOMBIA: unpublished internal study prepared by Rodrigo Mello and NCL for Greystar Resources Limited, 28 February 2011.

NCL, Alquimia 2011: Greystar Resources, Angostura Underground Mine Scoping Study, Final Report, January 2011. Unpublished internal study.

In addition, reports and documents listed in the Reference section of this Report were used to support the press release published on March 18, 2011.

2.6 Technical Report Sections and Required Items under NI 43-101

Table 2.6-1 relates the sections as shown in the contents page of this Report to the Prescribed Items Contents Page of NI 43-101. The main differences are that Item 25 “Additional Requirements for Technical Reports on Development Properties and Production Properties” is incorporated into the main body of the Report, following Item 19, “Mineral Resource and Mineral Reserve Estimates”.

Table 2.6-1: Contents Page Headings in Relation to NI 43-101 Prescribed Items—Contents

NI 43-101 Item Number	NI 43-101 Heading	Report Section Number	Report Section Heading
Item 1	Title Page		**Cover page of Report**
Item 2	Table of Contents		**Table of contents**
Item 3	Summary	Section 1	**Summary**
Item 4	Introduction	Section 2	**Introduction**
Item 5	Reliance on Other Experts	Section 3	**Reliance on Other Experts**
Item 6	Property Description and Location	Section 4	**Property Description and Location**
Item 7	Accessibility, Climate, Local Resources, Infrastructure and Physiography	Section 5	**Accessibility, Climate, Local Resources, Infrastructure and Physiography**
Item 8	History	Section 6	**History**
Item 9	Geological Setting	Section 7	**Geological Setting**
Item 10	Deposit Types	Section 8	**Deposit Types**
Item 11	Mineralization	Section 9	**Mineralization**
Item 12	Exploration	Section 10	**Exploration**
Item 13	Drilling	Section 11	**Drilling**
Item 14	Sampling Method and Approach	Section 12	**Sampling Method and Approach**
Item 15	Sample Preparation, Analyses and Security	Section 13	**Sample Preparation, Analyses and Security**
Item 16	Data Verification	Section 14	**Data Verification**
Item 17	Adjacent Properties	Section 15	**Adjacent Properties**
Item 18:	Mineral Processing and Metallurgical Testing	Section 16	**Mineral Processing and Metallurgical Testing**
Item 19	Mineral Resource and Mineral Reserve Estimates	Section 17	**Mineral Resource and Mineral Reserve Estimates**
Item 20	Other Relevant Data and Information	Section 19	**Other Relevant Data and Information**
Item 21	Interpretation and Conclusions	Section 20	**Interpretation and Conclusions**
Item 22	Recommendations	Section 21	**Recommendations**
Item 23	References	Section 22	**References**
Item 24	Date and Signature Page	Section 23	**Date and Signature Page**
Item 25	Additional Requirements for Technical Reports on Development Properties and Production Properties	Section 18	**Additional Requirements for Technical Reports on Development Properties and Production Properties**
Item 26	**Illustrations**		**Incorporated in Report under appropriate section number**

3.0 RELIANCE ON OTHER EXPERTS

The QPs, authors of this Report, state that they are qualified persons for those areas as identified in the "Certificate of Qualified Person" attached to this Report.

The QPs have fully relied upon and disclaim information relating to the surface rights status for the Project through the information presented by Greystar legal department.

4.0 PROPERTY DESCRIPTION AND LOCATION

4.1 Location

The Project is located approximately 400 km north–northeast of the Colombian capital city of Santa Fé de Bogotá, and approximately 67 km northeast of the city of Bucaramanga. The project area and the area for which mineral resource have been estimated fall within the Santander Department.

Project centroids are 7º 23' north and 72º 54' west.

4.2 Mineral Tenure

Greystar holds 14 concessions covering over 31,000 ha (Table 4.2-1, Figure 4.2-1 and Figure 4.2-2) which are held in its Branch in Colombia.

Greystar has acquired, through purchase and direct acquisition, a 100% interest in all the mineral licenses and permits itemized in Table 4.2-1. Greystar acquired the original Permit 3452 covering an area of 250 ha in the Municipality of California, Santander, Colombia through a purchase agreement dated September 8, 1994. On 17 April, 2006 Greystar submitted an application to Ingeominas to integrate the titles in the original group application into a new concession, to include claim fractions. License 3452 was converted to an Integration Mining Concession No. 3452 contract (Permit 3452) with the Colombian Government on February 14, 2007 and registered in the National Mining Register on August 9, 2007, and is known as the "Angostura Block". In april 2010, Greystar submitted an extension application for the exploration phase of mining title 3452 for an additional two years. This application was approved by Ingeominas in December 2010 therefore the exploration phase will expire on August 8 2012.

Excluded from the Angostura Block, but enclosed within it, are two Mining Licenses, L101-68 and L127-68 which are currently subjet to the 1988 mining code. Permit 3452 incorporates the following titles previously held by Greystar as individual licences: 110-68, 102-68, 140-68, 302-68, 3452, 13929, 45-68, 47-68, 13356, 300-68, HDB-082, GB3-091 and 370-68. The total Permit 3452 area is 5,244.9 ha, and provides for gold, silver and other precious metals exploitation.

Table 4.2-1: Mineral Tenure Summary Table

License	Designation	Area (ha)	Expiry Date	Notes
CC 3452	Concession	5,244.9	August 8, 2027	1, 2
L 101-68	Exploitation License	5.7	April 19, 2010	3-4
L 127-68	Exploitation License	3.5	April 19, 2010	4
CC 6979	Concession	40.0	July 09, 2026	
L 300-68	Exploration License	9.2	October 13, 2008	5-6
L 22346	Concession	1,184.1	June 17, 2026	
CC AJ5-142	Concession	4,061.1	November 14, 2034	7
CC AJ5-143	Concession	3,890.5	June 21, 2037	7
CC AJ5-144	Concession	4,336.0	February 11, 2037	7
CC EJ1-159	Concession	814.9	March 8, 2037	
CC EJ1-163	Concession	8,424.7	March 15, 2037	
CC EJ1-164	Concession	1,439.3	May 23, 2037	
CC 343	Concession	600.0	February 9, 2037	
L 13921	Exploitation Licence	78.63	December 17, 2013	
Totals		**31,132.53**		

(1) Angostura Block. Validity of the exploration phase until August 8 2012.
(2) Two of the original claims incorporated into the Angostura Block are subject to a net profits royalty (NPI). These are the original Permit 3452 (7.5% NPI on 230 ha) and concession 47-68 (10% NPI on 53.9 ha).
(3) The concept of mining licenses has been abandoned as part of the 2001 Mining Code revision, but there are three Greystar mining licenses that continue under this designation until their expiry date.
(4) These exploitation licenses are under application of renewal of extension of the mining concessions for an additional ten year period.
(5) Regarding the exploration licenses herein, as defined by the 1988 Mining Code and as defined in the 2001 Mining Code, they grant the holder the exclusive right to conduct exploration activities. The exploration licenses formerly held by Greystar at the Angostura site have now been integrated and incorporated into the Angostura Block (Concession 3452). Under 2001 Law prior to their expiry of the term of the exploration stage of the Mining License, and in order to qualify for the designation as a Concession Contract, a production plan (PTO) and an environmental impact study must be submitted. Also, an exploration license can be converted into a mining license after 10 years under the 1988 rules, or into a Concession Contract.
(6) Change to concession contract in process.
(7) Application in process for the addition of limestone as new mineral for exploration.

Figure 4.2-1: Mineral Tenure Plan

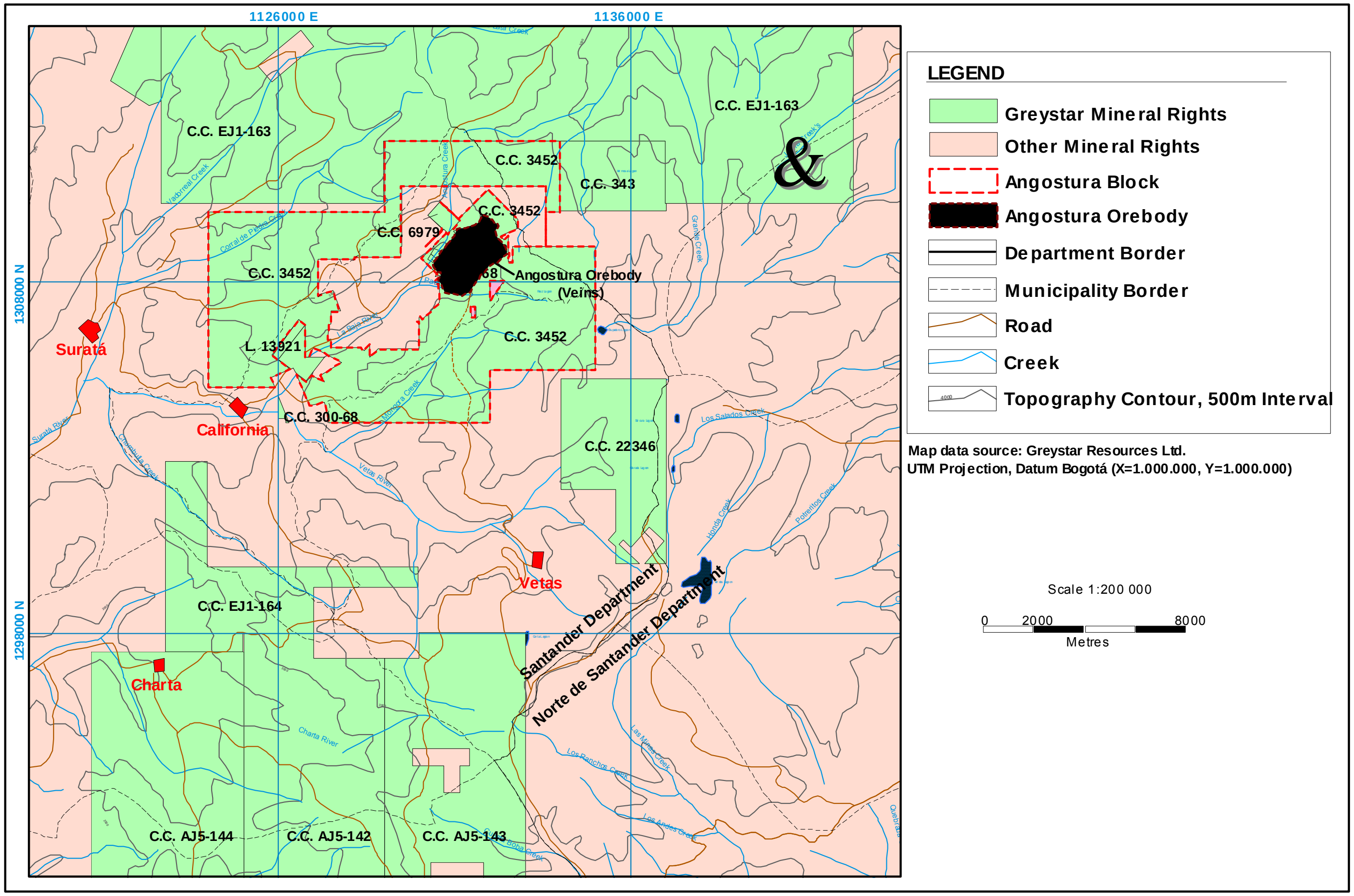
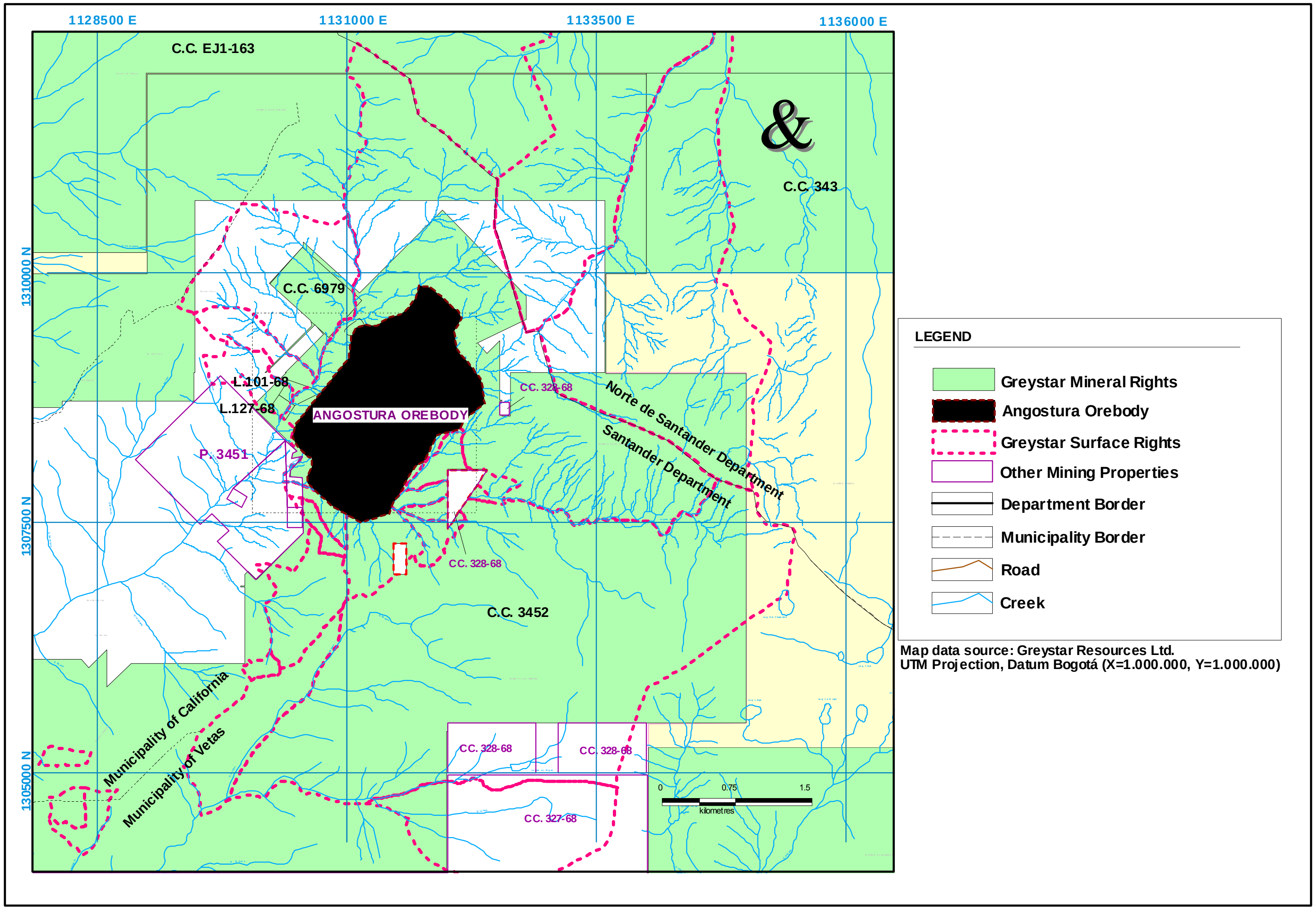


Figure 4.2-2: Surface Rights Plan – Angostura Block Area



Under the integrated concession, concluded in 2007, Permit 3452 had a three-year period, extended for a further two years until August 8 2012, to finish exploration, and start construction before going into production. All obligations and exploration conditions under the licenses incorporated into the new Permit 3452 were successfully fulfilled.

The term of the Concession Contract of the Angostura Block expires on 8 August 2027. Under the 2001 Mining Code, Greystar can then apply for a further 30 year extension of the contract.

Greystar holds mineral exploration rights covering approximately 1,833 ha located adjacent to Permit 3452, including exploitation licenses 101-68, 127-68, and 6979 covering a total area of 49 ha.

Two contracts requiring annual fee payments that are based on the number of hectares and a Colombian wage factor which fluctuates on an annual basis have the fees payable for 2011 as shown in Table 4.2-2.

Greystar also has concession contracts. These contracts require annual fee payments based on the number of hectares and a Colombian wage factor which fluctuates on an annual basis. Each of the concession contracts is for an initial exploration period of three years from the date of registration, with an option to extend for an additional eight years in two-year periods. The total period of these concession contracts is approximately 30 years. The fees payable for 2011 are summarized in Table 4.2-3.

In November 2009, Greystar entered into an option agreement with a private Colombian company to acquire the La Plata property, an area of 78 ha. The consideration was US$1.9 million, 160,000 share purchase warrants and minimum annual exploration expenditure commitments over a four-year period. Greystar is also required, if an economic deposit is identified in the property, to pay a one-time payment of US$7 per ounce of gold and US$0.10 per ounce of silver for extractable reserves up to a maximum of 750,000 ounces. Greystar has competed payments of the property and transfer of 100 % of the property has been undertaken.

The vertices of the mining titles polygons have been defined by topographic surveying and appropriately defined with ground markers. This process was approved by Ingeominas.

Table 4.2-2: Contracts Payable, Mineral Exploration Rights Areas

Contract Area	(Ha)	Expiration date	Annual Fee (US$)
22346	1,184.1	September 18, 2032	10,387
343	600.0	February 8, 2037	5,263

Table 4.2-3: Contracts Payable, Mineral Exploration Rights Areas

Contract Area	(Ha)	Expiration date	Annual Fee (US$)
3452	5,244.9	August 08, 2027	138,026
EJI-159	814.9	March 08, 2037	7,149
EJI-163	8,424.66	May 15, 2037	221,708
EJI-164	1,439.34	May 23, 2037	12,626
AJ5-142	4,061.1	November 14, 2034	71,250
AJ5-143	3,890.5	June 21, 2037	68,256
AJ5-144	4,336.0	February 11, 2038	76,072

4.3 Surface and Water Rights

Currently Greystar has outright ownership of aproximately 3,700 ha subject to certain deferred payments being made. These are summarized in Table 4.3-1 and were shown in Figure 4.2-1. Figure 4.2-1 also shows the location of the Angostura Project in relation to the acquired surface rights.

Greystar has implemented clear land and property acquisition procedures which must be applied when purchasing lands for mining, access or other activities which are required for project development. The land acquisition procedures generally follow the recommendations laid out in IFC Performance Standard 5.

Table 4.3-1: Surface Rights Acquisition Summary Table

Purchased Rural Land			
#	**Land Property Name**	**Area (ha)**	**Municipality/Department**
1	Angostura (consolidation of several lots)	736.09	California/Santander
2	Padilla	15.98	Suratá/Santander
3	La Herrera	18.52	California/Santander
4	La Casita	31.00	California/Santander
5	Romeral-Carrizal	383.50	Cucutilla/Norte de Santander
6	Romeral	535.55	Cucutilla/Norte de Santander
7	La Armenia	175.24	California/Santander
8	Miraflores	36.95	California/Santander
9	La Casita	11.11	California/Santander
10	La Berenciana	16.23	California/Santander
11	Cruz	8.80	California/Santander
12	Los Llanitos	14.63	California/Santander
13	Esmeralda – DIVISO	8.89	California/Santander
14	El Cadillal	68.95	California/Santander
15	El Bosque	9.00	California/Santander
16	El Salibal	28.83	California/Santander
17	Carbón	89.98	Vetas/Santander
18	Las Pavas	6.19	Vetas/Santander
19	Los Robles	14.57	California/Santander
20	Las Puentes (*)	1,034.35	Vetas/Santander
21	La Esmeralda (*)	86.38	California/Santander
22	El Jordán-El Carbón	34.85	Vetas/Santander
	TOTAL	**3,365.60**	
(*) Right over the land property as part of a process of succession			
Purchased Urban Land (In California Town)			
#	**Land Property Name**	**Area (ha)**	
1	Lot Cra 4 No 3-10-California (Greystar's House)	0.19	California/Santander
2	Lot San Francisco	6.37	California/Santander
3	Lot 6	0.07	California/Santander
4	Lot 7	0.07	California/Santander
5	Lot Cra 6 No. 3-26 and cll 4No. 5-41/45/49 (Core Shack)	0.3570	California/Santander
	TOTAL	**7.05**	
Agreement Signed – Pending Judicial Process			
#	**Land Property Name**	**Area (ha)**	
1	El Alta - Deed 2176 Reg. 300-242	30.68	California/Santander
2	La Virgen	188.72	Vetas/Santander
3	San Julián	18.00	California/Santander
4	Padillo	3.54	Suratá/Santander
	TOTAL	**240.94**	
(*) Right over the land property as part of a process of succession			

Greystar currently holds three water licenses to carry out exploration works in the Angostura Block. Currently la Plata area has a water license held by Sociedad Minera

La Plata which will be requested for assignment to Greystar. The Company has 8 water rights under request before the environmental authority.

4.4 Rights of Way and Easements

The Colombian Mining Code grants broad rights to the owner of a mining concession to establish easements or rights of way for exploration activities and mine infrastructure construction.

In accordance with applicable law, the owner of a mining concession is entitled to request from judicial authorities the application of easements or rights of way, as well as to request expropriation of lands needed for the project, when is not possible to have an agreement with the land owner.

4.5 Royalties

The underlying vendors of original License 3452 retained a 10% net profits royalty.

The underlying vendors of License 47-68 covering an area of approximately 54 ha hold a 10% net profits royalty.

During 2008, Greystar purchased one-half of the 10% net profit royalty in the original License 3452 from one of the underlying vendors in consideration for $850,201 (US$800,000) and issued 100,000 common share purchase warrants. As at December 31, 2009, one underlying vendor of the original License 3452 covering an area of approximately 150 ha holds a 5% net profits royalty while the second underlying vendor covering an area of approximately 100 ha retains a 10% net profits royalty.

In addition, a royalty will be payable to the Colombian Government. According to Colombian Law, exploitation for gold production is subject to a 4% royalty on 80% of the London Price Fixing for the gold and silver production at pithead.

4.6 Permits

The Project requires that a work and investment plan (PTO) be prepared and approved prior to any exploitation activities being permitted.

Greystar submitted an application under the 2001 Mining Code for the PTO based on the 2009 pre-feasibility study (2009 PFS) on October 23, 2009. Ingeominas is the government agency that evaluate and approve the PTO, and could be parallel process to an environmental permitting approval. On March 23, 2011 Greystar withdrew the

PTO application because the company considered it necessary to reformulate the project addressing the government and the community's concern. Greystar will study the viability of alternative options for the project, including the underground exploitation option, considered in the scoping study presented in this report.

4.7 Environment

4.7.1 Exploration Activities

Field and exploration activities are permitted for the Angostura Project under an approved environmental management plan or “plan de manejo ambiental” (PMA).

Greystar submitted required environmental action plans to the CDMB, including an environmental plan on January 20, 2004 for the underground development on the 2,850 level at Perezosa and on November 2, 2007 for an environmental plan for the Veta de Barro tunnel at level 3,095. Greystar has been granted all necessary permits for field activities, including for drilling in the Móngora and Animas areas.

4.7.2 Development Activities

Greystar filed the EIA for a open pit operation (based on the pre-feasibility study) on December 22, 2009. The Ministerio de Ambiente, Vivienda y Desarrollo Territorial (MAVDT) initially requested that the EIA be amended to comply with provisions of the 2010 Mining Code amendments, which according to MAVDT, prohibit mining and exploration activity within any “paramo" ecosystem. However, in early 2010, this decision was reversed by MAVDT, and study continued.

Part of the planned pit and associated mine infrastructure are located within the “paramo” ecosystem, based on cartographic co-ordinates defined by the Alexander Von Humbolt Investigation Institute. However, the competent environmental authority for Colombia has not legally defined the “paramo” for the Project area. Declaration is contingent upon technical, social, and environmental studies.

Two public Project information hearings have been held, on 3 November and 4 November, 2010, in the municipalities of California and Vetas respectively. A public hearing was held in the municipality of California on 21 November, 2010. MAVDT requested an additional public consultation be held in Bucaramanga on 4 March, 2011. The second public hearing in Bucaramanga was terminated prematurely due to disorders presented during the event. As with the PTO, Greystar withdrew the EIA application from the MAVDT on March 23, 2011.

When Greystar defines a new exploitation project, a new EIA application will have to be presented to the competent environmental government agency. If the project is considered as a large mining operation, more than 2 million tonnes of rock movement per year, the MAVDT will be the authority in charge of evaluating and approving the environmental license. For small or medium mining operation, the EIA is evaluated and approved by the local environmental government agency, Corporación Autónoma

Regional para la Defensa de la Meseta de Bucarmananga, CDMB, established in Bucaramanga.

4.7.3 Project Design Principles

Greystar will apply environmental standards recognized by the international community for the Angostura Project and will adopt these standards for pre-feasibility and feasibility engineering studies. Standard environmental design criteria will be used in all stages of the Project in order to meet both national and international requirements and minimize the potential environmental and social impacts that might result from the Project development. International criteria being followed include those from the International Finance Corporation, the World Bank Equator Principles and the International Cyanide Management Code.

5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Accessibility

Current Project access from Bucaramanga is via the partially-paved Matanza–Surata–California road, a distance of 67 km and travel time of two to three hours, depending on weather conditions. Within the Project area, access is by a network of unpaved tracks and horse and foot trails.

Bucaramanga has an airstrip, with daily flights to Bogota, Medellin and Panama City. Helicopter flights to the Project are also conducted from Bucaramanga.

The closest port is Santa Marta, 550 km from the Project site.

5.2 Climate

The Project experiences two wet seasons, from April to June and from September to November. January is the driest month, April the wettest. The climate is highland tropical, with the average annual temperature being 8.7–9.8°C, ranging between 0.8°C and 26.9°C. Rainfall averages 1,182 mm, and the evaporation rate is about 576 mm. Relative humidity averages 73%.

5.3 Local Resources and Infrastructure

The Angostura Project is located in a relatively undeveloped region in the department of Santander. The closest communities are the small towns of California, 1,800 inhabitants; Vetas, 2,400 inhabitants, Suratá, 3,500 inhabitants and Matanza, 5,700 inhabitants. These towns can provide basic services. Bucaramanga, 1,500,000 inhabitants, can supply goods and services.

5.4 Physiography

The Project is located in steep and relatively rugged mountainous terrain at elevations ranging from 2,600 masl to 3,800 masl.

The Project is situated at the upper end of the La Baja creek drainage basin, a catchment area of approximately 124 km^2 above the town of California. The local catchment area of the Angosturas creek drains an area of approximately 10 km^2.

The principal economic activity in the area is the small-scale exploitation of gold, while agriculture, cattle husbandry and basic commercial activities are of lesser significance. Agriculture is carried out using traditional methods with low yields, and cattle are primarily grown for meat production.

Vegetation in the higher part of the Angostura Project area can be described as light “alpine scrub” consisting of grasses and shrubs such as "Frailejon" (*espeletia humbolt*), typical for the high elevations of the northern Andes mountains of Venezuela, Colombia, and Ecuador. There is significant growth of oak trees along the watercourses in the lower elevations on the Project.

6.0 HISTORY

Early mining activity comprised artisanal activities that ranged from pre-Colombian time, and later Spanish excavations. At the end of World War 1, the British company Colombian Mining Association and French company Francia Gold and Silver operated in the area.

In 1947, The Anaconda Company (Anaconda) completed detailed geologic surface and underground mapping and core drilling (746 m) between the La Baja and La Alta areas. The Nippon Mining Company in 1967 undertook drilling in the La Baja area. Exploration activity was undertaken by Placer Development and Ingeominas in the 1970s and 1980s respectively.

Modern exploration by Greystar commenced in 1994, and to 1999, geologic mapping, surface rock sampling, core drilling (181 drill holes, 52,000 m), and metallurgical testwork were completed. A small part of the underground development created by artisan miners was mapped and sampled, and based on areas that were safely able to be inspected, about 13,000 t at about 8 g/t Au has been excavated. Mineral resource estimates were undertaken in 1997, and updated in 1999. An engineering study, termed a “pre-feasibility study” at the time was also undertaken in 1998, and envisioned either; an open pit/ heap leach operation, or an open pit feeding agitated leach and heap leach facilities. Kinross Gold Corporation, who at the time was a significant shareholder in Greystar, performed a mineral resource estimate update in 1999.

From 2000 to 2003, due to security constraints, no work was undertaken. From 2003, work has included geochemical sampling, geologic mapping, adit and tunnel excavation, core drilling, and condemnation drilling. Mineral resource estimates were performed in 2005, 2006, 2007, 2008, and 2010. Preliminary assessment (PA) studies were completed in 2008. Mineral reserve estimation was undertaken in 2009 together with additional metallurgical testwork.

The pre-feasibility study, completed in 2009 by GRD Minproc (now part of AMEC), envisaged open pit mining, followed by a conventional process flowsheet using two process routes, cyanide heap leaching of oxide, transition and low sulfur ore to produce doré, and grinding and flotation of high sulfur/high gold content ore to produce concentrates. Based on the assumptions in the study, the Project returned positive economics.

A feasibility study for a open pit operation was commissioned during 2010 and completed in 2011. This study did not progress to implementation and some of the technical studies executed in this phase will be used to support the evaluation of alternative exploitation options for the project.

Greystar withdrew the Environmental Impact Assessment (EIA) and the work and investment plan for exploitation (PTO) from the MAVDT and Ingeominas respectively. Both studies were prepared based on the exploitation project defined in the pre-feasibility study.

The company considered that regional and national government and the community of Bucaramanga did not support the project as configured for an open pit operation. Greystar will study the viability of alternative options for the project, including the underground exploitation option, considered in the preliminary economic assessment presented in this report.

7.0 GEOLOGICAL SETTING

7.1 Regional Geology

The Project is situated in the northern Andes ranges, north of the point of bifurcation of the Eastern Cordillera into the western and eastern branches. The eastern branch, hosting the Project, includes the Santander Massif. The oldest rocks in the Massif are Precambrian gneisses and schists that were part of the Guyana Shield, and which have been regionally metamorphosed to upper amphibolite grade in the Palaeozoic.

Intruding the metamorphic rocks are diorite to granite composition rocks that belong to the Triassic–Jurassic Santander Plutonic Group. These intrusions were accompanied by felsic to andesitic volcanism. During subsequent back-arc development, a number of basins formed, and were filled with marine transgressive sediments. During the Late Cretaceous to Paleocene/Eocene, folding and thrusting of the Eastern Cordillera resulted in basin inversion and uplift, and intrusion of Middle Miocene porphyritic bodies of rhyodacitic and dacitic composition.

Uplift and erosion occurred during the Late Eocene to Early Oligocene, with reactivation of older structures and continued uplift during the Middle to Late Miocene. As a result of ongoing tectonic plate movements, the area is currently undergoing additional deformation, with rapid basin inversion and uplift.

Gold mineralization occurs within the Angostura–California gold province, a belt of epithermal gold occurrences that has developed along the regional-scale Rio La Baja fault in association with the Middle Miocene stocks.

A regional geology plan of the Project area is shown in Figure 7.2-1.

7.2 Deposit Geology

7.2.1 Lithologies

The Angostura deposit currently has a strike extent of 2 km, a width of 1 km, and has currently from the 2,400 masl level and up to 3,470 masl. The deposit is delimited to the northwest by the Angostura Fault and to the southeast by the Móngora Fault. Mineralization continues southward across the Páez Fault, but the steep topography provides an impediment to exploration drilling. To the north, the deposit appears to terminate fairly abruptly against an unnamed fault in the Cristo Rey area, beyond which only narrow, isolated veins have been encountered (Figure 7.2-2).

Figure 7.2-1: Regional Geology Plan

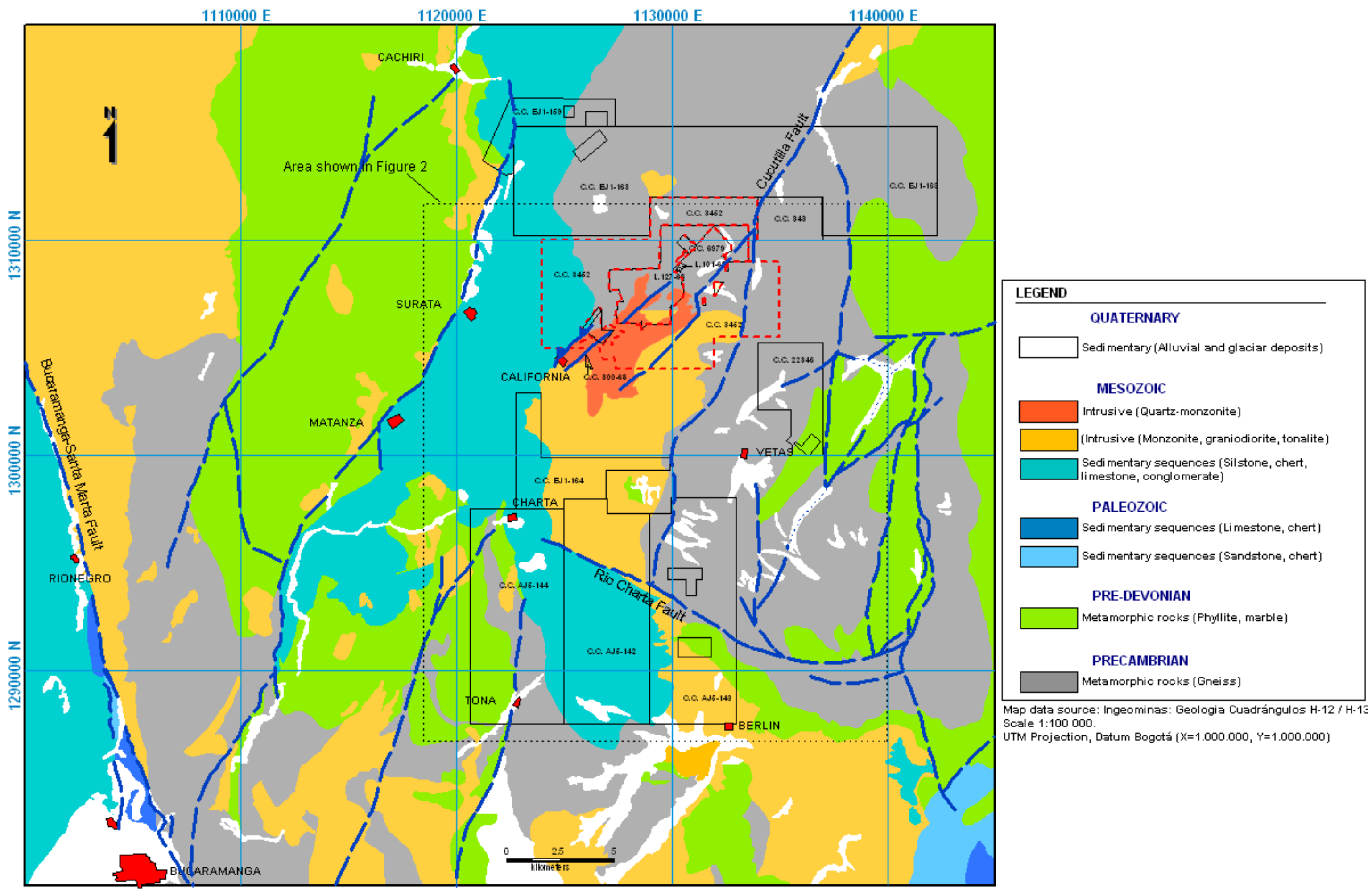
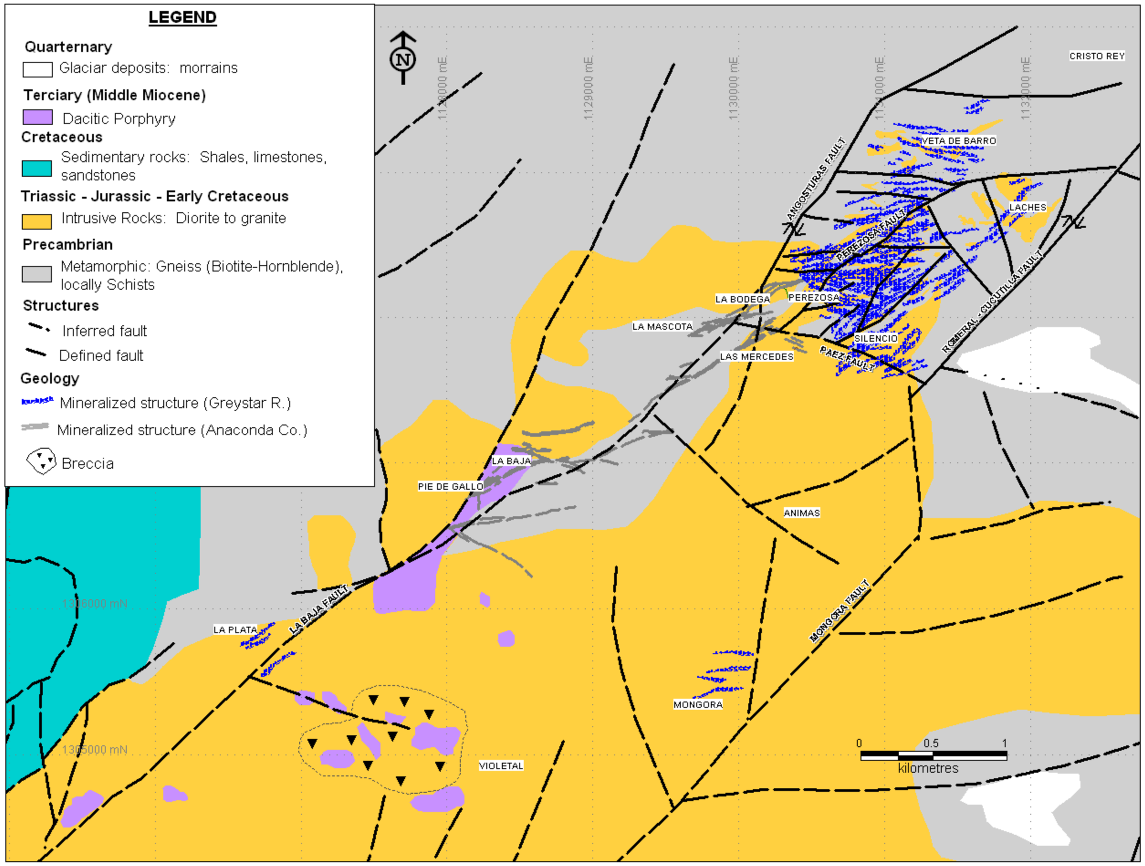


Figure 7.2-2: California District Geology Plan



A suite of porphyritic diorite to quartz monzonite bodies and dyke swarms of Triassic age, are intruded into the amphibolite facies Bucaramanga Gneiss, a series of meta-sediments of Proterozoic age. These rocks have been intersected by a swarm of generally east–northeast trending, steeply north-dipping structures. More than 200 individual veins and composite veins have been identified to date by means of surface and underground mapping, and interpretation of drill hole data.

Mineralization occurs in bands, veinlets, stringers, and silicified hydrothermal breccias within the structures. In the upper parts of the mineralized system, alteration and mineralization are stronger in the intrusive host rocks, and the meta-sediments appear to make a poorer host for the gold–silver mineralization.

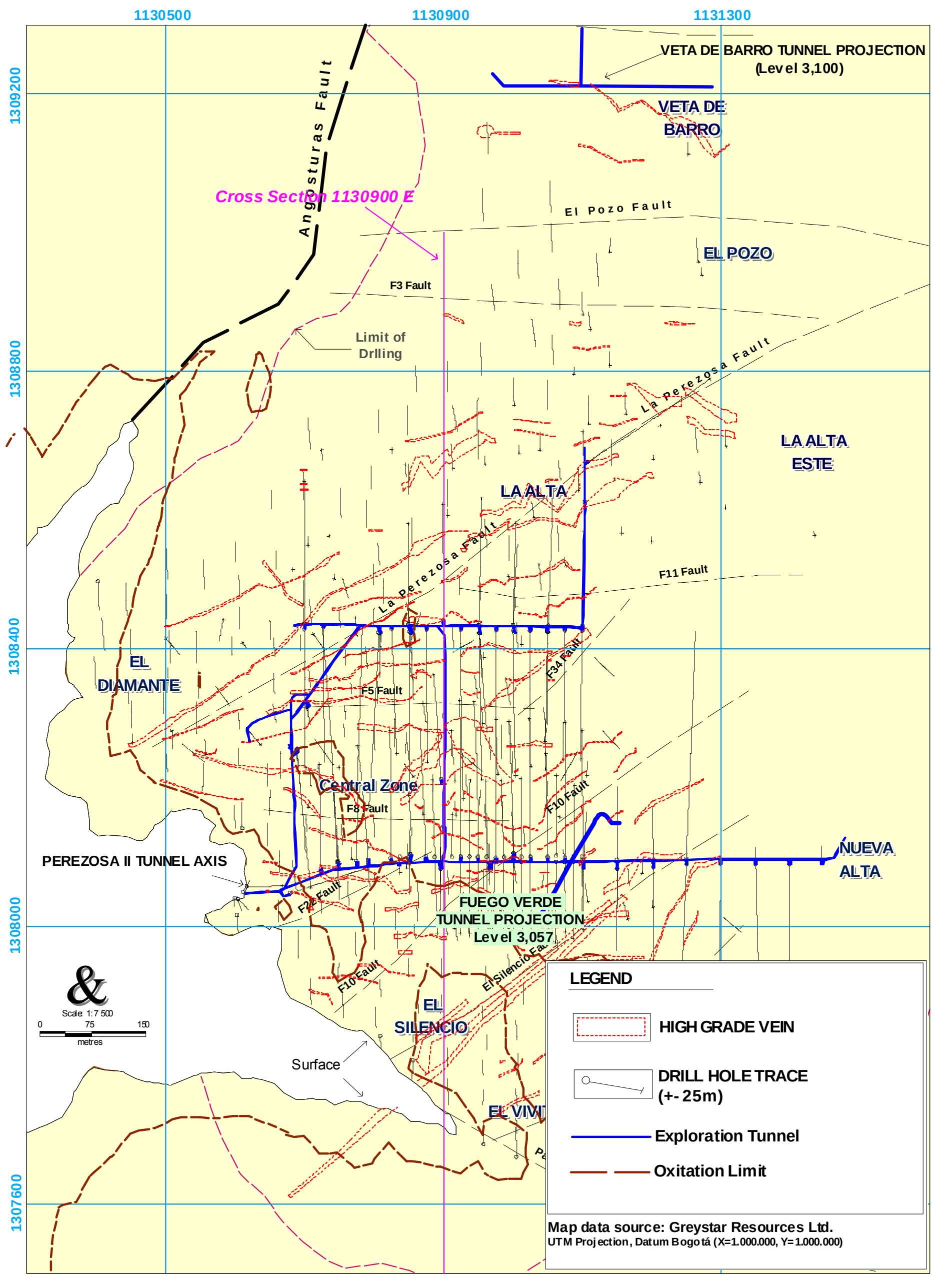
Mineralized structures vary from less than 2 m for individual veins to over 40 m for composite structures, and strike lengths range from less than 50 m to over 1 km. The intensity of fracturing, and the degree of secondary porosity and permeability of the host rocks controls the density of structures, and therefore of mineralization. Flexures along mineralized structures, vein–vein intersections, and vein–fault intersections are preferred mineralization sites typically display higher gold and silver grades. Such higher-grade pods can display ranges from >2–30 m in width, 30–100 m in strike, and 30–300 m down-dip.

The Angostura deposit is sub-divided geographically into a number of areas or sections that, from south to north, are referred to as El Vivito, El Silencio, Nueva Alta, La Perezosa, El Diamante, La Alta and its eastern neighbour La Alta Este, El Pozo, Veta de Barro, Veta de Barro Este, and Cristo Rey.

Figure 7.2-3 shows an outline of the vein systems and sub-areas, at plan level 2,850 m. Figure 7.2-4 is a geological section through the deposit at 1,130,900 E.

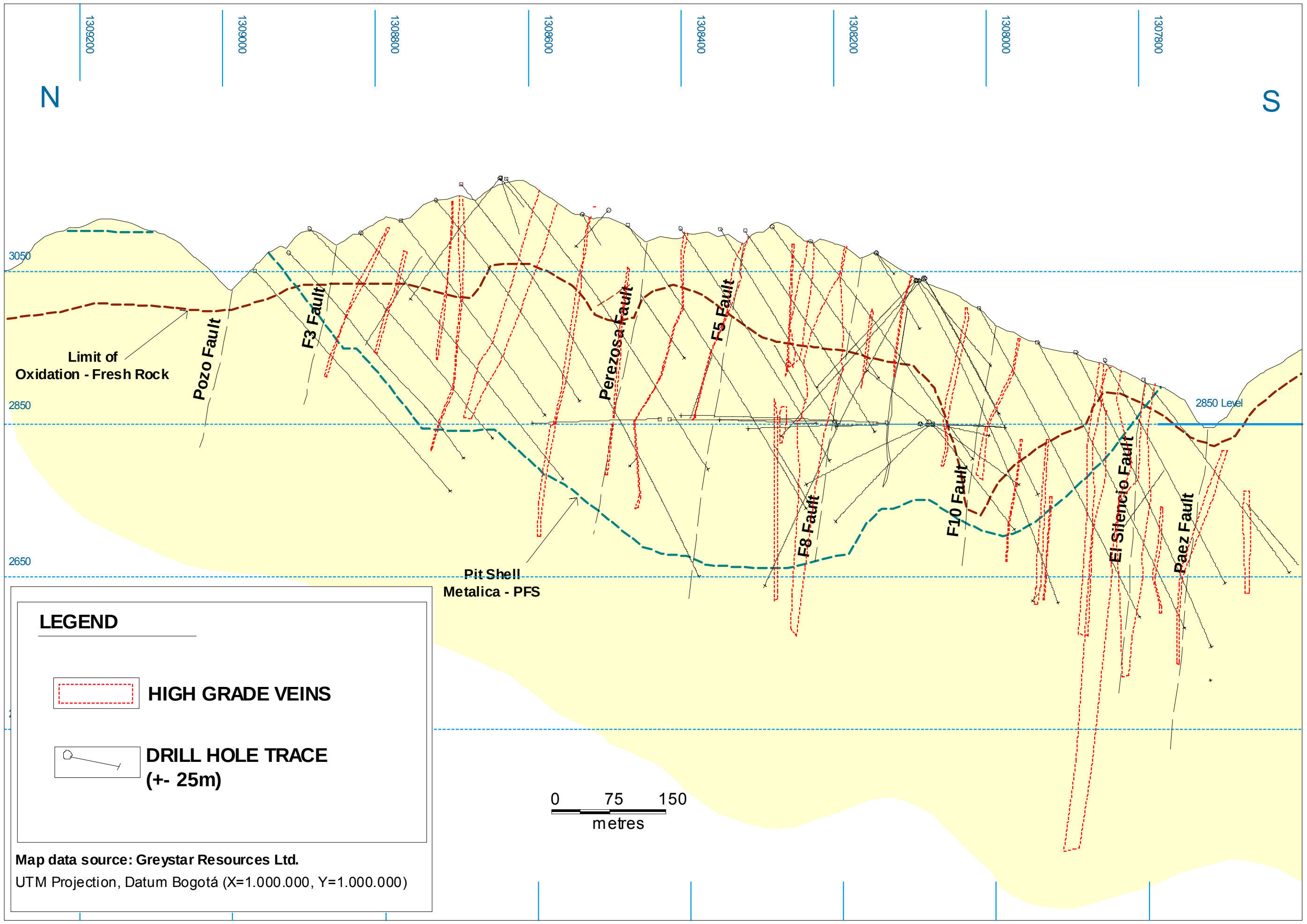
Surface oxidation has affected the rocks at Angostura to a depth of 10–30 m at the edge of the deposit, and attains depths that vary from 40 m to 100 m in its central parts. The oxidation profile is irregular, following increased permeability along mineralized structures and later faults and fractures, sometimes exceeding 400 m in depth along specific structures.

Figure 7.2-3: Detailed Geological Plan View, 2,850 Level



Note: Cross-section indicated in Figure 7.2-3 on plan is Figure 7.2-4 in this Report.

Figure 7.2-4: Geological Section, 1,130,900 E



7.2.2 Alteration

Hydrothermal alteration overprinting alteration processes related to regional metamorphism, such as chloritization and epidotization, is complex. Different intensity levels of alteration are observed in the gneiss, varying from partial alteration to a total replacement of the original constituents such as mafic minerals. The highest degree of hydrothermal alteration is found surrounding quartz–alunite veins, where alteration completely replaces the original host rock. More distal to these veins sericite, illite, smectite, kaolinite and finally chlorite can occur.

Illite was identified as the most common alteration mineral, followed by sericite. Alunite, kaolinite, chlorite and an interlayer illite–smectite are also frequent mineral phases. Some supergene alunite was observed.

