Minerals and Metals

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Inmaculada Project, Peru Technical Report on the Feasibility Study

11 January 2012

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1 Summary

Ausenco Peru S.A.C (Ausenco) was contacted by International Minerals Corporation (IMZ) to assist in the production of a technical report summarizing the Feasibility Study (FS) that was completed by Ausenco on the Inmaculada project on behalf of Minera Suyamarca S.A.C (Suyamarca). IMZ announced on 12 October 2010 that it had signed a framework agreement with Hochschild Mining plc (Hochschild) where by Hochschild would hold a 60% interest (IMZ 40%) and become the operator of the Inmaculada Property. Ausenco was responsible for the overall management of the FS, as well as the design and cost estimation of the process plant and project infrastructure.

The purpose of this report is to provide an independent, National Instrument (NI) 43-101 compliant, technical report on the feasibility study, as well as an updated resource estimate on the Angela Vein deposit located on the Inmaculada property in the provinces of Parinacochas and Paucar de Sara Sara, Department of Ayacucho, Southern Peru (the “Property”).

The Angela Vein is a free milling, non-refractory gold/silver deposit with a mineral reserve of 7.8 million (M) metric tonnes (including allowance for mining dilution and ore loss) with grades of 3.37 grams per metric tonne (g/t) Au and 120.2 g/t Ag, corresponding to 0.84 M ounces (oz) Au and 30.14 Moz Ag. The objective of the project is to mine and process this deposit at a rate of 1.27 Mt/y. The expected life of processing operations from commencement of ramp-up is approximately 7 years. During this time, a total of 0.78 Moz of Au and 26.5 Moz of Ag will be produced.

The Feasibility Study (FS) determined that the project is economically viable and generates a non-discounted pre-tax net present value (NPV) of $323 million, and at a 5% discount rate, and a pre-tax NPV of $181 million. The pre-tax internal rate of return (IRR) is 18% and the payback period is 4.3 years. This analysis was completed using only the proven and probable mineral reserves with gold and silver prices of $1,100 and $18 per troy ounce respectively.

All currency amounts are expressed in United States dollars ($).

1.1 Resources and Reserves Estimate

At a revenue cutoff of $48.30 /t, the veins in the Quellopata area of the Property are estimated to have Measured and Indicated Resources of 7.07 Mt, with grades of 4.07 g/t Au and 144 g/t Ag, with an in situ metal content of 0.93 Moz Au and 32.79 Moz Ag. In addition, there are estimated to be 4.94 Mt of Inferred Resources, with grades of 3.91 g/t Au and 152 g/t Ag, with an in situ metal content of 0.62 Moz Au and 24.17 Moz Ag. These mineral resources were estimated using ordinary kriging within wireframes that define the geometry of eight veins, splays and branches.

These Mineral Resource Estimates have an effective date of 11 January 2012, and use all drill hole data available as of June 2011. A total of 326 core drill holes with a total length of approximately 102,867 m were drilled.

The mineral reserve estimation only considered the Angela Vein (not the minor splays and branches) and was based on the resource block model, mining method selected, geomechanical recommendations and potential stope sizes which assisted in the definition of the Mining Units (MU) to which dilution and ore loss was applied.

Measured and indicated resources were used as input for the respective conversion to proven and probable reserves, while the material corresponding to inferred resources was considered waste.

Total Inmaculada project reserves were estimated at 7.8 Mt of ore, with grades of 3.37 g/t Au and 120.2 g/t Ag, corresponding to 0.84 Moz Au and 30.14 Moz Ag. The reason for the mineral reserve tonnage being higher than the resource tonnage is due to the dilution expected during the mining of the vein, and not due to inclusion of any inferred resources.

1.2 Mine Geomechanical Conditions

Evaluation of the stability of rock excavations was developed using empirical and deterministic methodologies including;

- Maximum dimension for excavations and self-support time based on the rock quality classification Rock Mass Rating (RMR) of the hangingwall (HW) and footwall (FW) and the vein;
- Stability graph method to establish the hydraulic ratio; and
- 2D finite element stress-strain analysis for the excavations using Phase 2 software.

The mean RMR obtained from the geomechanical model indicated moderate to poor rock conditions. For the hanging and foot walls, the maximum unsupported excavation is 4 m with a self-supporting time of 2 days, while for the vein zones the expectation is 2.7 m of unsupported excavation for 7.2 hours. These conditions require immediate support in the vein zones after blasting activities. Accordingly adequate support specifications and stope sizes were defined, as well as sufficient backfill rate capacity.

1.3 Mining

The main design elements considered in the selection of mining method and ultimately the mine design were:

- Ore and host rock geomechanical characteristics;
- Dip and thickness of the Angela Vein;
- The production and productivity requirements of each mining method; and
- Ore dilution.

The geomechanical results indicated poor rock conditions in areas of the Angela Vein. As a result a conservative approach was taken to the mining method selected, as well as the stope size and sequencing. The sub level stoping mining method was applied in the central part of the vein (larger vein width) with a stope height of 5 m. The cut and fill method was applied in the north and south areas of the vein with stope heights of 4.5 m.

Based on geomechanical rock conditions and orebody geometry the sub level stoping zones were developed with sub levels of 9m vertical intervals to limit hangingwall exposure, drill hole deviation and dilution. In the cut and fill areas the accesses to the vein will be developed at 12.5 m vertical intervals. Both methods will use cemented backfill. The 2 km long vein has good continuity and sufficient height to allow a number of simultaneous mining sections to ensure a production of 3,506 t/d.

1.4 Processing and Metallurgy

A testwork program was developed by Ausenco for the Angela Vein deposit and representative samples were collected with the specific objective of verifying the technical feasibility and providing data for the preparation of the FS. The metallurgical testwork was completed at ALS Ammtec (Ammtec) in Perth, Australia.

Findings from the testwork indicated that the ore is moderately competent for semi-autogenous (SAG) milling, moderately hard and moderately abrasive. Gold and silver can be recovered from the ore using conventional cyanide leaching techniques with an optimum grind size of P_{80} 50 micron, 96 hours leaching and 1,500 mg/L sodium cyanide concentration.

The testwork included cyanidation leaching of bulk (20 kg) samples at the optimum conditions identified through the course of the program. The gold and silver extractions in these tests are summarized in

Table 1.1.

Table 1.1 - Bulk Cyanidation Testwork Results

Composite Identity	Test No	% Au Extraction 96 Hours	Au Leach Residue (g/t)	% Ag Extraction 96 Hours	Ag Leach Residue (g/t)	Consumption (kg/t)	
						Lime	NaCN
Domain Comp #1	HS25805	97.5	0.11	92.6	8.2	0.64	1.08
Domain Comp #2	HS25806	97.6	0.09	93.9	8.0	0.59	1.05
Domain Comp #3	HS25807	97.5	0.15	90.4	22	0.67	1.56

Gold and silver recoveries of 95.6% and 90.6% respectively were used for mineral reserve calculations and include reductions to accommodate scale-up and solution losses.

The testwork indicated that the leached slurry could be thickened to allow solid-liquid separation for recovery of precious metals through a conventional Merrill Crowe processing plant. Testwork also indicated that cyanide could be detoxified using the INCO SO_2/Air process prior to the tailings being discharged to the tailings storage facility (TSF), or filtered and used for paste backfill in the underground mine.

The unit operations used to model the plant throughput and metallurgical performance are well proven in the gold and silver processing industry. The overall approach was to provide a robust process plant flowsheet that could handle the variability in the metallurgical performance of the ores tested. The flowsheet incorporates the following major process operations:

- Primary crushing direct feeding the milling circuit via a stockpile;
- SAG mill grinding;
- Ball mill grinding;
- Leaching;
- Counter-current decant solution washing;
- Pregnant solution clarification and precious metal recovery by zinc precipitation;
- Refinery incorporating mercury retort and smelting facilities;
- Tailings detoxification, thickening and disposal;
- Thickened tailings filtration and paste backfill;
- Fresh and reclaim water supply; and
- Reagent preparation and distribution.

The process design criteria were developed from the testwork program and from benchmarked data for similar operations. The key criteria selected for the plant design are:

- An average throughput of 3,506 t/d for 365 days per year;
- Design availability of 91.3%, equivalent to 7,998 operating hours per year, with standby equipment in critical areas, and
- Sufficient plant design flexibility for treatment of all zones, at design throughput, within the Angela Vein as per testwork completed.

1.5 Capital Cost

Table 1.2 provides a summary of the overall capital costs estimated for the Inmaculada project starting on 1 January 2012 until ramp-up is completed and full scale production commences in Q1 2014. The costs are expressed in October 2011 United States dollars ($). There is no allowance for escalation. The estimated costs include all mining, site preparation, process plant, first fills, buildings, road works and permanent accommodation camp. The estimates are considered to have an overall accuracy of ±15% and assume the project will be developed on an Engineering, Procurement and Construction Management (EPCM) basis.

Table 1.2 - Overall Project Capital Costs by Area

Description	Total Cost ($M)
Process Plant	88.9
Process Plant Infrastructure	12.0
High Voltage Power Supply	13.5
Mine	69.6
Non Process Plant Infrastructure	36.6
Temporary Facilities	11.5
Indirects	35.1
Owners Costs	23.3
Contingency	24.9
Grand Total	**315**

1.6 Operating Cost

The operating cost estimate was developed on an annual process plant throughput basis of 1.28 Mt per year. The costs were divided into the key cost centers and all costs are as of the 4th quarter of 2011 (calendar year). The overall operating costs are shown in Table 1.3.

Table 1.3 - Unit Operating Cost Summary (based on 3,506 t/d milled)

Parameter	Unit	Value
Underground mine - Cut and Fill:	$/t ore milled	44.5

Parameter	Unit	Value
- Sub Level Stoping: - Average:		32.2 39.1
Processing	$/t ore milled	25.1
General and Administrative	$/t ore milled	8.70
TOTAL	**$/t ore milled**	**72.9**

1.7 Economic Analysis

Two economic analysis cases have been provided in this report. The "overall project economic analysis" indicates the parameters of the overall project. The "IMZ specific analysis" provides the economic parameters of the project specific to IMZ based on the IMZ – Hochschild terms of agreement as outlined in Section 4.2.

The overall project base case analysis resulted in a non-discounted net present value (NPV) of $323 million before tax and $194 million after tax. With a 5% discount applied the NPV before tax was $181 million and $90 million after tax. The overall project pre-tax payback period is 4.3 years with an IRR of 18%.

The sensitivity analysis indicated that the project is most sensitive to feed grade and metal price as shown in Table 1.4.

Table 1.4 - Overall Project Economic Sensitivity Summary

Case	Variable	Pre-tax NPV5% (M$)		
		-10% Variance	0% Variance	+10% Variance
Case 1 (base case)	Capital Cost	217	181	144
	Operating Cost	224	181	137
	Metal Price or Feed Grade	82	181	280

Table 1.5 below indicates IMZ's attributable production and economic parameters for the Inmaculada project based on the terms of the joint venture agreement.

Table 1.5 - IMZ's Attributable Production and Economic Parameters

Item	Units	
Average annual gold production	ounces/year	49,600
Average annual silver production	ounces/year	1,682,000
Average annual gold equiv. production[2]	ounces/year	78,000
Life-of-mine gold production	ounces	313,000
Life-of-mine silver production	ounces	10,600,000
Life-of-mine gold equiv. production[2]	ounces	488,000

Item	Units	
Initial capital	$ millions	91
Direct site costs[1]	$ per tonne processed	74
Direct site costs[1,3]	$ per ounce Au (with Ag credit)	133
Total cash operating costs[1,3,4]	$ per ounce Au (with Ag credit)	262
IRR Pre-tax/post-tax	%	26 / 21
Pre-tax /post-tax cash flow (non-discounted)	$ millions	136 / 95
Pre-tax/post-tax NPV, 5% discount rate	$ millions	85 / 57
Pre-tax/post-tax NPV, 8% discount rate	$ millions	63 / 40

1) Direct site costs include mining, processing and mine administration. Total cash operating costs include direct site costs plus estimates of the management fee, refining charges and government royalty (but do not include workers profit sharing which is 8% of net income).
2) Gold equivalents are estimated using a silver-to-gold ratio of 60:1 calculated by using the ratio of the base case metal prices.
3) By-product accounting subtracts the revenue generated by silver from the total operating costs to determine the cost per ounce of gold.
4) For comparative purposes, if IMZ had selected co-product accounting, the resulting cash operating costs would be $560/oz of gold and $9.15/oz of silver.

1.8 Project Implementation Plan

Project construction is expected to take 15 months due to climatic conditions (seasonal snow and ice) and the reduced productivity at 4,800 meters above sea level (masl).

The following critical key dates and milestones have been identified for the project. They are subject to receipt of environmental approvals, financing, award of contracts and meteorological conditions. Delays in meeting these target dates would likely result in a delay in project completion. At the time of this report basic engineering on the process plant was essentially completed, with recommendations made for purchase of all long lead items.

Commence placement of orders for long lead items……………………………………February 2012

Commencement civil works….…………………………………………………………….....May 2012

Complete detailed engineering…………………………………………………….....December 2012

Commence mechanical installation..……………………………………………….........March 2013

Complete civil works……………………………………………………………………….…May 2013

Commence commissioning……………………………………………………….….…November 2013

Production……………………………………………………………………………..….December 2013

1.9 Conclusions and Recommendations

The following are the key conclusions of this report:

- The Inmaculada deposit is extensively drilled and geologically well understood. The data set is robust and reliable. Development of the orebody, particularly within the measured region of the resource is now recommended to validate the resource model and to test mining assumptions.
- Mine geomechanical investigations indicated that the rock conditions and quality in the hanging and foot walls are classified as poor to fair, with the improved rock quality areas in the central zones.
- The mine design and support was based on the geomechanical model for the hanging and foot walls. This includes immediate support in the vein zones after blasting activities.
- The mine design utilizes sub level stoping in the central part of the vein (larger vein width) with a stope height of 5 m and cut and fill method was applied in the north and south areas of the vein with stope heights of 4.5 m.
- Results of the metallurgical testwork conclude that processing the Angela Vein Zone ore types can be successfully achieved using industry standard crushing, grinding and cyanide leaching technology.
- The process plant flowsheet and unit operations used to model the plant throughput and metallurgical performance are well proven in the gold and silver processing industry.
- There is sufficient capacity in the tailings storage facility (TSF) for 7.8 Mt of tailings. A tailings filtration and paste backfill plant will deliver cemented paste backfill to the underground mine, and hence reduce the overall TSF volume requirements for the life of mine.
- Surface water will be utilized to provide the raw water requirements with any excess water being discharged under controlled conditions to surrounding areas.

The following are the key recommendations of this report:

- The project advance to detailed engineering, procurement and construction.
- Complete detailed mapping and sampling of level development and construction of grade control models.
- Closer spaced drilling from a footwall position to determine if development of any parallel veins is warranted.
- Additional density measurements are required to establish whether or not there is a density-grade relationship, and to establish appropriate densities to be used in the splays, branches and minor veins.
- Design of the stope dimensions should be regarded as a first step in the design process and local adjustments to the design should be expected, depending upon actual conditions observed in the stopes.
- Once backfill properties are known, numerical modeling should be completed to assess the support effectiveness of the fill, as well as to assess if sill or rib pillars should be left for overall mine stability.
- Process plant layout and equipment sizing optimization was identified as required during the FS and subsequently completed during Basic Engineering. This was based on the latest testwork information that became available in the latter stages of the FS.
- A more specific process plant site geotechnical investigation program is recommended for detailed design of the structures foundations, particularly for heavy loads such as the grinding mills.

- Investigations should include the possibility to remove the raw water dam from the design given the high amount of water inflow to the mine and subsequent possible use within the processing plant.
- Further detailed testwork and modeling should be completed on the expected mine discharge water quality to determine if it meets the regulatory requirements for discharge to the environment.
- Complete an overall detailed project risk assessment in detailed engineering.

2 Introduction and Terms of Reference

Ausenco was contracted by IMZ to assist in the production of a Technical Report on the Feasibility Study on the Inmaculada project property in Peru. The gold/silver deposit targeted for mining is the Angela Vein located in the provinces of Parinacochas and Paucar de Sara Sara, Department of Ayacucho, southern Peru (the “Property”).

This report considers a mine that will produce 1.28 Mt/y of gold/silver bearing ore for processing through a conventional cyanide leaching process plant.

This technical report was written by the independent Qualified Persons (QPs) shown in Table 2.1. Personal visits to the Project site were conducted by Ian Dreyer on 20 July 2011 and Angel Mondragon on 15 March 2011.

Table 2.1 - Qualified Persons and Areas of Responsibility

Name of Qualified Person	Area of Responsibility
Clinton Donkin	1, 2, 3, 4, 5, 6, 13, 17, 18, 19, 21.1.1, 21.1.2, 21.1.4, 21.1.5, 21.2.1, 21.2.2, 21.3, 21.5, 21.6, 22, 23, 24, 25.2, 25.4, 25.5, 26.3, 26.4, 26.5, 26.7, 27, 28, 29
Ian Dreyer	7,8,9,10,11,12, 25.1, 26.1
Angel Mondragon	15,16, 21.1.3, 21.4, 25.3, 26.2
Tony Sanford	20, 26.6
Mohan Srivastava	14

Each QP in this report takes sole responsibility for their work as outlined in their QP Certificates reporting in Section 29.

Any previous technical reports or literature used in the compilation of this report are referenced in References (Section 27).

All units in this report are based on the International System of Units (SI), except industry standard units, such as troy ounces for the mass of precious metals. All currency values are in United States Dollars ($) unless otherwise stated. The project operating and capital costs were developed to an accuracy of ±15%.

This report uses abbreviations and acronyms common within the minerals industry. Table 2.2 identifies several important terms and abbreviations used in this report.

Table 2.2 - Abbreviations and Symbols

Term	Abbreviation or Symbol
average	ave
day	d
degree Celsius	°C
degree Fahrenheit	°F
diameter	dia

Term	Abbreviation or Symbol
foot	ft
gram	g
grams per liter	g/L
grams per metric tonne	g/t
hectare	ha
hour	h
inside diameter	ID
kilogram	kg
kilometer	km
kiloPascal	kPa
kilowatthour	kWh
life of mine	LOM
liter	L
maximum	max
megaPascal	MPa
meter	m
meter per second	m/s
meters above sea level	masl
meter per second squared	m/s^2
metric tonnes	t
metric tonnes per day	t/d
metric tonnes per hour	t/h
micron	µm
miles per hour	mph
Million tonnes per year	Mt/y
kilometers per hour	km/h
minimum	min
minute	min
mole percent	mol %
molecular mass (weight)	mol wt
parts per billion	ppb

Term	Abbreviation or Symbol
parts per million	ppm
pounds	lb
rock mass rating	RMR
run of mine	ROM
semi-autogenous grinding	SAG
second	s
specific gravity	SG
square meter	m^2
troy ounces	oz
unconfined compressive strength	UCS
volume by volume	v/v
weight (mass)	wt
weight (mass) percent	% w/w
weight by mass	w/w
weight by volume	w/v
year	y

3 Reliance on Other Experts

Preparation of this report is based upon public and private information provided by IMZ and information provided in various previous Technical Reports listed in Section 27 of this report. The authors believe that the information provided and relied upon for preparation of this report is accurate at the time of the report and that the interpretations and opinions expressed in them are reasonable and based on current understanding of mining and processing techniques and costs, economics, mineralization processes and the host geologic setting. The authors have made reasonable efforts to verify the accuracy of the data relied on in this report.

The results and opinions expressed in this report are conditional upon the aforementioned information being current, accurate, and complete as of the date of this report, and the understanding that no information has been withheld that would affect the conclusions made herein the authors reserve the right, but will not be obliged, to revise this report and conclusions if additional information becomes known to the authors subsequent to the date of this report.

The authors of this technical report are not qualified to provide extensive comment on legal issues associated with the Project. As such, portions of Section 4 dealing with the types and numbers of mineral tenures and licenses, the nature and extent of IMZ's title and interest in the Project, the terms of any royalties, back-in rights, payments or other agreements and encumbrances to which the Project is subject are descriptive in nature and are provided exclusive of a legal opinion.

Carlos Huaman of Ausenco Peru S.A.C was relied upon as a non-QP expert for review of the mine geomechanical sections of this report (Sections 1.2, 24.1, 25.2 and 26.2). He provided overall management and review of the mine geomechanical component of the Feasibility Study. He holds a master's degree in civil engineering specializing in geotechnical engineering and has 20 years of geotechnical engineering experience, with 10 years specific to rock mechanics.

Ausenco has also relied extensively on the accounting team from Hochschild and IMZ for the financial modeling and analysis discussed in Section 22. The taxation and depreciation information and calculations were provided to Ausenco and believed to be correct and in accordance with Peruvian laws and International Financial Reporting Standards (IFRS).

4 Property Description and Location

The Inmaculada Property is located in southern Peru within the provinces of Parinacochas and Paucar de Sara Sara in the Ayacucho Department situated at latitude 14°57'27"S and longitude 73°14'42"W (Figure 4.1). The Property is situated approximately 210 km southwest of the town of Cusco and 530 km southeast of Lima, the capital of Peru with elevations ranging between 3,900 and 4,800 masl in the Puquio-Caylloma Belt. The Property is located within the UTM coordinates in Table 4.1.

Table 4.1 - Inmaculada Property UTM Coordinates

Easting	Northing
677000	8358000
702000	8358000
677000	8341000
702000	8341000

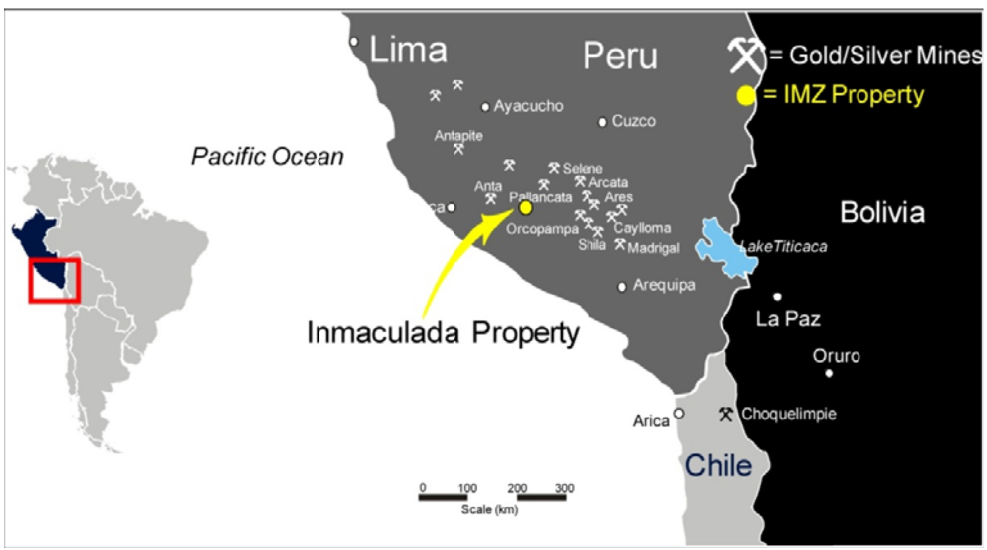

Figure 4.1 - Location Map for the Inmaculada Property

4.1 Permits

Four drilling permits have been acquired by IMZ or its predecessor Ventura Gold Corp (Ventura) during its tenure as operator of the Inmaculada joint venture project. These include:

- 2007: Declaración Jurada Proyecto Inmaculada. Permit to drill using 5 platforms;
- 2008: Evaluación Ambiental Proyecto Inmaculada Categoría C. Permit to drill using 15 platforms;
- 2008: Estudio de Impacto Ambiental Semidetallado Categoría II Proyecto Inmaculada. Permit to drill using 67 platforms. Expired November 2009;
- 2010: Segunda Modificatoria del Estudio de Impacto Ambiental Semidetallado Categoría II Proyecto Inmaculada. Permit to drill 79 platforms to expire in August 2012; and
- 2011: Tercera Modificatoria del Estudio de Impacto Ambiental Semidetallado Categoria II Proyecto Inmaculada. Permit to drill using 39 platforms to expire in July 2013.

4.2 Property Ownership and Agreements

Title to the Inmaculada Property is held by Minera Suyamarca S.A.C. (Suyamarca). This company (also the operator as well of the Hochschild/IMZ Pallancata mine) is a joint venture (JV) between IMZ and Hochschild (60% Hochschild and 40% IMZ). Suyamarca acquired Minera Quellopata S.A.C. the operating company that was established in August 2009 under the terms of an option and joint venture JV agreement between Ventura and Hochschild.

On 12 January 2010, IMZ acquired the Inmaculada property as part of the closing of a business transaction to acquire Ventura.

On 23 December 2010, IMZ signed a definitive agreement with Hochschild to fast track production at the Inmaculada project, whereby Hochschild committed to build a mining operation at Inmaculada with a process capacity of 3,000 tonnes per day (unless the parties agreed that such capacity was not optimal) by December 2013, subject to any unforeseen delays under defined “force majeure” conditions. Hochschild will provide 100% of the initial $100 million of funding required to complete a feasibility study and the planning, development and construction of a mining operation at Inmaculada. Any subsequent expenditures will be funded 60% by Hochschild and 40% by IMZ.

Below are the principal terms of the agreement:

- Hochschild paid to IMZ $15 million in cash on closing of the transaction;
- Hochschild made an equity investment in IMZ of $20 million in the form of a private placement at a price of CDN$5.525 per share;
- Hochschild will provide 100% of the initial $100 million of funding required for the planning, development and construction of a mining operation at the Angela Vein deposit at Inmaculada. Any subsequent expenditures will be funded 60% by Hochschild and 40% by IMZ;
- IMZ is no longer required to complete a feasibility study at Inmaculada, nor to issue 200,000 common shares to Hochschild;
- Hochschild committed to build a mining operation at the Angela Vein deposit with a process capacity of 3,000 t/d (unless the parties agree that such capacity was not optimal) within three years of closing of the transaction, subject to any unforeseen delays out of the control of Hochschild;

- If Hochschild fails to achieve the process capacity within the three-year period, then Hochschild must make quarterly pre-payments to IMZ during the period of any delay based on the parties' joint estimate of IMZ's 40% share of income / cash flows that would have been generated if production had started on schedule;
- Hochschild will be the operator of the project. Upon commencement of commercial production, Hochschild will charge the JV company a 7.0% management services fee based on the aggregate operating costs incurred by the JV during such mining operation;
- The management fee currently charged by Hochschild for the Pallancata Mine was reduced from 10.0% to 7.0%;
- IMZ and Hochschild contributed to the Inmaculada joint venture their respective ownerships in the Pacapausa property (80% Hochschild / 20% IMZ), located adjacent to the Pallancata Mine and the Puquiopata property (100% IMZ), situated on the northern edge of the Inmaculada Property. These contributions resulted in both properties being owned 60% by Hochschild and 40% by IMZ; and
- A minimum of 20,000 m of drilling per year for the first three years following the closing of the transaction must be carried out for evaluation of exploration targets outside of the main Angela Vein deposit. The drilling program will be funded 60% by Hochschild and 40% by IMZ.

4.3 Property Description and Tenure

The Inmaculada Property consists of 33 mining concessions with a total area of 20 799.46 ha (Figure 4.2).

The claims are in good standing until 30 June 2012, at which time fees and penalties estimated at US$224,404 must be paid (Table 4.2). As a result of the exploration expenditures made by Suyamarca during 2011, most of the "penalties" will not be required to be paid (Table 4.2). All of the concessions are held 100% by Suyamarca.

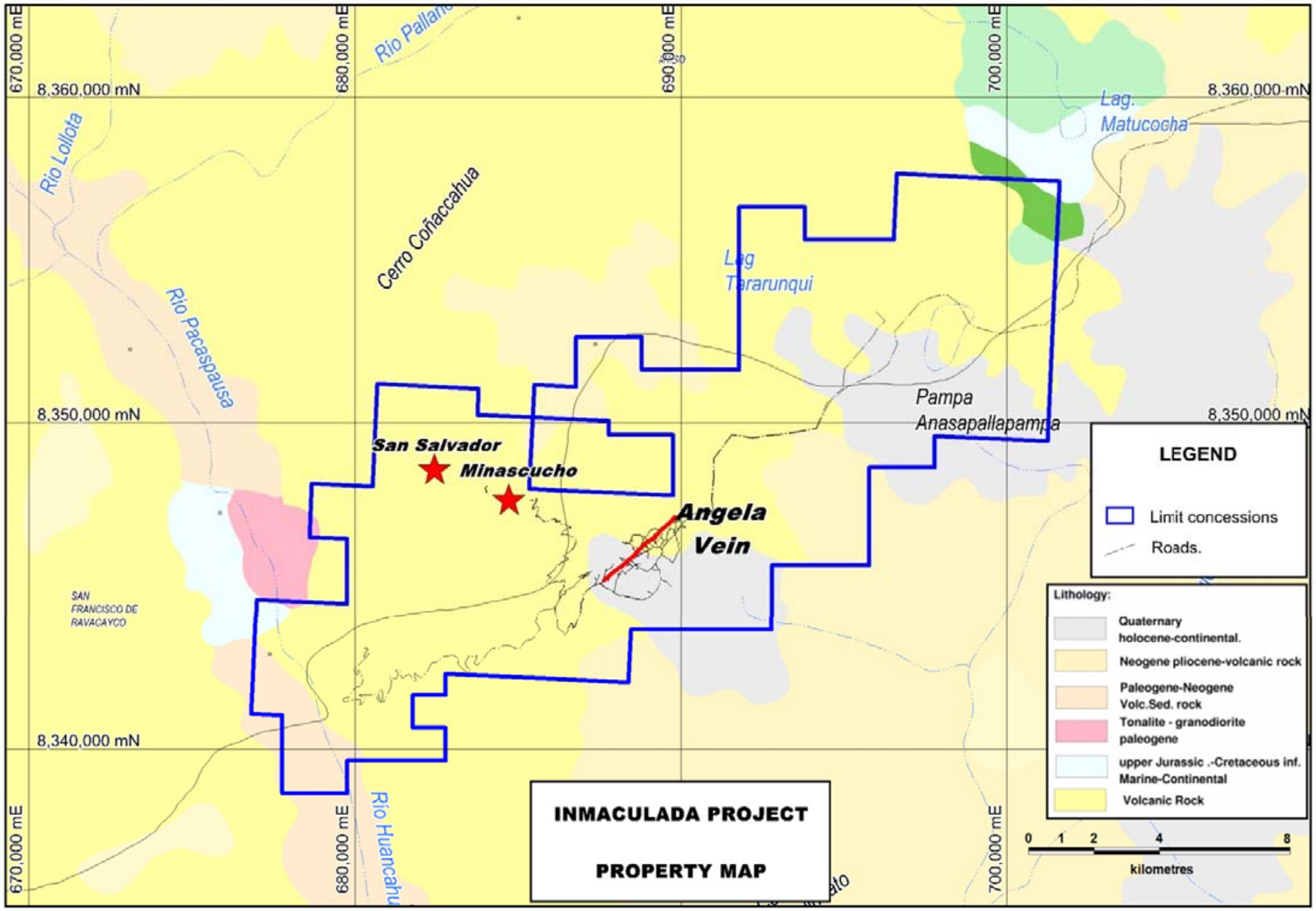

Figure 4.2 - Inmaculada Property Mining Concessions

Table 4.2 - Inmaculada Mining Concessions

Concession	Code	Area (ha)	Fees Paid 2011 (US$)	Penalties Paid 2011 (US$)	Fees & Penalties Estimated 2012 (US$)	Owner
CANCALLA 2	10584095	823.3169	$2,469.95	$16,466.34	$18,936.29	Suyamarca
INMACULADA 20 2006	10193006	999.9983	$2,999.99	$19,999.97	$2,999.99	Suyamarca
INMACULADA 2009-1	010170409	467.6250	$ 1,402.88	NA	$1,402.88	Suyamarca
INMACULADA 2009-2	010179609	555.3603	$ 1,666.08	NA	$1,666.08	Suyamarca
INMACULADA 2009-3	010218909	363.5240	$ 1,090.57	NA	$1,090.57	Suyamarca
INMACULADA 2009-4	010219009	165.6140	$ 496.84	NA	$496.84	Suyamarca
INMACULADA 2010-1	010204610	299.9991	NA	NA	$900.00	Suyamarca
INMACULADA 2010-2	010219910	1,000.0000	NA	NA	$3,000.00	Suyamarca
INMACULADA 2010-3	010219810	1,000.0000	NA	NA	$3,000.00	Suyamarca
INMACULADA 2010-4	010292910	900.0000	NA	NA	$2,700.00	Suyamarca
INMACULADA 67	010141704	300.0000	$900.00	NA	$2,700.00	Suyamarca
INMACULADA 77	010039305	177.3728	$532.12	NPPA	$532.12	Suyamarca
INMACULADA 78	010127005	250.4765	$751.43	NA	$751.43	Suyamarca
INMACULADA 80	010100806	4.7300	$14.19	NA	$14.19	Suyamarca
INMACULADA N°13	10010705X01	449.9998	$1,350.00	NPPA	$10,350.00	Suyamarca
INMACULADA N°14	10010706X01	449.9998	$1,350.00	NPPA	$10,350.00	Suyamarca
INMACULADA N°15	10010707X01	450.0016	$1,350.00	NPPA	$10,350.03	Suyamarca
INMACULADA N°18	10010854X01	999.9983	$2,999.99	NPPA	$22,999.96	Suyamarca
INMACULADA N°3	10010695X01	1000.0034	$3,000.01	$20,000.07	$23,000.08	Suyamarca

Concession	Code	Area (ha)	Fees Paid 2011 (US$)	Penalties Paid 2011 (US$)	Fees & Penalties Estimated 2012 (US$)	Owner
INMACULADA N°30	10011032X01	1000.0263	$3,000.08	$20,000.53	$23,000.61	Suyamarca
INMACULADA N°31	10011172X01	400.0001	$1,200.00	$8,000.00	$9,200.00	Suyamarca
INMACULADA N°33	1011457XX01	976.5025	$2,929.51	NPPA	$22,459.56	Suyamarca
INMACULADA N°34	1011457YX01	373.9997	$1,122.00	NPPA	$8,601.99	Suyamarca
INMACULADA N°6	10010698X01	999.3311	$2,997.99	$19,986.62	$22,984.61	Suyamarca
INMACULADA Nº 38	10197205	399.9985	$1,200.00	NA	$1,200.00	Suyamarca
MINASCUCHO 2007A	10035007	999.9981	$2,999.99	NA	$2,999.99	Suyamarca
MINASCUCHO 2007B	10034907	824.8503	$2,474.55	NA	$2,474.55	Suyamarca
PUQUIOPATA 2005	010218505	940.0106	$2,820.03	NA	$2,82003	Suyamarca
QUELLOPATA	010004999	162.1163	$486.35	NPPA	$3,728.68	Suyamarca
QUELLOPATA 2008A	010021908	564.6105	$1,693.83	NA	$1,693.83	Suyamarca
TARARUNQUI 2007B	010035207	999.9983	$2,999.99	NA	$2,999.99	Suyamarca
TARARUNQUI 2007A	010035107	999.9994	$3,000.00	NA	$3,000.00	Suyamarca
INMACULADA 2011-1	010314811	500.0000	$1,500.00	NA	$0.00	Suyamarca

* NPPA – Payment not required as minimum investment reached

4.4 Property Boundaries

The concessions which comprise the Inmaculada Property were first located by Mitsui Mining Corporation (Mitsui) using the old Peruvian claim-staking system, by which claims were demarcated by reference to local topographic points. In 1992, Peru adopted a new claim-staking system using UTM coordinates. At that time, all pre-existing mining concession corner points were converted to UTM coordinates. No surveys of the Property boundaries have been performed.

4.5 Peru's Mining Title Laws

Under Peruvian law, the right to explore for and exploit minerals is granted by way of mining concessions. A Peruvian mining concession is a property right, independent from the ownership of land on which it is located. The concession can be defended against possible claim by third parties, transferred or sold, leased, mortgaged and may be inherited in families. In general, a mining concession may be the subject of any transaction or contract. Government authorization is not required for this purpose.

According to the relevant mining law (*Ley del Catastro Minero Nacional, Ley No. 26615*), the basic unit for newly claimed mineral concessions is 100 ha to a maximum of 1,000 ha. The concession is irrevocable and indefinite as long as its holder fulfills the obligations prescribed by law to maintain them. There is no limit to the number of concessions that may be held by a company or individual.

Since the year 2001, the concession holder has had the obligation to pay an annual rent (derecho de vigencia) of US$3.00 per hectare by June 30 of each year. The annual obligation to pay concession rights can be postponed for one year but if it is not paid for two consecutive years then the concession expires.

According to the General Mining Act (the 'Mining Act'), which single unified text was approved by Supreme Decree No. 14-92-EM (Texto Único Ordenado de la Ley General de Minería) the mining concession holder must sustain a minimum level of annual commercial production of US$100 per hectare in gross sales within six years of the grant of the concession. If the mining concession has not been put into production within that period, then the mining concession holder must make an additional payment called Penalty (*Penalidad*) of US$6.00 per hectare for the 7th through 11th year following the granting of the mining concession and of US$20.00 per hectare thereafter. The mining concession holder shall be exempted from the Penalty if the investment made during the previous year was 10 times the Penalty (i.e.US$60 per hectare per year for 7th through to 11th years). The above mentioned applies for mining concessions granted before 10 October 2008. Pursuant to the Legislative Decrees No. 1010 and No. 1054, all concessions granted after such date will be ruled by the following: the mining concession holder must sustain a minimum level of annual commercial production of 1 Unidad Impositiva Tributaria (UIT) per hectare in gross sales within 10 years of the grant of the mining concession (the UIT is updated each year, and its value for 2009 is Peruvian sol (S/.) 3,350, which is approximately US$1,180). If the mining concession has not been put into production within that period, then the mining concession holder must pay an additional penalty of 10% of the UIT per hectare for the 11th through 15th year following the granting of the mining concession and, to avoid the expiration of the mining concession, a payment of the same penalty per hectare (10% of the UIT) must be made plus a minimum level of investments (i.e. exploration or development) of 10 times the penalty for the 16th through 20th year following the granting of the mining concession. Thereafter, if the mining concession holder does not put the mining concession into production the mining concession will expire. According to the Supreme Resolution No. 054-2008-EM the changes made by the Legislative Decrees above mentioned will be in force for the mining concessions granted before 10 October 2008.

Peruvian law also states that no payment is required on the portion of overlapping hectares with another concession.

4.6 Surface Rights

Suyamarca holds a 25-year lease from the community of Huallhua, on 870 ha of land for a period of 25 years effective 25 November 2009.

4.7 Environmental Liabilities

The reader is referred to Section 20 for further information on environment considerations.

5 Accessibility, Climate, Local Resources, Infrastructure & Physiography

The following section is sourced and summarized from the previous technical report on the Property by P&E Mining Consults Inc. (2010).

5.1 Accessibility

The total travel time from Lima to the Inmaculada Property is approximately 15.5 hours. The Property is accessible from Lima by means of the Pan-American highway to Nazca, then as far as Iscahuaca via the east-bound Nazca-Cusco highway, then by dirt road from Iscahuaca to the Huanacmarca turn-off and finally via dirt road through the communities of Sauricay and Sorani. Alternatively, access can be gained by travelling west on the Nazca-Cusco highway from Cusco, through Urubamba and Chalhuanca, to Iscahuaca and following the same gravel roads to the Property. The closest hotel to the Property is in Chalhuanca. Travel time by road from Cusco to Chalhuanca is about 5.5 to 6 hours, with a further hour travel to Iscahuaca. Cusco is served by several daily jet flights from Lima.

5.2 Climate

The climate in the area is characteristic of high altitude - ‘Puna’ climatic region in the Andes, with rain and snow between November and April, followed by a dry season between May and October.

5.3 Local Resources and Infrastructure

Laborers for the project are readily available from the nearby communities. People from the local villages have been used for various duties at the project. Peru has an adequate supply of university educated geologists to manage the exploration program.

The Property is located close to high tension power lines of the Electro Sur del Peru electric system, which reportedly can supply sufficient power for any mining operation. Hochschild currently operates several mines in the region.

5.4 Physiography

The Property is located in the ‘Puna’ region of Peru, with elevations on the Property varying between 3,900 and 4,800 masl. Surface topography is generally moderate in the immediate vicinity of the Angela Vein to locally very steep in the region. Vegetation is sparse consisting of grasses (ichu and pajonal).

6 History

6.1 Introduction

Exploration on the Inmaculada Property commenced in 1990 by Mitsui. The Property was sold and the concessions were transferred to Hochschild in 1992. Hochschild performed geological mapping, rock chip and soil sampling, geophysical surveys and core drilling on the Property between 1998 and 2005. In 2007, Ventura Gold started it's earn in process by drilling 15,000 m. IMZ exploration programs subsequent to the acquisition of Ventura Gold on 12 January 2010 are described in Section 9 of this report.

The following section is sourced and summarized from the previous technical report on the Property by P&E Mining Consultants Inc. (2010).

Exploration on the Inmaculada Property is summarized in Table 6.1.

Table 6.1 - Historical Exploration on the Inmaculada Property, 1990-2009

Year	Company	Exploration
1990 – 1992	Mitsui	Nature and extent of work not known. Property sold and concessions transferred to Hochschild.
1994	Hochschild / LAC Minerals	Hochschild signed a JV agreement with LAC Minerals. No work performed.
1995	Hochschild / Compañía North Minera S.A. (North)	Hochschild signed a JV agreement with North. Nature and extent of work not known. Property reverted to Hochschild at end of first year.
1998 – 2005	Hochschild	Through Hochschild subsidiaries Minera Ares S.A.C. and Minera Argento S.R.L. performed the following: • Regional mapping and sampling (1:25,000) • Construction of access road from Sauricay • Geological mapping of Tararunqui, Quellopata, Minascucho and San Salvador (1:5,000) • Rock chip sampling at Tararunqui (293 samples), Quellopata (1,665 samples), Minascucho – San Salvador (210 samples). • Soil sampling (186 samples). • Geophysical surveys: magnetometer (51.77 line-km), IP/Resistivity (29.2 line-km) and Gradient IP (21 line-km) at 3 targets. • DDH at Tararunqui (11 holes at 1,479.29 m), Quellopata (31 holes at 7,188.40 m) and Minascucho (2 holes at 440 m)
2007-2009	Ventura Gold	Core drilling focused on delineating the mineral resource of the Angela Vein, completed 99 core drill holes for a total of 25,590m

6.2 Historical Mineral Resource and Reserve Estimates

6.2.1 2009 Mineral Resource Estimates

In January 2009 Ventura announced an initial independent inferred mineral resource estimate of 483,000 contained ounces of gold and 16.6 million contained ounces of silver at the Angela Vein at the Inmaculada gold-silver project in southern Peru.

The inferred resource ounces are contained within an estimated 3.7 million tonnes at an average grade of 4.0 g/t gold and 139 g/t silver. This National Instrument 43-101 compliant resource estimate is based on approximately 15,000 m of core drilling and assay data obtained through December 31, 2008 and was calculated by Micon International of Toronto Canada.

6.2.2 2010 Mineral Resource Estimate

In February 2010, IMZ reported an updated mineral resource estimate for the Inmaculada project comprising the following resources on a 100% Project basis:

- Indicated Resource: 154,000 ounces gold and 4.9 million ounces silver (contained within 1.2 million tonnes at an average grade of 3.9 g/t gold and 122 g/t silver; and
- Inferred Resource: 512,000 ounces gold and 22.1 million ounces silver (contained within 4.7 million tonnes at an average grade of 3.4 g/t gold and 147 g/t silver.

The Inmaculada mineral resource estimate was classified in accordance with CIM guidelines by FSS Canada's R. Mohan Srivastava (P.Geo) and the estimate has an effective date of 3 February 2010. The mineral resource estimate is based on the results of 84 core drill holes for approximately 25,000 m of drilling, which have defined a strike length to the Angela Vein mineralization of over 1,500 m (and which is still open to the northeast) and a vertical extent of over 300 m.

6.2.3 2010 Mineral Resource Estimate and Preliminary Assessment

In September 2010, IMZ announced the results of an independent Preliminary Assessment (scoping study) and an updated, increased mineral resource estimate for the Angela Vein deposit at the Inmaculada project.

Results of the scoping study for the Project indicated that at base case gold and silver prices of $1,000/oz and $17/oz respectively and a 3,000 t/d throughput, an underground mining project could return a pre-tax non-discounted cash flow of approximately $660 million based on an anticipated Scoping study diluted mine production of 8.0 Mt at a grade of 3.8 g/t gold and 137 g/t silver.

The independent scoping study was overseen by P & E Mining Consultants Inc. of Brampton, Ontario Canada (P&E), with Golder Associates Peru S.A. (Golder) responsible for the tailing disposal facility and fresh water supply and FSS Canada (FSS) responsible for the updated resource estimate.

The results of the scoping study are summarized in Table 6.2.

Table 6.2 - Inmaculada Angela Vein Scoping Study (100% Project Basis, all in US Dollars)

Item	Units	
Base Case Gold price	$ per ounce	1000
Base Case Silver Price	$ per ounce	17
Initial Mine life	years	7.5
Average annual gold production [6]	ounces/year	117,000
Average annual silver production [6]	ounces/year	4,000,000
Average annual gold Eq. production	Au Eq ounces/year	180,000
Life-of-mine gold production [6]	ounces	858,000
Life-of-mine silver production [6]	ounces	29,300,000
Life-of-mine gold Eq. production	Au Eq. ounces	1,346,000
Plant processing rate (~3,000 t/d)	tonnes/year	1,095,750
Metallurgical recovery – gold	%	88
Metallurgical recovery – silver	%	83
Initial capital [2]	$ millions	168
Total Cash operating cost [3,]	per tonne of ore	$52.08
Total Cash operating cost [4]	per ounce Au Eq.	$311
Total Cash operating cost, inc capital [4]	per ounce Au Eq.	$517
Total Cash operating cost (by-product) [5]	per ounce Au (Ag credit)	-$94
Total Cash operating cost inc capital (by-product) [5]	per ounce Au (Ag credit)	$231
Pre-Tax IRR	%	41
Cash Flow (non-discounted)	$ millions	660
NPV, 5% discount rate	$ millions	434
NPV, 10% discount rate	$ millions	286

1) This Preliminary Assessment is preliminary in nature, in that 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, and there is no certainty that the results of the preliminary assessment study will be realized.
2) Initial Capital includes $32.9 million in contingency allowance. Costs are based on Q3 2010 estimates and no escalation factors have been applied. Value added tax has not been included in the cost estimates.
3) Total Cash Operating costs include smelting and refining, Peruvian Government royalties, but do not include employee profit sharing or depreciation, depletion or amortization.
4) Total Cash Costs per ounce of gold equivalent are calculated using a silver-to-gold ratio of 60:1.
5) By-product accounting subtracts the revenue generated by silver from the operating costs as a credit to determine the cost per ounce of gold.
6) Annual and life-of-mine production figures are after 5% mining losses, 20% mining dilution and the respective metallurgical recoveries for gold and silver.

Based on drill results received up to a cut-off date of 15 May 2010 (drill hole INMA 139), an updated mineral resource estimate was calculated by FSS Canada, an independent consulting firm. This new estimate was used as the basis for the scoping study.

The resource estimate was reported at a cut-off grade of 3 g/t gold equivalent (using a silver to gold ratio of 60:1), which approximates the cut-off grade for the underground mining and flotation process option selected for Inmaculada, using a base-case gold price of US$1,000 per ounce.

