PAN AMERICAN SILVER CORP
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SECURITIES AND EXCHANGE COMMISSION

Washington, D.C. 20549

FORM 6-K

Report of Foreign Private Issuer

Pursuant to Rule 13a-16 or 15d-16 of

the Securities Exchange Act of 1934

For the month of, September 2011

Commission File Number 000-13727

Pan American Silver Corp

(Translation of registrant s name into English)

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(Address of principal executive offices)

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Form 20-F o Form 40-F x

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Yes o No x

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DOCUMENTS INCLUDED AS PART OF THIS REPORT

Document	
1	Pan American Silver Corp. and Orko Silver Corp., La Preciosa Silver Property, Durango, México, Preliminary Economic Assessment Technical Report.

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1 Summary

This technical report has been prepared to disclose relevant information about the La Preciosa silver property (the Property). This information has resulted from additional mineral resource delineation drilling at the project (the Project), updated mineral resource estimates, and a preliminary economic assessment. This technical report supports the recently updated mineral resources at the La Preciosa silver property and the results of the preliminary economic assessment. The economic assessment is preliminary in nature. It includes Inferred mineral resources that are considered too speculative geologically to have the economic conditions applied to them that would enable them to be categorised as mineral reserves, and there is no certainty that the preliminary economic assessment will be realised.

1.1 Property description and ownership

This technical report refers to the Property, an advanced stage silver exploration project located approximately 84 kilometres (km) by road northeast of the city of Durango, in Durango State, México. In October 2009, Pan American Silver Corp. (Pan American), Orko Silver Corp. (Orko), PASMEX S.A. de C.V. (PASMEX, a subsidiary of Pan American which holds Pan American s interest in the Project) and Proyectos Mineros La Preciosa S.A. de C.V. (the La Preciosa JV Company) entered into a formal joint venture agreement in connection with the Project. Under the terms of the joint venture agreement, PASMEX acts as the Project operator and Pan American can earn a 55% interest in the joint venture by bringing the Project into production.

As of the end of 2010, Pan American had invested approximately $14 million on over 90,000 metres of exploration and infill mineral resource drilling, metallurgical testing, engineering activities to evaluate alternative extraction and development scenarios to maximize the Project s economics, baseline environmental studies, and community relations work.

1.2 Geology and mineralisation

The Property comprises a block of mineral exploitation concessions covering an area of approximately 1,134 hectares (ha) located on the eastern flank of the Sierra Madre Occidental mountain range. Cretaceous age conglomerate and Tertiary age andesitic volcanic rocks are the main hosts of Tertiary age epithermal quartz veins containing economic levels of silver and gold mineralisation, as well as barite and lesser quantities of base metals, primarily zinc and lead. Two major vein and vein breccia systems are exposed on hills and ridges on either side of an approximately 800 metre (m) wide valley. The dominant geological feature on the Property is the northwest-trending La Preciosa Ridge which hosts the dominantly north-striking and westward-dipping main vein system, which includes the Martha, Abundancia, Gloria, Pica, Luz Elena, Sur, and Nueva veins. These veins are cross-cut by east-striking, south-dipping Transversal veins. The major vein breccia system to the east of La Preciosa Ridge on the eastern side of the valley floor includes the northwest striking Zona Oriente and Zona Oriente Extension, which is believed to be the surface expression of the Martha vein.

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1.3 Status of exploration, development, and operations

On the order of 30,000 tonnes (t) of material from the La Gloria and Abundancia veins was removed by historical mining dating from the late 1800s. Underground workings are accessible on the La Gloria and Abundancia veins over a distance of approximately 2.5 km at the north end of the La Preciosa Ridge. Approximately 10,000 t of stockpiled mineralised material is present on the surface above the old workings. Historical underground workings are also found at the south end of the ridge. Aside from drill access roads and buildings to support drill core processing activities and core storage, no other development has been made on the Property and there are no active mining operations.

Numerous mineralised vein structures have been identified by nearly 1,000 drillholes and underground channel samples principally

distributed over an area of approximately 850 ha. A total of 18 different veins tested by 677 drillholes have demonstrated sufficient geological continuity for estimation of mineral resources.

1.4 Mineral resource estimate

The most recently estimated mineral resources effective 30 June 2011 (shown in Table 1.1) and the results of a preliminary economic assessment based on these mineral resources are disclosed in this technical report. No mineral reserves have been estimated for the Project. Additional infill drilling is recommended to upgrade Inferred mineral resources to Indicated for conversion to mineral reserves as the Project progresses. A number of continuous improvement recommendations have been made in the relevant sections of this report to increase confidence in the data used to estimate mineral resources.

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Table 1.1 La Preciosa Project mineral resource estimate effective 30 June 2011

Mining method	Classification	Cut-off grade Ag ppm	Tonnes (millions)	Ag ppm	Silver million ounces	Au ppm	Gold thousand ounces	Silver equivalent ppm	Silver equivalent million ounces
Open pit	Indicated	35	10.9	129	45	0.19	66	139	49
Open pit	Inferred	35	7.6	74	18	0.13	31	81	20
Underground	Indicated	85	13.9	152	68	0.35	156	170	76
Underground	Inferred	85	7.6	117	28	0.21	52	128	31
Total	Indicated		24.8	142	113	0.28	222	156	124
	Inferred		15.2	96	46	0.17	83	105	51

Notes:

Mineral resources that are not mineral reserves do not have demonstrated economic viability. CIM (2010) defines a mineral reserve as the economically mineable part of a Measured or Indicated mineral resource demonstrated by at least a preliminary feasibility study. No mineral reserves have been estimated. Mineral resources have accounted for minimum mining width and planned mining dilution.

The estimate of mineral resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues, but no such issues have been identified at this time.

Tonnes, grades, and ounces have been rounded and this may have resulted in minor discrepancies in the totals. Grades are expressed in parts per million (ppm) which is equivalent to grams per tonne (g/t).

Cut-off grades are based on operating cost estimates and metal prices of $25 per ounce silver (Ag) and $1,250 per ounce gold (Au). Metal prices are based on a weighted average of historical three year average daily silver prices and a two year future price forecast.

The division between open pit and underground mineral resources is set on a horizontal level at the 1920 m elevation, which is considered close to optimum at the metal prices and operating costs assumed in this preliminary economic assessment.

Silver equivalent grade values assume a gold to silver ratio of 50 to 1 based on the assumed metal prices. The metallurgical recoveries and refining charges are assumed to be the same for silver and gold for the purposes of the equivalence calculation only.

1.5 Mining methods and Project infrastructure

Mineralisation at the Project will be exploited using both open pit and underground mining methods. Open pit mining will be undertaken using conventional truck and shovel techniques. The veins at the Property vary widely in both width and dip to an extent that the choice of a single underground mining technique will not be appropriate for all veins. Shrinkage stoping was identified as the most appropriate method for steeply dipping veins (greater than 70°) and room and pillar (with backfill where the width requires it) was considered appropriate for shallow dipping veins (less than 35°).

The Property is not yet connected to the commercial electrical grid but a nearby village and a town are serviced by the commercial electrical grid. The Property is presently supplied electrical power by one 65 kilowatt (kW) diesel generator and two smaller 5.5 kW diesel generators. The main power grid for Durango follows a paved federal highway and a power connection is available for the Project from a substation located in the city of Canatlán, Durango, 41 km northwest of the Project site.

A nearby town has a gas station and the services of metal fabricators and mechanic shops. A railway line is present near the south boundary of the Property and the railway has a direct line to Torreon, the site of the nearest metal smelter.

The water for drilling and services is obtained from a water reservoir in a nearby village. Water for mining production is proposed to be supplied from an underground source

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in the thick gravels on the plain to the east of the Project, accessed by drilling a 200 m deep water well 7.2 km to the east of the Project.

Presently the Property has six core storage sheds, an office, lunch room, washrooms, small warehouse, flammable substances storage area, drilling company workshop, night watchman s accommodation, and a generator/core cutting shed.