The process of formation of the quartz veins is associated with partial silicification that is superimposed on the original altered rocks with patches of microgranular quartz. The most intensive alteration stage shows complete silicification of the rock, assimilating the primary quartz remnants with a final product of granular, sometimes vuggy, crustified quartz. A late event of high-temperature white quartz occurring in veinlets is observed in the lower levels of the deposit.

7.2.3 Structure

Northeast–southwest-trending right-lateral strike-slip faults are the major structural features of the Project area, and have defined a dilation zone that had increased ground preparation (porosity and permeability) for percolation of mineralizing fluids.

Five vein/fault stages were identified, from oldest to youngest:

- Northeast–southwest striking faults that have steep to moderate dips to the northwest and southeast
- Northwest and southwest striking structures that have dips from sub-vertical to about 60° to the northeast and southwest
- East–west to east–northeast–west–southwest striking structures that have dips ranging from 85° to 65° to the north and northeast, and south and southwest
- North–south and northeast–southwest striking, low-angle (50°–20°) structures that dip to the west, north and south
- Northwest–southeast, north–south, and northeast–southwest striking structures that have predominant steep to moderate dips to the west; east–west striking structures that dip steeply to the north and south.

7.3 Comment on Section 7

In the opinion of the QPs, knowledge of the deposit settings, lithologies, and structural and alteration controls on mineralization is sufficient to support Mineral Resource estimation.

8.0 DEPOSIT TYPES

The Angostura gold-silver project is part of the Angostura-California gold province of the Eastern Cordillera in northeastern Colombia, a belt of epithermal gold occurrences developed along the Rio La Baja regional fault that trends in a northeasterly direction from the town of California (Felder et al., 2000). The fault transects pre-Devonian granitoid bodies, high-rank metamorphic rocks and overlying Lower Cretaceous sediments – quartzite, shales, limestone and argillite. Middle Miocene (Tortonian) volcanic and subvolcanic stocks represent the last magmatic event in the California-Angostura district (Mantilla, et al., 2009; Leal, et al., this volume). These small, irregular shaped porphyritic bodies of rhyodacitic and dacitic composition have yielded radiometric ages of 8.4 ± 0.2 Ma and 9.0 ± 0.2 Ma (U-Pb) (Mantilla et al., 2009). Intense hydrothermal activity related with these porphyry intrusions gives rise to the gold¬ silver mineralization. The Figure 7.2-2 shows the geology of the district California – Angostura.

Angostura can be characterized as a structurally controlled, uplifted and deeply eroded, high-sulfidation epithermal gold deposit. The deposit is interpreted as being the lower part of a lithocap, as defined by Sillitoe (1995), that was eroded during uplift of the Northern Andes and that exposed the sericite – illite – quartz zone of a Cu-Mo-Au porphyry system, containing important contents of gold and silver, mainly associated to pyrite.

Most of the gold is contained within several sets of anastomosing veins and tabular silicified zones. The hydrothermal alteration in the center of the structures is mainly quartz-alunite, and more distal to this sericite, illite, smectite, kaolinite and finally chlorite, showing a gradual change from acid to basic alteration moving away from the veins.

The discovery of copper-molybdenum mineralization in an intrusive breccia some four kilometres to the south of Angostura suggests that the gold mineralization at Angostura may be associated with a porphyry system at depth. The exploratory works in this area have showed evidence of hydrothermal alteration, and contents of pyrite that suggest a potential that have to be properly investigated.

8.1 Comment on Section 8

In the opinion of the QPs, the Angostura deposit is considered to be an example of a high-sulfidation deposit because of the following features:

- Location in structural setting where second and later structures are associated with regional-scale crustal faults

- Spatial association with irregularly-shaped Middle Miocene porphyritic bodies of rhyodacitic and dacitic composition
- Acid-leached siliceous, clay and alunite-bearing 'lithocap' present; deposit has been eroded so that only the lower portions of the lithocap are exposed
- Alteration comprises intense silicification, sericitization, argillic alteration, locally advanced argillic alteration, and distal propylitic alteration
- Disseminated and fracture-controlled gold mineralization succeeded the main stage alteration
- High Grade Mineralization (Veins) is hosted by siliceous alteration
- Gold is associated with pyrite as the main sulfide phase
- Association of gold grades with elevated levels of silver, copper, arsenic (lower levels), bismuth, molybdenum and tellurium.

9.0 MINERALIZATION

At Angostura mineralization is controlled by a swarm of structures with generally NE-SW, E-W to ENE-WSW and NW-SE trends and commonly steep dips (Figure 7.2-3). The intensity of fracturing, and thus the degree of secondary porosity and permeability of the host rocks controls the density of mineralization containing important enrichment in high grade shoots. The width of individual high grade mineralized structures ("Veins") ranges from 2 m to more than 30 m, the lengths varies from less than 30 m to more than 500 m. Flexures of mineralized structures as well as vein-vein or vein-fault intersections are major sites for higher grade mineralization. The width of these high grade shoots is variable depending of the intensity of fracturing. The extent of these zones ranges from less than 5 m up to 30 m in width, 30 to 100 m in strike, and 30 m up to 300 m down dip. In general, these high grade shoots are irregular in shape and are flanked by lower grade material.

The mineralization is in bands, veinlets, stringers, silicified hydrothermal breccias and as stockwork composed predominantly of quartz, pyrite and alunite entirely replacing the primary host rock. Other ore minerals accompanying the pyrite dominant paragenesis include chalcopyrite, digenite, bornite, tetrahedrite (fahlore), marcasite, pyrrhotite and bismuthinite, and minor sphalerite, arsenopyrite, calcosine and coveline. Typical sulfide associations include pyrite-digenite-tethraedrite and pyrite-chalcopyrite (digenite, tetrahedrite); locally pyrite is the only sulfide.

At least two stages of pyrite formation have been recognized. The older is represented by relatively large crystals and does not appear to be associated with gold, or is so only moderately, while a younger, fine-grained pyrite/marcasite phase is intimately correlated with the intense stages of silicification and with gold deposition

Very fine-grained electrum and gold-silver tellurides occluded in pyrite are described by Thompson (2005b). Similarly SGS Lakefield Research Africa Pty (Lakefield Africa) have reported on the deportment of gold in the flotation concentrate from a composite sample of wide distribution within the Angostura deposit. Gold was only found as gold-telluride (probably calaverite – AuTe2) and as gold-silver telluride (probably petzite – Ag3 AuTe2). "No native gold or electrum was seen" (SGS Lakefield Africa 2007, page 6). Silver minerals identified included hessite (Ag2Te) and pearceite, a complex silver-copper-arsenic sulphosalt. At a grind of 80% passing 106 microns, about 30% of the observed gold-silver tellurides were still completely locked in sulfides (mainly pyrite). The fine¬grained character, and the frequent occurrence of gold in tellurides together explain the partly refractory nature of the primary mineralization at Angostura.

For this study the mineralized structures have been correlated as single veins, modelating the higher grade part of the structures. 203 individual veins were constructed using surface mapping, mapping of underground workings and interpretation of drill hole data. Widths vary from less than two metres for individual veins to over 30 metres, and identified strike lengths range from less than 50 metres to over 300 m. Figures 7.2-3 and 7.2-4 depict the interpreted economic geology of the deposit at the 2,850 m level and vertical section 1,130,900 E.

9.1 Comment on Section 9

The QPs are of the opinion that the mineralization styles and settings are well understood for the Project deposits, and can support declaration of Mineral Resources.

10.0 EXPLORATION

The Table 10-1 summarizes all the exploration work made by periods in Angostura and de data used for each resource estimation mad. Starting in 1994 the exploration had consisted of surface work which included geologic mapping, surficial rock sampling, soil sampling, stream sediment sampling and diamond drilling completing until March of 2011, of 326,894 metres in 992 diamond drill holes. 3,145 m of drifting have been constructed in the tunnels Perezosa 1, 32 m; Perezosa 2, 2,500 m (Figure 7.2-3) ; Veta de Barro, 415 m(Figure 7.2-3) and Fuego Verde (198 m). All the underground openings created by artisan miners in the Greystar's claims were also mapped and sampled. In 2010 13,068 m were drilled in Angostura, focused in the evaluation of the extensions in depth of the high grade structures in the areas of Los Laches, Cristo Rey, Veta de Barro and El Silencio. From January to March, 6,030 m have been completed continuing the 2010 program, focused in the high grade structures.

An underground development program was started in early 1997 with 198 metres in the the Fuego Verde tunnel on 3,056 metre elevation in the Silencio Area. In April 2004 on the 2,850-metre elevation consisting of two parallel east-west drives some 350 metres apart, with two connecting cross-cut in the Perezosa 2 tunnel. In 2008 415.3 metres and one cross cut were completed in the Veta de Barro tunnel located in the northern part of the deposit on 3 095 metres elevation.

Geochemical soil sampling campaigns have been undertaken in the project and some surrounding areas, and more than 4,000 samples have been taken on a grid with an initial spacing at 100 or 200 metres with later infill sampling. Samples were taken at an average depth of 0.8 m. The samples were analysed for 37 elements using ICP mass spectroscopy analysis of 15-gram aliquots after agua regia digestion. As a result of this work, gold anomalies were identified in such areas as Animas, Mongora, Violetal and La Plata (Figure 7.2-2).

Similar to the Angostura deposit, the Mongora prospect hosts higher-grade gold mineralization including for example 16.3 grams gold per tonne over 1.05 metres and 12.35 grams over 1.6 meters and 116 grams over 2 meters, within broader zones of lower-grade gold mineralization. The delineation for oxide and transitional gold mineralization at the Mongora area could be very important for the Angostura project. The potential of outlining a new oxide resource that could be added to the Angostura deposit resources could have favorable implications for the overall economics of the entire Angostura project. In 2010 38 drill holes, 13,264 metres, were drilled accumulating 57 drill holes and 19,549 metres since 2008. From January to March 2011, 402 m were drilled in this area as part of delineation drilling.

Table 10-1: Angostura Exploration Information by Period and Timing of Historical Resource Estimates

Period	Surface Sampling		Surface Diamond Drilling			Tunnelling			Underground Diamond Drilling		
	Metres Sampled	Assays	Holes	Metres Drilled	Assays	Metres (constructed)	Muck Samples	Channel Samples	Holes	Metres Drilled	Assays
1995 to late 1998	398	131	139	38 836	27 399	198	0	116	0	0	0
Mineral Resource Estimate MDA 1999 (KD Engineering Company, Inc., et al, 1999)											
Late 1998 and 1999	7 336	2 014	44	15 255	11 809	0	0	432	0	0	0
KTS Mineral Resource Estimate 1999 (Strathcona 2003 and predecessor reports)											
2000 to June 2003	756	275				0	0	343	0	0	0
June 2003 to May 2004			61	19 032	12 495	32	0	145	0	0	0
Greystar/Strathcona Mineral Resource Estimate May 2004 (Strathcona 2004)											
June 2004 to March 2005	1 272	738	61	23 041	13 763	476	322	356	12	2 463	1 556
Greystar/Snowden Mineral Resource Estimate March 2005 (Snowden 2005a)											
April to September 2005			38	13 353	7 000	369	214	169	14	2 817	1 681
Greystar/Snowden Mineral Resource Estimate November 2005 (Snowden 2005b)											
Sept. 2005 to June 2006			100	39 616	21 969	923	439	374	42	11 497	7 059
Greystar/Strathcona Mineral Resource Estimate June 2006 (Strathcona 2006)											
June 2006 to Dec. 2006			38	16 334	9 833	274	140	348	24	5 658	3 249
Hatch Scoping Study (Hatch Ltd., 2007)											
Dec. 2006 to Dec. 2007	87	42	121	50 423	27 353	228	295	718	23	6 118	3498
Greystar/Strathcona Mineral Resource Estimate Dec 2007 (Strathcona 2007)											
Dec. 2007 to May. 2008	167	76	111	27 148	15 015	458	54	150	42	5 409	3 014
Greystar/Metalica Mineral Resource Estimate Dec 2008 (Metalica 2,009)											
June. 2008 to July. 2010	2 726	1 031	68	25 834	13 120	187	304	165	0	0	0
Greystar / Rodrigo Mello; Mineral Resource Estimate for high grade veins (This report)											
Totals	12 742	4,307	**779**	**272 953**	**161 349**	3 145	1 768	3 316	**157**	**33 962**	**20 057**

La Plata comprises 78 hectares of mineral rights contiguous on the majority of its borders with the Issuer's existing mineral holdings (Figure 7.2-2).

The La Plata property lies within a mineralized belt related to the northeast-southwest trending La Baja Fault, which has given rise to a number of mineralized occurrences. This mineralization, which has traditionally been mined by local artisanal miners, is now the focus of more modern exploration methods. Within the La Baja structural domain, gold and silver mineralization is associated with flexures along the main fault.

Exploration carried out by the Issuer during the second quarter of 2009 and in early 2010 identified vein and stockwork mineralization associated with strong alteration hosted in dacite porphyry. Rock samples from mine tunnels on site returned gold assays ranging from no significant gold up to 9.66 grams per tonne gold and silver assays ranging from no significant silver up to 94.3 grams per tonne silver. The delineation of the mineralized structures was initiated in 2010 with a drilling program of 6,651 metres in 17 drill holes. From January to March 2011, 511.5 m were completed as part of the recognition of the mineralized structures to depth.

10.1 Grids and Surveys

The coordinate system used for the Project is based upon the Universal Transverse Mercator (UTM) projection (datum Bogota – Zone: 18N).

Topographic data used to delimit the Mineral Resources was provided by Greystar, and has a resolution of ± 5 m within the areas of the orebody. A regional topographic restitution was carried out in 2008 by the Colombian company Aeroestudios Ltda. from Medellin, who took the aerial photographies and developed it's processing for 16,000 hectares in the surrounding area of the project covering the mine infrastructure foreseen in the open pit project of the prefeasibility study. Detailed topography was carried out from 2008 to 2010 by Estudio-T Rural from Bucaramanga, for the individual areas of the mine infrastructure, using total stations for surveying.

10.2 Geological and Structural Mapping

Geological mapping has been performed by the geological staff of Greystar since 1995, at map scales that varied from 1:25000 for the surrounding areas of the project and detailed mapping up to 1:5000 on the deposit area and infrastructure area.

Results of the geological mapping of lithology, structure, hydrothermal alteration and mineralization supported the geological interpretations for the project used in Mineral Resource estimation, and provided vectors for channel sampling and drill targeting. Many old working exploited by the miners of the region have been used as an important input for the geological modelling of the ore body, including detailed mapping (scale 1:1000) and sampling. The exploration tunnels constructed by Greystar since 1997, Fuego Verde, PErezosa II and Veta de Barro tunnels have been in detail mapped for better understanding and construction of the geological model.

10.3 Geochemistry

As part of exploration evaluations, soil and stream geochemical samples that were collected to March 2011 for a combined total of 12,742 m sampled and 4,307 corresponding samples for Angostura project and 1,106 samples in different areas around the project. Mello and Felder (2010) report that:

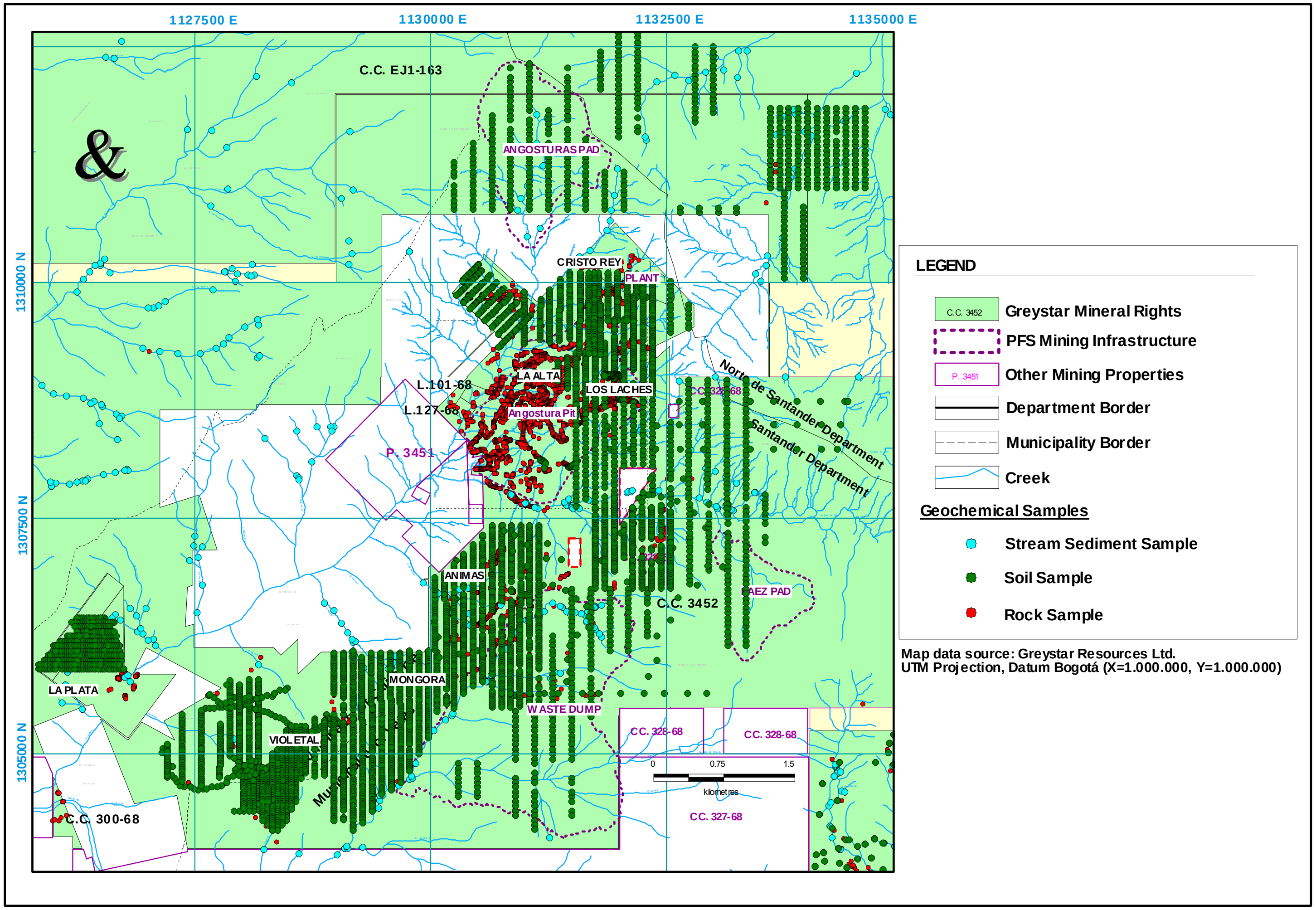
> *"Geochemical soil sampling campaigns have been undertaken in the project and some surrounding areas, and more than 4,000 samples have been taken on a grid with an initial spacing at 100 or 200 metres with later in-fill sampling. Samples were taken at an average depth of 0.8 m."*

This work identified gold anomalies at Cristo Rey, La Alta Este, Los Laches, Animas, Mongora,Violetal and La Plata within the Project area (refer to Figure 7.2-2 for prospect and anomaly locations).

Channel sampling from tunnels was performed on areas of single-lithology outcrop. To date 2010, 3,316 channel samples have been taken.

5,625 soil samples and 769 stream sediment samples have been collected in all of the mining titles owned by Greystar. The map of Figure 10.3-1 shows the rock (surface), soil and stream sediments samples collected in the area of Angostura project.

Figure 10.3-1: Geochemical Sample Location Plan



10.4 Underground Workings

Underground chip sampling was undertaken of adits and tunnels that had been excavated by artisanal miners, and to December 2010, a total of 899 samples were taken.

Greystar has developed four drifts for sampling purposes, totalling 3,145 m. These are the Perezosa 1 (32 m), Perezosa 2 (2,500 m), Veta de Barro (415 m) and Fuego Verde (198 m) drifts. Locations of these Greystar workings were included in Figure 7.2-3.

The Fuego Verde adit was excavated in 1997. The construction of the Perezosa 1 tunnel was started in February 2004 but stability problems due to its location on the Perezosa fault forced Greystar to start a new tunnel. The Perezosa 2 Tunnel, located 100 m to the southeast, was started in early April 2004 at the 2,850 m elevation and consists of two, parallel, east–west drives some 350 m apart, with a connecting cross-cut in the Perezosa area. These excavations served as a base for detailed underground core drilling and have provided access to some of the known mineralized structures for additional underground drifting and sampling. A fourth adit 415.3 m long was completed in 2008, in the Veta de Barro area at an elevation of 3,100 m.

To December 2008, when the tunnels advancing were stopped, 1,768 (muck) samples had been taken from the tunnels.

10.5 Drilling

Drilling completed on the Project is discussed in Section 11.

10.6 Bulk Density

Bulk density determinations are discussed in Section 12.

10.7 Petrology, Mineralogy and Other Research Studies

Petrography studies were performed in the late 1990s to elucidate details of the alteration petro genesis (Harris, 1998). The Table 10.7-1 presents a summary of the research studies developed for Angostura.

Table 10.7-1: Resarch Studies for Angostura

YEAR	STUDY	COMPANY	SAMPLES
1998	Petrography	Vancouver Petrographics	30
1999	Petrographic and Short-Wave Infared	PetraScience Consultants Inc.	3
2000	Fluid Inclusions Study	Micrometría y Assesoria Geologica-	3
2004	Petrography	PetraScience Consultants Inc.	5
2004	Petrography	Universidad Industrial de Santander	3
2005	Petrography	PetraScience Consultants Inc.	6
2005	SEM Analysis	PetraScience Consultants Inc.	6
2005	Petrography	PetraScience Consultants Inc.	10
2005	Petrography	PetraScience Consultants Inc.	3
2005	Structural Geology and Tectonics	iC Consulenten	-
2006	SEM Analysis	PetraScience Consultants Inc.	3
2006	Mineralogical Association Study	Lissete A Diaz & Margareth Gerrero A.	-
2007	Petrography	Universidad Industrial de Santander	2
2007	Petrography	Universidad Industrial de Santander	4
2007	Pyma Analysis of drill core	Anglogold Ashanti	-
2008	Petrography	Universidad Industrial de Santander	7
2008	LA-ICP-MS Laser-Ablation-Coupled	Rhodes University	9

X-ray fluorescence (XRF) studies, using a Niton XLt3 hand-sampler were performed on soil samples in areas that required condemnation evaluation as they were proposed infrastructure sites. Terraspec Mineral Spectrometer is being used for hydrothermal alteration minerals characterization and for alteration mapping in all of the areas in exploration.

10.8 Exploration Potential

Significant additional exploration potential exists in the Project area.

10.8.1 Angostura Deposit

Ongoing drill program in Angostura is focused on targeting extensions and define their continuity in direction and depth of high grade veins that exist within the Angostura deposit. In addition, the drill program will probe the unexplored potential for the mineralization underlying Angostura, where the mineralization continues to depth, and the depth limit has not been defined.

The drilling activities will improve the category of the inferred resources defined to support a pre-feasibility and a feasibility study of a underground exploitation project.

10.8.2 Regional Exploration

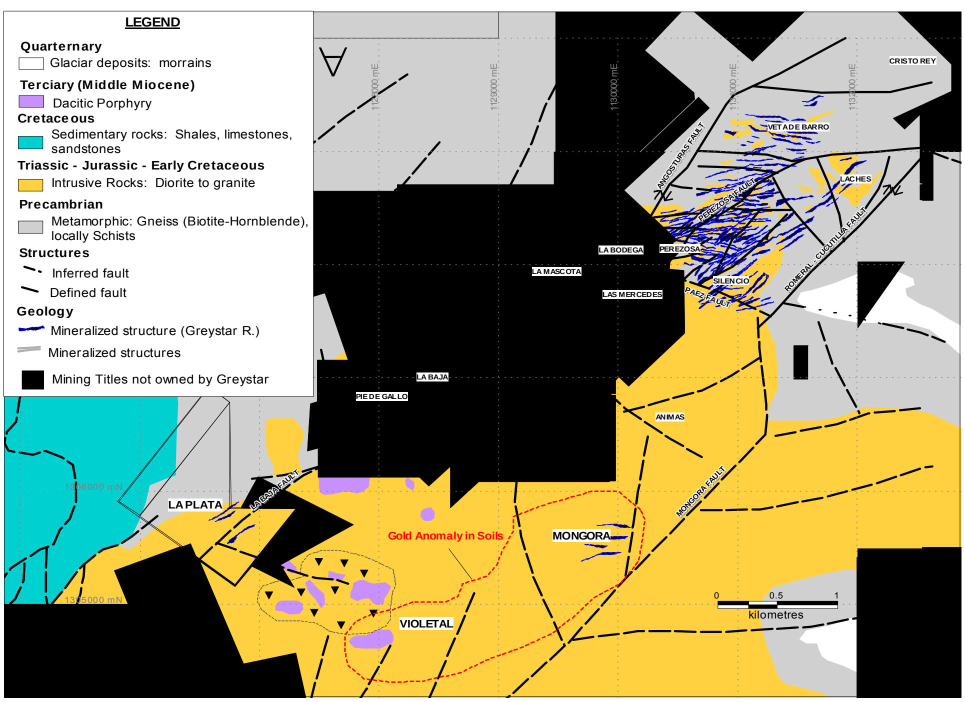
Three major regional targets have been identified, Móngora, Violetal, and La Plata, to the south of the Angostura deposit (Figure 10.8-1).

Móngora

The Móngora prospect is located 3 km southwest of the planned Angostura pit and has many similarities with the geological environment of Angostura. Structurally-controlled mineralization is hosted in Triassic–Jurassic intrusive rocks in association with pyrite. The intensity of hydrothermal alteration does not appear to be as strong as in Angostura, but sericite, illite and chlorite are common.

The Móngora–Animas trend is defined by a series of geochemical anomalies that form a semi-continuous pattern starting 1 km south of Angostura and extending for over 3.5 km in a southerly direction on the west side of the Móngora fault. The actual Móngora prospect is defined by a large, 500 m x 300 m gold-in-soil anomaly. Core drilling to date consists of 58 drill holes, 20,276 m (March 2011), the majority of which have intercepted anomalous gold grades. Greystar is continuing to evaluate the prospect, as it may potentially provide additional resources.

Figure 10.8-1: Location Plan, Regional Exploration Targets



Violetal

Soil sampling extended the gold-in-soil anomaly south of Móngora to the Violetal area, where hydrothermal alteration associated with porphyritic outcrops has been recognized. Elevated copper and molybdenum values were also returned from the soil sampling. In 2008, 2,819 metres in six drill holes were carried out and returned anomalous grades of cooper, gold and silver. Additional work is required to define the extents of the mineralization.

La Plata

The gold, silver, copper mineralization discovered on the La Plata property is part of the mineralizing system following the northeast faulting trend, parallel to La Baja creek. This mineralization gives rise to the Angostura and La Mascota deposits some 4 kilometres to the northeast. Mineralization is structurally controlled and is hosted in Triassic - Jurassic intrusives and in the Precambrian gneisses. The mineralization is associated with hydrothermal fluids possibly generated by small irregular porphyry bodies of 8.4 ± 0.2 Ma and 9.0 ± 0.2 Ma (U-Pb) (Mantilla et al., 2009) age. At La Plata this mineralization occurs in sheeted faulted veins striking to northeast and east-west, steeply dipping to north and south. The veins are commonly silicified and allunitized with halos of Illite-Sericite and Kaolin and smectite alterations. At surface, the mineralized structures have returned grab sample values as high as 9.3 g/t gold, 2,030 g/t silver, 2% copper, 736 parts per million ("ppm") molybdenum, 0.4% lead and 1% zinc

Drilling, comprising 18 drill holes, 7,162 m (March 2011), has intersected anomalous gold and silver grades, and additional work is in process to define the geometry of the mineralization.

Limestone

Cretaceous rocks that crop out to the west of California town and in the municipalities of Suratá, Matanza, Charta and Tona include limestones that could have potential for exploitation and production of lime or limestone for metallurgical uses in an Angostura exploitation project.

10.9 Comment on Section 10

In the opinion of the QPs, the exploration programs completed to date are appropriate to the style of the deposits and prospects within the Project.

11.0 DRILLING

Drilling completed between 1995 and 2011 comprises 1,158 drill holes (365,459 m). Drilling is summarized in Table 11-1, and drill hole locations shown in Figure 11-1. Those numbers include the geotechnical, hydrogeological and condemnation drilling in areas of infrastructure defined in the prefeasibility study.

Table 11-1: Drill Summary Table to March 2011

Activity	Area	# Drill holes	Metres
Exploration	Angostura	992	326,894
Exploration	Animas	7	2,630
Exploration	Mongora	58	20,276
Exploration	Violetal	6	2,819
Exploration	La Plata	19	7,162
Geotechnical - Condemnation	Feasibility studies	76	5,678
Totals		**1,158**	**365,459**

11.1 Drill Contractors and Methods

All drilling to date has been by core methods. Drill contractors used on the Project are summarized in Table 11.1-1. A variety of drill rigs were utilized. Drill contractors also administered three hydraulic drill rigs owned by Greystar, one Hagby 1,000, one Hagby 1,500 and a Atlas Diamec 180. The contractor have used mainly Longyear drill rigs (38 and 44).

Core size varied from BQ (36.5 mm diameter) to NQ (47.6 mm) to HQ (63.5 mm). By far the majority of the core size has been NQ (77% of the total), with HQ and BQ core making up 20% and 3% of the total, respectively. Since November 2007, 2,981 m of PQ-size (85 mm) core has been drilled to collect material for metallurgical testwork, mainly from within the oxide zone.

Table 11.1-1: Drill Contractors

Contractors	Year on Project
Norbert Reinhart	1995
Terramundo Drilling	1996 to 1998
Major Drilling Inc	1999
Geominas S.A.	2003–2011
Perfotec Ltda	2003–2011

11.2 Core Logging

From 1997, all drill core has been photographed, with film records from 1997–1999, and digital records from June 2003 to date.

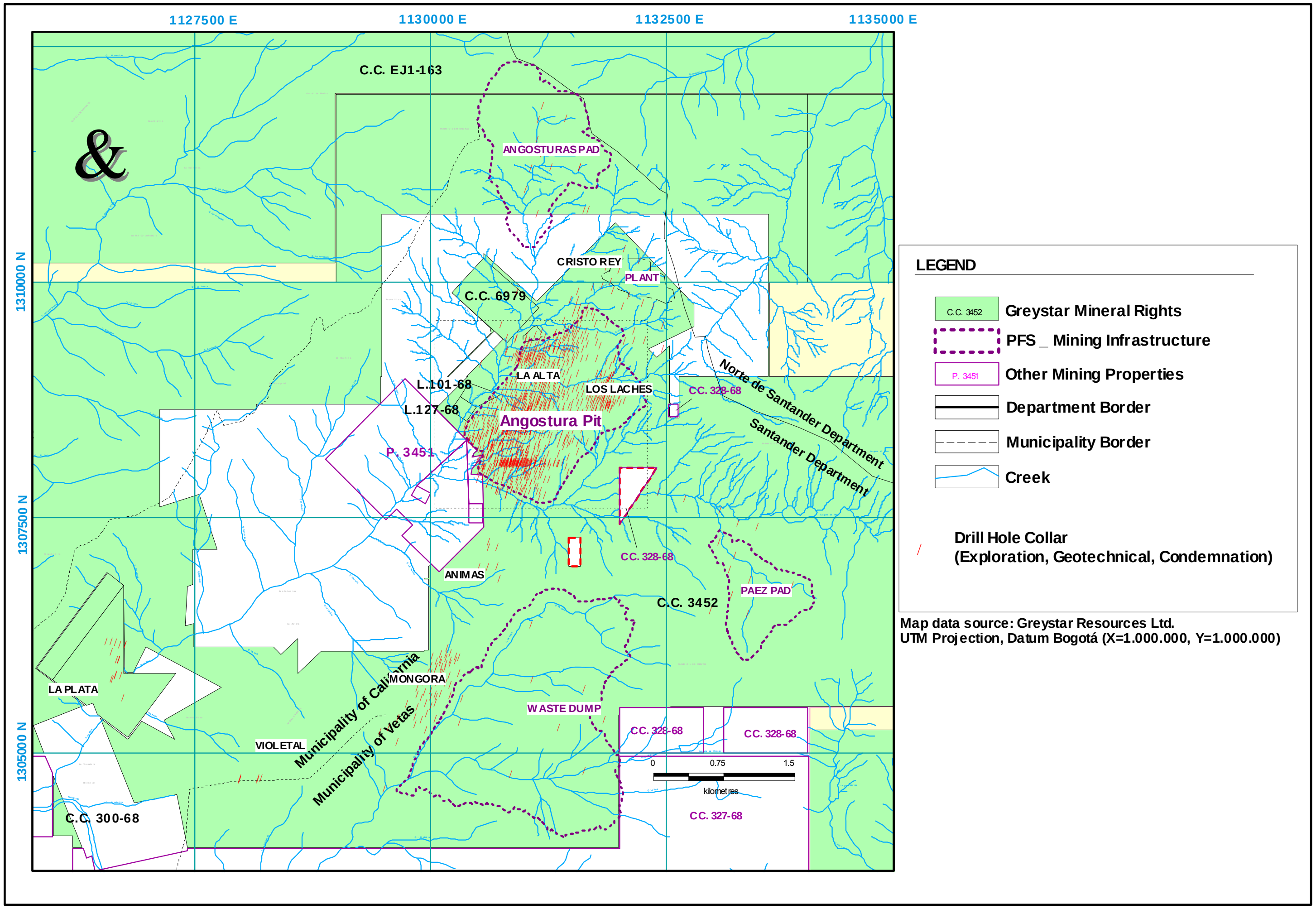
After photography, the Greystar geologists log the core in detail. Data recorded include the major and minor lithologies, mineralization style, intensity, and key minerals, the type and intensity of alteration, rock colour, grain size, structural information such as brecciation and faulting, rock quality designation (RQD) since November 1997, and the degree of oxidation and weathering. The data were initially entered into paper log sheets and later into a computerized relational database. As a result of unsatisfactory logging procedures in the earlier years, a major program of re-logging to improved standards set by Greystar was undertaken in early 1999.

From October 2009 a new oxidation level classification was introduced such that all core is classified as either oxidized, transitional or sulfide (or fresh rock).

11.3 Collar Surveys

Drill hole collars in the field are clearly marked by wooden stakes bearing the information of hole number, azimuth, inclination, and coordinates. Drill collar locations have been verified by survey, and Greystar contracted a professional surveyor to perform the survey readings using total station equipment.

Figure 11-1: Drill Hole Location Plan



11.4 Down-hole Surveys

Initial drill holes, until 1997, were measured using a Tropari instrument. From 1997 to 2003, a Sperry Sun instrument was used. The original deviation measurement pattern was at 50 m intervals, starting at a depth of 150 m down-hole.

Down hole surveys were completed at surface and then at 25 m intervals until the end of the hole using a Reflex EZ-Shot instrument. Sometimes, where there are problems of stability, the measurements were taken at different interval spacings.

In Greystar's experience, the 25 m readings indicate a systematic steepening of the holes by about 1.5° in the first 100 m. Azimuth deviations are of a similar order of magnitude, but can be in either direction, right or left. The degree of steepening and change of azimuth at greater depths is somewhat smaller, reducing to 1° for the dip and 0.3° for the azimuth per 100 m. These changes in drill hole attitude are small and reasonably predictable, so that, despite the absence of deviation determinations above a depth of 150 m in the earlier drill holes, the actual location of these holes will not be far from where they are plotted.

11.5 Recovery

The average core recovery for the entire drill-hole database is approximately 93%. About 80% of the intervals are above a 90% recovery, a figure which includes near-surface recoveries that are typically very poor to a depth of about 5–10 m. Core recovery below a depth of approximately 20 m increases to an average 95%.

11.6 Drilling Used to Support Mineral Resource Estimation

Although up to March 2011, 992 drill holes have been carried out in Angostura, for the geological model and resource estimation in this report, the data used was that available to July 2010, when the veins wireframes were updated. A total of 936 drill holes were used to prepare the geological model and the resource estimation. The average sample length of the core drill holes is approximately 1.68 m, the longest being 17.5 m long.

Drill holes were generally orientated perpendicular to the mineralization. Dips vary depending on the target and range from -76º to 57º. Average drill spacing in the core of the deposit is approximately 50 m; but in areas of high-grade mineralization drill holes can be at 25 m spacing and in some small areas at 12.5 m. This widens to 150 m drill spacing on the deposit edges.

Example drill intercepts for the mineralization at Angostura, sourced from various drilling programs are summarized in Table 11.6-1, and are illustrative of nature of the mineralization. The example drill holes contain oxide, transition, and sulfide intersections, including high-grade veins and disseminated mineralization intervals, and are sourced from drilling programs conducted between 2003 and 2010.

Table 11.6-1: Drill Intercept Summary Table

Drill Hole ID	Intercept Depth From (m)	Intercept Depth To (m)	Drilled Width (m)	Au Grade (g/t Au)	Ag Grade (g/t Au)
DM03-01	278.0	284.0	6.0	2.45	1.4
	338.7	340.2	1.5	10.54	42.2
NA04-01	84.0	98.0	14.0	1.19	9.7
	191.8	212.0	20.2	1.09	9.7
	241.0	253.5	12.5	1.26	7.7
AL05-01	56.0	79.0	23.0	1.27	2.43
	95.4	134.0	38.6	2.07	3.0
	209.4	220.5	11.1	1.06	5.74
	285.9	303.05	17.15	1.01	5.15
	355.0	366.0	11.0	1.57	6.73
SI06-02	20.9	43.7	22.8	1.71	7.9
	66.0	98.0	32.0	1.72	28.6
	110.0	121.0	11.0	0.53	5.6
	237.0	357.0	30.0	0.66	2.1
	362.25	366.0	3.75	9.39	74.0
	374.2	385.0	10.8	1.41	2.8
AL07-15	189.8	196.6	6.8	12.09	66.6
AL08-01	71.85	84.6	12.75	0.51	5.3
	151.0	168.15	17.15	0.59	2.2
	33.0	42.45	9.45	3.09	12.7
ALE07-39	275.0	294.55	19.55	0.51	8.0
	344.0	358.0	14.0	1.48	18.9
DM08-01	94.0	106.0	12.0	1.01	5.0
	143.05	180.1	37.05	0.71	0.8
USI07-08	245.35	246.85	7.5	13.65	136.0
	36.0	48.0	12.0	1.19	9.7
QPO09-06	201.75	231.4	29.65	0.63	2.7
	390	421.65	31.65	1.30	21.2
	447	457.55	10.55	1.27	7.7
LL10-01	84	86.5	2.5	4.56	0.1
	129	137.2	8.2	1.85	22.8
QPO10-02	530.5	537	6.5	11.79	263.7
	537	552	15	1.21	6.2

11.7 Comment on Section 11

In the opinion of the QPs, the quantity and quality of the lithological, geotechnical, collar and downhole survey data collected in the exploration and infill drill programs completed by Greystar are sufficient to support Mineral Resource estimation as follows:

- Core logging undertaken by Greystar meets industry standards for gold and silver exploration within a high-sulfidation epithermal-style setting
- Collar surveys have been performed for the Greystar programs using industry-standard instrumentation
- Downhole surveys were performed using industry-standard instrumentation
- Geotechnical logging of drill core meets industry standards.
- Drilling is normally inclined. Depending on the dip of the drill hole, and the dip of the mineralization, drill intercept widths are typically greater than true widths
- Drill orientations are generally appropriate for the mineralization style, and have been drilled at orientations that are optimal for the orientation of mineralization for the bulk of the deposit area
- Drill orientations are shown in the example cross-section presented as Figure 7.2-4, and can be seen to appropriately test the mineralization.
- Drill hole intercepts as summarized in Table 11.6-1 appropriately reflect the nature of the gold and silver mineralization
- No material factors were identified by with the data collection from the drill programs that could affect Mineral Resource estimation.

12.0 SAMPLING METHOD AND APPROACH

12.1 Surface Sampling

Surface and trench sampling was conducted by channel sampling where the lithology did not change in an outcrop. Discernable vein structures were sampled by panel sampling, with individual panels measuring one to four square metres. Chip samples were collected over pre-defined sections of outcrop showing no discernable difference in lithology or alteration. The sample locations were determined by tape and compass, tying into surveyed drill hole collars. The Table 10-1 contains the information of the surface rock sampling executed by periods.

In 2004, many of the channel samples were still easily recognized. However, following the recommendations of Strathcona, from 2005 rock re-sampling using an electrical saw was implemented to give a more representative sampling of the veins themselves. The surface channel samples are used during resource estimation for geometrical purposes (Veins modeling), but not for the estimate.

12.2 Adit Sampling

The Perezosa 2 (2,850 level, 2,501 m length) and Veta de Barro (3,100 level, 415 m length) tunnels were sampled as follows:

- Continuous chip sampling was originally completed along the walls of the drifts and cross-cuts, but is incomplete. Individual fractures with obvious mineralization were sampled separately to pinpoint the location of the gold.
- The broken material of the majority of the rounds on the Perezosa 2 tunnel and on the Veta de Barro tunnel was systematically sampled by shovel from each mine car, creating 1,768 samples of typically 150–160 kg per round, a sample ratio of nearly 1%. A round was typically 2.3 m x 2.3 m x 1.3 m in dimension.
- A program of systematic channel sampling using an electrical saw was performed along both walls of the 2,850 drifts and cross-cuts. Analysis of the comparative data (nearly 350 m of drift length) for channel and muck sampling showed the muck samples to be systematically higher for gold as well as other elements such as Ag, S, and As, on average by 10–20%, indicating a systematic sampling bias between the two types of samples.

Strathcona investigated this potential bias (Hendricksen, 2007) and concluded that:

> *"as the sample size increased, so did the gold grade. This is may be due to the fact that as the volume of material increases so does the actual degree of systematic sampling of the microfractures containing the*

mineralization. Thus the smaller samples including drill core tend to under estimate grade."

The results of the samples taken from the adits were used for geometrical modeling of the veins but not for the resource estimate of the high grade veins, due to the differences in support.

12.3 Core Samples

Mineralized sections of core are sampled as follows:

- Silicified rocks and zones of sulfides: 0.5 – 1.0 m
- Altered porphyry with between 1% and 10% alteration/sulfides: 1.0 – 2.0 m
- Porphyry and gneiss with minor alteration and sulfides 2.0 m
- Mafic gneiss and dyke without alteration: 2.0 - 3.0 m

Sample intervals are marked on the core boxes and a paper ticket is placed with the core. The portion of core to be assayed is placed in a plastic sample bag with a sample ticket. Before 2004, the plastic bags containing individual samples were combined into larger heavy plastic bags, and three of these in turn are packed into plastic fiber bags for transport. In 2004 a preparation laboratory was constructed in the Angostura's main exploration camp and each sample is prepared as is described in the section 13.2.

The average sampled core length was 1.3 m in the 1990s drilling, and has increased to nearer 1.7 m since 2003. In general, longer samples were taken in areas believed to be of below economic cut-off grade. Few samples are less than 0.5 m long. Sampling observes obvious lithological, alteration, and mineralization breaks.

12.4 Density/Specific Gravity

Greystar have undertaken 9,700 density measurements to March 2011 on drill core samples selected according the lithology, alteration and mineralization, using a wax immersion (ASTM C914-98) methodology. Section 17.10 describes the use of the specific gravity measurements to the block model.

12.5 Comment on Section 12

A description of the drilling programs, including sampling and recovery factors, are included in Section 11 and Section 12. All collection, splitting, and bagging of core samples were carried out by Greystar personnel from 1994 to 2011. No material factors were identified with the drilling programs that could affect Mineral Resource estimation.

Data validation of the drilling and sampling program is discussed in Section 14, and includes review of database audit results.

In the opinion of the QPs, the sampling methods are acceptable, meet industry-standard practice, and are adequate for Mineral Resource and mine planning purposes, based on the following:

- Data are collected following Project-approved sampling protocols
- Sample collection and handling of core was undertaken in accordance with industry-standard practices, with procedures to limit potential sample losses and sampling biases
- Sample intervals, which have been defined on the basis of lithology, alteration, and sulfide content, are considered to be adequately representative of the true thicknesses of mineralization
- Specific gravity determination procedures completed are consistent with industry-standard procedures.

13.0 SAMPLE PREPARATION, ANALYSES, AND SECURITY

From Project acquisition to date, Project staff employed by Greystar was responsible for the following:

- Sample collection
- Core splitting
- Density determinations
- Sample preparation from March 2004 to date
- Sample storage
- Sample security.

13.1 Analytical Laboratories

To March, 2004, Rossbacher Laboratories Ltd. (Rossbacher) of Vancouver was the primary analytical laboratory. The laboratory accreditations are not known for this period; however, the laboratory was independent of Greystar.

Bondar Clegg Laboratories in Vancouver, now ALS Chemex Laboratories (ALS Chemex), performed check assays on pulp duplicate materials during the period 1996 to 1999. The laboratory accreditations are not known for this period. The laboratory was independent of Greystar.

ALS Chemex in Vancouver performed sample preparation during early 2004, was the umpire laboratory until 2007, and has performed assays on samples since 2007. ALS Chemex also re-assays all samples that return a gold grade of >0.4 g/t Au. The laboratory is independent of Greystar, and holds both ISO:9001:2000 and ISO17025 accreditation.

ACME Analytical Laboratories Ltd. in Vancouver (ACME) was the analytical laboratory that performed assays on low-grade samples between 2004 and 2007; the laboratory was also the umpire laboratory for the period between late 2007 and March 2010. The laboratory is independent of Greystar, and holds ISO:9001:2000 accreditation.

Assayers Canada Limited performed umpire assays during the period April 2004 to March 2007. The laboratory is independent of Greystar and holds ISO:9001:2000 accreditation.

13.2 Sample Preparation

Sample preparation has been performed by independent laboratories, and by an on-site sample preparation facility staffed by Greystar personnel.

Rossbacher, from 1995 to 1999, prepared all core, geochemical and tunnel samples. The preparation method comprised drying the samples at 50 to 60° C and then crushing to minus 10 mesh (1.7 mm). A sub-sample of 250–350 g was obtained from the crushed sample by Jones splitter and pulverized to 90% passing 150 mesh (106 µm) in a ring pulverizer.

From June 2003 to March 2004 (, sample preparation was initially undertaken by a local laboratory in Bucaramanga; however, the laboratory could not keep up with the volume. Samples that were considered to be non-mineralized continued to be prepared by the local laboratory, but all other samples were air-freighted to ALS Chemex. No information is available was available to Greystar on the preparation used during this timeframe.

In March 2004 a preparation laboratory was established on site. The site facility employs one Rhino and one Terminator jaw crusher. A charge of barren limestone or granite is passed between samples. After crushing, the sample, of an original mass of typically 1.5 kg to 3 kg, is blended and a sub-sample of nominally 250 g obtained by riffle splitting. Quality control (QC) measures include the weighing and screen analysis of one in 10 samples. Actual crusher output is usually close to 95% passing 1.7 mm. The sample preparation facility capacity is currently around 150 samples per shift, and the facility employs six people.

13.3 Sample Analysis

At Rossbacher, samples were initially assayed for gold by aqua regia digest followed by atomic absorption spectroscopy (AAS). Samples with gold values between 0.5 g/t Au and 1.5 g/t Au were reanalyzed, until 1996, using a one assay-ton aliquot and fire assay (FA). The remainders of the pulps of those samples yielding above 1.5 g/t Au from the initial geochemical method were re-assayed using a pulp-and-metallics method, screening at 150 mesh.

Silver and copper were originally determined by Rossbacher, using AAS, based on a 0.5 g aliquot, with an aqua regia digestion. Silver values equal to or greater than 15 g/t Ag were later re-assayed using Fire Assay.

A large number of samples from early drilling at Angostura (1996 to 1998) were also assayed by Rossbacher using a cyanide-leach method, for the purpose of comparing total gold (as determined by FA or metallics assay) to cyanide-soluble gold. A 30 g sample was shaken for three hours in 60 mL of a 0.5% NaCN solution, and the dissolved gold was determined by AAS.

ACME performed assays using a 15 g aliquot and a 30 element geochemical inductively-coupled plasma (ICP) gold method after aqua regia digestion.