Table 6.3 - Angela Vein, Inmaculada Project – Mineral Resource Estimates (as of 9 September 2010 at US$1,000/oz gold and $17/oz silver)

Resource Estimate Category	Tonnes	Gold Grade (g/t)	Silver Grade (g/t)	100% Project Contained Ounces			
				Gold	Silver	Gold Equivalent	Silver Equivalent
Measured	1,080,000	5.1	107	178,000	3,717,000	240,000	14,395,000
Indicated	2,747,000	4.0	137	354,000	12,128,000	556,000	33,392,000
Measured and Indicated	**3,827,000**	**4.3**	**129**	**532,000**	**15,845,000**	**796,000**	**47,788,000**
Inferred	4,388,000	4.6	200	645,000	28,283,000	1,116,000	66,959,000

1) Resources are shown on a 100% project basis. IMZ controls a 70% interest (Hochschild Mining 30%) with a current 51% ownership and is earning a 70% interest by completing a feasibility study before September 2013 and issuing to Hochschild 200,000 IMZ shares.
2) Numbers are rounded to reflect the precision of a resource estimate.
3) The estimated mineral resources are not mineral reserves and do not have demonstrated economic viability.
4) To limit the influence of individual high-grade samples, grade cutting was used. Gold assay grades were capped at 100 g/t and silver grades were capped at 1,500 g/t.
5) Average dry bulk densities of 2.51 tonnes per cubic meter (t/m^3) for all mineralized rocks.
6) The grades were interpolated using "Ordinary Kriging" estimation technique.
7) Descriptions of parameters to determine "Measured", "Indicated" and "Inferred" resources are provided below.
8) The contained metal estimates remain subject to factors such as mining dilution and process recovery losses.

6.2.4 2011 Mineral Resource Estimate

In February 2011, IMZ reported an updated, increased mineral resource estimate for the Inmaculada project.

Table 6.4 contains the updated resource estimate (on a 100% project basis) using a 3.0 g/t gold equivalent cut-off grade.

Table 6.4 - Inmaculada Project, Angela Vein Deposit - Mineral Resource Estimate – 24 February 2011

Resource Estimate Category	Cut-Off (g/t gold Equiv)	Tonnes	Gold Grade (g/t)	Silver Grade (g/t)	100% Project Contained Ounces			
					Gold	Silver	Gold Equivalent	Silver Equivalent
Measured	4.2	698,000	6.0	153	135,000	3,420,000	192,000	11,520,000
	3.0	**941,000**	**5.1**	**136**	**155,000**	**4,110,000**	**223,500**	**13,410,000**
	1.6	1,094,000	4.7	125	164,000	4,410,000	237,500	14,250,000
Indicated	4.2	2,951,000	6.1	226	578,000	21,450,000	935,500	56,130,000
	3.0	**3,806,000**	**5.2**	**198**	**640,000**	**24,210,000**	**1,043,500**	**62,610,000**
	1.6	4,518,000	4.7	177	676,000	25,700,000	1,104,333	6,260,000
Measured and Indicated	4.2	3,649,000	6.1	212	713,000	24,880,000	1,127,667	67,660,000
	3.0	**4,747,000**	**5.2**	**186**	**795,000**	**28,320,000**	**1,267,000**	**76,020,000**
	1.6	5,612,000	4.7	167	840,000	30,110,000	1,341,833	80,510,000
Inferred	4.2	1,998,000	7.4	295	473,000	18,950,000	788,833	47,330,000
	3.0	**2,648,000**	**6.1**	**247**	**521,000**	**21,030,000**	**871,500**	**52,290,000**
	1.6	3,553,000	5.0	199	568,000	22,770,000	947,500	56,850,000

1) **Resources are shown on a 100% project basis. IMZ owns a 40% interest.**
2) Metal prices used are US$15/oz for silver and US$900/oz for gold.
3) An overall average bulk density of 2.51 tonnes per cubic meter has been used for the tonnage estimation.
4) Gold equivalent grade is calculated at a silver: gold ratio of 60:1, using metallurgical recoveries of 88% for gold and 83% for silver and metal prices as stated in Note 2.
5) **The resources are reported at a 3 g/t gold equivalent cut-off grade. This case is shown in bold text in the Table above.**
6) The estimated mineral resources are not mineral reserves and do not have demonstrated economic viability.
7) Numbers have been rounded in all categories to reflect the precision of the estimates.
8) The mineral resources were estimated by Hochschild using the ordinary kriging capability of MineSight software to estimate metal grades for the main vein body and inverse distance to the power of three for the peripheral veins. A block size of 10m by 5m by 10m in size was used and outlier high grades were top-cut to 30 g/t for gold and 860 g/t for silver.

7 Geological Setting and Mineralization

The information in this Section was sourced from a previous technical report by P&E Mining Consultants Inc. completed in 2010.

7.1 Regional Geology

The Inmaculada Property is located in the Western Cordillera of southern Peru, which consists of Cretaceous and Tertiary volcanics and to a lesser extent sedimentary sequences with Tertiary intrusive (Figure 7.1). Gold deposits are located within the Cenozoic Puquio-Caylloma Belt and are associated with volcanics and intrusions. Ore zones are hosted in volcanic rocks as epithermal Ag-Au mineralized quartz vein systems including the Low Sulfidation (LS) Pallancata, Ares and Explorador deposits, Intermediate Sulfidation (IS) Arcata and Caylloma deposits and High Sulfidation (HS) Shila, Paula, Selene, Suyckutambo, Chipmo and Poracota deposits.

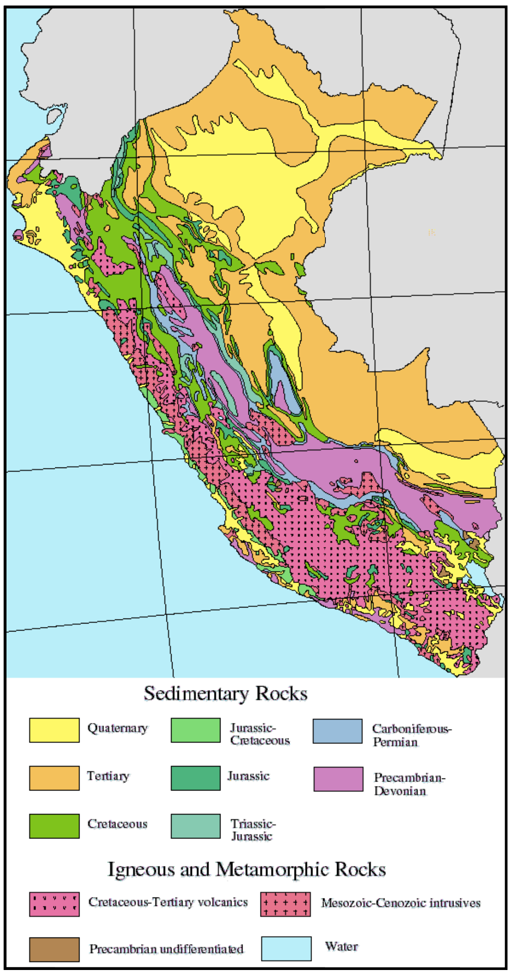

Figure 7.1 - Simplified Regional Geology of Peru

7.1.1 Property Geology

The oldest rocks within the Property are Mesozoic clastic marine sediments of the Soraya Formation of probable Mid Cretaceous age. The Soraya Formation consists of fine- to medium-grained sandstones and calcareous sandstones. Overlying the Soraya Formation are continental red beds of the Cretaceous Mara Formation. The Mara Formation consists of thick-bedded siltstones, sandstones and conglomerates. Both Mesozoic formations outcrop in the vicinity of the Minascucho and San Salvador targets on the Property. At these localities, the Mesozoic rocks are unconformably overlain by volcanic rocks of the Mid Oligocene (30 Ma) Tacaza Group, which reach a thickness of 600 to 800 m.

The known mineral occurrences on the Property are hosted by the Tacaza Group volcanics (Figure 7.2). The Tacaza sequence consists of a thin, basal unit of rhyodacite lapilli tuff, overlain by a thick sequence of andesitic flows, breccias and tuffs. Some local epiclastic sediments also occur intercalated within the andesites. Small stocks and dykes of andesitic composition are found within the Mesozoic basement rocks at Minascucho and San Salvador. These are thought to represent the feeders to the more voluminous flows and breccias. Small rhyolite domes, emplaced within Tacaza Group andesites, outcrop in the southwestern portion of Minascucho and at Tararunqui.

At Minascucho and San Salvador, the uppermost portions of the Tacaza are represented by laminated sandstones, tuffaceous sandstones and conglomerates which were deposited in a lacustrine environment, within a graben-like setting (the Minascucho Graben). The lacustrine sediments attain a thickness of approximately 40 m. Similar types of sediments of lacustrine origin also occur in the southwest corner of the Quellopata area.

The Tacaza Formation is overlain by the Miocene Alpabamba Formation in the southeastern part of the Property. The Alpabamba Formation consists of thin sequences of rhyodacite lithic tuffs, with the total thickness of the formation approximately 800 m.

The Miocene Aniso Formation overlies the Alpabamba Formation and consists mainly of crystal tuff up to 150 to 200 m in thickness. The Aniso Formation outcrops to the north of the Property boundary and is overlain by the Pleistocene to Pliocene aged Barroso Group. The Barroso Group mainly consists of andesitic lavas, lahars and breccias with an aggregate thickness of more than 400 m and occurs at the highest elevations within the Property.

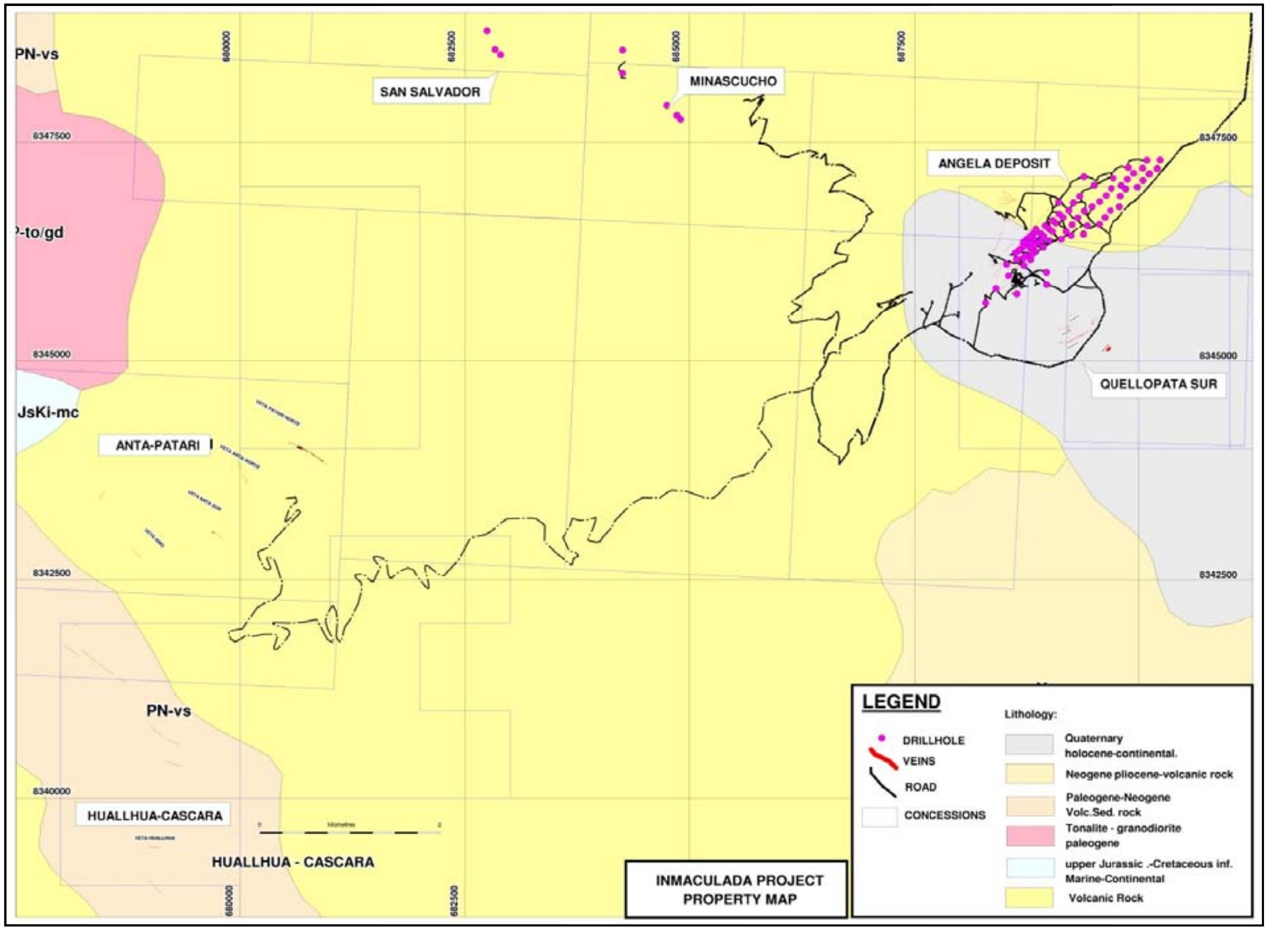

Figure 7.2 - Inmaculada Property Geology Map

7.2 Geology of the Quellopata Veins

The Quellopata area, which hosts the Angela Vein, is underlain by intercalated andesitic lavas and breccias of the Tacaza Formation. The andesites are greenish to purplish in color and porphyritic. The breccias appear to be autochthonous.

Up to four lava flows have been delineated at Quellopata and are intercalated with volcaniclastic breccias that consist of andesite clasts within an andesitic groundmass.

Northwest trending, southeast dipping faults are the oldest structures at Quellopata and host the eight veins known as Angela, Roxana, Martha, Teresa, Lourdes, Shakira, Juliana and Lucy (Figure 7.3). Relative displacements of marker horizons in cross-sections constructed from core logging and surface mapping, suggest that these structures are normal faults. These faults appear to have been active at various times, as evidenced by repeated brecciation of the fault breccia which makes up a portion of the mineralization (along with vein and stockwork types).

Both northeast trending and east-west trending structures occur at Quellopata. East-west trending, south dipping faults appear to displace the earlier vein structures. Where offsets have been observed in surface mapping, the apparent displacement is sinistral.

Four new quartz veins were identified in 2009 to the southeast of Quellopata, namely the Organa, Marina, Verónica and Rebeca veins, located 650 m to the southeast of Quellopata's veins. The veins trend to the northeast-east with azimuths ranging from 050 to 070° and dip steeply to the southeast

7.2.1 Geology of the Anta Patari Veins

The Anta-Patari veins are located 8 km to the southwest of the Quellopata veins (Figure 7.2). Six low sulfidation Ag-Au epithermal veins are recognized: North Patari, South Patari, Patari, Anta Norte, Menor and Anta Sur. The vein system strikes northeast to easterly and is approximately 300 m in length. Individual veins trend predominantly to the northeast and range between 0.3 to 6.0 m in width. The veins consist mainly of a vein / breccia of colloform-crustiform, stockwork and saccharoidal quartz. Replacement textures of carbonates by quartz are common, carbonate veins are minor. Ilmenite-smectite occurs in patches. Cubic pyrite occurs usually associated with iron and manganese oxides in many places.

7.2.2 Geology of the Cascara Huallhua Veins

The Cascara-Huallhua area is located to the southwest of Anta-Patari (Figure 7.2). Three low sulfidation Ag-Au epithermal veins have been identified: Huallhua, Chaguaya and Ismo. The Huallhua vein has been mapped for a strike length of approximately 600 m.

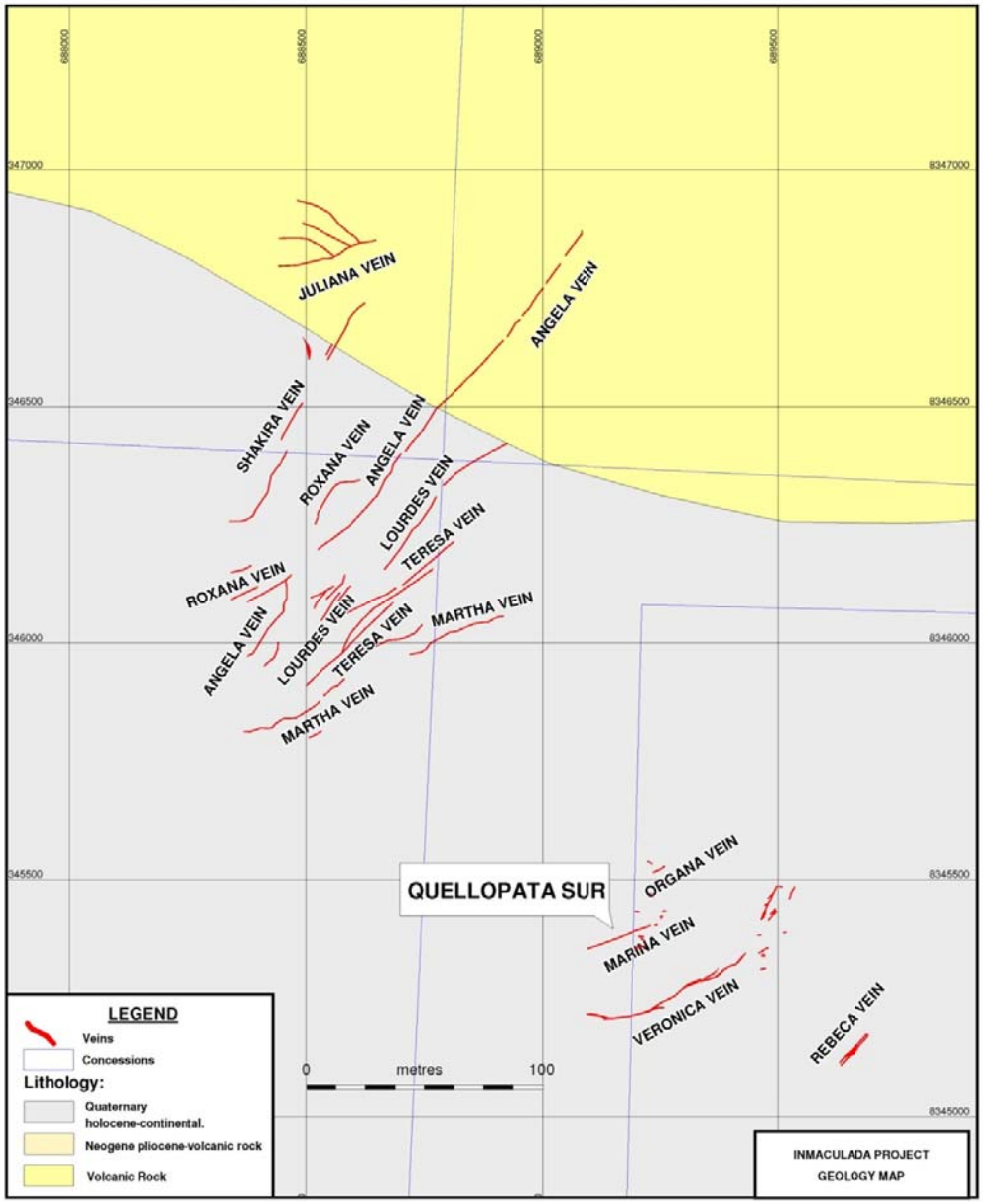

Figure 7.3 - Geology of the Quellopata Veins

7.3 Mineralization of the Angela Vein

Target vein structures are hosted in Tertiary-age volcanics and are associated with several episodes of mineralization. Average vein thicknesses range between 0.8-4.0 m in width and the majority of vein systems tend to be silver rich, although local variations do occur with zones of lower Ag:Au ratios.

Detailed mapping showed that the dominant rock type which hosts the Quellopata veins is andesite / andesite breccia of the Oligocene Tacaza Formation. The mapping also showed that the veins were predominantly northeast-trending, with the exception of the southwestern portion of the area where they trend more westerly. Rock chip sampling has shown that consistently high gold values (from 1 to over 10 g/t Au) are present in the Angela Vein, in the vicinity of line 10,000N of the local grid. This exposure of higher grade veining occurs in the valley. The better-grade material does not reach surface over much of the vein's strike length although quartz veining can be seen. The veining at surface has opalescent quartz with only rare bladed calcite pseudomorphs or colloform banding. The likely explanation is that the boiling level was just below the current topographic surface except in the valley, and that precious metal deposition stopped there.

Northwest trending, southeast dipping faults are the oldest structures at Quellopata and host the eight known veins: Angela, Roxana, Martha, Teresa, Lourdes, Shakira, Juliana and Lucy.

The Angela Vein outcrops in the central portion of the Quellopata vein system (Figure 7.3). The vein strikes northeasterly (050º), dips to the southeast (45º to 90º) and outcrops on surface along a strike length of 700 m (from line 9600N to 10300N). The vein has been intersected in drilling as far to the northeast as line 12000N (Figure 7.4). The portion of the vein that carries potentially economical quantities of precious metals so far occurs between 10000N and 11800N, a strike length of some 2,000 m.

The vein varies in thickness from 0.5 m to as much as 16.0 m, averaging approximately 6.0 m, and has been tested over a vertical extent of up to 300 m.

Two generations of mineralization have been observed in the Angela Vein, an early lead-zinc event and a later gold-silver event. The early mineralization consists of white quartz veinlets with sphalerite, galena, pyrite and argentite (minor). These veinlets form a broad, low-grade envelope (0.2 to 1.0% Pb + Zn) which surrounds, and overlaps, the Angela Vein mineralization.

The second mineralizing event at the Angela Vein is the most important economically, and consists of a white chalcedony vein with associated breccia and stockworks. The chalcedony contains small amounts (generally < 1%) of electrum, argentite, pyrargyrite, chalcopyrite, pyrite and marcasite.

The Angela Vein is composed of a gangue of white chalcedony, quartz, calcite (minor), smectite and illite. Pseudomorphic, quartz-after-calcite textures, and colloform banding, both indicative of boiling, are quite common in the vein. Metallic minerals, which rarely constitute more than 1% of the vein, occur as disseminations and colloform banding, and consist of pyrite, marcasite, argentite, pyrargyrite, chalcopyrite, sphalerite and electrum.

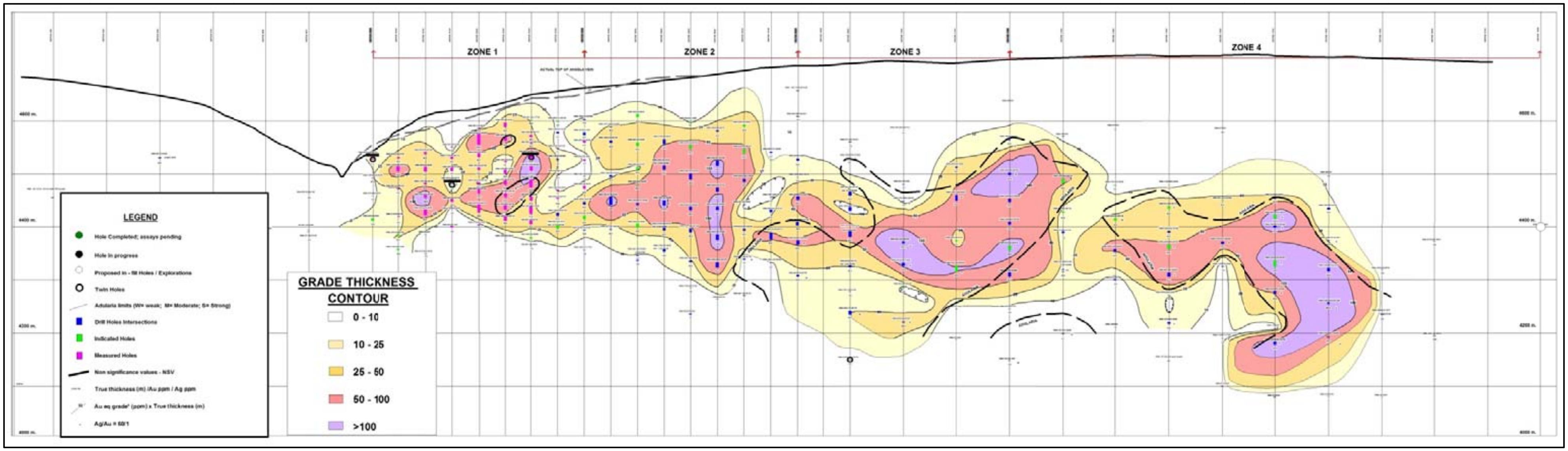

Figure 7.4 - Longitudinal Section with Grade Contours for the Angela Vein

The andesitic wall rock surrounding the Angela Vein is altered to a propylitic assemblage consisting of smectite, chlorite and disseminated pyrite. Quartz stockworks, some containing significant quantities of base metals, are common in the wall rock adjacent to the vein. Figure 7.5 shows a typical cross section through the Angela Vein wherein the vein changes laterally into breccia or stockworks.

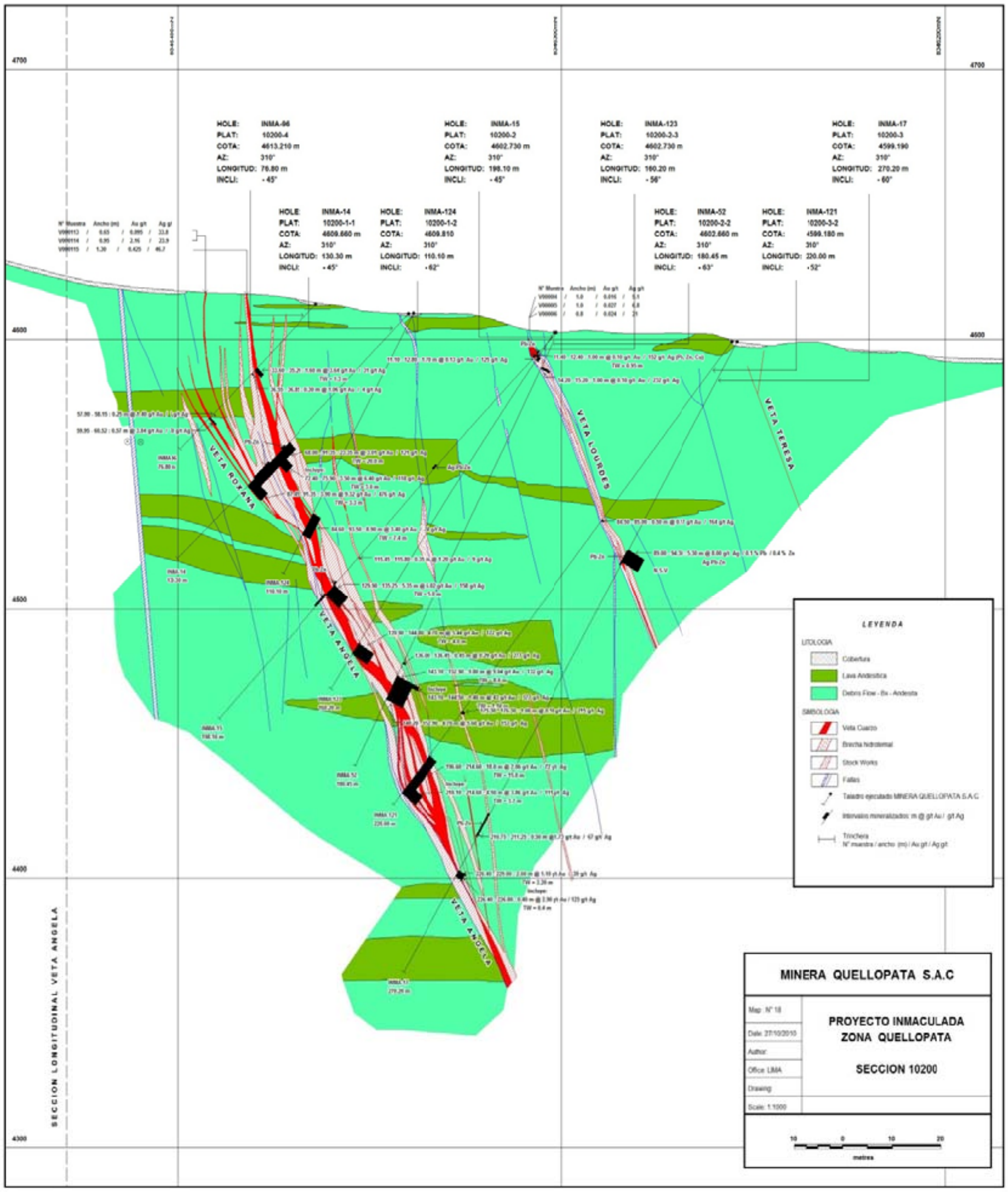

Figure 7.5 - Cross Section 10200N of the Angela Vein (looking north east)

8 Deposit Types

The information in this Section was sourced from the previous technical report on the property by P&E Mining Consultants Inc. completed in 2010.

The Inmaculada deposits can be broadly classified as low to high sulfidation epithermal silver-gold deposits.

In terms of metallogenesis, the Property lies in the Cenozoic volcanic belt of Southern Peru, known as the Puquio-Caylloma Belt. This belt hosts a number of important epithermal deposits with Ag-Au mineralization distributed as veins, mantos and disseminated breccias. Deposit sub-types on the Property include LS type which includes the Au-Ag quartz veins at Quellopata including the Angela Vein, IS type and HS type which includes mineralized breccias at Minascucho, Central and San Salvador and disseminated mineralization at Tararunqui.

The following sections are summarized from the following reports on epithermal systems: Panteleyev (1996), Corbett (2002, 2007) and Taylor (2007).

8.1 Epithermal Au-Ag Deposit Classification Systems

Epithermal Au-Ag deposits form in near-surface environments, from hydrothermal systems at shallow crustal levels (< 1 km) or low temperatures. They are commonly associated with centers of magmatism and volcanism, but form also in shallow marine settings. Hot-spring deposits and both liquid- and vapor-dominated geothermal systems are commonly associated with epithermal deposits. The deposits contain precious metals deposited by the mixing of upwelling mineralized fluids which contain a magmatic component, with oxidizing ground water.

Much of the gangue mineralogy comprising quartz, adularia, and carbonate forms in response to the boiling of dominantly meteoric fluids upon periodic, structurally-controlled pressure release, and so may develop the characteristic banded fissure vein ores.

Historically, epithermal deposits have been exploited for a wide variety of metals and minerals, however, many of the more economically significant deposits are mined for their precious metals.

8.2 Epithermal Deposit Sub-Types

Epithermal deposits are primarily distinguished using criteria of varying gangue and ore mineralogy, deposited by the interaction of host rocks and groundwaters with different ore fluids. The deposits are commonly considered to comprise one of two sub-types: LS and HS (Figure 8.1 and Table 8.1). Each sub-type is denoted by characteristic alteration mineral assemblages, occurrences, textures, and in some cases, characteristic suites of associated geochemical elements.

LS epithermal deposits are distinguished from HS primarily by their sulfide mineralogy. Many low sulfidation veins are well banded and each band represents a separate episode of hydrothermal mineral deposition. LS deposits develop from dilute near neutral pH fluids and can be subdivided further into two groups: those which display mineralogies derived dominantly from magmatic source rocks (arc low sulfidation) and others with mineralogies dominated from circulating geothermal fluid sources (rift low sulfidation).

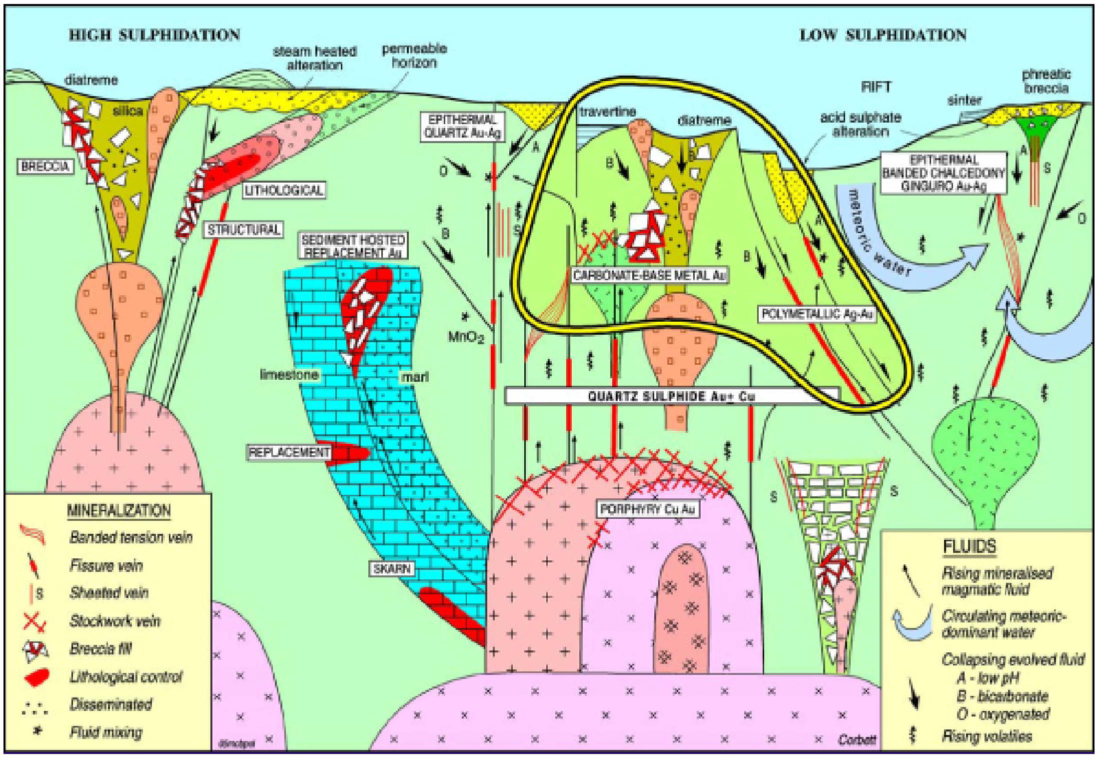

Figure 8.1 - LS and HS Epithermal Deposit Model

From Corbett (2007)

Styles of LS Au-Ag are distinguished according to mineralogy and relation to intrusion source rocks and influence precious metal grade, Ag:Au ratio, metallurgy and Au distribution. The following subsets of LS have been distinguished by Corbett (2007): quartz-sulfide Au ± Cu; carbonate-base metal Au; polymetallic Ag-Au; epithermal quartz Au-Ag; chalcedony-ginguro epithermal Au-Ag. Polymetallic Ag-Au deposits dominate in the Americas as fissure Ag-rich veins.

IS sub-types are considered to be a subset of LS types. In some epithermal deposits, notably those of IS sub-type, base metal sulfides may comprise a significant ore constituent.

HS systems vary with depth and permeability control, and are distinguished from several styles of barren acid alteration. HS systems develop due to the reaction of hot acidic magmatic fluids with the host rocks, producing characteristic zoned alteration and later sulfide and Au±Cu±Ag deposition. Ore systems display permeability controls governed by lithology, structure and breccias and changes in wall rock alteration and ore mineralogy with depth of formation.

Taylor (2007) subdivides based on LS and HS which are further subdivided into those hosted by volcanic and plutonic rocks and those that are hosted in sedimentary and mixed host rocks.

The polymetallic Ag-Au vein systems of South America typically occur with the following sulfides: pyrite > sphalerite > galena > chalcopyrite with electrum, Ag sulphosalts (tennantite-tetrahedrite, argentite) and quartz, carbonate and barite gangue. These systems are considered the Andean equivalent of the SW Pacific carbonate-base metal gold epithermal systems.

Table 8.1 - Characteristics of Epithermal Deposit Sub-Types

	Low Sulfidation	Intermediate Sulfidation	High Sulfidation
Metal Budget	Au - Ag, often sulfide-poor	Ag - Au ± Pb - Zn; typically sulfide-rich	Cu - Au - Ag; locally sulfide-rich
Host Lithology	bimodal basalt-rhyolite sequences.	andesite-dacite; intrusion centred district.	andesite-dacite; intrusion centred district.
Tectonic Setting	rift (extensional)	arc (subduction)	arc
Form and Style of Alteration / Mineralization	vein arrays; open space veins dominant; disseminated and replacement ore minor; stockwork ore common; overlying sinter common; bonanza zones common.	vein arrays; open space veins dominant; disseminated and replacement ore minor; stockwork ore common; productive veins may be km-long, up to 800 m in vertical extent.	veins subordinate, locally dominant; disseminated and replacement ore common; stockwork ore minor.
Alteration Zoning	ore with quartz-illite-adularia (argillic); barren silicification and propylitic (quartz-chlorite-calcite ± epidote) zones; vein selvedges are commonly narrow.	ore with sericite-illite (argillic-sericitic); deep base metal-rich (Pb-Zn ± Cu) zone common; may be spatially associated with HS and Cu porphyry deposits.	ore in silicic core (vuggy quartz) flanked by quartz-alunite-kaolinite (advanced argillic); overlying barren lithocap common; Cu-rich zones (enargite) common.
Vein Textures	chalcedony and opal common; laminated colloform-crustiform; breccias; bladed calcite (evidence for boiling).	chalcedony and opal uncommon; laminated colloform-crustiform and massive common; breccias; local carbonate-rich, quartz-poor veins; rhodochrosite common, especially with elevated base metals.	chalcedony and opal uncommon; laminated colloform-crustiform veins uncommon; breccia veins; rhodochrosite uncommon.
Hydrothermal Fluids	low salinity, near neutral pH, high gas content (CO_2, H_2S); mainly meteoric.	moderate salinities; near neutral pH.	low to high salinities; acidic; strong magmatic component?
Examples	McLaughlin, CA; Sleeper and Midas, NV; El Peñón, Chile; Hishikari, Japan; Pallancata, Peru.	Arcata Peru; Fresnillo Mexico; Comstock NV; Rosia Montana Romania.	Pierina Peru; Summitville CO.

Compiled from Hedenquist et al. (2000) and Hedenquist and White (2005)

9 Exploration

The following information on the exploration program on the Inmaculada Property from 2007-2008 is taken from the Hennessey and Pressacco (2009) and updated below to 2010. Exploration and drilling programs prior to 2007 are described in Section 5 of this Report and drilling from 2007 onwards is described in detail in Section 10 of this Report.

In 2007, Ventura acquired an option to explore the Property and began the exploration of the Quellopata veins. The initial work consisted of detailed mapping, in-house topographic surveying and rock chip sampling. Encouraging gold and silver assay results were received from the Angela Vein at Quellopata, exposed in outcrop, and in the initial drill holes drilled in the southwestern part of the vein. Ventura then followed the mineralization along strike to the northeast where the vein disappeared under the ridge. The drilling program identified significant 'blind' mineralization in the Angela Vein.

9.1 2007 Ventura Exploration Program

Ventura, through its Peruvian subsidiary, performed the following exploration work:

- The initial work in Inmaculada project between April and September 2007 was the rehabilitation of the access road to camp and to the Quellopata and Minascucho targets;
- The camp construction at Quellopata;
- Detailed topographic surveys at Quellopata (approximately 6 km^2);
- Rock chip sampling at: Quellopata (360 samples) and regionally (133 samples); and
- Core drilling at Quellopata (15 holes totaling 2,901 m).

9.2 2008 Ventura Exploration Program

Ventura, through its Peruvian subsidiary, performed the following exploration work:

- Detailed topographic surveys of the Minascucho and San Salvador targets (approximately 2 km^2);
- Detailed geological mapping of Minascucho (1:2,500 scale, approximately 1 km^2);
- Detailed geological mapping of San Salvador (1:1,000, approximately 1 km^2);
- Rock chip sampling at Minascucho (50 samples) and San Salvador (10 samples);
- Soil sampling at Minascucho (132 samples);
- 46 Pima samples were taken at Minascucho and San Salvadors;
- Magnetic surveying at Minascucho (9.1 line-km) and San Salvador (4.2 line-km);
- IP / resistivity surveying at Minascucho (9.1 line-km) and San Salvador (4.2 line-km);
- Core drilling at Quellopata (39 holes totaling 10,252 m);
- Core drilling at Minascucho (7 holes totaling 1,460 m); and
- Core drilling at San Salvador (3 holes totaling 450 m).

Ventura drilled a total of 15,059.2 m during 2007-08, thus completing one of the requirements for earning its 51% equity position in the Inmaculada property.

9.3 2009 Ventura Exploration Program

The 2009 program explored new targets on the Inmaculada Property with three new gold targets identified to the south and south-east of Quellopata and the Angela Vein: the Anta-Patari zone, the Cascara-Huallhua zone and the Quellopata Southeast zone. These occurrences are associated with quartz veins and veinlets hosted in volcanic sequences. Geochemical trends correlate with detailed geological mapping confirming that the gold is associated with the volcanic sequences.

The program at Anta Patari consisted of detailed geological mapping at 1:1,000 scale of the Anta Patari (Patari, Anta Norte, Anta Sur and Minor) veins and rock chip sampling from vein outcrops and trenches, totaling 286 samples.

The Huallhua vein in the Cascara-Huallhua area has been geologically mapped, with an outcropping strike length of approximately 600 m. Sixteen rock channel samples were taken from accessible workings on the Huallhua vein and a total of 50 samples were taken from the Cascara-Huallhua area.

Mapping and sampling of the Quellopata Southeast prospective area commenced in 2009. Four new quartz veins were identified in this area: Organa, Marina, Verónica and Rebeca vein. The veins are located approximately 650 m to the southeast of Quellopata's Martha vein. A total of 11 rock chip samples were taken from the new veins. The Rebeca vein reported low values in gold and silver.

Table 9.1 gives the average gold and silver grades for the veins as well as the structural details of each vein on the three prospects.

Table 9.1 - Rock Chip Results for the Anta Patari, Cascara-Huallhua and Quellopata Southeast Prospect Areas

Zone	Vein	Strike / Dip	Average Width (m)	Average Length (m)	Average Values		
					Au (g/t)	Ag (g/t)	Width (m)
ANTA-PATARI	Patari Norte	300/82NE	0.1-0.5	30	2.400	265.0	0.5
	Patari	310/50-65NE	0.5-6	600	1.451	178.0	1.0
	Anta Norte	330/75NE	0.3-4	280	5.600	453.0	1.2
	Menor	340/63E	1-2.5	100	4.881	222.0	1.0
	Anta Sur	310/85NE	0.5-3.5	120	8.600	215.0	1.0
CASCARA-HUALLHUA	Huallhua	270/81N	2.1	30	1.081	14.4	5.0
	Isno	320/75S	13	70	0.005	0.2	4.7
	Chaguada	330-315/70-82NE	1.5	450	0.478	11.2	1.4

Zone	Vein	Strike / Dip	Average Width (m)	Average Length (m)	Average Values		
					Au (g/t)	Ag (g/t)	Width (m)
QUELLO-PATA SE	Organa	70/76SE	0.7	15	0.005	91.6	0.7
	Marina	70/75SE	0.6-3.4	110	0.005	36.6	1.2
	Verónica	55/85SE	0.2-2.4	490	0.005	230.0	0.4
	Rebeca	40/85SE	6	33	0.071	2.9	1.8

9.4 2010 IMZ Exploration Program

Following the acquisition of Ventura by IMZ in January 2010, IMZ continued exploring the Quellopata area. The exploration program consisted of checking three targets; Quellopata central (Chaparral fault); Quellopata south east (Rebeca, Veronica, Marina and Organa Vein) and Quellopata South west (Melissa, Kattia, Jimena Vein). One hundred twenty seven channel samples were taken in these new veins identified in Quellopata targets.

Eleven channel samples were taken in Quellopata central area of Chaparral fault, 26 channel samples were taken in Quellopata south east of Rebeca, Verónica, Marina, Organa Veins, these veins were delimited an area of 1.5 by 1.5 km; In Quellopata south west, 85 samples were taken in Melissa, Kattia and Jimena Veins, these structures were filled with quartz, banding colloform opal, and hydrothermal breccias with chalcedonic and gray silica. Finally five rock chip samples were taken as a Jimenas Veins reconnaissance.

Table 9.2 gives the average gold and silver grades for rock channel samples taken from the veins as well as the structural details of each vein on the three target areas.

Table 9.2 - Summary of Channel Samples in Quellopata area

Zone	Vein	Azimuth	Width average (m)	Average Long. (m)	Averages Grade			Ratio Ag/Au	Observation
					Au (g/t)	Ag (g/t)	Width (m)		
QUELLOPATA	Chaparral	N60º/75ºSE	0.4	60	0.104	17.4	0.3	>100	Sb, Hg
QUELLOPATA SW	Jimenas	N75º/60º-80ºN - N75º/60º-80ºS	0.5-3.9	400	0.048	0.5	1	46	Hg
	Lucy I	N90º/70ºS	0.3-1.2	300	2.58	43	1	33.8	As, Mo, Sb, Hg, Mn, Zn
	Lucy II	N70º-N90º/70ºS	0.3-1.2	300	1.625	18.8	1.1	44	Ba, Mo, Hg, Zn
	Kattia	N80º/75ºS	0.3-0.9	70	0.912	8.7	0.6	18	As, Mo, Sb, Hg
	Melissa	N295º/80ºS W	0.5-3.0	240	0.596	15	1.7	45	As, Mo, Sb, Hg

Zone	Vein	Azimuth	Width average (m)	Average Long. (m)	Averages Grade			Ratio Ag/Au	Observation
					Au (g/t)	Ag (g/t)	Width (m)		
QUELLOPATA SE	Organa	N70º/76ºSE	0.7	15	0.005	91.6	0.7	>1000	Ba, Pb, Zn, Mn, Hg
	Marina	N70º/75ºSE	0.6-3.4	110	0.005	36.6	1.2	>1000	Ba, Pb, Zn, Mn, Hg
	Verónica	N55º/85ºSE	0.2-2.4	490	0.005	230	0.4	>1000	As, Pb, Zn, Mn, Hg
	Rebeca	N40º/85ºSE	6	33	0.071	2.9	1.8	40	As, Hg, Mn

9.5 2011mHochschild Exploration Program

During 2011 Hochschild continued with surface sampling of rocks, channel samples were taken from veins Jimenas, Martha, Angela (SW extension), Shakira and Juliana. The sampling was done selectively, taking rock chips of quartz veins and wall rock in 0.4 m wide channels. This sampling was carried out in soil covered areas, for this reason heavy machinery was used.

The trenching resulted in a total of 700 m of Angela SW extension being uncovered as well as 300 m of Shakira, 300 m of Juliana, 300 m of Jimenas (vein system), 100 m of Lucy and 100 m of Martha.

Table 9.3 - Summary of 2011 surface exploration

Zone	Vein	Azimuth	Width average (m)	Average Long. (m)	Averages Grade			Ratio Ag/Au	Nº Samples
					Au (g/t)	Ag (g/t)	Width (m)		
QUELLOPATA NE	Angela NE	N40/65-80SE	1.2	250	No significant values				342
	Shakira NE	N45/70-80SE	1	500	No significant values				-
	Juliana NE	N48/75SE	3	200	No significant values				-
	Martha	N65/70SE	0.45	100	5.06	61.1	0.45	12.08	36
QUELLOPATA SW	Angela SW	N60/65SE	0.35	300	1.539	16.9	0.35	10.98	107
	Cimoide Angela SW	N65-80/70NW	1.2	700	2.21	77.3	0.3	34.98	44
	Tensional Angela 1	N40/70NW	0.7	250	No significant values				
	Tensional Angela 2	N40/60NW	0.7	150	No significant values				
	Shakira SW		0.55	200	2.15	63.8	0.55	29.67	17
	Lucy	N80/65S	0.45	100	3.81	129	0.45	33.86	41
	Jimenas		3	300	2.615	22.6	0.3	8.64	275

10 Drilling

10.1 Core Drilling

The following information on the drill program on the Inmaculada Property from 2007-2008 is taken from the Hennessey and Pressacco (2009) and updated below to 2011.