The proposed main processing facilities for the Project will include primary crushing, grinding, leaching, counter current decantation, tailings detoxification, silver and gold precipitation, refining, and tailings disposal facilities. In addition to the main process facilities, there will be several surface buildings constructed to support the mining and process operations. These facilities include administration, security, warehouse, change house, explosives storage, and truck shop buildings, and truck wash and mill maintenance facilities.

1.6 Capital and operating costs and economic analyses

Based on the mineral resource estimate and some economic assumptions, Snowden has developed a detailed mine plan and schedule, and incorporated assumptions with respect to metallurgical recovery, mining recovery, and dilution. This plan has resulted in an estimation of capital costs as detailed in Table 1.2 and operating costs as summarised in Table 1.3. The average life-of-mine cash costs were estimated at $11.84 per ounce of silver, net of gold by-product credits.

Table 1.2 La Preciosa Project preliminary capital cost estimates

Type	**$ Million**
Open pit mining equipment	25.9
Underground mining equipment	15.8
Open pit pre-strip	4.4
Underground capital development	3.7
Plant	143.3
EPCM	22.1
Owner s costs	47.9
Escalation of plant and EPCM costs	6.4
Total estimated capital(1)	**269.5**

Note(1): Excludes sustaining capital. EPCM is engineering, procurement, and construction management costs

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Table 1.3 La Preciosa Project preliminary operating cost estimates

Cost type	$/Tonne	$/Tonne milled
Open pit reference mining ($ per tonne of waste)(1)	1.11	
Open pit reference mining (($ per tonne of feed)	1.45	
Underground mining ($ per tonne of feed)	31.49	
Total mining		26.60
Processing ($ per tonne of feed)		16.64
General and administration ($ per tonne of feed)		2.57
Average total cost(2) ($ per tonne of feed)		**45.81**

Note(1): Reference costs are at pit crest. Mining cost increases with depth.

Note(2): Excludes taxes and royalties.

The metal prices used in this technical report are based on a weighted average calculation of the historical prices (weight 60%) and future forecast prices (weight 40%) as of the end of June, 2011.

For the calculation of the silver price used in this technical report, the historical price of US$19.44 is the average of the London Bullion Market daily silver prices for the 36 months prior to the end of June, 2011. The future price of $33.70 is the 24 month future price forecast on the Chicago Mercantile Exchange (previously the Comex futures division of NYMEX). The weighted average price of $25.14 was rounded down to $25.00.

For the calculation of the gold price used in this technical report, the historical price of US$1,110.70 is the average of the London Bullion Market daily PM gold prices for the 36 months prior to the end of June, 2011. The future price of $1,491.38 is the 24 month future price forecast on the Chicago Mercantile Exchange. The weighted average price of $1,262.98 was rounded down to $1,250.

Based on the costs, the plan and schedule, and the commodity prices, the financial outcomes that were derived for the Property are shown in Table 1.4. These outcomes include allowances for taxes and royalties as detailed in Section 22.1.

Table 1.4 La Preciosa Project preliminary financial metric estimates(1)

Measure	Units	Value(2)
Undiscounted present value	$ Million	497.0
Net present value at 5% discount p.a.	$ Million	314.6
Net present value at 10% discount p.a.	$ Million	191.2
Internal rate of return	%	24.3
Payback period	Years	3.3

Note(1): The economic assessment is preliminary in nature. It includes Inferred mineral resources that are considered too speculative geologically to have the economic conditions applied to them that would enable them to be categorised as mineral reserves, and there is no certainty that the preliminary economic assessment will be realised.

Note(2): Based on metal prices of $25/oz silver and $1,250/oz gold. Inclusive of taxes, royalties, and a management fee of 5% of operating costs ($50.9 million) and 5% of initial capital costs ($13.5 million) paid to the operator.

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1.7 Mineral processing, metallurgical testing, and recovery methods

In the period from 2007 to 2010 a total of 865 samples from diamond drill core were used to prepare 44 metallurgical composites, including 28 variability composites. They were studied at two independent commercial laboratories to evaluate the response to gravity and flotation concentration and agitation cyanidation.

Whole ore agitation cyanidation was determined to be the most favourable processing option. At the currently estimated global plant feed grade of 137 parts per million (ppm) silver (Ag), the data forecasts a laboratory silver extraction of 86%. Gold extraction is affected by oxidation with a nominal 70% extraction in the sulphide zones increasing to 90% in some of the oxidised zones, with an overall forecast of 78% gold extraction.

The consumption of cyanide has been quite variable. Although the majority of the test work reported consumption of less than 1.3 kilograms per tonne (kg/t), some tests were as high as 6 kg/t (which is considered to be a high rate of cyanide consumption), where favourable metallurgical results were reported. Additional testing will be required to better understand the reported high cyanide consumptions. It is anticipated that to the extent that elevated cyanide consumption is encountered in operation, the cyanide concentration in the plant solutions will be permitted to decrease below the optimum of 2 grams per litre (g/l), incurring a few percent loss in silver extraction but a net gain as a result of lower cyanide costs.

The deposit benefits from fine grinding to at least P_{80} = 74 microns (80% passing 200 mesh) with a leaching time that is expected to be in the range of three to four days. Small quantities of copper and zinc are leached in the process and report to the pregnant solution as cyanides. They will be destroyed in the operating plant cyanide destruction circuit.

A two staged laboratory test programme is recommended in which the first stage will determine the role of pH. The second stage should comprise another variability testing programme to confirm the silver and gold tailings grades by ore type, establish crushing and grinding work indices including JK drop weight, and cyanide destruction. The results of this programme may alter the feasibility study stage design criteria and therefore impact the capital and operating costs, particularly so in the grinding circuit.

The plant facilities for the Project have been designed for a throughput of 5,000 dry metric tonnes per day. Feed grade material will be processed by crushing and grinding prior to cyanide leaching. Recovery of soluble metals will be accomplished by multi-stage counter current decantation followed by zinc precipitation. The precipitate will be further refined through smelting and will yield a doré product for sale that will contain the silver and gold produced at the Project. Leach tailings will go through cyanide detoxification with sulphur dioxide prior to discharge at the tailings facility. The tailings facility will be of conventional construction utilising multiple spigot discharge points and a reclaim barge. A tailing facility containment dam will be constructed from mined waste or surface quarry material.

1.8 Environmental studies, permitting, and social and community impact

Reporting of the environmental baseline data collected for a full year between 2010 and 2011 has been completed for the Project. The baseline data will be used to compile the environmental impact statement (MIA), which will be submitted to the Mexican government for approval prior to the issuing of construction and operation permits. The most likely significant environmental issues that may be related to the permitting of the Project include long-term water quality and quantity management, securing water rights, protracted approval and permitting processes, long-term management of metal leaching and acid rock drainage (MLARD), construction and operation of the access road, social issues, securing surface rights, and management and liner considerations for the tailings impoundment facility. The main documents to be prepared and submitted to obtain construction and operation permits are the MIA, a forest land use modification, a risk analysis, and an archaeological study report.

Likely social concerns surrounding the development of the Project include acquisition of surface land, water use, perceptions of the cyanide facility and the use of cyanide, operation of the access road, potential imposition of access restrictions to the area, and the expectations that the Project development may generate in surrounding communities with respect to employment and quality of life.

It is recommended that a programme be commenced to acquire rights to the land in the area of the Project which will allow the infrastructure to be commissioned on the Property. The access rights will include the area for the well, transmission corridors, mine and mill areas, tailings storage facility, buffer zone, etc. As water supply is critical to the success of the Project, it is recommended that water rights required for the facility also be acquired. An extensive community engagement process should also be immediately established.

Consideration should be given to continue to collect environmental data beyond the baseline data required for the MIA. The activities to be considered include:

- The installation of a weather station for meteorological data collection.
- The installation of monitoring wells and piezometers for open pit areas as well as upstream and downstream from the tailings storage facilities and process plant to monitor ground water characteristics.
- Design and implementation of an environmental monitoring programme.
- Dust monitoring.
- Complete static and kinetic testing for MLARD which will be coupled with an ongoing monitoring programme for waste rock and tailings.