ALS Chemex analyses are by fire assay with an atomic absorption spectrometer (FA/AAS) finish using a one assay-ton (29.2 g) aliquot (Code Au-AA23). Gold assays above 10 g/t Au and silver assays above 100 g/t Ag are re-assayed by one assay-ton FA with a gravimetric finish (Codes Au-GRA21, Ag-OG62). Separate splits of these samples are subjected to a multi-element ICP assay, including silver and sulfur, following a four-acid digestion. The limit of the reported sulfur assays by this method is 10%. All samples with ICP results that show a sulfur grade of >10% are re-assayed using the Leco method (Code S-IR08) with an upper limit of 50% S.

13.4 Quality Assurance and Quality Control

There was no Greystar-sponsored quality assurance/quality control (QA/QC) program in place for the drilling campaigns from 1995 to 1999. However, a substantial program of check assaying of pulp duplicates was undertaken at Bondar Clegg Laboratories during those years, and in 2003–2004 a number of high-grade core intervals were re-sampled and rejects submitted for check assaying at ALS Chemex. Results of this program are discussed in Section 14.

In June 2003, a QA/QC program external to the assay laboratory was instituted, consisting of submission of blanks and standard reference materials (SRMs).

A total of 12 different SRMs were employed. All of the standards were prepared and certified by CDN Resource Laboratories Ltd. (CDN) of Vancouver, British Columbia. Standards are inserted into the sample stream every 15–20 samples or when a particularly mineralization-prospective intersection is logged. Triggers for an individual standard to have failed were generally set at reference value plus or minus three standard deviations (SD). If two adjacent standards were both more than two SD values above or below the reference value, then both standards were failures as well. The SD values were determined during the certification process. When Greystar receives SRM results outside of an acceptable range, a request is made to the laboratory to re-analyze the affected batch or batches.

From September 2003 to March 2004, the field blanks were core samples from Angostura drill core that had previously been found to be barren; this was subsequently, to June 2006, changed to limestone or barren gneiss material. There were no blanks inserted by Greystar from July 2006 to August 2007, so that a significant number of samples cannot be assessed with respect to any contamination that may have occurred. Burns (2005) noted that blank materials were crushed and bagged in 200 g portions at Greystar's onsite preparation laboratory. Blanks are inserted into the sample stream at the rate of one in 25 to 30 by the Project geologists.

From 2007, Greystar geologists have inserted control samples during core sampling. In October 2010, sampling protocols were changed to ensure that a control sample was inserted with each batch dispatched to the laboratory, one blank, one SRM, one core duplicate and a pulp duplicate per batch (35 samples) to follow the industry standards.

Collection of duplicate samples are triggered by a geologist inserting a "repeat" ticket into the sample stream; this indicates to the preparation laboratory that a second split of the sample is required (Burns, 2005). Duplicate samples have been collected at irregular intervals since early 2004.

Assayers Canada Limited performed secondary assays on pulp duplicate materials from April 2004 (with a three-month interruption at the end of 2005) to March 2007 at the rate of one in 25 to 30 by batch. Evaluation of the data indicated no analytical biases between the primary and umpire laboratories (Smee, 2007).

Acme performed secondary assays on pulp duplicate materials from late 2007 to June 2010. Evaluation of the data indicates a potential low bias at ALS Chemex in the gold range of 50–100 g/t Au (Mello and Felder, 2010).

A total of 30 samples that submitted to the SGS test facility in South Africa for flotation testwork were re-analyzed. These indicated a close correlation between the flotation sample head grades and the original assay data (Mello and Felder, 2010).

13.5 Databases

Greystar has implemented SQL software, produced by the Bucaramanga firm of Systemas Integrados de Informacion y Digitalizacion (SIID), to manage the Angostura database. The system is installed in the Greystar Bucaramanga office and is fully integrated with the data acquisition activities in the field and downloaded to Bucaramanga by satellite phone. A strict, controlled and structured set of fields and columns is used to manage the data flow, and there are checks to alert the database manager of any import issues (Burns, 2005).

Assays are received electronically from the laboratories and imported directly into the database. Drill hole logging, collar and down hole survey data are manually entered into the database. Data are verified on entry to the database by means of in-built program triggers within the SQL database, and further checked on import to the mining estimation software. Checks are performed on surveys, collar co-ordinates, lithology data, and assay data.

Paper records are kept for all assay and QA/QC data, geological logging and bulk density information, downhole, and collar co-ordinate surveys.

13.6 Sample Security

Sample security relied upon the fact that the samples were always attended or locked at the sample dispatch facility. Sample collection and transportation have always been undertaken by company or laboratory personnel using corporately-owned vehicles.

Chain of custody procedures consisted of filling out sample submittal forms that were sent to the laboratory with sample shipments to make certain that all samples were received by the laboratory.

13.7 Sample Storage

Core is stored in boxes that are numbered and ordered, and housed in two dry, clean, and well-maintained permanent facilities near the village of California. Older drill core that was stored originally in Bucaramanga were moved to the facilities in California in late 2008.

13.8 Comment on Section 13

The QPs are of the opinion that the quality of the gold analytical data collected during the Greystar drill programs are sufficiently reliable (also see discussion in Section 14) to support Mineral Resource estimation and that sample preparation, analysis, and security are generally performed in accordance with exploration best practices and industry standards as follows:

- Drill sampling has been adequately spaced to first define, then infill, gold and silver anomalies to produce prospect-scale and deposit-scale drill data. Drill hole spacing varies with depth. Drill hole spacing increases with depth as the number of holes decrease and holes deviate apart.
- Sample preparation is in line with industry-standard methods for high-sulfidation epithermal deposits. Preparation prior to 2004 was by independent laboratories; since 2004, preparation has been undertaken by Greystar personnel at an on-site preparation laboratory.

- From 2004, Greystar drill programs have included insertion of blank and standard reference material samples. Duplicate submission is irregular. QA/QC submission rates meet industry-accepted standards of insertion rates. The QA/QC program results that have been validated by independent consultants do not indicate any problems with the analytical programs, therefore the gold and silver analyses from the core drilling are suitable for inclusion in Mineral Resource estimation
- Data that were collected were subject to validation, using in-built program triggers that automatically checked data on upload to the database. This includes checks on surveys, collar co-ordinates, lithology data, and assay data. The checks are appropriate, and consistent with industry standards
- Sample security has relied upon the fact that the samples were always attended or locked in the on-site sample preparation facility. Chain-of-custody procedures consist of filling out sample submittal forms that are sent to the laboratory with sample shipments to make certain that all samples are received by the laboratory
- Current sample storage procedures and storage areas are consistent with industry standards.

14.0 DATA VERIFICATION

A number of data verification programs and audits have been performed over the Project history, primarily in support of compilation of technical reports on the Project.

14.1 Mine Development Associates, 1998

As part of the 1998 mineral resource estimate, Mines Development Associates (MDA) completed a database audit on data collected at the end of 1998. Thalenhorst (2002) reports that MDA found the database to be “generally satisfactory”. Thalenhorst and Barton (2003) also noted, in relation to the MDA audit results:

> *“This audit detected a number of clerical errors and shortcomings, and addressed remnants of a problem that had been recognized earlier, namely the duplication of a number of sample numbers by Rossbacher. The problem was subsequently resolved, but a relatively small number of assay results could not be assigned to their original drill hole intervals which therefore are labelled as not having been assayed.”*

14.2 Strathcona Mineral Services Limited, 2002, 2003

The analytical data undertaken by Rossbacher on original and coarse reject samples were compared with analytical results from pulp duplicate assays performed by Bondar Clegg Laboratories. Thalenhorst and Barton (2003) concluded that:

> *“There is a tendency of the Rossbacher results to be high by around 10% as compared to Bondar Clegg. An analysis of the data shows that this difference is mainly caused by data from the assay intervals >20 g/t where there is a fairly large difference between the two labs.”*

A systematic QA/QC program was recommended for implementation as was a systematic density determination program.

14.3 Strathcona Mineral Services Limited, 2004

In 2003–2004, a number of high-grade core intervals were re-sampled and rejects submitted for check assaying at ALS Chemex. Thalenhorst (2004) examined the data, finding that:

> *“There is a tendency of the Rossbacher results for gold to be high by around 10% as compared to Bondar Clegg and Chemex. In contrast, the Rossbacher silver results may be slightly low, but how much of this is due*

to the different digestion methods used is currently unclear. Most of the difference is in the assay population >10 g/t gold."

Greystar reviewed this finding, and showed that the differences between the Rossbacher and Bondar Clegg results were restricted to assays above about 2 g/t Au and appeared to be traceable to certain groups of contiguous samples, where the Rossbacher results were consistently and obviously high (the "bad" set, corresponding to 891 samples), while other batches did not show this bias. A total of 520 samples with a Rossbacher mean gold value of 6.6 g/t Au were re-assayed at ALS Chemex who reported a mean of 6.2 g/t Au. Kinross, in 1999, estimated the mineral resources with, and without, the Rossbacher data, and found little impact.

The overall conclusions were that the data could support mineral resource estimation.

Thalenhorst (2004) recommended that Greystar replace those Rossbacher assays for which Bondar Clegg check assays were available and re-assay those other samples assayed by Rossbacher that had a direct impact on the mineral resource estimated gold grade.

14.4 Snowden, 2005

Burns (2005) reviewed Greystar's database, the geological interpretation, the collection of drill hole data, surface and underground showings, the preparation laboratory, the core logging facility and the core sheds in California and Giron. Seven drill holes were inspected, and in all instances the lithologies, mineralization, alteration and sample intervals were found to agree with the drill logs. Assay checks between the primary database and the compiled Datamine database were undertaken, and selected analytical data were compared between ALS Chemex's posted website results and the database values. No discrepancies were found. Burns (2005) concluded that the data were acceptable to inform mineral resource estimation.

14.5 Strathcona Mineral Services Limited, 2006

QA/QC data from blank, SRM and duplicate samples were reviewed and no significant biases or analytical errors noted. Thalenhorst (2006) was of the opinion that:

"The full QA/QC system as practised since 2003 conforms to industry standards, with the available check assay and standards assay data indicating the assay results for the years 2003 to 2006 as reliable, and that any contamination is a short-lived problem. Because of the generally fine-grained nature of the gold at Angostura, assay precision is

> *uncommonly good, giving a high degree of confidence in individual assays."*

Thalenhorst (2006) noted that the Rossbacher data, by 2006, constituted <1% of the total analytical database, and was therefore no longer of concern. The specific gravity determinations available were acceptable to support mineral resource estimation.

14.6 Hatch Limited, 2007

No independent data verification was performed by Wells et al (2007), instead, the conclusions of Thalenhorst (2006) were considered acceptable for the preliminary assessment completed.

14.7 Metálica Consultores S.A., 2009

As part of database verification for mineral resource estimation, Sironvalle (2009) concluded:

"While the QA/QC system as practiced since 2003 largely conforms to industry standards, it could have been somewhat more systematic and regular. The available check assay and standards assay data indicate that the assay results for the years 2003 to 2008 collectively are reliable, that they are fairly precise individually, and that any contamination was a short-lived problem.

Bulk density data added since December 2007 confirmed the analyses made in 2007."

14.8 GRD Minproc Limited, 2009

Greig et al (2009) did not perform any independent verification, noting:

"From the checks made by Greystar, previous consultants and Metalica, it is concluded that the data has been verified to a sufficient level to permit its use in a 43-101 compliant resource estimate."

14.9 Smee Consultants, 2006–2010

Barry Smee inspected the preparation laboratory on three occasions, and prepared reports on the QA/QC programs in 2006, 2007, and 2008. Smee (2007) recommended that two high-grade standards be acquired; these were incorporated into the QA/QC program for the 2008 drill program.

Smee (2010) reviewed the QA/QC program conducted between September 2008 and September 2010. The program was conducted as part of exploration drilling at Angostura and Móngora. Smee recommended the following:

- Increase the frequency of control samples to 4 control sample (standard, blank, preparation duplicate and core duplicate) per batch (35 samples).
- Include a Standard Au > 10 g/t
- Prepare a custom standard (1.2 - 4 g/t Au).
- Prepare and analyze 200 stored rejects to act as preparation duplicates for estimating sample preparation representativity.

14.10 NCL, 2010

NCL conducted a review of the database quality, concluding that it was robust and well managed, and noted that security measures precluded data tampering.

Four drill holes, representing approximately 1,000 assays, were randomly selected, and checked against the corresponding database entries. No inconsistencies between digital and hardcopy data were identified.

NCL concluded, from these reviews, that:

"While the QA/QC system as practiced since 2003 conforms to industry standards. The available check assay and standards assay data indicate that the assay results for the years 2003 to 2008 collectively are reliable, that they are fairly precise individually, and that any contamination was a short-lived problem.

Considering the results of the verification completed by NCL and the extent of external quality assurance and quality control measures implemented by Greystar as well as external data verification and QA/QC controls conducted by Barry Smee, NCL did not consider that further independent verification sampling was required.

In the opinion of NCL, Greystar used industry best practices to explore for gold and silver on the Angostura project. The exploration data was collected with care and is appropriately managed to ensure the safeguard of exploration information. The resulting exploration data is generally reliable for resource estimation."

14.11 Comment on Section 14

The process of data verification for the Project has primarily been performed by external consultancies. The QPs consider that a reasonable level of verification has been completed, and that no material issues would have been left unidentified from the programs undertaken.

The QPs, who rely upon this work, have reviewed the appropriate reports, and are of the opinion that the data verification programs undertaken on the data collected from the Project adequately support the geological interpretations, the analytical and database quality, and therefore support the use of the data in Mineral Resource estimation, and in the feasibility study:

- No significant sample biases were identified from the QA/QC programs undertaken by Greystar. A small portion of the initial 1996 to 1999 Rossbacher data were identified as being biased high for gold values; however, check analytical data by ALS Chemex are used instead of the original data in estimation, and the remainder of the data are now a very minor component of the total assay database (<1%)
- Sample data collected adequately reflect deposit dimensions, true widths of mineralization, and the style of the deposit
- External reviews of the database were undertaken between 1998 and 2010, producing independent assessments of the database quality. No significant problems with the database, sampling protocols, flowsheets, check analysis program, or data storage were noted,
- Changes and adjustments the QA/QC program suggested by Smee 2010 has been implemented, except the preparation and introduction of a custom standard (1.2 – 4 g/t Au) that is in process of certification to date.
- Drill data are typically verified prior to Mineral Resource estimation, by running a software program check.

15.0 ADJACENT PROPERTIES

There are a number of small-scale mining operations in the area of the Angostura project run by Colombian nationals and cooperatives.

Ventana Gold Corp. ("Ventana") owns 100% of the mineral rights of La Bodega property (Ventana Gold Corp., 2010) which immediately adjoins the Angostura property. Ventana has drilled more than 100,000 metres in more than 275 drill holes defining 4 minieralized zones, La Bodega (Extension of Angostura), La Mascota, Las Mercedes and Aserradero (www.ventanagold.com, 2010). On November 7 2008 Ventana Gold Corp. was listed in the Toronto Stock Exchange (TSX:VEN).

Ventana also holds the California-Vetas property most of which is located well to the south of the Angostura deposit, to the west of the municipality of Vetas. However, two small parcels of concession 328-68 are located immediately to the east of the Angostura deposit.

On March 16 2011 AUX Canada Acquisition Inc ("AUX") completed the acquisition of Ventana Gold Corp. To date, the de-listing of Ventana's shares from the Toronto Stock Exchange is in process.

Galway Resources Ltd. (TSX-V:GWY) and Calvista Gold Corporation have made agreements over the California-Vetas district, including a land position along strike and adjacent to (south west) Ventana's La Mascota area and are executing a drilling program in California. Galway acquired the Reina de Oro gold property in Vetas (www.galwayresources.com, 2010).

16.0 MINERAL PROCESSING AND METALLURGICAL TESTING

16.1 Metallurgical Testwork

In order to design the process circuit, Greystar carried out several testwork campaigns, including mineralogical characterization, flotation, sulfur oxidation and cyanidation of three samples: oxide, transition and sulfide.

The main flotation and cyanidation testwork campaigns were conducted by SGS, METCOM and McCLELLAND Lab INC (the last one also carried out pressure oxidation tests), but the following laboratories also carried out some others metallurgical tests: G&T (mineralogical characterization), Hazen (roasting tests) and BIOMIN Technologies S.A (bio-oxidation tests).

The following are the main results and conclusion obtained from all testwork.

16.1.1 Mineralogical composition

The mineralogical composition of three samples was analyzed by G&T, according to Table 16.1-1:

Table 16.1-1: Mineralogical composition of three Angostura samples

Sample	Composition- percent or g/tonne		
	Gold	Pyrite	Gangue
Sulphide Composite 1	3.10	6.6	93.4
Sulphide Composite 2	1.18	4.6	95.4
Transition Composite 1	1.92	6.3	93.7

The Table 16.1-2 shows the fragmentation of the mineral, information that is used for grinding and regrinding power requirement estimation.

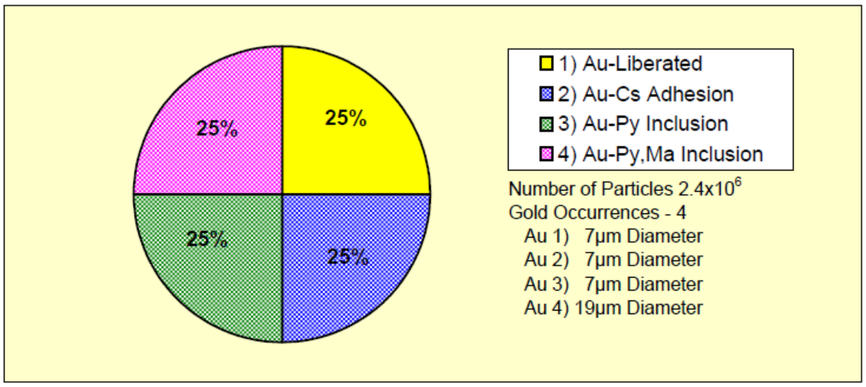
Table 16.1-2: Mineral fragmentation for three mineral composites

Mineral Status	Sulphide 1 (93 µm K_{80})		Sulphide 2 (90 µm K_{80})		Transition 1 (87 µm K_{80})	
	Py	Gn	Py	Gn	Py	Gn
Liberated Grains	60	98	71	98	76	98
Binary with Pyrite	-	2	-	2	-	2
Binary with Gangue	40	-	29	-	24	-
Multiphase Particles	<1	<1	<1	<1	<1	<1

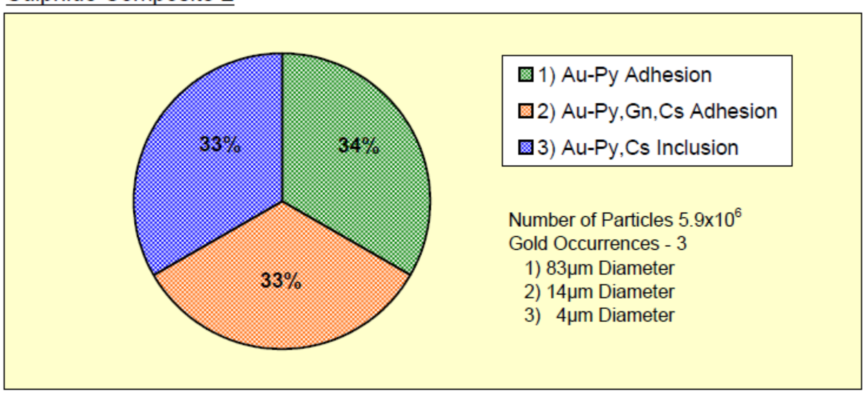
The following figures show the distribution of gold occurrence in concentrates obtained from rougher flotation, carried out with the analyzed three composites:

Figure 16.1-1: Gold distribution in rougher concentrates, for three samples, where Py: pyrite; Cs: calcosine; Gn: gangue

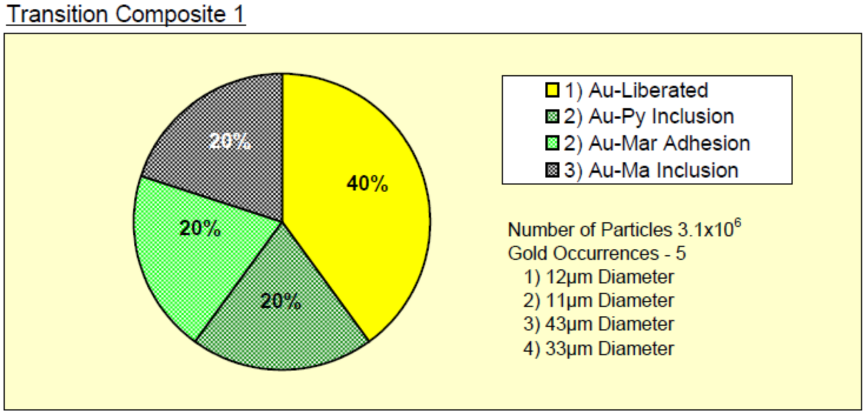
Sulphide Composite 1



Sulphide Composite 2



Transition Composite 1



Regarding the rougher flotation performance, the following results were obtained:

- Sulfide composite 1 has a mass recovery in rougher flotation of 12 – 14%, with a gold recovery of 90%. Rougher flotation doesn't seem to be influenced by feed size, in the range of P80 70 to 150 µm.

- Sulfide composite 2 has a rougher mass recovery of 10%, with 90% of gold recovery. Again, feed size doesn't have an effect in flotation performance, in the range of P80 90 to 150 µm.

- Transition composite 1 has a rougher mass recovery of 10 – 11%, with 90% of gold recovery. Flotation feed size is not a factor influencing gold recovery.

The Table 16.1-3 shows the estimated mineralogical composition of a bulk rougher flotation concentrate. The analysis was carried out by SGS, by means of QUEMSCAN Bulk Modal Analysis (BMA).

Table 16.1-3: Mineralogical composition of bulk flotation concentrate

Mineral	Mass %
Quartz	28.34
Muscovite	31.47
Alunite	3.11
Kaolinite	0.47
Chlorite	0.28
Total Silicate Gangue	*63.68*
Pyrite	34.75
Chalcopyrite/Bornite	0.04
Calcocite/Covellite	0.12
Tetrahedrite	0.17
Sphalerite	0.10
Total Sulphides	*35.18*
Rutile	0.25
Zircon	0.02
Apatite	0.01
Magnetite	0.01
Chromite	0.14
Carbonate	0.06
Other	0.64
All Other	*1.14*
Total	100

Data validation was made by comparing the calculated chemical composition with the chemical measured composition of selected elements, according to Table 16.1-4:

Table 16.1-4: Data validation for mineralogical composition

Element (mass %)	Calculated from mineralogy	Chemical Analysis
Mg	0.05	0.43
Al	7.12	7.41
Si	20.3	19.5
Si	19.2	18.0
K	3.38	3.33
Ca	0.03	0.05
Ti	0.15	0.35
Cr	0.07	0.08
Fe	16.3	15.5
Cu	0.16	0.14

As can be noted in the Table 16.1-4 above, the values obtained are not exactly the same, but the correlation is good enough to validate the BMA results.

16.1.2 Whole ore cyanidation tests

Whole ore cyanidation tests were conducted on oxides, transition and sulfide samples, under two different conditions:

- Bottle roll tests for three particle sizes: P80 ¾ inches, 35 Mesh and 150 Mesh and two leaching times: 72 and 96 hours.
- Open cycle column leach tests for two particles sizes: P80 ¾ inches and 4 inches and for two leaching times: 61 and 144 days, app.

The tables 16.1-5 and 16.1-6 show the results obtained in the direct cyanidation tests. The data corresponds to the average results obtained from all the samples tested in each case.

Table 16.1-5: Bottle Roll tests

CYANIDATION BOTTLE ROLL TESTS							
Ore type	Number of samples	Particle Size	Leaching time hr	Extraction %		Reagent Consumption kg/t	
				Au	Ag	NaCN	CaO
Oxide	7	P_{80} 3/4 Inch	96	82.7	56.3	1.07	1.14
	7	P_{100} 35 Mesh	72	93.4	81.6	0.80	1.77
	7	P_{100} 150 Mesh	72	93.7	88.1	0.92	2.21
Transition	4	P_{80} 3/4 Inch	96	69.9	60.6	1.12	0.77
	5	P_{100} 35 Mesh	72	75.1	74.2	1.31	1.28
	5	P_{100} 150 Mesh	72	83.2	82.3	1.34	1.38
Sulfide	13	P_{80} 3/4 Inch	96	34.5	43.5	2.33	1.19
	13	P_{100} 35 Mesh	72	41.0	60.3	3.28	1.68
	13	P_{100} 150 Mesh	72	49.8	64.8	3.54	2.02

Table 16.1-6: Open cycle column leach tests

OPEN CYCLE COLUMN LEACH TESTS							
Ore type	Number of samples	Crush Size (P_{80})	Leach Cycle days	Extraction %		Reagent Consumption kg/t	
				Au	Ag	NaCN	CaO
Oxide	7	-	61	90.3	59.8	0.49	1.35
Transition	4	4 inch	142	41.5	43.8	1.36	2.32
	4	3/4 inch	61	70.5	61.3	0.54	1.08
Sulfide	5	3/4 inch	60	35.6	27.7	1.06	1.25
	1	3/4 inch	144	44.9	33.2	4.59	0.17

As it can be noted from the tables above, poor results were obtained for the sulfide ore, with Au recoveries about 40% and Ag recoveries about 30%.

On the other hand the results indicate that both transition and oxide ores are amenable to recovery of precious metals by heap or agitation leaching techniques. For transition samples, Au recoveries vary from 70 to 83%, while Ag recoveries vary from 60 to 82%. For oxide sample Au recoveries are in the 90% range, while Ag recoveries are in the 70% range.

16.1.3 Flotation

Several flotation tests were performed with sulfide and transition ores and with mixed of both, and include rougher flotation and locked cycle flotation tests.

The Table 16.1-7 shows the results obtained in locked cycle tests, carried out by two laboratories, MLI and SGS. The data shown correspond to the average results obtained in all samples tested.

Table 16.1-7: Locked cycle tests

LOCKED CYCLE FLOTATION TESTS													
Lab.	Ore Type	Number of samples	Weight %		Au grade g/t		Ag grade g/t		Au %		Ag %	
			Cl. Conc.	Ro Tail	Cl. Conc.	Ro Tail	Cl. Conc.	Ro Tail	Cl. Conc.	Ro Tail	Cl. Conc.	Ro Tail
MLI	Sulfide	4	11.1	88.9	37.1	0.64	103	2.38	87.1	12.9	84.3	15.7
	Transition	2	10.7	89.4	67.8	0.74	166	3.40	90.4	9.6	84.3	15.7
	Mixed	1	10.8	89.2	39.7	0.70	155	2.70	87.3	12.7	87.6	12.4
SGS	Sulfide	9	16.2	83.8	31.6	1.19	279	9.00	83.3	16.6	82.3	17.7
	Transition	1	6.4	93.6	39.2	3.15	544	22.00	45.9	54.1	62.7	62.7

As is can be seen from the table above the mixed ore test, performed by MLI, shows a recovery of 88%, for both Au and Ag, in the cleaner concentrate, which represents a 11% of the rougher flotation feed weight.

Regarding the sulfide ore, an average recovery of 83% gold and 82% silver is obtained in cleaner concentrate for test performed in SGS; values that reach 87% gold and 84% silver when the tests were performed in MLI.

The Table 16.1-8 shows the results obtained from variability flotation tests, in which 6 sulfide and 6 transition ore samples were tested in order to study their behavior in rougher flotation.

Table 16.1-8: Variability flotation tests

ORE VARIABILITY FLOTATION TESTS											
Ore Type	Number of samples	Weight %		Au grade g/t		Au distribution %		Ag grade g/t		Ag distribution %	
		Ro Conc.	Ro Tail	Ro Conc.	Ro Tail	Ro Conc.	Ro Tail	Ro Conc.	Ro Tail	Ro Conc.	Ro Tail
Sulfide	6	14.1	85.9	26.3	0.38	90.7	9.3	149	1.7	85.5	14.0
Transition	6	11.0	89.0	51.8	1.38	73.8	26.3	640	16.3	77.5	22.5

Finally, tests were made with the objective of study the effect of grinding size in both, rougher and cleaner flotation. The Tables 16.1-9 and 16.1-10 shows the results obtained, indicated by mineral type.

Table 16.1-9: Effect of feed size in rougher flotation in sulfide ore

Rougher flotation	Weight		Au recovery		Ag recovery	
	80% 106 µm	80% 75 µm	80% 106 µm	80% 75 µm	80% 106 µm	80% 75 µm
Central	15	13	93	87	75	89
Los Laches-Silencio	12	8	84	74	82	78
Perezosa	15	10	89	76	92	89
Veta Barro	13	15	88	94	85	87
Average	14	12	88	83	84	86

Table 16.1-10: Effect of feed size in cleaner flotation in sulfide ore

Cleaner flotation	Weight		Au recovery		Ag recovery	
	80% 106 µm	80% 75 µm	80% 106 µm	80% 75 µm	80% 106 µm	80% 75 µm
Central	62	49	85	58	79	58
Los Laches-Silencio	57	49	81	69	72	67
Perezosa	64	57	92	86	81	78
Veta Barro	54	45	86	91	83	85
Average	59	50	86	76	79	72

For both, rougher and cleaner flotation, the recovery of mass, Au and Ag decreases as the feed size decreases too, in most of the scenarios.

For both, rougher and cleaner flotation, the recovery of mass, Au and Ag decreases as the feed size decreases too, in most of the scenarios.

The Table 16.1-11 shows the results obtained in rougher and cleaner flotation tests performed with sulfide ores, with and without the addition of a depressant. The tests were carried out in SGS in 2010.

Table 16.1-11: Flotation tests with and without depressant

FLOTATION TESTS							
Ore Type	Number of samples	Weight %		Au %		Ag %	
		Cl	Ro	Cl	Ro	Cl	Ro
Sulfide	21	14.6	28.9	90.8	93.2	89.1	93.9
Sulfide with depressant	20	16.7	24.4	86.9	91.3	88.1	91.3

Rougher recoveries go from 91.3% to 93.2% for Au, and from 91.3% to 93.9 for Ag. Cleaner recoveries go from 86.9% to 90.8% for Au and from 88.1% to 89.1% for Ag.

16.1.4 Flotation concentrate and flotation tails cyanidation

Cyanidation tests were performed to both, flotation concentrate and flotation tails. The Table 16.1-12 shows the results obtained in each case.

Table 16.1-12: Rougher concentrate cyanidation for oxide, transition and sulfide ores

FLOTATION AND CYANIDATION OF ROUGHER CONCENTRATE						
Ore type	Number of samples	Flotation			Ro concentrate cyanidation	
		Weight %	Recovery %		Recovery %	
			Au	Ag	Au	Ag
Oxide	4	61.0	74.9	78.8	91.9	61.0
Transition	8	37.7	79.7	77.8	73.9	72.4
Sulfide	22	31.8	83.2	74.9	41.3	50.3

Table 16.1-13: Tails concentrate cyanidation for sulfide, transition and oxide ores, carried out by two laboratories

FLOTATION AND CYANIDATION OF ROUGHER TAILS								
Lab.	Ore Type	Number of samples	Au recovery %			Ag recovery %		
			Flot.	Tails CN	Combined	Flot.	Tails CN	Combined
MLI	Sulfide	4	87.1	43.1	92.8	84.3	55.2	93.0
	Transition	1	93.1	91.9	99.0	90.7	65.4	97.0
	Mixed	1	87.3	57.3	95.0	87.6	73.0	97.0
SGS	Sulfide	10	84.0	63.2	93.9	83.1	46.6	90.2
	Transition	1	45.9	93.0	96.0	62.7	53.7	96.0

Rougher concentrate cyanidation tests show relatively low recoveries in cyanidation, in particular for sulfide samples.

On the other hand, for flotation tails cyanidation, very good results were obtained for Au and Ag combined recoveries (flotation and cyanidation).

16.1.5 Sulphur oxidation

Sulfur oxidation tests were made, including roasting, pressure oxidation (POX) and biooxidation test. The Table 16.1-14 shows the main results obtained in each of the campaign. Roasting and POX tests were carried out by HAZEN, while biooxidation tests were carried out by BIOMIN Technologies SA.

Table 16.1-14: Roasting results

Roast Test	Roast No.	BK-1		BK-2		BK-3		BK-4		BK-5		BK-6		BK-7	
		Stg 1	Stg 2	Stg 1	Stg 2	Stg 1	Stg 2	Stg 1	Stg 2	Stg 1	Stg 2	Stg 1	Stg 2	Stg 1	Stg 2
Roast Conditions															
Temperature, °C	na	475	475	675	675	675	675	475	530	475	600	475	650	475	700
Gas Atmosphere	na	Air	Air	Air	Air	Air	Air+O_2	Air	Air+O_2	Air	Air+O_2	Air	Air+O_2	Air	Air+O_2
Gas Flow, sL/min	na	1.0	2.0	1.0	2.0	1.0	2.4	1.0	2.4	1.0	2.4	1.0	2.4	1.0	2.4
Retention Time, min	na	20	120	20	120	20	120	20	120	20	120	20	120	20	120
Test Charge,[a] g	na	199.0	-	197.8	-	149.0	-	198.3	-	199.7	-	198.0	-	198.0	-
Calcine (-30 mesh)															
Mass, g	na		172.8	-	155.0	-	106.6	-	160.4	-	157.6	-	153.5	-	154.3
Weight Loss, %	na	100.0	13.2	-	21.6	-	28.5	-	19.1	-	21.1	-	22.5	-	22.1
Leach Evaluation															
No.	3201-18		3201-19		3201-20		3201-21		3201-22		3201-23		3201-24		3201-25
Au Head, g/t	33.7		39.0		44.3		44.2		41.2		43.3		42.8		42.9
Ag Head, g/t	120		140		147		148		152		160		160		153
S^{total}, %	30.5		16.3		1.65		1.21		3.47		1.59		1.42		0.85
SO_4^{2-}, %	0.24		5.4		2.7		2.6		8.8		4.7		4.0		2.7
S^{2-} (by diff.), %	30.4		14.5		0.74		0.35		0.55		0.02		0.09		-0.04
Au Tails,[b] g/t	16.4		4.80		6.60		5.80		3.20		4.00		4.70		5.3
Au Extraction,[c] %	**52.1**		**87.3**		**84.9**		**86.9**		**91.8**		**90.7**		**89.0**		**87.8**
Ag Extraction, %	**53.2**		**75.6**		**36.8**		**31.1**		**61.6**		**36.5**		**23.9**		**27.9**
Lime Requirement, kg/t	2.58		50.9		24.0		15.2		74.9		31.0		24.3		11.5
CN^- Consumption, kg/t	6.39		23.8		11.4		12.2		18.5		15.5		14.0		11.9
Leach Balance (Au),[d] %	101.6		103.2		104.5		101.2		101.6		98.1		98.9		99.1

na = not applicable
[a]Roaster charge is 400 g coarse sand plus ~200 g ore
[b]Leach tails assay is average of 2 assay measurements
[c]Gold extraction calculated from Au in head and tails
[d]Balance is Au out/Au in as a percent.

In the table above, the first column corresponds to the baseline results; the baseline considers direct cyanidation of the flotation concentrate, that is, without any sulfur oxidation pre-treatment.

Silver extraction decreases considerably (with respect to the baseline) when roasting is performed at temperatures above 600 °C. Without taking into consideration those results, silver extraction ranged from 67 to 76%. Gold extraction ranged from 87 to 92%.

The best performance was achieved for kiln operation, with a first stage at 475 °C, followed by a second stage at 530 °C. Gold extraction for a baseline leach of the concentrate was 52%, while silver extraction was 53%.

Table 16.1-15: Pressure oxidation (POX) and roasting results

TEST	POX Head [g/t] Au	POX Recovery [%] Au	POX % Sulphide Oxidation	CYANIDATION (AuCN) leach [g/t]	ROASTING Head [g/t] Au	ROASTING Recovery [%] Au	ROASTING % Sulphide Oxidation	CYANIDATION (AuCN) leach [g/t]
Baseline A1	30,8		0,0	3,7	30,8		0,0	3,7
Baseline A2	32,6		0,0	3,5	32,6		0,0	3,5
Baseline A3	30,0		0,0	3,6	30,0		0,0	3,6
A1: Silica [2]	3,9	95,2	94,9	0,87	2,9	84,8	99,4	0,6
A2: Silica [2]	3,8	97,5	97,6	0,81	3,1	91,3	97,3	0,6
A3: Silica [2]	3,7	95,2	96,4	0,87	3,1	91,8	99,7	0,6
Average	3,8	96,0	96,3	0,85	3,1	89,3	98,8	0,6

2. Test conducted at a mix ratio of 10% A con:90% silica

Autoclave tests (POX) indicate gold recoveries in the mid 95% range. It must be noted that autoclave feeds were blended with silica because of the high fuel value (sulfide content). See Table 16.1-15.

Table 16.1-16: Sulfur oxidation profiles per phase of the BIOX mini pilot plant operation

Phase of Operation	Retention days	Sulphide Oxidation % P1	S1	S2	S3	O/F
Phase D, average	6	21.0	63.7	74.6	85	91.1
Phase E, average	6	19.3	59.2	59.4	74.5	81.8
Phase F, average	5	18.6	60.9	75.2	83.7	85.1
Phase G, average	5.5	-	41.8	67.2	69.2	84.0

The plant achieved steady state conditions during phase F and fairly good overflow sulfide oxidation values were recorded. The BIOX product and acid solution generated during this phase was used for downstream testing. Phase F had 5 days of retention time. See Table 16.1-16.

Table 16.1-17: Average gold dissolution and reagent consumption in biooxidation tests

Phase	NaCN Addition kg/t	Consumption NaCN* kg/t	Consumption CaO* kg/t	Residual NaCN in solution g/l	Assayed Head g/t	Residual Au Grade g/t	Gold Dissolution %
D	20	13,4	3,2	1,52	13,5	1,02	92,8
E	20	14,5	4,4	1,29	14,7	1,58	89,4
F	20	14,9	5,7	1,19	13,5	1,07	92,3
G	23	17,1	6,9	1,29	13,1	0,96	92,8

Gold recoveries in bio oxidation tests are between 90% and 93%, for 5 to 6 days of retention time. See Table 16.1-17.

16.1.6 Conclusions

The conclusions that can be reached from the testwork are the Table 16.1-18.:

Table 16.1-18: Summary flotation & tails cyanidation testwork metal recoveries

Ore	Flotation recoveries			Au distribution flotation products		Recovery tails cyanidation	Flotation & tails cyanidation recovery
	Rougher	Cleaner	Overall	Cl concentate	Flotation tails	Global Tails	
	P80=106 µm	P80=106 µm		106 µm	106 µm	106 µm	106 µm
Sulfide	**93,0**	**93,0**	**86,5**	**86,5**	**13,5**	**58,5**	**94,4**
Transition	**75,0**	**65,7**	**49,3**	**49,3**	**50,7**	**92,3**	**96,1**
Oxide					**100,0**	**95,0**	**95,0**

Ore	Flotation recoveries			Ag distribution flotation products		Recovery tails cyanidation	Flotation & tails cyanidation recovery
	Rougher	Cleaner	Overall	Cl concentate	Flotation tails	Global Tails	
	P80=106 µm	P80=106 µm		106 µm	106 µm	106 µm	106 µm
Sulfide	**87,0**	**96,3**	**83,8**	**83,8**	**16,2**	**47,8**	**91,5**
Transition	**74,8**	**84,0**	**62,8**	**62,8**	**37,2**	**59,8**	**85,0**
Oxide					**100,0**	**84,5**	**84,5**

Testwork showed that sulfide and transition ores respond well to a flotation stage performed on the whole mineral, followed by flotation tails cyanidation; for sulfides this circuit allows a gold recovery of 94.4% and a silver recovery of 91.5%.

- The flotation recoveries are 86.5% gold and 83.8% silver for sulfide samples, and 49% gold and 63% silver for transition samples
    - The rougher recoveries are 93% gold and 87% silver for sulfide samples, and 75% gold and silver for transition samples
    - The cleaner recoveries are 93% gold and 96% silver for sulfide samples, and 66% gold and 84% silver for transition samples
- The recoveries for rougher tails cyanidation are 59% gold and 48% silver for sulfide samples, and 92% gold and 60% silver for transition samples.

The Table 16.1-19 shows the metal recoveries of the sulfides oxidation and cyanidation testwork.

Table 16.1-19: Sulphur oxidation & cyanidation testwork metal recoveries

Test	Sulfur oxidation %	Gold extraction for cyanidation %	Observations
Roasting	-	91	first stage at 475 °C Second stage at 530 °C
POX	95	96	Test conducted at a mix ratio: of 10% flotation concentrate/90% silica
BIOX	85	92	Retention time: 5 days first stage in one primary reactor second stage in three secundary reactors

The Tables 16.1-20, 16.1-21, 16.1-22, 16.1-23 and 16.1-24 show the design values for the project. It is considered that the rougher concentrate shall be regrinding to a size of 37 µm. This should allow the production of a reduced quantity of cleaner concentrate and would reduce the capacity, size and cost, of the expensive refractory process unit operation. At the same time it would probably liberate more gold from the pyrite concentrate. Alquimia have assumed a recovery of 90% gold and 80% silver in the cleaner tailings cyanidation. However, Alquimia's assumptions would require confirmatory testwork, particularly to study the effect of regrinding size in both, cleaner flotation and cleaner-scavenger tails cyanidation.

Table 16.1-20: Flotation & tails cyanidation design metal recoveries

Ore	Flotation recoveries			Au distribution flotation products			Recovery tails cyanidation		Flotation & tails cyanidation recovery
	Rougher P80=106 µm	Cleaner P80=37 µm	Overall	Cl concentate P80=37 µm	Ro tails P80=106 µm	Scav tails P80=37 µm	Rougher Tails P80=106 µm	Scavenger Tails P80=37 µm	
Sulfide	**93,0**	**65,1**	**60,5**	**60,5**	**7,0**	**32,5**	**58,5**	**90,0**	**93,8**
Transition	**75,0**	**45,0**	**33,8**	**33,8**	**25,0**	**41,3**	**92,3**	**95,0**	**96,0**
Oxide					**100,0**		**95,0**	**95,0**	**95,0**

Ore	Flotation recoveries			Ag distribution flotation products			Recovery tails cyanidation		Flotation & tails cyanidation recovery
	Rougher P80=106 µm	Cleaner P80=37µm	Overall	Cl concentate P80=37 µm	Ro tails P80=106 µm	Scav tails P80=37 µm	Rougher Tails P80=106 µm	Scavenger Tails P80=37 µm	
Sulfide	**87,0**	**86,7**	**75,4**	**75,4**	**13,0**	**11,6**	**47,8**	**80,0**	**90,9**
Transition	**74,8**	**75,0**	**56,1**	**56,1**	**25,3**	**18,7**	**59,8**	**85,0**	**87,0**
Oxide					**100,0**		**84,5**	**85,0**	**84,5**

Table 16.1-21: Overall Au recoveries – Roasting alternative

Ore Type	Au distribution [%] flotation products			Cyanidation recoveries [%]			Au distribution [%] after roasting			Overall recovery
	Cl concentate	Ro tails	Scav tails	Sulfur oxidated Concentrate	Rougher tails	Scavenger tails	Sulfur oxidated Concentrate	Rougher tails	Scavenger tails	
	P80=37 µm	P80=106 µm	P80=37 µm		P80=106 µm	P80=37 µm		P80=106 µm	P80=37 µm	
Sulphide	**60,5**	**7,0**	**32,5**	**91,0**	**58,5**	**90,0**	**55,1**	**4,1**	**29,2**	**88,4**
Transition	**33,8**	**25,0**	**41,3**	**91,0**	**92,3**	**95,0**	**30,7**	**23,1**	**39,2**	**93,0**
Oxide		**100,0**			**95,0**			**95,0**		**95,0**

Table 16.1-22: Overall Au recoveries – POX alternative

Ore Type	Au distribution [%] flotation products			Cyanidation recoveries [%]			Au distribution [%] after POX			Overall recovery
	Cl concentate	Ro tails	Scav tails	Sulfur oxidated Concentrate	Rougher tails	Scavenger tails	Sulfur oxidated Concentrate	Rougher tails	Scavenger tails	
	P80=37 µm	P80=106 µm	P80=37 µm		P80=106 µm	P80=37 µm		P80=106 µm	P80=37 µm	
Sulphide	**60,5**	**7,0**	**32,5**	**96,0**	**58,5**	**90,0**	**58,1**	**4,1**	**29,2**	**91,4**
Transition	**33,8**	**25,0**	**41,3**	**96,0**	**92,3**	**95,0**	**32,4**	**23,1**	**39,2**	**94,7**
Oxide		**100,0**			**95,0**			**95,0**		**95,0**

Table 16.1-23: Overall Au recoveries – BIOX alternative

Ore Type	Au distribution [%] flotation products			Cyanidation recoveries [%]			Au distribution [%] after BIOX			Overall recovery
	Cl concentate	Ro tails	Scav tails	Sulfur oxidated Concentrate	Rougher tails	Scavenger tails	Sulfur oxidated Concentrate	Rougher tails	Scavenger tails	
	P80=37 µm	P80=106 µm	P80=37 µm		P80=106 µm	P80=37 µm		P80=106 µm	P80=37 µm	
Sulphide	**60,5**	**7,0**	**32,5**	**92,0**	**58,5**	**90,0**	**55,7**	**4,1**	**29,2**	**89,0**
Transition	**33,8**	**25,0**	**41,3**	**92,0**	**92,3**	**95,0**	**31,1**	**23,1**	**39,2**	**93,3**
Oxide		**100,0**			**95,0**			**95,0**		**95,0**

Table 16.1-24: Overall Ag recoveries – All alternatives

Ore Type	Ag distribution [%] flotation products			Cyanidation recoveries [%]			Ag distribution [%] after roasting			Overall recovery
	Cl concentate	Ro tails	Scav tails	Sulfur oxidated Concentrate	Rougher tails	Scavenger tails	Sulfur oxidated Concentrate	Rougher tails	Scavenger tails	
	P80=37 µm	P80=106 µm	P80=37 µm		P80=106 µm	P80=37 µm		P80=106 µm	P80=37 µm	
Sulphide	**75,4**	**13,0**	**11,6**	**60,0**	**47,8**	**80,0**	**45,2**	**6,2**	**9,3**	**60,7**
Transition	**56,1**	**25,3**	**18,7**	**60,0**	**59,8**	**85,0**	**33,6**	**15,1**	**15,9**	**64,6**
Oxide		**100,0**			**84,5**	**85,0**		**84,5**		**84,5**

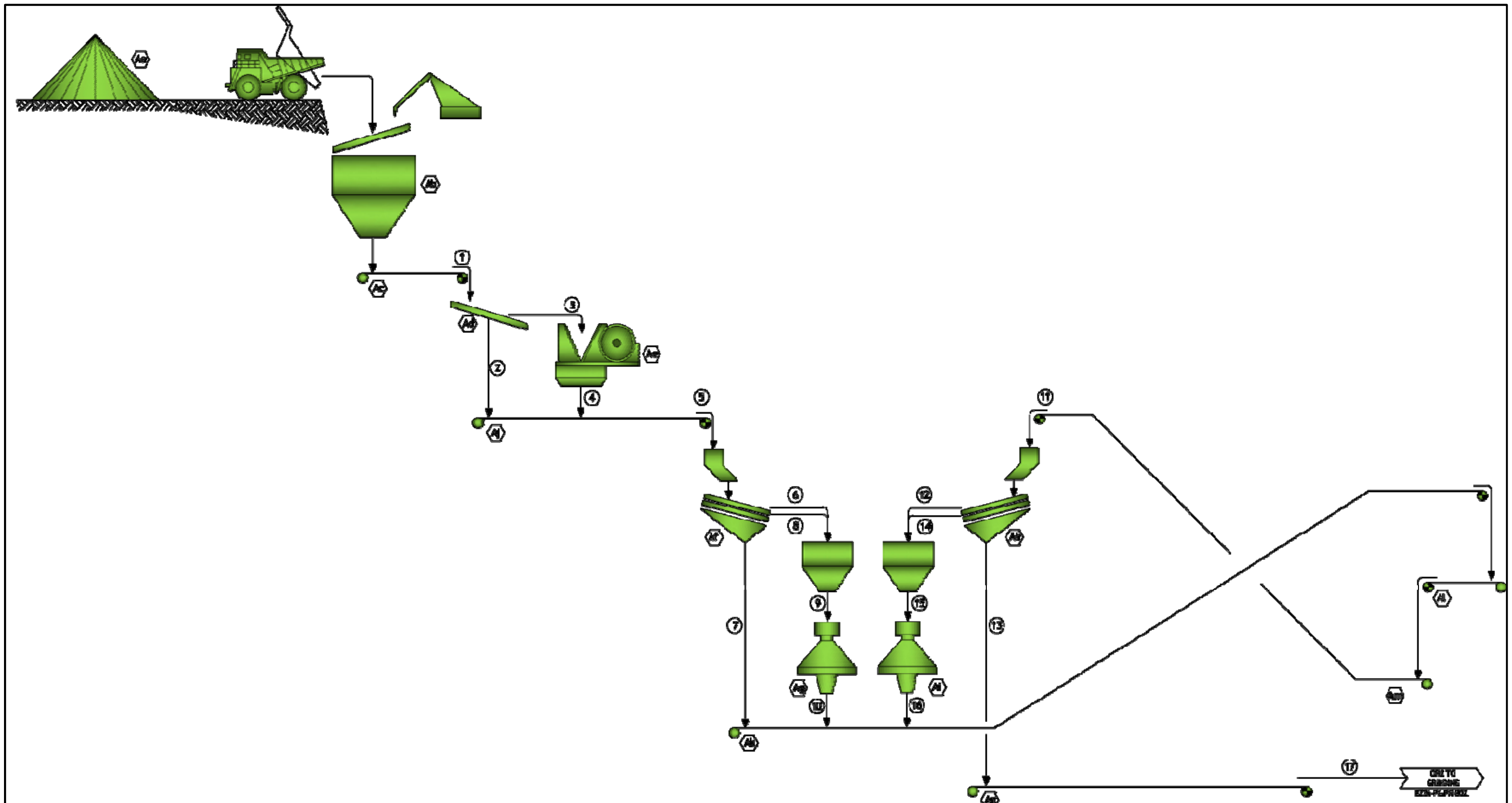
16.2 Plant Design

The processing operation was designed for a nominal throughput of 3,288 tpd (tonnes per day) with an average head grade of 5.5 g/t Au, 18 g/t Ag and 770 g/t

Cu. According to the mining plan, the ore type composition is: 75% sulfur; 15% transition and 10% oxide.