From 2007 to December 2011, 326 core drill holes totaling 102,867 m have been drilled on the Property. Table 10.1 provides the details on the program for each year to December 2011. This does not include ancillary drilling such as specific geomechanical and hydrogeology holes.

Table 10.1 - Drill Holes on the Inmaculada Property, 2007 – December 2010

Company	Phase	Number of Drill Holes	Total (m)	Target Area
Ventura Gold (2007-2009)	I	15	2,910	Quellopata
	II	15	3,226	Minascucho
	III	15	4,426	Quellopata
	IV	19	4,506	San Salvador
	V	9	4,042	Quellopata
	VI	36	15,437	Quellopata
IMZ 2010	VII	114	35,025	Quellopata
	VIII	9	3,244	Quellopata
Hochschild 2011	IX	94	30,051	Quellopata
Total		**326**	**102,867**	

10.1.1 2007 - 2008 Ventura Drill Program

The 2007 drilling campaign on the Inmaculada Property focused on the initial testing of the Angela, Shakira, Martha, Teresa and Lourdes veins at depth at Quellopata, totaling 2,901 m in 15 holes. The best intersections were obtained from the Angela Vein along sections 10100N and 10200N of the local grid. A list of drill holes and a summary of significant intercepts for the 2007 drill program is reported in Hennessey and Pressacco (2009).

Based on the 2007 drilling results, a second drill program commenced in 2008 during which a total of 39 core drill holes and 10,247 m was drilled at the Angela Vein using a nominal separation between holes of 100 m along strike and between 50 to 100 m down dip. The Angela Vein was tested along a strike length of 1,000 m (between lines 10000N and 11000N) as well as across a vertical distance of approximately 200 m. Drilling extended the Angela Vein to a strike length of more than 900 m and a depth of 300 m below surface and continues to show continuity of thickness and grade throughout the vein (Ventura News Release 3 December 2008).

The vein remained open at depth and down-plunge to the northeast. A list of drill holes and a summary of significant intercepts for the 2008 drill program on the Angela Vein is reported in Hennessey and Pressacco (2009). The San Salvador target is located approximately 7 km northwest of the Angela Vein. Three core drill holes, totaling 450 m, were drilled below an outcropping, mineralized breccia at San Salvador which represents a disseminated, high-sulfidation gold target. The drill results indicated the presence of gold mineralization, requiring additional evaluation and drilling to better understand the underlying structure (Ventura News Release 3 December 2008).

Highlights of the drilling include the following (true widths unknown):

- SS-01: From 107.4 to 121.5 m; 14 m at 0.4 g/t Au and 26 g/t Ag
- SS-03: From 20.0 to 38.0 m; 18 m at 0.7 g/t Au and 20 g/t Ag

The Minascucho breccia, located 5 km north of the Angela Vein, was drilled in 2008 with 5 core drill holes completed in the breccia and two additional holes drilled on the Central zone (another exploration target adjacent to Minascucho), totaling 1,460 m. The Central zone drill holes returned no significant values. Positive results were reported from the Minascucho breccia (Ventura News Release 8 September 2008) with significant intersections as follows:

- MIN-01: From 70.65 to 107.05 m; 32 m at 2.0 g/t Au and 8 g/t Ag
- MIN-02: From 118.6 to 143.05 m; 18 m at 3.6 g/t Au and 45 g/t Ag
- MIN-04: From 104.9 to 126.95 m; 20 m at 0.9 g/t Au and 3 g/t Ag

The 2007 - 2008 core drilling program on the Angela Vein in the Quellopata area was performed by Bradley-MDH (Bradley) of Lima, Peru. All holes were drilled using HQ-diameter rods up to a depth of 250 m and NQ-diameter rods to the end of hole.

The azimuth and inclination of the drill stem were measured using topographic surveying instruments. Dip and azimuth measurements were taken every 100 m down the hole with a Reflex 'EZ-Shot' instrument which electronically records the azimuth, dip and local magnetic field.

10.1.2 2009 Ventura Drill Program

In 2009, 45 core drill holes totaling 15,4312 m were drilled on the Property (Table 10.2). The drill program commenced on 3 June 2009 and was completed by December 15, 2009. Holes were drilled by Bradley using a modified Hydracore LD-250 portable drilling machine with HQ-diameter rods to 250 m then NQ-diameter rods to the bottom of the hole and two LF70 drilling machine using HQ-diameter rods.

The program focused on the Angela Vein at Quellopata in order to expand the known mineral resources, extend the vein laterally to the northeast and to further define some of the possible feeder zones that are still open at depth (IMZ News Release 19 January 2010). Infill drilling was undertaken between sections 10000N and 10400N and exploration drilling between sections 10500N and 11600N. As a result of the program, the Angela Vein was extended to a 1,500 km strike with an average width of 8.3 m. An additional 2,000 m of the 2009 drill program tested the down-dip extent of mineralization at the Martha Vein and the untested Anta and Patari Vein systems. The Martha Vein was first intersected during the 2007 drill program (Ventura News Release 2 July 2009).

10.1.3 2010 IMZ Drill Program

One hundred - twenty one core drill holes totaling 38,269 m have been drilled on the Property from 15 January 2010 to 15 December 2010.The holes were drilled by Bradley using a modified Hydracore LD-250 portable drilling machine with HQ-diameter rods to 250 m then NQ-diameter rods to the bottom of the hole and two LF70 drilling machine using HQ-diameter rods. The drilling extended the Angela Vein mineralization for an additional 500 m of strike length to more than 2,000 m with a vertical extent of up to 300 m and the vein system averaging 12.5 m in width. Mineralization remains open along strike to the northeast (IMZ News Release, 4 May 2010). Table 10.2 lists significant intercepts of the 2007 to 2010 drill program.

Table 10.2 - Significant Representative Intercepts for the 2007 to 2010 Drill Program

Section	DDH	DDH length (meters)	Az	Incl	From (m)	To (m)	Inters (m)	True Width (meters)	Au (gpt)	Ag (gpt)	Au Eq ppm (Ag/Au=60)	Au Eq-Width ppm-m (Ag/Au=60)
10000N	Inma-207	100.50	310	-52	26.88	31.22	4.34	2.4	7.38	124	9.45	22.7
10050N	Inma-117	101.30	310	-58	52.35	70.60	18.25	10.00	4.07	145	6.49	64.9
			including		52.35	61.15	8.80	4.80	6.98	158	9.61	46.1
10100N	Inma-04		310	-45	50.50	56.65	6.15	5.00	6.32	166	9.09	45.4
			including		53.20	56.65	3.45	2.80	8.98	129	11.13	31.2
10100N	Inma-16		310	-45	204.75	217.15	12.40	12.00	3.90	99	5.55	66.6
			including		212.75	217.15	4.40	4.30	8.60	237	12.55	54.0
10100N	Inma-75	187.30	310	-52	151	157.65	6.65	6.20	22.39	156	24.99	154.9
10150N	Inma-78	230.90	310	-45	187.50	192.98	5.48	5.40	4.97	104	6.70	36.2
			including		192.23	192.98	0.75	0.70	29.38	495	37.63	26.3
10200N	Inma-14	130.30	310	-45	68.00	91.35	23.35	20.00	3.01	121	5.03	100.5
			including		72.40	75.90	3.50	3.00	6.54	118	8.51	25.5
10200N	Inma-14		310	-45	87.45	91.35	3.90	3.30	9.32	476	17.25	56.9
10250N	Inma-105	210.00	310	-59	170.20	180.00	9.80	8.00	4.20	143	6.58	52.7
			including		170.20	173.85	3.65	3.00	6.30	183	9.35	28.1
10300N	Inma-19		310	-45	156.00	163.75	7.75	7.20	16.66	294	21.56	155.2
10400N	Inma-152	270.40	310	-51	240.58	246.56	5.98	5.10	5.00	180	8.00	40.8
10550N	Inma-155	269.10	310	-63	231.30	242.80	11.50	10.00	6.34	232	10.21	102.1
10650N	Inma-163	320.40	310	-80	281.24	296.75	15.51	11.00	6.78	265	11.20	123.2
10700N	Inma-203	330.80	310	-57	302.20	306.9	4.70	4.20	6.03	330	11.53	48.4
10800N	Inma-197	320.30	310	-56	297.5	304.4	6.90	6.85	3.03	169	5.85	40.0
10900N	Inma-64	457.45	310	-51	395.60	400.76	5.16	5.00	2.30	179	5.28	26.4
			including		395.60	399.30	3.70	3.60	2.78	240	6.78	24.4

Section	DDH	DDH length (meters)	Az	Incl	From (m)	To (m)	Inters (m)	True Width (meters)	Au (gpt)	Ag (gpt)	Au Eq ppm (Ag/Au=60)	Au Eq-Width ppm-m (Ag/Au=60)
11100N	Inma-61	484.10	310	-55	413.00	424.85	11.85	11.00	4.08	324	9.48	104.3
			including		416.30	421.10	4.80	4.50	6.70	554	15.93	71.7
11200N	Inma-73	606.50	310	-55	470.45	473.55	3.10	2.90	2.29	306	7.39	21.4
11300N	Inma-104	324.10	310	-48	242.00	250.20	8.20	7.60	5.02	136	7.29	55.4
			including		242.00	245.90	3.90	3.60	7.92	195	11.17	40.2
11500N	Inma-91	412.40	310	-55	298.25	303.75	5.50	4.60	5.36	178	8.33	38.3
11600N	Inma-118	629.20	310	-61	338.70	341.20	2.50	1.90	3.42	242	7.45	14.2
11700N	Inma-137	567.60	310	-59	465.45	471.70	6.25	5.00	7.15	192	10.35	51.8
11800N	Inma-164	414.40	310	-43	374.48	378.62	4.14	3.60	3.87	87	5.32	19.2

10.1.4 2011 Hochschild Drill Program

During 2011, infill drilling was carried in the Angela Vein with a total of 4,520 m in 12 core drill holes. Resource delineation and brownfield drilling was carried out in Angela SW, Shakira, Martha, Theresa Juliana, Jimenas and Nia, the total drilling was 25,531 m in 82 core drill holes, all performed from 15 January to 20 December.

At the same time drill core and reverse circulation was performed as part of the Feasibility Study with a total of 4,782 m in 49 holes (46 core drill holes and 3 reverse circulation drill holes). This consisted of geotechnical drilling performed in the proposed locations for the tailings dam, waste dump and raw water dam. Also geomechanical drilling was completed along the hangingwall and footwall strikes of the Angela Vein. This program was carried out from 7 June to 10 November.

11 Sample Preparation, Analysis, and Security

11.1 Sample Preparation

All samples are received by SGS-Lima and are processed through a sample tracking system that is an integral part of that company's Laboratory Information Management System (LIMS). This system utilizes bar coding and scanning technology that provides complete chain-of-custody records for every stage in the sample preparation and analytical process and limits the potential for sample switches and transcription errors.

After receipt and logging into the LIMS system, samples were dried in a large oven at 105 ºC (70 ºC if the sample is to be sent for mercury analysis). Samples were then crushed to 95% passing -10 mesh (1.7 mm). A 250 g subsample of the crushed material was then pulverized to 95% passing -140 mesh (106 micron). The sample preparation equipment was cleaned with barren cleaning material between sample preparation batches and, where necessary, between highly mineralized samples. Sample preparation stations were also equipped with dust extraction systems to reduce the risk of sample contamination. Pulps and coarse rejects from the prepared samples were returned to IMZ and stored in a secure warehouse in Lima for future reference.

11.2 Shipment and Storage of Samples

Following the logging and core marking procedures described above, the core was sampled by experienced samplers who have worked for Suyamarca or related companies on other projects. Quality control was maintained through regular verification by on-site geologists.

Core was broken, as necessary, into manageable lengths. The pieces were removed from the box without disturbing the sample interval markers and cut in half, lengthwise, with a diamond saw.

One half of the core was placed in a plastic sample bag and the other half was carefully repositioned in the core box. The tops of the sample bags were folded over several times and stapled shut with the sample tag visible in the fold for easy identification of the sample once shipped to the laboratory. The sample tag number was also written on the bag in black marker. The individual sample bags were placed in heavy fiberglass rice bags and sealed with heavy plastic tape.

The sealed sample bags were shipped by company truck to the field office in Chalhuanca where they were sent onward to Lima, either by commercial bus or by company truck. The samples were shipped to IMZ's Lima office where the quality assurance/quality control (QA/QC) samples were inserted prior to shipment to SGS. From the IMZ's Lima office the samples were transported to SGS by SGS laboratory personnel.

Once sampling was complete, the ends of the core boxes were painted white and information regarding drill hole number, box number and from and to intervals was recorded on the box ends. They were then stored in the core shed on-site at the Inmaculada Property.

The sample submission forms for the assay laboratory were prepared in Lima by the geologist responsible for insertion of the QA/QC samples.

11.3 Gold Analysis

Prepared sample pulps were analyzed for gold by fire assay of a 30 gram aliquot with atomic absorption spectrophotometry (AAS) finish (SGS code FAA313). Samples that assayed in excess of 5 g/t Au were re-analyzed with a gravimetric finish to ensure a more accurate result at the higher grades (SGS code FAG303).

A 35-element ICP package was also utilized for base metal and silver determinations. This package uses an aqua regia acid digestion with atomic emission spectroscopy (AES) finish (SGS code ICP12B). Over limit samples returning more than 100 g/t Ag were re-run using a multi-acid digestion method with AAS determination of silver, a method optimized for higher grades (SGS code AAS41B). SGS reports that the multi-acid digestion used for AAS41B is a “combination of HCl (hydrochloric acid), HNO_3 (nitric acid), HF (hydrofluoric acid) and $HClO_4$ (perchloric acid)” often referred to as a ‘near- total digestion’.

11.4 Bulk Density Data

The density information used in the resource and reserve estimates comes from a study done by Ventura in 2008 in which density was calculated for 100 samples of drill core from six lithologies. Each sample was dried, weighed in air, sealed in wax and weighed in water. Using Archimedes’ Principle, the dry bulk density of the material is based on the following formula:

Dry bulk density = Dry weight in air ÷ (Dry weight in air – Weight of sealed sample in water)

Note that with the weight of the sealed sample was adjusted to take into account the weight of the wax used to seal the sample before it was immersed in water.

Fifty one (51) of the 100 samples in this study were taken from the Angela Vein. The average density of these 51 samples was 2.51 t/m^3. The average density of the 29 samples of unaltered andesite material surrounding the mineralized vein was 2.54 t/m^3. The average density of the 20 samples showing stockwork alteration in andesitic material was 2.44 t/m^3. The 51 samples of mineralized material suggest that there may be a very slight tendency for density to decrease with increasing grade. The ten samples with highest gold-equivalent grades (> 10 g/t) had an average density of 2.49 t/m^3, while the ten samples with the lowest gold equivalent grades (< 1 g/t) had an average density of 2.53 g/t. With these differences being small, only 1-2%, and with too few samples to confirm whether or not they are statistically significant, the current resource and reserve estimates use a single density constant, 2.51 t/m^3, for the Angela Vein. If future studies establish that there is a statistically significant difference between the density of well-mineralized Angela Vein and weakly mineralized Angela Vein, then future resource studies should make use of densities that capture this relationship.

12 Data Verification

12.1 Verification of Database from Assay Certificates

The Database was recompiled in September 2010 by R. Mohan Srivastava, P. Geo. of FSS Canada Consultants Inc. (FSS), after spot-checking of the electronic assay data base against original assay certificates identified several types of errors, including assay values entered in the wrong interval, assay values not entered into the electronic data base, and typographical errors. The December 2011 mineral resource update corresponds exactly to the data provided by SGS. The only drill holes for which this verification was not possible were the earliest core drill holes, the QU series, which were drilled by Hochschild. Of these, the two which affect the resource estimates are QU-07 and QU-25. QU-07 intersects the Angela Vein, and contributes sample intervals whose gold and silver assays are used for grade estimation. QU-25 does not intersect the Angela Vein, but its lack of strong mineralization forces closure of the Angela Vein wireframe nearby; in this sense, QU-25 does affect the volume (and tonnage) estimates, but not the grade estimates. For these two drill holes, the assay data in the electronic data base was checked against tabulations of assay values, and average values across the Angela Vein, that appear in earlier project reports.

12.2 Independent Checks

The Inmaculada Property was visited by Mr. David Burga, P. Geo. on 10 and 11 June 2010 and Mr Ian Dreyer, CP on 13 July 2011. A tour of the drill sites was undertaken and core handling, core logging and sample protocols were observed. Check coordinates for selected drill hole collars were collected by GPS in 2010.

Ten assay verification samples were taken from nine drill holes during the June 2010 site visit. Sample intervals were taken from a variety of low, medium and high grade mineralized material. The chosen intervals were then sampled by taking the remaining half-split core. The samples were documented, bagged and sealed with packing tape and taken by Mr. Burga to ALS Chemex in Lima, Peru for analysis.

At no time, prior to the time of sampling, were any employees or associates of IMZ advised as to the location or identification of any of the sample intervals to be collected nor did they, at any time, have access to the sampled material.

A comparison of the independent sample verification results versus the original assay results for gold and silver are depicted in Figures 12.1 and 12.2 respectively. The results for gold and silver were satisfactory and clearly demonstrate that the tenor of the mineralization is similar to what was originally reported.

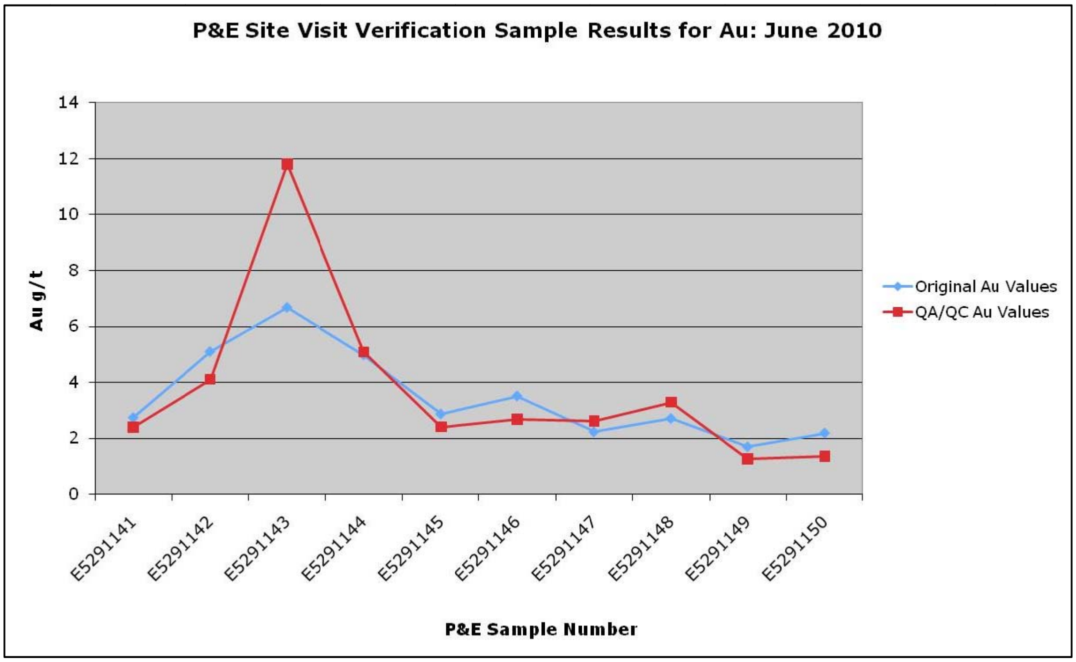

Figure 12.1 - Verification Sample Results for Gold, June 2010

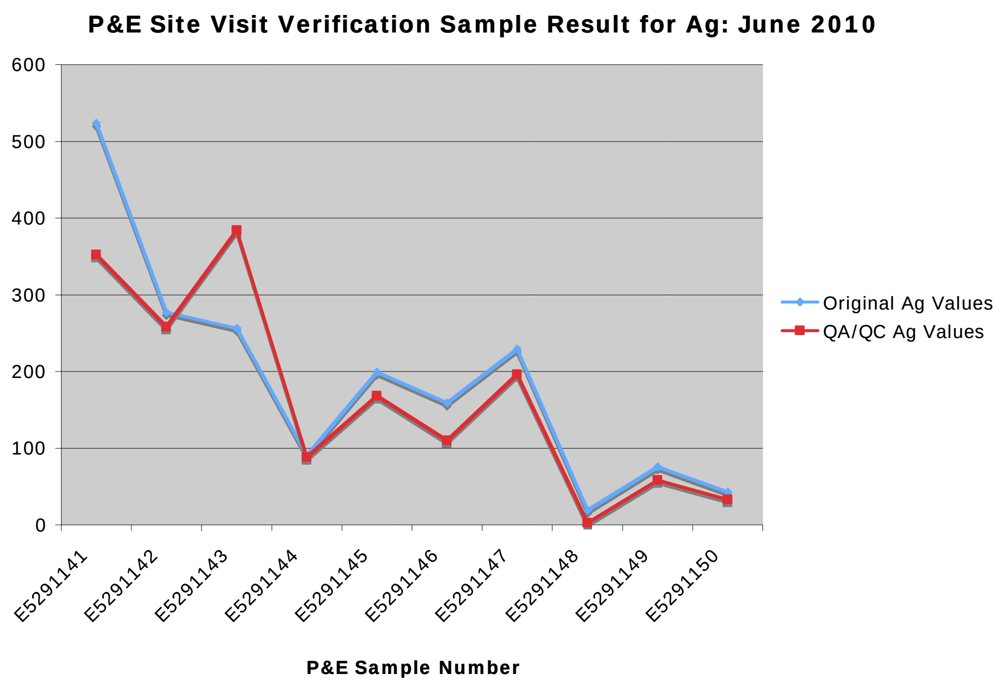

Figure 12.2 - Verification Sample Results for Silver, June 2010

12.3 Quality Assurance and Quality Control Programs

Core samples from Inmaculada were shipped to IMZ's Lima office where the standards and blanks were inserted and the entire sample shipment prepared for delivery to the SGS laboratory in Lima. Either a standard or a blank was inserted after every ten core samples, with standards and blanks being used alternately; this results in one of the five standards being used after every 20th core sample.

The position of the QA/QC samples was determined in the field during logging and sample tags, in sequence with the core samples, were reserved for QA/QC purposes so that the sample numbering of the standards and blanks was seamlessly integrated with the core samples.

The reliability of the Inmaculada assay results was monitored by a QA/QC program conducted by IMZ, that included the use of standards (to monitor accuracy), blanks (to monitor contamination) and duplicates (to monitor precision).

IMZ has a head office staff geologist who implements the QA/QC program and monitors the results of the standards, blanks and duplicates, looking for significant discrepancies in duplicate results, anomalously high values for the blank samples or sample results which are significant deviations from the accepted values for the standards. All anomalous results were reported to IMZ's exploration manager and followed up with the laboratory.

The standards consist of five standards, three of which were manufactured by SGS from local gold-silver mineralization, and two of which were commercial standards purchased from Rocklabs Ltd., a commercial provider of standard reference materials. The gold standards range from roughly 0.3 g/t to nearly 9 g/t; the silver standards range from trace quantities to over 1,000 g/t. As discussed earlier, the internal silver standards used by SGS covered only the lower silver grades up to 30 g/t; the external silver standards used by IMZ span the entire range, and thus serve as the better source of information on the accuracy of the high-grade silver assays.

12.3.1 Duplicate Samples

IMZ selected both pulp and coarse reject duplicates to be checked by re-assaying. Approximately 10% of all samples were sent for check assay. About 5% of the coarse reject samples were selected randomly, renumbered and sent back to the SGS laboratory for re-assay. Approximately 5% of the pulps were selected for re-assay, usually based on unexpected assay results, and were sent to a secondary laboratory for check assay. ALS Chemex in Lima was used as the secondary laboratory by IMZ; it analyzed the samples using similar methods to those employed by the primary laboratory, SGS.

The precision of duplicate assays was monitored using scatter plots, examples of which are shown in Figures 12.3 and 12.4. For the pulp duplicates, almost all of them (over 99%) have the two assays within ± 10% of their average; with the pulp duplicates having been assayed by two different labs, this indicates a very high degree of precision in the assay values. Furthermore, these inter-laboratory checks do not indicate any problem with systematic bias in either laboratory. The coarse reject duplicates show slightly less precision than the pulp duplicates, but still have almost 90% of the duplicates within ± 10% of their average.

Since 2007, none of the duplicate scatter plots has indicated any problem with sample mix-ups due to poor inventory control, or with typographical or transcription errors.

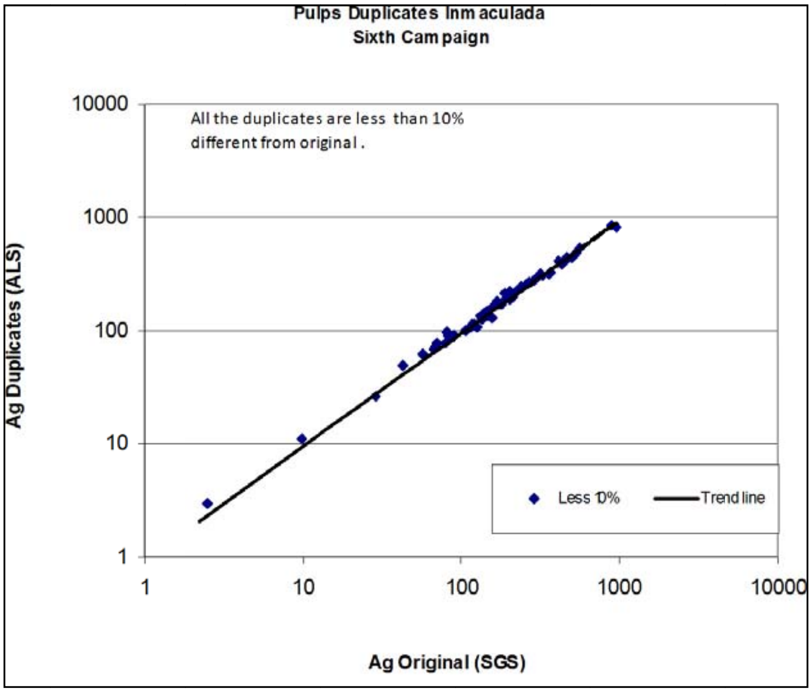

Figure 12.3 - Scatter Plot for Pulp Duplicates, Showing the Comparison of the Original Silver Assays from the SGS Laboratory to the Duplicate Silver Assays from the ALS Chemex Laboratory

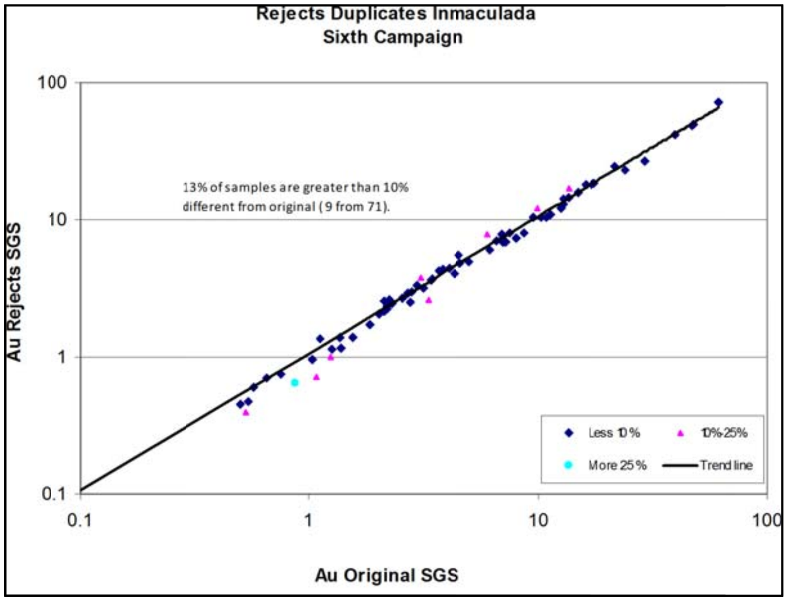

Figure 12.4 - Scatter Plot for Coarse Reject Duplicates, Showing the Comparison of the Original Silver Assays from the SGS Laboratory to the Duplicate Silver Assays from the ALS Chemex Laboratory

12.3.2 Blanks

The material used as blanks for the Inmaculada Property is locally purchased stone from Agregados Calcareos. A blank was inserted after every 20th sample, ten samples before (and after) each standard.

Blanks were monitored using control charts as shown in Figures 12.5 and 12.6. The magenta line on the control chart shows the assay values reported by the lab; the blue line defines the acceptable limit for blank assays. Since the locally purchased stone does, in fact, contain trace amounts of gold and silver, the acceptable upper limit is set at a threshold designed to detect significant cross-contamination of gold or silver from mineralized core samples.

Since the beginning of the Inmaculada drilling, there have been no blank samples that have shown any significant cross-contamination between samples. On rare occasions, the silver blanks assay exceeded 1.5 g/t with an example of this in Figure 12.6. This level of silver is extremely low, around the 1st percentile of the silver grade distribution and is entirely consistent with the trace amounts observed in one of the local sources used by Agregados Calcareos.

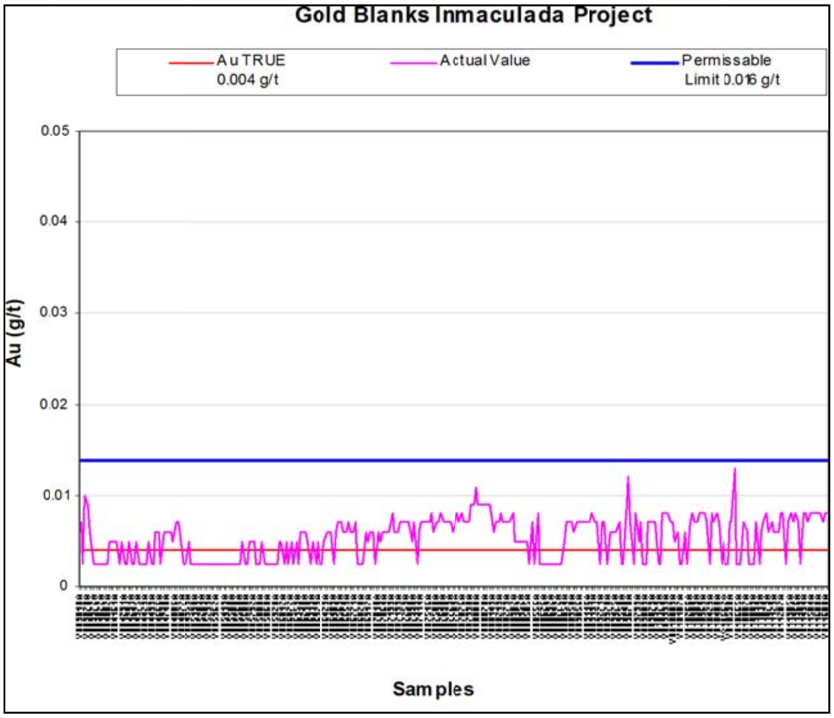

Figure 12.5 - Control Chart for External Gold Blanks

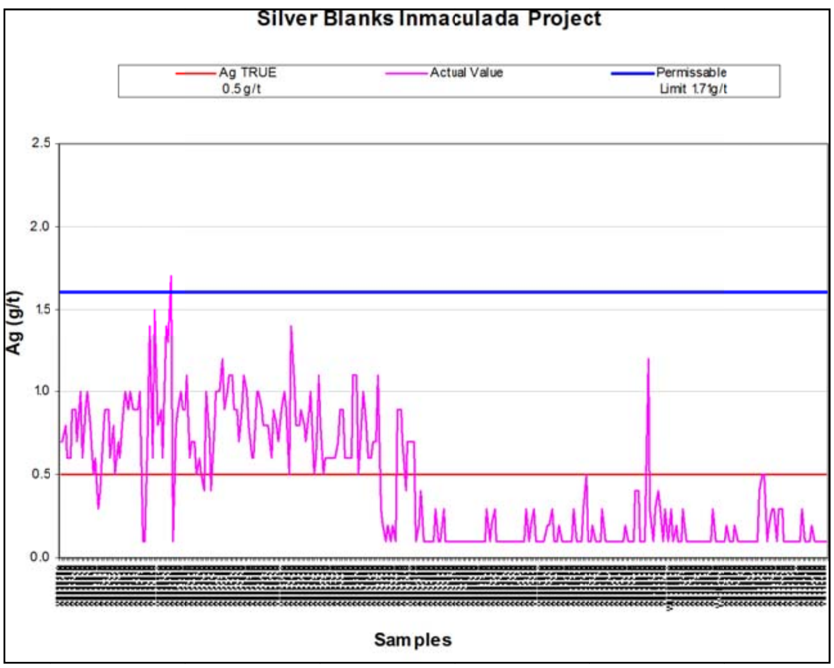

Figure 12.6 - Control Chart for External Silver Blanks

12.3.3 Standards

The external standards consisted of five standards, three of which were manufactured by SGS from local gold-silver mineralization, and two of which were commercial standards purchased from Rocklabs Ltd., a commercial provider of standard reference materials. The gold standards range from roughly 0.3 g/t to nearly 9 g/t; the silver standards range from trace quantities to over 1,000 g/t. As discussed earlier, the internal silver standards used by SGS covered only the lower silver grades up to 30 g/t; the external silver standards used by IMZ span the entire range, and thus serve as the better source of information on the accuracy of the high-grade silver assays.

Either a standard or a blank was inserted after every ten core samples, with standards and blanks being used alternately; this results in one of the five standards being used after every 20th core sample.

External standards were monitored using control charts like the one shown in Figure 12.7 below. The red solid line on the control chart shows the reference value for the standard; the magenta line shows the assay values reported by the lab; and the blue dashed lines define the acceptable range for each standard, and are set at ± 10% of the reference value.

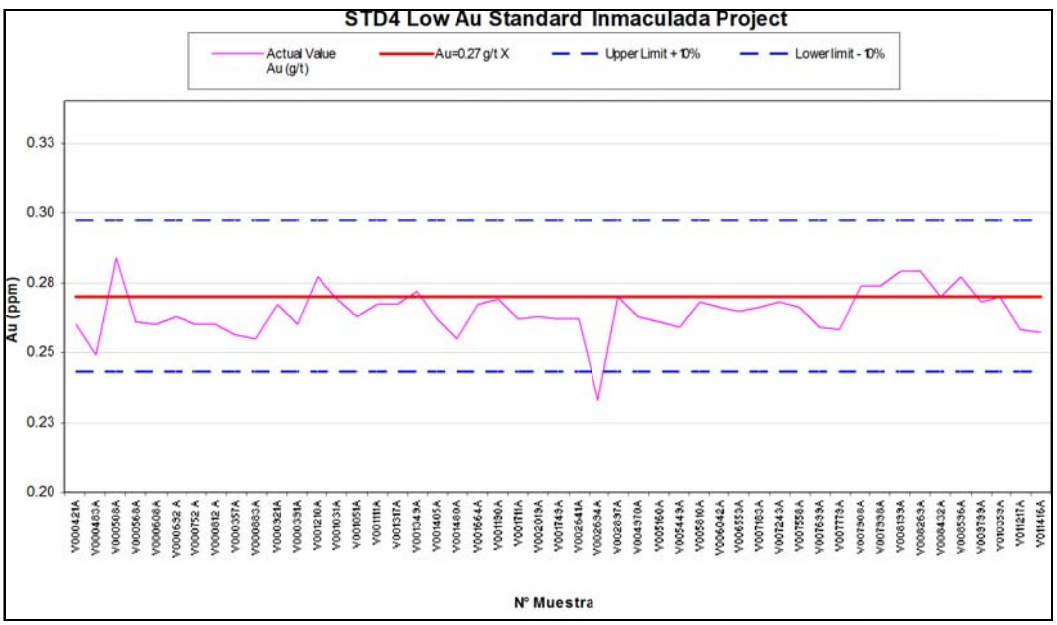

Figure 12.7 - Control Chart for External Gold Standards

13 Mineral Processing and Metallurgical Testing

13.1 Introduction

The Inmaculada project is being developed by Minera Suyamarca S.A.C. with ore from underground mining operations. The gold-silver bearing ores will be processed at a nominal rate of 3,506 t/d, with 91% availability, in a plant recovering the precious metal into doré bars. The process plant design feed grades are 4.5 g/t Au and 164 g/t Ag.

The process flowsheet includes single stage crushing, followed by grinding in a SAG-ball milling circuit (SAB) operating in closed-circuit with cyclones. The ground slurry is then fed to a leach circuit, for precious metal dissolution in cyanide solution. The gold in leach solution will be recovered in solution in a counter current decantation (CCD) circuit and precipitated in a Merrill Crowe circuit.

The precipitated gold and silver will be dried and smelted on site to produce gold and silver doré.

The leach circuit tailings are treated for cyanide destruction by the INCO SO_2/Air method to reduce the weak acid dissociable cyanide (CNwad) level to <50 ppm, prior to release into the tailings storage facility (TSF). Water is reclaimed from the TSF and used as make-up process water. Fresh water is provided for specific use within the processing circuits. Figure 13.1 provides an overall process block diagram.

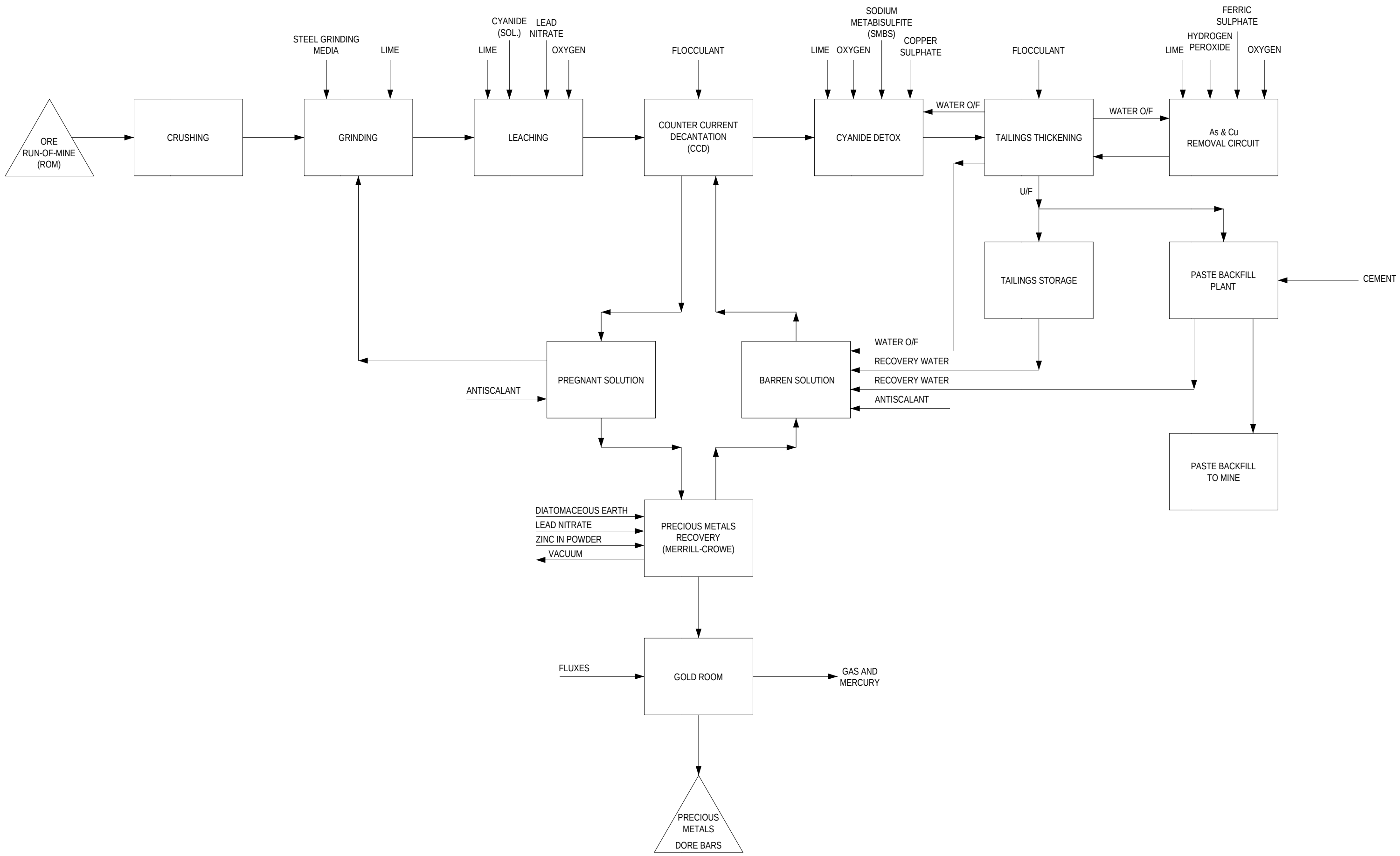

Figure 13.1 - Block Diagram of the Processing Facility

13.2 Key Sample Selection Criteria

The Angela Vein was divided into three zones or domains along the strike of the vein for the purposes of selecting metallurgical samples. The first domain represents the upper zone of the vein, with the third domain representing ore at the deepest section of the vein. Core drill hole (DDH) ¼ core and ½ core samples were selected from these domains to represent a bulk composite of the material in each domain.

In addition, variability samples were selected to provide an understanding of the change in ore characteristics spatially within the deposits, and determine any potential impact on the metallurgical response of these ores.

The drill hole locations from where the zone composites and variability composites were taken are depicted in Figure 13.2. The intercepts selected are shown in Tables Table 13.1 and Table 13.2.

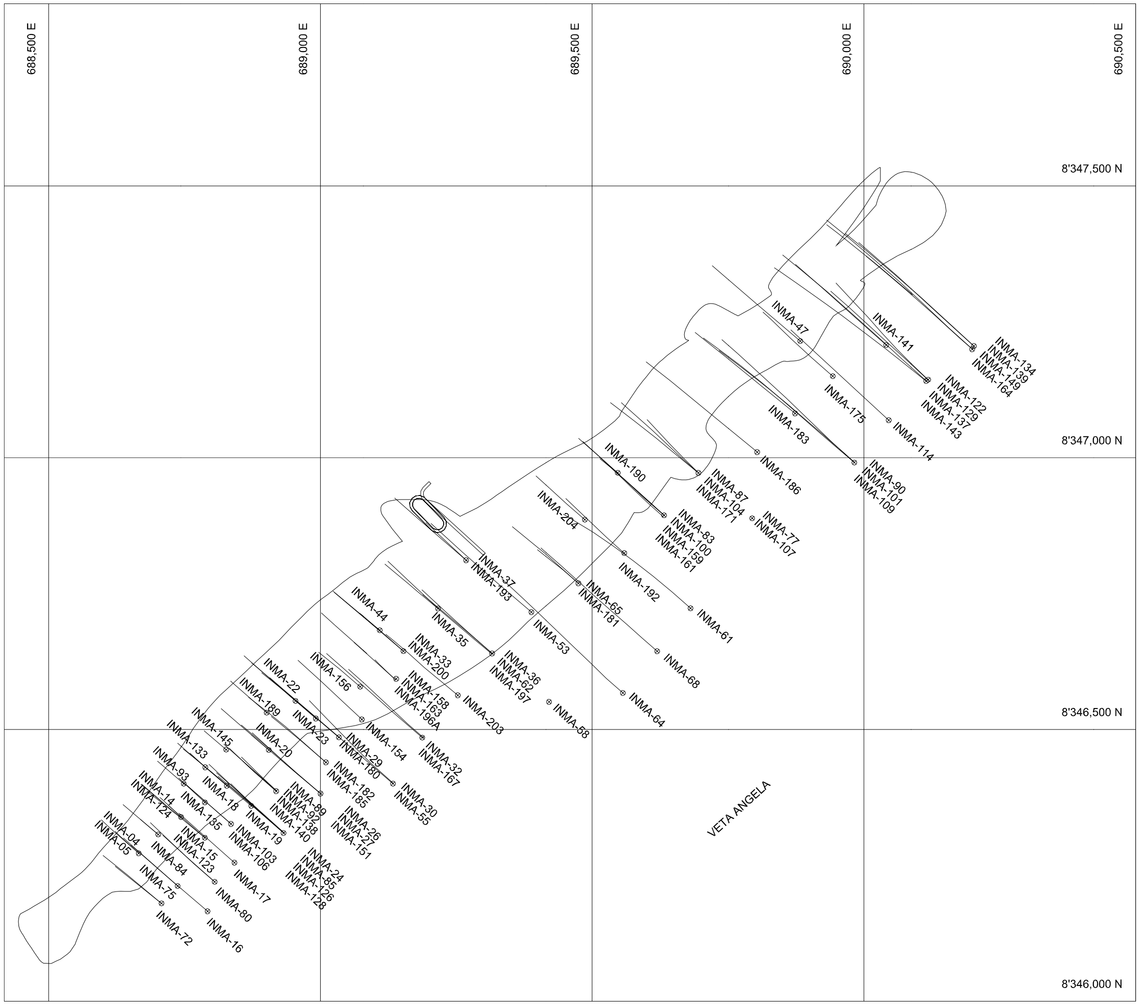

Figure 13.2 - Metallurgical Drill Hole Location

Table 13.1 - Zone Composites Drill Hole Identification

Zone 1 Composite			Zone 2 Composite			Zone 3 Composite		
Hole ID	From	To	Hole ID	From	To	Hole ID	From	To
INMA-05	75.6	79.6	INMA-22	104.6	107.2	INMA-65	299.4	305.4
INMA-14	71.9	78.9	INMA-23	155.0	162.9	INMA-68	474.0	478.5
INMA-15	129.9	135.5	INMA-29	202.3	207.1	INMA-73	470.5	474.2
INMA-16	204.8	217.2	INMA-30	311.8	324.1	INMA-77	361.0	364.9
INMA-18	104.7	107.7	INMA-32	290.4	297.2	INMA-90	465.3	471.3
INMA-19	156.0	163.8	INMA-33	172.6	182.1	INMA-101	535.4	543.2
INMA-20	127.7	133.6	INMA-35	197.0	203.7	INMA-104	241.5	246.6
INMA-24	218.3	226.1	INMA-36	274.8	281.3	INMA-107	504.3	506.3
INMA-26	224.8	233.3	INMA-44	124.1	130.2	INMA-114	379.7	384.3
INMA-27	254.1	261.8	INMA-53	275.3	287.8	INMA-122	424.8	440.9
INMA-72	189.0	192.9	INMA-55	176.0	182.1	INMA-125	479.8	486.7
INMA-80	215.2	218.5	INMA-58	379.3	382.8	INMA-137	465.5	491.3
INMA-84	86.6	91.3	INMA-62	308.5	315.5	INMA-139	543.2	548.3
INMA-85	246.0	253.6	INMA-64	394.2	399.3	INMA-143	540.3	546.8
INMA-89	191.0	195.3	INMA-156	182.4	195.8	INMA-149	432.7	442.8
INMA-92	235.0	244.5	INMA-158	197.9	207.0	INMA-159	239.6	246.9
INMA-93	36.0	40.6	INMA-167	301.2	309.2	INMA-161	366.1	378.9
INMA-123	139.9	145.6	INMA-182	207.2	212.7	INMA-164	374.5	378.6
INMA-124	83.8	91.3	INMA-189	120.6	129.8	INMA-171	305.9	310.8
INMA-126	232.5	240.9	INMA-196A	322.9	339.2	INMA-175	349.7	359.0
INMA-128	268.5	272.7	INMA-203	301.3	307.7	INMA-181	333.8	344.9
INMA-133	67.6	75.9				INMA-192	272.4	285.4
INMA-138	194.1	198.3				INMA-204	188.7	197.1
INMA-140	214.0	223.4						
INMA-141	277.5	284.0						
INMA-145	116.8	122.1						

Table 13.2 - Summary of Variability Samples Locations

Zone 1 Variability		
Hole ID	From	To
INMA-04	49.5	707
INMA-75	148.6	158.5
INMA-135	101.1	110.9
INMA-106	178.7	193.8
INMA-151	175.2	283.6
INMA-103	154.7	169.5
INMA-017	201.2	231.9

Zone 2 Variability		
Hole ID	From	To
INMA-163	283.2	299.0
INMA-200	200.8	218.1
INMA-197	291.8	305.0
INMA-154	197.0	207.0
INMA-185	228.3	246.9
INMA-180	227.3	258.5
INMA-193	185.1	220.8

Zone 3 Variability		
Hole ID	From	To
INMA-61	411.2	424.9
INMA-100	270.8	280.7
INMA-186	359.2	369.5
INMA-129	361.5	371.5
INMA-109	421.4	434.5
INMA-183	333.9	358.8
INMA-83	329.1	341.7
INMA-137	420.0	455.0

13.2.1 Variability Composites

A sub-sample was taken from each variability composite for a duplicate head assay prior to extraction testwork. The results are summarized in Table 13.3

Table 13.3 - Variability Composite Head Assays

Composite	Au g/t	Ag g/t	As ppm	$C_{organic}$ %	Cu ppm	Hg ppm	Pb ppm	$S_{sulfide}$ %	Zn %
VC1	0.84	33	50	<0.03	46	0.40	155	0.70	0.04
VC2	3.46	132	40	<0.03	324	0.20	310	0.64	0.09
VC3	11.4	74	40	<0.03	110	0.20	75	0.50	0.08
VC4	1.42	39	190	<0.03	26	0.10	270	0.66	0.09
VC5	2.98	110	30	<0.03	116	0.10	315	0.42	0.20
VC6	2.89	63	100	<0.03	48	0.30	175	0.44	0.06
VC7	4.34	133	30	<0.03	72	0.20	205	0.20	0.04
VC8	6.25	221	10	<0.03	126	0.20	260	0.20	0.05
VC9	3.03	140	160	<0.03	146	0.20	615	0.74	0.18
VC10	1.92	111	20	<0.03	36	<0.1	175	0.14	0.04
VC11	3.42	78	410	<0.03	214	0.20	190	1.36	0.11
VC12	4.69	249	1050	<0.03	1540	0.90	1435	1.92	0.16
VC13	3.07	223	30	<0.03	416	<0.1	6640	1.26	1.16
VC14	6.25	188	2670	<0.03	98	1.80	575	4.06	0.12

Composite	Au g/t	Ag g/t	As ppm	$C_{organic}$ %	Cu ppm	Hg ppm	Pb ppm	$S_{sulfide}$ %	Zn %
VC15	5.77	199	40	<0.03	112	<0.1	560	0.18	0.15
VC16	11.55	294	2480	<0.03	134	0.60	170	2.94	0.04

The Inmaculada variability composites gold head assays ranged from 0.8 to 12 g/t, while silver head grades ranged from 32 to 296 g/t. The silver-to-gold ratios varied from 23:1 to 43:1 with some outliers of very low gold grade.