1.9 Conclusions and recommendations

With a net present value of $315 million at a 5% discount rate, an internal rate of return of 24.3%, and a Project payback of 3.3 years, the Project is sufficiently robust that, at the metal prices used in this analysis ($25 per ounce silver and $1,250 per ounce gold), there is a reasonable likelihood that the Project will proceed following the completion of a feasibility study.

On the basis of the results of this preliminary economic assessment, the recommendation is made to proceed to a feasibility study. The expected cost of completing the feasibility study, from the issuing of the preliminary economic assessment forward, has been estimated at $8 million (excluding the acquisition costs of land and water rights).

Inferred mineral resources will be excluded from the estimates forming the basis of the recommended feasibility study, as required by CIM (2010), which will impact the feasibility study financial outcomes.

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1.10 Cautionary note regarding forward-looking information and statements

This preliminary economic assessment is preliminary in nature. It includes Inferred mineral resources that are considered too speculative geologically to have the economic conditions applied to them that would enable them to be categorised as mineral reserves, and there is no certainty that the preliminary economic assessment will be realised.

Certain of the statements and information in this Technical Report constitute forward-looking statements within the meaning of the United States Private Securities Litigation Reform Act of 1995 and forward-looking information within the meaning of applicable Canadian Provincial securities laws. All statements, other than statements of historical fact, are forward-looking statements. When used in this Technical Report the words estimates , expects , projects , plans , contemplates , calculates , objective , potential , and other similar words and expressions, id forward-looking statements or information. These forward-looking statements or information relate to, among other things: the future successful development of the Project; the estimates of expected or anticipated economic returns, as reflected in the preliminary economic assessment; the timing for completion of a feasibility study and environmental impact assessment on the Project; future production of silver and gold and mine-life of the Project; future cash costs per ounce of silver; the price of silver and gold; the sufficiency of Pan American s current working capital, anticipated operating cash flow or its ability to raise necessary funds; the capital necessary to construct a mine at the Project and the time-line for such construction; the accuracy of mineral resource estimates; estimated production rates for silver and other payable metals produced at the Project; timing of production and the cash and total costs of production; the estimate of metallurgical recoveries for silver and gold; the estimate for mining dilution; the estimated cost of and availability of funding necessary for sustaining capital; and ongoing or future development plans and capital replacement, improvement or remediation programmes.

These statements reflect current views with respect to future events and are necessarily based upon a number of assumptions and estimates that, while considered reasonable, are inherently subject to significant business, economic, competitive, political and social uncertainties and contingencies. Many factors, both known and unknown, could cause actual results, performance or achievements to be materially different from the results, performance or achievements that are or may be expressed or implied by such forward-looking statements contained in this Technical Report and assumptions and estimates have been made based on or related to many of these factors. Such factors include, without limitation: fluctuations in spot and forward markets for silver, gold, base metals and certain other commodities (such as natural gas, fuel oil and electricity); fluctuations in currency markets (such as the Mexican Peso versus the United States Dollar); changes in national and local government, legislation, taxation, controls or regulations and political or economic developments, particularly in Mexico and in Canada; risks and hazards associated with the business of mineral exploration, development and mining (including environmental hazards, industrial accidents, unusual or unexpected geological or structural formations, pressures, cave-ins and flooding); employee relations; relationships with and claims by local communities and indigenous populations; availability and increasing costs associated with mining inputs and labour; the speculative nature of mineral exploration and development, including the risks of obtaining necessary licenses and permits and the presence of laws and regulations that may impose restrictions on mining; diminishing quantities of grades of mineral reserves as properties are mined; global financial conditions; challenges to, or difficulty in maintaining, title to properties and continued ownership thereof; the actual results of current exploration activities, conclusions of economic evaluations, and changes in Project parameters to deal with unanticipated economic or other factors; increased competition in the mining industry for properties, equipment, qualified personnel, and their costs; and, with respect to Pan American, those factors identified under the caption Risks related to Pan American s business in Pan American s most recent Form 40F and annual information form filed with the United States Securities and Exchange Commission and Canadian provincial securities regulatory authorities. Investors are cautioned against attributing undue certainty or reliance on forward-looking statements. Although Pan American and Orko have attempted to identify important factors that could cause actual results to differ materially, there may be other factors that cause results not to be as anticipated, estimated, described, or intended. The companies do not intend, and do not assume any obligation, to update these forward-looking statements or information to reflect changes in assumptions or changes in circumstances or any other events affecting such statements or information, other than as required by applicable law.

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2 Introduction

This technical report has been prepared by Snowden Mining Industry Consultants Inc. (Snowden) for Pan American Silver Corp. (Pan American) and Orko Silver Corp. (Orko), in compliance with the disclosure requirements of Canadian National Instrument 43-101 (NI 43-101), to support disclosure of the results of a preliminary economic assessment of the Property. This disclosure includes information from additional mineral resource delineation drilling, updated mineral resource estimates, and a preliminary economic assessment.

The effective date of this technical report is 30 June 2011. The Project drilling data cut-off date for mineral resource estimation was 25 October 2010. The economic analysis to determine the appropriate cut-off grades for reporting mineral resources and for the subsequent mining study was completed on 30 June 2011. No new material information has become available between these dates and the signature date given on the certificate of the qualified persons.

This preliminary economic assessment is preliminary in nature. It includes Inferred mineral resources that are considered too speculative geologically to have the economic conditions applied to them that would enable them to be categorised as mineral reserves, and there is no certainty that the preliminary economic assessment will be realised. Mineral resources that are not mineral reserves do not have demonstrated economic viability. No mineral reserves have been estimated.

Pan American is a silver mining and exploration company listed on the Toronto Stock Exchange (TSX:PAA) and the NASDAQ (NASDAQ:PAAS) stock exchange. Orko is a silver exploration company listed on the TSX Venture Exchange (TSX.V: OK.).

Unless otherwise stated, information and data contained in this report or used in its preparation have been provided by Orko and Pan American. This technical report has been compiled from sources listed in the References Section and cited in the text by Mr. Anthony Finch, P.Eng., M.AusIMM, Divisional Manager, Mining Engineering of Snowden, Mr. Michael Stewart, M.AIG, Principal Consultant of Quantitative Geoscience Pty. Ltd. (QG), Mr. Joshua Snider, P.E., Engineer with M3 Engineering & Technology Corporation. (M3), Mr. Thomas Drielick, P.E., Senior Vice President with M3, and Mr. Gary Hawthorn, P.Eng., Owner of Westcoast Mineral Testing, Inc.. Mr. Finch, Mr. Stewart, Mr. Snider, Mr. Drielick, and Mr. Hawthorn are qualified persons as defined by NI 43-101 and are independent of Pan American and Orko. The responsibilities of each co-author are provided in Table 2.1.

Mr. Snider visited the Project site on 9 June 2010, accompanied by Hernán Dorado Smith, Senior Planning Engineer of Pan American. Mr. Snider reviewed the site with emphasis on process plant and tailings locations.

Mr. Finch and Mr. Stewart visited the Project site on 5 July 2011, accompanied by Ms. Pamela De Mark, Director of Resources, Samuel Coronado, La Preciosa Project Manager, Sergio Morfín, Exploration Manager México, and Hernán Dorado Smith, Senior Planning Engineer, all representatives of Pan American. Mr. Finch reviewed the potential pit locations, the portal and the dump locations, as well as the general infrastructure and access to the Project site. Mr. Stewart reviewed representative drill core intersections of the veins and surrounding host rock located at the core storage facility on the Project site, confirmed the collar coordinates of selected drillholes, visited

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outcrops of the Martha, Gloria, Abundancia, and Transversal veins, reviewed paper and digital geological interpretations, and reviewed the geological database.

Because a current personal inspection has been conducted by other qualified persons responsible for the preparation of this technical report, and no additional beneficial information would have been derived from a site visit at this stage of the Project, Mr. Hawthorn and Mr. Drielick did not conduct a site visit.