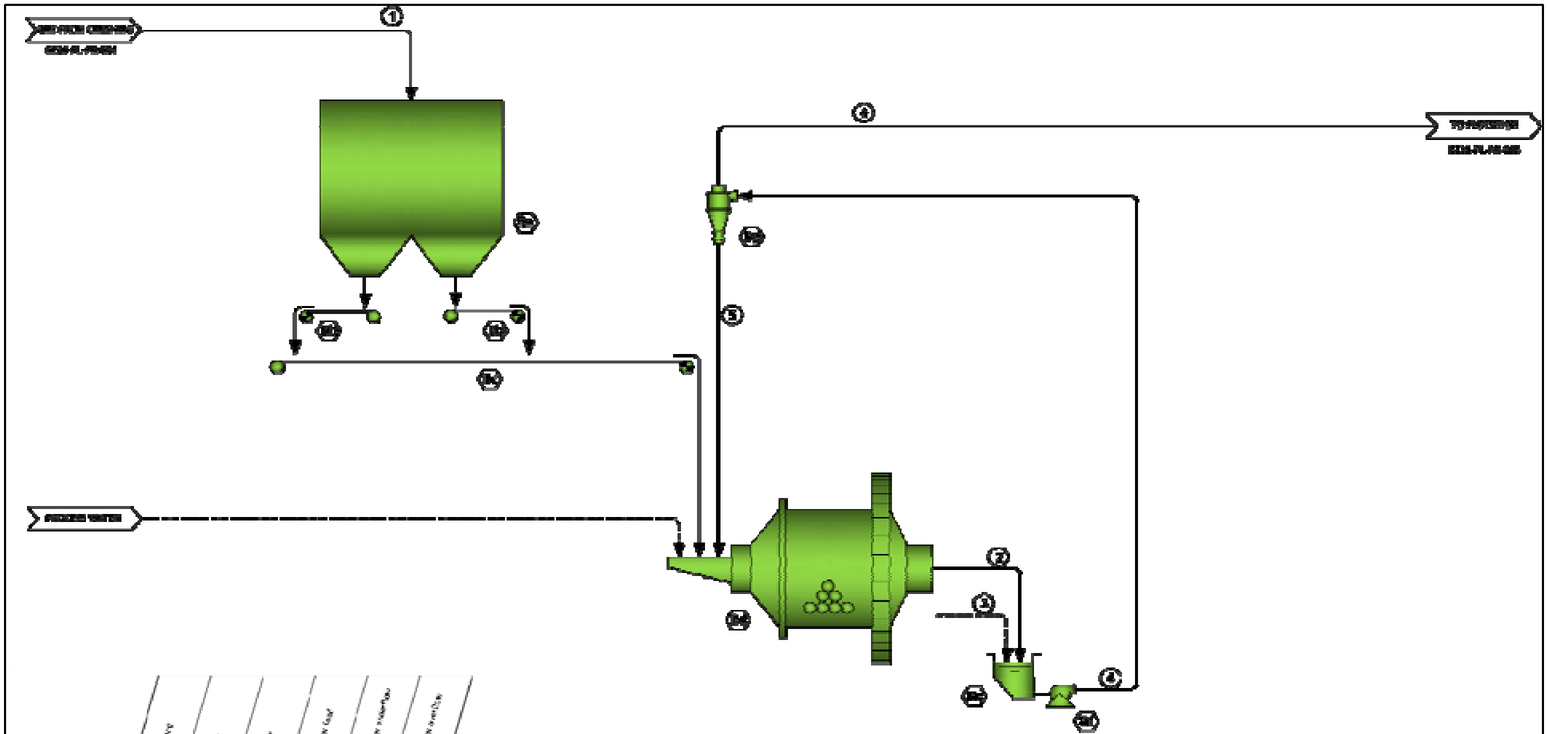
Considering the ore type composition, the process was designed in order to recover the most gold and silver contained in each type of ore. The Figures 16.2-2 to 16.2-11 correspond to the flowsheets for the different stages, and to the design plant lay out. A description of all the different stages in the plant can be found next.

Figure 16.2-2: Crushing flow sheet



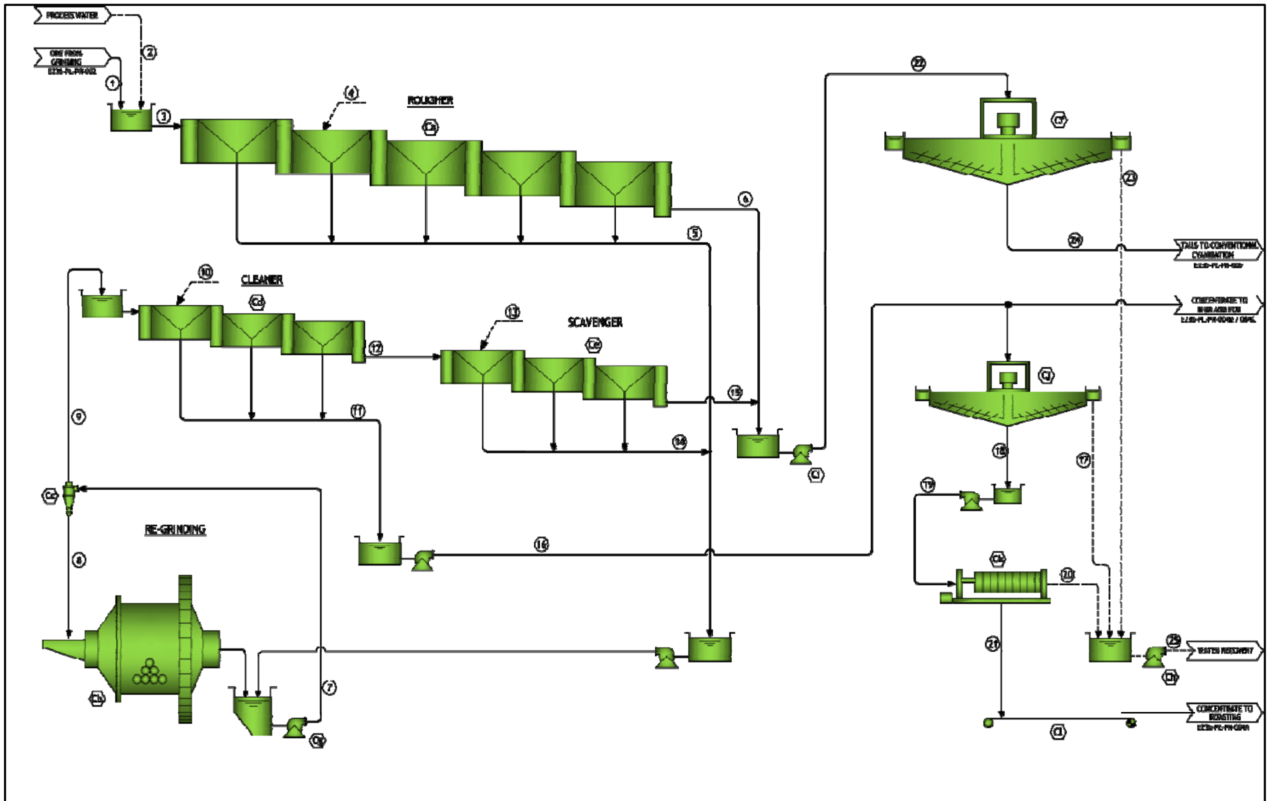
Item	Units	Feed to crushing	Grizzly undersize	Grizzly oversize	Primary crusher discharge	Secondary screen feed	Secondary screen oversize 1st deck	Secondary screen undersize 2nd deck	Secondary screen oversize 2nd deck	Secondary crusher feed	Secondary crusher discharge	Tertiary screen feed	Tertiary screen oversize 1st deck	Tertiary screen undersize 2nd deck	Tertiary screen oversize 2nd deck	Tertiary crusher feed	Tertiary crusher discharge	Final product
		1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
Nominal Balance																		
Solid	t/d	3,288	1,716	1,570	1,570	3,288	1,751	811	725	2,476	2,476	5,142	1,084	3,288	770	1,854	1,854	3,288
	t/h	183	95	87	87	183	97	45	40	138	138	286	60	183	43	103	103	183
Moisture	%	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4
-1/2"	%	70	38	1	13	76	3	91	9	79	65	77	33	99	67	100	88	99
F_{80}	mm	142	50	182	95	71	91	10	33	79	17	13	21	8	15	19	11	8
Design Balance																		
Solid	t/d	3,945	2,061	1,884	1,884	3,945	2,102	974	870	2,972	2,972	6,170	1,301	3,945	925	2,225	2,225	3,945
	t/h	219	115	105	105	219	117	54	48	165	165	343	72	219	51	124	124	219

Figure 16.2-3: Grinding flow sheet



Item	Units	1 Feed to grinding	2 Mill discharge	3 Dilution water	4 Cyclone cluster feed	5 Cyclone cluster underflow	6 Cyclone cluster overflow
Nominal Balance							
Solid	t/d	3,288	13,152		13,152	9,864	3,288
	t/h	149	598		598	447	149
Grade	g/t Au	5.5	5.5		5.5	5.5	5.5
	g/t Ag	18	18		18	18	18
	g/t Cu	720	770		770	770	770
Water/solution	m3/h	6	198	279	466	191	277
Slurry	t/h	155	793		1064	638	425
	m3/h	2	352		679	350	329
SG		2.82	2.82		2.82	2.82	2.82
Slurry density	t/m³	1.63	2.25		1.57	1.82	1.29
Cw	%	96	75		56.0	70	35
Design Balance							
Solid	t/d	3,946	15,782		15,782	11,837	3,946
	t/h	179	715		715	536	179

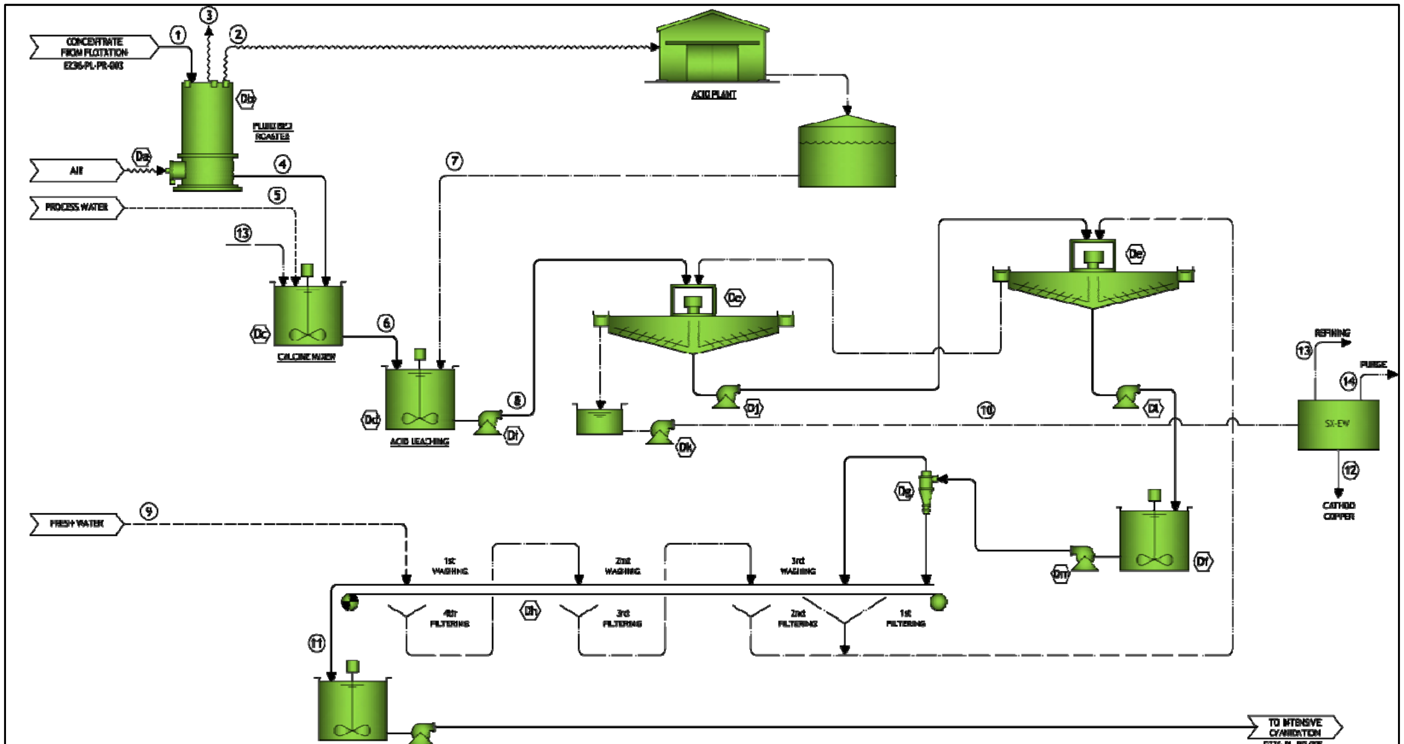
Figure 16.2-4: Flotation flow sheet



Item	Units	1	2	3	4	5	6	7	8	9	10	11	12
		Feed to flotation	Dilution water for flotation feed	Rougher flotation feed	Dilution water for rougher flotation concentrate	Rougher flotation concentrate diluted	Rougher flotation tails	Cyclone cluster feed	Regrinding mill feed	Cleaner flotation feed	Dilution water for cleaner flotation concentrate	Cleaner flotation concentrate diluted	Cleaner flotation tails - scavenger flotation feed
Solid	t/d	3,288		3,288		493	2,795	2,700	1,620	1,080		329	758
	t/h	149		149		22	127	122	73	49		15	34
Grade	g/t Au	5.5		5.5		34	0	32	32	32		33	31
	g/t Ag	18		18		104	3	62	62	62		136	30
	g/t Cu	770		770		4,877	45	2,487	2,487	2,487		6,949	534
Water/solution	m3/h	277	0	277	15	67	224	193	31	161	4	38	123
Slurry	t/h	425		425		89	351	315	105	210		53	157
	m3/h	329		329		74	270	231	54	176		42	134
SG		2.82		2.82		3.20	2.76	3.20	3.20	3.20		4.00	2.94
Slurry density	t/m³	1.29		1.29		1.21	1.30	1.36	1.93	1.19		1.27	1.17
Cw	%	35.0		35		25	36	39	70	23		28	22

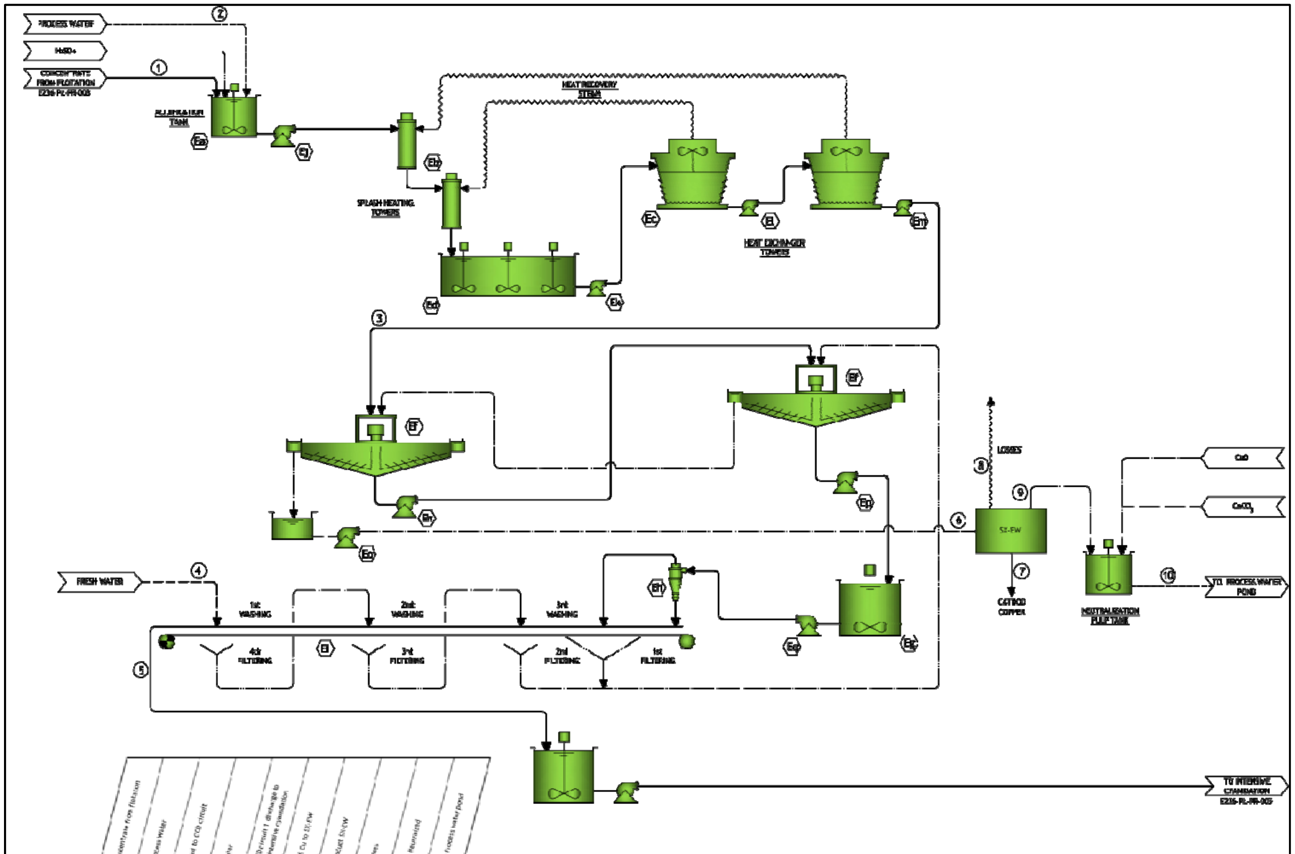
Item	Units	13	14	15	16	17	18	19	20	21	22	23	24	25
		Dilution water for scavenger flotation concentrate	Scavenger flotation concentrate diluted	Scavenger flotation tails	Concentrate thickener feed	Water recovery from concentrate thickener	Concentrate thickener discharge	Concentrate filtering feed	Water recovery from concentrate filtering	Concentrate to sulphur oxidation	Tails thickener feed	Water recovery from tails thickener	Tails thickener discharge	Total water recovery
Solid	t/d		587	164	329		329	329		329	2,959		2,959	
	t/h		27	7	14		14	17		17	126		126	
Grade	g/t Au		30	36	33		33	33		33	2.4		2.4	
	g/t Ag		27	41	136		136	136		136	4.9		4.9	
	g/t Cu		478	731	6,949		6,949	6,949		6,949	83		83	
Water/solution	m3/h	14	94	29	36	25	11	14	11	3	238	170	68	199
Slurry	t/h		121	36	50		25	31		20	363		194	
	m3/h		103	32	39		15	18		7	284		114	
SG			3.20	2.29	4.00		4.00	4.00		4.00	2.73		2.73	
Slurry density	t/m³		1.18	1.15	1.27		1.70	1.70		2.76	1.28		1.70	
Cw	%		22	21	28		55	55		85	35		65	

Figure 16.2-5: Sulphur oxidation (Alternative Roasting) flowsheet



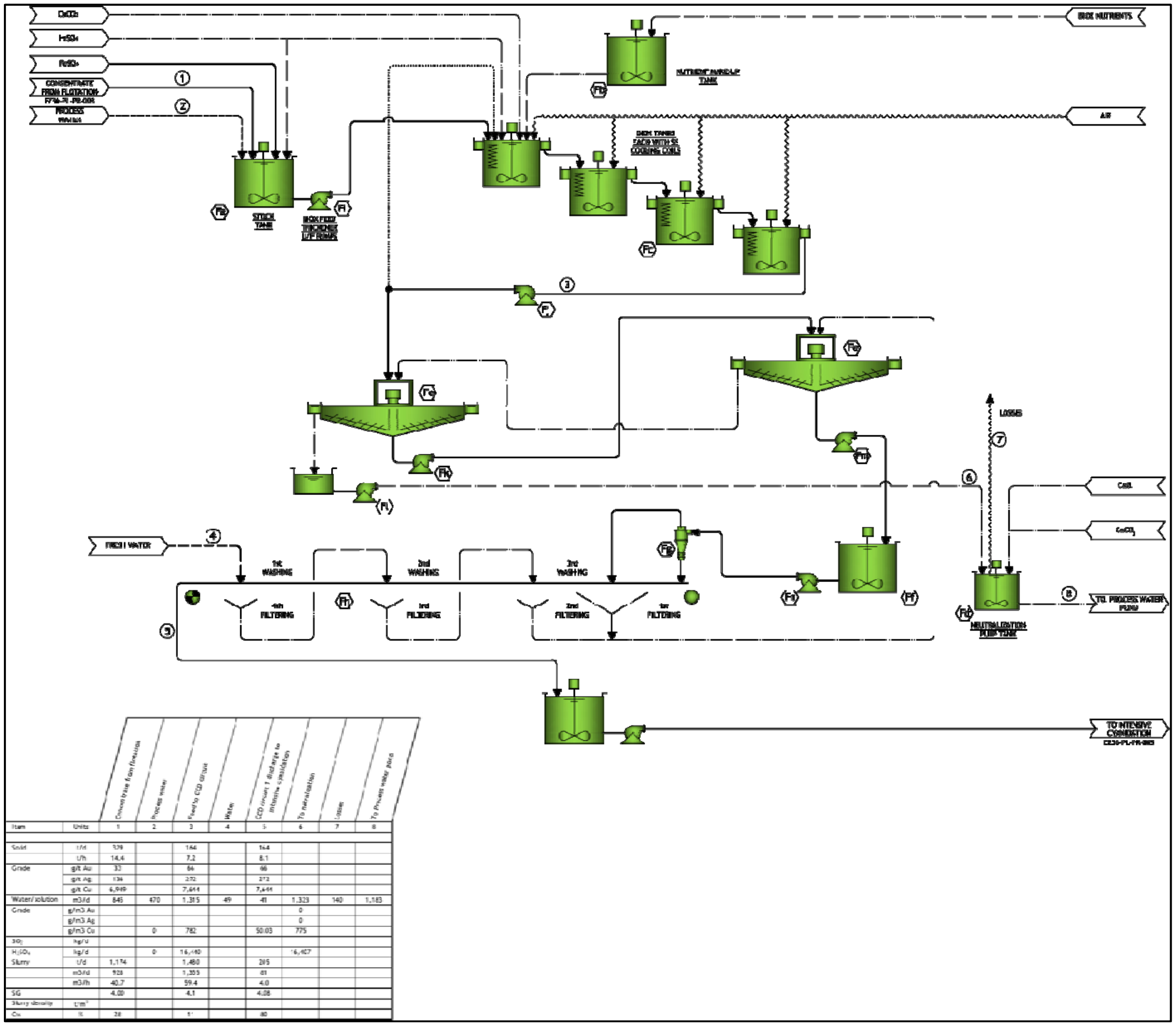
Item	Units	1 Concentrate from flotation	2 SO_2 generated	3 Steam	4 Calcined concentrate	5 Process water in acid leaching	6 Feed to acid leaching	7 H_2SO_4 in acid leaching	8 Product leaching - feed to CCD circuit 1	9 Wash water CCD circuit 1	10 PLS Cu to SX-EW	11 CCD circuit 1 discharge to intensive cyanidation	12 Product SX-EW	13 Recycled to acid leaching	14 Purge
Solid	t/d	329			249		249		247			247			
	t/h	14			11		11		12			12			
Grade	g/t Au	33			44		44		44			44			
	g/t Ag	136			180		184		181			181			
	g/t Cu	6,949			9,157		9,167		1,386			1,386			
Water/solution	m3/d	58		58	0	305	305		305	74	317	62		250	63
Grade	g/m3 Au	0				0.0	0		0	0	0	0		0	0
	g/m3 Ag	0				0.0	0		0	0	0	0		0	0
	g/m3 Cu	0				51	51		6,475	0	6,163	63		67	67
SO_2	kg/d		265,161												
H_2SO_4	kg/d					9,826	9,816	2,637	12,463		12,438	25		9,826	2,488
Slurry	t/d	387					554		554			309			
	m3/d	140					365		365			122			
	m3/h	6					16		18			6			
SG		4.00					4.10		4.06			4.08			
Slurry density	t/m^3	2.76					1.52		1.52			2.53			
Cw	%	85			100		45		45			88			

Figure 16.2-6: Sulphur oxidation (Alternative POX) flow sheet



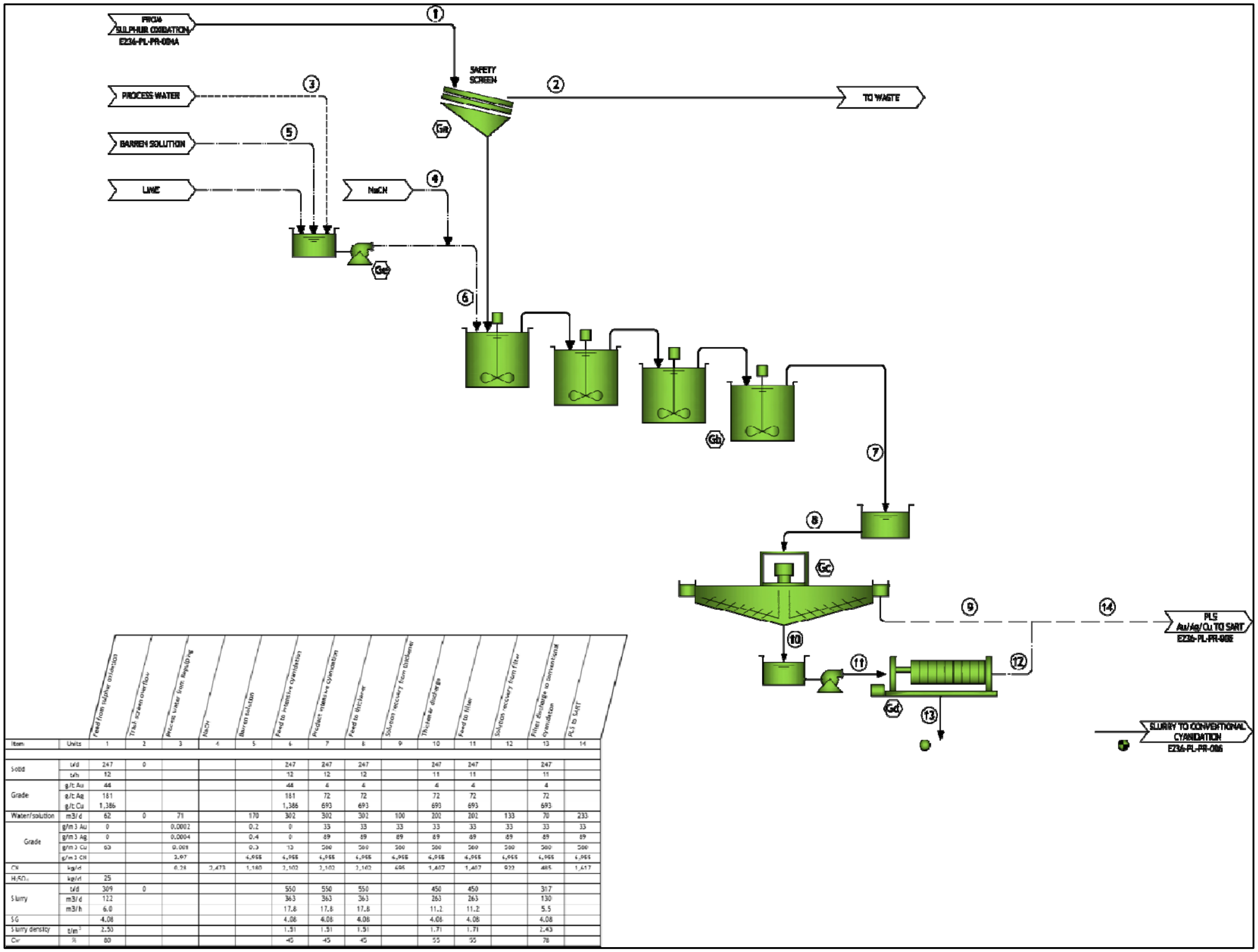
Item	Units	1 Concentrate from flotation	2 Process water	3 Feed to CCD circuit	4 Water	5 CCD circuit 1 discharge to intensive cyanidation	6 PLS Cu to SX-EW	7 Product SX-EW	8 Losses	9 To neutralized	10 To process water pond
Solid	t/d	329		164		164					
	t/h	14		7		8					
Grade	g/t Au	33		65		66					
	g/t Ag	136		272		272					
	g/t Cu	6,949		2,085		2,085					
Water/ solution	m3/d	845	0	845	49	41	854		140	714	714
Grade	g/m3 Au						0				
	g/m3 Ag						0				
	g/m3 Cu			2,297		95	2,270				
SO_2	kg/d										
H_2SO_4	kg/d			16,440			16,407			16,407	
Slurry	t/d	1,174		1,010		205					
	m3/d	928		886		81					
	m3/h	41		37		4					
SG		4.80		4.1		4.38					
Slurry density	t/m³										
Cw	%	28		15		60					

Figure 16.2-7: Sulphur oxidation (Alternative BIOX) flowsheet



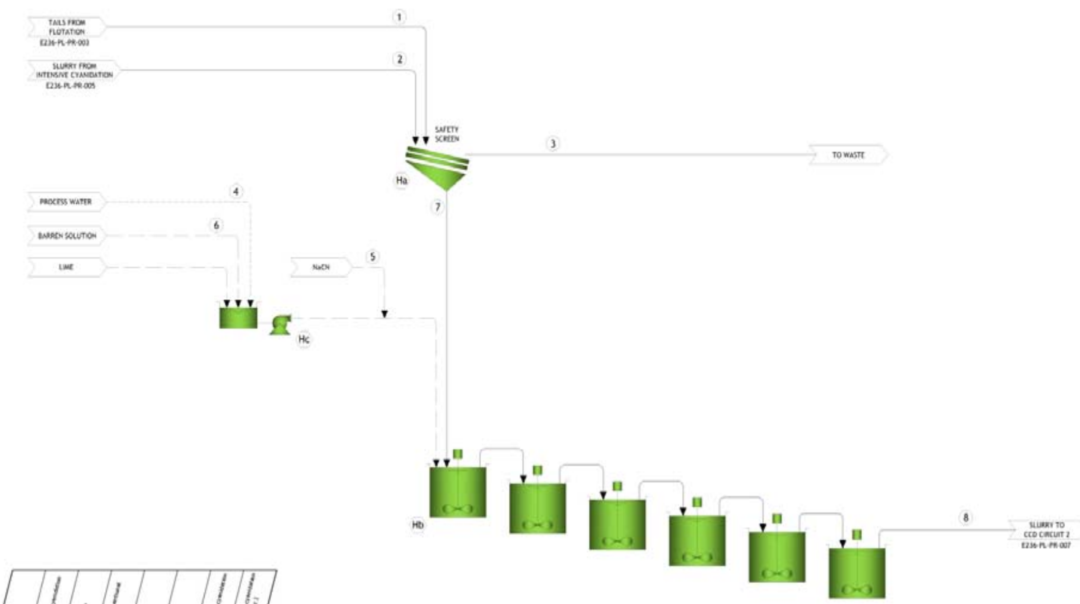
Item	Units	1 Concentrate from flotation	2 Process water	3 Feed to CCD circuit	4 Water	5 CCD circuit 1 discharge to intensive cyanidation	6 To neutralization	7 Losses	8 To Process water pond
Solid	t/d	329		164		164			
	t/h	14.4		7.2		8.1			
Grade	g/t Au	33		64		66			
	g/t Ag	136		272		272			
	g/t Cu	6,949		7,644		7,644			
Water/solution	m3/d	845	470	1,315	49	41	1,323	140	1,183
Grade	g/m3 Au						0		
	g/m3 Ag						0		
	g/m3 Cu		0	782		50.03	775		
SO_2	kg/d								
H_2SO_4	kg/d		0	16,440			16,407		
Slurry	t/d	1,174		1,480		205			
	m3/d	928		1,355		81			
	m3/h	40.7		59.4		4.0			
SG		4.00		4.1		4.08			
Slurry density	t/m^3								
Cw	%	28		11		80			

Figure 16.2-8: Concentrate cyanidation flow sheet



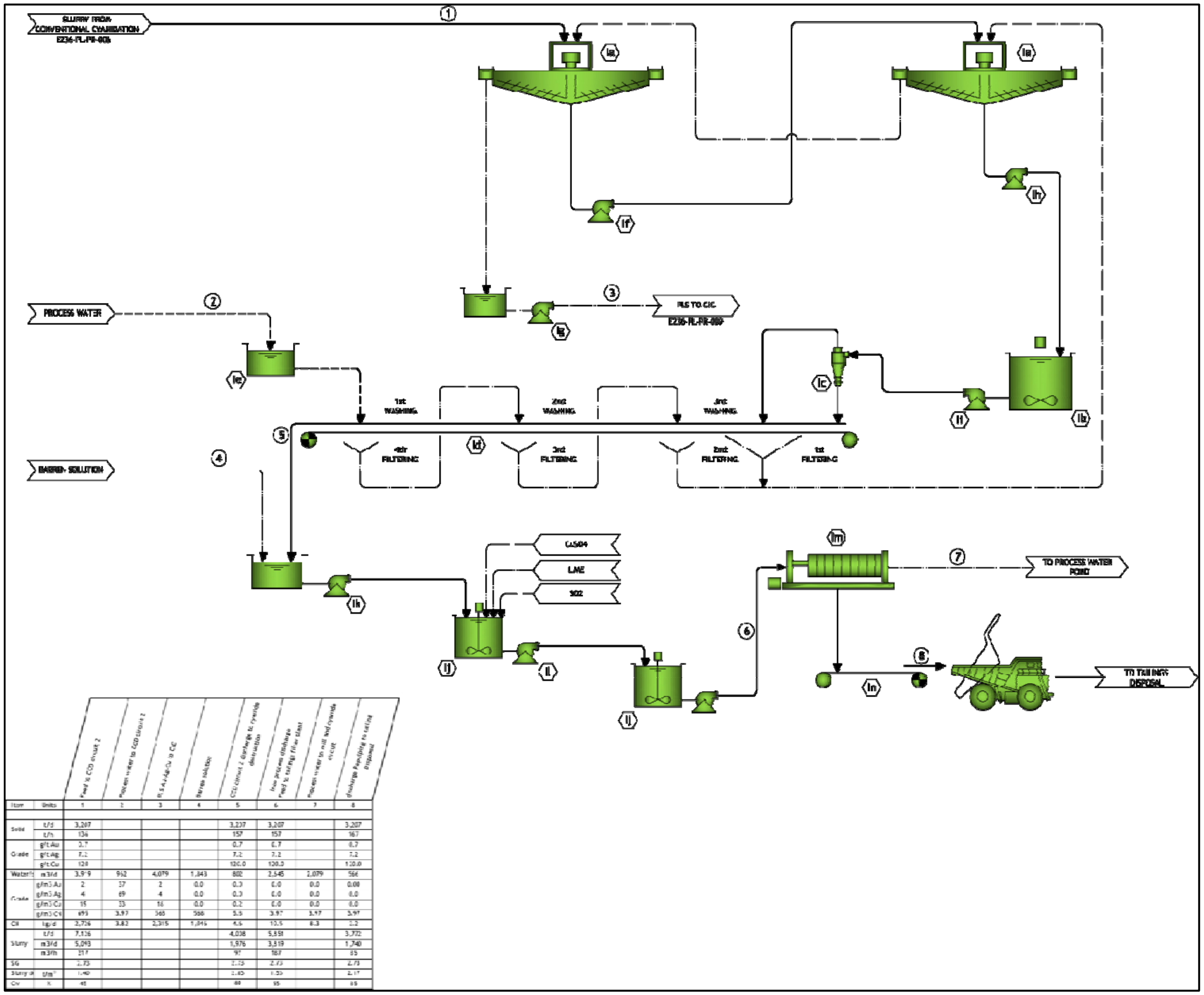
Item	Units	1 Feed from sulphur oxidation	2 Trash screen overflow	3 Process water from Repulping	4 NaCN	5 Barren solution	6 Feed to intensive cyanidation	7 Product intensive cyanidation	8 Feed to thickener	9 Solution recovery from thickener	10 Thickener discharge	11 Feed to filter	12 Solution recovery from filter	13 Filter discharge to conventional cyanidation	14 PLS to SART
Solid	t/d	247	0				247	247	247		247	247		247	
	t/h	12					12	12	12		11	11		11	
Grade	g/t Au	44					44	4	4		4	4		4	
	g/t Ag	181					181	72	72		72	72		72	
	g/t Cu	1,386					1,386	693	693		693	693		693	
Water/solution	m3/d	62	0	71		170	302	302	302	100	202	202	133	70	233
Grade	g/m3 Au	0		0.0002		0.2	0	33	33	33	33	33	33	33	33
	g/m3 Ag	0		0.0004		0.4	0	89	89	89	89	89	89	89	89
	g/m3 Cu	63		0.001		0.3	13	500	500	500	500	500	500	500	500
	g/m3 CN			3.97		6,966	6,966	6,966	6,966	6,966	6,966	6,966	6,966	6,966	6,966
CN	kg/d			0.28	2,473	1,180	2,102	2,102	2,102	696	1,407	1,407	922	485	1,617
H_2SO_4	kg/d	25													
Slurry	t/d	309	0				550	550	550		450	450		317	
	m3/d	122					363	363	363		263	263		130	
	m3/h	6.0					17.8	17.8	17.8		11.2	11.2		5.5	
SG		4.08					4.08	4.08	4.08		4.08	4.08		4.08	
Slurry density	t/m³	2.53					1.51	1.51	1.51		1.71	1.71		2.43	
Cw	%	80					45	45	45		55	55		78	

Figure 16.2-9: Tails cyanidation flowsheet



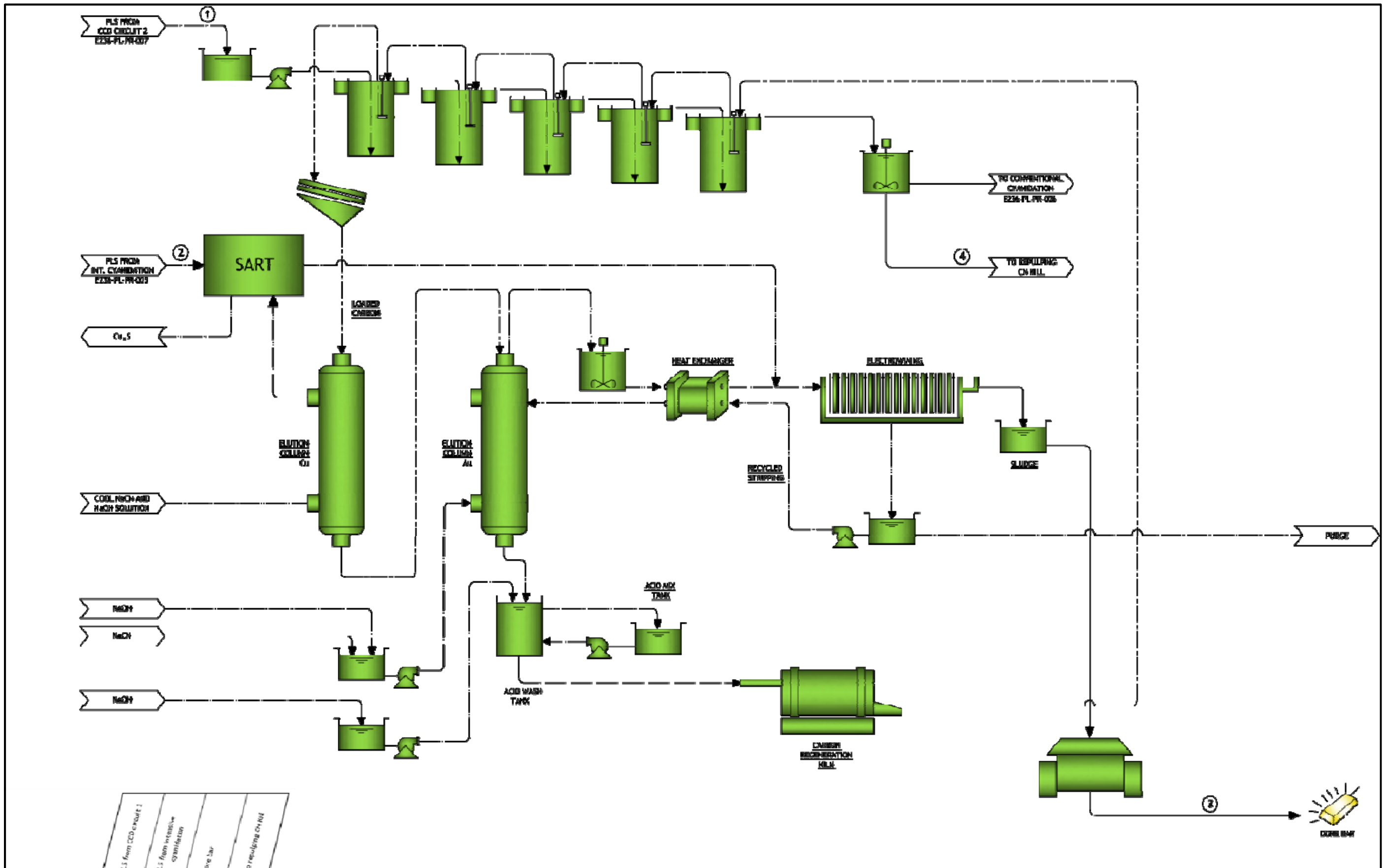
Item	Units	1 Feed from flotation	2 Feed from intensive cyanidation	3 Trash screen overflow	4 Process water to conventional cyanidation	5 NaCN	6 Barren solution	7 Feed to conventional cyanidation	8 Product conventional cyanidation to CCD circuit 2
Solid	t/d	2,959	247	0				3,206	3,206
	t/h	145	12					157	157
Grade	g/t Au	1	4					1	0.1
	g/t Ag	3	180					17	2
	g/t Cu	108	884					168	154
Water/solution	m3/d	1,593	70	0	2,195		0	3,918	3,918
Grade	g/m3 Au	0	66		0.079		0	1	2
	g/m3 Ag	0	148		0.144		0	3	15
	g/m3 Cu	0	721		0.023		0	13	24
	g/m3 CN	0	13		568		0	1	1
CN	kg/d		903		1,246	1,057		3,206	2,725
H_2SO_4	kg/d								
Slurry	t/d	4,553	316	0				7,124	7,124
	m3/d	2,677	130					5,092	5,092
	m3/h	131	6					250	250
SG		2.73	4.08					2.73	2.73
Slurry density	t/m³	1.70						1.40	1.40
Cw	%	65	78					45	45

Figure 16.2-10: Dewatering - washing and cyanide destruction flow sheet



Item	Units	1 Feed to CCD circuit 2	2 Process water to CCD circuit 2	3 PLS Au-Ag-Cu to CIC	4 Barren solution	5 CCD circuit 2 discharge to cyanide destruction	6 Inco process discharge Feed to tailings filter plant	7 Process water to mill and cyanide circuit	8 discharge Repulping to tailing disposal
Solid	t/d	3,207				3,207	3,207		3,207
	t/h	134				157	157		167
Grade	g/t Au	0.7				0.7	0.7		0.7
	g/t Ag	7.2				7.2	7.2		7.2
	g/t Cu	120				120.0	120.0		120.0
Water	m3/d	3,919	962	4,079	1,843	802	2,545	2,079	564
Grade	g/m3 Au	2	37	2	0.0	0.3	0.0	0.0	0.00
	g/m3 Ag	4	69	4	0.0	0.3	0.0	0.0	0.0
	g/m3 Cu	15	33	16	0.0	0.2	0.0	0.0	0.0
	g/m3 CN	695	3.97	568	568	5.5	3.97	3.97	3.97
CN	kg/d	2,726	3.82	2,315	1,046	4.6	10.5	8.3	2.2
Slurry	t/d	7,126				4,038	5,851		3,772
	m3/d	5,093				1,976	3,519		1,740
	m3/h	217				97	187		85
SG		2.73				2.73	2.73		2.73
Slurry density	t/m3	1.40				1.85	1.50		2.17
Cw	%	45				80	55		85

Figure 16.2-11: SART - ADR - EW & smelting



Item	Units	1 (PLS from CCD circuit 1)	2 (PLS from intensive cyanidation)	3 (Dore bar)	4 (To repulping CN kill)
Solid	t/d			0.03	
	t/h			0.001	
Grade	g/t Au			272,561	
	g/t Ag			500,000	
	g/t Cu			242,707	
Water/solution	m3/h	200	11		90
Grade	g/m3 Au	2.1	33.0		0.02
	g/m3 Ag	3.6	69.2		0.04
	g/m3 Cu	18.1	580.1		0.02
	g/m3 CN	[illegible]	[illegible]		[illegible]

16.2.1 Crushing Circuit

The extracted ore is transported to a 1 hour capacity crusher feed bin either by direct tipping from the mine trucks or by front end loaders. An apron feeder delivers ore from the crusher feed bin to a vibrating grizzly and the grizzly oversize feeds a C80 primary jaw crusher with a setting of 76.2 mm. The grizzly undersize and crusher discharge are transported by a conveyor belt to a double deck secondary screen (first deck: 38.1 mm and second deck: 15.9 mm.). The screen oversize feeds a surge bin which then discharges into a 149 kW secondary cone crusher with a setting of 15 mm.

The secondary crusher discharge and the secondary screen undersize are transported by a two-conveyor/transfer tower system to a tertiary double deck screen (first deck: 15.9 mm and second deck: 12.7 mm). The oversize gravitates to a surge bin and then to a tertiary crusher with a setting of 12 mm. The tertiary screen undersize discharges on to the final product conveyor and is transported to the crushed ore bin.

The crushed ore has a particle size calculated by simulation software to be P80= 8.5 mm.

The crushing plant considers a dust management system by suppression and collection. Suppression consists of wetting the generated dust with specially-designed spray nozzles, avoiding its suspension in the atmosphere, and thus keeping it in the process. Collection consists of gathering the generated dust through a bag filter system then sending it to a wet process.

16.2.2 Grinding and Flotation

The crushed ore bin, with one day live capacity, feeds a conveyor which transports the ore to a 2,237 kW ball mill. The mill reduces the particle size from 80% - 8,500 µm to 80% - 106 µm. The milled ore discharges, with a solids concentration of 75%, to a pump box where it is diluted to achieve a solids concentration of 56%. The slurry feeds a cyclone cluster composed of 10 x 15 inch cyclones (8 operating and 2 stand-by). The cyclone overflow, with a solids concentration of 35%, gravitates directly to the flotation stage. The underflow, with a solids concentration of 70%, returns to the ball mill.