The majority of the samples contained sulfide sulfur in the range 0.2 to 0.8%, with four variability composites recording high sulfide values in the range 1.4 to 4%. Arsenic levels varied from 20 ppm to 2,480 ppm in variability composites. Mercury levels ranged from 0.1 ppm to 0.9 ppm with a few composites below the detection limit.

Organic carbon content was below detection of 0.03% in all variability composites. This is considered below the limit that would cause preg-robbing in a leach circuit.

The variability sample assays showed a general increase in the sulfide sulfur, mercury and arsenic content at depth. This was confirmed with the domain samples. Domain three has higher gold and silver assays but also higher copper, mercury, sulfur and arsenic which indicates a slightly higher refractory gold proportion at depth.

13.2.2 Domain Composites

The Inmaculada domain composites gold head assays in Table 13.4 ranged from 3.3 to 5.3 g/t, while silver head grade ranged from 101 to 244 g/t. The silver-to-gold ratios varied from 24:1 to 46:1.

The samples contained sulfide sulfur in the range 0.4 to 1.04%. Arsenic levels varied from 40 ppm to 370 ppm in domain composites. Mercury levels ranged from 0.1 ppm to 0.2 ppm.

Organic carbon content was below detection of 0.03% in all domain composites. This is considered below the limit that would cause preg-robbing in a leach circuit.

Table 13.4 - Domain Composites Head Assays

Analyte	Unit	Domain Composite #1	Domain Composite #2	Domain Composite #3
Au	g/t	4.24	3.34	5.26
Ag	g/t	101	134	244
As	ppm	160	40	370
$C_{organic}$	%	<0.03	<0.03	<0.03
Cu	ppm	60	140	448
Hg	ppm	0.1	0.2	0.2
Pb	ppm	205	535	865
$S_{sulfide}$	%	0.64	0.40	1.04

Analyte	Unit	Domain Composite #1	Domain Composite #2	Domain Composite #3
Zn	ppm	666	1,460	1,694

13.3 Sample Preparation

A total of 443 drill core intervals were selected for variability testwork and a total of 736 drill core composites were selected for composite samples for use in the Inmaculada testwork program. The half core and quarter core received at ALS Ammtec (Ammtec) were crushed to <25 mm and a portion split for gold and silver assay.

13.4 Testwork Programs

Metallurgical testing was initiated in mid-2009 at McClelland Laboratories (McClelland) in Reno, Nevada. The testwork included:

- Determination of standard “Bond” grinding and abrasion indices;
- Response to whole ore cyanidation (heap and agitation leaching);
- Response to flotation to produce concentrate for third party processing; and
- Rougher concentrate and tailings response to cyanidation.

The latest feasibility level testwork program was designed and supervised by Ausenco, and formed the basis for preparing the design criteria, process flow diagrams, mass balance and equipment sizing. The testwork was conducted by Ammtec in Perth Australia, and commenced in May 2011. It was completed October 2011. The program included testwork to establish:

- Mineralogy;
- Comminution characteristics of the three domain composites;
- Gravity recovery;
- Whole ore cyanidation;
- Oxygen uptake;
- Rheology;
- Thickening and vacuum filtration of tailings; and
- Cyanide detoxification.

13.4.1 Mineralogy

Mineralogical examination on three samples was completed. The three samples had similar compositions. The host lithology has been described elsewhere (in Section 8.2). Low sulfide ores are hosted in bimodal basalt-rhyolite sequences, while the intermediate and high sulfide ores are hosted in andesite-dacite intrusions. Quartz is the dominant gangue mineral and pyrite the dominant sulfide mineral in all three composites. Other sulfide minerals detected in the sample were sphalerite, chalcopyrite and tetrahedrite, galena and Ag-sulfides The Au/Ag (wt %) in the electrum was about 60/40 in all three composites

Table 13.5 - Summary of the Detected Gold Bearing Particles in the Samples

Sample		Gold Phase	Liberation	Size
Composite # 1	More in pan	Electrum	Mainly	Mainly

Sample		Gold Phase	Liberation	Size
	concentrate than pan tail		encapsulated or exposed (pyrite, chalcopyrite and quartz)	<20µm but one liberated grain is 47 x 33µm
Composite # 2	Pan concentrate	Electrum	Liberated	<10µm
Composite # 3	Several in pan tail, also in pan concentrate	Electrum	Mainly encapsulated (argentite or quartz) or liberated	Mainly <20µm but one liberated grain is 45 x 17µm

13.4.2 Comminution

- Unconfined Compressive Strength.

A selection of 15 domain composites, 8 variability composites and 10 waste composites were selected to determine unconfined compressive strength (UCS). The average UCS for all samples tested was 56 MPa and did not indicate any strong correlation to mineralization or drillhole depth.

- Bond Work Indices.

For the Bond rod mill indices (BRWi), the closing screen aperture was 1,180 µm. For the Bond ball mill indices (BBWi), the closing screen aperture was 75 µm. Bond abrasion indices (Ai) were also completed and are summarized in Table 13.6.

Table 13.6 - Summary of Bond Work Indices

Sample	Ai (g)	BRWi (kWh/t)	BBWi (kWh/t)
Domain # 1	0.367	17.7	20.1
Domain # 2	0.463	18.2	20.6
Domain # 3	0.439	17.6	19.9
Variability composite # 1	0.196	18.5	18.9
Variability composite # 7	0.364	16.3	19.9
Variability composite # 12	0.449	16.8	21.2
Waste # 1	0.090	17.1	17.0
Waste # 2	0.168	17.1	18.0
Waste # 3	0.248	20.9	22.4

All the samples tested are considered moderately abrasive and typically classified as moderately hard for ball milling.

- SAG Mill Comminution Testwork.

Drop weight indices (DWi) were obtained from 3 Domain composites, 3 variability comminution composites and three waste composites. These showed a range of 3.93 to 5.50 kWh/m³ with an average of 4.6 kWh/m³ which indicates a low to moderate resistance to breakage.

13.4.3 Gravity concentration and Intensive Leaching

Gold recovery by gravity ranged from 35 to 50% and recovery of silver by gravity ranged from 18 to 23%. Subsequent leaching of the gravity concentrate resulted in average overall gold and silver extractions of 97% and 91%, respectively. The present flow sheet developed by Ausenco does not incorporate a gravity circuit.

Table 13.7 - Gravity Separation / Direct Cyanidation: Concentrate Lead Nitrate Addition

Composite Identity	Test No	Gravity Lead Nitrate (g/t)	% Au Extraction		Au Leach Residue (ppm)	% Ag Extraction		Ag Leach Residue (ppm)	Consumption (kg/t)	
			Gravity	96 hrs		Gravity	96 hrs		Lime	NaCN
Domain Comp #1	HS25488	0	43.5	96.8	0.15	17.9	92.1	10	0.64	1.67
	HS25583	250	52.2	96.2	0.13	22.6	87.2	14	0.56	1.74
Domain Comp #2	HS25489	0	35.9	97.9	0.10	18.8	94.7	8	0.53	2.02
	HS25584	250	46.0	97.4	0.09	19.3	94.6	8	0.43	1.95
Domain Comp #3	HS25490	0	40.6	97.3	0.18	20.0	87.6	30	0.67	2.17
	HS25585	250	50.3	96.9	0.17	22.4	92.7	18	0.61	2.24

13.4.4 Whole Ore Cyanidation

All three domain composites responded well to direct cyanidation with average extractions of Au and Ag being 98% and 92% respectively. The recovery of gold and silver was sensitive to the grind size and cyanide concentration. A grind size 80% passing (P80) 50 µm and cyanide concentration of 1,500 ppm were selected for the design. Other factors that resulted in improved precious metal leaching kinetics were the use of oxygen sparging and the addition of lead nitrate.

Table 13.8 - Standard Direct Cyanidation

Composite Identity	Test No	% Au Extraction 96 Hours	Au Leach Residue (g/t)	% Ag Extraction 96 Hours	Ag Leach Residue (g/t)	Consumption (kg/t)	
						Lime	NaCN
Domain Comp #1	HS25485	96.5	0.18	89.9	12	0.37	2.37
Domain Comp #2	HS25486	97.7	0.10	93.3	10	0.65	1.55
Domain Comp #3	HS25487	97.4	0.18	91.7	22	0.67	2.03

Bulk cyanide leach tests using 20 kg of each domain composite were completed and the results summarized in Table 13.9. These tests are likely to better represent the expected plant scale extractions and reagent consumptions due to the sample size.

Table 13.9 - Bulk Direct Cyanidation

Composite Identity	Test No	% Au Extraction 96 Hours	Au Leach Residue (g/t)	% Ag Extraction 96 Hours	Ag Leach Residue (g/t)	Consumption (kg/t)	
						Lime	NaCN
Domain Comp #1	HS25805	97.5	0.11	92.6	8.2	0.64	1.08
Domain Comp #2	HS25806	97.6	0.09	93.9	8.0	0.59	1.05
Domain Comp #3	HS25807	97.5	0.15	90.4	22	0.67	1.56

The results showed similar leach extractions to those achieved in the standard tests using 3 kg samples (Table 13.8).

Results from the direct cyanidation tests on the variability samples are shown in Table 13.10.

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Table 13.10 - Direct Cyanidation Variability Composites

Composite Identity	Test No	% Au Extraction 96 Hours	Au Leach Residue (ppm)	% Ag Extraction 96 Hours	Ag Leach Residue (ppm)	Consumption (kg/t)	
						Lime	NaCN
VC #01	HS25708	96.8	0.03	93.7	2.3	0.82	1.57
VC #02	HS25709	97.3	0.10	94.9	7.0	0.59	2.00
VC #03	HS25710	99.1	0.11	95.8	3.5	0.82	1.81
VC #04	HS25711	94.5	0.08	92.7	2.8	0.65	1.62
VC #05	HS25712	97.7	0.07	88.2	13.7	0.67	1.63
VC #06	HS25713	96.0	0.13	92.9	5.0	0.74	1.68
VC #07	HS25714	98.2	0.08	94.4	8.0	0.83	1.70
VC #08	HS25715	98.9	0.07	96.1	9.8	0.58	1.55
VC #09	HS25716	96.5	0.11	91.6	13.1	0.62	1.71
VC #10	HS25717	97.7	0.05	93.0	8.5	0.42	1.75
VC #11	HS25718	94.1	0.22	88.0	10.7	0.87	2.25
VC #12	HS25719	93.0	0.34	73.4	68.0	0.61	2.70
VC #13	HS25720	97.0	0.10	95.2	11.5	0.53	2.25
VC #14	HS25721	92.7	0.52	85.6	28.0	0.89	2.10
VC #15	HS25722	97.1	0.17	89.9	21.0	0.50	1.51
VC #16	HS25723	89.3	1.19	84.2	48.0	0.83	1.70

The average extractions for all direct cyanidation tests completed at the optimum leach conditions (including domain composites and variability samples) was 96.4% and 91.0% for gold and silver respectively.

13.4.5 Rheology

Slurry viscosity measurements were undertaken at 40, 50, 60 and 65% (w/w) pulp density.

Table 13.11 - Rheology Results

Sample Identity	Screen Size P80 (µm)	Pulp Temp.	pH	Pulp Density % Solids (w/w)	Viscosity @ Shear Rate (sec-1) (cP)							
					4.2	7.4	13.0	21.8	38.9	67.1	118.7	209.9
Domain Comp #1	50	Ambient	11.6	40	Low	Low	Low	Low	Low	53	48	59
				50	Low	Low	29	38	53	68	74	80
				60	2931	1780	1068	681	410	253	159	142
				65	5929	3033	2093	6313	791	495	298	188
Domain Comp #2	50	Ambient	11.6	40	Low	Low	Low	Low	Low	32	41	60
				50	Low	Low	39	42	57	60	66	82
				60	2629	1506	653	633	412	265	188	173
				65	6810	4457	3056	2062	1307	853	541	343
Domain Comp #3	50	Ambient	11.6	40	Low	Low	Low	Low	Low	36	47	57
				50	Low	28	37	41	54	62	66	78
				60	3082	1829	1099	704	431	268	173	157
				65	7667	5553	3691	3546	1351	837	509	312

13.4.6 Thickening and Filtration

Thickening and filtration testwork was completed at two separate consultant companies that specialize in this area (Outotec and FLSmidth).

The thickening testwork included flocculant screening and feed dilution tests and were completed using both batch and dynamic testing methods. The results indicate that the ores can be thickened to between 55 – 60% solids with the addition of 50 g/t flocculant.

Filtration testwork was conducted to determine the filtration characteristics of the tailings for use as paste backfill in the underground mine. The results indicated that standard disk filters are suitable for tailings filtration, and will produce a filter cake of 73% solids at a filtration rate of 217 kg/h/m².

13.4.7 Cyanide Destruction

The INCO SO_2/air oxidation process was used to test amenability of cyanide detoxification for the plant tailings slurry. The results indicated that the leach tailings from all ore types are amenable to cyanide detoxification.

A single bulk leach under optimized conditions was conducted on each domain composite to provide samples for cyanide detoxification work. Observations from the cyanide detoxification testwork include:

- An SO_2 dosage of 3.0 g/g SO_2/CN_{WAD} appeared necessary to obtain the target 25 mg/L CN_{WAD} in the treated effluent for each of the samples; and
- A single stage of detoxification stage with 60 minutes residence time was sufficient to achieve the required detoxification requirements.

13.5 Metallurgical Recoveries

The metallurgical recoveries were estimated based on all tests completed at the selected optimum leach conditions. The recoveries used for both reserve estimations and the project financial analysis were 95.6% and 90.6% for gold and silver respectively. These results represent an average across the complete Angela Vein and include reductions of 0.8% and 0.4% for gold and silver respectively to account for scale-up and solution losses.

13.6 Conclusions

The following optimum leaching conditions were selected based on the Ammtec testwork campaign:

- Grind size of 80% passing (P_{80}) 50 micron;
- 100 g/t of lead nitrate addition;
- Increased pulp dissolved oxygen concentration through oxygen sparging;
- Initial free cyanide concentration in the leach solution of 1,500 ppm;
- 96 hours of leach residence time; and
- Pulp pH of 11.

The leached ore can be thickened to allow solid liquid separation and the recovery of pregnant leach solution. The thickened residue can be filtered using standard disk filters to provide paste backfill for the mine.

14 Mineral Resource Estimate

14.1 Mineral Resource Estimates

14.1.1 Veins included in mineral resource estimates

Figure 14.1 shows the horizontal footprints of the eight veins included in the feasibility study resource estimates:

- The main Angela Vein;
- The Angela Splay, a split from the main vein, that occurs at depth to the northeast;
- Four small branches from the Angela Vein, referred to as “R1” through “R4”; and
- The Lourdes and Martha veins, two small and unconnected hangingwall veins that lie south of the main vein.

Figure 14.2 shows the relative contribution of these veins to total metal content, as measured by the contained metal in the drill hole assays that fall within these veins. The most important vein, by far, is the Angela Vein, which accounts for more than 80% of the metal content.

14.1.2 Data

Assay Database

All drill holes with assay data available by 30 June 2011 were used for resource estimation. These include:

- Holes QU-01 through QU-31, the early group of holes drilled by Hochschild in the project area. Of these, only QU-04, QU-05, QU-07, QU-24, QU-25 and QU-31 intersect the veins included in these resource estimates. These are, therefore, the only early holes that provide assay data that affect the grade estimation. The other QU holes affect the definition of the geometry of the veins, their lack of vein intersections forcing closure on the wireframes used to model each of the veins;
- Holes INMA-01 through INMA-214, the main series of drill holes begun by Ventura and IMC and now continued by Hochschild. Of these, the vast majority (198) intersect one or more of the resource veins and therefore provide assay data that affects grade estimation. The INMA holes that do not intersect any of the resource veins, and that affect the resource estimation only by forcing closure on the interpreted geometry of the veins, are: INMA-08, INMA-39 through INMA-42, INMA-45 through INMA-51, INMA-63, INMA-120, INMA-153, INMA-157, INMA-169 and INMA-208; and
- The ANG11, ASW11, JIM11 and MAR11 series of holes drilled in 2011 by Hochschild. Of these, the following holes intersect one or more of the resource veins, and therefore affect grade estimation: ANG11-001 through ANG11-005, ANG11-016, ANG-018, ANG-019, ANG-022, ANG-026 through ANG-028, MAR11-001 through MAR11-010, and MAR11-012. The remaining holes drilled in the past year do not intersect any of the resource veins and affect resource estimation only by forcing closure on the interpreted geometry of the veins.

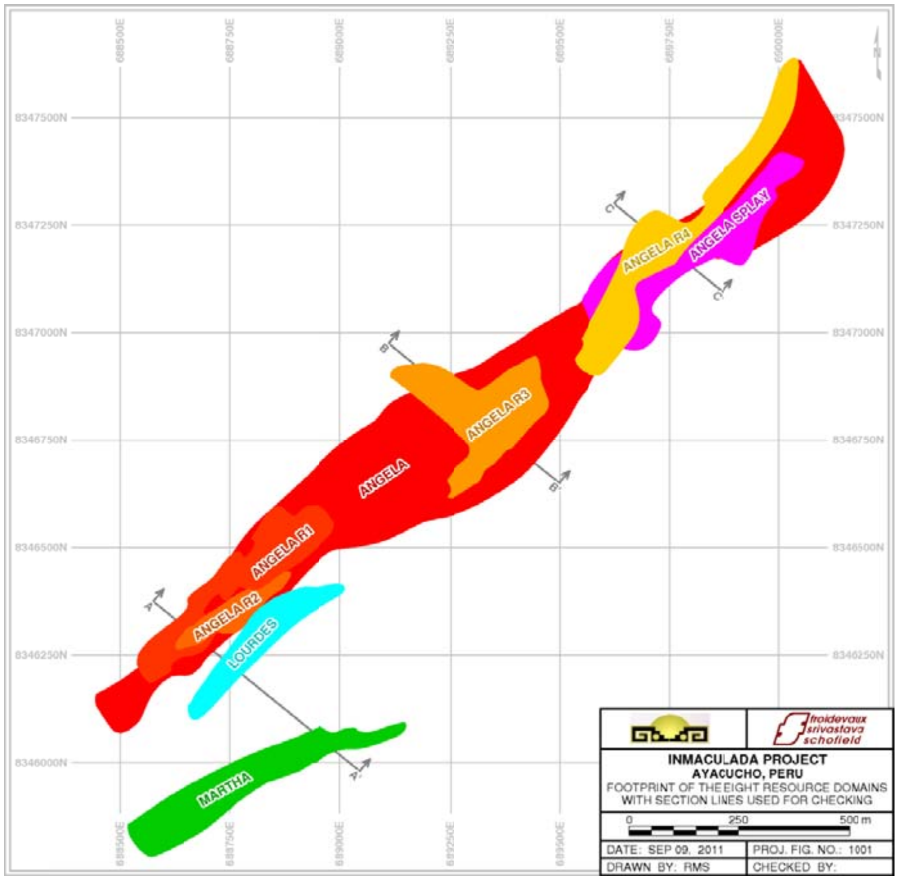


Figure 14.1 - Horizontal footprint of the eight domains used for resource estimation, with section lines used in later figures

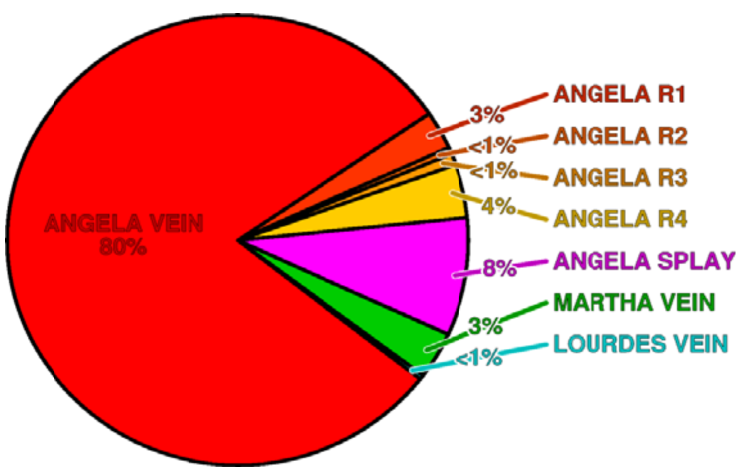


Figure 14.2 - Relative contribution of each domain to total metal content in the drill hole assays

None of the assays from chip samples from surface outcrops were used for grade estimation; surface mapping of the exposed veins was used, however, in the interpretation of the 3D geometry of the resource veins.

The assay data base was recompiled by Hochschild from the original lab certificates. For intervals with duplicate assays, only the first assay value was retained for the purpose of resource estimation.

Topography

Information on the topography in the project area is sourced from a digital elevation model developed from a 1:5,000 aerial topography survey completed in 2010 by IMZ. Ground survey data used as control points for this survey had to be shifted by 2-3 m to align the vertical datum used for local surveying with the PSAD56 datum used for the aerial topography.

The consistency between the aerial topography model and the drill hole collars was checked. The vast majority of hole collars (92%) are within ±1m of the ground elevation provided by the aerial topography. The maximum discrepancy of 2.7 m was observed at the collar of ASW11-002, one of the holes that does not intersect any of the resource veins. Of the holes that do intersect resource veins, and that provide grade data used for grade estimation, all of their collars are within ±2 m of the elevation provided by the digital elevation model developed from aerial topography. Most of the hole collars that are now available post-date the development of the digital elevation model in 2010; so the excellent agreement between the surveyed elevations of new hole collars and the digital elevation model confirms that the topographic and survey control remains excellent, and suitable for the purposes of a feasibility study.

Density

The density information used for the resource estimates comes from a study done by Ventura Gold in 2008 in which density was calculated for 100 samples of drill core from six lithologies. Each sample was dried, weighed in air, sealed in wax and weighed in water. Using Archimedes' Principle, the dry bulk density of the material is:

Dry bulk density = Dry weight in air ÷ (Dry weight in air – Weight of sealed sample in water)

with the weight of the sealed sample being adjusted to take into account the weight of the wax used to seal the sample before it was immersed in water.

51 of the 100 samples in this study were from the Angela Vein; the average density of these 51 samples was 2.51 t/m^3. The average density of the 29 samples of unaltered andesite material surrounding the mineralized vein was 2.54 t/m^3. The average density of the 20 samples showing a stockwork alteration in andesitic material was 2.44 t/m^3.

The 51 samples of mineralized material suggest that there may be a very slight tendency for density to decrease with increasing grade. The samples with highest silver-equivalent grades (> 500 g/t) had an average density of 2.49 t/m^3, while the samples with the lowest silver-equivalent grades (< 50 g/t) had an average density of 2.53 t/m^3. With these differences being small, only 1-2%, and with too few samples to confirm whether or not they are statistically significant, the feasibility study resource estimates use a single density constant, 2.51 t/m^3, for all eight of the resource veins.

14.1.3 Geological Domains

The eight resource veins shown in map view in Figure 14.1 are shown in cross-sections in Figures 14.3 through 14.5. All of the veins dip to the southeast, with the average dip being about 70°. In some places, the dip is as low as 45°; in other places, the veins are vertical.

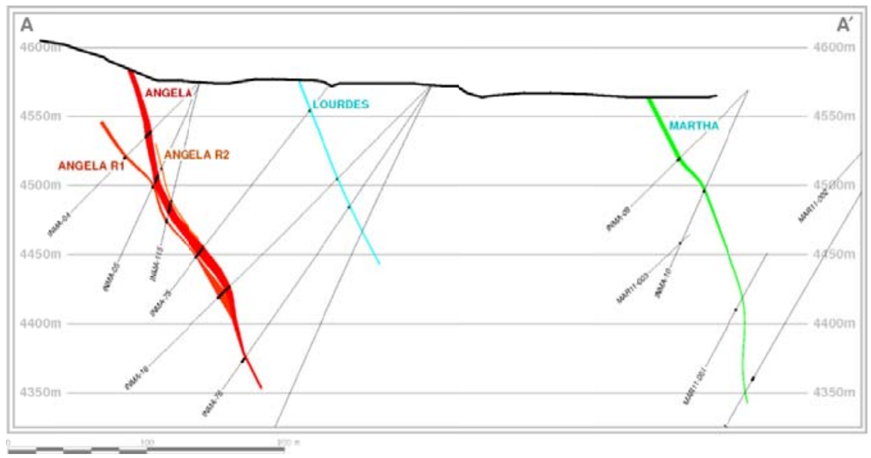


Figure 14.3 - Vertical cross-section, facing northeast, showing the interpreted vein geometries on A-A' (see Figure 14.1) along with the drill holes within ±50m of the section line, with vein intersections shown as thicker black lines

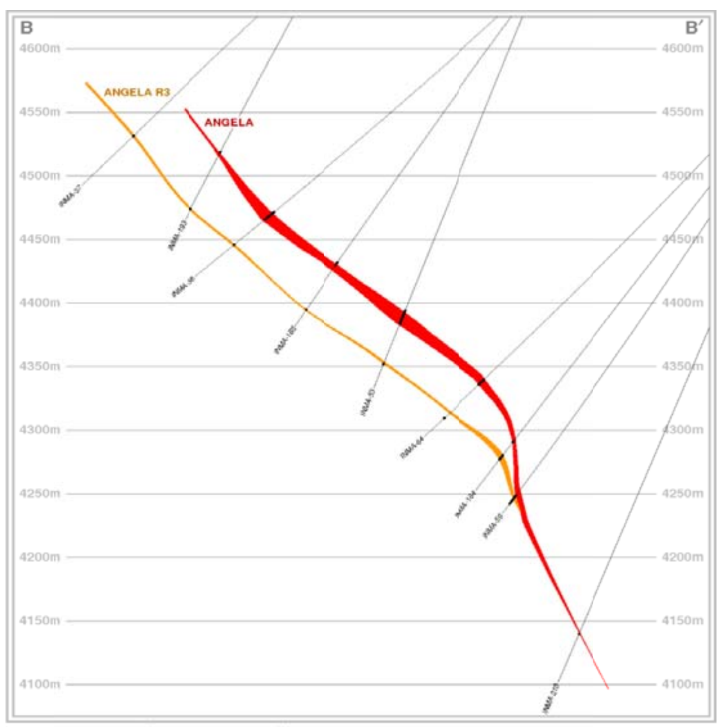


Figure 14.4 - Vertical cross-section, facing northeast, showing the interpreted vein geometries on B-B' (see Figure 14.1) along with the drill holes within ±50m of the section line, with vein intersections shown as thicker black lines

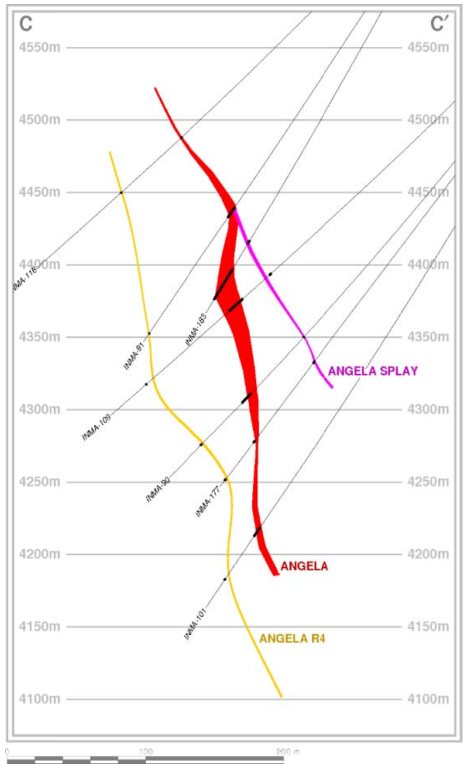


Figure 14.5 - Vertical cross-section, facing northeast, showing the interpreted vein geometries on C-C' (see Figure 14.1) along with the drill holes within ±50m of the section line, with vein intersections shown as thicker black lines

Wireframing of the domains was done by identifying the hangingwall and footwall intercepts of the vein in the drill holes, and then using the Leapfrog software to create wireframed solids that honor these control points, that follow the predominant strike and dip, that honor the location of surface outcrops, and that close before they reach drill holes that do not intersect the vein. The wireframes were edited to ensure that the wireframes for the branches and the splay fit together exactly with the wireframe for the main Angela Vein, with no gaps or overlaps where adjacent wireframes meet. The wireframes were also clipped exactly to the topography so that the volume within each wireframe does not include any air.

The hangingwall and footwall intercepts of the veins were initially identified from geological logging. The gold and silver assay values were then checked and sample intervals adjacent to the geologically-logged vein were incorporated in the vein if they had a silver-equivalent above 70 g/t, with silver-equivalent defined as Ag + 70*Au. This grade threshold for expanding the geological domains was chosen to be lower than the economic break-even cutoff to avoid the overestimation of resources that occurs when break-even cutoffs are used to define grade zones. With marginal and sub-economic material included at the edges of the veins, it is possible for block grade estimates to be below the cutoff later used for reporting purposes.

The choice of hangingwall and footwall intercepts also included a minimum-mining-width constraint to avoid running wireframes through thin, single intervals of high-grade mineralization. In drill holes where the true width of the vein was less than 0.8 m, additional intervals were included, either on the hangingwall or footwall, to bring the true width up to at least 0.8 m.

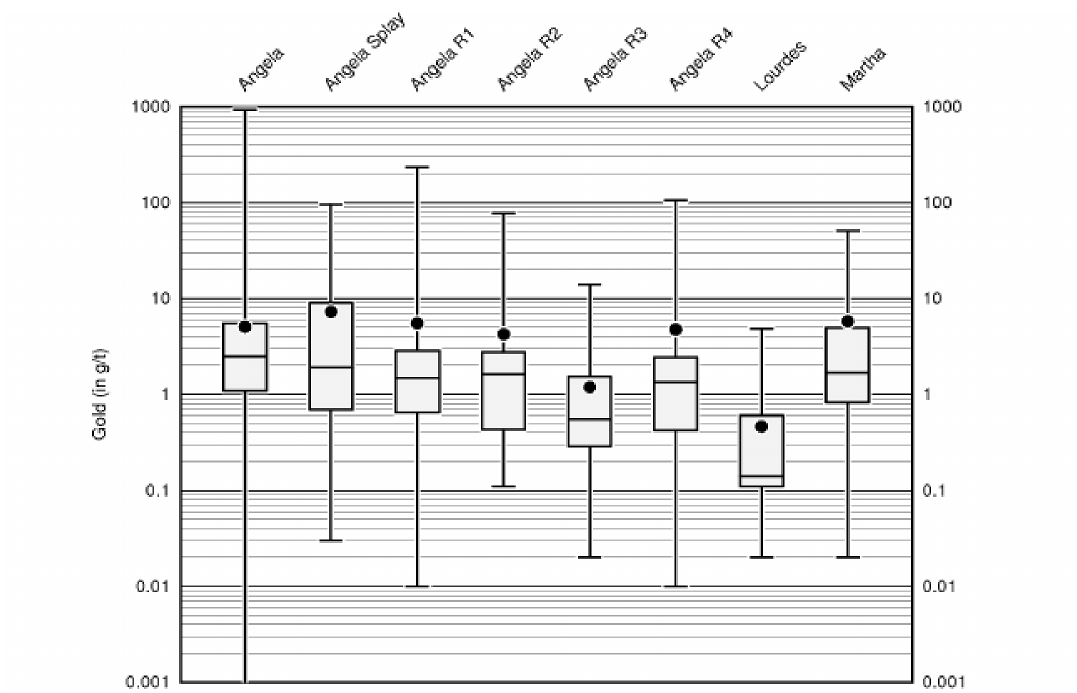
14.1.4 Statistical Analysis of Gold and Silver Assays

Figure 14.6 shows the side-by-side boxplots of the distribution of gold assays in the eight geological domains. Figure 14.7 shows the corresponding comparison for silver; and Figure 14.8 shows how the silver-gold ratio compares in the eight veins.

The silver in the Angela Vein runs, on average, about 50x the gold grade, the lowest Ag:Au ratio of all eight domains. With the value of a gram of gold being worth roughly 70x a gram of silver, gold accounts for more than half of the in situ value of the Angela Vein. With the Angela Vein accounting for the vast majority of the tonnage and metal content (see Figure 14.2), the majority of the metal value in the resource block model (about 60%) is due to gold.

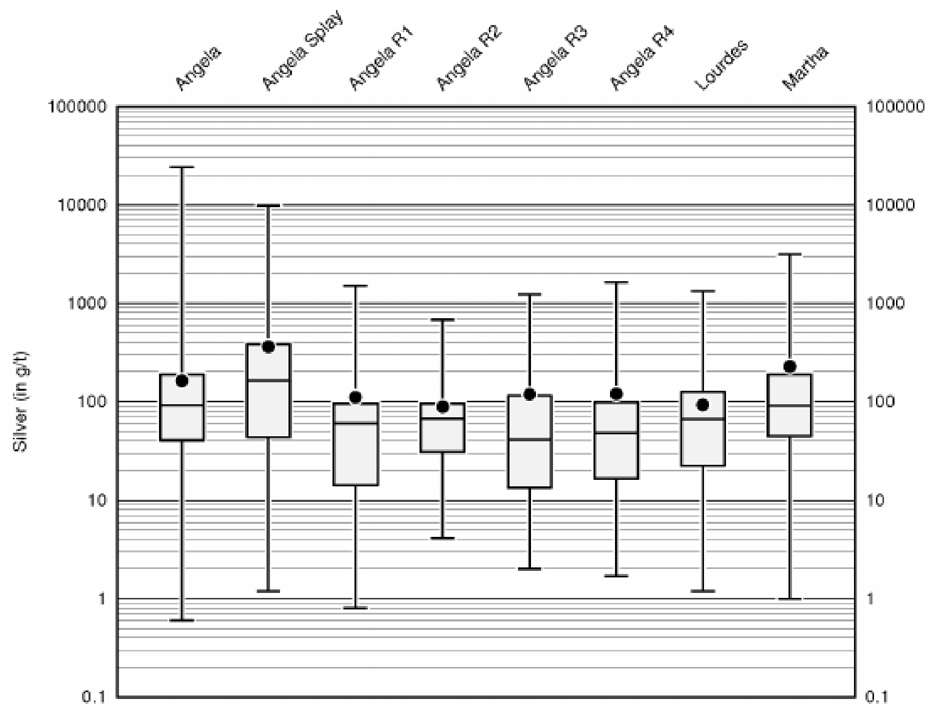
The Angela Splay has the highest average gold grade and silver grade of all domains, approximately 50% higher gold grade than the Angela Vein, and approximately twice the silver grade of the main Angela Vein. With an Ag:Au ratio of approximately 150:1, the Angela Splay is predominantly a silver vein, with more than two-thirds of its metal value in silver.

The branches off the Angela Vein, R1 through R4, generally have lower gold and silver grades than the main Angela Vein. With these branches having a Ag:Au ratio that approaches the 70:1 relative value of gold to silver, these veins have about half their value in silver and half in gold.



	Angela	Angela Splay	Angela R1	Angela R2	Angela R3	Angela R4	Lourdes	Martha	
Number	1920	72	153	38	44	48	47	38	Number
Mean	5.04	7.26	5.49	4.22	1.18	4.73	0.46	5.76	Mean
Standard deviation	13.95	14.71	22.64	13.61	2.07	16.45	0.71	10.18	Standard deviation
Coefficient of variation	2.77	2.03	4.13	3.22	1.76	3.48	1.55	1.77	Coefficient of variation
Maximum	929.02	95.55	233.65	77.01	13.91	104.9	4.83	50.22	Maximum
75th percentile	5.48	8.94	2.83	2.73	1.53	2.41	0.6	4.95	75th percentile
Median	2.47	1.9	1.47	1.61	0.55	1.32	0.14	1.68	Median
25th percentile	1.08	0.69	0.64	0.43	0.29	0.42	0.11	0.82	25th percentile
Minimum	0.0	0.03	0.01	0.11	0.02	0.01	0.02	0.02	Minimum

Figure 14.6 - Side-by-side boxplots of the distribution of gold assays in the eight geological domains



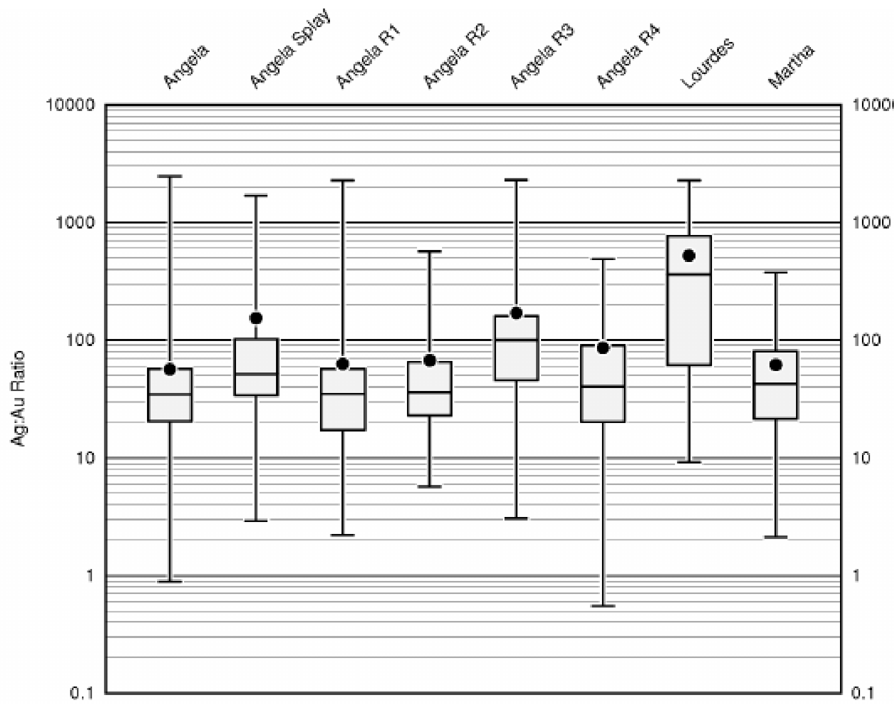
	Angela	Angela Splay	Angela R1	Angela R2	Angela R3	Angela R4	Lourdes	Martha	
Number	1914	72	152	38	44	48	50	38	Number
Mean	162.27	359.87	110.37	88.41	118.02	119.25	92.22	226.58	Mean
Standard deviation	360.74	843.35	201.36	120.62	217.69	260.78	109.06	502.07	Standard deviation
Coefficient of variation	2.22	2.34	1.82	1.36	1.84	2.19	1.18	2.22	Coefficient of variation
Maximum	24193.0	9867.0	1513.0	679.0	1228.0	1637.0	1324.0	3133.0	Maximum
75th percentile	187.0	383.1	95.47	94.76	113.8	97.91	124.63	187.75	75th percentile
Median	91.9	163.71	60.1	66.62	41.09	47.77	65.94	90.38	Median
25th percentile	40.75	43.33	14.2	30.96	13.4	16.65	22.53	44.56	25th percentile
Minimum	0.6	1.2	0.8	4.1	2.0	1.7	1.2	1.0	Minimum

Figure 14.7 - Side-by-side boxplots of the distribution of silver assays in the eight geological domains

The Lourdes Vein has the lowest gold and silver grades, with the vast majority (90%) of the value in silver. The Martha Vein is similar to the main Angela Vein, slightly higher in gold and silver grade, and with a very similar Ag:Au ratio.

With clear changes in the statistical characteristics of the eight veins, each of the veins is treated as a separate domain for the purposes of grade estimation; grade estimates for each vein use only the assay data from that same vein.

The coefficients of variation are generally between 2.0 and 2.5; the only very high coefficients of variation are in the small branches from the Angela Vein. With the coefficient of variation above 2, local grade estimates will occasionally be strongly influenced by extreme assay values. Fortunately, the problem of erratic extremes is least problematic in the domains that carry the vast majority of the metal: the main Angela Vein and the Angela Splay. In all veins, the smearing of extreme values is well controlled by the use of tight wireframes that spatially restrict the extreme grades to very narrow bands. This, combined with the capping of grades (discussed in the following section), ensures that erratic high-grade assays do not unduly influence resource estimates.



	Angela	Angela Splay	Angela R1	Angela R2	Angela R3	Angela R4	Lourdes	Martha	
Number	1908	72	150	38	44	47	47	38	Number
Mean	56.06	153.18	62.47	66.97	169.31	85.48	519.53	61.34	Mean
Standard deviation	124.91	313.3	154.64	92.38	315.58	107.55	556.76	70.34	Standard deviation
Coefficient of variation	2.23	2.05	2.48	1.38	1.86	1.26	1.07	1.15	Coefficient of variation
Maximum	2459.63	1692.31	2267.69	567.01	2296.88	484.95	2274.51	374.21	Maximum
75th percentile	56.69	102.0	56.57	64.83	158.72	89.33	761.34	79.62	75th percentile
Median	34.41	50.8	34.57	35.67	99.26	39.98	359.01	42.04	Median
25th percentile	20.34	33.71	17.03	22.81	45.17	20.13	60.88	21.29	25th percentile
Minimum	0.89	2.93	2.21	5.72	3.05	0.55	9.22	2.13	Minimum

Figure 14.8 - Side-by-side boxplots of the distribution of the Ag:Au ratio in the eight geological domains

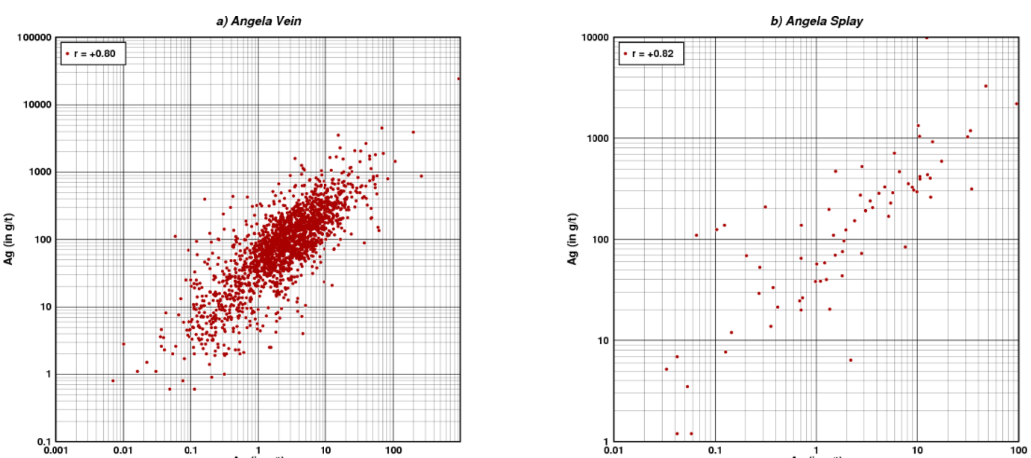


Figure 14.9 - Silver-gold scatterplots in the Angela Vein and the Angela Splay

Although the Ag:Au ratio changes from domain to domain, there is a strong correlation between silver and gold within each domain. Figure 14.9 shows two examples, one from the Angela Vein (with the lowest silver-gold ratio), and the other from the Angela Splay (one of the domains with a very high silver-gold ratio).The strong silver-gold correlation in two domains with very different Ag:Au ratios is consistent with the view that geological events responsible for gold mineralization are also responsible for silver mineralization. This, in turn, suggests, that the two metals have similar patterns of spatial continuity that can be modeled using the same variogram.

14.1.5 Grade Capping

In each domain, the gold and silver assays were capped at levels customized to that domain. Cumulative probability plots were used to select the capping levels close to the point at which the upper tail of the grade distribution starts to break up. The Ag:Au ratio for each vein was also taken into account when setting the capping levels; when the cumulative probability plots suggested different possibilities for the capping value, the Au and Ag caps were chosen so that they reflected the average ratio seen in that domain.