Table 2.1 Responsibilities of each qualified person

Qualified person	Company	Responsible for sections
Anthony Finch	Snowden Mining Industry Consultants Inc.	1: Summary; 2: Introduction; 3: Reliance on Other Experts; 4: Property Description and Location; 5: Accessibility, Climate, Local Resources, Infrastructure and Physiography; 12: Data Verification; 15: Mineral Reserve Estimates; 16: Mining Methods; 19: Market Studies and Contracts; 20: Environmental Studies, Permitting and Social or Community Impact; 21: Capital and Operating Costs; 22: Economic Analysis; 24: Other relevant data and information; 25: Interpretation and Conclusions; 26: Recommendations; 27: References
Michael Stewart	Quantitative Geoscience Pty. Ltd.	1: Summary; 2 :Introduction; 6: History; 7: Geological Setting and Mineralisation; 8: Deposit Types, 9: Exploration; 10: Drilling; 11: Sample Preparation, Analyses and Security; 12: Data Verification: 14: Mineral Resource Estimates; 23: Adjacent Properties; 25: Interpretation and Conclusions; 26: Recommendations
Joshua Snider	M3 Engineering & Technology Corp.	1: Summary; 2: Introduction; 12: Data Verification; 18: Project Infrastructure; 21: Capital and Operating Costs; 25: Interpretation and Conclusions; 26: Recommendations
Thomas Drielick	M3 Engineering & Technology Corp.	1: Summary; 2: Introduction ; 12: Data Verification; 17: Recovery Methods; 25: Interpretation and Conclusions; 26: Recommendations
Gary Hawthorn	West Coast Mineral Testing Inc.	1: Summary; 2: Introduction ; 12: Data Verification; 13:Mineral Processing and Metallurgical Testing; 25: Interpretation and Conclusions; 26: Recommendations

Unless otherwise stated, all units are metric and currencies are expressed in US dollars ($). Project data coordinates are based on the International Terrestrial Reference System (ITRS), which is similar to within a few centimetres of the World

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Geodetic System (WGS) 84, the coordinate system used by the Global Positioning System (GPS).

3 Reliance on other experts

The qualified persons preparing this technical report have relied on the reports, opinions, and statements of experts who are not qualified persons as defined by NI43-101. Information regarding environmental aspects of the Property has been provided by Wade Stogran, Director, Environmental Affairs of Pan American, who is not a qualified person. Information regarding land occupancy and lease agreements has been provided by one of the La Preciosa JV Company s legal counsel in Durango, Mexico, Mr. Eduardo Bravo Campos, now deceased.

In development of the mineral inventory for this assessment Snowden has based its geotechnical design criteria on a report written by Golder Associates (Golder, 2010). This report was commissioned by Pan American to assess the geotechnical aspects of mining at the Project in both the underground and the open pit environments.

4 Property description and location

Information in this section is updated from Mine Development Associates (MDA, 2009).

4.1 Location, mineral tenure, and surface rights

The Property is located approximately 84 kilometres (km) by road northeast of the City of Durango in Durango State, México. The centre of the mineral resources are at 2,702,000 North, 555,400 East in the Universal Transverse Mercator (UTM), North American Datum of 1927 (NAD 27).

The Project is located within eight concessions with a total area of 1,134.1 hectares (ha). Each corner of each concession is surveyed by a licensed surveyor with reference to the location of a claim monument. The Property is surrounded by the Santa Monica and San Juan properties, which are also controlled by the La Preciosa JV Company. A map of the Property land position is shown in Figure 4.1 at three different scales.

Details of the concessions, including the expiration date of the claims, are given in Table 4.1. The La Preciosa JV Company holds 100% of the registered and beneficial title in the Properties, free and clear of all encumbrances (other than liens in favour of government authorities as reflected by the terms of the mineral leases, licenses and permits and all obligations arising from any royalty or similar agreements existing on the effective date of the joint venture agreement in favour of government authorities), except as discussed in Section 4.3.

The La Preciosa JV Company has entered into a series of contracts with local landholders and Ejido Councils (farm owners collective) in order to conduct exploration on their land. Separate contracts are held to maintain free access to the site. New negotiations will be required to either purchase land or to obtain long term agreements for the future Project operations.

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Figure 4.1 La Preciosa Property land position location map

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Note: black lines are lease boundaries controlled by the La Preciosa JV Company, black shaded areas are third party land holdings, and red shaded area is the vein interpretation.

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Table 4.1 La Preciosa Property concession details

Claim name	Expedient	Title	Issue date	Area (ha)	$MXP(1) per ha	Total $MXP(2)	Expiry date
La Preciosa property concessions							
La Preciosa	321.1-2/398	182517	15/07/1988	143.6119	111.270	15,980	14/07/2038
Lupita	321.1/9-303	182584	12/08/1988	27.1878	111.270	3,025	11/08/2038
Fracción La Preciosa	321.1/2-399	185128	14/12/1989	2.5249	111.270	281	13/12/2039
San Patricio	321.42/919	189616	05/12/1990	29.474	111.270	3,280	04/12/2040
La B	2/1.3/01962	214232	06/09/2001	28.2006	111.270	3,138	05/09/2051
El Choque Tres	2/1/02251	218953	28/01/2003	10.0	63.220	632	27/01/2053
El Choque Cuatro	25/30812	220251	02/07/2003	644.1296	31.620	20,367	01/07/2053
El Choque Seis	25/31144	220583	02/09/2003	249.0	31.620	7,873	01/09/2053
Adjacent concessions controlled by the La Preciosa JV Company							
Santa Monica	25/31208	221288	20/01/2004	16385.457	31.620	518,108	19/01/2054
Santa Monica Sur	25/31411	223097	15/10/2004	900.0	31.620	28,458	14/10/2054
San Juan	25/31434	226663	17/02/2006	14003.4737	15.72	220,135	16/02/2056

Note(1): MXP = Mexican Pesos. Note(2): Fees are payable twice per year and are due every January and July.

4.2 Issuer s interest

On 13 April 2009, Pan American and Orko announced that they had signed a letter agreement setting out the basic terms under which they may jointly develop the Project. Pursuant to the letter agreement, Pan American agreed to act as operator and, in order to retain a 55% interest in the Project, to (i) make a minimum of $5,000,000 in expenditures relating to exploration of the properties within 12 months of executing the letter agreement, (ii) prepare a feasibility study in respect of the Project within 36 months of executing the letter agreement, and (iii) following a positive construction decision by the parties, contribute 100% of the funds necessary to develop and construct an operating mine as contemplated in the feasibility study.

In October 2009, Pan American, Orko, (PASMEX, a subsidiary of Pan American which holds Pan American s interest in the La Preciosa Project and which is the operator) and the La Preciosa JV Company entered into a formal joint venture agreement in connection with the Project.

Pursuant to the terms of the joint venture agreement dated 13 April 2009, if PASMEX fails to complete the required expenditures during the first 12 months or to complete the feasibility study within the 36-month period, PASMEX will surrender its interest in the La Preciosa Project. If, however, PASMEX incurs the required expenditures during the first 12 months and completes the feasibility study within the allotted period, but elects not to proceed with funding the construction of the Project, PASMEX will surrender its 55% interest but will be entitled to receive a 1.5% net smelter return royalty on the Project as provided in the joint venture agreement. Orko or the La Preciosa JV Company have the right to purchase the net smelter return royalty from PASMEX for $8,000,000 for a period of three years from the date of the surrender of the Pan American interest.

4.3 Royalties, back-in rights, payments, agreements, and encumbrances

The Project is subject to the San Juan Property Option Agreement dated April 10, 2006, among Orko and the Silver Standard Group, in accordance of which the La Preciosa JV Company pays the La Cuesta Royalty, comprised of $5,000 or 2% of direct exploration costs, and 0.25% of net smelter returns.

Additionally, there is a Net Smelter Return Royalty Agreement dated 19 June 2002 among Minas Luismin S.A. de C.V., Minas Sanluis, S.A. de C.V. and Corporación Turística Sanluis, S.A. de C.V. (CTS) which runs with the Property and grants a 3% net smelter returns royalty to CTS on minerals derived from the La Preciosa, Lupita, Fracción La Preciosa, San Patricio, El Choque Tres, and La B claims.