The cyclone overflow feeds the rougher flotation stage, consisting of a bank of five 70 m^3-cells, with a residence time of 45 minutes. The rougher tails gravitate to the thickening pump box and the concentrate gravitates to the regrinding pump box. This pulp is then pumped from the mill discharge box to a cyclones cluster, composed of 5 x 10 inch cyclones (3 operating and 2 stand-by). The cyclone overflow, with a solids concentration of 25%, gravitates directly to the cleaner flotation stage, which consists of a bank of three 20 m^3-cells, with a residence time of 25 minutes, while the cyclone underflow, with a solids concentration of 70%, returns to the ball mill of 298 kW. The cleaner tails gravitate directly to the scavenger flotation stage, consisting of a bank of three 20 m^3-cells, with a residence time of 45 minutes. The concentrate gravitates to a pump box, from where it is pumped either to a dewatering stage, in order to achieve a solids concentration of 85%, for alternative A – Roasting - or to the sulfur oxidation stage for alternatives B and C – POX and BIOX. The scavenger tails join the rougher tails and gravitate together to the thickening pump box, while the cleaner concentrate joins the rougher concentrate and gravitate together to the regrinding pump box.

The flotation stage recovers about 10% of the total mass, and 60%, 50% and 85% of gold, silver and copper, respectively.

The tails from rougher and scavenger flotation stages are pumped to a tails thickener, which discharges with a solids concentration of 65%..

16.2.3 Roasting – Acid plant – Cu leaching – CCD circuit (Alternative A)

The product obtained in the flotation concentrates filtering is transported by conveyor to the fluid bed roaster, where air is added in order to perform the oxidation. In the process 95% of the sulfurs are oxidized, with a mass loss of about 25%. The reaction produces SO_2, which is used to produce sulfuric acid in an acid plant, and calcine. The calcine is sent to a calcine mixer tank, and from there to the acid leaching stage, to leach the copper contained in the mineral. A part of the acid produced is used in the copper leaching stage; both, the copper produced in this stage and the residual acid, are considered as credits.

The slurry obtained in the acid leaching stage is then pumped to a CCD circuit, which objective is to separate the solids from the Cu-rich PLS. The PLS is pumped to a SX-EW stage, while the solids are sent to the next stage.

All CCD circuits considered in the plant consist of two thickeners operating in a countercurrent fashion and two belt filters, operating in parallel. It was established, through mass balances simulations, that this arrangement has a high efficiency in recovering in the solution, most of the precious metals contained in the slurry. The CCD efficiency is estimated to be at least 99%.

16.2.4 POX – CCD circuit (Alternative B)

The product obtained in the flotation concentrates thickening is pumped to the pressure oxidation stage (POX), which starts with an acidification stage, where acid is added to the concentrate in an acidification tank. The resulting slurry is pumped to the splash heating towers, where the required operational temperature is achieved. The heated slurry gravitates to the autoclave-agitated tank, where the sulfur is oxidized under high pressure conditions. The process allows a sulfide oxidation of 95% and a Cu dissolution of 71%, with mass losses of about 50%.

The oxidized slurry is pumped to the heat exchanger towers, where is cooled down. The cooled slurry is pumped to a CCD circuit, which objective is to separate the solids from the Cu-rich PLS. The PLS is pumped to a SX-EW stage, while the solids are sent to the next stage.

16.2.5 BIO-OX – CCD circuit (Alternative C)

The cleaner flotation concentrates feed the bio-oxidation stage (BIO-OX), which starts with the addition of acid and FeSO4 in a stock tank. The conditioned slurry is pumped to the BIO-OX tanks, where nutrients and other reagents required for the oxidation are added. The process allows a sulfide oxidation of 90% and a Cu dissolution of 54%, with mass losses of about 50%.

The oxidized slurry is pumped to a CCD circuit, where the solids are separated from the solution. The solution is then pumped to a neutralization stage, where Cao and CaCO3 are added. A solution bleed stream is sent to the tailings dam. The CCD solids go to the next stage.

16.2.6 Concentrate cyanidation

The product from the sulfur oxidation is pumped to the concentrate cyanidation stage, which starts with a vibrating trash screen to remove oversize waste, such as roots, plastic residues and others. The screen undersize feeds the first cyanidation tank, where also is added a pulp composed of lime, barren solution and process water. The cyanidation circuit is composed of 4 tanks with agitators which allow a residence time of 72 hours.

The cyanide consumption is 10 kg/t of solid feed. The pulp gravitates from one tank to another. Every tank can be bypassed in order to permit maintenance when required.

The slurry is pumped to a dewatering stage from which a PLS rich in Au/Ag/Cu is recovered and a slurry that is transported by conveyor to the next stage.

16.2.7 Conventional cyanidation, CCD circuit, cyanide destruction and filter plant

The thickened flotation tails and the concentrate cyanidation product are pumped to the conventional cyanidation stage leaching tanks, where is also added lime, cyanide, barren solution and process water. The cyanidation circuit is composed of 6 agitated tanks at 15 meter high for 15 meters diameter, which allow a residence time of 48 hours.

The cyanide consumption is 1.0 kg/t of solid feed. The slurry gravitates from one tank to another. Every tank can be bypassed in order to permit maintenance when required.

The slurry product from conventional cyanidation is fed to the CCD circuit, where the solids are separated from the metal-rich PLS. Since this is a final stage, a CCD circuit must be considered in order to recover most of the Au and Ag contained in the slurry, otherwise they would remain in the solids and would be sent to the tailings dam.

The PLS reports to the CIC process, while the solids are sent to the INCO process for CN destruction, together with the barren solution. The INCO process consumes SO_2, $CuSO_4$ and CaO with doses of 6 g/g of CN, 0.12 g/g of CN and 0.3 g/g of CN, respectively, and is performed in two aerated tanks at 10.5 meters high for 7 meters diameter, with a residence time of 2 hours. The INCO process has an efficiency of 99%.

The slurry resulting from the INCO process is sent to a filtering stage, in order to recover process water. The solids obtained from the filters are sent to tailings disposal trough trucks.

16.2.8 SART – CIC – Elution – EW – smelting

The PLS obtained in concentrate cyanidation is sent to the SART process, where copper is extracted as Cu_2S with a 90% of efficiency. The Au/Ag PLS, now with a low content of Cu, is sent to EW.

The PLS obtained in conventional cyanidation is sent to the activated carbon adsorption stage – CIC. The barren solution obtained in CIC, is sent to a storage pond, while the loaded carbon is first sent to the copper elution stage and then to the Au/Ag elution stage. Finally, the stripped carbon is sent to the regeneration stage, while the pregnant solution is sent to EW.

The EW sludge reports to the smelting stage, where Dore bars are obtained. Dore bars have a minimum composition of 70% of Au+Ag and 30% of Cu.

The CIC – Elution – EW - smelting circuit allows a recovery of 99% Au and Ag.

17.0 MINERAL RESOURCE AND MINERAL RESERVE ESTIMATES

17.1 Introduction

This chapter describes the methodology and results related to the block model used in this present report. This work was developed in three stages:

In September 2009, NCL Ingenieria y Construccion S.A. of Chile (NCL) developed the geological model of the high grade veins and assisted Greystar in the block model construction and resource evaluation. Rodrigo Mello, a senior geologist from NCL, MAusIMM, led the team who, working together with Giovanny Ortiz, Exploration Manager from Greystar, MAusIMM, obtained a resource block model amenable for UG mining.

In 2010, Greystar updated a portion of this block model, to reflect new results from the Los Laches zone. Other zones were not modified.

In April 2011, Mr. Mello undertook a review of the block model used for the mine planning activities presented in this study, checking all items necessary to allow him to act as Qualified Person (QP) for this mineral resource evaluation.

This estimate used data available up to July 2010.

Besides reviewing the mineral resource estimation, the QP has audited the database and verified aspects of data quality and security, to assure that the resource estimate complies with the standards set by the Canadian norm NI 43.101, companion policy NI43-101CP and Form 43-101F1. Mr. Mello visited the Angostura site and the Greystar office in Bucaramanga, Colombia, from June 15th to 20th, 2,009 and from February 2nd to 5th, 2010. During these visits, Mr. Mello reviewed the exploration activities, audited the database and reviewed resource estimation prepared by Mr. Ortiz.

17.2 Software Used

The modelling and geostatistics analysis of the deposit was carried out using three different software packages: Gemcom 6 (geologic modelling), Datamine Studio 3 (kriging, block model construction and model validation) and GSLIB (variography and exploratory data analysis).

17.3 Database

The database used for this estimate is composed of over 300 km of diamond drilling. Differently than the previous estimate, no bulk sample was used, due to concerns with the support effect. The Table 17.3-1 depict the database details:

Table 17.3-1: Database basic statistics

Type	Nº of holes	Nº Meters drilled
DD	936	306,915

Element	Nº of assays	Nº of meters sampled
Au	181,406	304,854
Ag	180,384	303,221
Cu	180,383	303,220
S	142,693	254,868

17.4 3D Modelling

In total, 202 different veins were modelled, based on geological descriptions and a nominal grade of 2 g/t Au as a reference. The solids had restricted extension away from the drilling intersects, since the vein continuity is not well known yet. Normally solids are terminated 15 m away from the last drilling information, which is a conservative measure but necessary, at this stage of the work.

Two surfaces were constructed by Greystar and used for modeling:

Topographic surface, obtained from the 2008 topographic survey completed by Estudio-T Rural (Colombian Company).

Surface separating oxide + mixed material from fresh material

17.5 Oxidation State Level Model

The modeling of the three oxidation state levels at Angostura – oxide, transition and fresh (sulfide), is based on the characterization of each Angostura assay interval using the indicators in the Table 17.5-1:

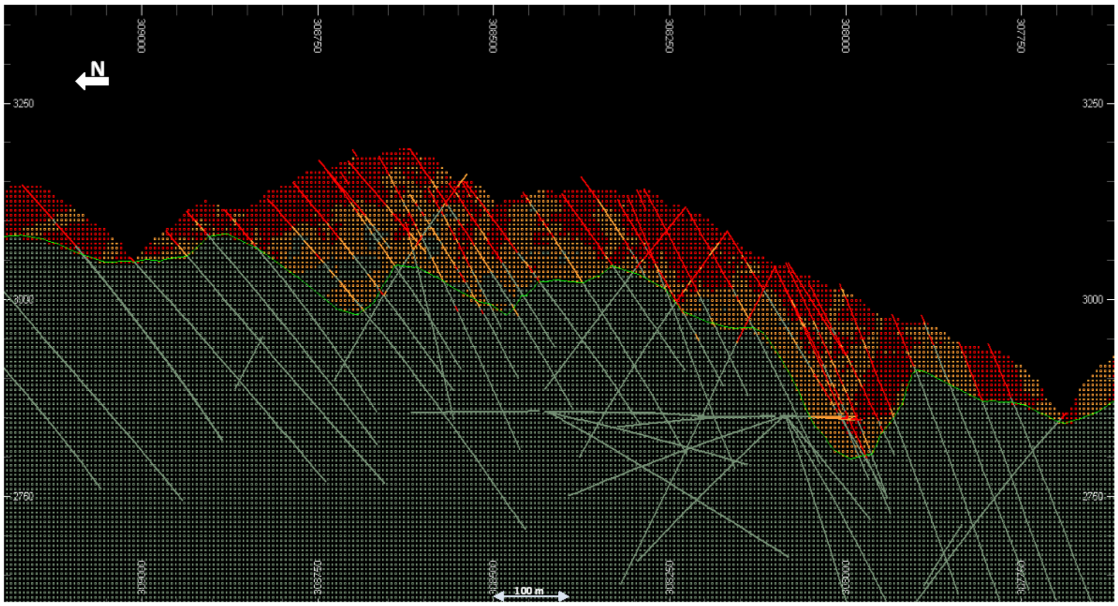
Table 17.5-1: Criteria to define the oxidation state level of core

Characterization as	Limonite	Total S (%)	Visual Sulfides
Oxide	Present	≤ 1%	Trace to ≤ 0.5%
Transition	Present	≥ 1%	≥ 1%
Fresh or Sulfides	Absent	Not a criterion	Present

The characterization was accomplished using core photographs, geological drill-hole logs and the total sulfur (Stotal) assay information available for most drill holes. Core pictures and sulfur assays are not available for drill holes completed before 2000, and these were re-logged for this characterization.

After the drill-hole intervals had been characterized as oxide, transitional or fresh in the Angostura block model database, the distribution of the three zones was modelled by Greystar staff for the entire deposit. In a first step, the top of the fresh zone, below which no oxidation is present, was correlated on section and translated into a digital terrain model (DTM) called the “Green Line”. Above the Green Line, all three zones are present due to the incomplete nature and variable intensity of the oxidation process. In a second step, their distribution above the Green Line (Figure 17.5-1) was interpolated using an indicator kriging (IK) approach that estimates the likelihood of a block to belong to one or more of the three different oxidation zones based on the surrounding drill-hole information. For each indicator a variogram was obtained and used to define the variogram model for the kriging and the search radii. The IK interpolation resulted in the assignment of oxidation-zone probabilities into each block above the Green Line. All blocks below the Green Line are automatically 100% fresh zone. As a result, each block in the deposit now has an estimate of the proportion of oxide, transition and fresh rock (the “Oxidation Composition”). The block model with the oxidation level identifier was superimposed to the high grade veins block model. Although this approach is acceptable at this point, future evaluation of the oxidation level inside the veins should be reviewed.

Figure 17.5-1: Oxidation state level. Vertical section



Note: Colour codes for oxidation state levels in blocks and drill holes: Red: Oxide; Orange: Transition; Grey: Sulfides or Fresh. Apparent local discrepancies between drill-hole information and block model are due to drill holes and blocks not being exactly in the same plane

Based on the combined geological and metallurgical evidence, the definition of the oxidation zones in terms of their oxide composition is as follows:

A block is designated as Oxide if the proportion of oxide rock is equal to or higher than 60% and the proportion of fresh rock is 10% or less, the highest such proportion in the column- test composite samples. In addition, the S_{total} grade of the block should be 1% or less (the highest S_{total} assay in the oxide-zone column test samples was 0.9%);

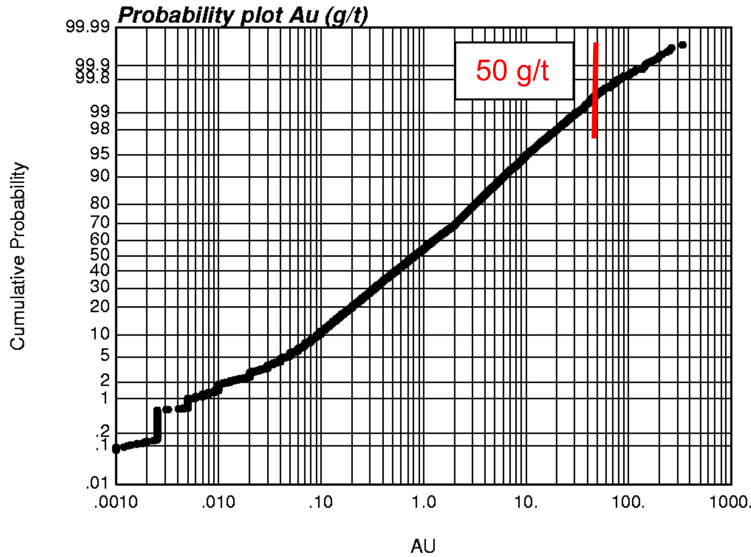
A block is designated as Fresh or Sulfide if the proportion of fresh rock is equal to or higher than 45%, and the proportion of oxide rock is 8% or less;

Any block not identified as Oxide or Fresh (Sulfide) in accordance with these provisions is automatically designated as Transition

17.6 Outlier Analysis

The Figure 17.6-1 is an example of the probability graphs that were used to define the threshold to cap the outliers of the studied population. This figure refers to gold. Silver, copper and sulfur had the same analysis.

Figure 17.6-1 Probability plot, for identification of outliers - Au



The objective is to limit the influence of very high values on the interpolation of grades. If the high values stay in the expected position (a straight line in the high end of the probability graph) they may be considered part of the population and used in the estimative. Otherwise, they may be capped, to have their value reduced to a selected threshold. Other factors are also considered, like the adherence of the kriging values to the moving average, the geology, etc.

Capping was applied to Au, Ag and Cu over the raw values, i.e, before compositing, to avoid the smearing of any spurious value in the process of compositing. Sulfur does not appear to possess outliers value. Capping values and the effect of compositing can be seen in the Table 17.6-1.

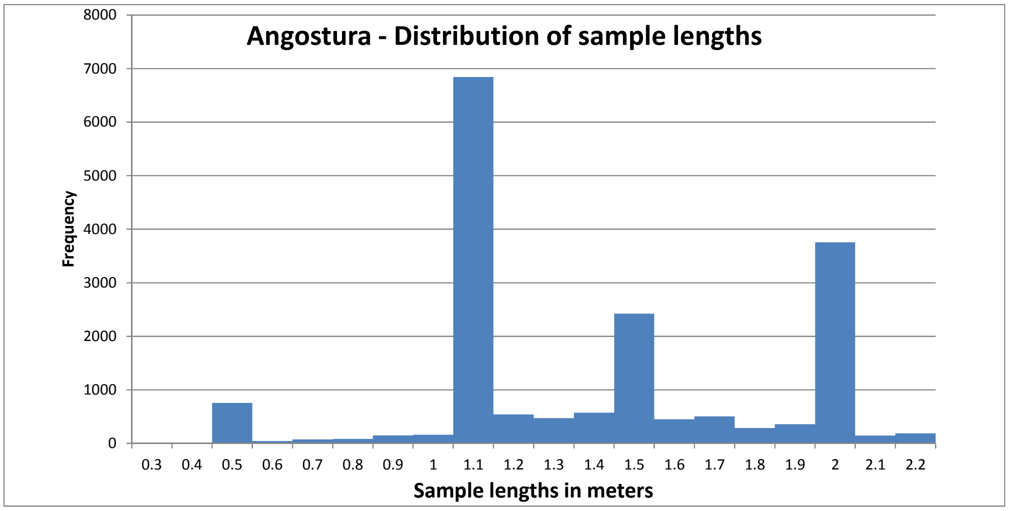
Table 17.6-1: Statistics of samples inside the veins, before and after capping

Element/Unit	Capping value	Raw mean	Capped mean	Decrease	Nr Samples capped	Percentile	Raw CV	Capped CV
Au g/t	40	2.79	2.53	9%	118	99.37	3.06	2.02
Ag g/t	500	13.53	12.97	4%	35	99.81	3.43	2.87
Cu %	4	0.06	0.06	2%	8	99.96	1.00	3.56
S %	no capping	3.57		100%	0	100.00	0.31	0.31

17.7 Compositing

Compositing, i.e. transforming the samples to a fixed length in order to have all values at a similar support, is a necessary step before interpolation of results. For the Los Laches area, 1.5 m length was utilized and for the remaining veins, 1.0 m was chose for compositing. These lengths were selected because they best represent the mode of the sample length for each population. Choosing this length for composition would preserve the detail obtained in the sampling, while still having a good statistical agreement between samples and composites. Figure 17.7-1 depicts the histogram of sample lengths, used to support this decision.

Figure 17.7-1 Distribution of sample lengths



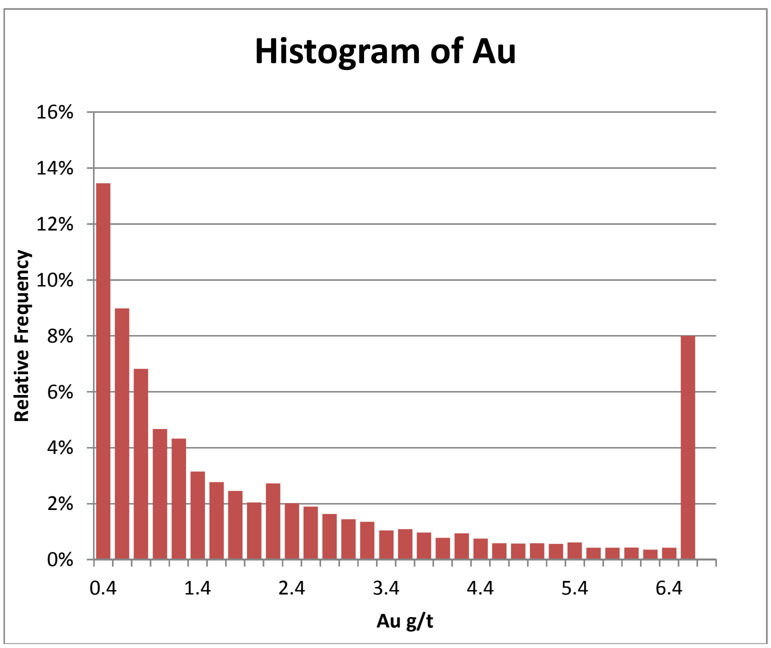
17.8 Exploratory Data Analysis

The Table 17.8-1 below depicts the basic statistics of the composites and for the samples and composites, contained within the geologic solids. The Figure 17.8-1 is the gold histogram for composites used in the estimation.

Table 17.8-1: Basic statistics of samples and composites

Samples Inside solid								
Element	*Number*	*Total length*	*Mean g/t*	*Std Dev*	*Var*	*CV*	*Min*	*Max*
Au	18,631	26,533	2.79	8.56	73	3.06	0.0001	336
Ag	18,616	26,533	13.53	46.48	2,160	3.43	0.0500	1,500
Cu	18,616	26,533	0.06	0.23	0.053	1.00	0.0001	9
S	13,345	26,533	3.57	3.27	10.662	0.31	0.0000	39
Samples Inside solid - After Capping								
Element	*Number*	*Total length*	*Mean g/t*	*Std Dev*	*Var*	*CV*	*Min*	*Max*
Au	18,631	26,533	2.58	5.52	30	2.13	0.000	50
Ag	18,616	26,533	12.97	37.16	1,381	2.87	0.000	500
Cu	18,616	26,533	0.06	0.21	0.045	3.56	0.000	4
S	13,345	26,533	3.57	3.27	10.66	0.31	0.00	38.50
Composites Inside solid								
Element	*Number*	*Total length*	*Mean g/t*	*Std Dev*	*Var*	*CV*	*Min*	*Max*
Au	25,568	25,833	2.29	4.48	20	1.96	0.000	50
Ag	25,555	25,833	10.93	27.49	756	2.51	0.050	400
Cu	25,555	25,833	0.10	0.88	0.766	9.17	0.000	18
S	19,967	25,833	3.26	2.90	8.420	0.89	0.000	34

Figure 18.8-1 Histogram of Au in composites



17.9 Population Analysis

Since 200 separate veins were modeled, it is unpractical to estimate each one separately. After several approaches studied, a simplistic division of the veins in four separate groups was adopted. The division was based on the dominant direction of the strike of each group of veins, except for the Los Laches veins, which were separated due to their higher grade characteristics and also due to the fact that they were estimated posterior to the other veins. Los Laches has the same attitude as the Group 1. All vein groups dips roughly to 85° to North, see Table17.9-1.

The Table 17-9-2 shows the Gold statistics by veins group.

Table 17.9 1: Vein Groups details

Group	Dominant direction	Azimuth strike
Los Laches	NE	50
1	NE	50
2	NW	106
3	WE	85

Table 17.9-2: Basic stats for each group

Au g/t							
ZONE	Number	Mean g/t	Std Dev	Var	CV	Min	Max
Los Laches	1,053	2.42	5.41	29	2.23	0.0025	50
Group 1	13,241	1.97	4.04	16	2.05	0.0019	40
Group 2	2,730	2.66	4.94	24	1.86	0.0005	40
Group 3	8,544	2.64	4.81	23	1.82	0.0001	40

17.10 Specific Gravity Measurements

The same density values as used in the last resource study were used in the present one. To reach those numbers, Greystar has undertaken more than 9,000 density measurements on drill core samples, selected according the lithology, alteration and mineralization. The method used is the wax immersion method in most cases.

A single value of bulk density was assigned to the blocks for each of the oxidation zones, based on the average of the density measurements and considering some safety factors, to adjust downward to account for expected rock porosity that is not reflected in the measurements for oxide and transition zones. The Figure 17.10-1 and Table 17.10-1 shows the statistics of the specific gravity measurements and the correction factors applied by oxidation zone to account for vugs and porosity in the different oxidation levels.

Figure 17.10-1: Summary Density Data

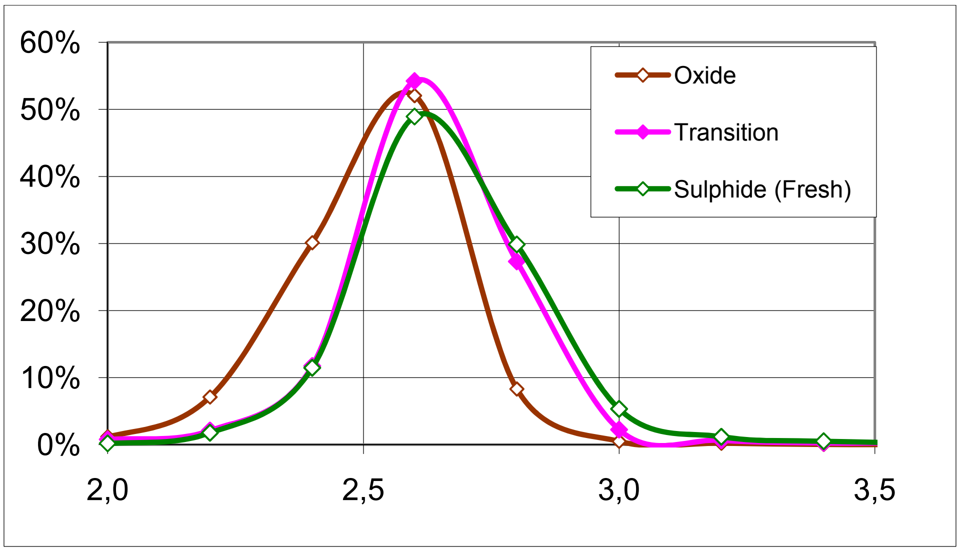


Table 17.10-3: Average Density by Oxidation level

	Specific Gravity (g/cm^3)	Correction Factor	Bulk Density (g/cm^3)
	Oxide		
Number of values	1,756	95%	**2.31**
Mean	2.43		
Standard deviation	0.16		
Median	2.45		
Outlier high	2.92		
Outlier low	1.94		
New mean	2.43		
New standard deviation	0.15		
	Transition		
Number of values	758	98%	**2.49**
Mean	2.54		
Standard deviation	0.18		
Median	2.55		
Outlier high	3.09		
Outlier low	2.00		
New mean	2.54		
New standard deviation	0.14		
	Sulfide (Fresh)		
Number of values	6,498	100%	**2.57**
Mean	2.57		
Standard deviation	0.19		
Median	2.56		
Outlier high	3.15		
Outlier low	1.99		
New mean	2.56		
New standard deviation	0.16		

17.11 Block Model Parameters

The block size used was 12 x 12 x 12 m, used to estimate grades. For volumetric purposes, sub-blocking was used, with a limit of 1 x 1 x 1 m minimum sub-block size. A block of this size would be adequate for mine planning. The parameters are as follows:

Table 17.11-1: Block model parameters

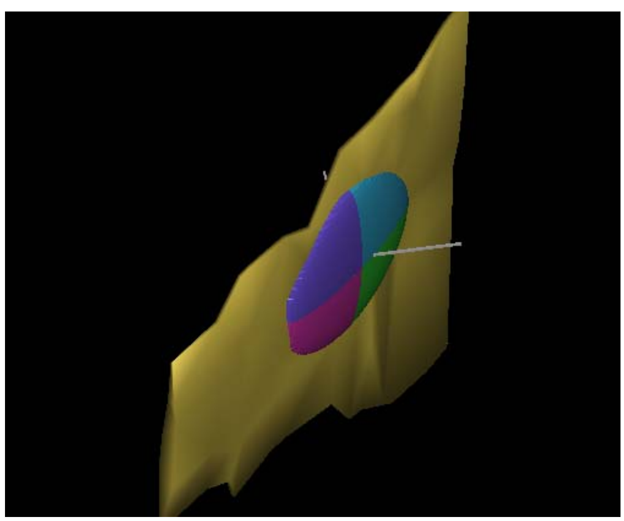
Angostura - UG model			
	X	Y	Z
Minimum Coordinates	130170	307400	2300
Maximum Coordinates	132330	310160	3692
No. blocks	180	230	116
User Block Size	12	12	12
Rotation	0		
Extension	2160	2760	1392

17.12 Variography

Two different types of software were used to carry out the anisotropy analysis, GSLIB and DATAMINE. Fan of variograms were studied for each of the three veins families established, analyzing the anisotropy in intervals of 22.5°, along the principal planes of mineralization: Veins NW (Family 1), 320°/-79°; Veins NW (Family 2), 16°/-85°; and Veins EW (Family 3), 355°/-78°.

Semi-variograms were used for variogram modeling. 1-metre composites were used for the calculation of the semi-variogram using populations by vein families. The semi-variograms for gold, silver, cooper and sulfur were obtained and a variographic model for each element and population was obtained. The Figure 17.12-1 shows an example of a search ellipse for semi-variogram construction.

Figure 17.12-1: Search Ellipse view



The variography parameters used in the kriging are listed below on Table 17.12-1.Table The nugget effect was obtained from the down the hole variogram using the 1.5 m composites.

Table 17.12-1: Variogram parameters

	Veins NE	Veins NW	Veins EW
Metal	**Au**	**Au**	**Au**
1st Azimuth	39.2	291	355
1st dip	-44	-45	-78
2nd Az	240	101	85
2nd dip	-44	-45	0
3rd Azimuth	230	196	175
3rd dip	-79	-5	-12
nugget	0.3	0.3	0.3
1st sill	0.55	0.2	0.2
range 1	80	100	100
range 2	40	40	120
range 3	15	70	90
2nd sill	0.15	0.5	0.5
range 1	2000	1000	450
range 2	400	80	220
range 3	38	2000	100

The Figure 17.12-2 presents the down the hole gold variograms (used to identify the nugget effect) and the variograms for the veins families, Figures 17.11-3, 4 and 5. The first in the direction with best continuity, and the third to the poorest. All of the variograms were calculated with a lag separation of 15 m, and using a tolerance on azimuth and dip of 22.5°. All models are spherical. The search ratios were established visually.

Figure 17.12.2: Down the hole variogram . 1.5 m composites (Nugget Effect, Gold).

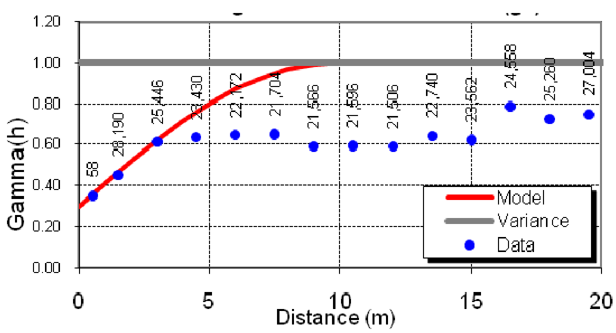


Figure 17.12.3: Variograms for gold calculated for 1.5 m composites, NE veins family.

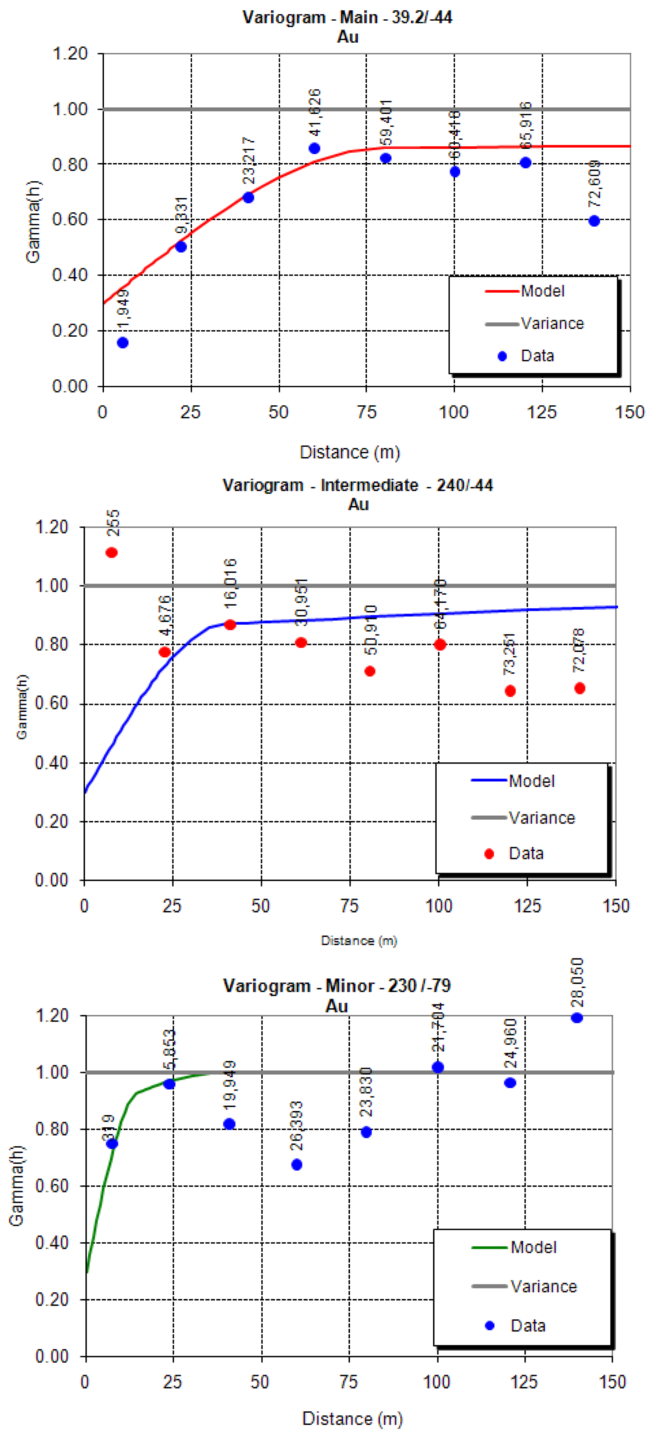


Figure 17.12.4: Variograms for gold calculated for 1.5 m composites, NW veins family.

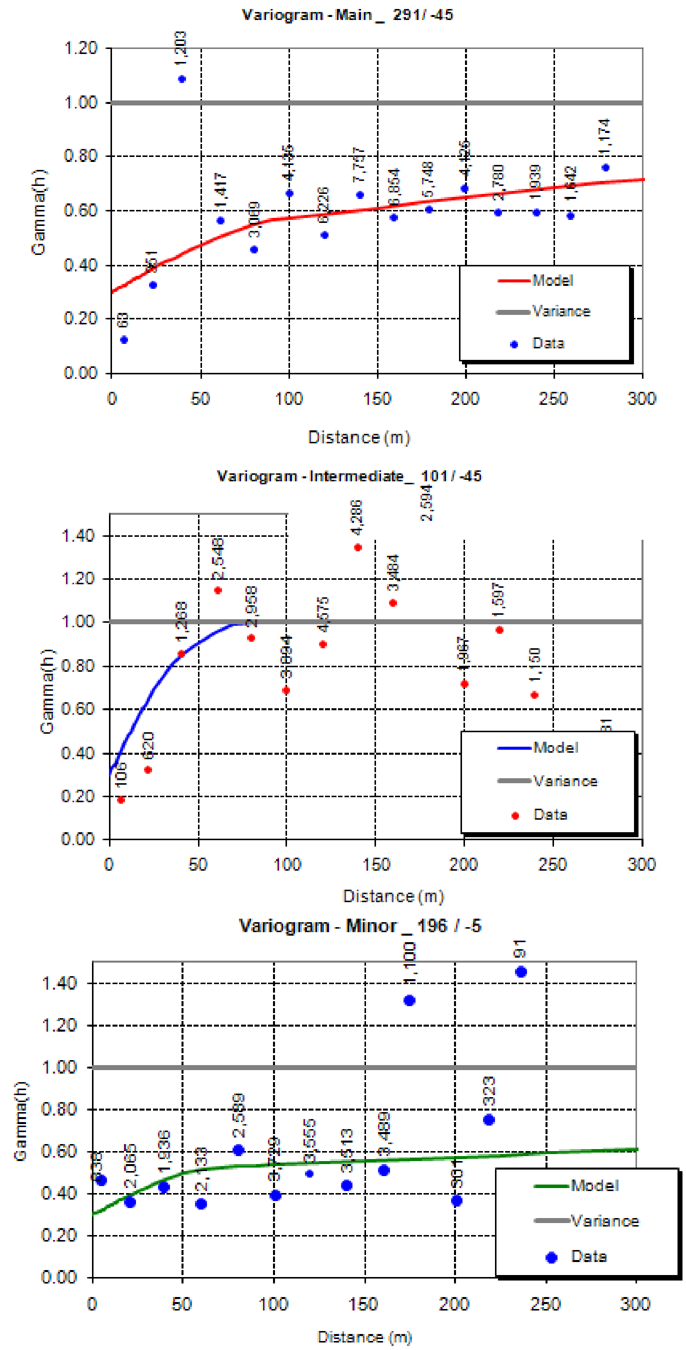
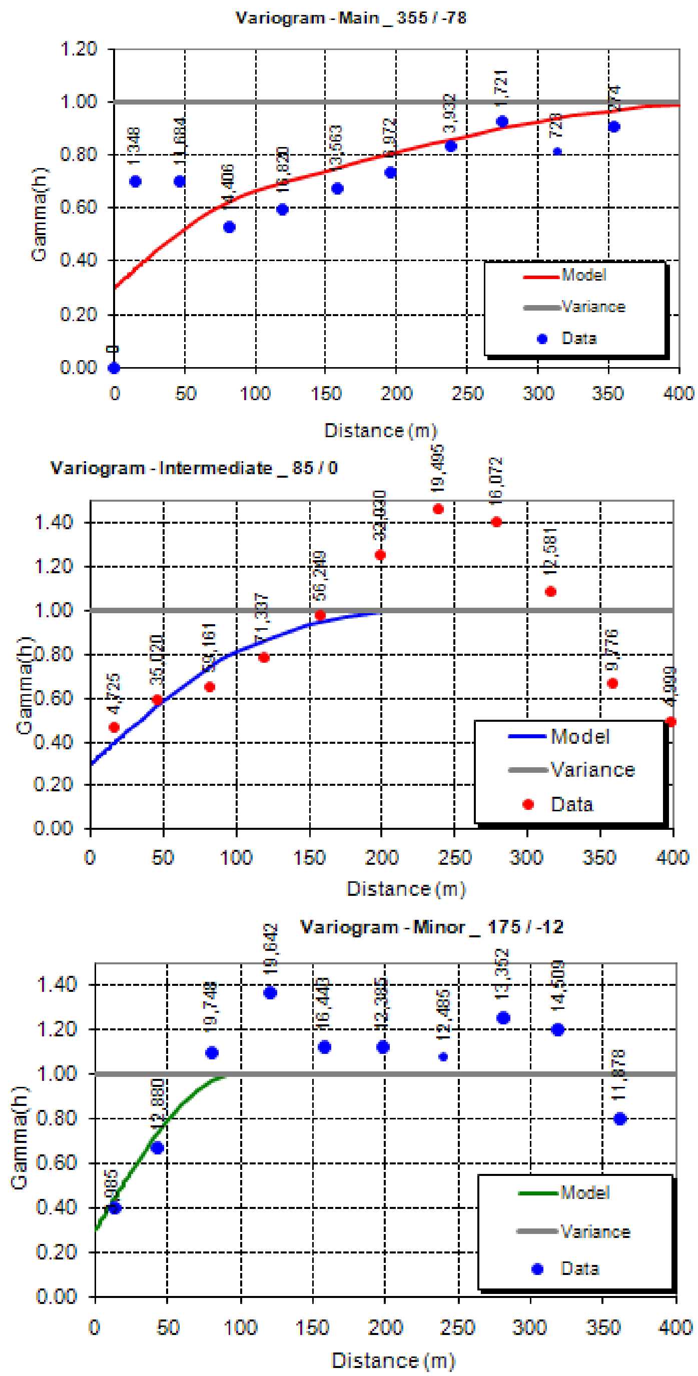


Figure 17.1.5: Variograms for gold calculated for 1.5 m composites, EW veins family.



Variography analysis were performed for silver, copper and sulfur for kriging interpolation.

17.13 Kriging Strategy

Ordinary kriging was used for gold, silver, copper and sulfur interpolation. Each vein was interpolated with its own vein composites.

Two passes were used, to successively interpolate grades with the parameter showed in the Table 17.13-1. These passes were used for grades interpolation and not for categorization of the resources.

Table 17.13-1: Kriging strategy for grade interpolation for veins

Kriging strategy (Gold - Silver)			
1st Search Ellipse			
	NE Veins	NW Veins	EW Veins
Strike	37.5	37.5	50
Down Dip	25	25	50
Cross Strike	20	25	25
Min. Nr. Drillholes	1	1	1
Min. Nr.	3	3	3
Max. Nr.	3	3	3
Max. Nr.	12	12	12
Nr of	4x4x4	4x4x4	4x4x4

Kriging strategy (Copper - Sufhur)			
1st Search Ellipse			
	NE Veins	NW Veins	EW Veins
Strike	37.5	37.5	50
Down Dip	25	25	50
Cross Strike	20	25	25
Min. Nr. Drillholes	1	1	1
Min. Nr.	4	4	4
Max. Nr.	-	-	-
Max. Nr.	48	48	48
Nr of	4x4x4	4x4x4	4x4x4

Note: Each vein was interpolated with an ellipse that follows its strike and dip

17.14 Veins Model Construction

The sequence of block model construction in the Studio 3 (Datamine) software is the following:

1. Construction of block model inside the vein wireframes using sub-cells of minimum 1x1x1 m and a maximum block size of 6x6x6 m, and a parent cell of 12x12x12 m. (WIREFILL command)

2. Extract the blocks above surface and outside of the mining property

3. Print the vein code from modeled veins solids

4. Kriging of the Au

5. Kriging of Ag, Cu and S grades, for oxidized and sulphides separately.

6. Print the oxidation code blocks according to the geological model

7. Classify the resources into indicated and inferred.

17.15 Resource Classification

The blocks interpolated with gold in the grade interpolation were categorized as inferred. For the definition of the indicated resources, a new interpolation was made using the following criteria, Table 17.15-1.

Table 17.15-1: Interpolation strategy for indicated categorization of resources

Interpolation Strategy	
Run 1	
Strike	37.5
Down Dip	37.5
Cross Strike	37.5
Min. Nr. Drillholes	2
Min. Nr. Composites	1
Max. Nr. Octants	2
Category	**Indicated**

According to the criteria defined, the classification methodology adopted by NCL follows:

- Indicated resources: blocks which have at least two different drillholes in the neighbourhood, considering a distance corresponding to the fist sphere in the Table 17.15-1. The rock code of these intercepts must be same as the block being classified.

- Inferred resources: The blocks estimated using the first and second search ellipses defined in the Table 17.13-1. A single drillhole is enough to estimate inferred resources.

17.16 Model Validation

To verify the results of the estimation, a set of checks were performed on the model for each area:

•Visual validation of grades and the classification, comparing with the drilling

•Comparison using the drift analysis: compare the average grade of composites and kriged values (Figure 17.16-1, Figure 17.16-2 and Figure 17.16-3) along the major axis of the deposit.

In all tests the models were considered consistent and robust.

Figure 17.16-1: Floating window along West-East.

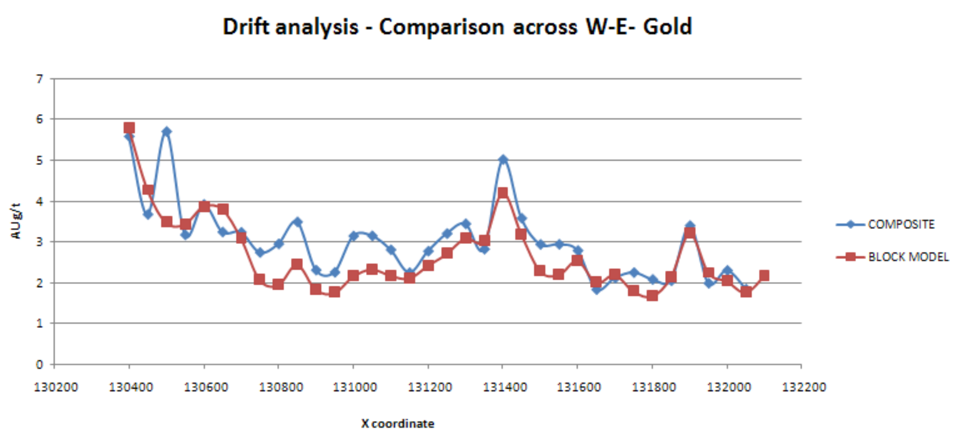


Figure 17.16-2: Floating window along South-North.

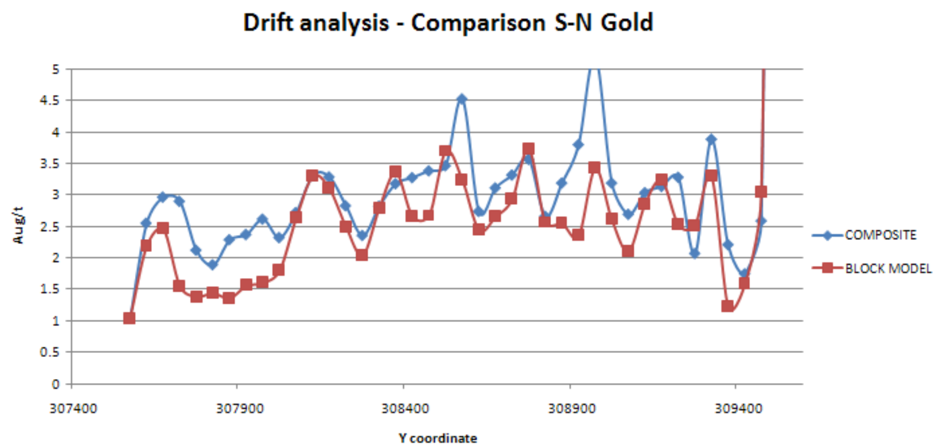
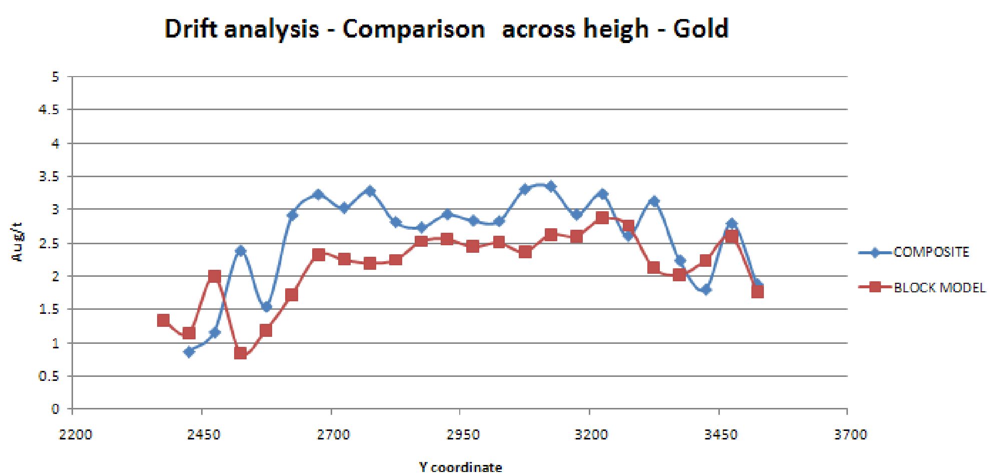


Figure 17.16-3: Floating window along levels (height).



17.17 Resource Reporting Criteria

The basic criteria followed in this estimation are as follows:

- Cut off grades of 1.5, 2.0, 2.5 and 3.0 g/t Au were used to report the mineral resources outside the stopes defined by the preliminary economic assessment for underground mining.

- The blocks of isolated veins (based on a visual analysis) and the blocks of veins with less than 1,000 Au ounces were not reported.

  Veins not reported are those identified with the following numbers: 2, 66, 128, 135, 137,158, 62, 63, 64, 111, 122, 128, 169 and 170.