Figure 14.10 shows the details of the selection of the capping levels for the main Angela Vein. The cumulative probability plot of gold shows that the high-grade tail starts to break up between 100 and 200 g/t; the cumulative probability plot of silver shows that the high-grade tail starts to break up between 1,500 and 5,000 g/t. With the silver values between 1,500 and 5,000 g/t following the same linear trend as shown by the rest of the distribution, and with the Ag:Au ratio being approximately 50:1, the silver cap was set at 5,000 g/t.

Table 14.1 shows the capping levels used in each of the domains.

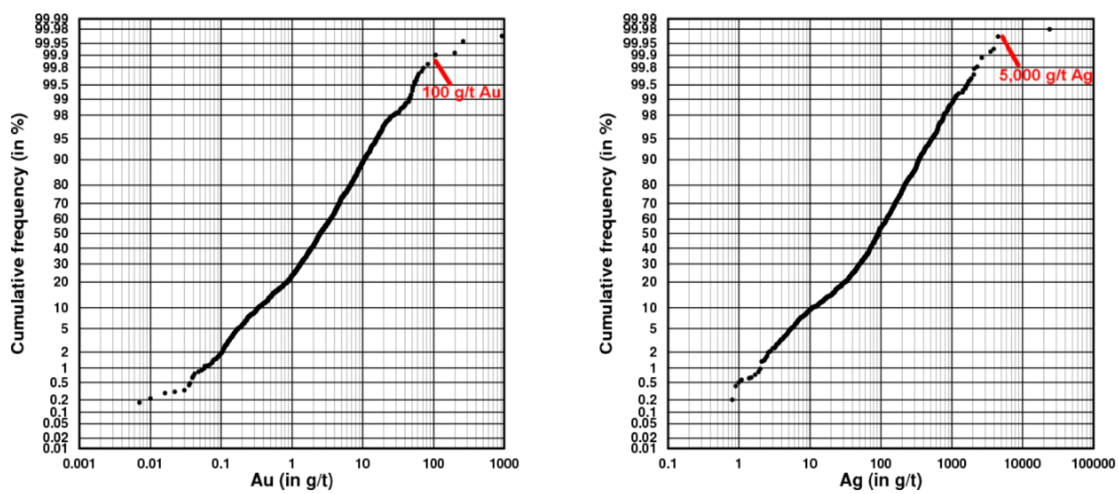


Figure 14.10 - Cumulative probability plots of gold and silver assays in the Angela Vein used to select capping levels

Table 14.1 - Capping levels used for gold and silver assays in each domain

	Au cap (g/t)	Ag cap (g7t)
Angela Vein	100	5,000
Angela Splay	25	1,250
Angela R1	50	1,000
Angela R2	10	500
Angela R3	10	500
Angela R3	10	500
Lourdes Vein	5	500
Martha Vein	25	1,000

14.1.6 Variograms

Since gold and silver are very well correlated within the mineralized domains (see Figure 14.9), their patterns of spatial variation will be very similar, and one variogram model will suffice for both metals. Variogram analysis was therefore done on silver-equivalent, defined as:

$$AgEq = Ag + 70 \cdot Au$$

The factor of 70 reflects the combined effect of the metal prices and the metal recoveries: a gram of gold, in situ, has roughly the same value as 70 grams of silver, in situ.

Figure 14.11 shows the experimental variograms for silver-equivalent, along with the spherical variogram models for interpolation of both gold and silver. In the direction perpendicular to the vein system, the range of correlation is 10 m; in the plane of the vein system, both along strike and down-dip, the range of correlation is 110 m. The nugget effect is 35% of the sill.

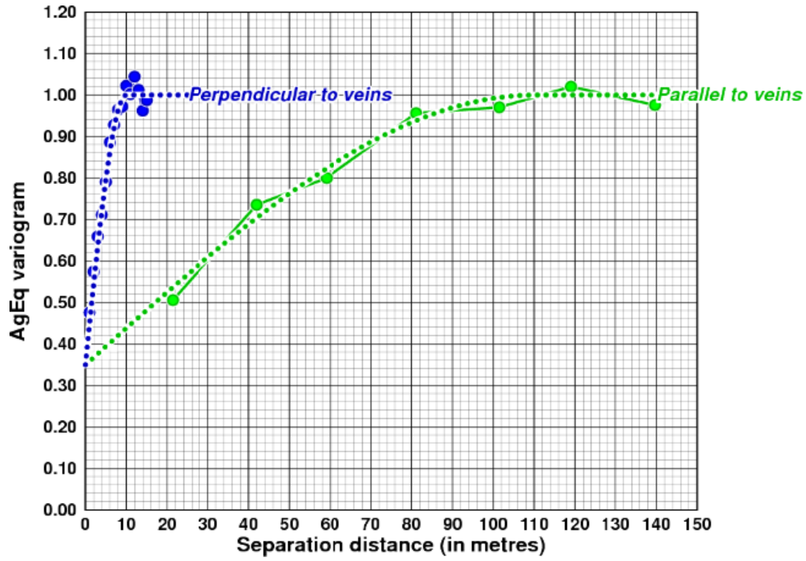


Figure 14.11 - Variogram model (dotted line) and experimental variogram (solid line) of silver-equivalent

14.1.7 Block Model Configuration

As shown in Figure 14.12, the resource block model has been oriented so that it runs parallel to the northeasterly strike of the veins. The corner of the block model is at 688755.6E, 8345603.0N; its southeastern edge runs at an angle of 50.04° from east (an azimuth of N39.96°E). In the northeasterly direction, along the strike of the veins, there are 240 columns of blocks that are 10 m long. In the northwesterly direction, across the strike of the veins, there are 300 rows of blocks that are 2 m wide. In the vertical direction, there are 64 10 m levels, from an elevation of 4 050 m to an elevation of 4 690 m.

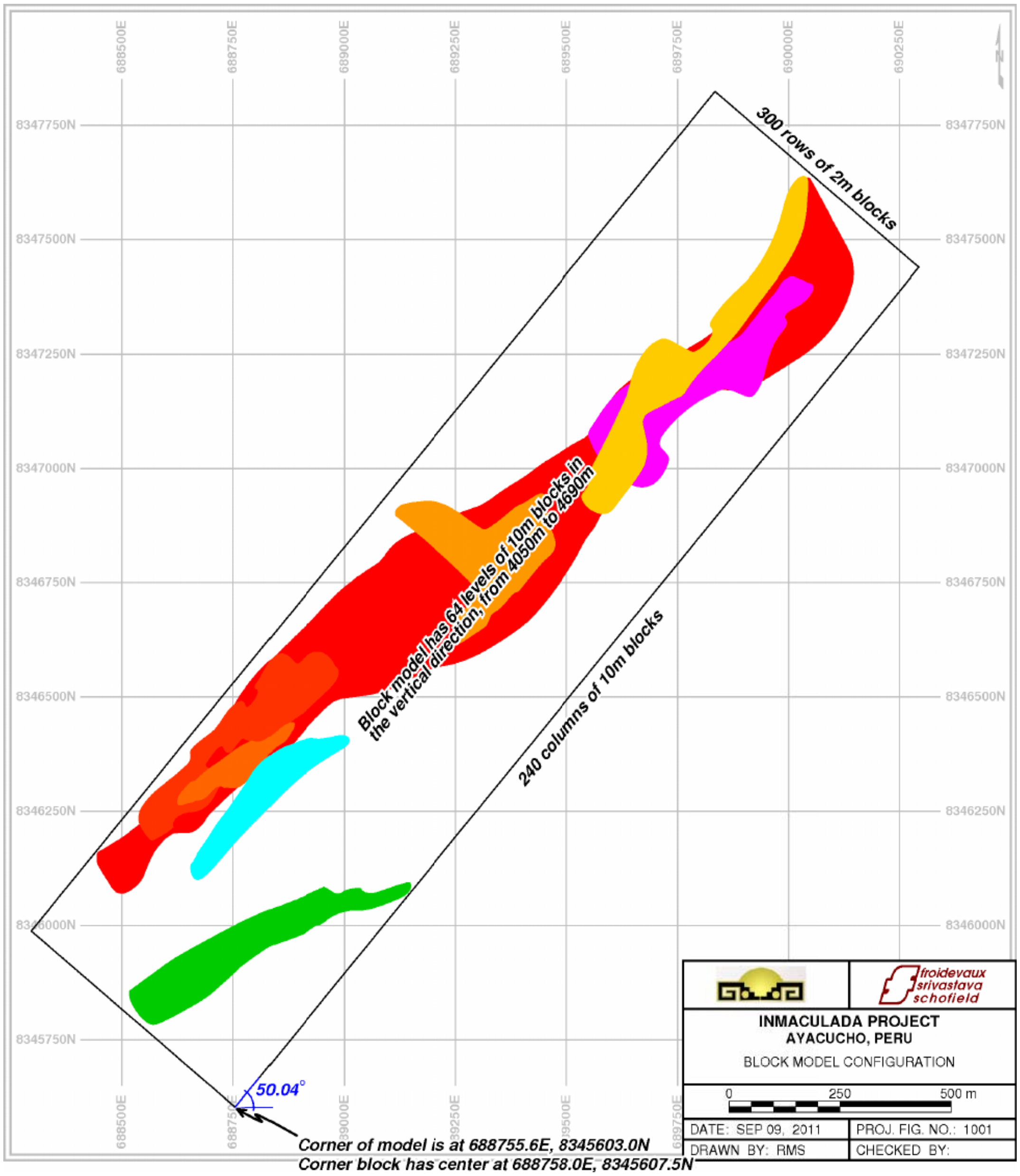


Figure 14.12 - Resource block model configuration

With the Angela Vein having an average width of 4-6 m, the resource model will include several 2m wide blocks from the hangingwall side of the Angela Vein to the footwall side.

14.1.8 Ordinary Kriging

Estimation of Tonnage

Within each 10×10×2 m block, the mineralized tonnage contributed by each of the eight domains was estimated by calculating the volume of the intersection of the domain wireframe with the rectangular block and multiplying this by the assumed dry bulk density of 2.51 t/m^3.

The estimated resource tonnage in each block is the sum of the tonnages within each of the eight mineralized domains.

Estimation of Grade

In any block that has non-zero estimates of the tonnage for any domain, the gold and silver grades for that domain's tonnage were calculated using ordinary kriging of the nearby assays that fall within that domain.

The ordinary kriging search neighborhood was a sphere with a search radius of 110 m (the range of the variogram along the vein). Because the domain wireframes serve as hard boundaries, the distance to nearby samples in the direction across the vein is controlled by the wireframes, and not by the 110 m radius of the spherical search. This makes the search neighborhood like a "pancake" that is parallel to the vein, with a circular radius of 110 m in the plane of the vein, and a thickness equal to the full width of the vein.

An octant search was used to limit the effects of sample clustering, with the octants configured like slices of a pie, looking perpendicular to the vein at the 110 m circular projection of the search neighborhood. Within each octant, only the closest four samples were retained for estimation.

For the purposes of the average covariance calculations needed by the ordinary kriging equations, the 10×10×2 m blocks were approximated by a 4×4×2 grid.

The estimation of grade directly used the capped drill hole assays; no compositing was performed. Once the ordinary kriging weights had been calculated, these weights were multiplied by the assay length, and then renormalized to sum to one. This ensures that the variable sample length is correctly accounted for in the grade estimation.

Within each 10×10×2 m block , the grade estimates for each of the eight mineralized domains were combined into a grade estimate for the mineralized tonnage within the entire block by calculating the tonnage-weighted average of the grade estimates for each of the mineralized domains that contributes tonnage to that block.

14.1.9 Checks of Block Model Estimates

Software

All of the resource estimates were performed in-house software developed by FSS Canada. The tonnage and grade estimates were checked using two different commercial software systems: Micromine® and MineSight®. Globally, the tonnage estimates from the three independent calculations were within ±1% of each other, and the grade estimates were within ±5% of each other for the blocks classified as measured and indicated (and within ±7% for the blocks classified as inferred). The differences between the systems are due to the different ways the search strategy is implemented, and to the use of composites versus the use of assays.

Tonnage Checks

Tonnage estimates were checked visually, domain by domain, using the 3D viewing capability of Micromine® and MineSight® to ensure that the individual domain tonnage estimates correctly respect the domain boundaries, and also to check that there were no gaps or overlaps between contiguous wireframes, i.e. between the Angela Vein and its branches and splays.

Figure 14.13 is a typical cross-section showing the estimated total tonnage from all domains along with the outlines of the domain wireframes. The tonnage estimates in this figure have been color-coded using a gray scale, with solid black blocks having 100% of their tonnage (502 t) within one or more of the veins and solid white blocks having no tonnage in any of the eight veins.

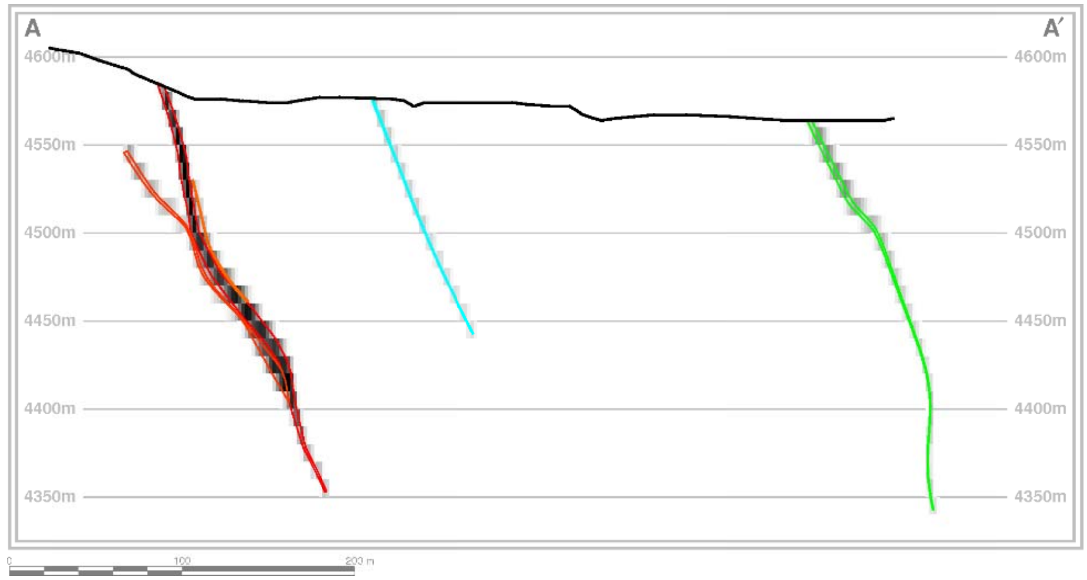


Figure 14.13 - Check of estimates of volume proportions

Grade Checks

Grade estimates were checked visually, domain by domain, using the 3D viewing capability of Micromine® and MineSight® to ensure that the individual gold and silver grade estimates are consistent with the nearby drill hole data.

Figure 14.14 is a typical cross-section showing the estimated silver-equivalent grade for each block along with the original drill hole data on the same section. This example confirms that the geological domains have been used as hard boundaries. The low-grade estimates in the Lourdes Vein (in the middle of the cross-section in Figure 14.14) are due to the fact that the only data available for estimation in this vein are the low-grade assays in the Lourdes Vein itself; the higher-grade assays in the Angela Vein to the north are within the 110m search radius but are excluded because they belong to a different domain.

Although the cross-section in Figure 14.14 may create an initial impression that the grades have been spread beyond their vein boundaries, it should be noted that the grade estimates apply only to the proportion of each block that falls within the wireframes. As shown on the previous cross-section, Figure 14.13, the volume proportions within each block correctly represent the fact that, in most of the blocks, the tonnage of vein within the block is less than 100%.

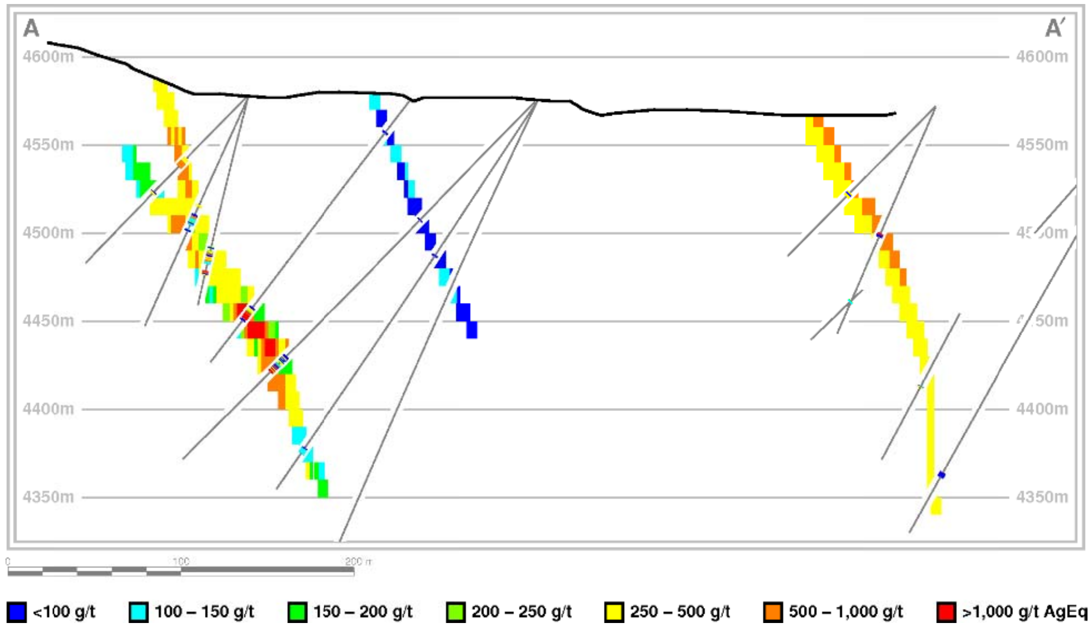


Figure 14.14 - Check of silver-equivalent grade estimates

Grades were also checked by doing “swath plots” that compare the average grade estimates to the length-weighted average of the original drill hole data along the columns, rows and levels of the block model. Figure 14.15 shows these swath plots comparisons for silver-equivalent. The block estimates show less total variability than the original assay data since they represent larger volumes of material; but the peaks and troughs in the drill hole data are picked up in the estimates.

There is a slight shift in the swath plot done along the rows of the model; this is due to the fact that the vein system is not vertical, but dips steeply to the southeast. Calculations of average assays done along the vertical rows of the model will be slightly offset from the same calculations done for the block estimates since deep assays with low row numbers will contribute to the estimate of blocks with higher row numbers.

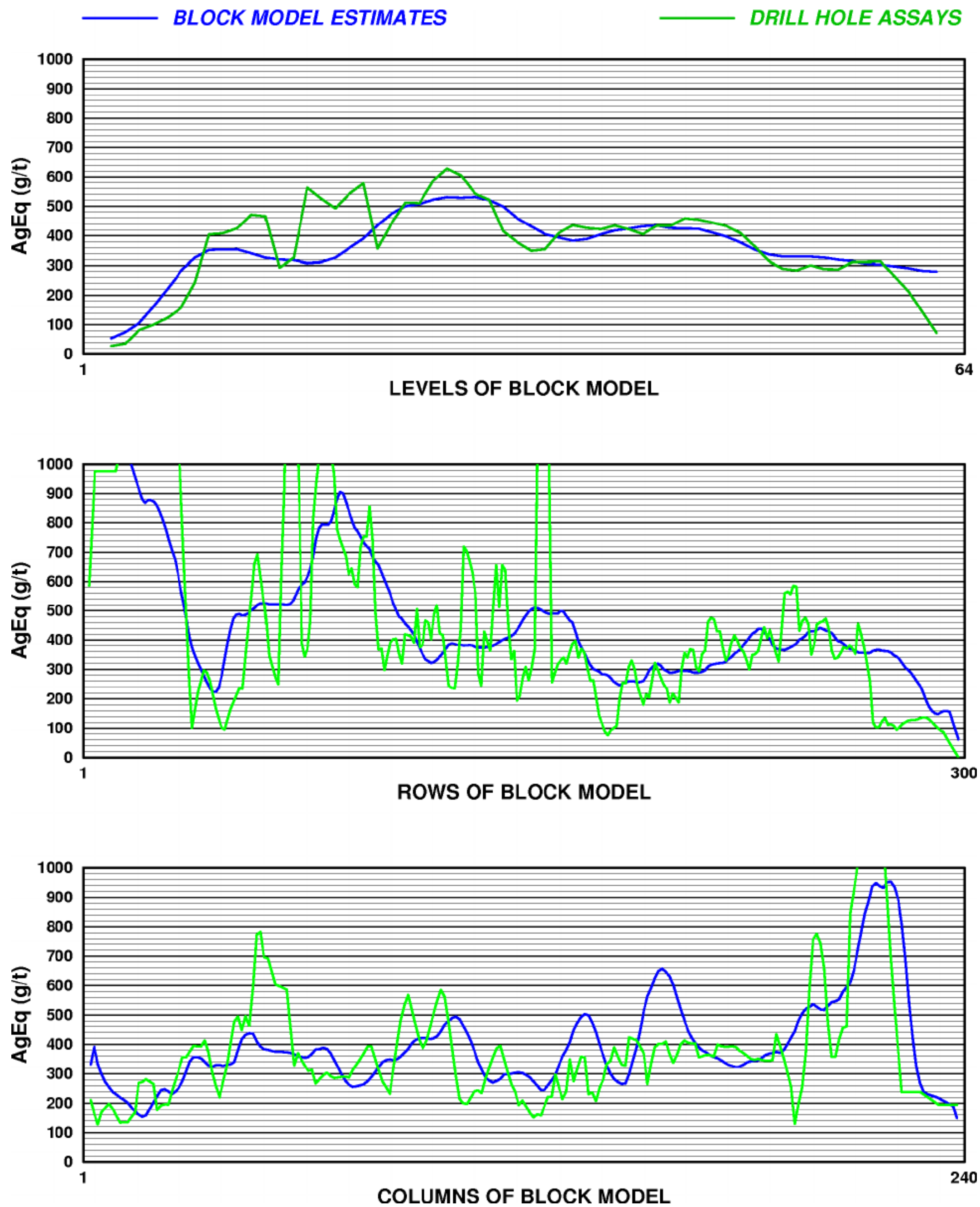


Figure 14.15 - Swath plots comparing block estimates to original drill hole data

14.1.10 Resource Classification

Mineral Resources were classified according to the CIM Definition Standards, which require that resource classification conform to the following terminology and definitions:

A Measured Mineral Resource is that part of a Mineral Resource for which quantity, grade or quality, densities, shape, and physical characteristics are so well established that they can be estimated with confidence sufficient to allow the appropriate application of technical and economic parameters, to support production planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration, sampling and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes that are spaced closely enough to confirm both geological and grade continuity.

An Indicated Mineral Resource is that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics can be estimated with a level of confidence sufficient to allow the appropriate application of technical and economic parameters, to support mine planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes that are spaced closely enough for geological and grade continuity to be reasonably assumed.

An Inferred Mineral Resource is that part of a Mineral Resource for which quantity and grade or quality can be estimated on the basis of geological evidence and limited sampling and reasonably assumed, but not verified, geological and grade continuity. The estimate is based on limited information and sampling gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes.

Classification compliant with these definitions was achieved using a two-step procedure. In the first step, each block was assigned an integer code, from one to three, using the following criteria:

1. Blocks within 25 m of a drill hole sample, and with samples closer than 110 m (the variogram range) in at least four octants, were assigned a value of 1.
2. Blocks that were not assigned a 1, but that were within 40m of a drill hole sample, and with samples closer than 110 m (the variogram range) in at least four octants, were assigned a value of 2.
3. Blocks that were not assigned a 1 or 2, but that were within 110 m (the range of the variogram) of a drill hole sample were assigned a value of 3.
4. Regardless of the previous classification, blocks in sparsely-drilled minor veins are assigned a classification code of 3. These include Angela R2, Angela R3, Angela R4, Lourdes and Martha. The Angela R1 branch is much better drilled, and outcrops to the southwest, which creates less uncertainty in the geometry of this vein. Similarly, with the Angela Splay also being better drilled, with almost twice the drilling of the other minor veins, the classification codes in this vein are based on proximity to nearby data.

The first criterion gives a value of 1 to blocks that are well-informed by nearby data, and that are well-surrounded by data. The second gives an intermediate value to blocks that have data well within the range of the variogram, but that are slightly less well-informed than the first group. The third criterion gives a value of 3 to all blocks that are correlated with at least one nearby sample. The fourth criterion ensures that the minor veins are given a classification code that reflects the higher degree of uncertainty in their grade and tonnage estimates.

Figure 14.16 shows these integer codes on a longitudinal section of the main Angela Vein, averaged across the width of the vein. In a broad sense, the 1s correspond to “measured”, the

2s to “indicated” and the 3s to “inferred”. As often occurs with numerical criteria evaluated on a block-by-block basis, however, there are locations where the classification changes abruptly, creating small islands of 1s surrounded by 3s, and vice versa.

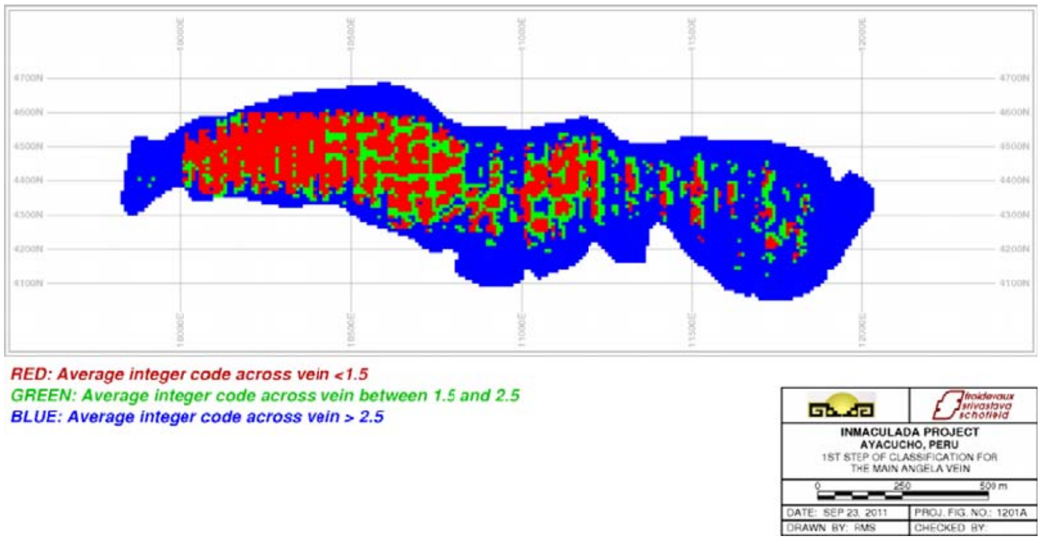


Figure 14.16 - Longitudinal section showing the first step of classification procedure for the Angela Vein

In order to provide a classification that pertains to the tonnages involved in mid-term and long-term mine planning, the initial codes shown in Figure 14.16 were spatially averaged to remove small one-block islands of one classification code that are surrounded by a different classification code. With this spatial smoothing, the classification pertains to volumes larger than a single 200 m^3 block; the final classification pertains to volumes that are approximately the same volume as quarterly production planning increments.

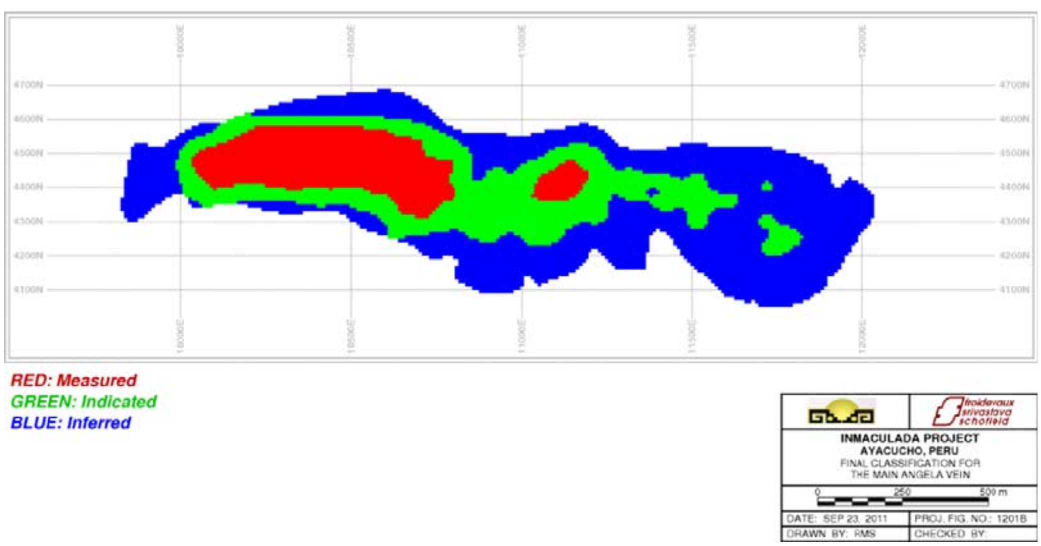


Figure 14.17 - Longitudinal section showing final classification for the Angela Vein

"Measured" blocks are those that have a value of 1.5 or less after the spatial smoothing step; "indicated" blocks are those that have a value of 2.5 or less; and "inferred" blocks are those that have a value of 2.5 or more. Figure 14.17 shows a longitudinal section of the final classification of the Angela Vein.

Most of the Measured Resources lie in the southwestern portions of the Angela Vein, where it outcrops and where it has been drilled on 50 m sections, with holes that pierce the vein at roughly 25 m spacing down dip. A small amount of the Measured Resource lies in the central part of the Angela Vein, where recent drilling has brought the drill hole spacing below the 50 x 50 m spacing that is typical outside the southwestern region. There are Indicated Resources in the central area, where there is drilling on 50 m sections, with holes that pierce the vein at roughly 50 m spacing down dip. In the remainder of the deposit, where the average drill hole spacing is greater than 50m, there are Inferred Resources.

14.1.11 Reporting Cutoff

Mineral resources for the Inmaculada project are reported at a cutoff at which the revenue produced by the gold and silver is sufficient to cover the costs of mining and processing the ore. Table 14.2 lists the economic and technical parameters required to calculate the revenue produced by a gram of in situ metal.

Table 14.2 - Parameters used for the calculation of the revenue produced by a gram of metal in situ

	Gold	Silver
Price	1,100 $/g	18 $/g
Recovery[1]	95.46%	90.37%
Offsite costs[2]	24.11 US$/kg	--

[1]Includes metallurgical and commercial losses.

[2]Includes transportation, treatment, refining, umpire analyses, supervision of shipments

Using the parameters in Table 14.2, at an average diluted head grade of 3.15 g/t Au and 110.77 g/t Ag a gram of in situ gold produces $ 32.79 of revenue and a gram of silver produces $ 0.5188 of revenue.

The total revenue of a block, in $/t, is:

$$\text{Total Revenue} = 32.79 * \text{Au} + 0.5188 * \text{Ag}$$

The reporting cutoff is a Total Revenue of $48.30 /t, the sum of the mining, processing and G&A costs. This is equivalent to a cutoff of 93.1 g/t AgEq or 1.47 g/t AuEq, with silver-equivalent and gold-equivalent defined as follows:

$$\text{AgEq} = \text{Ag} + 63.2 * \text{Au} \quad \text{and} \quad \text{AuEq} = \text{Au} + \text{Ag} \div 63.2$$

14.1.12 Summary of Classified Mineral Resources

Tables 14.3 through 14.10 summarize the estimated mineral resources in each of the eight domains, using a cutoff of $48.30 /t on the estimated revenue of each block. This is equivalent to a 93.1 g/t AgEq cutoff, and also to a 1.47 g/t AuEq cutoff. In these tables, numbers have been rounded to reflect the precision of the estimates. The effective date of these mineral resource estimates is 11 January, 2012.

These summaries are based on the entire block grade, and not on the individual contribution of each vein to each block. For blocks with contributions from two or more veins, a dominant vein code was assigned to the vein that contributed the greatest tonnage.

Table 14.3 - Estimated Mineral Resources for the Angela Vein, at a revenue cutoff of $ 48.30 /t, as of 11 January 2012

		GRADE				IN SITU METAL CONTENT			
CLASSIFICATION	**TONNAGE (Mt)**	Au (g/t)	Ag (g/t)	AuEq (g/t)	AgEq (g/t)	Au (Moz)	Ag (Moz)	AuEq (Moz)	AgEq (Moz)
Measured	3.18	4.17	129	6.02	421	0.43	13.20	0.61	42.99
Indicated	3.55	4.11	161	6.40	448	0.47	18.36	0.73	51.17
Measured+Indicated	**6.73**	**4.14**	**146**	**6.22**	**435**	**0.89**	**31.57**	**1.35**	**94.16**
Inferred	3.17	4.49	164	6.84	479	0.46	16.75	0.70	48.84

Table 14.4 - Estimated Mineral Resources for the Angela Splay, at a revenue cutoff of $ 48.30 /t, as of 11 January 2012

		GRADE				IN SITU METAL CONTENT			
CLASSIFICATION	**TONNAGE** (Mt)	Au (g/t)	Ag (g/t)	AuEq (g/t)	AgEq (g/t)	Au (Moz)	Ag (Moz)	AuEq (Moz)	AgEq (Moz)
Measured	0.00	–	–	–	–	–	–	–	–
Indicated	0.07	4.26	216	7.35	514	0.01	0.48	0.02	1.14
Measured+Indicated	**0.07**	**4.26**	**216**	**7.35**	**514**	**0.01**	**0.48**	**0.02**	**1.14**
Inferred	0.54	4.45	207	7.41	519	0.08	3.58	0.13	8.95

Table 14.5 Estimated Mineral Resources for the Angela R1 Branch, at a revenue cutoff of $ 48.30 /t, as of 11 January 2012

		GRADE				IN SITU METAL CONTENT			
CLASSIFICATION	**TONNAGE** (Mt)	Au (g/t)	Ag (g/t)	AuEq (g/t)	AgEq (g/t)	Au (Moz)	Ag (Moz)	AuEq (Moz)	AgEq (Moz)
Measured	0.11	2.09	76	3.18	222	0.01	0.27	0.01	0.78
Indicated	0.16	2.70	91	3.99	279	0.01	0.48	0.02	1.47
Measured+Indicated	**0.27**	**2.45**	**85**	**3.67**	**257**	**0.02**	**0.74**	**0.03**	**2.24**
Inferred	0.02	1.90	82	3.07	215	0.00	0.05	0.00	0.12

Table 14.6 - Estimated Mineral Resources for the Angela R2 Branch, at a revenue cutoff of $ 48.30 /t, as of 11 January 2012

		GRADE				IN SITU METAL CONTENT			
CLASSIFICATION	TONNAGE (Mt)	Au (g/t)	Ag (g/t)	AuEq (g/t)	AgEq (g/t)	Au (Moz)	Ag (Moz)	AuEq (Moz)	AgEq (Moz)
Inferred	0.05	2.04	83	3.23	226	0.00	0.12	0.00	0.33

Table 14.7 - Estimated Mineral Resources for the Angela R3 Branch, at a revenue cutoff of $ 48.30 /t, as of 11 January 2012

		GRADE				IN SITU METAL CONTENT			
CLASSIFICATION	TONNAGE (Mt)	Au (g/t)	Ag (g/t)	AuEq (g/t)	AgEq (g/t)	Au (Moz)	Ag (Moz)	AuEq (Moz)	AgEq (Moz)
Inferred	0.15	1.01	120	2.73	191	0.00	0.59	0.01	0.93

Table 14.8 - Estimated Mineral Resources for the Angela R4 Branch, at a revenue cutoff of $ 48.30 /t, as of 11 January 2012

		GRADE				IN SITU METAL CONTENT			
CLASSIFICATION	TONNAGE (Mt)	Au (g/t)	Ag (g/t)	AuEq (g/t)	AgEq (g/t)	Au (Moz)	Ag (Moz)	AuEq (Moz)	AgEq (Moz)
Inferred	0.51	2.01	92	3.32	232	0.03	1.49	0.05	3.79

Table 14.9 Estimated Mineral Resources for the Martha Vein, at a revenue cutoff of $ 48.30 /t, as of 11 January 2012

		GRADE				IN SITU METAL CONTENT			
CLASSIFICATION	TONNAGE (Mt)	Au (g/t)	Ag (g/t)	AuEq (g/t)	AgEq (g/t)	Au (Moz)	Ag (Moz)	AuEq (Moz)	AgEq (Moz)
Inferred	0.41	3.14	99	4.55	319	0.04	1.30	0.06	4.21

Table 14.10 - Estimated Mineral Resources for the Lourdes Vein, at a revenue cutoff of $ 48.30 /t, as of 11 January 2012

		GRADE				IN SITU METAL CONTENT			
CLASSIFICATION	TONNAGE (Mt)	Au (g/t)	Ag (g/t)	AuEq (g/t)	AgEq (g/t)	Au (Moz)	Ag (Moz)	AuEq (Moz)	AgEq (Moz)
Inferred	0.10	0.50	93	1.82	128	0.00	0.28	0.01	0.39

Table 14.11 below shows the total mineral resource inventory for the entire project, i.e. the sum of the resources estimates across all eight domains. As with the previous tables, numbers have been rounded to reflect the precision of the estimates, and the effective date of these estimates is 11 January 2012.

Table 14.11 - Estimated Mineral Resources for the Inmaculada Project, at a revenue cutoff of $ 48.30 /t, as of 11 January 2012

		GRADE				IN SITU METAL CONTENT			
CLASSIFICATION	TONNAGE (Mt)	Au (g/t)	Ag (g/t)	AuEq (g/t)	AgEq (g/t)	Au (Moz)	Ag (Moz)	AuEq (Moz)	AgEq (Moz)
Measured	3.28	4.10	128	5.92	415	0.43	13.47	0.63	43.76
Indicated	3.78	4.05	159	6.32	442	0.49	19.32	0.77	53.78
Measured+Indicated	**7.07**	**4.07**	**144**	**6.13**	**429**	**0.93**	**32.79**	**1.39**	**97.54**
Inferred	4.94	3.91	152	6.08	426	0.62	24.17	0.97	67.57

14.1.13 Grade-tonnage Curves

Figures 14.18 through 14.20 show how the tonnage, gold grade and silver grade of the project's total resources are affected by changes in the reporting cutoff. With the main veins having sharp walls, and the grade climbing quickly from well below cutoff to well above cutoff within ±1 m of the edges of the veins, the resource tonnages and grades are relatively insensitive to the reporting cutoff. Regardless of the cutoff, the resource inventory essentially covers almost all of the material within the vein wireframes.

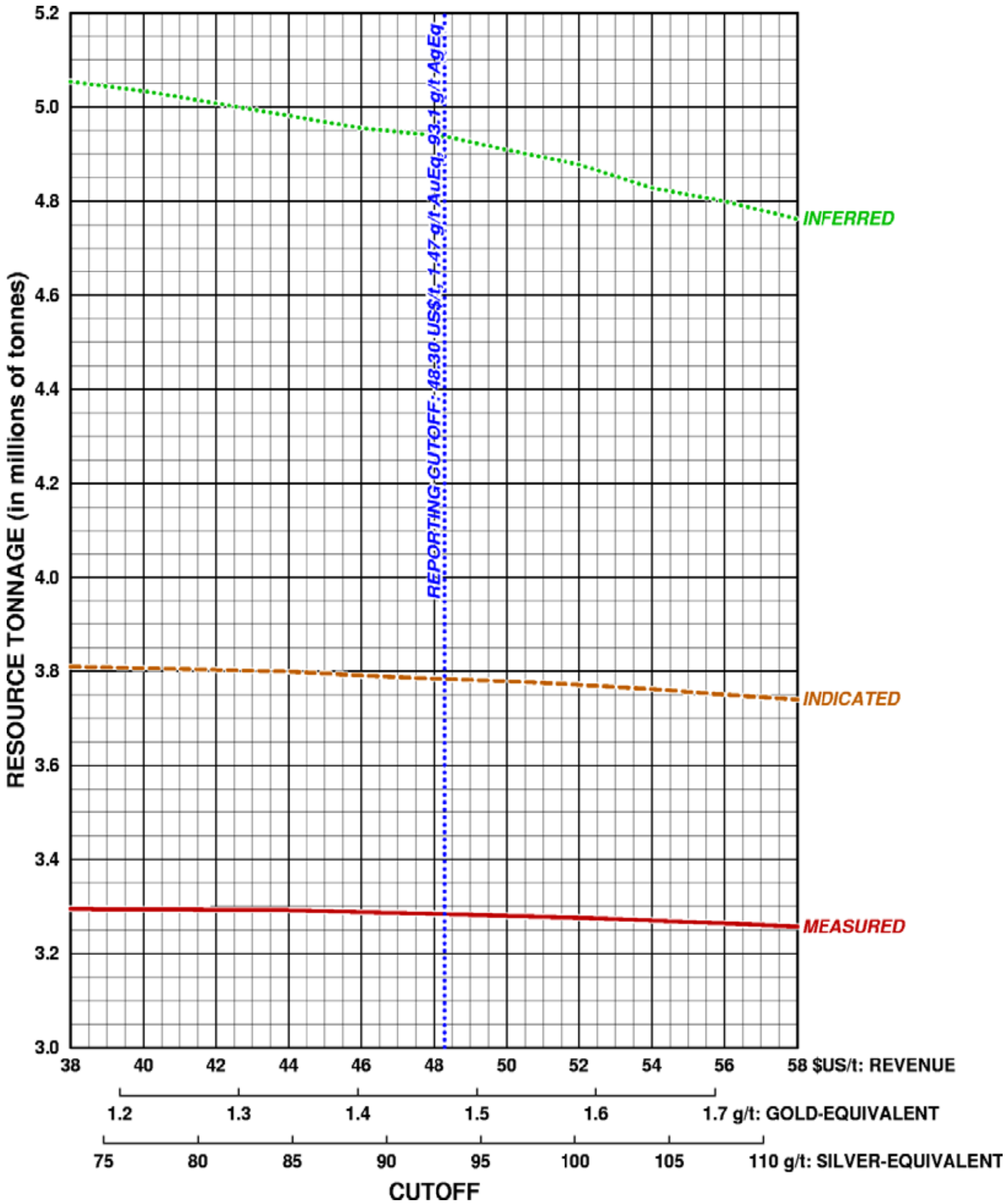


Figure 14.18 - Resource tonnage versus reporting cutoff

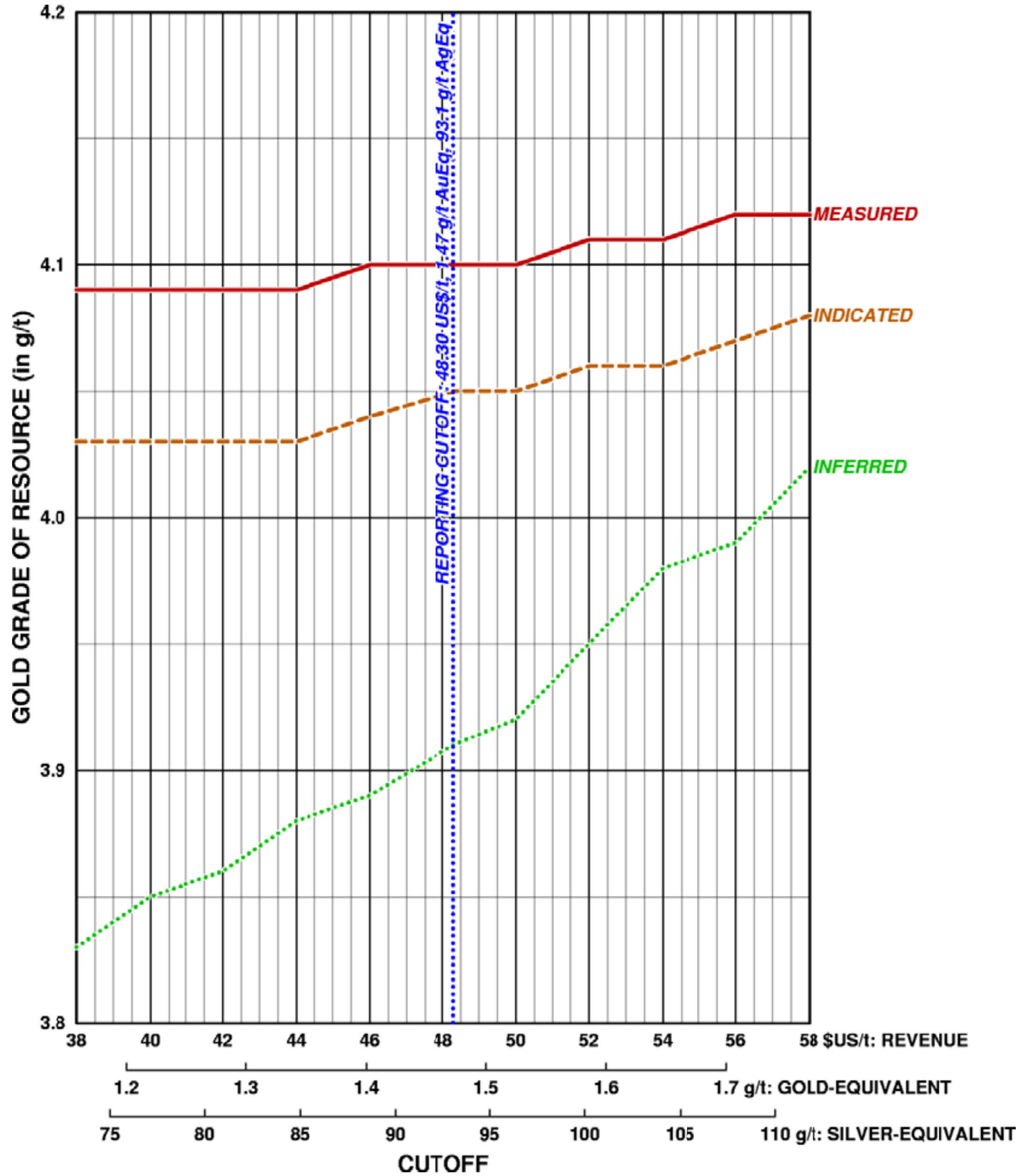


Figure 14.19 - Resource gold grade versus reporting cutoff

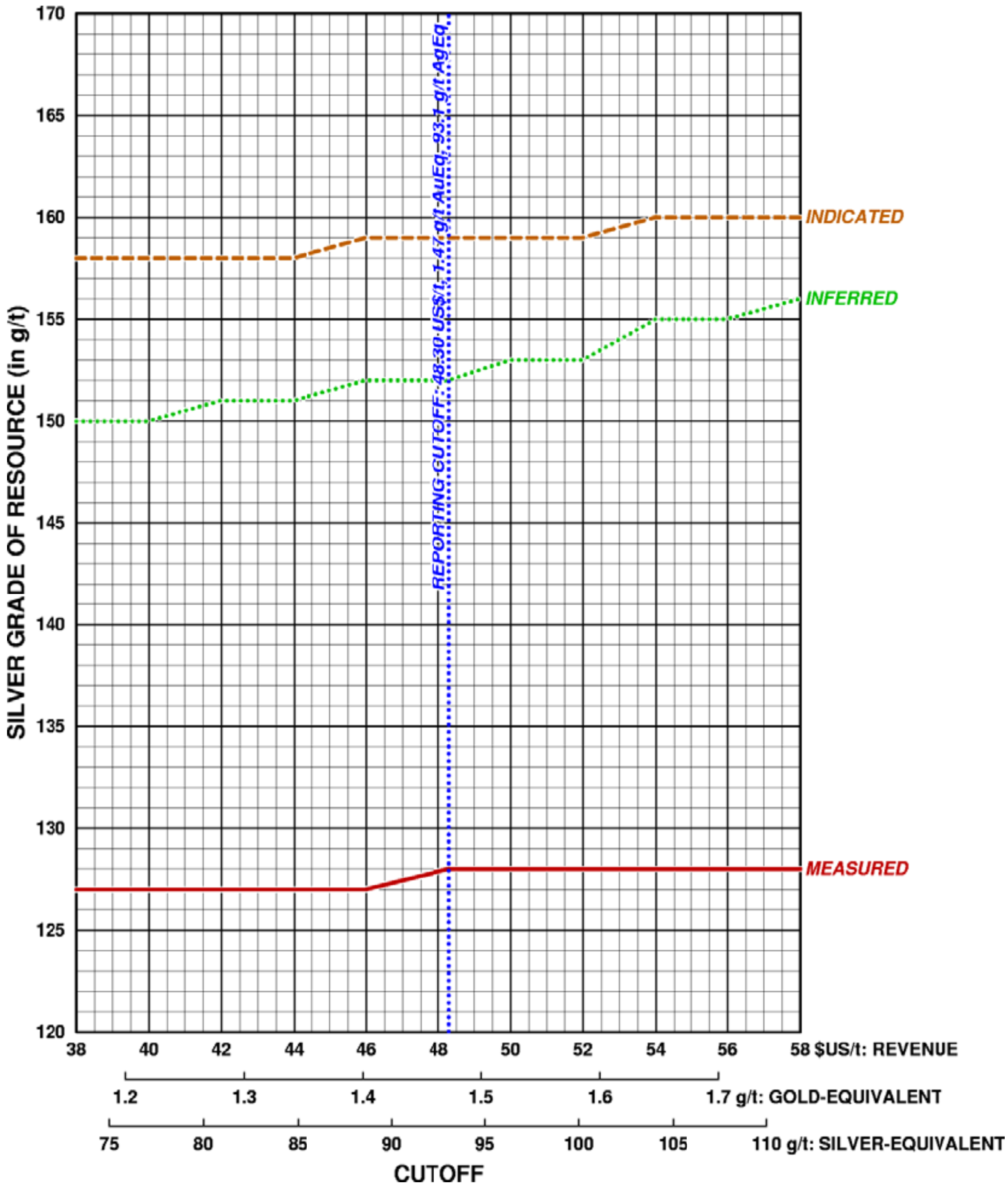


Figure 14.20 - Resource silver grade versus reporting cutoff

With the gold and silver grade increasing slightly with cutoff, and the tonnage decreasing slightly with cutoff, the in situ metal content is insensitive to the cutoff.