For this preliminary economic assessment, El Choque Cuatro and El Choque Seis are also included in economic model but they are not subject to any royalties. Therefore, based on the estimated feed grade tonnes to be extracted from the various claims, a weighted average net smelter royalty of 2.5% is applied in the economic model.

There are no other known royalties, back-in rights, payments, agreements, or encumbrances.

4.4 Environmental liabilities

A stockpile of approximately 10,000 tonnes (t) of material produced from the historical workings is located towards the north end of La Preciosa Ridge and there are small historical workings over some of the veins. These do not present a significant environmental liability and will be processed as plant feed.

Nearby farmers produce beans and maize and local cattle graze on neighbouring land. There are a number of unpaved access roads on the Property as well as minor infrastructure to support exploration and drilling activities. The major environmental issue in the area is the current cattle, goat and agricultural regime that tends to exceed the carrying capacity of the area.

4.5 Permits

Pan American has obtained five exploration permits from the Secretaría de Medio Ambiente y Recursos Naturales (Ministry of Environment and Natural Resources, SEMARNAT) since the start of Pan American s involvement in the Project. Data collection work is complete to compile the database of vegetation, wildlife, ground and surface water quality and quantity, climate, and other items necessary for submission of the MIA (Environmental Impact Assessment). The database summary report has been completed and the information from that report will be used for compiling the MIA for submission to and approval by SEMARNAT. Pan American has obtained contracts and agreements with the owners of the Property for exploration and will need the same for mining.

4.6 Significant factors and risks

The Property is subject to the same risks as any other mining project in México or in other newly industrialised nations, including under-estimation of capital and/or operating costs, delays or inability in securing land agreements, the inability to attract qualified personnel, personal security risks, poor communication with stakeholders, over-estimation of mineral resources and metal prices, inflation of capital and/or operating costs, complicated geotechnical conditions, and changes in the Project mine plan. There are no other known significant factors and risks that may affect access, title, or the right or ability to perform work on the Property.

5 Accessibility, climate, local resources, infrastructure and physiography

Information in this section is updated from MDA (2009).

5.1 Access

The Property is located approximately 84 km by road northeast of the city of Durango and can be accessed by vehicle from Durango in approximately 90 minutes. A Google satellite map showing the location of the Property relative to Durango and the access roads is shown in Figure 5.1. From Durango, travel is northeast toward Torreon by the sealed Federal Highway 40 to the town of Francisco I. Madero. From this point a secondary paved road is followed northwest to the village of Lázaro Cardenas, then by a newly paved road to the village of Francisco R. Serrano. After 9 km there is a turnoff on a newly paved road southwest to the village of Francisco Javier Mina, then travel is to the south for 5.5 km by gravel road to the access road to the Project site. The access road is a 3.5 km gravel road heading southeast and leads to the portal of the historic workings and the main camp of the Project.

Figure 5.1 La Preciosa Property location and access map

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5.2 Climate and length of operating season

The Property area has a semi-arid climate, with an annual average temperature of about 25°C and an average annual precipitation of about 600 millimetres (mm), usually occurring between May and October. Temperatures can fall below freezing on winter nights but snow is rare. Activities can take place year round. The dominant wind direction is southeast.

5.3 Proximity to population centre and transport

Durango is the capital city of Durango State and has a population of approximately 600,000. One of the major industries in Durango is mining, particularly for silver, and the region is a good source of skilled personnel, support services, and mining equipment. The city of Durango is served by an international airport with daily flights connecting to destinations in México and the United States. Durango is located on Mexican Federal Highway 40 which connects Durango to Mazatlan approximately 310 km to the southwest on the Pacific coast and to Torreón approximately 245 km to the northeast. A rail line runs between Durango and Torreón and connects to other cities in México and the United States.

5.4 Surface rights, land availability, infrastructure, and local resources

5.4.1 Surface rights, land availability, and mining areas

The Property has ample land for the construction of the mine, mill, and supporting facilities including tailings and waste disposal. A plan of the proposed pits, dumps, tailings dam, plant, and infrastructure is shown in Figure 5.2

Surface rights are coordinated through agreements with the Ejido Councils (farm owners collective). Any future surface utilisation or construction will require negotiating with the Ejido Councils involved. Separate surface access contracts are also in place with some independent ranch owners who are not members of Ejido Councils. Details of the landowners in the area of the proposed mine and plant facilities are shown in Table 5.1. A plan of the land owner boundaries is shown in Figure 5.3.

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Figure 5.2 Plan of proposed pits, dumps, tailings dam, plant, and infrastructure

Table 5.1 Land owners in the area of proposed mine and plant facilities

Name and property type	Area (ha)

Candelaria Uves Solórzano	private	200
Petra Higareda	private	55
Ciro Diaz	private	100
Ricardo Flores Magón	Ejido	150
Lázaro Cárdenas	Ejido	101
La Preciosa	La Preciosa Resources	45

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Figure 5.3 Plan of land owner boundaries in the area of the proposed mine and plant facilities

5.4.2 Power, infrastructure, and water

Both the quality of infrastructure and the population density increases towards the city of Durango. The Property is not connected to the commercial electrical grid but the nearby village of Francisco Javier Mina (population around 920) and the town of Francisco I. Madero (population around 4,550) are serviced by the commercial electrical grid. The Property is presently supplied electrical power by one 65 kilowatt (kW) diesel generator and two smaller 5.5 kW diesel generators. The main power grid for Durango follows a paved federal highway and a power connection is available for the Project from a substation located in the city of Canatlán, Durango, 41 km northwest of the Project site.

The town of Francisco I. Madero has a Pemex gas station and the services of metal fabricators and mechanic shops. A railway line is present near the south boundary of the Property and the railway has a direct line to Torreon, the site of the nearest metal smelter.

Presently the Property has six core storage sheds, an office, lunch room, washrooms, small warehouse, flammable substances storage area, drilling company workshop, night watchman s accommodation, and a generator/core cutting shed.

The water for drilling and services is obtained from a water reservoir in Francisco Javier Mina, charged at a rate of $500 Mexican Pesos per 1.75 cubic metres (m^3), including the cost to haul the water to the Project by tanker trucks to water tanks located adjacent to the drilling areas. Water for mining production is proposed to be supplied from an underground source in the thick gravels on the plain to the east of the Project. The underground source will be accessed by drilling a well 200 metre (m) deep on the Ejido Lázaro Cardenas property located 7.2 km to the east of the Project. The cost for the installation of the well is anticipated to be on the order of $700,000.

The fee for industrial use of the water as imposed by the Mexican National Water Commission (CONAGUA) is $7.1623 Mexican Pesos per m^3 of water. The area in which the water well is proposed does not require a permit to initialise construction, however, once the well is complete and operational it must be registered with CONAGUA, which administers water management at the federal level.

5.4.3 Local resources and mining personnel

There is a sufficient local work force available in Durango and the surrounding region for Project construction and operators, details are shown in Table 5.2.

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Table 5.2 La Preciosa Project work force availability

Federal code	State	Municipality code	Municipality	Local code	Locality	Total population	Active population	Distance to Project (km)
10	Durango	1	Canatlán	1	Canatlán	11,495	4,285	43
10	Durango	1	Canatlán	110	Ricardo Flores Magón	1,467	479	13
10	Durango	5	Durango	1	Victoria de Durango	518,709	204,350	84
10	Durango	5	Durango	295	Vicente Suárez	92	31	48
10	Durango	20	Pánuco de Coronado	1	Francisco I. Madero	4,550	1,601	32
10	Durango	20	Pánuco de Coronado	8	Francisco Javier Mina (Corralejo)	919	201	9
10	Durango	20	Pánuco de Coronado	14	General Lázaro Cárdenas	389	121	27
10	Durango	20	Pánuco de Coronado	9	Francisco Rueda Serrano	541	160	23

5.5 Topography, elevation, and vegetation

La Preciosa lies on the western edge of the high plains of northern México, an extensive volcanic plateau characterised by narrow, northwest trending ranges separated by wide, flat-floored filled basins. In the Durango area, the basins have elevations of between 1,900 m to 2,100 m above sea level and the higher peaks rise to 3,000 m. The Property elevation in the area of the mineralised zones at La Preciosa is between 1,990 m and 2,265 m. The highest elevations on the Property are at the northwest trending La Preciosa Ridge which overlies the La Gloria and Abundancia veins. A broad valley forms to the east of the ridge and extends approximately 1 km toward another lower lying ridge to the northeast. Grasses, small shrubs, and cactus comprise the typical vegetation on the steep hillsides with larger bushes and mesquite trees in the lower lying areas near springs and streams. Nearby farmers produce beans and maize with groundwater sourced from thick gravel beds in the surrounding plains. Local cattle graze on land dominated by litho-soils supporting nopal (prickly pear) and huizache (acacia) scrubland.