- A crown pillar of 15 metres was used to limit the mineral resources in veins close to surface.

17.18 Results

The mineral resources for the veins of Angostura Project outside of the stopes defined in the PEA are tabulated in Tables 17.18-1 to 1.18-4, using different cut off grades, 1.5, 2.0, 2.5 and 3.0 g/t Au. The resources outside of the veins (Disseminated) were not evaluated and are not reported. .

Table 17.18-1: Mineral Resources, outside the stopes @ 1.5 g/t Au COG

	Ore (t)	Au (g/t)	Au Oz	Ag (g/t)	Cu (%)
			INDICATED		
Oxides	1,233,974	3.48	138,058	13	0.023
Transition	4,548,357	3.57	521,421	21	0.032
Sulfides	14,614,648	3.47	1,629,434	20	0.082
Sub-total	**20,396,979**	**3.49**	**2,288,913**	**20**	**0.067**
			INFERRED		
Oxides	761,366	3.62	88,572	15	0.027
Transition	1,411,480	4.18	189,471	17	0.050
Sulfides	10,224,700	3.69	1,212,061	23	0.093
Sub-total	**12,397,546**	**3.74**	**1,490,104**	**22**	**0.084**

Table 17.18-2: Mineral Resources, outside the stopes @ 2.0 g/t Au COG

	Ore (t)	Au (g/t)	Au Oz	Ag (g/t)	Cu (%)
			INDICATED		
Oxides	925,381	4.06	120,805	14	0.024
Transition	3,357,309	4.21	454,857	22	0.034
Sulfides	10,484,414	4.15	1,398,448	23	0.090
Sub-total	**14,767,105**	**4.16**	**1,974,111**	**22**	**0.073**
			INFERRED		
Oxides	581,949	4.20	78,548	16	0.028
Transition	1,103,422	4.86	172,324	18	0.052
Sulfides	7,378,145	4.43	1,051,960	28	0.099
Sub-total	**9,063,515**	**4.47**	**1,302,832**	**26**	**0.089**

Table 17.18-3: Mineral Resources, outside the stopes @ 2.5 g/t Au COG

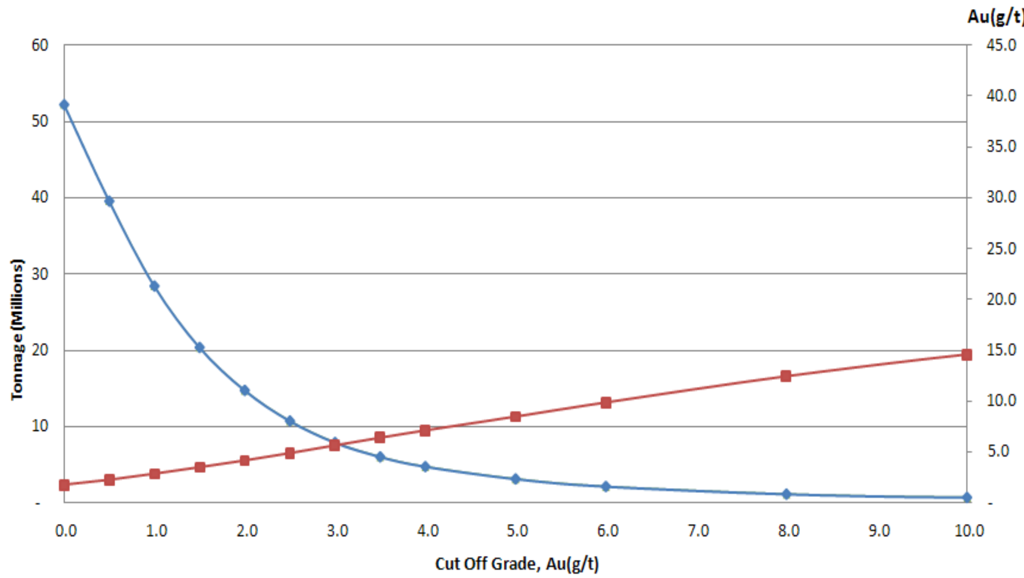
	Ore (t)	Au (g/t)	Au Oz	Ag (g/t)	Cu (%)
			INDICATED		
Oxides	681,946	4.70	103,152	14	0.024
Transition	2,455,658	4.94	389,967	23	0.035
Sulfides	7,616,340	4.87	1,192,721	27	0.096
Sub-total	**10,753,944**	**4.88**	**1,685,840**	**25**	**0.078**
			INFERRED		
Oxides	403,684	5.06	65,624	14	0.028
Transition	866,067	5.57	155,216	18	0.051
Sulfides	5,498,970	5.19	917,334	32	0.104
Sub-total	**6,768,721**	**5.23**	**1,138,174**	**29**	**0.092**

Table 17.18-4: Mineral Resources, outside the stopes @ 3.0 g/t Au COG

	Ore (t)	Au (g/t)	Au Oz	Ag (g/t)	Cu (%)
			INDICATED		
Oxides	499,214	5.43	87,198	15	0.025
Transition	1,783,624	5.77	330,880	24	0.037
Sulfides	5,642,124	5.62	1,019,271	31	0.102
Sub-total	**7,924,963**	**5.64**	**1,437,349**	**28**	**0.083**
			INFERRED		
Oxides	308,467	5.78	57,328	14	0.028
Transition	666,322	6.42	137,632	18	0.050
Sulfides	4,207,439	5.94	803,700	35	0.107
Sub-total	**5,182,227**	**5.99**	**998,661**	**32**	**0.095**

Figure 17.18-1 presents the tonnage – grade curve for the indicated resources in high grade veins, outside stopes.

Figure 17.18-1: Tonnage-Grade Curve for Indicated Resources outside stopes



The mineable resources inside the stopes defined in the Preliminary Economic Assessment (PEA), are considered Inferred and are tabulated in the Table 18.1-6 (Mineable resources per Oxidation level @ 3.0 g/t Au COG (diluted) (Includes Additional Stopes))

17.19 Comment on Section 17

The QPs are of the opinion that the Mineral Resources for the Project, which have been estimated using core drill data, have been performed to industry best practices, and conform to the requirements of CIM Definition Standards (2005).

Areas of uncertainty that may materially impact the Mineral Resource estimates include:

- Long-term commodity price assumptions
- Long-term exchange rate assumptions
- Constraints on the mine and/or process design as a result of designation of what constitutes "paramo", which in turn will affect the economic parameters used in the underground mine design that constrain Mineral Resources and Mineral Reserves.

18.0 ADDITIONAL REQUIREMENTS FOR TECHNICAL REPORT ON DEVELOPMENT PROPERTIES AND PRODUCTION PROPERTIES

18.1 Preliminary Mining Study

18.1.1 Introduction

This section presents a description of the work performed and the results of the Mine Scoping Study carried out by NCL to evaluate the potential of underground exploitation of the Angostura gold deposit, located in the Santander Province, Colombia.

The scope of work includes the following topics:

- Mineable resources.
- Preliminary Mine Design.
- Mine Plan.
- Materials Handling Analysis.
- Fleet Dimensioning.
- Services and Infrastructure.

The level of the work corresponds to a scoping study with a contingency level of 35%. All units in this report are metric, unless specified.

18.1.2 Definition of Case Scenario

Mining Methods

Considering the geometry and geotechnical conditions of the orebody, different mining methods were analysed for the underground exploitation of the Angostura deposit. According to the rock conditions presented, a geotechnical assessment was provided by the specialist consultants AKL S.A. (see Appendix A), whose recommendations for mining methods are:

- Veins with less than 5 m width = Bench and Fill Stoping
- Veins within 5 m and 20 m width = VCR (Vertical Crater Retreat)
- Veins within 20 m and 40 m width = Open Stoping

Backfill and support will be required in any of these options. The principal mining method will be Bench & Fill Stoping, considering:

- The distribution of the mineralization (sub-vertical veins, 1 to 30 m width, average of 5 m).
- Poor rock quality relative to the areas of interest.
- Experience in other mines with the same mineralization geometry.

The method consists of:

- Ramps that will allow for access to the veins.
- Along the vertical development of these ramps, a ventilation raise will provide fresh air to the operations and an ore pass will connect with the main transport level.
- Each 24 m (vertically measured), an access is designed allowing access to 2 drift levels (12 m each)
- Production starts with drifts, after accesses reach the vein, and continue in opposite directions.
- As soon as the last drift is completed (at the bottom of the stope), benching starts from the end of the drift towards the center
- Mucking will be completed with remote control 7 yd^3 LHD in the base drift
- Once all ore has been mined out, the stope will be filled from the upper drift with waste material coming from developments and/or surface.
- A new cycle is started when the stope is completely filled

The Figures 18.1-1, 18.1-2 and 18.1-3 illustrate these mining methods:

Figure 18.1-1: Bench & Fill

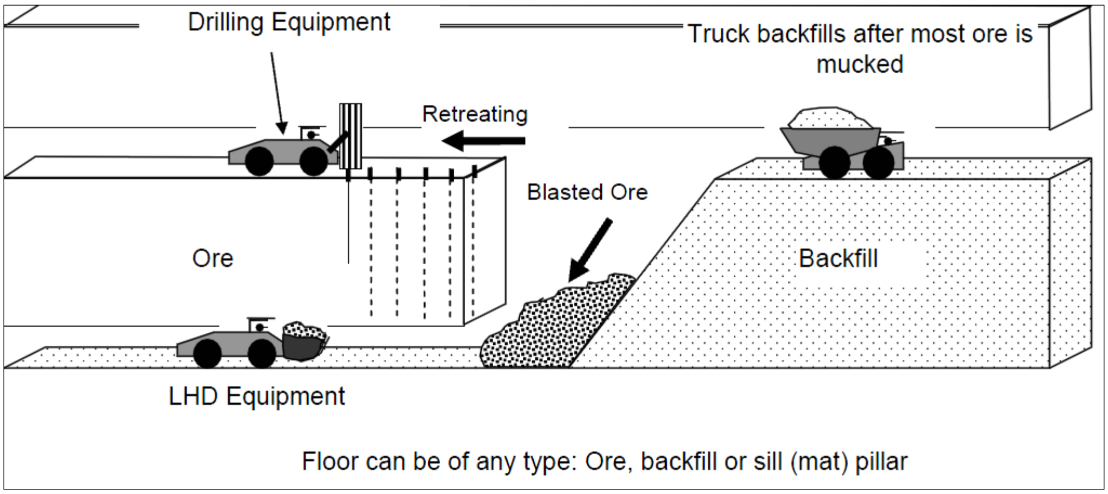


Figure 18.1-2: VCR

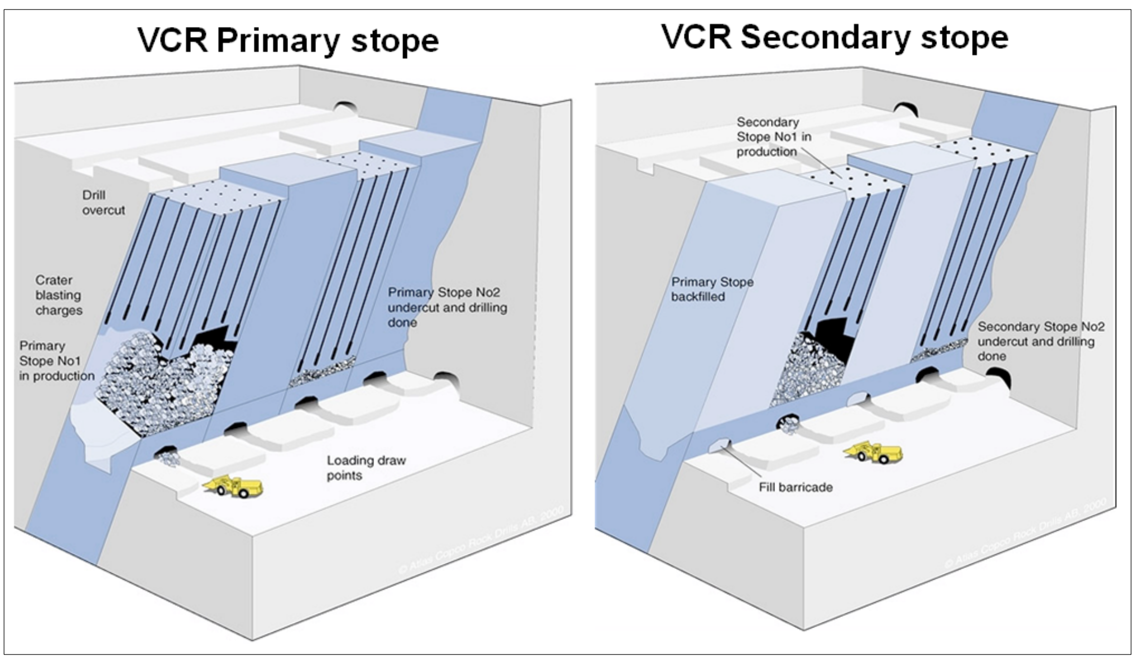
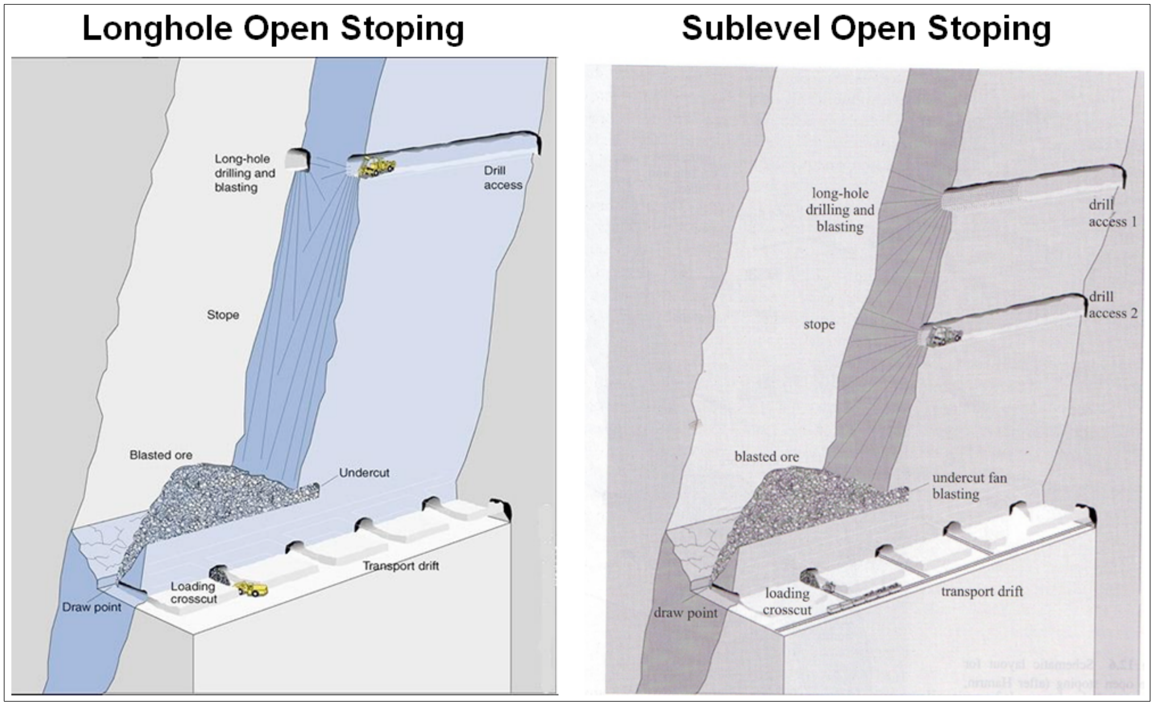


Figure 18.1-3: Open Stoping



Underground Mine Parameters

Dilution

Stopes were created from 20 m level contours. These polygons were tied between levels to delineate the corresponding solids representing the stopes. A minimum width of 2 m was applied for the construction of the solids.

Given the separation of the levels and the width of the veins, the delineation does not accurately follow the limits of the high grade veins, incorporating dilution to the content of the generated solids. For this reason, no additional dilution factors have been applied to the calculation of the mineable resources.

Cut-off grade (cog)

The following technical and economic parameters were used for the definition of the cut- off grade that corresponds to the main mining method considered, Table 18.1-1.

Table 18.1-1: Cut-off grade parameters

Total Mine Cost (Production & Maintenance)	40.0	US$/t
Process Cost	20.0	US$/t
G&A	10.0	US$/t
Selling	10.0	US$/oz Au
Recovery Au	85	%
Au Price	850.0	US$/oz

The assumed mining costs correspond to a bench and fill option and have been estimated from the basis of similar operations known to NCL.

The resultant cut off grade values is 3.0 g/t Au.

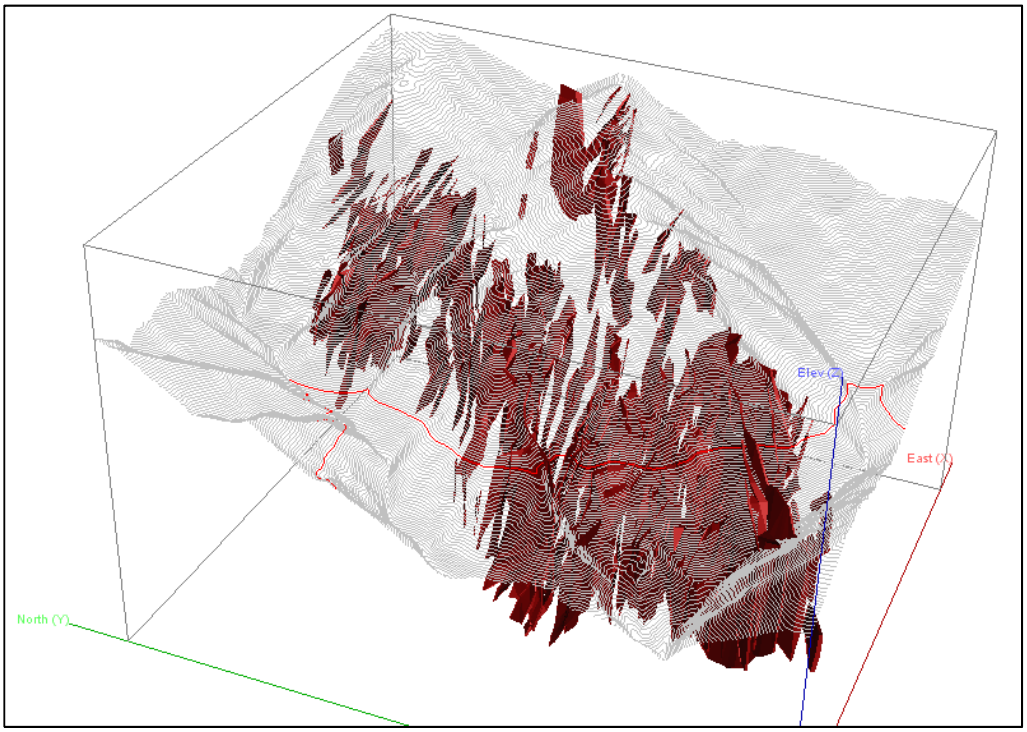
18.1.3 Economic Underground Mineable Resources

Input Data

The primary input data provided by Greystar for an estimate of the economic underground mineable resources was:

- Ore resources block model based on wireframes for the definition of the high grade zones (this model is different to the one used for the open pit DFS).
- Vein wireframes, Figure 18.1-4

Figure 18.1-4: Veins wireframes



Applying the defined cut-off grade to the model, the following resources available for mining were established, Table 18.1-2 .

Table 18.1-2: Resources – Selected veins @ 3.0 g/t Au COG.

Resources	16.06	Ore (Mt)
	6.33	g/t Au
	19.16	g/t Ag

Mineable Resources

Mineable resources were determined from the selected veins by generating a contour at 3.0 g/t Au cut-off grade. These contours were created from plan views at 20 m.

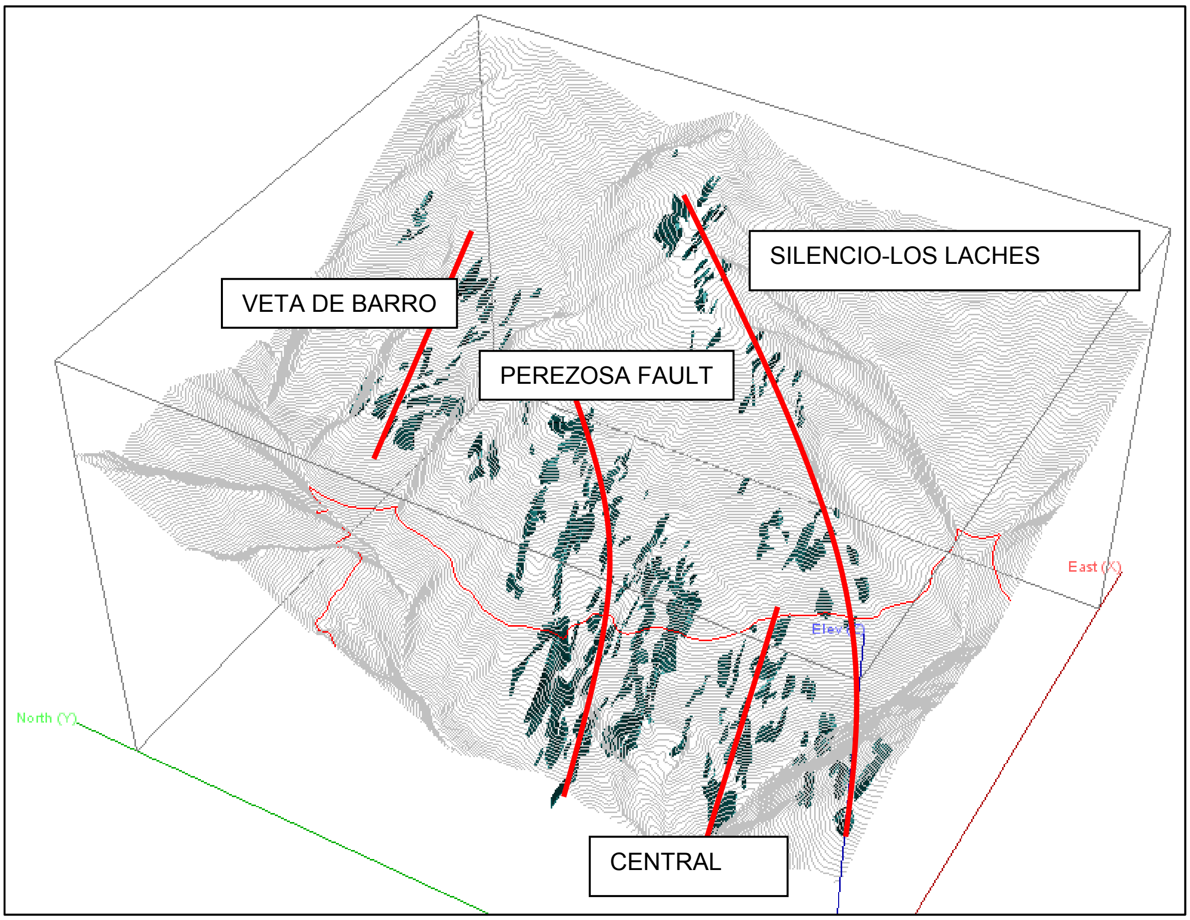
Four main sectors were identified:

- Veta de Barro.
- Central Area.
- Perezosa Fault.

- Silencio-Los Laches.

The Figure 18.1-5 shows these sectors and resulting mineable stopes.

Figure 18.1-5: Mineable Stopes - created from 20m contours



Final stopes were evaluated against the resource block model. These results are considered diluted (as explained in 2.2.2). Solid generation was based on 20 m contours and veins are sub-vertical, therefore, there is an inherent dilution associated. In addition to this, contours were created with a minimum width of 2 m, which will include dilution for veins narrower than 2 m.

Table 18.1-3: Mineable resources @ 3.0 g/t Au COG (diluted)

Sector	Ore (Mt)	Au (g/t)	Ag (g/t)	Cu (%)	S (%)
Veta de Barro	1.65	5.432	15.545	0.036	2.702
Central Area	0.97	4.742	18.031	0.141	3.221
Perezosa Fault	7,21	5.486	14.841	0.087	3.866
Silencio-Los	3,60	5.801	25.073	0.058	3.697
Total	**13,43**	**5.510**	**17.901**	**0.077**	**3.631**

Oxidation level is not included in the wireframe block model. To estimate how oxidation is distributed in these resources, the open pit block model was used as the data in this block model includes this variable, and oxidation level applied to the wireframe model. The resulting oxidation level by sector is shown in following Table 18.1-4.

Table 18.1-4: Mineable resources per Oxidation level @ 3.0 g/t Au COG (diluted)

	Ore (Mt)	Au (g/t)	Ag (g/t)	Cu (%)
Oxide	**0.62**	**5.746**	**18.514**	**0.0265**
VETA DE BARRO	0.14	5.836	8.287	0.0088
CENTRAL	0.11	6.000	18.662	0.0481
PEREZOSA FAULT	0.23	6.028	11.302	0.0224
SILENCIO-LOS LACHES	0.13	4.928	42.477	0.0347
Mixed	**2.29**	**5.676**	**21.990**	**0.0433**
VETA DE BARRO	0.44	6.200	15.984	0.0247
CENTRAL	0.17	3.975	25.204	0.0904
PEREZOSA FAULT	0.95	5.507	13.029	0.0533
SILENCIO-LOS LACHES	0.74	5.987	36.298	0.0302
Sulfur	**10.52**	**5.460**	**16.974**	**0.0871**
VETA DE BARRO	1.07	5.065	16.345	0.0438
CENTRAL	0.69	4.731	16.103	0.1691
PEREZOSA FAULT	6.03	5.462	15.260	0.0944
SILENCIO-LOS LACHES	2.73	5.793	21.223	0.0672
Total	**13.43**	**5.510**	**17.901**	**0.0768**

Additional Resources

The block model was reviewed by GSL during the development of this work, resulting in the addition of 556 kt of sulfides in the area of Silencio – Los Laches, with the detail shown in the Table 18.1-5.

The stopes added have a low gold grade and are mainly justified by silver grade. Copper values are also high compared to the average of the other stopes.

Table 18.1-5: Additional Ore at Silencio – Los Laches

Additional ore at Silencio - Los Laches, COG>3g/t							
Vein	Ore (Mt)	Au (Oz)	Ag (Oz)	Cu (%)	S (%)	Au (g/t)	Ag (g/t)
1	0.54	26,171	5,612,970	0.4552	6.94	1.50	321.87
201	0.03	4,423	25,521	0.0312	3.22	4.68	26.99
202	0.02	3,384	839	0.0061	2.88	4.75	1.18
Total	0.59	33,978	5,639,329	0.4443	6.80	1.47	312.38

With the addition of these stopes the total mineable resources per oxidation level results as follows, Table 18.1-6.

Table 18.1-6: Mineable resources per Oxidation level @ 3.0 g/t Au COG (diluted) (Includes Additional Stopes)

	Ore (Mt)	Au (g/t)	Ag (g/t)	Cu (%)
Oxide	**0.62**	**5.746**	**18.514**	**0.027**
VETA DE BARRO	0.14	5.836	8.287	0.009
CENTRAL	0.11	6.000	18.662	0.048
PEREZOSA FAULT	0.23	6.028	11.302	0.022
SILENCIO-LOS LACHES	0.13	4.928	42.477	0.035
Mixed	**2.29**	**5.676**	**21.990**	**0.043**
VETA DE BARRO	0.44	6.200	15.984	0.025
CENTRAL	0.17	3.975	25.204	0.090
PEREZOSA FAULT	0.95	5.507	13.029	0.053
SILENCIO-LOS LACHES	0.74	5.987	36.298	0.030
Sulfide	**11.08**	**5.260**	**31.806**	**0.105**
VETA DE BARRO	1.07	5.065	16.345	0.044
CENTRAL	0.69	4.731	16.103	0.169
PEREZOSA FAULT	6.03	5.462	15.260	0.094
SILENCIO-LOS LACHES	3.29	5.062	70.463	0.131
Total	**13.98**	**5.349**	**29.612**	**0.091**

The reader is cautioned that the underground mining study is a preliminary assessment and it includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them *that would enable them to be categorized as mineral reserves. There is no certainty that the preliminary assessment will be realized. No Mineral Reserves have been estimated.*

18.1.4 Mine Layout

Mine layout was designed considering the following restrictions and criteria:

- Avoid surface accesses and roads above 3,000 m level.
- Main transport levels should connect the different sectors of production.
- Portals for access to the main transport levels located below 3,000 m level.
- If possible, use accesses from surface avoiding development of long internal ramps.
- Ore passes will take the ore to the transport level, and ventilation shafts will provide fresh air; for every ramp created. The ventilation shafts will be equipped with fans (range between 200 to 300 thousand cfm) that have been sized according to the requirements of the mine.

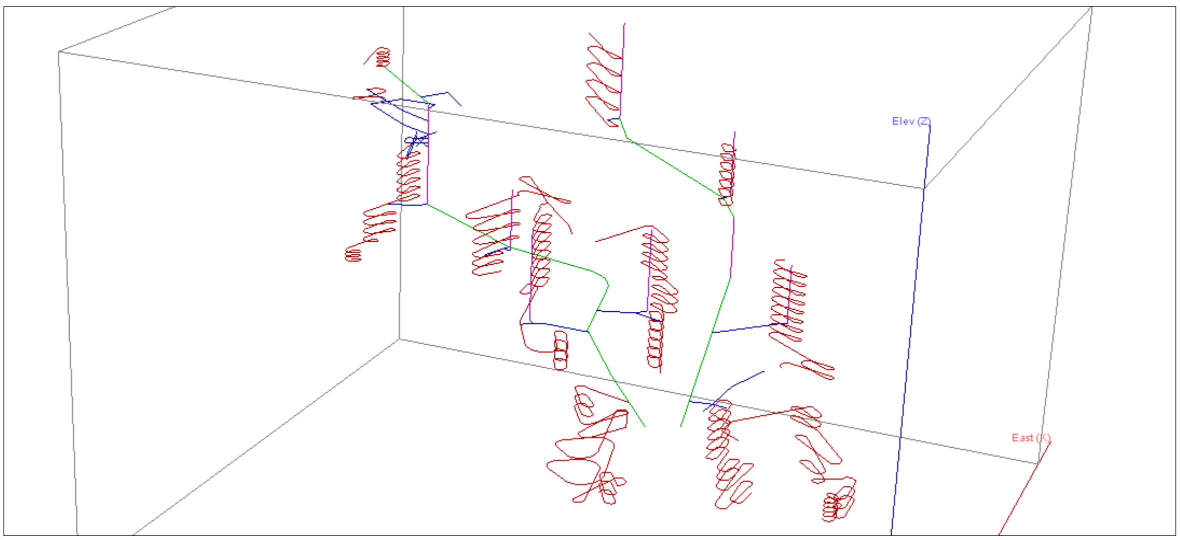
The main characteristics of the mine layout defined are:

- A minimum of 6 portals from topographic surface are needed to develop ramps for access to all levels and sectors where the mineral is located.
- Mineral will be taken from production levels to a transport level via ore passes.
- As mineral will be brought to surface via trucks, two main transport adits were designed. One will take mineral from ore passes of the northern sectors, while the other will collect from the ore passes of the south-east sectors. These two galleries portals are located at 2,844 m level.
- All material below 2,844 m level is to be accessed by ramps from developments above this level, as any surface access is not possible due to owner's requirement or limit impact in any areas which may be potentially considered as Paramo.

Process plant should be close to transport portals and below 3,000 m. Best location is approximately 300 m (straight line) north-west of the portals, as this is within owner's property, has the lowest topographic slope and enough open area for plant facilities.

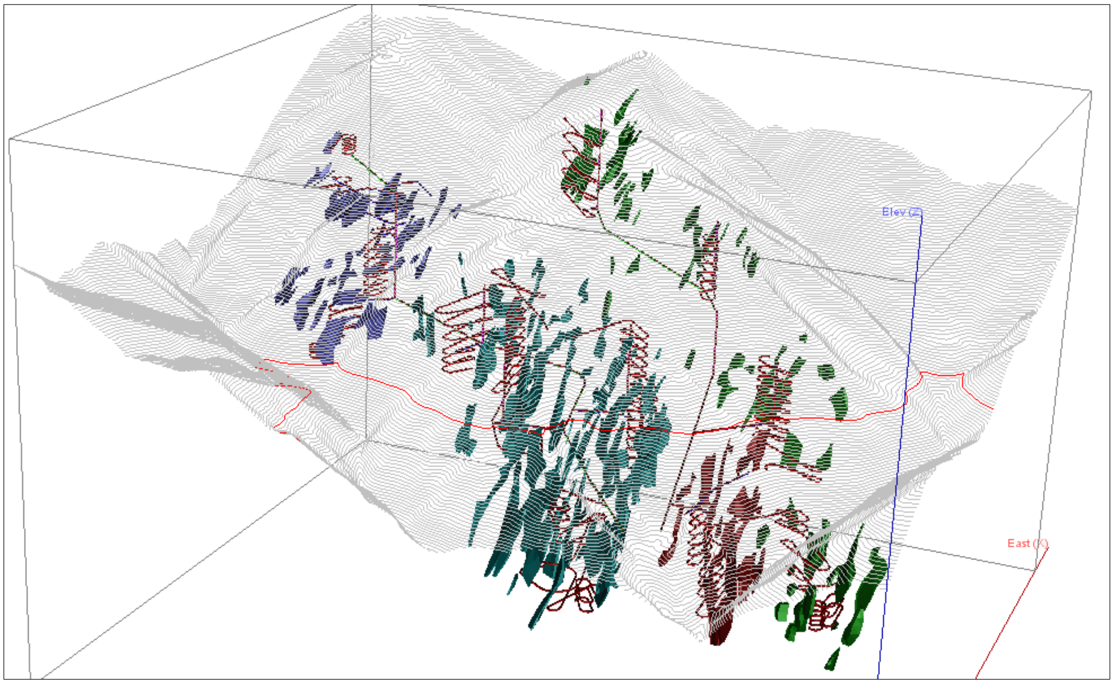
The Figure 18.1-6 illustrates the final mine design. The green lines identify the transport levels, red lines the ramps, magenta lines ore passes and blue lines for main accesses.

Figure 18.1-6: Mine layout 3D view (Ramps, Transport & Ore Passes)



A general view is presented below, Figure 18.1-7, including topography, stopes and mine design.

Figure 18.1-7: General mine 3D view.



The total development work estimated from the mine design is presented in the Table 18.1-7.

Table 18.1-7: Development requirements

Development		
Horizontals	**69,513**	**m**
Transport	3,280	m
Accesses	3,493	m
Ramps	30,111	m
Preparations	32,630	m
Verticals	**5,112**	**m**
Ore Passes	1,673	m
Ventilation Rises	3,438	m
Total	**74,625**	**m**

18.1.5 Mine Schedule

Development Plan

The horizontal development capacity has been estimated based on cycles of 1 blast per day with 3 m effective advance per gallery, for 5x4 and 4x4 m sections. All development was measured from drawings and scheduled in a logical sequence. A general summary of the development sequence is shown in the Table 18.1-8:

Table 18.1-8: Proposed Development Schedule

		Year							
Development	**Length (m)**	**-2**	**-1**	**1**	**2**	**3**	**4**	**5**	**6**
Horizontal	**69,513**								
Accesses	**3,493**								
A01	448		1	447					
A02	352				352				
A03	240		240						
Preparation	**32,630**								
L24	650			111	132	132	132	132	12
L43	4,030				1,315	1,692	1,023		
L45	3,770			165	278	277	277	277	278
Transport	**3,280**								
L21	1,412	666	746						
L23	358		358						
Ramps	**30,111**								
L24	665	408	256						
L39	2,814		315	783	785	783	148		
Vertical	5,112								
Ore Passes	**1,673**								
L22	280		280						
L42	168			168					
L47	216				179	37			
Ventilation	**3,438**								
L24	98	42	56						
L29	86			50	36				
L38	338		86	91	92	69			
L45	383				5	83	83	83	83

Development sequences to access all stopes were determined. The resulting development plan is presented in Table 18.1-9.

Table 18.1-9: Development plan

Year	Horizontal					Vertical		
	Acc.	Prep.	Transp.	Ramps	Total	OP	Vent.	Total
Pre-Production Development								
-2	709		666	408	1,783		42	42
-1	736		1,103	1,305	3,144	280	174	453
Production starts								
1	522	1,041		2,473	4,036	408	286	694
2	352	2,967	125	3,419	6,863	179	414	593
3	150	3,671	679	5,102	9,602	37	567	604
4	260	3,337		4,061	7,658		454	454
5		1,849	706	3,595	6,150	169	404	574
6		2,434		3,793	6,227	268	426	695
7	420	2,804		2,346	5,570	332	267	599
8	345	2,996		1,566	4,906		174	174
9		2,775		1,566	4,341		174	174
10		2,543		477	3,020		55	55
11		2,466			2,466			
12		2,101			2,101			
13		1,180			1,180			
14		466			466			
Total	3,493	32,630	3,280	30,111	69,513	1,673	3,438	5,112

Production Plan

The production plan was prepared for each stope independently, and then integrated into a global plan to establish the maximum production capacity for the underground mine. Stopes were divided vertically, to allow maximum productivity. Table 18.1-10 shows the identification for each of the stopes generated.

The production plan was prepared estimating productivities per area involved in a sector. Productivity estimation is a function of the stopes width and the mining method applied to the area. A general assumption of a bench and fill method has been made in order to estimate the productivities. The list of these areas was identified as follows, Table 18.1-10:

Table 18.1-10: Area/Sector Productivity Estimates

Area	Ore (t)	Au (g/t)	Ag (g/t)	Cu (%)	t/month
B01	128,727	3.70	28.60	0.02	2500
B02	184,844	6.65	7.60	0.01	4500
B03	411,452	4.48	15.74	0.03	10000
B04	488,744	5.80	11.93	0.04	8000
B05	111,514	6.38	15.37	0.04	3000
B06	202,119	5.55	15.62	0.08	8000
B07	122,672	6.10	27.62	0.04	4000
Veta de Barro	**1,650,071**	**5.43**	**15.55**	**0.04**	
C01	246,683	4.34	8.93	0.09	3000
C02	223,189	5.57	21.40	0.11	10000
C03	207,363	4.31	17.60	0.28	7500
C04	295,189	4.76	23.39	0.12	6300
Central	**972,424**	**4.74**	**18.03**	**0.14**	
P01	1,147,951	4.56	2.05	0.02	25000
P02	776,812	5.10	8.87	0.09	6500
P03	1,527,539	5.55	22.83	0.11	6000
P04	463,072	4.80	9.25	0.10	6000
P05	580,205	6.83	19.42	0.06	6300
P06	398,069	7.56	11.44	0.02	16000
P07	993,950	4.58	19.13	0.17	8500
P08	1,319,465	6.16	17.99	0.08	4500
Perezosa Fault	**7,207,064**	**5.49**	**14.84**	**0.09**	
S01	834,540	9.23	28.13	0.03	40000
S02	576,626	4.61	41.49	0.03	9000
S03	399,832	5.55	52.39	0.04	5850
S04	356,978	5.62	13.68	0.09	6400
S05	958,440	3.76	8.94	0.09	12000
S06	471,521	5.70	17.84	0.07	5040
S additional	550,000	1.47	312.38	0.44	
Silencio-Los Laches	**4,147,937**	**5.23**	**63.17**	**0.11**	

The Tables 18.1-11 to 18.1-16 and graph (Figure 18.1-8) present the resulting production plan for the mine and per sector.

Table 18.1-11: Production plan - with Oxidation Level Distribution

Year	Ore (t)				Au (g/t)	Ag (g/t)	Cu (%)	S (%)
	Oxides	**Mix**	**Sulfides**	**Total**				
1	100,645	192,112	8,343	301,100	5.51	17.33	0.05	2.78
2	132,929	393,498	369,673	896,100	5.62	13.25	0.05	3.13
3	108,012	250,707	907,053	1,265,772	5.50	12.30	0.05	3.27
4	64,090	260,972	914,821	1,239,883	5.24	12.56	0.07	3.44
5	75,600	214,376	930,951	1,220,927	5.11	13.38	0.09	3.56
6	5,076	258,247	944,532	1,207,855	6.38	19.03	0.09	3.92
7	19,795	204,360	1,021,299	1,245,454	6.76	22.86	0.07	3.91
8	110,178	170,970	938,053	1,219,201	4.55	68.50	0.14	3.96
9		234,581	932,548	1,167,129	3.92	108.59	0.19	4.57
10		82,852	718,628	801,480	4.94	21.41	0.09	3.63
11		28,617	740,546	769,163	4.96	21.93	0.09	3.67
12			648,846	648,846	4.98	20.74	0.09	3.74
13			373,462	373,462	4.97	19.37	0.09	3.99
14			292,920	292,920	5.04	15.86	0.09	4.12
>14			1,328,206	1,328,206	5.89	19.98	0.09	4.13
Total	**616,325**	**2,291,292**	**11,069,881**	**13,977,498**	**5.35**	**29.49**	**0.09**	**3.76**

Table 18.1-12: Ore Production per day (tpd)

Year	Tpd			
	Oxides	Mix	Sulfides	Total
1	335	640	28	1,004
2	443	1,312	1,232	2,987
3	360	836	3,024	4,219
4	214	870	3,049	4,133
5	252	715	3,103	4,070
6	17	861	3,148	4,026
7	66	681	3,404	4,152
8	367	570	3,127	4,064
9		782	3,108	3,890
10		276	2,395	2,672
11		95	2,468	2,564
12			2,163	2,163
13			1,245	1,245
14			976	976

Figure 18.1-8: Mine production plan (Oxidation levels)

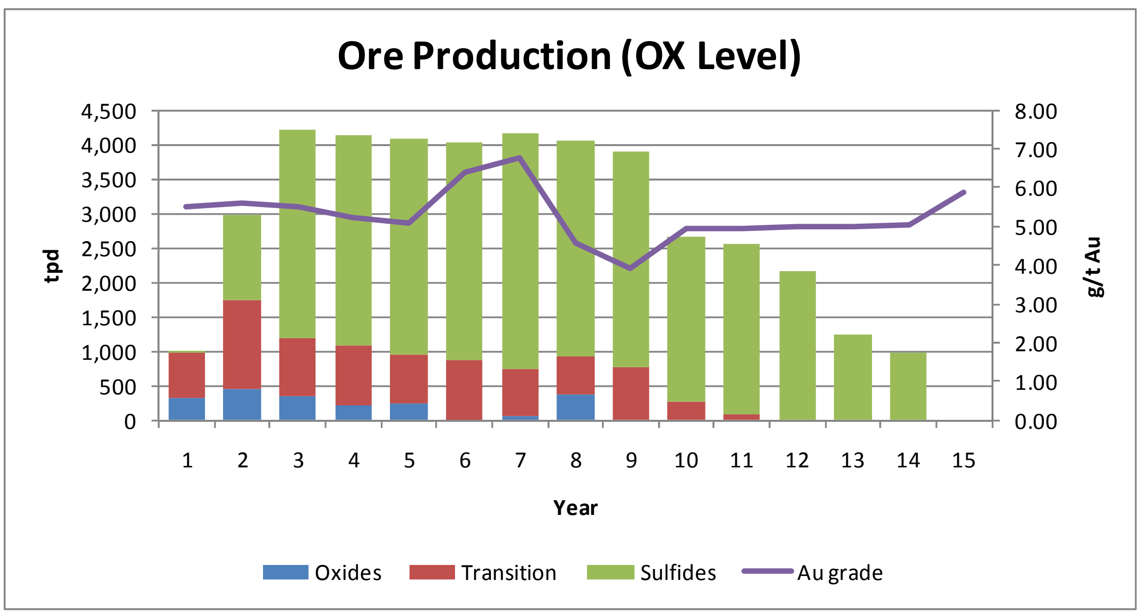


Table 18.1-13: Veta de Barro Sector production plan

Veta de Barro						
Year	**Ore (t)**	**Tpd**	**Au (g/t)**	**Ag (g/t)**	**Cu (%)**	**S (%)**
1	185,000	617	5.299	16.893	0.026	2.143
2	351,000	1,170	5.278	17.175	0.039	2.471
3	382,672	1,276	5.370	16.984	0.040	2.535
4	288,414	961	5.254	15.116	0.031	2.827
5	145,727	486	5.743	14.349	0.036	3.139
6	96,514	322	5.800	11.946	0.036	3.205
7	96,000	320	5.797	11.928	0.036	3.205
8	96,000	320	5.797	11.928	0.036	3.205
9	8,744	29	5.797	11.928	0.036	3.205
10						
11						
12						
13						
14						
>14						
Total	**1,650,071**		**5.432**	**15.545**	**0.036**	**2.702**

Table 18.1-14: Central Sector production plan

Central						
Year	**Ore (t)**	**Tpd**	**Au (g/t)**	**Ag (g/t)**	**Cu (%)**	**S (%)**
1						
2						
3						
4	135,800	453	5.077	18.385	0.131	3.646
5	321,600	1,072	4.888	19.409	0.154	3.581
6	224,789	749	4.594	18.550	0.175	3.573
7	116,463	388	4.612	18.679	0.113	2.246
8	98,089	327	4.606	18.084	0.105	2.153
9	36,000	120	4.336	8.934	0.088	2.770
10	36,000	120	4.336	8.934	0.088	2.770
11	3,683	12	4.336	8.934	0.088	2.770
12						
13						
14						
>14						
Total	**972,424**		**4.742**	**18.031**	**0.141**	**3.221**

Table 18.1-15: Perezosa Fault Sector production plan

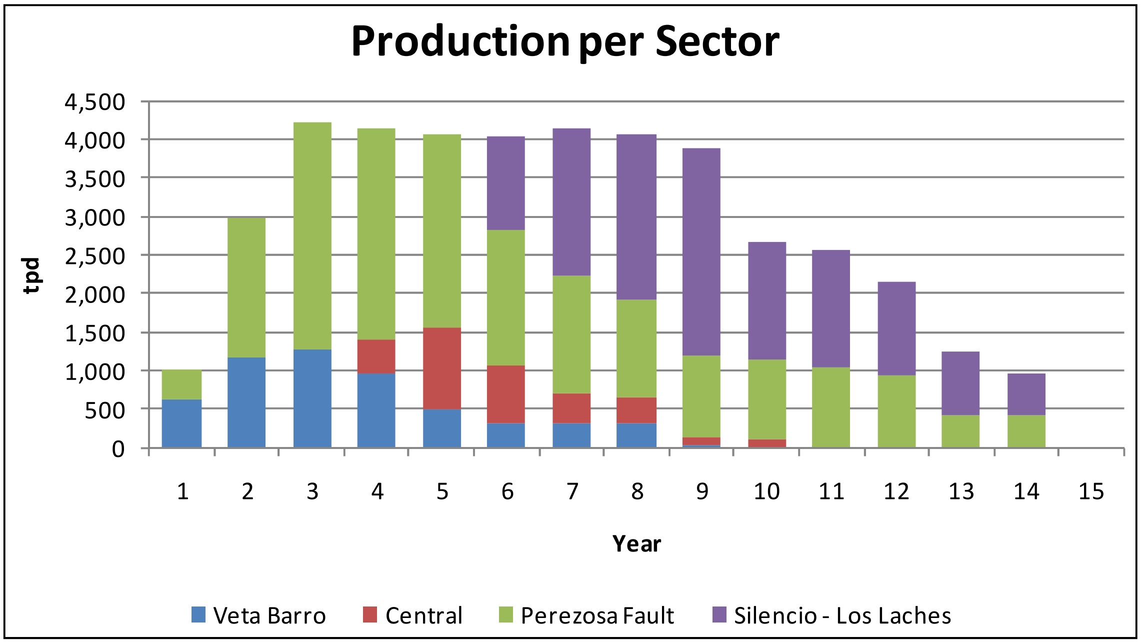
Perezosa Fault						
Year	**Ore (t)**	**Tpd**	**Au (g/t)**	**Ag (g/t)**	**Cu (%)**	**S (%)**
1	116,100	387	5.843	18.023	0.092	3.803
2	545,100	1,817	5.835	10.716	0.048	3.558
3	883,100	2,944	5.555	10.270	0.058	3.592
4	815,669	2,719	5.265	10.687	0.067	3.626
5	753,600	2,512	5.076	10.625	0.071	3.632
6	526,551	1,755	5.300	14.322	0.095	3.848
7	453,600	1,512	5.420	16.296	0.107	3.964
8	382,672	1,276	5.536	17.602	0.108	3.833
9	312,905	1,043	5.253	17.233	0.118	4.097
10	306,000	1,020	5.217	17.183	0.119	4.130
11	306,000	1,020	5.217	17.183	0.119	4.130
12	278,762	929	5.262	17.350	0.117	4.136
13	126,000	420	5.811	20.752	0.099	4.222
14	126,000	420	5.811	20.752	0.099	4.222
>14	1,275,004		5.897	20.072	0.095	4.155
Total	**7,207,064**		**5.486**	**14.841**	**0.087**	**3.866**

Table 18.1-16: Silencio-Los Laches Sector production plan

Silencio-Los Laches						
Year	Ore (t)	Tpd	Au (g/t)	Ag (g/t)	Cu (%)	S (%)
1						
2						
3						
4						
5						
6	360,000	1,200	9.23	28.13	0.03	4.43
7	579,390	1,931	8.40	30.66	0.03	4.32
8	642,440	2,141	3.77	114.97	0.18	4.42
9	809,480	2,698	3.36	149.38	0.23	4.85
10	459,480	1,531	4.80	25.19	0.07	3.37
11	459,480	1,531	4.80	25.19	0.07	3.37
12	370,084	1,233	4.77	23.29	0.07	3.43
13	247,462	824	4.54	18.66	0.08	3.87
14	166,920	556	4.46	12.17	0.09	4.04
>14	53,201		5.70	17.84	0.07	3.42
Total	3,597,938		5.80	25.07	0.06	3.70

The area Silencio – Los Laches enters the production **s**chedule after year 6, even though it has better grade**s**. This is due to its depth and greater mine development requirement. The Figure 18.1-9 presents the production by sector.