14.1.14 Estimate of Grade of Dilution

For the calculation of diluted resources and reserves, it is necessary to have an estimate of the grade of the waste rock immediately adjacent to the veins.

This estimate is based on all of the assays less than 1m outside of the three veins that contain measured or indicated resources: Angela, Angela R1 and Angela Splay. These assays, 506 in all, have a declustered and length-weighted average grade of 0.30 g/t Au and 11.2 g/t Ag. Expressed as metal-equivalents, these averages are 0.48 g/t AuEq and 30.2 g/t AgEq.

A check was done for the possibility of a systematic difference between the average grade on the footwall versus the hangingwall, but no significant difference could be found between the two.

A check was also done for the possibility of systematic differences between the veins. The thickest of the three veins, the main Angela Vein, has slightly higher grades in its dilution fringe; but, as with the previous check, the differences were not statistically significant.

15 Mineral Reserve Estimate

Of the eight recognized mineralized structures included in the resource estimate, measured and indicated resources were assigned to only three veing. Of these three veins, 95% of the total measured and indicated resources are found in the Angela Vein, which presents sufficient continuity for exploitation. Considering this, the mineral reserves estimate relates exclusively to the Angela Vein.

The mineral reserves were estimated considering two underground mining methods: cut and fill and sub level stoping both using cemented paste backfill and including the relative effects of dilution and ore losses for each method.

Measured and indicated mineral resources were used as inputs for mine design and conversion to proven and probable reserves, while the material corresponding to inferred resources was considered waste.

The Inmaculada project mineral reserve estimate is summarized in Table 15.1.

Table 15.1- Inmaculada Project - Mineral Reserves

Category	Mt	Au		Ag		AuEq	
		g/t	Moz Content	g/t	Moz Content	g/t	Moz Content
Proven	3.84	3.40	0.42	106	13.13	5.18	0.64
Probable	3.96	3.33	0.42	134	17.01	5.56	0.71
Total	**7.80**	**3.37**	**0.84**	**120**	**30.14**	**5.37**	**1.35**

The reason for the mineral reserve tonnage being higher than the resource tonnage is due to the dilution expected during the mining of the vein, and not due to inclusion of any inferred resources.

15.1 Mineral Reserve Estimate Calculation Procedure

The mineral reserve estimation only considered the Angela Vein (not the minor splays and branches) and was based on the resource block model, mining method selected, geomechanical recommendations and potential stope sizes which assisted in the definition of the Mining Units (MU) to which dilution and ore loss was applied.

The MU's were defined by areas of 90 m along the vein by 25 m in height suitable to accommodate stopes for both mining methods (sub level stoping and cut and fill). An initial assessment of the economic contribution of each mining unit was undertaken based on its gold and silver revenue compared to the costs of mining, processing, and metal sale costs.

The next stage included designing infrastructure and access to these economically minable units depending on the mining method selected for each MU.

15.2 Dilution

After evaluating the block model within the defined mining limits, blocks with a grade below the cut-off represented less than 2% of the total tonnage within the designed stopes. For the mineral reserves determination these blocks were incorporated into the MU's and represents an internal dilution.

Two mining dilution estimates were defined for the mining methods selected:

- In the areas mined by cut and fill, 25% of the average vein width was added based on the footwall (FW) and hangingwalls (HW) adjacent to the vein being of poor rock quality as determined by the geomechanical studies.
- In the areas mined by sub level stoping, 30% of the average vein width was added for the same reasons, even though the height of the stope was limited to only 5 m.

These values have been discussed and agreed with Suyamarca personnel based on previous experience from existing operating mines.

The grades of the dilution described above were estimated as included under the Resource Section 14.1.14.

15.3 Ore Loss

The ore loss expected is due to ore body geometry and operational aspects:

- Practical mine design will not exactly follow the vein geometry and part of the mineralized structure will not be fully extracted, remaining "in situ". Comparing the designs developed with the interpreted mineralized structure an ore loss of 1% has been estimated.
- Operational factors; due to drilling, blasting and stope extraction activities will result in some inevitable loss of ore. The impact of these factors is estimated at 3% of the in situ stope quantities.

15.4 Mineral Reserves

The main geometries and dilution width for each mining method are summarized in the following table:

Table 15.2 - Geometrical and Dilution Parameters

<table>
<tr><th>Type</th><th>Width</th><th>Length</th><th>Height</th><th>Footwall Dilution</th><th>Hangingwall Dilution</th></tr>
<tr><td colspan="6">Cut and Fill</td></tr>
<tr><td rowspan="2">Base Heading</td><td>Max: 6 m</td><td>45 m each side of access</td><td rowspan="2">4 m</td><td rowspan="2">+ 0.80 m</td><td rowspan="2">+ 0.80 m</td></tr>
<tr><td>Min: 4 m</td><td>90 m Mining Unit</td></tr>
<tr><td rowspan="2">Stope</td><td>Max: 6 m</td><td>45 m each side of access</td><td rowspan="2">4.5 m</td><td rowspan="2">+ 0.80 m</td><td rowspan="2">+ 0.80 m</td></tr>
<tr><td>Min: 4 m</td><td>90 m Mining Unit</td></tr>
<tr><td colspan="6">Sub Level Stoping</td></tr>
<tr><td rowspan="2">Strike Heading</td><td>Max: vein width</td><td>15 m each side of access</td><td rowspan="2">4 m</td><td rowspan="2">+ 1.15 m</td><td rowspan="2">+ 1.15 m</td></tr>
<tr><td>Minimum: 4 m</td><td>90 m Mining Unit</td></tr>
<tr><td rowspan="2">Stope</td><td>Maximum: vein width</td><td>15 m each side of access</td><td rowspan="2">5 m</td><td rowspan="2">+ 1.15 m</td><td rowspan="2">+ 1.15 m</td></tr>
<tr><td>Minimum: 4 m</td><td>90 m Mining Unit</td></tr>
</table>

Table 15.3 summarizes the economic and metallurgical parameters used in the Mining Unit assessment.

Table 15.3 - Economic and Metallurgical Parameters

Metal Prices	Value	Units
Gold	1,100	US$/oz
Silver	18	US$/oz
Costs		
Cut and Fill Mining	44.47	US$/t
Sub Level Stope Mining	32.15	US$/t
Processing	25.30	US$/t
General and Administration	9.99	US$/t
Metallurgical Recoveries		
Gold	95.6	%
Silver	90.6	%
Payable Percentages		
Gold	99.85	%
Silver	99.75	%

From above the Inmaculada project mineral reserves totals 7.8 Mt of ore with grades of 3.37 g/t Au and 120.2 g/t Ag, containing 0.84 Moz Au and 30.14 Moz Ag as shown in Table 15.1.

16 Mining Methods

16.1 Mine Design Basis

The main design elements considered were:

(a) ore and host rock geomechanical characteristics;

(b) dip and thickness of the Angela Vein;

(c) production and productivity requirements of each mining method; and

(d) estimated ore dilution and losses.

Practical input from the Suyamarca Technical and Operational teams was taken into consideration as well as results from initial geomechanical 2D and 3D modeling. The recommendations from Suyamarca were based on operating experience mining similar narrow vein deposits. The author and QP of this Section agreed with these recommendations and operating policies provided by Suyamarca, and subsequently included in the mine design and operational parameters. Figure 16.1 shows a longitudinal section showing the areas of each mining method.

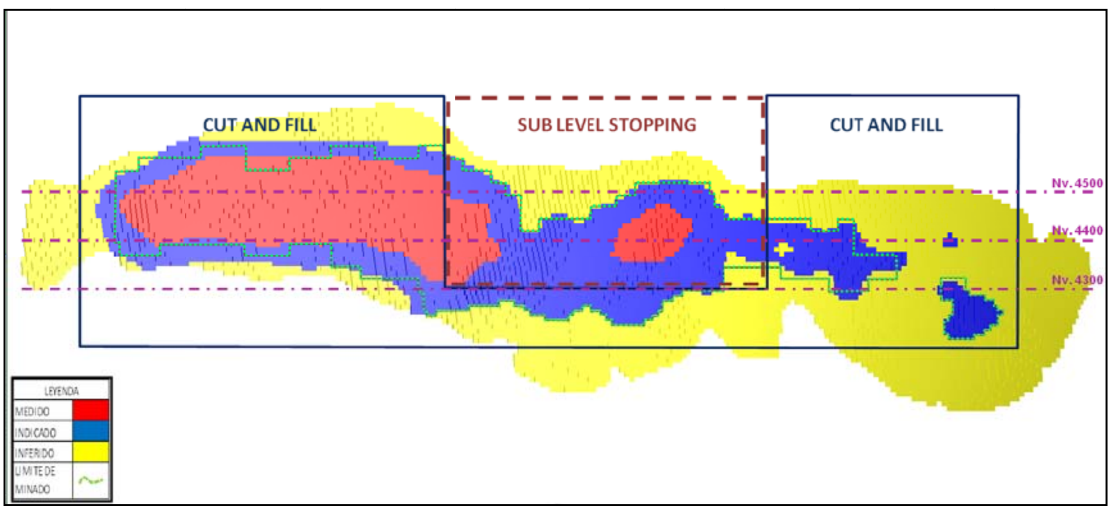


Figure 16.1 - Mining methods applicable to the Angela Vein

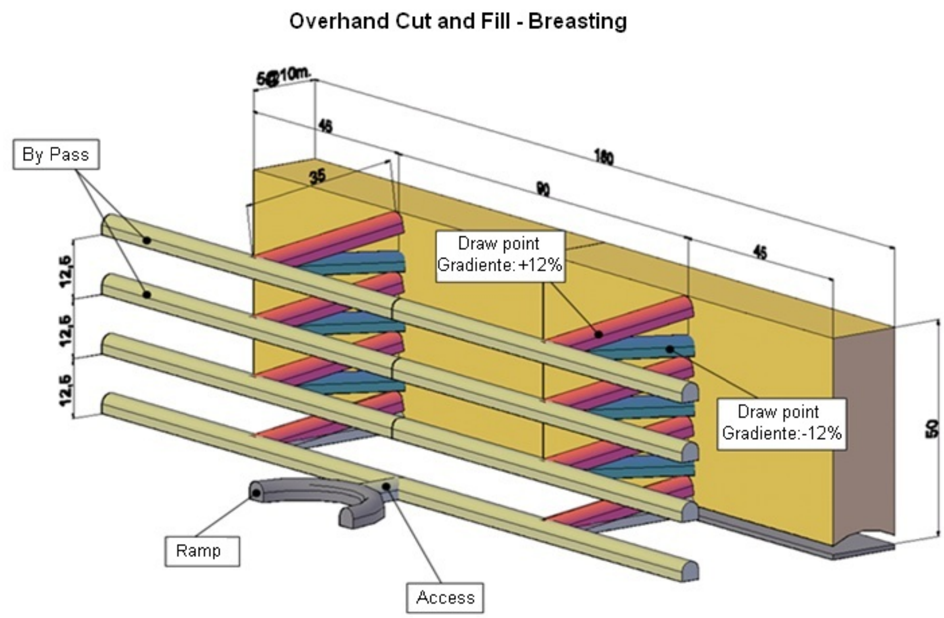
16.2 Mine Design Description

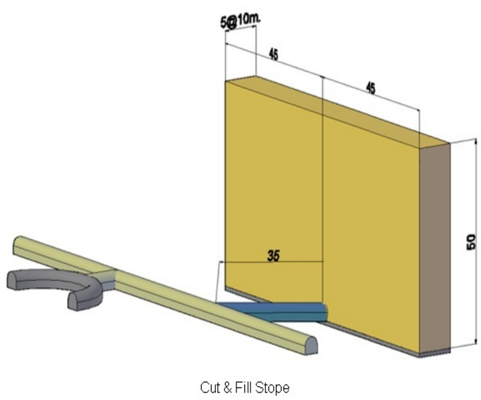
The underground infrastructure will be located in the FW of the Angela Vein, where the better geomechanical rock conditions are found compared to the HW, providing safer construction conditions**.**

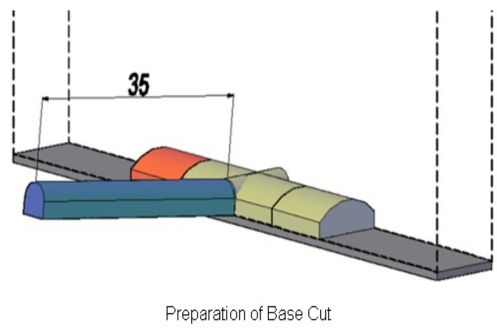
Permanent works such as main ramps, main production levels and operating ramps, crosscuts and production levels off vein (also called by-passes) will be located at a minimum distance of 35 m from the vein. Likewise the ore and waste passes will be developed adjacent to the bypasses and located in the center of every operating area delimited by the internal access ramps.

Based on geomechanical rock conditions, orebody geometry, vein width and dip angle; two highly mechanized mining methods were selected, namely cut and fill and sub level stoping. The sub level stoping was selected for the better rock conditions, wider veins and near vertical dip angles with sub-levels developed at 9 m vertical intervals to limit HW exposure, drill hole deviation and dilution. The cut and fill method was selected generally on weaker ground, narrower veins and lower dip angles; accesses to the vein will be developed at 12.5 m vertical intervals. Both methods will use cemented backfill. The 2 km long vein has good continuity and sufficient height to allow a number of simultaneous mining sections to ensure an ore production of 3,506 t/d. A robust mine infrastructure has been designed to handle development and production material, ventilation, backfill, ore transfers and water handling.

The following Figures 16.2 through 16.5 depicts the main geometrical characteristics of the mining methods selected.

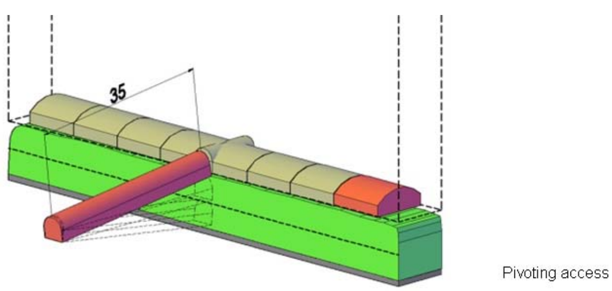


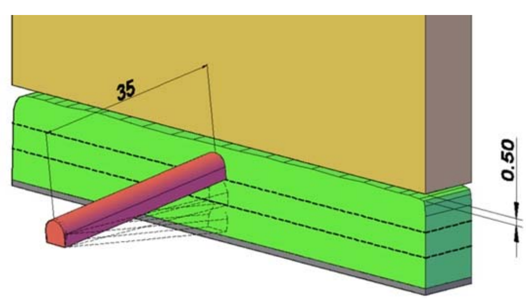




Cut & Fill Stope

Preparation of Base Cut





Pivoting access

Figure 16.2 - Main Infrastructure for Cut and Fill and Mining Details

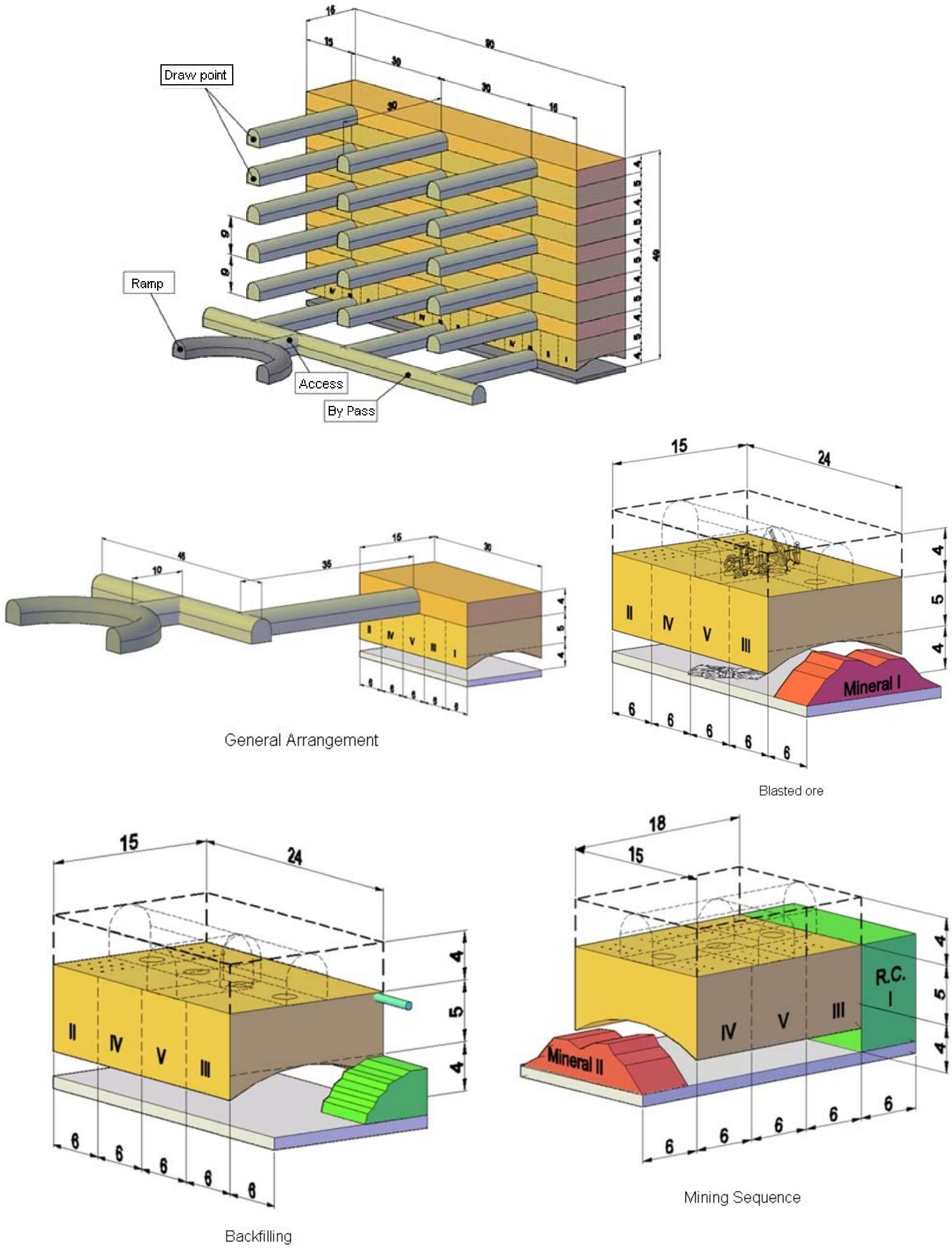


Figure 16.3 - Main Infrastructure for Sub Level Stoping and Mining Details

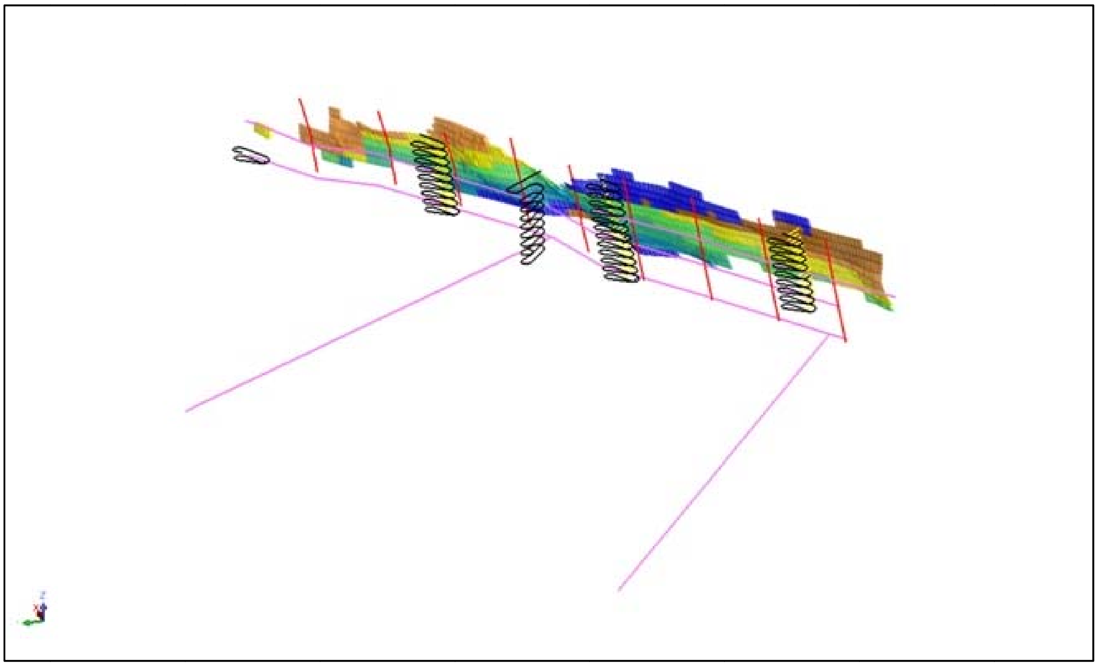


Figure 16.4 - Stopes and main infrastructure view (looking southeast)

A general arrangement showing accesses necessary for the mining of the Angela Vein is shown in Figure 16.5.

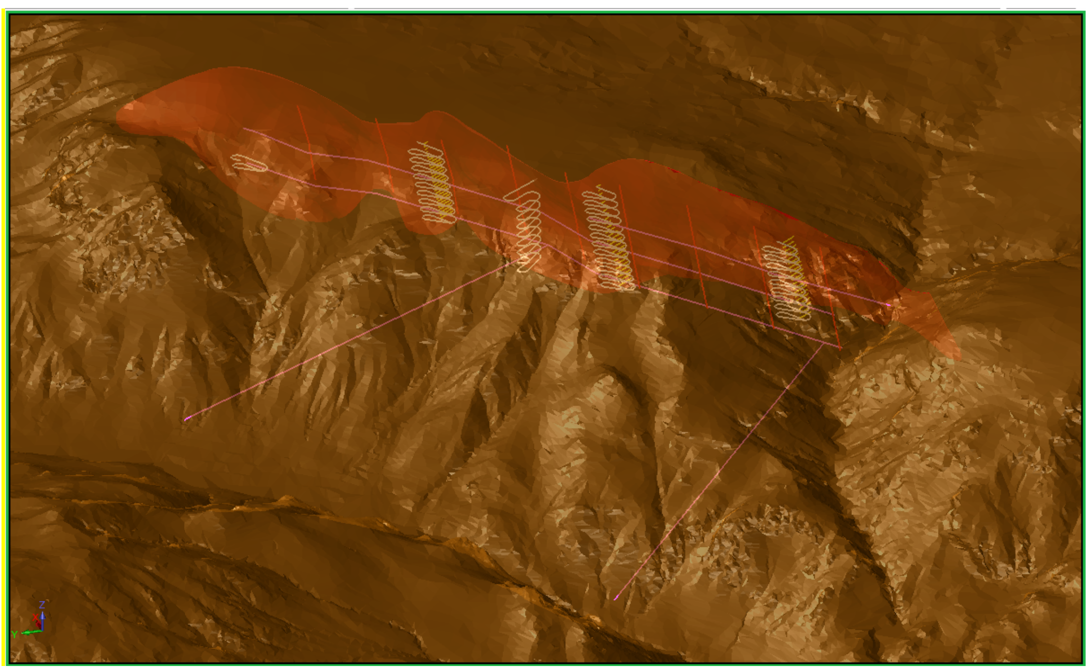


Figure 16.5 - Main access ramps, levels and internal ramps with a schematic view of the Angela Vein (looking southeast)

16.3 Mine Operation Criteria

The following criteria were considered for the estimation of capital and operating costs for mine development and operation:

In accordance with Suyamarca's operating policy, mine development work and ground support of main ramps, working levels (on and off vein), operating ramps and stope access shall be conducted by third party mining contractor, which shall employ its own resources.

Likewise the ore and waste transport from underground shall be conducted by a contracting company specialized in internal mine transport.

Production mining and support work of the Angela Vein shall be undertaken using Suyamarca's own resources (personnel and equipment).

16.4 Mine Operations

Development and production drilling will use electro hydraulic jumbos fitted with long hole kits (designed by the equipment manufacturer for this kind of equipment) and telescopic booms.

Development and production explosive loading will be done manually using retractable scaffolds where needed as the presence of ground water precludes the use of a mechanized ANFO loader. Vertical production drill holes will also be loaded manually.

The ground support work will use jumbo bolters which will install split sets, mechanical, helicoidal bars, Swellex or Hidrabolts bolts. Manual methods will be used for wet-mix shotcreting, complemented by a concrete mixer, and formwork construction.

The slot or free face opening for production in the sub level stoping method will be done with raise borers which can drill vertical raises up to 2.1 m in diameter.

Loading and tramming of blasted material will be performed by diesel scoop trams, covering distances no greater than 150 m, based on ore passes design and location. For greater haulage distances low profile mining trucks will be used. The scoop trams used in sub level stoping production will have remote control operation.

Ore produced at levels 4,400 and 4,500 will be trammed to the ore passes that will transport it to the 4,300 level, from where it will be transported using dump trucks with a capacity of 20 m3 (between 27 and 30 t) by way of the 4,400 level ramp to the mine entrance.

Paste backfill will be used employing a surface plant and an internal distribution system. The proportion of cement used in the fill for the sub level stopes is 5% and 2% in cut and fill stopes.

16.5 Mine Development

The following types of development were designed:

- Main ramp 5 m x 5 m;
- Main levels 4 m x 4 m;
- Sub levels or mining headings 4.5 m x 4 m in cut and fill and 4 m x 4 m in sub level stoping areas;
- Internal connection ramps 4 m x 4 m;
- Ventilation raises with raise boring at a diameter of 1.8 m;

- Service raises with raise boring at a diameter of 1.5 m; and
- Bypasses, crossroads, drawpoints and accesses at 4 m x 4 m sections.

16.5.1 Development Support

MICSAC developed the following mine support criteria for the FS:

- Systematic roofboltings and meshing for all development and preparation work;
- Applying a 2-inch thickness of shotcrete in up to 50% of mine areas including the main ramps, internal ramps, levels and bypasses;
- A ratio of 4 metal straps every 40 m for areas under development, and 6 metal straps every 40 m in the drawpoints / accesses from the bypass to the ore deposit; and
- A ratio of 5.1 bolts per meter for 5 m x 5 m ramps was assumed whereas a ratio of 3.9 bolts per meter was assumed for 4 m x 4 m ramps or levels. A 1.5 m x 1.5 m systematic meshing was assumed in both cases.

16.6 Main Equipment

The equipment selected for the underground works are listed below:

- Slot drilling – Raise borers mounted on crawlers will be used on the free face openings for stope blasting in sub level stoping areas;
- Production drilling will be done using electro-hydraulic jumbos with a telescopic boom for horizontal headings and for cut and fill areas. The jumbos are equipped with a kit for 5.5 m long holes and 2.5 inch diameter with an estimated effective drill rate of 25 m/h for sub level stoping;
- Pre-splitting will be done on sub level stoping using electro-hydraulic jumbo drills with a kit of long holes at a diameter of 2.0 inch to reduce the impact to sidewalls;
- Mine headings and sub level support: installation of roofbolts and mesh using automated Jumbo units. The application of shotcrete will be done using 6 m^3/hr manual equipment; and
- Tramming on production levels will use 4.6 m^3 and 3.0 m^3 diesel scoop trams and 20 t haul trucks.
- Main Transport will be completed using haul trucks with a capacity of 20 m^3

The number of equipment units required was estimated in accordance with the mining cycle, equipment performance and estimated effective availability and utilization rates.

16.7 Mine Production Support Work

Split sets and mesh were used in 100% of production support work, with the following specifications:

- 7 foot helicoidal bolts injected with cement grouting for permanent work;
- 7 foot split set friction bolts for non-permanent / occasional work;
- Electro-welded wire mesh No. 8 in 100% of area in preparation workings; and
- 100% shotcrete without steel fibers for sub levels, drawpoints and access ramps.

The placement of metal straps using a ratio of 6 straps per intersection was used in areas in which drawpoints, access ramps and mineralized structures intersect.

No active support system will be applied in the sub level stope void generated during the ore extraction process.

16.8 Production Schedule

A two year pre-operational period has been considered during which three mine access ramps, the main levels and internal ramps to support the first year production will be developed.

The production schedule has been developed to access the higher grade areas first to optimize the project´s net present value in conjunction with the productivity offered by the mining methods selected.

The mine schedule shown in Table 16.1 has a steady ramp-up in production during the first year of operation reaching 3,506 t/d at the start of the second year of operation.

Table 16.1 - Production Ramp-up during the first year of operation

Month	Ore Production (t/d)
1	250
2	500
5	1,000
7	2,000
10	3,240
13	3,506

The overall project schedule anticipates the completion of the process plant and necessary infrastructure construction in Q4 2013, and assumes the commencement of mine production operations in Q1 2014.

Prior to the start of operations the construction of approximately 3.5 km of main access ramps is required in addition to internal ramps and the preparation of mine workings.

The annual production program is shown in Table 16.2 with a graphical sequence shown in Figure 16.6.

Table 16.2 - Annual Production Plan

Year	Tonnage (Mt)	Grade Au (g/t)	Grade Ag (g/t)	Grade AuEq (g/t)	Au (x 1000 ozAu)	Ag (x1000 ozAg)	AuEq (x1000 ozAgEq)
2014	0.62	3.81	98.0	5.44	76	1,945	108
2015	1.26	3.41	121.0	5.43	138	4,905	220
2016	1.26	3.01	147.2	5.46	122	5,969	222
2017	1.26	2.65	130.9	4.83	107	5,296	195
2018	1.26	3.03	109.4	4.85	123	4,426	196
2019	1.25	3.64	96.8	5.25	147	3,904	212

Year	Tonnage (Mt)	Grade Au (g/t)	Grade Ag (g/t)	Grade AuEq (g/t)	Au (x 1000 ozAu)	Ag (x1000 ozAg)	AuEq (x1000 ozAgEq)
2020	0.89	4.60	128.9	6.75	132	3,696	194
Total	7.80	3.37	120.2	5.37	845	30,141	1,347

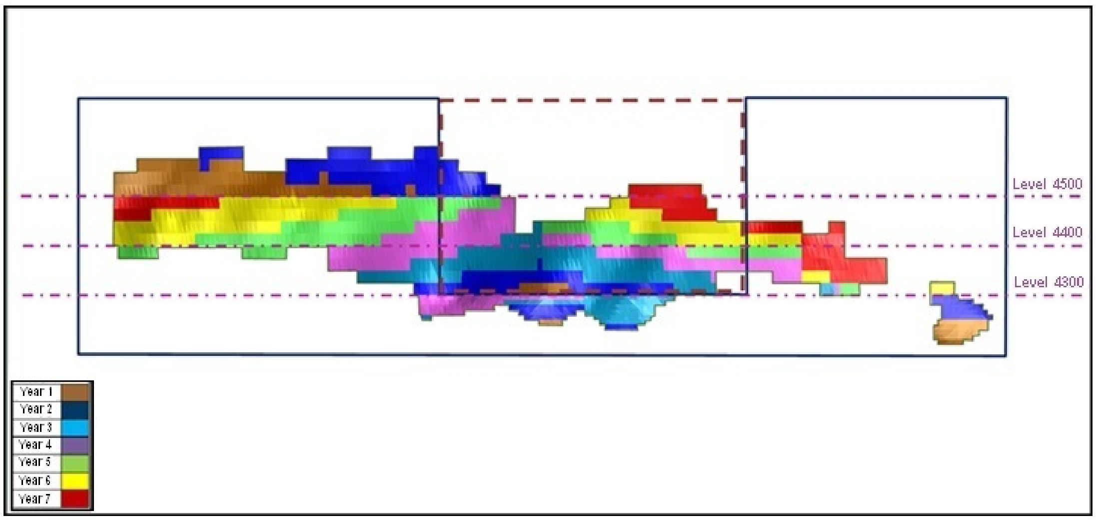


Figure 16.6 - Annual Mining Graphic Sequence looking north-westerly

16.9 Ancillary Services

The design and/or size of the following facilities were established during the FS:

- Ventilation System;
- Dewaterning system, taking into account available information indicating the need for the pumping of approximately 100 l/s during the first two years of project operation;
- Power supply;
- Compressed air supply; and
- Main Backfill system lines through 4 raises (or boreholes).

17 Recovery Methods

17.1 General

The process plant and associated service facilities will process run of mine (ROM) ore delivered to the primary crusher, to produce doré bars and tailings. The process encompasses crushing and grinding of the ROM ore, agitated tank cyanidation leaching, a Merrill Crowe circuit and smelting of the precipitate to produce doré bars that are then shipped to a refinery for further processing. Following cyanide destruction, the leached tailings will be thickened before placement in the tailings storage facility (TSF) or used as mine backfill.

17.2 Plant Design Basis

The FS design criteria were based on detailed testwork and experience from similar operations.

The key criteria selected for the plant design are:

- An average throughput of 3,506 t/d for 365 days per year;
- Design availability of 91.3%, being 7,998 operating hours per year, with standby equipment in critical areas, and
- Sufficient plant design flexibility for treatment of all ore types based on testwork completed at design throughput.

17.2.1 Throughput and Availability

An overall plant availability of 91.3% or 7,998 h/y was nominated. Benchmarking indicates that similar well operated plants with abrasive ores similar to those tested have consistently achieved 91.3% overall plant availability.

The throughput selected is mainly a function of the mining production schedule. From the review of testwork data, a plant throughput of 160 t/h is sustainable based on 100% of the SAG feed material from the Angela Vein. With 91.3% availability an average of 3,506 t/d can be processed. This equates to an annual mill capacity of 1,279,660 tonnes of ore.

17.2.2 Head Grade

The plant is designed to treat various tonnages of ore with a maximum feed rate of 160 t/h and average feed grades of 3.4 g/t Au and 120 g/t Ag.

17.2.3 Processing Strategy

The process design is based on treating the different ore types tested individually at the nominated design throughput rates. Inputs for the Ausenco comminution model were based on testwork completed on samples from Zone 1, Zone 2 and Zone 3 of the Angela Vein. Ore hardness parameters were selected based on the 75th percentile, i.e. 75% of the ore to be processed is expected to be similar or softer in hardness than the ore hardness parameters used for design.

17.2.4 Design Criteria Summary

The overall approach was to provide a robust process plant flowsheet that could handle the variability in the metallurgical performance of the three Zones based on the testwork. A key design strategy was to reduce the unit process stages required to reduce the capital and operating cost of the plant whilst maintaining metallurgical performance.

The detailed process design criteria derived from the results of the metallurgical testwork program A summary of the key criteria is shown in Table 17.1.

Table 17.1 - Key Process Design Criteria Summary

Area	Description	Unit	Design
ORE PROCESSING AND CHARACTERISTIC OVERVIEW			
	Gold	g/t	3.4
	Silver	g/t	120
	Bulk density of crushed ore	t/m^3	1.6
	Humidity in RoM ore	% H_2O	4%
	Crushing Work Index	kWh/t	20
	UCS: Design	MPa	65
	Drop Weigh Index (design)	kWh/m^3	4.6
	Rod mill bond work index	kWh/t	18
	Ball mill bond work index	kWh/t	21
	Abrasion index		0.44
	Ore annual processing	t/y	1,279,660
	Available hours per day	h/d	24
	Plant availability	%	91.3
	Design feed rate	t/h	160
	Hours of operation per year	h	7,998
	Plant nominal processing per day	t/d	3,506
	Plant maximum processing per day	t/d	3,840
PRIMARY CRUSHER			
	Type		Jaw crusher
	Closed side opening	mm	115
SAG MILL			
	Size	m	6.4 x 3.1 EGL
	Installed Power	kW	2,000
BALL MILL			
	Type		Overflow
	Size	m	5.5 x 8.5 EGL
	Installed Power	kW	4,200

Area	Description	Unit	Design
MILL CYCLONES			
	Overflow P80	microns	50
	Cyclone diameter	mm	150
	Cyclone overflow density	% w/w	40
LEACHING			
	Number of tanks		7
	Leach residence time	h	96
COUNTER CURRENT DECANT			
	Number of stages		4
	Target washing ratio (m^3 of washing solution: m^3 of leach solution)		4
	Design underflow density	% w/w	55
MERRILL CROWE CIRCUIT			
	Feed flow, nominal	m^3/h	809
	Clarifier filters		2 operating, 1 standby
	Precipitate filters		2 operating, 1 standby
SMELTING			
	Number of mercury retorts		2
	Type of furnace		Reverbatory
	Smelting frequency	d/wk	5
DESTRUCTION OF CYANIDE IN TAILINGS			
	Method of destruction		SO_2/Air
	Target of destruction, CNWAD should not exceed	mg/L CN_{WAD}	<50
TAILINGS THICKENER			
	Type of thickener		High rate
	Underflow density	% w/w	55
	Addition of flocculant	g/t	50
PASTE FILL PLANT			
	Plant availability	%	50
	Fill production capacity	m^3/h	130

17.3 Flow Sheet Development

The FS plant flow sheet was designed based on the testwork results and benchmarked data from plants operating within the region on similar ore types. The overall plant flowsheet is shown in Figure 17.1.

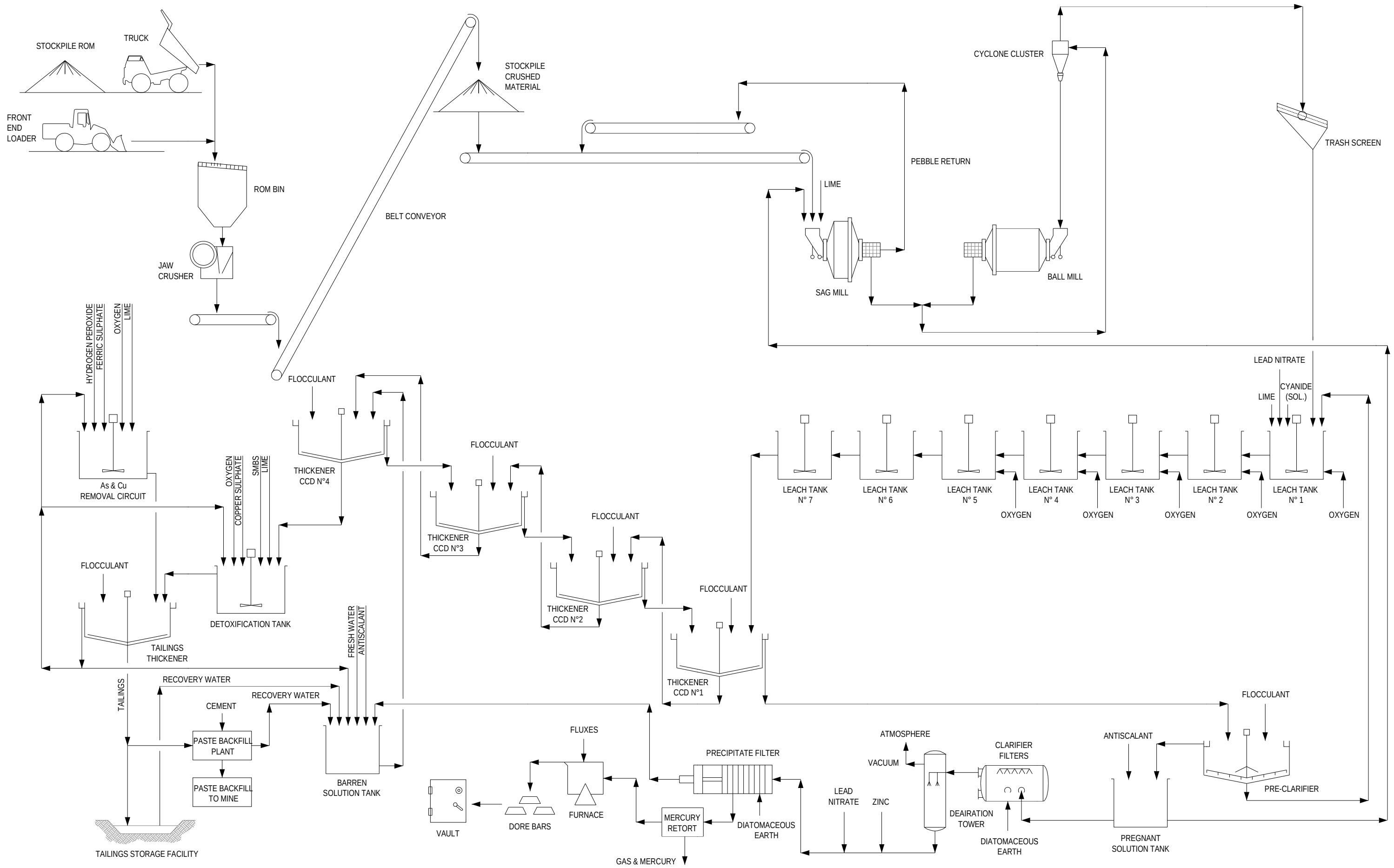


Figure 17.1 - Overall Process Plant Flowsheet

The process flowsheet includes single stage crushing, followed by grinding in a SAG-ball (SAB) milling circuit, operated in close-circuit with cyclones. The ground slurry is then fed to a leach circuit for precious metal dissolution in a cyanide solution. Precious metals leached into solution are in a counter current decantation (CCD) circuit and precipitated in a Merrill Crow circuit.

The precipitated gold and silver will be dried and then smelted in a reverbatory furnace to produce gold and silver doré bars.

The leach circuit tailings are treated for cyanide destruction by the INCO SO_2/Air method to reduce the cyanide (CN_{WAD}) level to <50 ppm, prior to release into the tailings facility. Water is reclaimed from the tailings storage facility and used as make-up process water.

17.3.1 Unit Process Selection

The unit operations used to model the plant throughput and metallurgical performance are proven in the gold/silver processing industry. The flowsheet incorporates the following major process operations:

- Ore from the underground mine is crushed using a primary jaw crusher with a close size setting of 115 mm and fed to a stock pile;
- A stock pile with live capacity of 1,920 t to provide crusher product surge capacity prior the SAG mill;
- A 2.0 MW SAG mill with pebble return;
- A 4.2 MW ball mill in closed circuit with hydrocyclones;
- 7 x 4,085 m^3 live capacity cyanidation leach tanks in series;
- 4 x CCD thickener washing stages and 1 solution clarifier thickener;
- An 810 m^3/h Merrill Crowe circuit;
- Retort system to volatilize and remove mercury from the precipitate cake;
- Diesel fired reverbatory smelting furnace;
- Leached tailings cyanide destruction utilizing the INCO SO_2/Air process;
- Tailings thickening in a high rate thickener to an underflow density of 55% solids;
- Centrifugal pumping of tailings to a conventional TSF;
- Raw process plant water supply from the site raw water dam throughout the plant as required. Process water is supplied from water reclaimed from the TSF, mine decant and process operations;
- Paste backfill plant to produce 130 m^3/h of cemented tailings to be used as part of mine filling;
- Potable water generated by treatment of raw water in a filtration/UV disinfection/chlorination treatment plant. Potable water is distributed to the plant and to various other locations around the site;
- Reagent preparation and distribution;
- Plant and instrument air services and associated infrastructure; and
- Pressure swing adsorption oxygen plant to provide oxygen for use in the leaching and cyanide destruction circuits.

17.3.2 Crushing

Run-of-mine (ROM) ore will be dumped from haul trucks or a front-end loader through a 800 mm square-grid static grizzly into the 100 t capacity ROM hopper. Ore is recovered from the ROM bin by a vibrating pan feeder that reports directly to the 1.1 x 0.86 m single toggle jaw crusher. The jaw crusher is designed to operate with a closed side setting of 115 mm and crushes ore to 80% passing (P80) 93 mm. The primary crusher product is conveyed to the crushed ore stockpile.

The crushed ore stockpile will have a live capacity of 1,920 t or approximately 12 hours of mill feed. Three reclaim vibrating feeders will be installed at the base of the crushed ore stockpile to reclaim ore for milling.

17.3.3 Grinding

The grinding circuit selected by Ausenco will be a standard SAB configuration with provision to return SAG mill pebbles to the SAG mill feed. The reclaimed crushed ore will be fed at a controlled rate to a 6.40 m diameter by 3.05 m effective grinding length (EGL) SAG mill. The SAG mill will be equipped with a single 2,000 kW wound rotor induction motor variable speed drive system and single pinion, allowing the mill to operate at 60 – 80% of critical speed.

The ball mill will be equipped with a single 4,200 kW wound rotor induction motor, trommel screen and retractable feed spout/chute. The combined mills discharge slurry will be pumped to the mill cyclone cluster operating in closed circuit configuration with the ball mill. Pregnant solution from the clarifier overflow is added to the cyclone feed hopper to achieve the required cyclone feed pulp density. Mill cyclone underflow will gravitate to the ball mill feed, and cyclone overflow will gravitate to the leach feed vibrating trash screen.

17.3.4 Leaching

Ground ore slurry will flow by gravity from the vibrating trash removal screen to the leaching circuit. Leaching of precious metals by cyanide occurs in a series of seven agitated leach tanks to provide a total leach residence time of 96 hours.