6 History

6.1 Prior ownership, exploration, and development work

6.1.1 Late 19th century work

Mining has occurred on the Property area since the late 19th century. The earliest known workings from this time are concentrated on the Abundancia and La Gloria veins at the north end of La Preciosa Ridge, and are known as the Mina La Preciosa. Drifting on a vein at the south end of the ridges at Mina El Orito may also have taken place during the same time period. Mining ceased during the start of the Mexican Revolution in 1910 and the Property area lay idle until selective small scale mining operations took place between 1970 and 1979. Previous mining is believed to total no more than 30,000 t of material.

6.1.2 Work by Luismin from 1981 to 1982 and 1994

In 1981, Compañía Minera Minas San Luis (Luismin), operating as Minera Thesalia through a joint venture between Tormex S.A. and Luismin, conducted detailed channel sampling of surface outcrops in the eastern breccias (Zona Oriente) and main vein systems as well as a single east-west line of induced polarisation (IP) resistivity across the Property. Luismin also drilled seven diamond drill core holes, two from underground and five from surface, for 1319 m. The holes targeted the Abundancia and La Gloria veins 50 m to 75 m below the primary underground workings on the 2065 level. The drill programme terminated in 1982, reportedly due to falling metal prices. The half core is still intact and stored on site and the drillhole data is available in the database.

Luismin enlarged the main 2065 level within the Abundancia and La Gloria vein underground workings to a 3 m by 3 m size over approximately 60% of the drifts to provide access for trackless mining and drilling equipment. Luismin collected underground channel samples at two to three metre intervals along the drifts. In addition, Luismin collected chip samples from a total of 450 m of underground workings along the Abundancia vein and 408 m along the La Gloria vein. In total, Luismin collected 1,365 chip samples from underground.

Luismin extracted approximately 11,730 t of material at an estimated grade of 0.43 parts per million (ppm) gold (Au) and 157 ppm silver (Ag) from the underground workings and placed it in stockpiles at the portal, which remain largely intact.

Luismin completed only limited work after 1982, including a single 313 m long drillhole in the eastern vein breccia system (Zona Oriente) in 1994. The drillhole intersected a series of variably silicified zones and veinlet stock work with anomalous silver, gold, lead, zinc, and mercury grades, but the hole did not pass through the entire width of the structure. In 1988 a small scale bench metallurgical test was performed on a sample of material extracted from the Abundancia and La Gloria veins, the results of this test work are discussed in Section 13.

Luismin staff prepared several historical internal mineral resource estimates which relied heavily on underground chip sample assays and only limited drilling. The results of these estimates are no longer relevant.

6.1.3 Work by Orko from 2003 to 2008

In December 2003, Orko Gold Corp., now known as Orko Silver Corp., negotiated a joint-venture option agreement with Luismin. An independent property examination including the selection of eight surface rock samples for verification of historical data was performed in early 2004 on behalf of Orko. Geological mapping and surface sampling began in 2004 and target areas were identified for more detailed follow up.

In January 2005, 40 line kilometres of 3D induced polarisation resistivity and chargeability surveys were completed on the Property. The geophysical surveys were conducted over the north part of the main structures (Mina La Preciosa), extending across the central valley and the eastern vein breccia (Zona Oriente), and northward to Cerro Prieto and the northern projection of the main structures (La Preciosa Norte). Weak geophysical signatures were noted on the ridge where the Mina La Preciosa veins are known, but the method provided inconclusive results beneath the basalt cover to the north along the same vein trend. A large multi-line chargeability anomaly was observed in the central valley beneath the basaltic cover.

Orko s diamond drill core programme began in 2005 with the initial drilling targeting the Mina La Preciosa veins. The 2005 to 2008 drilling programmes successfully intersected the Abundancia, La Gloria, and Luz Elena veins in multiple intercepts, as well as oblique intercepts of the Esperancita and Carmen veins. The deeper and thicker Martha vein structure was discovered in late 2006 in drillhole BP06-77. In total, Orko drilled 388 diamond drill core holes for 152,368 m.

Orko prepared a series of five mineral resource estimates from 2006 to 2008. These estimates were prepared by interpretation of multiple mineralised zones defined by structure and by grade on longitudinal sections using minimum cut-off grades of 100 ppm Ag equivalent and 150 ppm Ag equivalent and a minimum true vein width of 1.5 m (silver equivalent was calculated as the silver assay grade plus 60 times the gold assay grade, assuming a 100% relative recovery of each metal). The interpretations were projected to a maximum of 25 m vertically below the drillhole intercept with the vein. Immersion method specific gravity measurements were available for every sample submitted during the 2005 to 2007 drilling programmes and were used to derive tonnes and contained metal. Grades were estimated by true thickness weighted average grade of all drill intercepts within each mineralised zone. All mineral resource estimates were classified as Inferred.

In March 2009, Orko disclosed the results of an independent mineral resource estimate undertaken by MDA (2009), based on Orko s 388 drillholes. Geological models were prepared on paper cross sections, digitised to honour the three dimensional contacts on the drillhole trace, and then converted into wireframe solids. The solids were used to code the assay database, define separate silver and gold grade estimation domains, to prepare the block model, and to code the block model for density. Silver grade estimation domains were based on a cut-off grade of approximately 4 ppm Ag to 30 ppm Ag, depending on the style of mineralisation. Gold grade estimation domains were based on a cut-off grade of approximately 0.1 ppm Au.

Drillhole intersections lying within the grade estimation domains were top cut for extreme grade values defined by examination of the sample grade statistics in each domain. Samples were then composited to 3 m lengths honouring geological boundaries. Estimation was by inverse distance squared with ordinary kriging and nearest neighbour estimates as a check, using a minimum of one sample composite and

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a maximum of 12 composites required to return an estimate and a restriction of two composites per drillhole.

The percentage of the block lying within the wireframe was coded to the block model and the percentage was then used to calculate the weighted average of the tonnes and grades for mineral resource reporting. All high grade material estimated within or 10 m beyond the digitised historical workings was depleted from the model, resulting in the depletion of 410,000 t.

Mineral resources were classified as Inferred and Indicated and reported from vein material only above a range of silver equivalent cut-off grades using a ratio of 60 gold to 1 silver. Above a 100 ppm Ag equivalent, Indicated mineral resources were reported as 10.6 million tonnes (Mt) at 201 ppm Ag equivalent, 185 ppm Ag, and 0.27 ppm Au in the Indicated category and 12.1 Mt at 200 ppm Ag equivalent, 185 ppm Ag, and 0.25 ppm Au in the Inferred category. These mineral resource estimates have not been reviewed or validated by QG and are reported for historical purposes only. The results of these mineral resource estimates have been superseded by the updated mineral resource estimates presented in Section 14.

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7 Geological setting and mineralisation

Information in this section has been excerpted and updated from MDA (2009) and Coote (2010).

7.1 Regional geology

The Property is located in a geological sub-province known as the Altas Llanuras or High Plains, on the eastern flank of the Sierra Madre Occidental mountain range (Figure 7.1). The Altas Llanuras sub-province is a volcanic highland composed of Tertiary (Paleocene) to Quaternary (Pleistocene) age sequences of andesite, dacite-rhyolite, and basalt, which in turn rest on a basement of Cretaceous age conglomerate and Permian age metamorphic rocks. The present basin and range topography reflects a series of north to northwest trending linear grabens along the range fronts.