Figure 18.1-9: Production (tpd) by Sector



Materials Handling

Mucking will be completed with 7 cubic yards LHD`s. LHD`s will load into low profile trucks. Hauling will be performed by 20 ton trucks. Hauling activities will comprise the following:

- Ore hauling from the mine to the crushing station.
- Backfill material hauling from the dump to the stopes.

In order to determine the hauling requirements, the following activities were developed:

- Backfill balance between material (ore & waste) generated at the mine and material (waste) needed to backfill stopes.
- Average distance estimation for material differentiated by the location of origin. These distances are a function of the mine design (ramps, accesses and production levels).

Backfill Balance

To determine the quantity of backfill that will be required to haul from surface to the mine, a balance between the waste originated at the mine, from developments in waste rock, and the ore produced was carried out.

Ore production is used to estimate cubic meters of volume to be filled. Ore production was obtained directly from the production plan. Waste produced at the mine was obtained from the development and preparation plan, considering all meters developed in waste rock. These numbers were then compared, obtaining the amount of excess or additional backfill material required, Table 18.1-17.

Table 18.1-17: Backfill balance

Year	Volume Produced from Development (m3)				Volume required		Backfill from External Source
	5 x4 m	4x4 m	Vert.	Total	Prod (t)	Prod. (m3)	(m3)
-2	35,662	0	295	35,957			
-1	62,876	0	3,205	66,081			
1	59,903	16,657	4,903	81,463	301,100	167,278	85,814
2	77,920	47,476	4,195	129,591	896,100	497,833	368,243
3	118,618	58,741	4,266	181,624	1,265,772	703,207	521,582
4	86,417	53,387	3,209	143,014	1,239,883	688,824	545,810
5	86,027	29,584	4,056	119,668	1,220,927	678,293	558,625
6	75,861	38,938	4,911	119,710	1,207,855	671,030	551,320
7	55,309	44,870	4,235	104,414	1,245,454	691,919	587,505
8	38,213	47,930	1,232	87,375	1,219,201	677,334	589,959
9	31,314	44,406	1,232	76,952	1,167,129	648,405	571,453
10	9,547	40,689	392	50,628	801,480	445,267	394,639
11	0	39,459	0	39,459	769,163	427,313	387,854
12	0	33,609	0	33,609	648,846	360,470	326,862
13	0	18,885	0	18,885	373,462	207,479	188,594
14	0	7,449	0	7,449	292,920	162,734	155,284
Total	737,667	522,080	36,132	1,295,879	12,649,292	7,027,384	5,833,544

According to this table, it will be necessary to source about 5.8 million m^3 of material from external sources to provide enough backfill for the mine. Tailings from the process plant may be comingled with waste material to reduce the amount of external resource material required.

18.1.6 Equipment Fleet

The mine equipment estimate has been carried out based on the mine production and development plans. Equipment performances were estimated considering average distances. Estimation was made based on 8 hours/shift (5 effective operation hours), 3 shifts/day and 360 days/year.

Equipment requirement was separated into two main areas:

- Production: Involves ore production and ore handling, backfill material handling. Ore production considers a 4.0 m x 4.0 m drilling section to estimate drilling parameters.

- Development: Involves development of tunnels and the corresponding waste handling. These tunnels correspond to ramps with a 5.0 m x 4.0 m section and accesses & pivots with a 4.0 m x 4.0 m section. Waste material handling from vertical developments, is also included.

General parameters for performance estimation are summarized in the Table 18.1-18:

Table 18.1-18: General parameters for equipment estimation

Gallery Section		5 x 4 m	4 x 4 m
General parameters			
Density	t/m3	2.55	2.55
Swell Density	t/m3	1.53	1.53
Section	m2	20.00	16.00
- Width	m	5.00	4.00
- Height	m	4.00	4.00
Drilling diameter	mm	45.00	45.00
Burden	m	0.80	0.80
Spacing	m	0.80	0.80
Holes/blast		50.00	45.00
Drilling length	m	4.00	4.00
Efficiency		0.75	0.75
Advance meters/blast	m	3.00	3.00
Drilled meters/blast	m	200.00	180.00
Drilled meters/blasted meter	m	66.67	60.00
Tonnes/blast	t/blast	153.00	122.40
Tonnes/drilled meter	t/dm	0.77	0.68

Production Fleet

Drilling

The requirement of Jumbos was estimated as a function of meters developed (ramps, accesses and pivots) and the production from drifts and benches. Estimates are based on the production and preparation plans.

An advancing rate per shift was estimated per type of tunnel as a function of the section and then used to estimate the number of units. The Tables 18.1-19 to 18.1-22 show the estimation of the jumbo's performance in ore (4 m x 4 m) and in waste (5 m x 4 m and 4 m x 4 m) developments.

Table 18.1-19: Jumbo performance for 5 m x 4 m gallery

Equipment	Jumbo	
Drilling speed	m/min	1.00
Boom	Boom	1.00
Drilling time	min	208.00
Hole switch time	min	52.00
Spot changing time	min	40.00
Operational losses (transport, blast, etc.)	min	0.47
Performance	d.m./hour	18.80
Time/blast	hr/blast	5.00

Table 18.1-20: Jumbo performance for 4 m x 4 m gallery

Equipment	Jumbo	
Drilling speed	m/min	1.00
Boom	Boom	1.00
Drilling time	Min	188.00
Hole switch time	Min	47.00
Spot changing time	Min	40.00
Operational losses (transport, blast, etc.)	Min	0.47
Performance	d.m./hour	18.46
Time/blast	hr/blast	4.58

Table 18.1-21: Bolting Jumbo performance

Equipment	Bolting Jumbo	
Drilling speed	m/min	1.00
Boom	Boom	1.00
Holes per Blast		33.00
Hole length	M	3.00
Drilling time	min	99.00
Hole switch time	min	33.00
Spot changing time	min	40.00
Operational losses (transport, blast, etc.)	min	0.47
Performance	d.m./hr	16.23
Time/blast	hr/blast	2.87

Table 18.1-22: DTH performance for 8 m benches

Equipment		DTH
Drilling speed	m/min	0.30
Drilling depth	m	8.00
Drilling time	min	26.67
Rod switch time	min	2.67
Rods removal time	min	2.13
Hole switch time	min	2.00
Total	hr/hole	0.67
Performance	d.m./hr	11.95
Time/blast	hr/blast	3.35

Loading

For loading activities 7 yd^3 LHDs were selected. LHD numbers were estimated based on the mine plans and the expected performance of the equipment in the different activities (Tables 18.1-23, 18.1-24 and 18.1-25). The equipment is allocated to ore loading, backfill dumping in the production stopes and mucking in the development tunnels.

Different performance estimations were made for LHDs in the following two loading conditions:

Ore Loading: Includes work in the ore stopes, loading into 20 t trucks, and considers an average hauling distance of 250 meters from the stope to the truck loading station.

- Waste Loading: Includes work in development tunnels and backfill of stopes, and considers an average hauling distance of 50 meters for ramps and accesses, and 100 meters for preparation developments, from the face to the loading station.

Table 18.1-23: Loading performance at developments

Equipment	LHD 7 yd3	
Nominal Capacity	yd3	7.00
Nominal Capacity	m3	5.22
Fill factor		0.90
Effective Load	Ton	7.18
Loading	min	0.50
Dumping	min	0.25
Average distance	m	50.00
Tramming	min	1.50
Delays	min	0.50
Cycle	min	2.75
Operational losses (transport, blast, etc.)		0.70
Performance	t/hr	109.67
Time/blast	hr/blast	1.40

Table 18.1-24: Loading performance at preparations

Equipment	LHD 7 yd3	
Nominal Capacity	yd3	7.00
Nominal Capacity	m3	5.22
Fill factor		0.90
Effective Load	Ton	7.18
Loading	min	0.50
Dumping	min	0.25
Average distance	m	100.00
Tramming	min	3.00
Delays	min	0.50
Cycle	min	4.25
Operational losses (transport, blast, etc.)		0.70
Performance	t/hr	70.97
Time/blast	hr/blast	1.72

Table 18.1-25: Loading performance at production

Equipment	LHD 7 yd3	
Nominal Capacity	yd3	7.00
Nominal Capacity	m3	5.22
Fill factor		0.90
Effective Load	Ton	7.49
Loading	min	0.50
Dumping	min	0.25
Average distance	m	250.00
Tramming	min	4.29
Delays	min	0.50
Cycle	min	5.54
Operational losses (transport, blast, etc.)		0.70
Performance	t/hr	54.48
Time/blast	hr/blast	2.25

Hauling

Trucks were estimated for ore transported to the plant, backfill material to the stopes and for waste from development drifts. An estimate of the average performance is shown in the Tables 18.1-26 to 18.1-28.

Table 18.1-26: Hauling performance at developments

Equipment	20 t Truck	
Nominal Load	Ton	20.00
Fill Factor		1.00
Empty speed	km/hr	12.00
Loaded speed	km/hr	8.00
Bucket passes/Truck		3.00
Loading	min	8.25
Dumping	min	0.50
Delays	min	2.00
Distance	m	3,500.00
Traveling Loaded	min	26.25
Traveling Empty	min	17.50
Traveling	min	43.75
Cycle	min	54.50
Performance	t/hr	18.35
Time/blast	hr/blast	8.34

Table 18.1-27: Hauling performance at preparations

Equipment	20 t Truck	
Nominal Load	Ton	20.00
Fill Factor		1.00
Empty speed	km/hr	12.00
Loaded speed	km/hr	8.00
Bucket passes/Truck		3.00
Loading	min	12.75
Dumping	min	0.50
Delays	min	2.00
Distance	m	4,500.00
Traveling Loaded	min	33.75
Traveling Empty	min	22.50
Traveling	min	56.25
Cycle	min	71.50
Performance	t/hr	13.99
Time/blast	hr/blast	8.75

Table 18.1-28: Hauling performance at production

Equipment	20 t Truck	
Nominal Load	Ton	20.00
Fill Factor		1.00
Empty speed	km/hr	12.00
Loaded speed	km/hr	8.00
Bucket passes/Truck		3.00
Loading	min	16.61
Dumping	min	0.50
Delays	min	2.00
Distance	m	4,500.00
Traveling Loaded	min	33.75
Traveling Empty	min	22.50
Traveling	min	56.25
Cycle	min	75.36
Performance	t/hr	13.27
Time/blast	hr/blast	9.22

Support Fleet

The type of equipment and its quantity was estimated taking into account the following considerations:

- Number of stopes in production per mine, actually the maximum active stopes at a given time.
- Overall requirements for a modern standard operation.

The following support equipment was considered for the project:

- Scaler
- Utility trucks:
  - For explosive distribution: These can be conventional diesel trucks, 1,500 kg loading capacity, equipped with all the requirements established in regulations for explosive transportation.
  - For materials transportation inside the mine (all-purpose): Conventional flat – bed trucks, 1,500 kg loading capacity.
  - For general maintenance services, in order to reduce the number of trips of the equipment to the maintenance shop, particularly the Jumbos.

Main and Ancillary Fans: Main fans (200 – 300 kcfm) were estimated by the number of main areas projected in operation. Ancillary fans are estimated as a function of active stopes and tunnels in development at a given time, according to production and development plans. See Table 18.1-29.

Table 18.1-29: Support fleet

Main Fans	**5**
Auxiliary Fans	24
Maintenance Trucks	4
Explosive Trucks	3
Scaler	3
All purpose truck	4

Fleet Requirement

To calculate the number of equipment per fleet, the previously estimated performances were used in conjunction with the requirements defined in the mine and preparation plans. The Table 18.1-30 summarizes the number of units required per year for the main type of equipment:

Table 18.1-30: Production equipment fleet

Year	LHD 7 yd^3	Jumbo	DTH	20 t Truck	Bolting Jumbo
-2	2	1	0	3	2
-1	2	2	0	3	2
1	7	5	2	13	3
2	9	10	2	34	3
3	11	13	3	48	4
4	11	12	3	48	3
5	11	11	3	46	3
6	11	11	3	46	3
7	11	11	3	47	3
8	10	10	3	40	3
9	9	7	2	32	3
10	9	7	2	32	3
11	7	6	2	29	2
12	6	5	2	25	2
13	5	4	2	15	2
14	5	3	2	12	2

Based on the previously estimated requirements, an acquisition calendar was defined including initial purchasing and replacement.

The Table 18.1-31 shows the number of units and the proposed period for acquisition:

Table 18.1-31: Total fleet requirement & acquisition schedule.

Year	-2	-1	1	2	3	4	5	6	7	8	9	10	11
Main Equipment													
LHD 7 yd3	2	2	7	9	11	11	11	11	11	10	9	9	7
Fleet increment	2		5	2	2								
Replacement					2		5	2	4		3	2	2
Total Acquisition	2		5	2	4		5	2	4		3	2	2
Jumbo	1	2	5	10	13	12	11	11	11	10	7	7	6
Fleet increment	1	1	3	5	3								
Replacement					1		2	5	4			3	3
Total Acquisition	1	1	3	5	4		2	5	4			3	3
DTH			2	2	3	3	3	3	3	3	2	2	2
Fleet increment			2		1								
Replacement							2		1		1		1
Total Acquisition			2		1		2		1		1		1
Truck 20 t	3	3	13	34	48	48	46	46	47	40	32	32	29
Fleet increment	3		10	21	14								
Replacement					3		8	21	18			14	15
Total Acquisition	3		10	21	17		8	21	18			14	15
Bolting Jumbo	2	2	3	3	4	3	3	3	3	3	3	3	2
Fleet increment	2		1		1								
Replacement					2				3				2
Total Acquisition	2		1		3				3				2
Support													
Main Fans		1	1	1	1	1							
Auxiliary Fans		4	5	5	5	5							
Maint. Trucks		1	1	2									
Explosive Trucks	1		1		1								
Scaler	1		1		1								
All purpose truck		1	1	1	1								

18.1.7 Services and Infrastructure

The following main items are considered as part of the mine services and infrastructure:

- Workshops and offices at surface.
- Workshops and offices equipment.
- Explosive magazine at surface.
- Cement plant.

- Main accesses (portals).
- Fire-proof refuge.
- Transformers.
- Cables, protections and assembly.
- Communications

18.2 Tailings Disposal

The following are the main criteria considered for the filtered tailings design:

- The tailings disposal facilities shall be constructed near the process plant location, preferably at a lower elevation to optimize energy consumptions.
- The tailings disposal shall be located preferably within Greystar's properties.
- Tailings disposal shall be designed to include the whole tonnage of the Angostura project, considered as 14,000,000 tonnes. The disposal facilities shall be designed below 3,000 masl.
- The design considers filtered tailings disposal, with a deposition density of Cw=85%. The material deposit density for design is 1.6 t/m3.
- The tailings disposal will be placed in terraces of 10m (approximately), with a global slope of 3:1. In order to avoid terrain failure, it is considered to design a supporting platform to contain the tailings material.
- Diversion channels shall be designed to avoid rainwater conduction to the supporting platform.
- The filtered tailings material will be transported by trucks to the disposal facilities.

The area was reviewed, including conditions and dimensions, using available satellite images and topography. After studying the preliminary main areas selected for tailings disposal and the mine properties, a location was selected 200 meters south from the mine portals for tailings disposal facilities.

The analysis took into consideration the following items: rough grade surface, supporting platform, under drains installations, place and compact soils bedding fill, supply and install 2mm waterproof membrane, diversion channel.

The following are the material take-off estimations, required for the filtered tailings disposal facilities.

- Terrain preparation includes organic material removal from the entire disposal area, according to 351,400 m3.
- The supporting platform wall has an altitude of 130m and requires 1,850,000 m3 of filling material.
- Tailings disposal platform starts on 2,890 masl and finish on 3,040 masl, at the end of the plant operation.
- The disposal construction also implies the installation of 49,200 m2 of 2mm waterproof membrane and 80,260 m2 of filling material.
- The diversion channel requires the cut of 217,000 m3 of material, in order to conduct the rainwater outside of the tailings disposal.
- The disposal facilities are 2.6 km away from the process plant.
- For tailing transportation to the final disposal, 4 trucks (30 tonnes each), 1 loader and 1 bulldozer are required.

18.3 Water Management

One of the objectives in the process plant design was the efficient use of fresh water, mainly through internal recirculation. This is in order to minimize solution purges, in addition to minimizing fresh water and cyanide consumption.

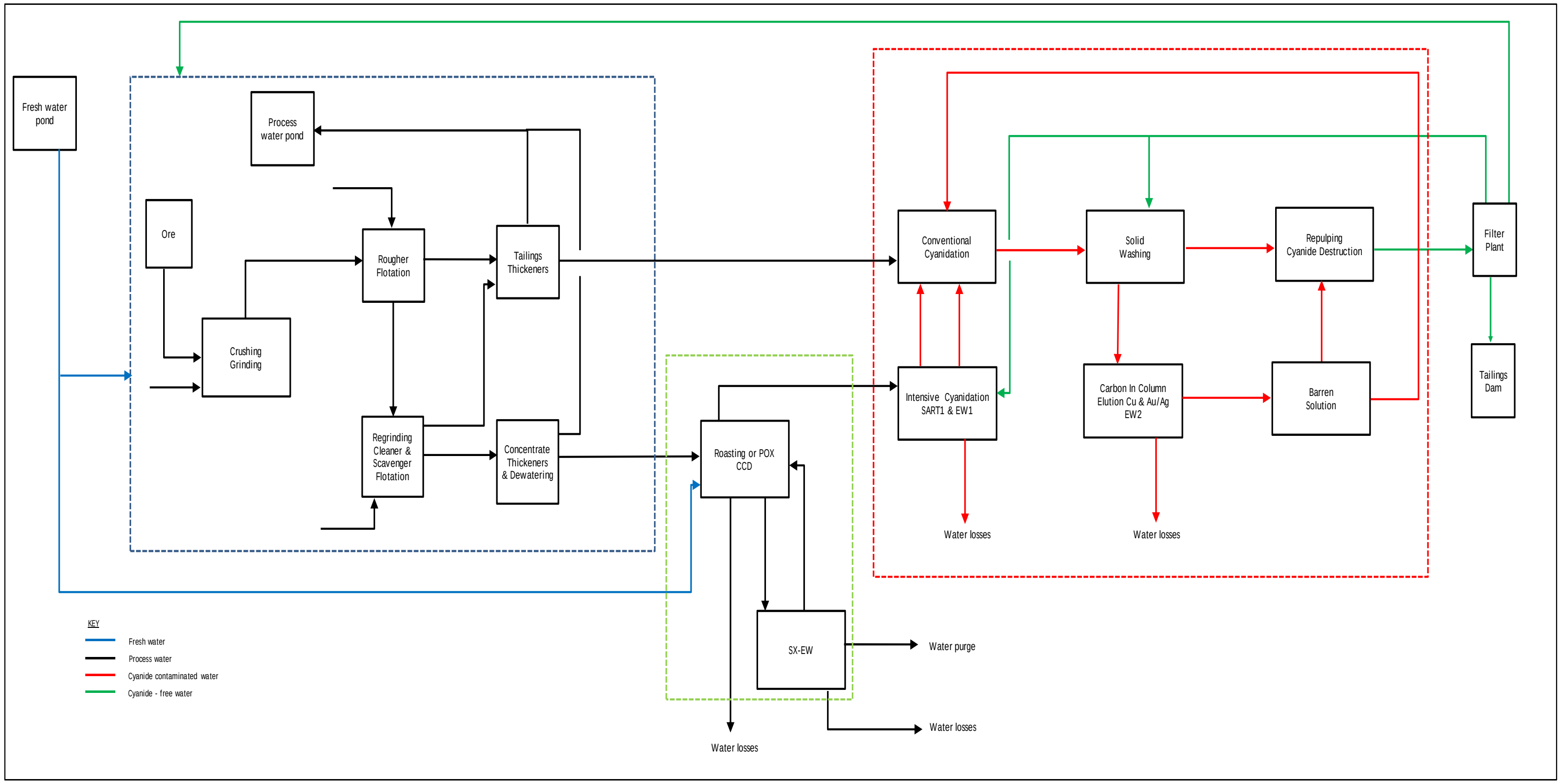
For water management purposes, three water circuits were established for each analyzed process alternative, corresponding to: grinding and flotation circuit, sulfur oxidation circuit and cyanidation-CIC-SART-EW-Cyanide destruction circuit. Each circuit is fed with both fresh and process water; the latter could come from an internal recirculation or from other water circuit.

The plant fresh water make up is 596 m^3/d, which considers the reutilization of treated water from CN destruction and recovered water from the tailings filter plant.

Fresh water is used in the cyanide-free area of the plant, which comprises grinding & flotation and oxidation circuits. It is used mainly as dilution water in the grinding stage, to dilute the slurry prior to all flotation stages (rougher, scavenger and cleaner) and as washing water in the CCD circuit in the sulfur oxidation stage (for the three alternatives).

The diagram in the Figure 18.3-1 schematize the water movement in the plant, for the three process alternatives, and considering the three water circuits previously defined:

Figure 18.3-1: Water diagram for all alternatives.



18.4 Personnel

18.4.1 Mine Personnel

Mine personnel includes all the salaried supervisory and staff people working in mine operations, maintenance, and engineering and geology departments, and the hourly people required to operate and maintain the drilling, blasting, loading, hauling, and mine support activities.

Salaried Staff

The Table 18.4-1 summarizes the mine administrative personnel staff estimation:

Table 18.4-1: Mine Administration personnel

Administration	Production	Pre-Production
Operations Superintendent	1	1
Mine Captains	4	2
Head of Drill & Blasting	3	1
Shift Leaders	8	2
Senior Engineering	2	1
Engineers	2	1
Technician	2	1
Statistician	2	
Surveyor	2	1
Assistant	4	2
Service driver	3	1
Services	3	1
Geology Superintendent	1	1
Geologist	2	
Draftsmen	2	1
Geology helper & sampler	4	2
Rock Mechanic Engineer	1	
Maintenance Superintendent	1	1
Head of Maintenance	2	1
Maintenance Engineer	1	1
Maint. Shift Leaders	3	1
Statistician	2	1
TOTAL	55	23

Direct Labor

The total number of active operators has been estimated as a function of the number of main equipment, assuming 4 operators per machine per day. An allowance of 5 % for absenteeism, vacations and sickness has also been included in the estimates.

The Table 18.4-2 shows theestimated operation manpower requirement per year.

Table 18.4-2: Mine Direct Manpower

Year	LHD	20 t	Jumbo	DTH	Bolting	Explosive	Mechanics &	TOTAL
-2	7	10	4	0	7	4	15	47
-1	7	10	7	0	7	4	16	51
1	19	35	13	7	10	7	51	142
2	26	89	26	7	10	7	95	260
3	29	124	35	10	13	10	130	351
4	29	124	32	10	10	7	125	337
5	29	117	29	10	10	7	121	323
6	29	117	29	10	10	7	121	323
7	29	121	29	10	10	7	122	328
8	26	102	26	10	10	7	108	289
9	26	83	-19	7	10	7	87	239
10	26	83	19	7	10	7	87	239
11	19	76	16	7	7	4	75	204
12	16	64	13	7	7	4	65	176
13	13	38	13	7	7	4	46	128
14	13	32	10	7	7	4	40	113

Explosive charger labor has been estimated from the total units of equipment considered to load explosives in the development tunnels. Bench blasting will be performed by the same DTH crew.

Total Mine Labor

Total labor required in the mine and the corresponding productivity per year indicator is presented in the Table18.4-3:

Maintenance staff was calculated assuming a factor of 2.5 people per equipment in maintenance plus holidays and absenteeism factors.

Table 18.4-3: Total Mine personnel

Year	Administrative	Direct	TOTAL	t/man-day
-2	23	47	70	
-1	23	51	74	
1	55	142	197	4.2
2	55	260	315	7.9
3	55	351	406	8.7
4	55	337	392	8.8
5	55	323	378	9.0
6	55	323	378	8.9
7	55	328	383	9.0
8	55	289	344	8.2
9	55	239	294	7.7
10	55	239	294	7.6
11	55	204	259	8.2
12	55	176	231	7.8
13	55	128	183	5.7
14	55	113	168	4.8

The average productivity per man-day is 7.6 t/man-day, which compare to the average of similar operations known to NCL.

18.4.2 Plant Personnel

Process plant personnel have been estimated as follows:

- Operations staff were classified in two roles; A for shift chiefs and B for control room operators and operators.
- Maintenance personnel were classified in two roles; A for maintenance chiefs and B for supervisor, mechanics, electricians and instrument technician.
- To determine role A human resources, an estimation was made taking into account a typical administrative day's work as well as the number of workers necessary per shift, considering absenteeism, vacations and substitute workers.
- To determinate role B human resources it is estimated the total allowance required by shift, considering absenteeism, vacations and substitute workers.
- Labor includes salaries, additional benefits and safety at work.
- For regular operations, four shifts per day are considered in order to satisfy the three shifts per day working requirement (plus one "swing" shift).
- For maintenance operations, one shift per day is considered in order to satisfy the one shift per day working requirement.

The Figures 18.4-1, 18.4-2 and 18.4-3 correspond to the process plant organization chart, which includes both operations and maintenance personnel.

The operations personnel are organized by operation and maintenance and by plant areas, according to: crushing, grinding, flotation, oxidation stage (roasting + acid plant, POX or BIO-OX) & CCD, concentrate cyanidation & SART-CIC-elution-EW-smelting, conventional cyanidation & cyanide destruction and tailings disposal.

Figure 18.4-1: Personnel – Processing & Maintenance – Roasting

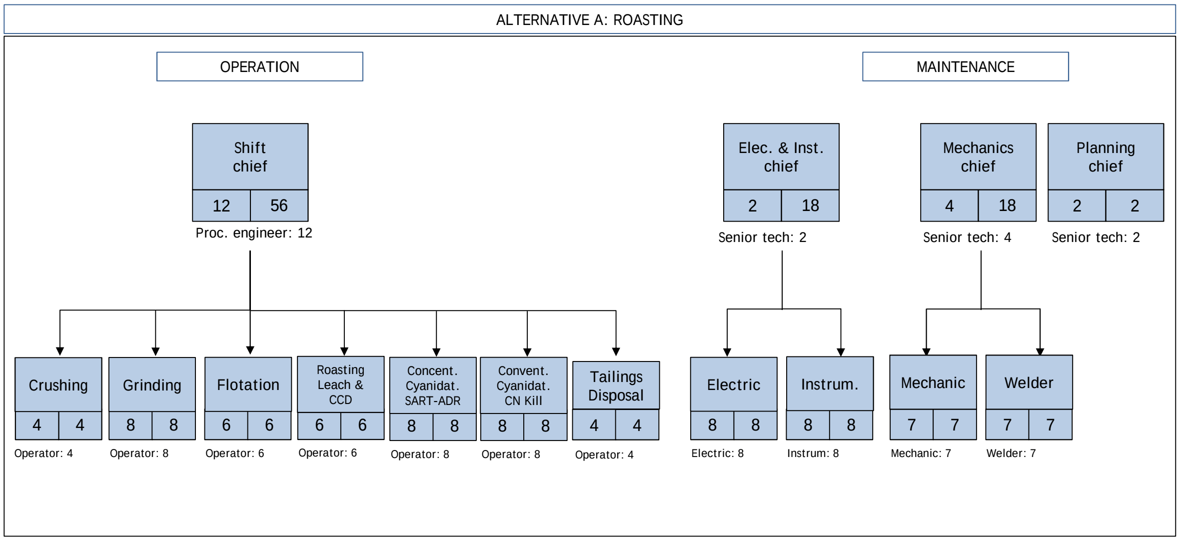


Figure 18.4-2: Personnel – Processing & Maintenance – POX

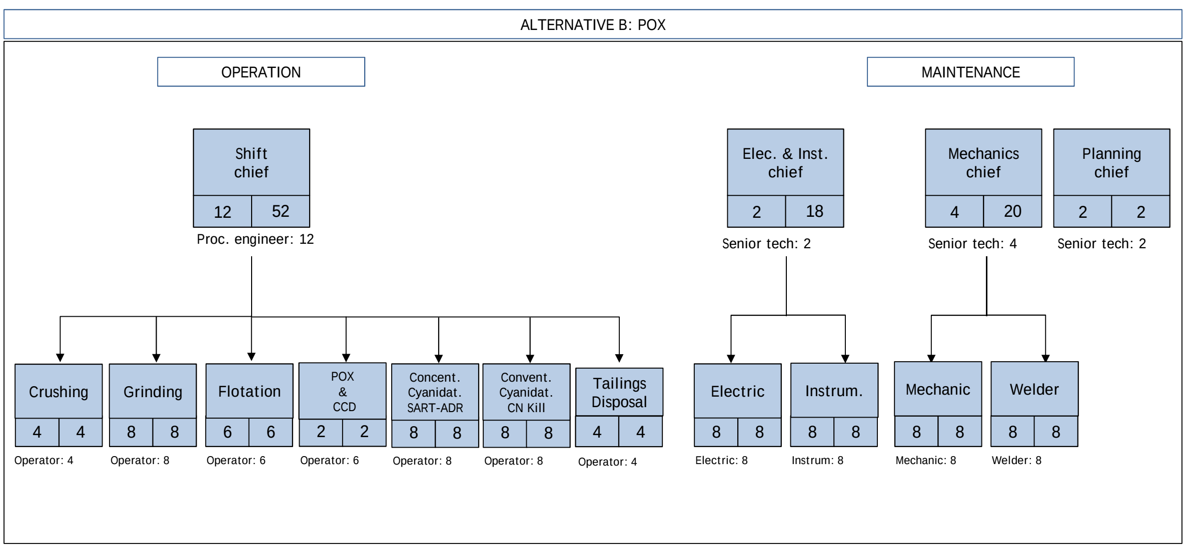
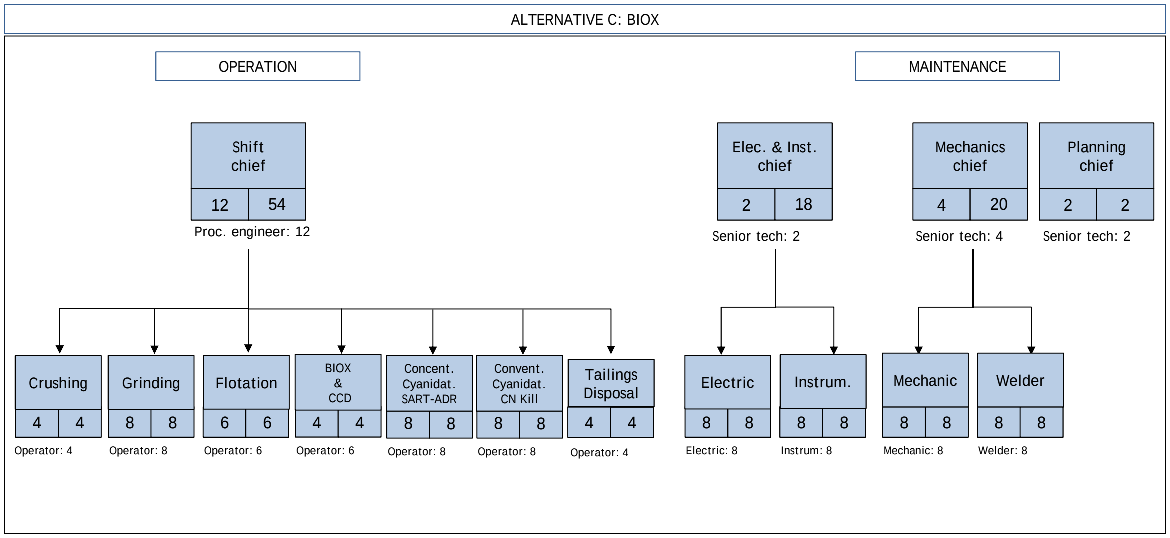


Figure 18.4-3: Personnel – Processing & Maintenance – BIOX



18.5 Capital Cost Estimate

18.5.1 Mine Capital Cost

Mine Capital expenditures have been separated in three items:

- Mine development.
- Equipment acquisition.
- Infrastructure and services.

Items considered as mine development are:

- Main accesses and Ramps (all with a 5 m x 4 m section).
- Ventilation rises and Ore passes.

Quantities of meters associated to each one of these items were estimated as a yearly profile, according to the mine development plan. Total expenses in development, not including labor, are presented in Table 18.5-1:

Table 18.5-1: Development expenses

Year	5 m x4 m (1,200 US$/m)	Vert. Ø=3 m (1,000 US$/m)	Total (US$/m)
-2	2,140	42	2,181
-1	3,773	453	4,226
1	3,594	694	4,288
2	4,675	593	5,269
3	**7,117**	**604**	**7,721**
4	5,185	454	5,639
5	**5,162**	**574**	**5,736**
6	4,552	695	5,246
7	**3,319**	**599**	**3,918**
8	2,293	174	2,467
9	**1,879**	**174**	**2,053**
10	573	55	628

Unit costs for development were estimated assuming owner's equipment and owner's personnel.

Capital for equipment acquisition and replacement was defined based on the fleet requirement estimations developed in Section 6 of this report. A summary of capital expenditures for equipment is presented in Table 18.5-2.

Table 18.5-2: Equipment capital expenditures

Year	LHD 7 yd3 540	Jumbo 450	DTH 600	20 t Truck 500	Bolting Jumbo 590	TOTAL KUS$
-2	1,080	450		1,500	1,180	4,535
-1		450				870
1	2,700	1,350	1,200	5,000	590	11,615
2	1,080	2,250		10,500		14,380
3	2,160	1,800	600	8,500	1,770	15,505
4						250
5	2,700	900	1,200	4,000		8,800
6	1,080	2,250		10,500		13,830
7	2,160	1,800	600	9,000	1,770	15,330
8						
9	1,620		600			2,220
10	1,080	1,350		7,000		9,430
11	1,080	1,350	600	7,500	1,180	11,710

Infrastructure and service expenses estimates are shown in Table 18.5-3.

Table 18.5-3: Infrastructure and services capital expenditures

	Workshops and offices at surface	Workshops and offices equipment	Explosive magazine at surface	Cement plant	Main accesses (6 portals)	Fire-proof refuge	Transformers	Cables, protections and assembly	Communications	TOTAL
Year	400	100	80	250	600	400	250	400	200	**KUS$**
-2	100%	100%	100%		33%					**778**
-1										
1				100%	22%	30%	60%	60%	60%	**1,012**
2					22%	50%	20%	20%	20%	**502**
3					11%	20%				**146**
4					12%					**72**
5							20%	20%	20%	**170**

The total capital expenditures for the life of mine are presented in following Table 18.5-4. A contingency of 35% has been added to all the mine investment estimated.

Table 18.5-4: Capital costs summary.

Year	Administration KUS$	Mine Labor KUS$	Developments KUS$	Equipments KUS$	Infrastructure & Services KUS$	Contingencies 35% KUS$	TOTAL KUS$
-2	567	949	2,181	4,535	778	2,623	**11,633**
-1	1,134	1,020	4,226	870		1,784	**9,034**
1			4,288	11,615	1,012	5,920	**22,835**
2			5,269	14,380	502	7,053	**27,204**
3			7,721	15,505	146	8,180	**31,552**
4			5,639	250	72	2,086	**8,047**
5			5,736	8,800	170	5,147	**19,853**
6			5,246	13,830		6,677	**25,753**
7			3,918	15,330		6,737	**25,985**
8			2,467			863	**3,330**
9			2,053	2,220		2,770	**7,043**
10			628	9,430		2,456	**12,514**
11				11,710		3,889	**15,599**
Total	**1,701**	**1,969**	**49,372**	**108,475**	**2,680**	**56,184**	**220,381**

Initial capital cost for the underground mine is estimated at 20.6MUS$. Life of Mine capital requirement is estimated as 220.4MUS$.

18.5.2 Plant Infrastructure Capital Cost Estimate

The plant capital cost was calculated with a precision of +/-35%, and has been prepared in consideration with information provided by the previous design selection. The capital cost includes the cost estimate for the process plant and for tailings disposal.

A list of mechanical equipment was prepared for all the areas comprised in the process plant. The costs for main equipment, such as crushers, mills, filters and so on, were based on referential quotations made to vendors, DFS (definitive feasibility study) and on a historic data of similar projects. Tailings disposal capital costs contemplates mainly civil works costs.

The total mechanical equipment capital cost was US$ 77.8 million for roasting, US$ 80.1 million for pressure oxidation and US$ 63.7 million for biooxidation.

Civil work costs – excavation, landfill and concrete – were calculated by area, based on material take off estimations.

The costs of equipment installation, piping, electrical distribution, concrete, structural steel, instrumentation and spare parts, were calculated by percentage factors over the mechanical equipment capital cost.

Freight and insurances was calculated as 8% of foreign equipment and 2% of national equipment.

Two indirect costs were contemplated: contractor indirect costs and project indirect costs. Contractor indirect costs were calculated to be 35% of construction and assembly. Project indirect costs include the following items: permission and royalties, engineering and EPCM services, freight and insurances for bot, national and international equipment, and plant start up.

EPCM cost was considered to be 12% of direct and contractor indirect costs.

Contingency was estimated as 35% of direct and indirect costs.

The Tables 18.5-5 to 18.5-8 present summaries of both process plant and tailings disposal capital costs. For process plant, three alternatives are shown

Table 18.5-5: Process plant and tailings disposal capital costs – Alternative A, Roasting

PROCESS PLANT AND TAILINGS DISPOSAL - ALTERNATIVE A: ROASTING			
UNITARY OPERATION	CONSTRUCTION AND ASSEMBLY KUS$	ACQUISITIONS KUS$	TOTAL DIRECT COST KUS$
Primary Crushing - Secondary and Tertiary Crushing	5.745	3.967	**9.712**
Grinding	5.343	8.043	**13.386**
Rougher Flotation – Regrinding and Cleaner & Scavenger Flotation	6.968	8.539	**15.507**
Roasting - Acid Leaching - CCD circuit - SX/EW of Copper	8.982	21.972	**30.954**
Intensive Cyanidation and Dewatering	2.921	3.531	**6.452**
Conventional Cyanidation	4.267	7.043	**11.310**
CCD circuit - Cyanide Destruction - Filter plant	8.777	13.163	**21.940**
SART Process	393	5.000	**5.393**
CIC – Elution – EW - Smelting	1.766	2.222	**3.988**
Power Supply	1.000	3.000	**4.000**
Mining road to primary crushing and Process plant earth movement	6.568	153	**6.722**
Tailings disposal	25.041	0	**25.041**
Trucks and Bulldozer	0	1.150	**1.150**
I.- TOTAL DIRECT COSTS	**77.771**	**77.784**	**155.555**
CONTRACTOR INDIRECT COSTS		**35%**	**27.220**
PROJECT INDIRECT COSTS			**29.137**
II.- TOTAL INDIRECT COSTS			**56.356**
III.- CONTINGENCIES		**35%**	**74.169**
TOTAL INVESTMENT BUDGET		**KUS$**	**286.081**

Table 18.5-6: Process plant and tailings disposal capital costs – Alternative B, POX

PROCESS PLANT AND TAILINGS DISPOSAL - ALTERNATIVE B: POX			
UNITARY OPERATION	CONSTRUCTION AND ASSEMBLY KUS$	ACQUISITIONS KUS$	TOTAL DIRECT COST KUS$
Primary Crushing - Secondary and Tertiary Crushing	5.745	3.967	**9.712**
Grinding	5.343	8.043	**13.386**
Rougher Flotation – Regrinding and Cleaner & Scavenger Flotation	6.300	6.805	**13.106**
POX - CCD circuit - SX/EW of Copper	6.476	26.067	**32.543**
Intensive Cyanidation and Dewatering	2.921	3.531	**6.452**
Conventional Cyanidation	4.267	7.043	**11.310**
CCD circuit - Cyanide Destruction - Filter plant	8.777	13.163	**21.940**
SART Process	393	5.000	**5.393**
CIC – Elution – EW - Smelting	1.766	2.222	**3.988**
Power Supply	1.000	3.000	**4.000**
Mining road to primary crushing and Process plant earth movement	6.568	153	**6.722**
Tailings disposal	25.041	0	**25.041**
Trucks and Bulldozer	0	1.150	**1.150**
I.- TOTAL DIRECT COSTS	**74.597**	**80.146**	**154.743**
CONTRACTOR INDIRECT COSTS		**35%**	**26.109**
PROJECT INDIRECT COSTS			**29.443**
II.- TOTAL INDIRECT COSTS			**55.552**
III.- CONTINGENCIES		**35%**	**73.603**
TOTAL INVESTMENT BUDGET		**KUS$**	**283.898**

Table 18.5-7: Process plant and tailings disposal capital costs – Alternative C, BIOX

PROCESS PLANT AND TAILINGS DISPOSAL - ALTERNATIVE C: BIO-OX			
UNITARY OPERATION	CONSTRUCTION AND ASSEMBLY KUS$	ACQUISITIONS KUS$	TOTAL DIRECT COST KUS$
Primary Crushing - Secondary and Tertiary Crushing	5.745	3.967	**9.712**
Grinding	5.343	8.043	**13.386**
Rougher Flotation – Regrinding and Cleaner & Scavenger Flotation	6.300	6.805	**13.106**
BIO-OX - CCD circuit	7.635	9.620	**17.255**
Intensive Cyanidation and Dewatering	2.921	3.531	**6.452**
Conventional Cyanidation	4.267	7.043	**11.310**
CCD circuit - Cyanide Destruction - Filter plant	8.777	13.163	**21.940**
SART Process	393	5.000	**5.393**
CIC – Elution – EW - Smelting	1.766	2.222	**3.988**
Power Supply	1.000	3.000	**4.000**
Mining road to primary crushing and Process plant earth movement	6.568	153	**6.722**
Tailings disposal	25.041	0	**25.041**
Trucks and Bulldozer	0	1.150	**1.150**
I.- TOTAL DIRECT COSTS	**75.756**	**63.698**	**139.454**
CONTRACTOR INDIRECT COSTS		**35%**	**26.515**
PROJECT INDIRECT COSTS			**25.788**
II.- TOTAL INDIRECT COSTS			**52.303**
III.- CONTINGENCIES		**35%**	**67.115**
TOTAL INVESTMENT BUDGET		**KUS$**	**258.872**

Table 18.5-8: Summary process plant and tailings disposal capital costs

CAPITAL COST OF PROCESS PLANT AND TAILINGS DISPOSAL						
Alternative		Unit	Total Investment Budget by Year			
			Nominal	Year 0	Year 4	Year 8
Alternative A	Roasting	KUS$	286.081	280.963	2.559	2.559
Alternative B	POX	KUS$	283.898	278.780	2.559	2.559
Alternative C	BIOX	KUS$	258.872	253.754	2.559	2.559

18.6 Operating Cost Estimate

18.6.1 Mine Operating Costs

Mine operating costs were calculated using unit prices and consumption factors for the estimation. The basis for the calculation of labor costs includes the base salary and taxes for main categories of professionals and workers. Labor rates are presented in following Table 18.6-1.

Table 18.6-1: Labor rates

Category	US$/year
Superintendent	160,000
Chief of Department/Senior Geologist/Senior Engineer	63,000
Engineer and Geologist	40,000
Supervisor	35,000
Draftsman	20,000
Mechanics-Electrician	20,000
Skilled Operator	20,000
Semi-skilled operator	22,000
Assistants	17,000

These values were approximated from those used for the open pit feasibility study.

Labor costs charged to operation are considered as fixed costs depending on the manpower. A summary of the labor cost per year is shown in Table 18.6-2.

Table 18.6-2: Total mine labor cost

Year	Administrative (US$)	Direct (US$)	TOTAL (US$)
1	2,175	2,872	5,047
2	2,175	5,315	7,490
3	2,175	7,172	9,347
4	2,175	6,904	9,079
5	2,175	6,619	8,794
6	2,175	6,619	8,794
7	2,175	6,727	8,902
8	2,175	5,912	8,087
9	2,175	4,904	7,079
10	2,175	4,904	7,079
11	2,175	4,188	6,363
12	2,175	3,607	5,782
13	2,175	2,589	4,764
14	2,175	2,286	4,461

Prices for main consumables were obtained from other operations of similar conditions to Angostura Mine. Main consumables prices are presented as follow, Table 18.6-3:

Table 18.6-3: Consumable prices

DESCRIPTION	UNIT	US$
Explosives		
Detonating Cord 5gr/m	m	0.26
Softron	ea	0.10
Nonel MS/LP 6m	ea	2.24
Full Nº 8	ea	0.26
Anfo	kg	0.78
Emulsion 1 1/8 x 8"	ea	0.34
Steel Drills		
Bit R32 45 mm	ea	76.70
Hammer R38	ea	275.91
Couples R38	ea	67.74
Rods 12' R38/R32	ea	419.48
Crown 89 mm	ea	239.89
Bit 75 mm	ea	200.00
Hammer 114 mm	ea	5,000.00
Rods 1,5 m	ea	170.00
General		
Diesel	Lt	0.72
Lubricants	Lt	2.04
Electric Energy	Kwh	0.08
Industrial water	m3	1.00

These consumables prices are in line with the prices used in the DFS.

Mine support requirements were estimated based on the geotechnical recommendations made for the mine conditions. Item unit cost and recommendations are summarized in following Table 18.6-4:

Table 18.6-4: Support Recommendations & Cost

Bolt	**30.00**	**US$/bolt**	**A 1m x 1m grid is recommended, with 10 mm of shotcrete.**	
Grid	4.00	US$/m2		
Shotcrete	295.97	US$/m3		
Parameters		5x4	4x4	4x4p
Bolt	un	11.00	11.00	11.00
Grid	m2	9.43	9.14	9.14
Shotcrete	m3	0.94	0.91	0.91

According to previous data and equipment performance, unitary costs were estimated for production and developments. Tables 18.6-5 to 18.6-8

Table 18.6-5: Ramp & Access cost estimation.

5 m x4 m	(US$/m)	(US$/ton)
Drilling	125	2.5
Blasting	92	1.8
Loading	38	0.7
Transport	206	4.0
Support	739	14.5
Total	**1,200**	**23.5**

Table 18.6-6: Preparation cost estimation.

5 m x4 m	(US$/m)	(US$/ton)
Drilling	115	2.8
Blasting	82	2.0
Loading	48	1.2
Transport	218	5.3
Support	737	18.1
Total	1,200	29.4

Table 18.6-7: Production (Drifts) cost estimation.

5 m x4 m	(US$/ton)
Drilling	3.0
Blasting	2.1
Loading	1.6
Transport	6.0
Support	19.1
Total	31.9

Table 18.6-8: Production (Bench) cost estimation.

5 m x 4 m	(US$/ton)
Drilling	0.6
Blasting	0.4
Loading	1.6
Transport	6.0
Support	-
Total	8.6

The cost per tonne of ore produced has been estimated per year, generating the cost profile for the life of mine, the average cost for the project was also derived. The Table 18.6-9 shows the cost estimate per year, with an average cost of 40.4 US$/t.