Sodium cyanide solution and lead nitrate solution are dosed to the leaching circuit via pressurized ringmains. Oxygen is sparged into all leach tanks and slaked lime slurry is added to increase the pulp pH levels.

A hydrogen cyanide (HCN) gas detector is located above the leach tanks to monitor the airborne concentration of HCN gas. A small laboratory is located in the leach area to analyze weak acid dissociable (WAD) and free cyanide concentrations in solution.

Slurry exiting leach tank 7 will gravitate to the CCD recovery circuit.

17.3.5 Counter Current Decantation (CCD) Concentrate Solution Recovery

The CCD circuit recovers precious metals leached into solution via four-stage counter-current thickener washing.

The CCD circuit comprises four high rate thickeners. CCD thickener overflow solution will pass to the previous thickener stage (4→3→2→1), while the underflow will be pumped to the next thickener stage (1→2→3→4).

Slurry from the final leach tank will flow via gravity to the first CCD thickener. Barren solution from the Merrill Crowe circuit is used as wash solution in the fourth CCD thickener. The wash solution will flow counter current to the solids flow, increasing in precious metal concentration as it proceeds to the first CCD thickener. Thickened underflow slurry from the fourth CCD thickener will be pumped to the tailings cyanide detoxification circuit. The pregnant solution from the first CCD thickener will report to the pre-clarifier thickener prior to Merrill Crowe.

17.3.6 Solution Clarification and Zinc Precipitation

17.3.6.1 Solution Clarification

Pregnant solution gravitates from the first CCD thickener to the solution clarification thickener. This solution, combined with flocculant and a recycle stream of clarifier underflow, feed the clarifier. Clarifier overflow gravitates to the clarifier overflow tank and is pumped to the polishing filter circuit. The majority of the remaining suspended solids in the clarifier will be removed by the clarification pressure leaf filters. Diatomaceous earth is metered into the filter feed stream to maintain the filter cake porosity. The clarified pregnant solution then flows to the de-aeration tower.

17.3.6.2 Pregnant Solution De-Aeration

Clarified pregnant solution is pumped through a de-aeration tower to reduce the dissolved oxygen content in the solution to below 1 ppm to enhance precious metal precipitation.

17.3.6.3 Zinc Precipitation

The de-aerated pregnant solution is pumped to the zinc precipitation filters. Zinc dust will be metered into the zinc cone with lead nitrate to precipitate gold and silver from the de-aerated solution. This mix will be added to the suction line on the precipitate filter feed pumps with de-aerated solution and pumped through the precipitate filters. Recessed plate filters are used for the precipitate filtration duty.

17.3.7 Gold Room

After precipitate filtration, the filter cake will be dried with high pressure air prior to discharge. Filter cake discharges from the filters into trays that are loaded into a mercury retort oven for calcining. The mercury is separated from the precipitate, then recovered and stored in sealed containers. Once the mercury is removed from the calcine cake, the calcine is mixed with smelting fluxes and charged into the reverbatory furnace. The calcine is smeltedand the doré is poured into ingots.

Slag from the smelt is crushed and returned to the SAG mill.

17.3.8 Tailings Detoxification

The cyanide detoxification circuit is designed to reduce weak acid dissociable cyanide, (CN_{WAD}) to less than 50 ppm. The circuit consists of a single agitated detoxification tank with a residence time of 3 hours.

The tailings thickener overflow water is mixed with the incoming slurry to reduce the slurry to 42% solids prior to detoxification. A 20% w/v sodium metabisulfite (SMBS) solution (a source of SO_2) is also added to the slurry in the feed box. Oxygen is sparged into the detoxification tank to maintain a high redox potential to maximize oxidation of the cyanide present. Lime is dosed into the tank to maintain the desired pH. Copper sulfate is added to catalyze the SO_2/Air (INCO) process.

17.3.9 Tailings Disposal

Detoxified slurry discharges by gravity into the tailings thickener. Flocculant, diluted to 0.025% w/v with fresh water, is added to assist solid-liquid separation. Thickener underflow is discharged to the tailings disposal tank at 55% w/w solids. Tailings slurry is then pumped to the tailings dam or to the paste backfill plant. A fraction of the tailings thickener overflow is recycled back to the cyanide detoxification for feed dilution and the rest reports to the barren tank. Provision has been included in the design to bleed this stream intermittently to the Arsenic and Copper removal circuit.

17.3.10 Copper and Arsenic Removal

The design includes provision to bleed the tailings thickener overflow stream through a copper and arsenic precipitation circuit. It was identified during the study that copper and arsenic levels in the feed increase in Zone 3 (at depth) and this may have a detrimental effect on the zinc precipitation circuit as these elements build-up in the circulating water streams. This circuit comprises two reaction tanks where hydrogen peroxide and then ferric sulfate are added respectively followed by two precipitation tanks. The solution is then returned to the tailings thickener where the precipitates are settled and pumped to the tailings storage facility.

17.3.11 Paste Backfill

The underflow thickener tailings slurry is pumped at 55% solids in intermittent form to the paste backfill plant when mine backfilling is required. Tailings then filtered by disk filters to achieve approximately 72% solids and cement is added to give the adequate properties for mine backfill. Reclaimed filtrate water from paste backfill is returned to the barren solution tank. The paste is transported to the mine backfill boreholes either by positive displacement pumps or trucks.

17.3.12 Water Services

The separate water circuits include:

- Raw water;
- Tailings Storage Facility (TSF) decant water;
- Potable water; and
- Fire water.

Raw water will be sourced from the raw water dam to service the process plant, and general site buildings. Water will be delivered to a fresh water storage tank located at the plant. The fresh water storage tank will supply water for process make-up requirements, gland water, dust suppression, reagent mixing and fire services.

The plant raw water tank will have a dedicated reserve of firewater. Firewater will be distributed throughput the plant, laboratory, camp, truck workshop and fuel storage areas via a dedicated firewater pump system, which will include a back-up diesel powered pump.

Potable water is sourced from wells. The water is filtered and sterilized by ultra violet (UV) and chlorination prior to being distributed to the camp, buildings and process plant. Potable water is used for all eye wash stations and safety showers.

17.3.13 Reagents

The main reagents used in the process are outlined below:

- Lime is used to control the pH in the leaching and INCO detoxification circuits;
- Flocculant is used in the CCD thickeners, pre-clarifier and tailings thickener;
- Sodium metabisulfite (SMBS) is used in the INCO cyanide detoxification circuit;
- Copper sulfate is used as a catalyst in the INCO cyanide destruction circuit;
- Sodium cyanide (NaCN) is used in the leaching process for the dissolution of gold and silver;
- Lead nitrate is used in the leaching and zinc precipitation circuits;
- Zinc powder is used in the zinc precipitation circuit;
- Diatomaceous earth is used in the clarification and precipitate filters as a filtering media; and
- Ferric sulfate and hydrogen peroxide are both used in the copper and arsenic precipitation circuits.

17.3.14 Air Supply and Oxygen Services

Plant air is supplied by four rotary screw type compressors. One compressor is dedicated to the crushing area. The remaining three compressors (2 on duty and 1 on standby) provide plant and instrument air for the process plant. Instrument air is dried and filtered to remove entrained moisture and particles.

Oxygen is to be supplied to the leach and cyanide detoxification circuits by a pressure swing adsorption (PSA) plant of 13.7 t/d capacity.

17.4 Process Control Philosophy

The plant is designed to incorporate a modern level of automation. A supervisory control and data acquisition (SCADA) system will control the process programmable logic controllers (PLC), instrumentation and control valves. Condition monitoring of equipment for maintenance purposes will be recorded on the SCADA system.

18 Project Infrastructure

18.1 Introduction

The project consists of an underground mine, crushing, stockpile conveyor, coarse ore stockpile, SAG and ball mill grinding circuit, cyanidation leaching, CCD thickening, Merrill Crowe precious metal recovery, smelting, tailings detoxification, paste backfill, tailings storage facility, raw water dam, permanent camp, reagents, ancillary services and associated infrastructure.

The site layout takes into account site topography and limits imposed by the locations of the underground portal, stockpiles and waste dumps subject to the above constraints. The grinding area is founded on bedrock to reduce civil costs and take advantage of gravity flow where possible.

18.2 Site Access Road

There are two major access roads to Inmaculada project site; one coming from Lima (north) and another from Arequipa (south). Access to site from the north follows the route Lima – Ica- Nazca – Puquio – Iscahuaca – Inmaculada site (approximately 775 km). The road from Matarani port (SW of Arequipa) follows: Matarani - Chuquibamba – Cotahuasi – Inmaculada Site (approximately 607 km).

A route survey was undertaken on the two potential main access roads to site:

- From the port of Pisco (250km south of Lima) to Iscahuaca and onto Inmaculada site; and
- From the port of Matarani to Cotahuasi and onto the Inmaculada site.

The survey indicated that the road requiring the least effort to make acceptable for shipping construction freight safely to site would be the road from Pisco to Inmaculada via Iscahauca. However, road works are required in several sections to make this road safe for construction traffic.

Site access from the north is divided into two sections for the purpose of describing the road condition and repair work required:

<u>Section 1: Pisco – Ica – Nazca – Puquio – Iscahuaca</u>

This road is in an acceptable condition adequate to transport dimensions of 5 m wide, 4.5 m high and 18 m long (the project maximum transport envelope).

The road is asphalted with minimum bridge bearing capacity of 36 t up to Iscahuaca. Heavier loads may be transported through this section if minor road works such as river crossings are constructed.

<u>Section 2: Iscahuaca to Inmaculada Site</u>

From Iscahuaca the route continues as a dirt road for approximately 182 kms to the project site. The road follows: Iscahuaca - Selene - Huanacmarca - Huarcaya - Sauricay - Calaccapcha - Sorani - Oyolo Cotahuasi – Inmaculada camp.

After Iscahuaca and just past the turn-off to Selene, the road becomes difficult with winding terrain at an average altitude of 4 800 masl. In this section there are no height limitations but there are minimum road widths down to 3 m, tight curves, areas with unstable slopes and river crossings that require widening, upgrade and repair works.

To date this road has been in operation and used for medium sized truck deliveries to site. Road works however are required in several sections of the road to make the road suitable for 30 - 40 t low bed trailers required to carry the heaviest loads during construction. A summary of this preliminary work plan required to upgrade the access road from Iscahuaca to Inmaculada, and included in the FS capital cost estimation is:

- Signage along the road as there is currently very little;
- Armco guardrails required on road sections that have steep drop-off;
- Modification and/or construction of new culverts;
- Road widening works;
- Topographical and geotechnical investigations; and
- New road sub base for identified areas.

18.3 High Voltage Power Supply

The power supply to site was excluded from the scope of Ausenco and managed directly by Suyamarca. The supply of electrical energy for the Inmaculada project facilities is assumed to be furnished by the National Interconnected Electric System (SEIN – as per acronym in Spanish). Power will be sourced from a 220 kV sub-station located at Cotaruse and transmitted to the project via 63 km of high tension overhead power lines. The power line will also provide power to the Selene and Pallancata operations, and will be stepped down to 66 kV at Selene for the remaining 38 km to Inmaculada. The capital cost of this project was apportioned by IMZ and Hochschild between the three projects, and Ausenco was advised on the cost to be assigned to the Inmaculada project. The route of the power line is shown in Figure 18.1.

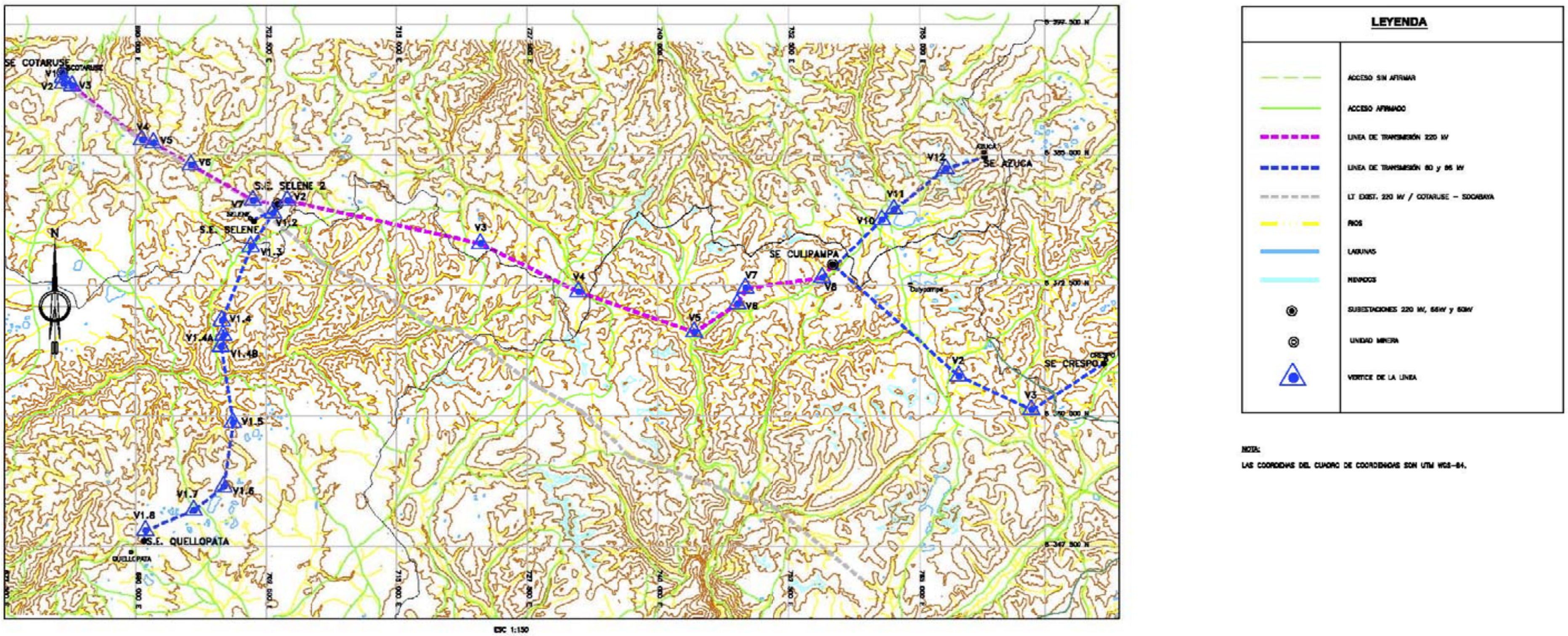


Figure 18.1 - Route of High Voltage Transmission Line

18.4 Site Power Distribution

Incoming high voltage power is stepped down at the main substation from 66 kV to 10 kV for distribution to the various remote substations located around the project site. These substations are located at:

- Process plant (multiple);
- Crushing plant;
- Paste backfill plant;
- Mine Portal; and
- Camp.

The power is then reduced at these sub stations to a low voltage supply for equipment of 460 V. Medium voltage motors such as the grinding mills are supplied at 4.16 kV. Each remote substation will consist of:

- Step-down transformer, pad or pole mounted;
- Set of pole fuses;
- Set of lightning arrestors for lightning protection;
- Motor control centre; and
- Lighting and small power board.

The various plant substations will be provided with load centres for the distribution of electricity to the process plant drives and other services. All process plant substations, including the main substation, will be of containerized transportable type. Each substation will include variable speed drives for the relevant equipment, light and small power board and control system input/output cubicle.

18.5 Raw Water Supply

Raw water for the project needs will primarily be sourced from a raw water dam. The raw water dam total volume is 458,000 m³ with an area of 72,400 m². The dam is constructed using a low permeability clay liner followed by the installation of a 1.5 mm thick high-density polyethylene (HDPE) liner.

Water will be pumped from the dam to the process plant raw water storage tank. This tank will also contain a dedicated fire water reserve.

18.6 Potable Water

A potable water treatment plant will be located near the camp and supply the potable water requirements for the project. Raw water from wells located near the camp will be pumped to the potable water plant. The treatment plant consists of sand filtration, UV sterilization and chlorination stages and will produce 122 m³ of potable water per day. Potable water will be distributed to the camp, administration buildings, workshops and process plant.

18.7 Sewage Treatment

A central sewage treatment plant will be located on the project site at the camp. Sewage and domestic waste water will be delivered to small pump pits and then pumped to the sewage treatment plant.

18.8 Administration and Plant Site Buildings

18.8.1 Maintenance Facilities and Fuel Storage

The maintenance shop will be divided into different areas: a washing station, mine equipment maintenance area, welding area, oil/lubricants area, spares storage area and maintenance staff office.

Fuel storage and gas stations will have an approximate area of 0.015 ha and consist of diesel storage and bowser facilities for truck and light vehicle filling. The fuel storage area will have a secondary impermeable containment system capable of storing 110% of its capacity in order to prevent potential leaks to the environment.

18.8.2 Laboratory

A laboratory will be provided within the process plant that is capable of providing fire assay analysis for gold and silver samples. In addition to this a separate sample preparation room will be provided for the mine samples outside of the process plant restricted area.

18.8.3 Accommodation

A 545-person construction camp will be built prior to Q1 2013 to supplement the existing 300 person permanent camp bringing the total camp capacity during construction to approximately 850 persons. The camp will be of modular construction to facilitate quick set-up as well as having the ability to expand and contract to suit the requirements of the project.

18.8.4 Administration Offices

The administration building includes both single and double occupancy offices, training room, medical room, server room and sanitary facilities. The building will have power to service computers and peripherals.

18.8.5 Warehouse

The main warehouse and storage yard shall have a total approximate area of 2.4 ha.

All chemical compounds supplied to the site and exclusively used by the process plant shall first be received at the warehouse, and then dispatched to the reagents warehouse within the plant area.

18.8.6 Ancillary Buildings

In addition to the buildings outlined above, the following ancillary buildings will be constructed for the project:

- Main site access and plant access security gate houses;
- Plant mess, sanitary facilities and training room;
- Plant office;
- Plant control room;
- Plant maintenance workshop and storage;
- Plant reagent storage;
- Gold room; and

- Switch room buildings (containerized type).

18.9 Security

The project will be surrounded by a single 3 strand wire fence, with a single guarded entry point to the site. This fence is designed to deter livestock and animals from entering the site, as well as people who may be un-aware of the activities occurring within the site.

The process plant will be provided with a designated single security fence. The security fence will enclose the plant, plant workshop and warehouse, laboratory, crib room and metallurgical offices. Access to the plant security area will be controlled at the gatehouse. The gatehouse is approximately 36 m^2 in area and will comprise:

- Personnel access via turnstile;
- Vehicle access via boom gate;
- Office for security personnel;
- Two search rooms; and
- Toilet.

18.10 Communications

A satellite system currently provides voice and data transmissions to site. This system will be upgraded to cater for the increased operating load. In addition to this the current VHF radio communication repeaters will be used for the surface operations.

18.11 Internal Plant Roads

Internal plant roads have been sized to suit traffic associated with the plant maintenance mobile equipment and vehicles used to deliver consumables and reagents for the plant. Road widths have been set at 8m, which allows for a clearance width of 6 m. This is adequate for an 80 tonne maintenance crane to have access to all areas of the plant. Turning circles at the warehouse have been set at 13.5 m to suit semi-trailer traffic delivering consumables. The fuel tank is located at the plant site and fuel will be pumped from this location.

18.12 Tailings Storage Facility

18.12.1 General

The tailings dam is located to the north of the process plant in a natural valley as shown in Figure 18.2.

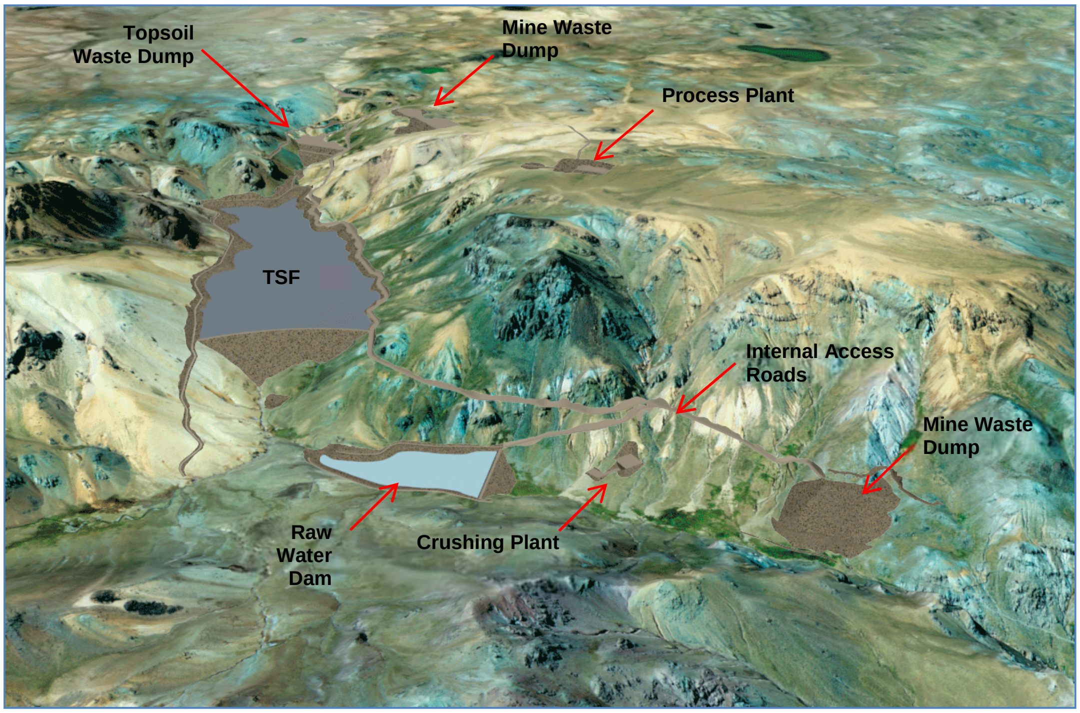


Figure 18.2 - Tailings Storage Facility in relation to the Process Plant (looking east)

The dam has been designed in two stages, with an initial storage volume of 3.72 Mm^3 and then a final volume of 7.72 Mm^3. The dam will be lined by a 1.5 mm HDPE impermeable plastic liner to prevent seepage to the environment. Rain water diversion channels will be constructed to prevent catchment water entering the TSF. In addition to this perforated 200 and 300 mm HDPE pipes are installed below the TSF to collect groundwater flows, and these report to the seepage pond at the basement of the TSF wall for quality measurement.

18.13 Water Management Plan

The water management plan must facilitate the operation of the mine development through a wide range of climatic conditions, while at the same time protecting the environment. The prime objectives of the water management plan will be to:

- Provide a reliable water supply to the plant;
- Facilitate mining of the ore deposit by managing the inflows and decantation of the mine;
- Provide sediment control;
- Collect and treat contact water that would otherwise impair water quality of receiving streams; and
- Protect mine infrastructure during extreme flood events.

The water management plan presented in summary form in this section must be viewed as a first iteration that will be optimized during the engineering of the project. The conceptualization of the water management plan considered the full evolution of the mine from construction to final closure.

18.13.1 Drainage

The plant site will be shaped, graded and sheeted to provide trafficable areas during construction and operation. Drainage of the site will fall into two general categories:

- Process area runoff; and
- Hard standing runoff.

All storm water runoff within the plant area is considered to be polluted and will be channeled to the tailings storage facility.

The plant site processing areas will be bunded, with concrete floors. All process, chemical or reagent spillage will be directed to area sump pumps and pumped back into the process plant.

All maintenance and chemical storage areas will be suitably bunded to allow collection and proper disposal of wastes.

18.13.2 Clean Water Diversions

Development of the TSF will require the diversion of clean rain water run-off in the tailings dam valley around the dam. The water will be diverted around the tailings dam and flow by gravity in open channels. As with all diversion and collection ditches at the mine, this diversion will be designed to convey the 100-year peak instantaneous discharge with an adequate freeboard.

18.13.3 Sedimentation and Seepage Ponds

A sedimentation pond has been included at the base of the mine waste dump. Preliminary geochemical analyses indicate that the waste rock is not acid generating with minimal leaching of heavy metals. Therefore no water treatment plant was included.

A seepage pond was included at the base of the TSF. Whilst the TSF will be a fully plastic lined facility, a pond was included in order to monitor and measure any seepage or sub-surface flows to ensure cyanide and heavy metals do not enter downstream water systems.

18.13.4 Tailings Storage Facility

The TSF will serve two key roles in the management of water at the mine. Firstly, runoff generated within the catchment of the process plant will report to the TSF. Secondly, the pond within the TSF will serve as a process water reservoir and will be recycled back to the process plant.

18.13.5 Project Site Water Balance

The overall project site water balance is shown in Figure 18.3.

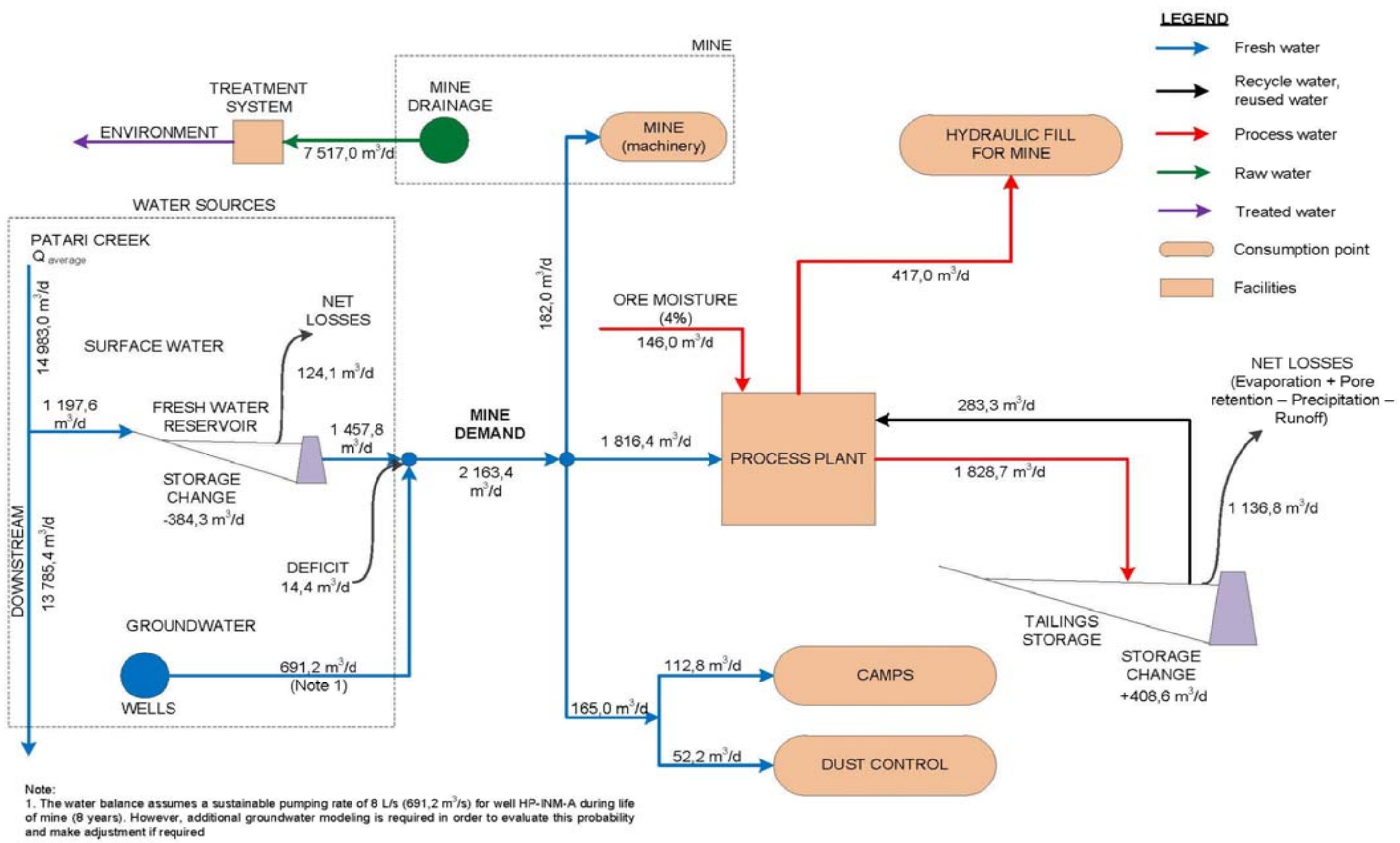


Figure 18.3 - Average Site Water Balance (Year 1)

In the latter stages of the FS, drilling and testing conducted for hydrogeological characterization showed that the planned underground mine would produce a significant quantity of water. For the purposes of the FS it was assumed that this water was of sufficient quality to be pumped and discharged under controlled conditions to surrounding areas. In the design stages further testwork is required to confirm this assumption. If proven, it is likely that the raw water dam could be removed and the water derived from underground mining operations can be used for the process plant requirements.

19 Market Studies and Contracts

Inmaculada will produce and sell a gold and silver doré to generate revenue for the project. Over the life of the mine, the process plant will produce doré bars containing a total of 0.78 Moz of gold and 26.5 Moz of silver. The doré will be sold to a third-party refinery, typically based on prices set by the London Bullion Market. IMZ will not hedge its gold or silver production.

19.1 Refining

19.1.1 Refinery Options

There are numerous refineries in Europe and North America that represent the shortest distance from the project.

- Johnson Matthey — Brampton, Canada and Salt Lake City, United States
- Royal Canadian Mint — Ottawa, Canada
- Argor-Heraeus — Mendrisio, Switzerland
- Cendres & Métaux — Biel-Bienne, Switzerland
- Metalor — Marin, Switzerland and North Attleboro, United States
- PAMP SA — Castel San Pietro, Switzerland
- Valcambi — Balerna, Switzerland

19.1.2 Commercialization Costs for Doré

Costs for the offsite commercialization of the doré, including transportation, insurance, agent fees and refinery treatment and metal retention charges were benchmarked based on contracts for similar operations in Peru. The overall cost over the life of mine is approximately 1.2% of the gross metal value.

20 Environmental and Social Studies

The information contained in this section is a summary of the environmental and social impact study (EISA) work completed by SVS Ingenieros S.A.C. on behalf of Suyamarca. Ausenco was not involved in the execution of the EIS but has reviewed the work completed to date and believes this summary to be a fair representation.

20.1 Legal Framework

The legal framework applicable to the EIS is based on Peru's 1993 Political Constitution. The main articles that regulate the environmental aspects are: Article 2.22, which sets forth the entitlement to a healthy environment, and articles 66, 67, 68, and 69, which set forth that natural resources are a national heritage, regulating their exploitation and establishing the State's obligations to the environment.

The regulations that form the basis of the EIS are the General Environmental Law (Law 28611), the Legal Framework of the National Environmental Management System (Law 28245), which regulates the EIS approval process; the Organic Law for Sustainable Natural Resource Exploitation (Law 26821); the Wild Forest and Fauna Law (Law 27308); the Natural Protected Area Law (Law 26834); the Water Resources Law (Law 29338); the General Solid Waste Law (Law 27314); the General Law of National Cultural Heritage (Law 28296); and other standards that regulate the citizen participation process, such as the Citizen Participation Regulation in the Mining Subsector (Supreme Decree 028-2008-EM). There are also environmental guides in various sectors that are used as references for the preparation of the EIS.

20.2 Environmental Impact Study

20.2.1 Introduction

Suyamarca contracted SVS Ingenieros S.A.C. to prepare an EIS for the Inmaculada project. The EIS was submitted to the Ministry of Energy and Mines (MINEM) in September 2011 and is currently being evaluated by this entity. At least one round of review and comments is expected on the document before the EIS permit is approved, sometime during 2012.

Baseline studies were completed to describe the natural and social environment associated with the project. This information was then processed and analyzed. Potential environmental and social impacts of the project were determined. This analysis allowed the establishment of the projects direct and indirect areas of social and environmental influence as well as the development of the project's environmental management plan.

20.2.2 Physical Environment

The Property is located in the district of Oyolo, province of Paucar del Sara Sara in the department of Ayacucho at an approximate height of 4,500 masl.

The climate is cold to mild, with an annual average temperature ranging between 9.5 and 12.5 ºC. There are two marked seasons, the rainy season, which starts in November and ends in April (maximum rainfall of 213 mm in February), and the dry season, which starts in May and ends in October (minimum rainfall of 5 mm in July). The total accumulated rainfall in one year is 870 mm. The predominant wind directions are east northeast and southeast.

Two air and noise quality monitoring campaigns were carried out. Two stations were placed within the area immediately surrounding the project and one station was placed near the town of

Huancute (town closest to the project). All data collected at these stations reflect levels within those established by current Peruvian regulations.

The project facilities will be located on essentially unoccupied terrain comprising meadows with occasional rocky outcrops and wetland patches.

The project is located in the Patarí Gorge (20.79 km^2) and Quellopata Gorge (8.05 km^2) micro-basins, which are in the high basin of the Ocoña River, in the Pacific watershed.

Two surface water monitoring campaigns were carried out in these micro-basins, at 10 sampling stations in the first campaign and 12 sampling stations in the second. Almost all the recorded surface water quality values are within the values established in the Environmental Quality Standards for Water, categories 3a and 3b. The waters of the Patarí- and Quellopata Gorge micro-basins exhibited a moderately acid to moderately alkaline character which is typical for the area.

Depth to ground water ranges between 5.3 m and 40.0 m and ground water quality is characterized by moderately alkaline to strongly alkaline character. Almost all the recorded ground water quality results reported are within the range stipulated in the Environmental Quality Standards for Water, categories 3a and 3b.

20.2.3 Biological Environment

The area of study is located at the Sub-Tropical Sub-Alpine humid bleak upland life zone between 4,000 masl and 4,300 masl and Subtropical Alpine very humid Tundra between 4,300 masl and 5,000 masl, corresponding to the Puna Eco-region (3,800 masl and above) where five ecological categories are recognized: scrubland or steppe of *gramineae*, scrubland and low bushes, puna sod, wetlands, and farm areas.

The flora, fauna and aquatic biota in the project area were assessed with 24 sensitive species being recorded including *Vultur gryphus* "condor", a species in one of the highest categories of conservation, with a further seven species being endemic to Peru. However, all 24 of these species have been recorded at various other locations outside the Department of Ayacucho, which enables, through the implementation of appropriate management programs, to reduce or offset the potential impact that the project can cause on their populations.

Table 20.1 - List of Sensible Flora and Fauna

Biological Group	Species	Conservation Status			Endemics Species[1], [2], [3]
		National Legislation[4],[5]	IUCN[6]	CITES[7]	
Flora	*Acaulimalva richii*		VU		X
	Gentianella dianthoides		VU		X
	Opuntia floccose			II	
	Perezia coerulescens	VU			
	Senecio nutans	VU			
	Xenophyllum staffordiae				X
Mammals	*Akodon juninensis*				X
	Calomys sorellus				X
	Hippocamelus antisensis	VU	VU	I	
	Leopardus colocolo			II	
	Lycalopex culpaeus			II	
	Puma concolor			II	
	Vicugna vicugna			II	
Birds	*Bubo virginianus*			II	
	Buteo polyosoma			II	
	Colibri coruscans			II	
	Falco femoralis			II	
	Falco sparverius			II	
	Phalcoboenus megalopterus			II	
	Phoenicopterus chilensis			II	
	Tyto alba			II	
	Vultur gryphus	EN		I	
Reptiles	*Liolaemus melanogaster*				X
	Liolaemus polystictus				X

[1] Carrillo, N. & Icochea, J. 1995. Lista taxonómica preliminar de los reptiles vivientes del Peru. Publ. Mus. Hist. Nat. UNMSM (A) 49, 1-27.

[2] León, B., J. Roque, C. Ulloa Ulloa, P.M. Jørgensen, N. Pitman y A. Cano (eds.) 2007d. Libro Rojo de las Plantas endémicas del Peru. Rev. peru biol. Edición Especial 13(2): 971 pp.

[3] Pacheco, V., R. Cardenillas, E. Salas, C. Tello, & H. Zevallos. 2009. Diversidad y endemismos de mamíferos del Peru. Rev. Peru biol. 16(1): 005 - 032.

[4] Supreme Decree Nº 043 - 2006 AG. Categorization of wild flora threatened species, and their collection, transportation or exportation with commercial purposes is prohibited. Lima - Peru.

[5] Supreme Decree Nº 034 - 2004 AG. Categorization of wild fauna threatened species, and their hunting, capture, possession, transportation or exportation with commercial purposes is prohibited. Lima - Peru.

[6] IUCN. 2011. List of Species. Red List of Threatened Species of the International Union for the Conservation of Nature and Natural Resources. Internet address: http://www.iucnredlist.org

[7] CITES. 2011. The CITES Species. Convention on the International Trade of Wild Fauna and Flora Threatened Species. Internet address: http://www.cites.org.

Meaning of the protection categories, according to the International Union for the Conservation of Nature and Natural Resources (IUCN):

- CRITICALLY ENDANGERED (CR) - A taxon is Critically Endangered when the best available evidence indicates that it meets any of the criteria A to E for Critically Endangered, and it is therefore considered to be facing an extremely high risk of extinction in the wild.
- ENDANGERED (EN) - A taxon is Endangered when the best available evidence indicates that it meets any of the criteria A to E for Endangered, and it is therefore considered to be facing a very high risk of extinction in the wild.
- VULNERABLE (VU) - A taxon is Vulnerable when the best available evidence indicates that it meets any of the criteria A to E for Vulnerable, and it is therefore considered to be facing a high risk of extinction in the wild.
- NEAR THREATENED (NT) - A taxon is Near Threatened when it has been evaluated against the criteria but does not qualify for Critically Endangered, Endangered or Vulnerable now, but is close to qualifying for or is likely to qualify for a threatened category in the near future.
- LEAST CONCERN (LC) - A taxon is Least Concern when it has been evaluated against the criteria and does not qualify for Critically Endangered, Endangered, Vulnerable or Near Threatened. Widespread and abundant taxa are included in this category.

20.2.4 Human Interest Environment

An Archaeological Reconnaissance Report has been prepared for the area of the project, with an area of 2,286.54 ha being assessed. 23 possible archaeological sites were identified, 9 of which are directly located on areas corresponding to components of the Inmaculada project footprint. The archaeological indicators are mainly rock shelters, corrals, and concentrations of lithic material. In order to protect potential archaeological evidence, Suyamarca is currently in the process of obtaining the Certificate of Inexistence of Archaeological Remains (CIRA) for the Project area. This step includes the presentation of an Archaeological Project to the Ministry of Culture, where the relevance of these cultural remains to the viability of the project will be determined; similarly, the document will also explain the mitigation policies to be applied, which may include an Archaeological Rescue or Salvage Project, if required.

20.2.5 Stakeholder Mapping

There are no major towns or villages located in the immediate surroundings of the Inmaculada project footprint. However, the project will be developed on surface land owned by the community of Huallhua and the inhabitants of the village of Huancute, located five hours and one hour drive respectively from the project. The key stakeholders identified are:

- For the social direct area of influence:
  - The inhabitants of the community of Huallhua (with 110 inhabitants);
  - The inhabitants of the village of Huancute (38 inhabitants);
  - Lieutenant Governor of the district of Oyolo, province of Páucar del Sara Sara (where the mining Project is located);
  - Lieutenant Governor of the district of San Javier de Alpabamba, province of Páucar del Sara Sara (where the community of Huallhua is settled);

- Lieutenant Governor of the district of San Francisco de Ravacayco, province of Parinacochas (where the small village of Huancute is located); and
- Representatives from local organizations of both towns.

- Key impacts and expectations are as follows:
  - Two families in Huancute may need to be relocated; and
  - The residents of the communities of Huallhua and Huancute expect to obtain employment and support for community-development projects.
- For the social indirect area of influence:
  - Municipal authorities of the districts of Oyolo, San Javier de Alpabamba, and San Francisco de Ravacayco, based on the fact that different components of the Project are developed in their district jurisdictions); and
  - The inhabitants of the small villages of Patarí, Belén, and Cascara, based on their expectations generated in terms of employment or support to their towns, as they are close to the project execution area.

20.2.6 Status of Land Ownership

Current status of the property

The Inmaculada mining project is located on surface land of the community of Huallhua and on non-agricultural land that belongs to proprietors of the village of Huancute.

The community of Huallhua is registered as such in the registration offices of the Ministry of Agriculture and, as a legal entity it is the owner of the lands. The community assembly assigns to each resident land suitable for the cultivation of crops, while the lands designated for grazing are the property of the community.

In the village of Huancute, the property rights are individual; however, the grazing lands are owned and used by the community.

Land acquisition

The Inmaculada project has acquired mining easement rights on 870 ha of rural land that belongs to the community of Huallhua, and also on 1,500 ha that belongs to individual proprietors of the village of Huancute.

As a result of the project development, at least two families from the village of Huancute will need to be re-settled.

20.2.7 Impact Identification and Evaluation

The identification and evaluation of potential foreseeable environmental impacts (positive and negative) of the project, in its different stages, comprised the systematic analysis of the projects insertion into the environment and the expected response of the environment to the project.

Based on the results, control and mitigation measures were identified in an environmental management plan, including monitoring plans that will comply with current environmental regulations in Peru and the environmental policy of Suyamarca.

The results of the impact prioritization in the EIS document shows that the potential impacts of higher importance have either "moderate significance" or "little significance". No potential impacts of high significance were identified by the study.

The main potential environmental impacts of moderate significance are described below:

- Alteration of the local relief and landscape;
- Increased surface and sub-surface vibration levels;
- Alteration of the superficial drainage network;
- Alteration of surface water flows;
- Alteration of groundwater quality;
- Alteration of grass quality from soil compaction;
- Alteration from changes in soil use; and
- Alteration of terrestrial flora.

Probable social impacts are as follows:

- Positive: Generation of non-qualified job positions for local inhabitants, especially during the construction phase;
- Negative: Decrease in land currently available for grazing;
- Negative: Physical re-settlement of families from the village of Huancute; and
- Negative: Alteration or relocation of archaeological sites.

20.2.8 Environmental Management Plan

The Environmental Management Plan (EMP) provides the environmental measures necessary to prevent, control and mitigate possible impacts that might be generated by typical project activities. The application of these measures is designed to ensure protection of the physical, biological and social environment.

The EMP consists of permanent monitoring activities of the project for each facility. It provides information for the assessment of identified impacts and the effectiveness of the mitigation measures applied.

20.2.9 Community Relations Plan

The community relations plan aims at enforcing compliance with legal regulations, environmental standards, and agreements signed with neighboring communities during the execution of diverse operations performed by Suyamarca and its contractors. Based on this, specific social programs will be designed for prevention and mitigation of identified social impacts. These programs are as follows:

- Land acquisition and procurement of easements (purchase of lands, compensation and relocation of affected proprietors);
- Local employment (hiring of local workers);
- Communication and consultation (strengthening of Suyamarca image by means of informative workshops, and permanent attention given to claims and complaints);
- Participative environmental and social monitoring;

- Training for local inhabitants and education of local organizations; and
- Sustainable development programs.

20.2.10 Closure Plan

A conceptual closure plan has been developed for the project within the EIS that ensures that the reclamation of impacted areas, will comply with all requirements of Peruvian regulations for closure. Approval will be necessary from the Ministry of Energy and Mines to ensure a post-closure sustainable environmental and social condition, which will be verified by future monitoring activities.

Surface Reclamation and Re-vegetation Plan

During the construction stage the topsoil and subsoil will be recovered and stockpiled, to provide adequate material for the cover reinstatement.

A re-vegetation plan, in order to allow satisfactory development of the species, adaptation to conditions predominant in the area of influence is proposed in the EIS. Native high Andean grass species from the zone will be used for re-vegetation, which show satisfactory conditions for optimum root-taking. This contributes to minimizing soil erosion and maintaining pre-project vegetation.

Reclamation and Closure by Facility

The conceptual closure activities of the most important project components are described next:

Progressive Closure

Progressive closure activities will be executed during the operative lifespan of the mine on those components where no further activities are envisaged.

Final Closure

- Decommissioning and demobilization

  In addition to decommissioning and removal of the installed infrastructure, the reclamation of access roads that will remain for post-closure monitoring is also considered.
- Removal, salvage and disposal

  This activity comprises the removal of concrete structures, backfilling with material from the zone, re-contouring and re-vegetating, and the disposal of solid waste generated from this activity.
- Establishment of terrain stability

  This work includes backfilling aspects, reconstruction, and integration of recovered surfaces with the natural landscape.
- Physical and chemical stability of previously mined area, waste dumps, top soil storage areas and the TSF.

Social Closure Activities

It has been anticipated that all social investments will decrease in a gradual manner in the last five years of operation, and that counterbalance amounts will be generated through sustainable development initiatives implemented during the mine life and/or social organizations or public entities.

Monitoring and Reporting

After closure activities of all components and the reclamation of the project area are completed, monitoring and maintenance tasks will be performed by Suyamarca for a minimum 5-year period. This will demonstrate that previously affected areas are stable with respect to physical, chemical and biological conditions, and that they will not generate negative impacts over the long-term.

20.3 Public Participation Process

As set out by Peruvian Legislation, D.S. N° 028-2008-EM, a public participation process must be implemented in order to assure active participation of the population that is directly- or indirectly affected by the mining project. In this context, the following consultations were held:

- First Workshop – Before the beginning of the Impact Assessment Study, with 53 and 43 attendees from the communities of Huallhua and Huancute respectively (24 and 25 March 2011);
- Second Workshop – Simultaneously held with the Impact Assessment Study, with 41 and 29 attendees from the communities of Huallhua and Huancute respectively (23 and 24 July 2011);
- Participative Environmental Monitoring, consisting of water quality monitoring during the dry season (28 to 30 June 2011);
- Participative Rural Assessment Workshops (16 and 17 July 2011); and
- Public Audience (15 December 2011).

The results of the public participation process were positive, and all the workshops were held in coordination with Suyamarca, the local population and local authorities.

21 Capital and Operating Cost Estimate

21.1 Capital Cost Summary

21.1.1 General

The capital cost of the project has been estimated based on the scope defined in the previous sections.

The following parties have prepared the capital cost estimate:

- Ausenco Peru S.A.C.
  - Process plant and tailings treatment;
  - Plant infrastructure and services with the exception of the high-voltage transmission line and high-voltage substation;
  - Permanent accommodation camp;
  - TSF and raw water dam;
  - Waste dumps;
  - Internal site access roads;
  - Working capital
  - Main access road and plant site bulk earthworks; and
  - EPCM costs relating to the process plant and infrastructure outlined above.
- MICSAC
  - Mine development; and
  - Mine equipment and infrastructure.
- Suyamarca
  - Site power supply including the high-voltage substation; and
  - Owner's cost.

21.1.2 Capital Cost Estimate Summary

Table 21.1 provides the summary of the overall capital costs (Capex) estimated for the Inmaculada project. The costs are expressed in October 2011 United States Dollars. There are no allowances for escalation. The estimated costs include all mining, site preparation, process plant, TSF, rock waste dumps, raw water dam, first fills, buildings, road works and permanent camp. The estimates are considered to have an overall accuracy of ±15% and assume the project will be developed on an EPCM basis.

The following parameters and qualifications are made:

- Estimate was based on October 2011 prices and costs;
- Estimate is only for Capex starting on January 2012; Capex prior to January 2012 is excluded. These past expenditures include mostly exploration, pre-feasibility and feasibility related studies and standby costs;

- The estimate excludes:
    - Exploration and development drilling post January 2012;
    - Financing related charges (e.g., fees, consultants, etc.);
    - Capitalized interest and standby fees from third party lenders;
    - Escalation;
    - Application of levies, taxes or customs duties; and
    - Permitting and community relations (assumed to be in the owners cost estimate).