In the region north of the city of Durango, sedimentary rocks of Cretaceous age are exposed in small windows through the Tertiary age volcanic rock cover. These consist of mudstone, shale, limestone, and conglomerate with volcanic, sedimentary, and limestone clasts. The Cretaceous age rocks are covered by a sequence of andesite tuff, flows, and agglomerate of the Paleocene-Eocene age Lower Volcanic Series. In the surrounding ranges, the Lower Volcanic Series is overlain by thick sequences of rhyolite and dacite ignimbrite, tuff, and volcanic breccia of the Oligocene age Upper Volcanic Series. The Upper Volcanic Series is not exposed at La Preciosa, but it is exposed in cliffs to the west of La Preciosa.

The basins and parts of the lower hills are covered with varying thicknesses of Pliocene to Pleistocene age basalt that erupted from numerous vents now marked by small volcanic cones and domes that dot the plains. Several volcanic vents have been mapped on the Property, including the prominent Cerro Prieto.

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Figure 7.1 Regional geological setting map

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7.2 Local geology

The oldest rocks within the Property, found only in deeper drill core intersections, are a Permian age metamorphic series consisting of graphitic schist, chlorite schist, and layers of quartzite. Above the metamorphic units is a thick package of Early Cretaceous age polymictic conglomerate composed of fragments of sub-angular to rounded schist, limestone, quartz, intrusives, and volcanic rocks and containing lenses of arkosic sandstone. This sedimentary package is overlain by andesitic tuff, andesite, and agglomerate of the regional Tertiary age Lower Volcanic Series. In places the flows are porphyritic and tuffs are partly welded. Rhyolites of the Tertiary age Upper Volcanic Series are not found in the immediate study area, but can be seen on cliffs further to the west on the San Juan property. The Cretaceous age conglomerate and Tertiary age Lower Volcanic Series andesitic rocks are the main host rocks of the mineralised veins, although vein mineralisation does extend into the basement metamorphic rocks. There are a few dacitic, rhyolitic, and andesitic dikes and micro-sills noted in deeper core intersections, but intrusive rocks are generally rare. The youngest rocks are basalt flows which erupted from several Pleistocene age volcanic vents and which now fill the lower valleys. A small number of basalt dykes related to this extrusive activity were encountered in drilling at La Preciosa. Cerro Prieto, Cerro Blanco, and Cerro La Chicharronera are prominent examples of the volcanic vents. An example cross section at 2701780 mN showing the drillhole traces and lithology at La Preciosa is shown in Figure 7.2.

Figure 7.2 Example cross section of drillholes and lithology at 2701780 mN

The Property covers a series of Tertiary age gold and silver-bearing epithermal quartz veins which also contain barite, calcite, fluorite, and quantities of base metals, primarily zinc, lead, and copper. There are two major vein and vein breccia systems exposed on a series of hills and ridges which are separated by a flat-floored valley roughly 800 m in width. The main vein system on La Preciosa Ridge (shown in Figure 7.3) consists of dominantly northward-striking and westward-dipping veins (e.g., Abundancia, La Gloria, Esperancita, Luz Elena, and Martha), plus east-west striking, south-dipping cross-cutting veins (e.g., Transversal). The eastern vein breccia system (Martha East) strikes northwest and is interpreted to be a surface expression of the shallowly dipping Martha Vein. A sub-parallel north-northwest trending vein system (La Plomosa, El Vaquero, and Nancy) is exposed on the hills immediately to the west of La Preciosa Ridge, mostly on the San Juan concession.

Figure 7.3 **Local geology plan**

7.3 Property geology

The La Preciosa Ridge vein system has been traced on surface for over 3.7 km and drilling has revealed that the veins continue to the north beneath the basalt cover. Exploration towards the northwest of the Martha vein was not conclusive because the drillholes did not extend to the contact of the volcanics and conglomerate. It is possible that the Martha vein has been truncated by a normal, east-west trending fault with dextral displacement, moving the vein to the east and at greater depth. Future drill programmes with deeper holes will explore this possibility. The veins are also interpreted to extend farther south of known exposures.

Individual veins have been traced for up to 3.1 km along strike. Within the main vein system, the Abundancia, La Gloria, Luz Elena, and Martha veins have been explored in the most detail. A representative cross section at 2701780 m N showing the Gloria, Abundancia, Luz Elena, Pica, Alacran, and Martha veins, drillhole traces, and the principal rock groups is shown in Figure 7.4. The Abundancia and La Gloria veins coalesce at depth with a shallow northward plunge at the intersection and the merged vein continues as the Abundancia vein. Vein thicknesses vary from 1.5 m to 26 m wide at Abundancia, 1.5 m to 17 m at La Gloria, and 1.5 m to 35 m at Martha veins. Continuity of the Abundancia and La Gloria vein structures has been demonstrated through a total of 2.5 km of underground drifts.

The Martha vein contains the largest part of the mineral resource on the Property. Mineralisation is found in metamorphic rocks at depth, through to the conglomerate unit above, and then follows the angular unconformity at the base of the andesite. The portion of the Martha vein with the greatest widths and highest grades occurs when the low angle structure develops on the contact of the volcanic and conglomerate units. The vein continues to depth in the shales but usually splits into several narrow veins. A similar effect occurs where the vein develops between the contact between the volcanic rocks and schists, or else when the vein extends into the volcanic units.

The east-west trending Transversal veins occupy a south-dipping normal fault, with the Zona Sur area representing the short offset, down dropped block. The Abundancia vein continues south of the Transversal vein in the Zona Sur. As it is in a structurally separate sector, it has been interpreted apart from the main Abundancia vein. The Esperancita vein is a northwest trending structure overlying the Abundancia vein.

Figure 7.4 **La Preciosa cross section at 2701780 mN showing drillhole traces, veins, and lithology**

7.4 Mineralisation

Mineralisation at the Property is hosted within multiple discrete poly-phase quartz veins, often displaying banded, smoky, drusy, and chalcedony textures. Fluorite, amethyst, a substantial number of barite laths, calcite, and rhodocrosite may also be present, and sulphide mineralisation in the form of sphalerite, galena, pyrite, chalcopyrite, acanthite, sparse native silver and free gold, as well as iron and manganese oxides have been noted in drill core. The principal silver bearing mineral at La Preciosa is acanthite, pseudomorphed after argentite or as microcrystalline to amorphous grains.

All veins on the deposit appear to be structurally controlled with movement on pre-, syn-, and post-mineralisation normal faults creating openings and fracturing in which the veins and mineralisation were deposited. The greatest thickness and highest grade portions are found where the largest amounts of opening and fracturing occurred. Away from these zones the expressions of the veins weaken. In each stage of variably crustiform banded fracture fill/breccia cement mineralogy, minor amounts of carbonate interstitial to quartz and chalcedonic quartz is succeeded by later carbonate-rich and quartz-poor assemblages. The trend of decreasing mosaic quartz/chalcedonic quartz and increasing carbonate/iron carbonate is repeated for each major stage of fracturing and brecciation.

In a recent petrologic study, Coote (2010) identified mainly argentite, tennantite/tetrahedrite, and Ag sulphosalts in samples. The majority of gold/electrum is inter-grown with or occupying the same paragenetic position as argentite, silver sulphosalts, sphalerite and galena, mostly transitional between quartz and carbonate/iron carbonate in formation. The minerals and textures identified in the studies indicate that gold/electrum has a similar hydrothermal paragenesis to silver sulphides/sulphosalts and base metal sulphides. Most of the identified gold/electrum is intergrown with argentite, silver sulphosalts, sphalerite, galena, and carbonate/iron carbonate. Only minor amounts of gold/electrum were identified in exclusive association with very fine grained quartz of early paragenesis in any given sequence of multiple stage fracture/cement mineralogy. There is a moderate to weak correlation between silver and gold, copper, and lead grades.