Service costs of 2.3 US$/t allow for ventilation and dewatering of the mine. These items have not been calculated in detail for this study, given the preliminary nature of the designs. The dewatering of all the areas above the haulage levels will require low power using mostly gravity to transfer the water to these levels.

A contingency of 35% has been applied, mainly related with possibilities for changes in the geotechnical conditions of the mine. Table 18.6-9 presents the total operational costs.

Table 18.6-9: Total operational cost

		1	2	3	4	5	6	7	8	9	10	11	12	13	14	US$/t
Drilling	KUS$	430	1,279	1,807	1,770	1,743	1,724	1,778	1,740	1,666	1,144	1,098	926	533	418	**1.4**
Blasting	KUS$	291	866	1,224	1,199	1,180	1,168	1,204	1,179	1,128	775	744	627	361	283	**1.0**
Loading	KUS$	487	1,448	2,046	2,004	1,973	1,952	2,013	1,971	1,886	1,295	1,243	1,049	604	473	**1.6**
Transport	KUS$	1,798	5,351	7,559	7,404	7,291	7,213	7,437	7,281	6,970	4,786	4,593	3,875	2,230	1,749	**6.0**
Support	KUS$	1,921	5,717	8,076	7,911	7,790	7,707	7,947	7,779	7,447	5,114	4,908	4,140	2,383	1,869	**6.4**
Preparation	KUS$	1,249	3,561	4,406	4,004	2,219	2,920	3,365	3,595	3,330	3,052	2,959	2,521	1,416	559	**3.1**
Services	KUS$	2,119	2,119	2,119	2,119	2,119	2,119	2,119	2,119	2,119	2,119	2,119	2,119	2,119	2,119	**2.3**
Admin. personnel	KUS$	2,175	2,175	2,175	2,175	2,175	2,175	2,175	2,175	2,175	2,175	2,175	2,175	2,175	2,175	**2.4**
Direct personnel	KUS$	2,872	5,315	7,172	6,904	6,619	6,619	6,727	6,619	6,276	4,904	4,188	3,607	2,589	2,286	**5.7**
Sub-total	KUS$	13,343	27,833	36,583	35,490	33,110	33,598	34,766	34,458	32,998	25,365	24,027	21,039	14,411	11,932	**30.0**
Contingencies 35%	KUS$	4,670	9,741	12,804	12,422	11,588	11,759	12,168	12,060	11,549	8,878	8,410	7,364	5,044	4,176	**10.5**
Total	**KUS$**	**18,012**	**37,574**	**49,388**	**47,912**	**44,698**	**45,357**	**46,934**	**46,518**	**44,548**	**34,242**	**32,437**	**28,403**	**19,454**	**16,108**	**511,585**
Total (US$/t)		60	42	39	39	37	38	38	38	38	43	42	44	52	55	**40.4**
Total (US$/oz)		338	232	221	229	223	183	173	230	239	269	264	273	326	339	231

18.6.2 Plant Operating Costs

The following criteria have been applied by Alquimia for the processing operating cost estimated in Table 18.6-10.

- The operating cost estimate assumes a nominal plant throughput of 3,288 tonnes per day.
- The manpower estimate was according to the organization chart, previously detailed in the chapter 18.5. The applied salaries were based on Alquimia's experience with similar projects and include additional benefits and safety at work.
- Reagent consumption rates are calculated according to laboratory results and benchmarking of similar projects. Annual consumption rates take into consideration both unit consumption rates and the plant mass balances. Reagent costs were taken from world standards and from Greystar background.
- Power consumption was calculated in MWh per year, using the estimate power consumption per equipment and considering efficiency factors. Power from grid cost is assumed to be 81 US$/MWh.
- Cost of maintenance includes maintenance and materials and spare parts. The cost of maintenance is calculated to be 35% of power cost. The cost of spare parts corresponds to 3% of the main equipment investment cost.
- The operating cost estimate includes a miscellaneous cost for any cost considered in the operating cost calculation. This is estimated to be 15% of the global operating cost. On the other hand, is considered a cost of 3.5 dollars per tonne for the filter plant, transport and disposal of tailings.
- A contingency of 10% is considered.
- The average estimated processing costs is 26 – 27 US$ per tonne of ore fed to the plant. The Table 18.6-10 shows the operating costs breakdown per alternative.

Table 18.6-10: Estimated processing costs

		Roasting	POX	BIO-OX
Total operating cost	**US$/t**	**26,0**	**26,2**	**27,1**
Labour	US$/t	4,9	4,6	4,7
Power	US$/t	4,3	4,7	3,5
Reagents and consumables	US$/t	5,6	5,2	7,8
Maintenance	US$/t	2,7	3,1	2,3
Miscellaneous	US$/t	6,1	6,2	6,3
Contingency	US$/t	2,4	2,4	2,5

18.6.3 General and Administrative Costs

NCL estimated an administrative cost of US$5.0 per tonne, which corresponds approximately to US$ 6.0 M per year.

18.7 Markets

Over the life of the mine, the process plant will produce doré containing approximately 1.9 Moz of gold, 7.7 Moz of silver and 228 Klb of copper. The doré will be sold to a refinery for separation into gold and silver bullion.

Greystar is of the opinion that sales contracts that may be entered into with refiners are expected to be typical of and consistent with standard industry practice and are similar to contracts for the supply of doré elsewhere in the world.

18.8 Taxation

The Colombian income tax rate for a corporation is set at 33%. There is also an export tax on gold doré which is 5%. The general rate for the value added tax (VAT) is 16%.

Nevertheless of the figures above, the economic valuation developed for the underground scoping study is free of tax.

A royalty of 3.2% was applied to gold and silver revenues.

18.9 Financial Analysis

A preliminary evaluation has been carried out by NCL, upon the basis of the presented mine schedule and mine capital and operating costs.

The processing data used for this evaluation has been provided by Alquimia Conceptos S.A., including plant and infrastructure capital and operating costs, metallurgical recoveries and metals production. The detail of Alquimia estimates is described in section 16 of this report.

18.9.1 Basis of Analysis

Three different process scenarios have been considered by Alquimia:

- Roasting
- Pressure oxidation (POX)
- Bio-oxidation (BIOX)

In all of the options the main final product is metal Dore, with a content of 75% of gold and silver and 25% of copper. In the case of roasting, small productions of copper cathodes and sulfuric acid were also accounted and included in the economic evaluation.

Pre-tax NPV at 5% discount rate and IRR of the cash flows have been calculated for a gold price of 1,015 US$/oz and a silver price of 15.85 US$/oz. Higher prices were applied to the two initial years of the plan (1,170 US$/oz Au and 18.25 US$/0z Ag).

The Table 18.9-1 summarizes the main parameters used for the evaluation.

Table 18.9-1: Summary of Evaluation Parameters

Rec Au	92.7	%
Rec Ag	66.1	%
Au price yr 1 -2	1,170	US$/oz
Ag price yr 1 -2	18.25	US$/oz
Au price	1,015	US$/oz
Ag price	15.85	US$/oz
Au payable	99.9%	
Ag payable	99.7%	
G&A	5.0	US$/t
Selling cost		
R/C Au	0.75	US$/oz pay
T/C Dore	0.25	US$/oz dore
Freight	0.38	US$/oz dore
Royalty	3.2%	of gold/silver revenue
Cu Price	2.5	US$/lb
	5,512	US$/t
Acid Price	70	US$/t

The capital costs estimated by Alquimia are shown in the Table 18.9-2:

Table 18.9-2: Process & Infrastructure Capital Expenditure

			Nominal	Year 0	Year 4	Year 8
Alternative A	Roasting	KUS$	286,081	280,963	2,559	2,559
Alternative B	POX	KUS$	283,898	278,780	2,559	2,559
Alternative C	BIOX	KUS$	258,872	253,754	2,559	2,559

18.9.2 Results of Analysis

For the calculation of cash flows, the initial process capital has been distributed in two years, with 40% in the first and 60% in the second. These costs include a contingency of 35% over the total of direct and indirect costs.

The operating costs were also estimated by Alquimia, and are presented in Section 16. The average cost varies between 26.06 US$/t (Roasting) to 27.09 US$/t (BIOX).

The Table 18.9-3 summarizes the cash flow evaluation of the three different scenarios evaluated.

Table 18.9-3: Summary of Economic Evaluation

		Roasting	POX	BIOX
Dore Produced	Oz	12,983,907	13,040,538	12,995,233
Gold in dore	Oz	1,928,577	1,985,209	1,939,904
Silver in dore	Oz	7,725,719	7,725,719	7,725,719
Copper in dore	lb	228,316	228,316	228,316
Copper in cathodes	lb x 1000	17,758	17,758	
Sulfuric Acid	kt	881		
Mine Cost	US$/t	40.4	40.4	40.4
Process Cost	US$/t	26.02	26.25	27.09
G&A	US$/t	5.0	5.0	5.0
Selling Costs	US$/oz	5.00	4.89	4.97
Royalty	US$/oz	35.0	34.9	35.0
Cathodes Transport	US$/t Cu	70.0	70.0	
Total Cost	US$/oz	509.0	496.9	512.9
Initial Capital	KUS$	301,630	299,447	274,421
Mine	KUS$	20,667	20,667	20,667
Process & Infrastructure	KUS$	280,963	278,780	253,754
Total Capital	KUS$	506,462	504,279	479,253
Mine	KUS$	220,381	220,381	220,381
Process & Infrastructure	KUS$	286,081	283,898	258,872
NPV (5%)	KUS$	400,193	397,040	355,823
IRR	%	21.4%	21.5%	21.3%

Table 18.9-4 through Table 18.9-6 show the economic valuation details for Roasting, POX and BIOX options respectively.

Table 18.9-4: Cash Flow Summary – (Roasting Option)

ROASTING			-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14
Ore	Ton	12,649,292			301,100	896,100	1,265,772	1,239,883	1,220,927	1,207,855	1,245,454	1,219,201	1,167,129	801,480	769,163	648,846	373,462	292,920
Grade	Au [gpt]				5.51	5.62	5.50	5.24	5.11	6.38	6.76	4.55	3.92	4.94	4.96	4.98	4.97	5.04
	Ag [gpt]				17.33	13.25	12.30	12.56	13.38	19.03	22.86	68.50	108.59	21.41	21.93	20.74	19.37	15.86
	Cu [%]				0.0511	0.0447	0.0523	0.0659	0.0891	0.0846	0.0648	0.1421	0.1941	0.0878	0.0878	0.0911	0.0859	0.0917
Feed Insitu	Au [gr]	66,972,167			1,658,690	5,033,399	6,960,429	6,499,184	6,233,709	7,705,873	8,420,496	5,551,405	4,569,617	3,957,241	3,817,117	3,232,062	1,856,256	1,476,688
	Ag [gr]	385,639,549			5,217,655	11,869,649	15,568,829	15,573,518	16,340,150	22,990,331	28,474,488	83,513,746	126,734,347	17,155,983	16,867,248	13,454,890	7,233,201	4,645,514
	Cu [gr]	11,506,941,999			153,979,372	400,655,202	661,701,356	816,494,490	1,087,942,437	1,022,059,579	807,276,020	1,732,536,652	2,265,128,471	703,494,965	675,022,754	591,281,320	320,709,490	268,659,890
Recovery	Au [%]				93.5	91.4	89.9	89.7	89.6	89.4	89.3	89.6	89.3	88.9	88.6	88.4	88.4	88.4
	Ag [%]				71.2	66.0	63.5	62.8	62.9	61.7	61.7	63.4	61.5	61.1	60.9	60.7	60.7	60.7
	Cu [%]				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	0.9
Fine in Dore	Au [gr]	59,985,468			1,551,160	4,599,653	6,255,131	5,829,875	5,586,024	6,889,313	7,515,606	4,976,080	4,081,454	3,516,873	3,380,819	2,857,150	1,640,935	1,305,395
	Ag [gr]	240,296,735			3,712,364	7,829,076	9,890,204	9,776,139	10,274,996	14,175,417	17,581,049	52,965,241	77,952,456	10,487,417	10,267,609	8,170,979	4,392,628	2,821,160
	Cu [gr]	103,562,478			1,385,814	3,605,897	5,955,312	7,348,450	9,791,482	9,198,536	7,265,484	15,592,830	20,386,156	6,331,455	6,075,205	5,321,532	2,886,385	2,417,939
Dore Composition	% Au+AG	74%			79.2	77.5	73.1	68.0	61.8	69.6	77.5	78.8	80.1	68.9	69.2	67.5	67.6	63.1
	% Cu	26%			20.8	22.5	26.9	32.0	38.2	30.4	22.5	21.2	19.9	31.1	30.8	32.5	32.4	36.9
Recovery	Cu [%]	70			70	70	70	70	70	70	70	70	70	70	70	70	70	70
Fine in Cathode	Cu [t]	8,055			108	280	463	572	762	715	565	1,213	1,586	492	473	414	224	188
Acid production	H2SO4 [ton]	881,234			15,707	52,606	77,636	79,996	81,451	88,724	91,312	90,393	100,025	54,554	52,876	45,427	27,912	22,612
Dore	Oz	12,983,907			213,781	515,525	710,552	738,003	824,747	972,987	1,040,467	2,364,178	3,292,881	653,809	634,129	525,654	286,783	210,410
Au produced	Oz	1,928,577			49,871	147,882	201,107	187,435	179,595	221,497	241,632	159,985	131,222	113,070	108,696	91,859	52,757	41,969
Ag produced	Oz	7,725,719			119,355	251,711	317,977	314,310	330,349	455,750	565,244	1,702,872	2,506,229	337,178	330,111	262,703	141,226	90,702
Cu produced	Oz	3,329,611			44,555	115,932	191,468	236,258	314,803	295,740	233,591	501,321	655,430	203,561	195,322	171,091	92,799	77,739
Income Au	KUS$	1,986,170			58,291	172,849	203,920	190,056	182,106	224,594	245,012	162,222	133,057	114,651	110,216	93,144	53,495	42,556
Income Ag	KUS$	122,973			2,172	4,580	5,025	4,967	5,220	7,202	8,932	26,910	39,605	5,328	5,217	4,151	2,232	1,433
Total Income Dore	KUS$	2,109,143			60,462	177,429	208,944	195,023	187,327	231,796	253,944	189,132	172,661	119,980	115,433	97,295	55,727	43,990
Income Cu	KUS$	44,395			594	1,546	2,553	3,150	4,197	3,943	3,115	6,684	8,739	2,714	2,604	2,281	1,237	1,037
Income Acid	KUS$	61,686			1,100	3,682	5,435	5,600	5,702	6,211	6,392	6,328	7,002	3,819	3,701	3,180	1,954	1,583
Total Income	KUS$	2,215,224			62,156	182,657	216,932	203,773	197,226	241,950	263,450	202,143	188,402	126,513	121,738	102,757	58,918	46,609
Mine Cost	KUS$	511,585	0	0	18012	37574	49388	47912	44698	45357	46934	46518	44548	34242	32437	28403	19454	16108
	US$/t	40.4			59.82	41.93	39.02	38.64	36.61	37.55	37.68	41.61	41.61	42.72	42.17	43.77	52.09	54.99
Process Cost	KUS$	329,192			12,315	24,055	30,680	30,380	30,168	30,076	30,511	30,294	29,761	21,371	21,015	19,695	9,876	8,995
G&A	KUS$	63,246			1,506	4,481	6,329	6,199	6,105	6,039	6,227	6,096	5,836	4,007	3,846	3,244	1,867	1,465
R/C Au	KUS$	1,445			37	111	151	140	135	166	181	120	98	85	81	69	40	31
T/C Dore	KUS$	3,246			53	129	178	185	206	243	260	591	823	163	159	131	72	53
Freight	KUS$	4,934			81	196	270	280	313	370	395	898	1,251	248	241	200	109	80
Total selling	KUS$	9,625			172	436	598	605	654	779	837	1,609	2,173	497	481	400	220	164
Cathodes Transport	KUS$	564			8	20	32	40	53	50	40	85	111	34	33	29	16	13
	US$/oz	0.3																
	US$/oz eq	0.3																
Royalty	KUS$	67,493			1,935	5,678	6,686	6,241	5,994	7,417	8,126	6,052	5,525	3,839	3,694	3,113	1,783	1,408
	US$/oz	35.0			38.8	38.4	33.2	33.3	33.4	33.5	33.6	37.8	42.1	34.0	34.0	33.9	33.8	33.5
	US$/oz eq	31.3			36.2	36.1	31.2	31.0	30.8	31.0	31.2	30.3	29.6	30.7	30.7	30.7	30.6	30.6
Total cost	KUS$	981,705			33,947	72,243	93,713	91,377	87,673	89,718	92,674	90,655	87,953	63,991	61,505	54,885	33,217	28,152
	US$/oz	509.0			680.7	488.5	466.0	487.5	488.2	405.1	383.5	566.6	670.3	565.9	565.8	597.5	629.6	670.8
	US$/oz eq	455.1			635.0	459.7	437.5	454.0	450.0	375.5	356.2	453.7	471.9	512.0	511.4	540.6	570.6	611.3
Operational margin	KUS$	1,233,519			28,208	110,414	123,219	112,396	109,552	152,232	170,776	111,489	100,449	62,522	60,233	47,872	25,701	18,457
Capital Expenditure																		
Mine	KUS$	220,381	11,633	9,034	22,835	27,203	31,552	8,047	19,852	25,753	25,984	3,330	10,683	9,475	14,999	-	-	-
Process+ Infrast	KUS$	286,081	112,385	168,578				2,559				2,559						
Total	KUS$	506,462	124,019	177,611	22,835	27,203	31,552	10,607	19,852	25,753	25,984	5,890	10,683	9,475	14,999	-	-	-
Cash flow (B.T.)	KUS$	727,057	-124,019	-177,611	5,373	83,211	91,667	101,789	89,700	126,478	144,792	105,599	89,767	53,047	45,234	47,872	25,701	18,457
NPV (5%)	KUS$	$400,193																
IRR		21.4%																

Table 18.9-5: Cash Flow Summary (POX Option)

POX			-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14
Ore	Ton	12,649,292			301,100	896,100	1,265,772	1,239,883	1,220,927	1,207,855	1,245,454	1,219,201	1,167,129	801,480	769,163	648,846	373,462	292,920
Grade	Au [gpt]				5.51	5.62	5.50	5.24	5.11	6.38	6.76	4.55	3.92	4.94	4.96	4.98	4.97	5.04
	Ag [gpt]				17.33	13.25	12.30	12.56	13.38	19.03	22.86	68.50	108.59	21.41	21.93	20.74	19.37	15.86
	Cu [%]				0.0511	0.0447	0.0523	0.0659	0.0891	0.0846	0.0648	0.1421	0.1941	0.0878	0.0878	0.0911	0.0859	0.0917
Feed Insitu	Au [gr]				1,658,690	5,033,399	6,960,429	6,499,184	6,233,709	7,705,873	8,420,496	5,551,405	4,569,617	3,957,241	3,817,117	3,232,062	1,856,256	1,476,688
	Ag [gr]				5,217,655	11,869,649	15,568,829	15,573,518	16,340,150	22,990,331	28,474,488	83,513,746	126,734,347	17,155,983	16,867,248	13,454,890	7,233,201	4,645,514
	Cu [gr]				153,979,372	400,655,202	661,701,356	816,494,490	1,087,942,437	1,022,059,579	807,276,020	1,732,536,652	2,265,128,471	703,494,965	675,022,754	591,281,320	320,709,490	268,659,890
Recovery	Au [%]				94.7	93.4	92.4	92.3	92.2	92.1	92.0	92.2	92.1	91.8	91.5	91.4	91.4	91.4
	Ag [%]				71.2	66.0	63.5	62.8	62.9	61.7	61.7	63.4	61.5	61.1	60.9	60.7	60.7	60.7
	Cu [%]				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	0.9
Fine in Dore	Au [gr]	61,746,912			1,570,409	4,699,788	6,429,334	5,998,071	5,748,331	7,099,468	7,747,876	5,118,470	4,207,442	3,631,148	3,494,429	2,954,956	1,697,107	1,350,082
	Ag [gr]	240,296,735			3,712,364	7,829,076	9,890,204	9,776,139	10,274,996	14,175,417	17,581,049	52,965,241	77,952,456	10,487,417	10,267,609	8,170,979	4,392,628	2,821,160
	Cu [gr]	103,562,478			1,385,814	3,605,897	5,955,312	7,348,450	9,791,482	9,198,536	7,265,484	15,592,830	20,386,156	6,331,455	6,075,205	5,321,532	2,886,385	2,417,939
Dore Composition	% Au+AG	74.5%			79.2	77.7	73.3	68.2	62.1	69.8	77.7	78.8	80.1	69.0	69.4	67.6	67.8	63.3
	% Cu	25.5%			20.8	22.3	26.7	31.8	37.9	30.2	22.3	21.2	19.9	31.0	30.6	32.4	32.2	36.7
Recovery	Cu [%]	70			70	70	70	70	70	70	70	70	70	70	70	70	70	70
Fine in Cathode	Cu [t]	8,055			108	280	463	572	762	715	565	1,213	1,586	492	473	414	224	188
Dore	Oz	13,040,538			214,400	518,745	716,153	743,411	829,965	979,743	1,047,934	2,368,756	3,296,932	657,483	637,782	528,798	288,589	211,847
Au produced	Oz	1,985,209			50,490	151,102	206,708	192,842	184,813	228,253	249,100	164,563	135,272	116,744	112,348	95,004	54,563	43,406
Ag produced	Oz	7,725,719			119,355	251,711	317,977	314,310	330,349	455,750	565,244	1,702,872	2,506,229	337,178	330,111	262,703	141,226	90,702
Cu produced	Oz	3,329,611			44,555	115,932	191,468	236,258	314,803	295,740	233,591	501,321	655,430	203,561	195,322	171,091	92,799	77,739
Income Au	KUS$	2,044,188			59,014	176,612	209,599	195,539	187,398	231,445	252,584	166,864	137,164	118,377	113,920	96,333	55,326	44,013
Income Ag	KUS$	122,973			2,172	4,580	5,025	4,967	5,220	7,202	8,932	26,910	39,605	5,328	5,217	4,151	2,232	1,433
Total	KUS$	2,167,161			61,186	181,192	214,624	200,506	192,618	238,647	261,516	193,774	176,769	123,705	119,136	100,484	57,558	45,446
Income Cu	KUS$	44,395			594	1,546	2,553	3,150	4,197	3,943	3,115	6,684	8,739	2,714	2,604	2,281	1,237	1,037
Mine Cost	KUS$	511,585	0	0	18,012	37,574	49,388	47,912	44,698	45,357	46,934	46,518	44,548	34,242	32,437	28,403	19,454	16,108
	US$/t	40.4			59.82	41.93	39.02	38.64	36.61	37.55	37.68	41.61	41.61	42.72	42.17	43.77	52.09	54.99
Total Income	KUS$	2,211,556			61,780	182,738	217,176	203,656	196,815	242,590	264,630	200,458	185,508	126,419	121,741	102,765	58,795	46,483
Process Cost	KUS$	331,994			12,160	24,200	30,938	30,651	30,449	30,363	30,779	30,576	30,069	21,639	21,299	20,041	9,836	8,995
G&A	KUS$	63,246			1,506	4,481	6,329	6,199	6,105	6,039	6,227	6,096	5,836	4,007	3,846	3,244	1,867	1,465
R/C Au	KUS$	1,487			38	113	155	144	138	171	187	123	101	87	84	71	41	33
T/C Dore	KUS$	3,260			54	130	179	186	207	245	262	592	824	164	159	132	72	53
Freight	KUS$	4,955			81	197	272	282	315	372	398	900	1,253	250	242	201	110	81
Total selling	KUS$	9,703			173	440	606	613	661	788	847	1,616	2,178	502	486	404	223	166
Cathodes Transport	KUS$	564			8	20	32	40	53	50	40	85	111	34	33	29	16	13
	US$/oz	0.3																
	US$/oz eq	0.3																
Royalty	KUS$	69,349			1,958	5,798	6,868	6,416	6,164	7,637	8,369	6,201	5,657	3,959	3,812	3,215	1,842	1,454
	US$/oz	34.9			38.8	38.4	33.2	33.3	33.4	33.5	33.6	37.7	41.8	33.9	33.9	33.8	33.8	33.5
	US$/oz eq	32.2			37.0	37.0	32.1	31.9	31.7	31.9	32.1	31.3	30.9	31.7	31.7	31.7	31.7	31.7
Total cost	KUS$	986,441			33,816	72,512	94,160	91,831	88,130	90,234	93,195	91,091	88,398	64,384	61,914	55,337	33,238	28,201
	US$/oz	496.9			669.8	479.9	455.5	476.2	476.9	395.3	374.1	553.5	653.5	551.5	551.1	582.5	609.2	649.7
	US$/oz eq	458.6			638.5	463.0	439.4	457.0	453.7	376.9	356.9	460.1	482.2	516.0	515.3	545.6	572.8	614.7
Operational margin	KUS$	1,225,114			27,964	110,226	123,016	111,825	108,685	152,356	171,436	109,367	97,110	62,036	59,827	47,428	25,557	18,282
Capital Expenditure																		
Mine	KUS$	220,381	11,633	9,034	22,835	27,203	31,552	8,047	19,852	25,753	25,984	3,330	10,683	9,475	14,999	0	0	0
Process+ Infrast	KUS$	283,898	111,512	167,268				2,559				2,559						
Total	KUS$	504,279	123,146	176,302	22,835	27,203	31,552	10,607	19,852	25,753	25,984	5,890	10,683	9,475	14,999	-	-	-
Cash flow (B.T.)	KUS$	720,835	-123,146	-176,302	5,129	83,022	91,464	101,219	88,833	126,603	145,451	103,477	86,427	52,561	44,828	47,428	25,557	18,282
NPV (5%)	KUS$	$397,040																
IRR		21.5%																

Table 18.9-6: Cash Flow Summary (BIOX Option)

BIOX			-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14
Ore	Ton	12,649,292			301,100	896,100	1,265,772	1,239,883	1,220,927	1,207,855	1,245,454	1,219,201	1,167,129	801,480	769,163	648,846	373,462	292,920
Grade	Au [gpt]				5.51	5.62	5.50	5.24	5.11	6.38	6.76	4.55	3.92	4.94	4.96	4.98	4.97	5.04
	Ag [gpt]				17.33	13.25	12.30	12.56	13.38	19.03	22.86	68.50	108.59	21.41	21.93	20.74	19.37	15.86
	Cu [%]				0.0511	0.0447	0.0523	0.0659	0.0891	0.0846	0.0648	0.1421	0.1941	0.0878	0.0878	0.0911	0.0859	0.0917
Feed Insitu	Au [gr]	66,972,167			1,658,690	5,033,399	6,960,429	6,499,184	6,233,709	7,705,873	8,420,496	5,551,405	4,569,617	3,957,241	3,817,117	3,232,062	1,856,256	1,476,688
	Ag [gr]	385,639,549			5,217,655	11,869,649	15,568,829	15,573,518	16,340,150	22,990,331	28,474,488	83,513,746	126,734,347	17,155,983	16,867,248	13,454,890	7,233,201	4,645,514
	Cu [gr]	11,506,941,999			153,979,372	400,655,202	661,701,356	816,494,490	1,087,942,437	1,022,059,579	807,276,020	1,732,536,652	2,265,128,471	703,494,965	675,022,754	591,281,320	320,709,490	268,659,890
Recovery	Au [%]				93.7	91.8	90.4	90.2	90.1	89.9	89.8	90.1	89.9	89.4	89.2	89.0	89.0	89.0
	Ag [%]				71.2	66.0	63.5	62.8	62.9	61.7	61.7	63.4	61.5	61.1	60.9	60.7	60.7	60.7
	Cu [%]				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	0.9
Fine in Dore	Au [gr]	60,337,757			1,555,010	4,619,680	6,289,972	5,863,514	5,618,485	6,931,344	7,562,060	5,004,558	4,106,652	3,539,728	3,403,541	2,876,711	1,652,169	1,314,332
	Ag [gr]	240,296,735			3,712,364	7,829,076	9,890,204	9,776,139	10,274,996	14,175,417	17,581,049	52,965,241	77,952,456	10,487,417	10,267,609	8,170,979	4,392,628	2,821,160
	Cu [gr]	103,562,478			1,385,814	3,605,897	5,955,312	7,348,450	9,791,482	9,198,536	7,265,484	15,592,830	20,386,156	6,331,455	6,075,205	5,321,532	2,886,385	2,417,939
Dore Composition	% Au+AG	74.4%			79.2	77.5	73.1	68.0	61.9	69.6	77.6	78.8	80.1	68.9	69.2	67.5	67.7	63.1
	% Cu	25.6%			20.8	22.5	26.9	32.0	38.1	30.4	22.4	21.2	19.9	31.1	30.8	32.5	32.3	36.9
Dore	Oz	12,995,233			213,905	516,169	711,672	739,085	825,791	974,338	1,041,960	2,365,093	3,293,691	654,544	634,860	526,283	287,144	210,698
Au produced	Oz	1,939,904			49,995	148,526	202,227	188,516	180,638	222,848	243,126	160,900	132,032	113,805	109,426	92,488	53,118	42,257
Ag produced	Oz	7,725,719			119,355	251,711	317,977	314,310	330,349	455,750	565,244	1,702,872	2,506,229	337,178	330,111	262,703	141,226	90,702
Cu produced	Oz	3,329,611			44,555	115,932	191,468	236,258	314,803	295,740	233,591	501,321	655,430	203,561	195,322	171,091	92,799	77,739
Au Eq	Oz Eq	2,060,547			51,859	152,457	207,193	193,425	185,797	229,965	251,953	187,492	171,169	119,070	114,581	96,591	55,324	43,673
Income Au	KUS$	1,997,773			58,435	173,602	205,055	191,153	183,165	225,964	246,526	163,150	133,878	115,396	110,957	93,782	53,861	42,848
Income Ag	KUS$	122,973			2,172	4,580	5,025	4,967	5,220	7,202	8,932	26,910	39,605	5,328	5,217	4,151	2,232	1,433
Total	KUS$	2,120,746			60,607	178,182	210,080	196,120	188,385	233,166	255,458	190,060	173,483	120,725	116,173	97,933	56,093	44,281
Mine Cost	KUS$	511,585			18012	37574	49388	47912	44698	45357	46934	46518	44548	34242	32437	28403	19454	16108
	US$/t	40.4			59.82	41.93	39.02	38.64	36.61	37.55	37.68	41.61	41.61	42.72	42.17	43.77	52.09	54.99
Process Cost	KUS$	342,715			12,504	25,006	32,208	31,837	31,575	31,447	31,984	31,696	31,021	22,116	21,671	20,025	10,363	9,262
	KUS$	63,246			1,506	4,481	6,329	6,199	6,105	6,039	6,227	6,096	5,836	4,007	3,846	3,244	1,867	1,465
R/C Au	KUS$	1,453			37	111	152	141	135	167	182	121	99	85	82	69	40	32
T/C Dore	KUS$	3,249			53	129	178	185	206	244	260	591	823	164	159	132	72	53
Freight	KUS$	4,938			81	196	270	281	314	370	396	899	1,252	249	241	200	109	80
Total selling	KUS$	9,640			172	436	600	607	656	781	839	1,611	2,174	498	482	401	221	164
Royalty	KUS$	67,864			1,939	5,702	6,723	6,276	6,028	7,461	8,175	6,082	5,551	3,863	3,718	3,134	1,795	1,417
	US$/oz	35.0			38.8	38.4	33.2	33.3	33.4	33.5	33.6	37.8	42.0	33.9	34.0	33.9	33.8	33.5
	US$/oz eq	32.9			37.4	37.4	32.4	32.4	32.4	32.4	32.4	32.4	32.4	32.4	32.4	32.4	32.4	32.4
Total cost	KUS$	995,050			34,133	73,199	95,247	92,831	89,062	91,085	94,158	92,002	89,130	64,726	62,154	55,207	33,700	28,416
	US$/oz	512.9			682.7	492.8	471.0	492.4	493.0	408.7	387.3	571.8	675.1	568.7	568.0	596.9	634.4	672.5
	US$/oz eq	482.9			658.2	480.1	459.7	479.9	479.3	396.1	373.7	490.7	520.7	543.6	542.4	571.6	609.1	650.7
Operational margin	KUS$	1,125,696			26,474	104,982	114,833	103,288	99,323	142,081	161,300	98,058	84,353	55,999	54,020	42,726	22,393	15,865
Capital Expenditure																		
Mine	KUS$	220,381	11,633	9,034	22,835	27,203	31,552	8,047	19,852	25,753	25,984	3,330	10,683	9,475	14,999	-	-	-
Process+ Infrast	KUS$	258,872	101,502	152,253				2,559				2,559						
Total	KUS$	479,253	113,135	161,286	22,835	27,203	31,552	10,607	19,852	25,753	25,984	5,890	10,683	9,475	14,999	-	-	-
Cash flow (B.T.)	KUS$	646,443	-113,135	-161,286	3,639	77,779	83,282	92,682	79,471	116,328	135,316	92,168	73,671	46,524	39,021	42,726	22,393	15,865
NPV (5%)	KUS$	$355,823																
IRR		21.3%																

All the options show positive results. The option of Roasting shows better NPV and slightly lower IRR compared to the POX option due to the contribution of copper cathodes and sulfuric acid sales. Without that contribution this option results in an NPV of 332 MUS$ with a 19.3% IRR. It is important to ensure that this additional income is effective in order to obtain the return that this option shows.

18.10 Sensitivity Analysis

A sensitivity analysis was carried out to evaluate the effect on the NPV and IRR, varying the grade of the mineable resources (-5%), the metal price (±10%), the operating costs (+10%) and capital cost (+10%), obtaining the following results.

A 5% decrease in grades affects the results significantly as shown in the Table 18.10-1:

Table 18.10-1: Summary Sensitivity to Grade (-5%)

		Roasting	POX	BIOX
NPV (5%)	KUS$	332,626	327,637	287,889
IRR	%	19.13%	19.07%	18.70%

The mining method, as a function of the vein width and geotechnical conditions, and a detail estimate of the dilution will be relevant for any of the options applied.

Another relevant variable affecting the economic result of the project is metals price. A sensitivity analysis was carried out to +/- 10%, with the results shown in the following Tables, 18.10-2 and 18.10-3..

Table 18.10-2: Summary Sensitivity to Metal Price (-10%)

		Roasting	POX	BIOX
NPV (5%)	KUS$	283,086	276,615	238,052
IRR	%	17.73%	17.61%	17.09%

Table 18.10-3: Summary Sensitivity to Metal Price (+10%)

		Roasting	POX	BIOX
NPV (5%)	KUS$	517,300	517,464	473,593
IRR	%	24.69%	24.82%	24.88%

Additional sensitivities were carried out assuming 10% increase in operating cost and then 10% in capital costs. The results vary as shown in the Tables 18.10-4 and 18.10-5.

Table 18.10-4: Summary Sensitivity to Operating Costs (+10%)

		Roasting	POX	BIOX
NPV (5%)	KUS$	335,686	332,230	290,427
IRR	%	19.28%	19.27%	18.84%

Table 18.10-5: Summary Sensitivity to Capital Costs (+10%)

		Roasting	POX	BIOX
NPV (5%)	KUS$	357,659	354,708	315,807
IRR	%	18.85%	18.86%	18.63%

The project shows positive economic indices for all the scenarios evaluated and for the different sensitivity analyses performed.

Project sensitivity analysis indicates that the Project NPV is more sensitive to Feed Grade and Metal Price followed by OPEX and then CAPEX.

19.0 OTHER RELEVANT DATA AND INFORMATION

There are no additional data relevant to the Project.

20.0 INTERPRETATION AND CONCLUSIONS

In the opinion of the QPs, the following interpretations and conclusions are appropriate to the Project:

- The geologic understanding of the deposit settings, lithologies, and structural and alteration controls on mineralization is sufficient to support estimation of Mineral Resources.
- The mineralization style and setting is well understood and can support declaration of Mineral Resources.
- Work completed on the Project includes geochemical sampling, minor underground development, mineral resource estimation, core drilling including geotechnical, hydrological, confirmation and condemnation drill holes. Completed exploration programs were appropriate to the mineralization style.
- Sampling methods are acceptable, meet industry-standard practice, and are acceptable for Mineral Resource estimation purposes.
- The quality of the analytical data used in Mineral Resource estimation is reliable and sample preparation, analysis, and security are generally performed in accordance with exploration best practices and industry standards. Historic data used in estimation have been appropriately verified for support of estimation
- Mineral Resources, which were estimated using core drill data, have been performed to industry best practices, and conform to the requirements of CIM Definition Standards (2005).
- The unreacted solid from concentrate cyanidation is sent to conventional cyanidation together with flotation tails; the rich PLS obtained in conventional cyanidation is sent to CIC, copper elution, gold and silver elution and finally to EW and smelting, to produce Dore bars.
- To evaluate the feasibility of this, some metallurgical tests were performed with Angostura material, which was carried out by several laboratories. The main conclusions that can be reached from these tests are the following:
- Rougher flotation tests performed to sulfide ore showed gold and silver recoveries of 93% and 87%, respectively. Rougher tests performed to transitional ore showed gold and silver recoveries of 92% and 60%, respectively.
- Cyanidation of rougher tails obtained from sulfide flotation showed recoveries of 59% gold and 48% silver; when tails came from transitional flotation the recoveries increased to 92% gold and 60% silver.

- Alquimia assessment is that regrinding of the rougher concentrate to 37 microns, may be of benefit to the project. This should allow the production of a reduced quantity of cleaner concentrate and would reduce the capacity, size and cost, of the expensive refractory process unit operation. At the same time it would probably liberate more gold from the pyrite concentrate. Alquimia have assumed that a cyanidation recovery of 90% might be achieved on the cleaner tailings. However, all of this would require confirmatory testwork. Should a more conventional flotation concentrate be produced, without fine regrinding, it is Alquimia's view that the overall recovery would be very similar, in that more gold bearing material would be treated by the refractory process.
- Three alternatives can be used for sulfur oxidation: roasting, pressure oxidation and biooxidation.
- Roasting tests showed that a 91% of gold recovery can be reached.
- Pressure oxidation tests showed that a 96% of gold recovery can be reached.
- Biooxidation tests showed that a 92% of gold recovery can be reached.
- The production plan was prepared estimating productivities per area involved in a sector. Productivity estimation is a function of the stopes width involved and the mining method applied to the area. The mine production rate is 4,000 tonnes per day (tpd), maintained during 7 years.
- Loading will be made with 7 cubic yards Load Haul Dump units (LHD`s). LHD`s will load into low profile trucks. Hauling will be performed by 20 t trucks. Hauling activities will comprise ore hauling from the mine to the crushing station and backfill material hauling from the dump to the stopes.
- Fleet estimates indicate a maximum of 11 LHD units, 11 jumbos for development plus 3 bolting units, 3 DTH drilling rigs for bench drilling and 48 trucks.
- The total mine capital cost is 220 MUS$ for the life of the mine, with MUS$ 108 for equipment and 49 MUS$ for development. The initial capital is 20.6 MUS$. These numbers include a 35% contingency given the preliminary nature of the analysis
- Mine operating cost has been estimated at an average of 40.4 US$/t. Mine operating costs were calculated using unit prices and consumption factors

21.0 RECOMMENDATIONS

- Criteria for construction of vein wireframes should be reviewed and adjusted according to the requirements of an underground operation.

- To develop population analysis for different elements in the different areas of the deposit to better reflect the variations of the deposit, specially silver, copper, sulfur.

- To Re-evaluate the bulk density for high grade veins using the specific gravity measurements of the high grade population.

- To Re-evaluate the oxidation level model for the high grade veins population.

- To Use conditional simulation, for better quantification of uncertainty.

- Improve resource estimate classification by further drilling.

- Develop a more complete geotechnical analysis of the different areas selected to validate the recommendations for stopes dimensions and ground support.

- Develop a more detailed mine layout and include the design of the ventilation and dewatering systems, considering options for the treatment and use of the water extracted from the mine.

- Develop a more detailed analysis of the surface layout, including location of the mine portals and processing plant. The location and design of ore stockpiles needs to be considered.

- It is recommended to establish a geometallurgical model for the first five years of exploitation, focusing the exploration in the “Perezosa” and “Silencio – Los Laches” sulfide zones, which corresponds to the higher resources percentages. The metallurgical model can be established from representative samples, hopefully equidistant with each other, obtained from existing drilling campaigns, as well as programmed new ones.

- Further metallurgical testwork should be carried out, in order to study different parameters, such as the impact of grind size in flotation and cyanidation and the effect of the scheme of reagents addition and/or pulp density in gold and silver recoveries.

- Also it is recommended to study the effect of regrinding size in cleaner flotation and in cleaner-scavenger tails cyanidation.

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23.0 DATE AND SIGNATURE PAGE

The effective date of this Technical Report, entitled "MINERAL RESOURCE ESTIMATE AND PRELIMINARY ECONOMIC ASSESMENT FOR UNDERGROUND MINING.ANGOSTURA GOLD-SILVER PROJECT, SANTANDER, COLOMBIA" is February 28, 2011.

CERTIFICATE of AUTHOR – Rodrigo Mello

To accompany the technical report entitled: Mineral Resource Estimate and Preliminary Economic Assessment for Underground Mining, Angostura Gold-Silver Project, Santander, Colombia. April 25th, 2011

As one of the authors of this report about the Angostura property, pertaining to Greystar Resources Inc, I, Rodrigo Mello do hereby certify that:

1. I am independent consultant. Rua Eng. Sena Freire 193 Belo Horizonte, Brazil . Telephone: 5531-93910408.
2. Email: rodrigo.brito.mello@gmail.com

3. I hold the following academic qualifications:

B.Sc. (Geology)	Minas Gerais University 1985
Specialization (Computing)	Goiás Catholic University 1999

3. I am a registered Geologist with the Regional Council of Engineering, Minas Gerais and a member in good standing of the Australasian Institute of Mining and Metallurgy (AusIMM: 209332).

4. I have worked as a geologist and mineral resource analyst in the mineral industry for 25 years.

5. I am familiar with NI 43-101 and, by reason of education, experience and professional registration, I fulfill the requirements of a Qualified Person as defined in NI 43-101. My work experience includes 9 years as a exploration geologist or project manager, working in different terrains, including andean systems such as Angostura. I also have worked 9 years as a resource geologist working in the evaluation of gold, copper, zinc, nickel and silver deposits.

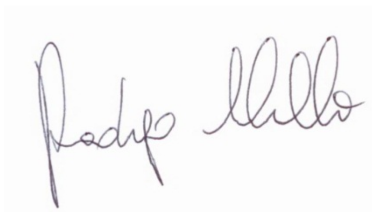
6. I have authored section 17 of this report and have reviewed all other sections, especially items linked to data quality and assurance.

7. I am not aware of any material fact, or change in reported information, in connection with the subject properties, not reported or considered by me, the omission of which makes this report misleading.

8. I am independent of the parties involved in the transaction for which this report is required, other than providing consulting services.

9. I have read NI 43-101 and, the Technical Report and I hereby certify that the Technical Report has been prepared in accordance with NI 43-101 and meets the form requirements of Form 43-101 F1.

Dated this 25th day of April, 2011



Rodrigo Mello

CERTIFICATE of AUTHOR – Carlos Guzman

As responsible for the overall preparation of the report entitled “Mineral Resource Estimate and Preliminary Economic Assessment for Underground Mining, Angostura Gold-Silver Project, Santander, Colombia. April 25th, 2011” prepared on behalf of Greystar Resources Limited (the “Technical Report”), I hereby state:

1. My name is Carlos Guzmán, Principal Mining Engineer and Project Director with the firm of NCL Ltda, Santiago, Chile. My address is General del Canto 235, Providencia, Santiago, Chile.
2. I am a practising mining engineer registered with the Australasian Institute of Mining and Metallurgy (MAusIMM 229036).
3. I am a graduate of the Universidad de Chile and hold a Mining Engineer title (1995).
4. I have practiced my profession continuously since 1995.
5. I have read the definition of “qualified person” set out in National Instrument 43-101 (the “Instrument”) and certify that by reason of my education, affiliation with a professional association (as defined in the Instrument) and past relevant work experience, I fulfil the requirements to be a “qualified person” for the purposes of the Instrument.
6. I most recently conducted a personal inspection of the Angostura Project in August 26, 2010.
7. I am responsible for the overall preparation of the Technical Report and specifically for Sections 18 of the Technical Report.
8. 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 that is required to be disclosed to make the Technical Report not misleading.
9. I am independent of Greystar Resources Limited pursuant to section 1.4 of the Instrument.
10. I have read the Instrument and Form 43-101F1 (the “Form”) and the Technical Report has been prepared in compliance with the Instrument and the Form.

Dated at Santiago, Chile, on April 27th , 2011.

Carlos Guzmán
NCL Ltda.
Mining Engineer (MAusIMM 229036)

Greystar Resources Limited
Angostura Gold–Silver Project
Santander Department, Colombia
NI 43-101 Technical Report

CERTIFICATE of AUTHOR – John Wells

To accompany the technical report entitled: Mineral Resource Estimate and Preliminary Economic Assessment for Underground Mining, Angostura Gold-Silver Project, Santander, Colombia. April 25th, 2011

As one of the authors of this report about the Angostura property, pertaining to Greystar Resources Inc, I, John Wells do hereby certify that:

1. I am independent consultant. Address-7445 Fleming Road, Vernon, BC, Canada V1H 1C1. Phone 1-250-549-7443.

2. I hold the following academic qualifications:
   B.SC. Mineral Engineering, Royal School of Mines, London, 1967.
   MBA-Business Administration, University of Sheffield, 1970.

3. I am an independent consulting metallurgical engineer, FSAIMM.

4. I have worked as a metallurgical engineer in the mining and mineral processing industry for 44 years.

5. I am familiar with NI 43-101 and, by reason of education, experience and professional registration, I fulfill the requirements of a Qualified Person as defined in NI 43-101. My work experience is 44 years in the mining and mineral processing industry, both in precious metals and base metals, with major operating companies as well as engineering/consulting companies. For the last five years I have worked as an independent consultant.

6. I have reviewed section 16 of this report.

7. I am not aware of any material fact, or change in reported information, in connection with the subject properties, not reported or considered by me, the omission of which makes this report misleading.

8. I am independent of the parties involved in the transaction for which this report is required, other than providing consulting services.

9. I have read NI 43-101 and, the Technical Report and I hereby certify that the Technical Report has been prepared in accordance with NI 43-101 and meets the form requirements of Form 43-101 F1.

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Greystar Resources Limited
Angostura Gold–Silver Project
Santander Department, Colombia
NI 43-101 Technical Report

Dated this 25th day of April, 2011

John Adrian Wells

CERTIFICATE of AUTHOR – Giovanny Ortiz

To accompany the technical report entitled: Mineral Resource Estimate and Preliminary Economic Assessment for Underground Mining, Angostura Gold-Silver Project, Santander, Colombia. April 25th, 2011

As one of the authors of this report about the Angostura property, pertaining to Greystar Resources Inc, I, Giovanny Ortiz do hereby certify that:

1. I am associated with, and have prepared and reviewed the study for Greystar Resources Ltd., Suite 1430-333 Seymour Street, Vancouver B.C. V6B 5A6 Canada, and Telephone 604 682 8212.

2. I hold the following academic qualifications:
   B.Sc. (Geology) — Universidad Industrial de Santander 1994
   Specialization (Management) — Universidad Autónoma de Bucaramanga 2004

3. I am a registered Geologist with the Colombian Council of Geology, Bogotá, Colombia and a member in good standing of the Australasian Institute of Mining and Metallurgy (AusIMM: 304612).

4. I have worked as exploration geologist, exploration management and mineral resource analyst in the mineral industry for 16 years.

5. I am familiar with NI 43-101 and, by reason of education, experience and professional registration, I fulfill the requirements of a Qualified Person as defined in NI 43-101. My work experience includes 13 years in Andean deposits of precious metals, as exploration geologist or project manager, and working in different terrains.

6. I have authored section Sections 7, 8, 9, 10, 11, 12 and 13 of this report.

7. I am not aware of any material fact, or change in reported information, in connection with the subject properties, not reported or considered by me, the omission of which makes this report misleading.

8. I am independent of the parties involved in the transaction for which this report is required, other than providing consulting services.

9. I have read NI 43-101 and, the Technical Report and I hereby certify that the Technical Report has been prepared in accordance with NI 43-101 and meets the form requirements of Form 43-101 F1.

Dated this 27th day of April, 2011

Giovanny Ortiz
Geologist (MAusIMM 304612)