Data for these estimates have been obtained from numerous sources including:

- Feasibility level engineering design;
- Mine plan;
- Topographical information obtained from site survey;
- Geotechnical investigation;
- Budgetary equipment proposals;
- Budgetary unit costs from local contractors for civil, concrete, steel and mechanical works;
- Data from recent completed similar studies and projects; and
- Information provided by Suyamarca.

The contingency of $24.9 M (7.8% of overall project CAPEX) covers changes which may arise and are not covered by any project accuracy provision. These would include exchange rate variations from the Estimate basis, escalation of field construction labor costs above the base date escalation of 1 October 2011, abnormalities in industrial relations, market conditions, weather or adverse political or regulatory developments.

Table 21.1 provides an overall summary of the capital costs for the project starting on 1 January 2012 until ramp-up is completed and full scale production commences in Q1 2014. The capital cost estimate was developed based on an EPCM project execution basis, with the EPCM labor and fees included within the indirect costs. This table does not show the IMZ capital cost component as per the JV agreements outlined in Section 4.2.

Table 21.1 - Overall Project Capital Costs by Area

Description	Total Cost ($M)
Process Plant	88.9
Process Plant Infrastructure	12.0
High Voltage Power Supply	13.5
Mine	69.6
Non Process Plant Infrastructure	36.6
Temporary Facilities	11.5
Indirects	35.1
Owners Costs	23.3

Description	Total Cost ($M)
Contingency	24.9
Grand Total	**315**

21.1.3 Mining Capital Cost Summary

Capex and operating costs (Opex) for the mine were sub-divided based on information supplied by Suyamarca and summarized in the following in Table 21.2.

Table 21.2 - Split Between Mining Capex and Opex

CAPEX	Mine Infrastructure	Ramps
		Raise Borers
		Main Levels
	Mine Preparation	By-Passes
		Raises
		Sub-Levels
OPEX	Stopes Preparation	Access to Stopes
		Drawpoints
	Vein Development	Raises
		Headings
	Mining	Stopes

The mine initial Capex summary (excluding sustaining capital cost) is summarized in Table 21.3.

Table 21.3 - Mining Capex Cost Summary

Description	Total Cost ($M)
Contract Mine Development	9.65
Contract Mine Development Support	5.16
Development Haulage	0.56
Ventilation Infrastructure	0.14
Mine De-watering Infrastructure	0.84
Underground Energy Distribution System	5.50
Energy Costs Associated with Development	0.74
Air Supply and Distribution	0.60
Paste Backfill Distribution	0.28
Mine Mobile Equipment	19.7
General (communications, software, tools etc.)	2.03
Sill Pillar Provision for Main Level	4.12

Description	Total Cost ($M)
Mine Offices and Change-rooms	0.58
Mine Sample Preparation Room	0.06
Mine Vehicle Workshop	2.81
Paste Backfill Plant	16.8
Total	**69.6**

21.1.4 Process Plant Capital Cost Summary

The process plant capital cost was derived based on the flowsheet and unit operations selected as outlined in Section 17. The costs per area are summarized in Table 21.4.

Table 21.4 - Process Plant Capital Cost Summary

Description	Total Cost ($M)
Primary Crushing and Overland Conveying	12.3
Stockpile and Reclaim	2.97
Grinding	19.8
Leaching	12.3
CCD and Solution Management	8.69
Merrill Crowe	11.3
Gold Room	2.22
Detoxification and Tailings Disposal	4.30
Reagents	4.80
Water Supply and Distribution (excluding dam cost)	2.84
Air Supply and Distribution	3.07
Plant Control System	1.24
General (fencing, lighting, diesel storage, fuel bowser)	3.10
Total	**88.9**

21.1.5 Sustaining Capital Cost Summary

Sustaining capital requirement for the production period is presented in Table 21.5 and totals $104 M over the mine life. This cost covers the development of the Angela Vein to production, ongoing tailings dam lifts, progressive re-habilitation and closure costs.

An allowance of $50,000 per annum has been made for sustaining capital associated with process plant equipment replacement.

The costs associated with closure and progressive rehabilitation were supplied by Suyamarca.

Table 21.5 - Sustaining Capital Expenditures

($M)	2014	2015	2016	2017	2018	2019	2020	Total
Mine	18.9	14.9	15.4	12.3	15.8	4.0	0.5	**81.8**
Process plant and Infrastructure	0.05	0.05	0.05	0.05	0.05	0.05	0.05	**0.35**
TSF progressive capital	-	1.5	1.5	1.5	1.5	1.5	-	**7.50**
Progressive Re-habilitation	0.3	0.09	0.09	0.09	0.07	0.15	0.5	**1.29**
Closure							13.3	**13.3**
Total	**19.25**	**16.54**	**17.04**	**13.94**	**17.42**	**5.7**	**14.35**	**104.2**

21.2 Operating Cost Summary

21.2.1 General

The operating cost (Opex) estimate was developed in United States dollars (US$) on an annual process plant throughput basis of 1,279,660 tonnes per year. The costs were divided into the key cost centers and all figures are as of the 4th quarter of 2011 (calendar year).

The operating costs presented do not include allowances for escalation or exchange rate fluctuations. The estimate is considered Feasibility Study level with an accuracy of ±15%.

21.2.2 Summary

A summary of the Inmaculada Opex is shown in Table 21.6

Table 21.6 - Unit Operating Cost Summary (based on 3,506 t/d milled)

Parameter	Unit	Value
Underground mine - Cut and Fill: - Sub Level Stoping: - Average:	$/t ore milled	 44.5 32.2 39.1
Processing	$/t ore milled	25.1
General & Administrative	$/t ore milled	8.70
TOTAL	**$/t ore milled**	**72.9**

The major common assumptions used for the cost estimation are:

- Diesel fuel price: $1.05/l
- Power cost: $0.07/kWhr
- Labor costs provided by Suyamarca are representative of similar operations in the area

- Three shifts on 14 days on followed by 7 days off rotation for all production, maintenance and technical personnel commuting from Lima or local villages.

21.3 Basis of Estimate

The mining costs estimate was developed from a number of sources. Cost determinations were based on fixed and variable components relating to yearly total amounts converted to unit cost based on the annual production and the production from each mining method. The source of data used for the Opex are summarized in Table 21.7.

Table 21.7 - Derivation of Mine Operating Cost

Cost Category	Source of Cost Data
Mine Labor	Manning schedule estimated based on benchmarking similar contract operations in the area and rates provided by Suyamarca.
Mine Development	Based on benchmarking similar contract operations in the area, equipment efficiency and rates provided by Suyamarca.
Mine Support	Based on recommendations from the geomechanical study and support standards provided by Suyamarca.
Auxiliary Services and Mine Infrastructure	Based on mine design, benchmarking similar operations, and MICSAC experience.
Mine Mobile equipment	Based on mine design, quotations and technical specifications from equipment providers, benchmarking similar operations and MICSAC and Suyamarca experience.

The plant operating and G&A costs estimate was developed from a number of sources. Cost determinations were based on fixed and variable components relating to ore throughput and plant flowsheet. The sources of data used for the Opex are summarized in Table 21.8

Table 21.8 - Derivation of Plant Operating Cost

Cost Category	Source of Cost Data
Process and G&A Labor	Manning schedule estimated based on benchmarking similar operations in the area and rates provided by Suyamarca.
Power	Consumption from the load estimate and power unit rate supplied by Suyamarca.
Reagents	Consumption rates based on testwork and benchmark from operating plants, unit prices as quoted by suppliers and compared with costs provided by Suyamarca.
Consumables	Consumption rates calculated and/or benchmarked off similar operations and Ausenco experience; unit prices as quoted by suppliers.
Maintenance Materials and Supplies	Estimated based on benchmarking similar operations and Ausenco experience.

Assay and Metallurgical Laboratory	Estimated based on industry benchmarking similar operations and Ausenco experience.
Camp	Number of persons in the camp calculated from manning schedules. Camp costs per person per day provided by Suyamarca.
G&A Incidentals	Estimated based on costs supplied by Suyamarca and estimated costs based on benchmarking similar operations and Ausenco experience.

21.4 Mine Operating Cost

Mine operating costs have been estimated for unit activities based on inputs from Suyamarca, MICSAC, cost of materials supplied from the Suyamarca database and quotations for equipment supply and services. The operating cost per period is obtained by multiplying the unit activities and production plan for the period.

Included within the operating costs are:

- Cost and labor insurance for the mine labor, according to the reference wage structure and general considerations in the market for underground mining operations;
- Variable cost of mining equipment stated as fuel, spare parts and consumables; and
- Personnel cost for support areas such as geology, safety and planning inclusive of overheads.

The overall mining costs associated with both cut and fill and sub level stoping are summarized in Table 21.9 and Table 21.10 respectively.

Table 21.9 - Unit Cost for Cut and Fill Mining (based on 3,506 t/d ore mined)

Item	Unit Cost
Mine Development	2.49
Ground Support for Mine Development	2.13
Mining Operations	11.11
Ground Support for Mine Operations	13.42
Hauling from Ore Pass to Mine Entrance	2.45
Paste Backfill	7.07
Power Supply Maintenance	0.27
Power Consumption	1.42
Ventilation	0.64
Dewatering	0.08
Labor Costs	3.39
Total Unit Cost - Cut and Fill	**44.47**

Table 21.10 - Unit Cost for Sub Level Stoping Mining (based on 3,506 t/d ore mined)

Item	Unit Cost
Mine Development	6.01
Ground Support for Mine Development	5.64
Mining Operations	5.28
Ground Support for Mine Operations	-
Hauling from Ore Pass to Mine Exit	2.35
Paste Backfill	7.07
Power Supply Maintenance	0.27
Power Consumption	1.42
Ventilation	0.64
Dewatering	0.08
Labor Costs	3.39
Total Unit Cost - Sub Level Stoping	**32.15**

21.5 Processing Cost

These operating costs are based on the process flowsheet as described in Section 17. The battery limits for the determination of the process operating costs commence from the crushing facilities and continue through to tailings discharge into the TSF.

Operating costs were sub-divided into plant and general and administrative (G&A). Plant costs include labor, power, reagents and consumables, maintenance spares and consumables, vehicles and mobile equipment, metallurgical testing, and cover plant areas from ore reclaim through to the goldroom, including plant services. G&A costs include labor, light vehicle running costs, camp, off-site office, and miscellaneous costs.

Reagent and consumable consumption rates were estimated based on the metallurgical testwork completed, and also benchmarked with plants that operate with similar ores.

Power consumptions for plant areas and G&A buildings were calculated from the electrical load list and assumed to remain constant over the life of operation.

Annual operating costs for maintenance spares and consumables were estimated at 6% of the capital cost estimate for process plant.

Included in the process plant operating cost estimate are:

- Labor for supervision, management and reporting of onsite organizational and technical activities directly associated with the processing plant;
- Labor for operating and maintaining plant mobile equipment and light vehicles, process plant and supporting infrastructure;
- Direct operation of the processing plant, including all fuels, reagents, consumables and maintenance materials;
- Fuels, lubricants, tires and maintenance materials used in operating and maintaining the plant mobile equipment and light vehicles;

- Operation of the TSF, including tailings discharge and management and return water, excluding construction and on-going dam raises;
- Power supplied to the process plant from the power facility, inclusive of labor, fuel, lubricants and maintenance supplies;
- Operation of raw water supply facility from the raw water dam and wells; and
- Labor and operational costs for the metallurgical and assay laboratories.

Table 21.11 is a summary of process operating cost per tonne of ore treated.

Table 21.11 – Process Operating Cost Estimate (Based on 3,506 t/d ore milled)

Category of Cost	Unit	Value
Process Operating Cost		
Process Labor	$/t ore milled	1.77
Power for Process	$/t ore milled	4.15
Reagents & Consumables	$/t ore milled	17.8
Maintenance Materials and Supply	$/t ore milled	1.41
Total Process Cost	**$/t ore milled**	**25.1**

21.6 General and Administrative (G&A) Cost

The overall project G&A costs are summarized in Table 21.12.

Table 21.12 - G&A Cost Estimate (based on 3,506 t/d ore milled)

Category of Cost	Unit	Value
G&A	$/t ore milled	5.46
G&A Labor	$/t ore milled	1.16
Permanent Camp	$/t ore milled	2.08
Total G&A	**$/t ore milled**	**8.70**

Below is a summary of the main cost items included in the G&A operating cost and the basis of the estimate:

- $1.49 M/y for G&A labor – manning schedule and rates based on benchmarking similar plants operating in the region;
- $0.17 M/y for communications including satellite phone communications and internet - benchmarking similar plants operating in the region;
- $0.79 M/y for insurances to cover general liability, risk, and vehicle insurance policies – advised by Suyamarca;
- $1.00 M/y for community project and relations – advised by Suyamarca;
- $0.23 M/y for environmental licenses and monitoring including costs associated with quality sampling and monitoring, analysis of surface and ground water, as well as surface flow measurement – as advised by Suyamarca;
- $2.24 M/y for G&A vehicles including fuel, consumables and maintenance – based on the G&A vehicle list and estimated run time hours, hourly maintenance cost and hourly fuel consumption;
- $0.46 M/y for G&A power costs associated with the camp, offices and water treatment plants; and
- $0.50 M/y for small equipment hire costs.

The camp unit cost was based on similar operations in the region. These camp costs include camp lodging and catering personnel costs. Labor rosters were used to determine occupancy at the camp.

All other G&A costs were estimated based on benchmarking similar plants.

22 Economic Analysis

Two economic analysis cases have been provided in this report. Firstly the “overall project economic analysis” shows the parameters of the overall project. Secondly the “IMZ specific analysis” provides the economic parameters of the project specific to IMZ based on the IMZ – Hochschild terms of agreement as outlined in Section 4.2.

22.1 Overall Project Economic Analysis

Information and calculation methods regarding taxation and duties participation were provided by Suyamarca and are believed by Ausenco to be correct at the time of this report. The taxes calculated in the FS cash flow are simplified and intended to provide only an indication of potential taxation. The actual tax that might be paid by Suyamarca is a complex subject and dependent upon factors that are beyond the scope of this report.

The following assumptions were used in this economic analysis and are based on the information provided by Suyamarca conforming to Peruvian Law and International Financial Reporting Standards (IFRS). Ausenco has not audited this information, but believes it to be correct.

22.1.1 Assumptions

- The gold and silver prices used for developing the ore reserve estimations and completing the overall project base case economic analysis as outlined in Table 22.2 were $1,100 and $18 per troy ounce for gold and silver respectively;
- 99.85% payable gold and 99.75% payable silver. $0.17 per total metal ounce refinery treatment charge, $0.75 per payable gold ounce refinery cost, and $13.31 per kilogram of doré transportation and insurance cost;
- Financial profit and loss (P&L) depreciation (based on the unit of production method) to calculate the net operating income and the applicable royalty and special mining tax (IEM);
- Royalties and IEM are calculated as per the legislative progressive scales, based on the percentage of pre-tax operating profit;
- For tax purposes, total mine development, closure and rehabilitation costs are depreciated in the same year of expenditure. Other capital expenditures use a 15% straight-line depreciation rate;
- 8% profit sharing tax applied to the net profit prior to income taxes;
- 30% corporate income tax applied to the final net profit;
- There are no applicable exportation taxes to be applied;
- Value added tax (or IGV) is excluded from the cashflow analysis. This tax will be paid and then recovered when the doré is exported. The calculation of the IGV paid and recovered during the life of mine is not expected to have a significant impact on the overall cashflow;
- Analysis begins in year 2012 (labeled as Year -2), therefore all costs prior to and including the feasibility study are assumed to be sunk costs;
- 5% discount rate for net present value calculation as per mineral industry practice
- The cash flow analysis uses “constant US dollars”, which means that no inflation of prices or costs was applied. This is consistent with mineral industry practice;

- The analysis has a base date of Q4 2011; and
- The analysis was performed based on 100% equity financing. No debt financing was considered.

The cash flow analyses all used the same LOM production schedule that were compiled using $1,100/oz gold and $18/oz silver reserves.

22.1.2 Methodology

Operating and capital costs were compiled by each responsible QP and submitted as summaries for input into the economic analysis spreadsheet. The flow chart for the overall analysis calculation is depicted in Table 22.1.

Table 22.1 - Economic Analysis Calculation

Calculations
Tonnes and grade delivered to the processing facility
x
Process recovery
=
Recovered gold/silver ounces
x
Percentage smelter payables
x
Gold/Silver price
-
Off-site costs (refining, transportation, etc.)
=
Revenue
-
Operating costs (mine, processing, G&A)
=
Sub-total pre-tax operating income
-
Financial P&L depreciation allow ance*
=
Sub-total P&L operating income for calculating royalties and duties
-
Royalties*
-
IEM mining tax*
+
Financial P&L depreciation allow ance*
=
Sub-total operating income for calculating taxes
-
Tax accounting depreciation allow ance*
-
Peruvian profit sharing tax*
=
Net taxable income
-
Corporate income tax*
+
Tax accounting depreciation allow ance*
=
Net operating cashflow after taxes
-
Capital expenditures
=
Net annual cash flow

*Excluded in Pre-tax calculation

The economic analysis is shown in Table 22.2.

Table 22.2 - Overall Project Economic Analysis (Base Case)

			Unit	Total	2012	2013	2014	2015	2016	2017	2018	2019	2020
Production	-												
Ore mined			Mt/a	7.8			0.617	1.260	1.261	1.258	1.258	1.255	0.892
Gold grade			g/t	3.37			3.81	3.41	3.01	2.65	3.03	3.64	4.60
Silver grade			g/t	120.2			98.0	121.0	147.2	130.9	109.4	96.8	128.9
Gold recovery			%	95.6			95.6	95.6	95.6	95.6	95.6	95.6	95.6
Silver Recovery			%	90.6			90.60	90.60	90.60	90.60	90.60	90.60	90.60
Revenue	-												
Gross Gold Value	Gold Payable	**99.85%**	$	860,148,765			77,050,404	140,586,283	124,390,440	109,179,935	124,934,017	149,567,667	134,440,019
Gross Silver Value	Silver Payable	**99.75%**	$	475,587,222			30,693,850	77,395,810	94,180,482	83,569,901	69,835,453	61,594,521	58,317,205
Gross Metal Value			$				107,744,254	217,982,092	218,570,922	192,749,836	194,769,470	211,162,188	192,757,225
Less realization costs													
	Refinery Treatment Cost		$	4,636,046			302,539	754,552	910,964	808,148	680,547	606,334	572,962
	Refining Cost		$	586,465			52,534	95,854	84,812	74,441	85,182	101,978	91,664
	Dore Transportation and Insurance		$	11,290,267			736,779	1,837,577	2,218,492	1,968,101	1,657,351	1,476,618	1,395,348
Net Metal Value			$	1,319,223,209			106,652,402	215,294,109	215,356,654	189,899,146	192,346,389	208,977,258	190,697,251
Operating Cost													
Mining			$	304,795,043			30,514,480	47,368,115	47,378,334	47,315,624	47,303,526	47,213,462	37,701,502
G&A			$	77,901,582			11,128,797	11,128,797	11,128,797	11,128,797	11,128,797	11,128,797	11,128,797
Plant			$	197,376,139			16,594,137	31,604,000	31,613,102	31,557,251	31,546,477	31,466,266	22,994,905
Operating Revenue (Pre-Royalties and IEM)													
Operating Cash Flow			$				48,414,988	125,193,196	125,236,421	99,897,473	102,367,589	119,168,733	118,872,046
Financial Depreciation (P&L)			$				-61,850,251	-56,347,594	-57,183,667	-58,247,562	-60,395,914	-63,788,024	-58,055,895
Operating Income (P&L)			$				-13,435,264	68,845,601	68,052,754	41,649,911	41,971,675	55,380,709	60,816,151
Duty and Royalties													
Operating Income Percentage to calculate IEM and Royalties													
Royalty			$	13,357,360			1,077,443	2,179,821	2,185,709	1,927,498	1,947,695	2,111,622	1,927,572
IEM Mining Tax			$	8,772,147			0	1,881,896	1,847,826	985,847	992,509	1,402,451	1,661,617
Pre-tax Operating income (P&L)	Pre-tax (after duties and royalties)		$	301,152,031			-14,512,706	64,783,885	64,019,219	38,736,565	39,031,472	51,866,635	57,226,961
Pre-tax Operating Cash Flow	Pre-tax (after duties and royalties)		$	717,020,939			47,337,545	121,131,479	121,202,885	96,984,128	99,427,386	115,654,659	115,282,857
				35.6%									
Depreciation and Amortization				6.8%									
				42.4%									
Financial Depreciation			$				61,850,251	56,347,594	57,183,667	58,247,562	60,395,914	63,788,024	58,055,895
Tax Depreciation			$	-415,868,908	-22,500,000	-77,389,756	-58,739,717	-54,698,512	-49,809,095	-53,495,895	-44,256,637	-27,050,452	-27,928,845
Tax loss Carryforward			$			-22,500,000	-99,889,756	-111,291,928	-44,858,961	0	0	0	0
Taxable Income			$	-114,716,878	-22,500,000	-99,889,756	-111,291,928	-44,858,961	26,534,830	43,488,232	55,170,749	88,604,207	87,354,012
Taxes													
	Peruvian Profit Sharing tax	**8.0%**	$	24,092,162			0	0	2,122,786	3,479,059	4,413,660	7,088,337	6,988,321
	Peruvian Income Tax	**30.0%**	$	83,117,960			0	0	7,323,613	12,002,752	15,227,127	24,454,761	24,109,707
Operating Revenue (After Tax)													

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			Unit	Total	2012	2013	2014	2015	2016	2017	2018	2019	2020
Operating Cash Flow	After-tax		$	609,810,816			47,337,545	121,131,479	111,756,486	81,502,317	79,786,599	84,111,562	84,184,829
Capital Cost													
Total Capital Cost			$		82,000,000	233,358,052	19,253,283	16,495,468	17,069,367	13,932,864	15,592,101	5,710,823	12,456,952
Pre-Tax Cashflow Analysis													
Project Cash Flow - Pretax, 100% Equity			$		-82,000,000	-233,358,052	29,161,705	108,697,727	108,167,054	85,964,610	86,775,489	113,457,910	106,415,095
Cumulative Project Cash Flow - Pretax, 100% Equity			$		-82,000,000	-315,358,052	-286,196,347	-177,498,619	-69,331,566	16,633,044	103,408,533	216,866,442	323,281,537
NPV - Pretax, 100% Equity @ 5% discount rate			$		-78,095,238	-289,757,870	-264,566,893	-175,141,004	-90,389,287	-26,241,171	35,428,548	112,221,328	180,817,447
Post-Tax Cashflow Analysis													
Project Cash Flow - After tax, 100% Equity			$		-82,000,000	-233,358,052	28,084,262	104,636,011	94,687,119	67,569,453	64,194,499	78,400,739	71,727,877
Cumulative Project Cash Flow - After tax, 100% Equity			$		-82,000,000	-315,358,052	-287,273,789	-182,637,779	-87,950,660	-20,381,207	43,313,292	122,214,031	193,941,908
NPV - After tax, 100% Equity @ 5% discount rate			$		-78,095,238	-289,757,870	-265,497,628	-179,413,323	-105,223,488	-54,802,122	-9,180,290	43,884,416	90,120,845

22.1.3 Sensitivity Analysis

Sensitivity analyses were done on the base case by individually modifying the capital cost, operating cost, metal price and mill feed grade up and down by 10% to show the sensitivity of the pre-tax net present value using a 5% discount rate ($NPV_{5\%}$). The results of the sensitivity analyses show that the project is most sensitive to gold price and mill feed grade. A 10% decrease in gold price or metal grade leads to a 55% decrease in pre-tax $NPV_{5\%}$ from $181M to $82M. The reverse occurs if the metal price or mill feed grade increases by 10%, the pre-tax $NPV_{5\%}$ increases from $181M to $280M. Table 22.3 shows the results of the sensitivity analyses.

Operating costs are the next most sensitive parameter. A 10% increase in operating costs reduces the pre-tax $NPV_{5\%}$ by $44M and a 10% decrease in operating cost increases $NPV_{5\%}$ by $43M. A 10% change in operating costs results in a 24% change in pre-tax $NPV_{5\%}$.

The project economics are also sensitive to capital cost. A 10% change in capital costs results in an approximately 20% change in pre-tax $NPV_{5\%}$. A summary of the sensitivity analysis is shown in Table 22.3.

Table 22.3 - Overall Project Economic Sensitivity Summary

Case	Variable	Pre-tax NPV5% (M$)		
		-10% Variance	0% Variance	+10% Variance
Case 1 (base case)	Capital Cost	217	181	144
	Operating Cost	224	181	137
	Metal Price or Grade	82	181	280

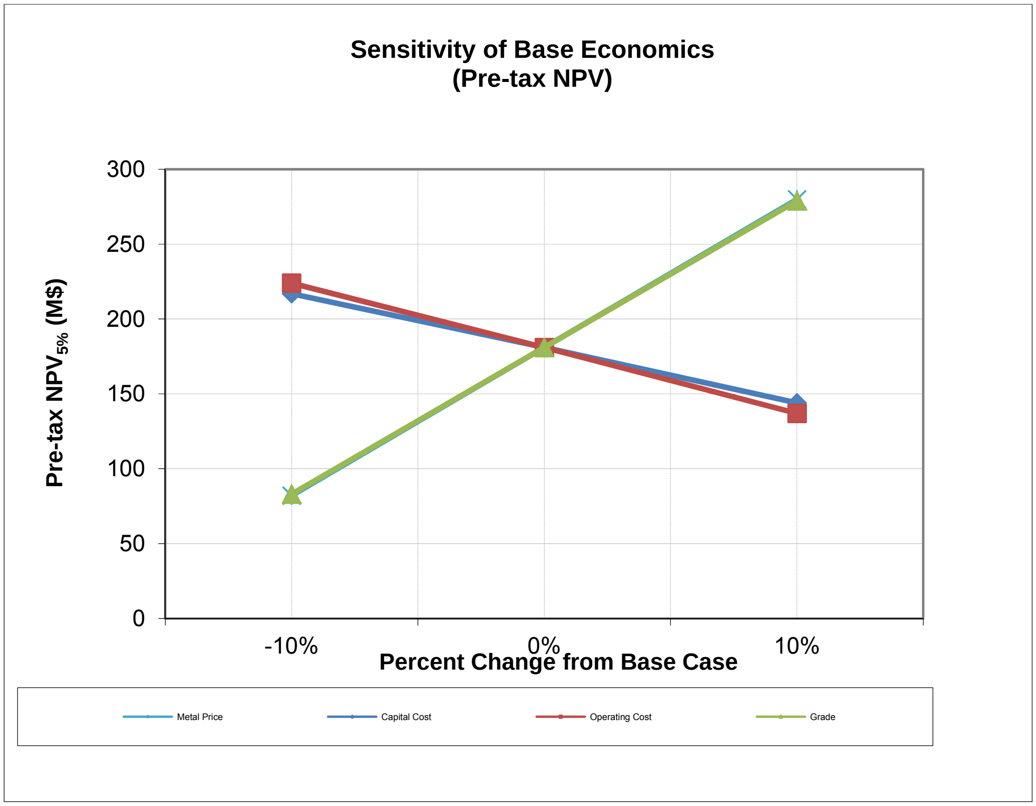

Figure 22.1 - Overall Project Sensitivity Analysis – Base Case

22.1.4 Payback

The overall project pre-tax payback period is 4.3 years with an internal rate of return (IRR) of 18%. The overall project post-tax payback period is 5.1 years with an IRR of 12%. The payback estimates do not include sunk costs spent prior to 2012 or on-going exploration costs not directly linked with the Feasibility Study.

22.1.5 Mine Life

The life of mine plan encompasses approximately nine years and is made up of two pre-production years (2012 to 2013 inclusive) and seven years of commercial production.

22.2 IMZ Specific Economic Analysis

This case outlines the project economic parameters that are attributable to IMZ based on the IMZ – Hochschild terms of agreement for the joint venture as outlined in Section 4.2.

22.2.1 Assumptions

- As per the assumptions outlined in Section 22.1.1;
- The principal terms of the framework agreement between IMZ and Hochschild as supplied by IMZ and outlined in Section 4.2 are accurate; and

- IMZ advised Ausenco that to date Hochschild has spent approximately $11.5 million on Feasibility Study and project development costs. This leaves a further $88.5 million to be expended by Hochschild before IMZ is required to start contributing its 40% share of the capital required for the construction of Inmaculada project.

22.2.2 Results

Table 22.4 below shows IMZ's attributable production and economic parameters for Inmaculada based on the terms of the joint venture agreement.

Table 22.4 - IMZ's Attributable Production and Economic Parameters

Item	Units	
Average annual gold production	ounces/year	49,600
Average annual silver production	ounces/year	1,682,000
Average annual gold equiv. production[2]	ounces/year	78,000
Life-of-mine gold production	ounces	313,000
Life-of-mine silver production	ounces	10,600,000
Life-of-mine gold equiv. production[2]	ounces	488,000
Initial capital	$ millions	91
Direct site costs[1]	$ per tonne processed	74
Direct site costs[1,3]	$ per ounce Au (with Ag credit)	133
Total cash operating costs[1,3,4]	$ per ounce Au (with Ag credit)	262
IRR Pre-tax/post-tax	%	26 / 21
Pre-tax /post-tax cash flow (non-discounted)	$ millions	136 / 95
Pre-tax/post-tax NPV, 5% discount rate	$ millions	85 / 57
Pre-tax/post-tax NPV, 8% discount rate	$ millions	63 / 40

1) Direct site costs include mining, processing and mine administration. Total cash operating costs include direct site costs plus estimates of the management fee, refining charges and government royalty (but do not include workers profit sharing which is 8% of net income).
2) Gold equivalents are estimated using a silver-to-gold ratio of 60:1 calculated by using the ratio of the base case metal prices.
3) By-product accounting subtracts the revenue generated by silver from the total operating costs to determine the cost per ounce of gold.
4) For comparative purposes, if IMZ had selected co-product accounting, the resulting cash operating costs would be $560/oz of gold and $9.15/oz of silver.

22.2.3 Sensitivity Analysis

Sensitivities to gold and silver prices based on IMZ's attributable production and economic parameters are shown in Table 22.5 below.

Table 22.5 - Pre-tax Sensitivity Analyses Attributable to IMZ (base case in bold)

	Gold Price/Silver Price ($/oz)					
Category	$900/ $15.00	**$1,100/ $18.00**	$1,300/ $21.00	$1,500/ $25.00	$1,700/ $28.00	$1,900/ $31.00
IRR	10%	**26%**	41%	55%	67%	79%
Cash Flow ($ millions)	$46	**$140**	$234	$339	$433	$528
NPV 5% ($ millions)	$17	**$87**	$158	$236	$306	$376
NPV 8% ($ millions)	$6	**$65**	$124	$190	$249	$309

23 Adjacent Properties

The Inmaculada project is located approximately 40 km south east of the Pallancata underground mine, also owned by IMZ (40%) and Hochschild (60% and operator). The Pallancata mine is a silver/gold mine operating at 3,000 t/d and in 2011 produced 8.8 million ounces of silver and 33,881 ounces of gold and is owned by the same Peruvian company, Suyamarca.

24 Other Relevant Data and Information

24.1 Mine Geomechanical Overview

Recommendations for design and support of underground excavations were based on geomechanical evaluation of the mineralized zone and the hangingwall (HW) and footwall (FW) as part of the FS. The evaluation included a review of information from previous exploration drilling, geological mapping, and orientated drill core drilled specifically for geomechanical purposes.

The field activities included:

- Geomechanical re-logging for 185 selected core drill holes. These focused on the vein and zones adjacent to the vein in the HW and FW;
- Surface mapping using cell and detail lines in 51 stations;
- Permeability measurements and piezometer installation; and
- Drilling and logging 6 orientated core holes.

Table 24.1 - Geomechanical Core Holes

Drillhole	Coordinates (m)		Elevation (m)	Azimuth (°)	Inclination (°)	Depth (m)
	North	East				
AUS_INM-01	8 347 275,4	690 035,4	4 692,1	311,1	-68,0	395,0
AUS_INM-02	8 347 148,9	689 600,0	4 694,7	140,0	-80,1	540,8
AUS_INM-03	8 346 744,0	689 414,0	4 657,0	304,4	-68,9	375,0
AUS_INM-04	8 346 781,6	689 138,6	4 688,2	124,2	-75,8	390,1
AUS_INM-05	8 346 486,7	688 954,0	4 658,3	308,5	-68,0	280,0
AUS_INM-06	8 346 299,2	688 651,6	4 583,9	139,1	-79,6	220,0

The testwork and analysis applied to the samples and information collected in the field included:

- Measurement of intact rock hardness through Schmidt hammer testing;
- Laboratory tests, which included: point load index, unconfined compression, physical properties, elastic properties, triaxial compression, direct shear on fractures and Brazilian traction test. These were applied over representative samples from the orientated drill holes;
- Geomechanical evaluation including the development of a 3D model of the rock mass rating (RMR) along the Angela Vein, based on exploration and orientated core holes;
- Stereographic analysis, based on the structural information from orientated drill holes; and
- Finite-element stress-strain analysis of proposed excavations, based on laboratory tests and field logging information.

The evaluation of the stability for the rock excavations was developed using empirical and deterministic methodologies including; maximum dimension for excavations and self-support time based on the RMR of the HW and FW and the vein; the stability graph method to state the hydraulic ratio; and the 2D finite element stress-strain analysis for the excavations using Phase 2 software.

According to the mean RMR obtained from the geomechanical model for the HW and FW, the maximum unsupported excavation is 4 m with a self-support time of 2 days, while for the vein the expectation is 2.7 m of unsupported excavation for 7.2 hours. These conditions require immediate support in the vein after blasting activities.

The basic RMR (without correction for oriented structures) was modeled in 3D along the vein showing an average value of 42 in the HW and 43 in the FW, while in the vein zone the average is 33. The model indicates that the predominant rock quality is Poor (IVA - RMR between 31 and 40) to Fair (IIIB - RMR between 41 and 60). The model also indicates a trend towards improved rock quality in the central zone.

The finite-element stress-strain analysis for the transversal sub level stoping showed that a pillar between sub-levels could be 4 - 5 m high; however it could be optimized with more detailed analysis to 7 m, assuming an average vein width of 6.5 m. Finite element stress-strain analysis shows that there are no expected problems with stability for the cut and fill mining methods utilizing an stope height of 5 m.

It was recommended that sub level stoping mining be applied in the central part of the vein (where the vein is wider) with a stope height of 5 m, and cut and fill mining be applied in the north and south parts of the vein with cut heights of 5 m.

24.2 Project Implementation Schedule

Project construction is expected to take 15 months due to climatic conditions (seasonal snow and ice) and the reduced productivity at 4,800 masl.

The following critical key dates and milestones have been identified for the project. They are subject to receipt of environmental approvals, financing, award of contracts and meteorological conditions. Delays in meeting these target dates would likely result in a delay in project completion. It should be noted that at the time of this report basic engineering on the process plant was essentially completed, with recommendations made for purchase of all long lead items.

Commence placement of orders for long lead items………………………………...February 2012

Commencement civil works…………………………………………………………………….May 2012

Complete detailed engineering………………………………………………………..December 2012

Commence mechanical installation..…………………………………………………...March 2013

Complete civil works……………………………………………………………………….…May 2013

Commence commissioning……………………………………………………………November 2013

Production………………………………………………………………………………December 2013

Procurement of the following items has been identified as critical to meet the proposed schedule, either due to long lead or the requirement for vendor data:

- Paste backfill plant and filters – long lead and vendor data;
- Merrill Crowe equipment and filters - long lead and vendor data;

- SAG and ball mills – long lead and vendor data;
- Thickeners and clarifier – long lead and vendor data;
- Crushing equipment - vendor data;
- Leach tank agitators - vendor data;
- Mill liner handler - vendor data;
- Cyclone cluster;
- Samplers; and
- Oxygen plant.

25 Interpretation and Conclusions

The Inmaculada project was a fast tracked feasibility study, progressing directly from scoping study to FS. As a result there are a number of areas identified for optimization and further definition to be completed during subsequent phases of the Project. This section outlines the key interpretations, risks and opportunities identified during the course of the FS.

25.1 Geology and Resources

The Inmaculada deposit is extensively drilled and well understood, geologically. The data set is robust and reliable as evidenced by examples of the QAQC data. The approach to interpretation has been conservative and realistic. The dilutant grades applied to the mining reserve are conservative.

Opportunities

- The location and definition of additional veins for mining in both the FW and HW of the Angela Vein.

Risks

- Small scale offsets of the veins may make mining more difficult than planned. Aggressive level development and careful mapping and sampling will mitigate this risk, to some extent.

25.2 Geomechanical

The geomechanical results indicated poor rock conditions in areas of the Angela Vein. As a result a conservative approach was taken to the mining method selected, as well as the stope size and sequencing. The sub level stoping mining method was applied in the central part of the vein (larger vein width) with a stope height of 5 m. The cut and fill method was applied in the north and south areas of the vein with cut heights of less than 5 m.

The maximum excavation and auto-support time analysis of the HW and FW showed that it is possible to have unsupported excavations up to 4 m with an auto-support time of 2 days. For the Angela Vein the expectation was 2.7 m of unsupported excavation for 7.2 hours. This last condition requires immediate support in the vein zones after blasting activities.

25.3 Mining

Mineral reserves for Inmaculada were estimated according to industry standards, adding dilution and applying ore losses, and converting measured and indicated resources into mineral reserves.

The underground mining methods applied to the Angela Vein are consistent with industry standards. In addition to utilizing good mining practices, a systematic program of sampling, mapping, laboratory analyses, and reporting will be required to minimize dilution and ore losses.

Opportunities

- Improve the ramp-up during the first year, increasing the production in the lower zones of the mine that generally contain higher gold and silver grades;
- Optimize the mine plan and schedule to maximize the plant throughput at the plant design feed rate of 3,506 t/d;

- Evaluate possibilities to increase the stope height for the sub level stoping areas within the current mine design based on further geomechanical work;
- Evaluate the grade of the HW and FW material during operations using short drill holes to improve the grade control;
- Complete more detailed testwork on the expected paste backfill strength achieved using various ratios of cement to allow optimization of the paste backfill cement consumption and operating costs; and
- Investigate opportunities to place as much waste rock as possible in the mined out areas, thereby reducing the requirement for paste backfill and reducing the operating costs.

Risks

- There is limited geomechanical and hydrogeological information of the zones that will be intercepted when developing the main access tunnels to the Angela Vein. It is possible that lower rock quality will be encountered than assumed, and the support cost for these developments could be higher;
- The vein geomechanical model was developed with drill holes spaced at between 20 to 50 m. It is possible that zones between these holes have different geomechanical conditions than that interpreted, and these could alter the mining method selected for various zones. There is flexibility in the mine design to switch between the mining methods; and
- The stereographic analysis showed that the main joint sets are sub-parallel to the orientation of the vein and have a high dip angle. In some zones the rock mass may be altered by these structures and produce ground support problems.

25.4 Process Plant

Outcomes of the testwork conclude that processing the Angela Vein ore types can be successfully achieved using industry standard process techniques.

The following optimum leach conditions were selected for Inmaculada based on the testwork results, as well as trade-off economic evaluations:

- primary grind size P_{80} of 50 µm
- pH 11.0 using lime
- cyanide concentration 0.15% (1,500 ppm)
- 96-hour leach
- 40% w/w solids
- oxygen sparging.

Mercury abatement by retort oven is required when processing ores especially from Zone 3.

25.5 Tailings Storage Facility

The following interpretations were made regarding the tailings storage facility:

- The proposed TSF design provides sufficient capacity to store the tailings produced during the life of mine (7.8 Mt);
- Based on the results of the deposition planning and water balance assessment, sequenced construction of the facility is possible; and

- It is possible to reduce the size of the TSF based on the amount of tailings used for paste backfill to the underground mine.

26 Recommendations

The investigations and analyses carried out are considered appropriate to feasibility level mine design. Risks and opportunities associated with the project are outlined in Section 25, with the key recommendations covered in this Section.

Based on the findings of the FS it is recommended that:

- The project advance to basic engineering followed by detailed engineering, procurement and construct.

26.1 Geology and Resources

To confirm the potential expansion of the current mineral resource base it is recommended that:

- Level development of the Angela Vein commence;
- Detailed mapping and sampling of level development and construction of grade control models be undertaken; and
- Closer spaced drilling be carried out from a FW position to determine if development of any parallel veins is warranted.

The only previous study of the density of the Inmaculada ore showed a tendency for high-grade material to have a slightly lower density. Furthermore, this study was limited to the main Angela Vein. It is recommended, therefore, that additional density measurements be done to establish whether or not there is a density-grade relationship, and to establish appropriate densities to be used in the splays, branches and minor veins.

26.2 Geomechanical & Mining

The geomechanical drilling results showed a sufficient level of variability in rock conditions in both the HW and FW to justify further and more detailed drilling campaigns. These oriented geomechanical drilling campaigns would provide further information to ensure the mining method selected and the key stope parameters and dilution estimates are further defined.

- It will be important to continually update the engineering geology model as more information becomes available from the exploration holes. In addition, engineering geological mapping should be carried out during construction of access tunnels and mine development;
- The engineering geology models should include the expected occurrence and location of any further major geological structures identified (e.g., faults and major dykes) in order to anticipate their potential intersections with the ramps, mine development tunnels and stopes to anticipate any potential formation. As much as feasible, the mine planning should take those locations into account and, eventually, locally change the orientation of these openings in relation to these structures or improve the support specifications;
- The magnitude and orientation of the in-situ stresses are not known, and will affect the analyses in stope stability assessments. Further investigation of in-situ stresses is recommended;
- It is recommended that additional laboratory testing be conducted to obtain more reliable data and improve the input data for the geomechanical model;

- The simple mining sequence used in 3D stress modeling does not simulate the planned mining sequence in any detail. It is considered adequate for investigating the rock mass behavior around the hangingwall development and ramps, but should be reviewed in the next stage of the mine planning. Further confirmatory 3D stress modeling should be performed once final stope dimensions and mining sequences are defined;
- The design stope dimensions should be regarded as a first step in the design process and local adjustments to the design should be expected, depending upon actual conditions observed in the stope;
- Further refined numerical modeling of the stope sequence can determine the requirements for dimensions of sill (or rib) pillars, as well as optimize the extraction from a geomechanical perspective to minimize induced stresses;
- Once backfill properties are known, numerical geomechanical modeling should be conducted to assess the support effectiveness of the fill, as well as to assess if sill or rib pillars should be left for overall mine stability;
- The completion of further and more detailed hydrogeology modeling is required for input into the geomechanical modeling to refine the water drainage and handling systems; and
- An ore stockpile should be created to assist with mine ramp-up and provide additional flexibility to the mine operations.

26.3 Plant Site Geotechnical

Based on the geological reconnaissance and the results of the investigations carried out during the FS, a more specific plant site investigation program and in-situ/laboratory testing program is recommended for detailed design of the structures foundations. This would include drilling to provide more detailed information regarding the different rock lithologies, rock mechanics parameters and geotechnical quality; and specifically to determine if highly weathered or weak rock occurs at the location of the heavy equipment foundations.

26.4 Metallurgical and Processing

Due to the nature of a fast-tracked study, certain metallurgical information was not available during the FS, and as such Ausenco recommends that the process plant design for the FS be optimized. This optimization includes consideration of:

- Latest rheology testwork – to confirm thickener, agitator and pumping requirements;
- Latest oxygen uptake testwork – to confirm the pressure swing adsorption (PSA) plant oxygen requirements;
- Latest filtration testwork – to confirm the paste backfill filter sizing;
- Investigate options to use a pre-leach thickener to increase the leach feed density from 40% solids to 50% solids. This will provide more operating flexibility in the hydro-cyclone classification circuit to ensure the correct feed size to leaching is achieved, and will also allow the leach tank sizes to be reduced. This will also reduce the size of the Merrill Crowe circuit due to the decreased barren wash solution volumes required; and
- Optimize the layout of the plant to reduce the overall civil works required for construction.

26.5 Infrastructure

During the feasibility study a logistics study was completed that included evaluation of the main site access road. This identified several areas where the road needs upgrading. It is recommended that this be further defined through geotechnical investigations (test pits). There were also three drainage crossings identified. Detailed engineering is required to better define structures required and their associated costs.

The hydrogeological investigations completed near the end of the FS indicated sufficiently high inflows of ground water into the mine during both the initial development and production phases to sustain the plant raw water make-up requirements. The quality of this water needs to be better determined through further modeling to determine the suitability for plant raw water requirements. It is also likely that excess mine inflow water will need to be discharged to the environment. Further investigations into this water quality are required to define any water treatment that may be required prior to discharge.

26.6 Environmental and Social

Ausenco recommends the following environmental and social items be addressed in follow-up work programs.

- Even though it is considered that the impact on air quality will be low, air modeling is recommended in order to show the dispersion of particulate matter in the project area;
- A water depth study should be made on the wetlands that will be impacted to be able to propose a specific management plan. It should be determined whether they are fed by aquifers or if they are influenced by surface or seepage waters;
- The environmental monitoring plan should include the use of biologic indicators for the aquatic biota, using benthos organisms as key groups;
- Control points should be included in the environmental monitoring plan for physical, chemical, and biological components;
- The biological monitoring plan should include an assessment of flying mammals (bats);
- The re-vegetation plan methodology should include a success indicator for re-vegetation, which is not detailed in the current plan;
- To reduce the amount of flora damaged from stripping and soil removal, the construction activities should be planned to minimize the areas subjected to earthworks and to prepare areas where components will be located. In addition, rescue activities should be implemented, to the greatest extent possible and as necessary, to protect threatened and/or endemic flora species;
- Re-vegetation is recommended on areas affected by the location of project components after they go through physical and chemical stabilization (if necessary) and disposal of organic soils;
- Threatened and/or endemic fauna species within the component footprint should be rescued to the greatest extent possible;
- Mine employees should be prohibited from hunting, capturing, and extracting fauna and flora species;
- Provisional bridges should be installed on access roads that inevitably cross body waters, to maintain their water supply and minimize impacts to the aquatic biota;
- The development of the community relations plan must continue; the social programs should incorporate active participation of local authorities and community leaders as stakeholders. This participation should be recorded using quantifiable indicators (attendance, number of trained people, number of workshops performed, etc.) which will feed the social monitoring system of the project;
- Social actor mapping should be utilized to manage the engagement of all stakeholders. All interactions should be recorded as activities within the communication plan;

- The communication plan should include a claim and complaint procedure, which should be prepared and disseminated amongst the population in a didactic manner; and
- An archaeological monitoring plan must be developed prior to the start of the construction stage. This is critical to avoid any damages to the project site's previously identified archaeological heritage.

26.7 Risk Assessment

It is recommended that a risk assessment be carried out during the next phase. This assessment should include, but not be limited to, the following items and issues:

- Commodity prices;
- Timing of construction permits being available;
- Mining methods selected and productivities;
- Exchange rates;
- Taxation;
- Logistics;
- Manpower;
- Sociopolitical considerations;
- Geotechnical/geomechanical considerations; and
- Environmental considerations.

26.8 Budget

The budget for the next two years (subject to successful and timely permitting) to advance the project through to operations and production is shown below:

- Basic and detailed engineering, procurement and construction management (EPCM) approximately $29 M;
- Owners costs of approximately $13 M; and
- Project construction approximately $273 M.

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28 Date and Signature Page

Effective Date 11 January 2012

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29 Certificates of Qualified Persons