Wall rocks hosting mineralisation are variably silicified, with proximal patchy illite-smectite alteration and distal chlorite alteration. The presence of mangano-calcite has been noted in several drillholes, but it is not uniformly distributed. In shallower drillholes, pyrolusite and limonite often appear on fracture surfaces.

The host rocks and veins have undergone intense weathering. The base of oxidation is erratically distributed as weathering is controlled by the presence of post mineralisation faults which allowed the percolation of oxidised meteoric groundwater to vertical depths of 350 m below surface. Weathering minerals include iron oxides, iron carbonates, manganese oxides, and unidentified clays.

The Martha vein in general contains more calcite and a higher proportion of pale, low-iron content sphalerite, galena, and pyrite compared to the Abundancia and La Gloria veins. It can be considered as a mineralised zone or lode of stock work, silicification, breccias, veins, vein breccias, veinlets, and a general mix of multiple styles of mineralisation. Within this broader zone, the Martha lode ranges from 1 m thick to 35 m thick and averages approximately 5 m. Generally one but occasionally more high grade veins or vein breccias exist within the thickness of the vein zone. The high grade vein zones range from less than 1 m thick to 10 m thick and average 5 m. The upper

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Martha vein (Martha Superior) has the highest silver grades and is primarily composed of quartz and contains dark sphalerite, pyrite, chalcopyrite, argentite, silver sulphosalts, and native silver. The main Martha vein and the lower Martha vein (Martha Inferior) are lower in silver tenor and are composed primarily of quartz and carbonate (calcite and rhodochrosite) with pyrite, galena, iron-poor sphalerite, and silver sulphosalts. In Martha East, pale green fluorite is present as a late stage mineral at a surface expression of the Martha vein where it flares into veinlets and silicified breccia bodies.

Eighteen geologically continuous veins have been defined by extensive diamond drilling intersections and three dimensional geological interpretations (wireframes) have been made of these veins for the purpose of mineral resource estimation. The principal vein, Martha Alta, extends along strike for a distance of 3 km and has a width of approximately 1.5 km. All the vein sets strike roughly north-south except for two cross cutting veins (Transversal Norte and Transversal Sur). The dip of the individual veins varies, with some steeply dipping vein sets to the northwest close to surface, and others dipping shallowly, either close to the surface or at depth. There are three main sets of veins present at La Preciosa, including:

- Martha veins dipping at moderate angles (approximately 20° to 30°) towards the southwest (235° to 260°).
- Steeper dipping veins sitting above Martha (approximately 55° to 70° dip towards around 260°).
- Transverse veins (50° dip towards 170°).

Details of each of the defined veins, including volume, strike length, thickness, orientation, and depth below surface are given in Table 7.1.

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Table 7.1 Vein volumes and orientations

Vein	Volume (m3)	Strike length (m)	Down dip width (m)	Thickness (m)	Dip	Strike	Depth below surface (m)
Carmen (2 lenses)	274,400	450	50 - 125	5	80	290	30 and 165
Gloria	743,600	900	200	4	80	350	0 to 50
Gloria Rama	162,300	380	160	3	60	350	0 to 50
Nieta	137,700	280	100	4	60	340	0
Pica (2 lenses)	543,600	900	200 - 400	2	60	350	50 and 100
Martha Alta	15,217,800	3,100	600 1,400	5-15 (average 5)	20	320	0
Martha Baja	2,045,800	1,400	200 1,000	3	20	330	0
Martha Media	1,030,000	580	200	9	40	360	185
Martha Media Alta	187,700	290	120 180	5 10 (average 5)	20	360	200
Martha Ramas (7 lenses)	978,100	2,150	100 220	2	10 30	360	30 to 300
Transversal Norte	242,600	500	150	2.5	45	075	0
Transversal Sur	423,700	650	150 400	3	50	090	0
Abundancia (2 lenses)	2,402,100	1,700	300 500	4	40	360	0
Alacran	244,000	500	250	2	30	350	60
Esperancita	165,600	450	200	2	45 70	330	0
Luz Elena (2 lenses)	237,300	550	300	2	30 and 65	350	50 and 150
Nueva	292,900	530	400	1.5	5	350	35
Olin	71,900	125	200	3	30	350	160
Sur	218,500	530	175 250	2	20	350	10

8 Deposit types

Information in this section is excerpted and updated from MDA (2009) and Coote (2010).

Mineralisation at the Property is hosted by quartz-carbonate-barite veins. The occurrence of adularia and the style of early quartz and chalcedonic quartz replacement amongst wall rock replacement and fracture-fill cement assemblages confirm that silver and base metal mineralisation is associated with low to intermediate sulphidation epithermal style systems typical of the Mexican silver belt.

Significant widths of mineralised quartz and carbonate dominated fracture fill and breccia cement assemblages have developed as a result of extended episodes of hydrothermal fluid flow and repeated rupturing of wall rock and pre-existing vein/cement assemblages. Multiple stages of silver and base metal mineralisation are associated with repeated fluid boiling and mixing events, defined by crustiform banded fill/cement assemblages.

The Martha vein, which contains the principal mineral resources at La Preciosa, is a low angle structure partly localised at the contact between sedimentary rocks (conglomerate and sandstone) and volcanic rocks (andesite/andesite clastic rocks).

The geology and style of mineralisation at La Preciosa are similar to those of other silver producing districts in the western Americas. México is host to many silver-gold mining districts, alternating with Peru as the largest silver producer in the world.

9 Exploration

Information in this section is excerpted and updated from MDA (2009).

No significant exploration has been reported on the Property prior to 1981, when Luismin began working on the Property. Only limited exploration took place between 1982 and 2004, when Orko acquired the Property.

9.1 Exploration by Luismin from 1981 to 1982 and 1994

Luismin conducted exploration between 1981 and 1982, including detailed channel sampling of surface outcrops in the main vein systems and Zona Oriente (now believed to be a surface expression of the Martha vein), geophysical surveying of a single IP resistivity line across the north end of La Preciosa Ridge extending eastward to Zona Oriente, and diamond drilling of the Abundancia and La Gloria veins. There are no available details of the results of the geophysical programme although it is reported that a weak response was interpreted for the area of the Abundancia and La Gloria veins and a weaker response was interpreted on the silicified mounds of Zona Oriente. The low sulphide content and low contrast between quartz veins and enclosing silicified andesite could be attributed to the lack of a stronger geophysical response. The 1981 and 1982 drilling programme comprising 7 drillholes and the 1994 programme comprising a single drillhole were reportedly terminated due to falling metal prices. More details on the diamond drilling programme are given in Section 10.

Luismin enlarged approximately 60% of the drifts to 3 m high by 3 m wide on the main 2065 m level within the Abundancia and La Gloria vein underground workings to provide access for trackless mining and drilling equipment. Underground channel samples were selected at 2 m to 3 m intervals along the drifts. In addition, 450 m of underground workings along the Abundancia vein and 408 m along the La Gloria vein were chip sampled. In total, 1,365 chip samples were collected underground by Luismin and 406 channels are preserved in the current drilling database.

Luismin staff prepared several historical internal mineral resource estimates which relied heavily on underground chip sample assays and only limited drilling. The results of these estimates are no longer relevant.

9.2 Exploration by Orko from 2004 to 2008

In early 2004, Orko commissioned an independent property examination which included the collection of eight surface rock samples for verification of historic data. Later in 2004, the Property was mapped at a scale of 1:5000, identifying target areas for more detailed work.

In January 2005, SJ Geophysics Ltd. of Delta, BC, Canada conducted a three dimensional IP resistivity and chargeability geophysical survey. The survey took place over the north part of the main structures at Mina La Preciosa, extending across the central valley and the eastern vein-breccia (Zona Oriente) and northward to Cerro Prieto and the northern projection of the main structures. 40 line-kilometres were run at 100 m line spacing and 25 m station spacing. A weak geophysical signature was noted on the shallow, near surface response high on the ridge where veins are known, but the resistivity method was not successful beneath the basalt cover to the north. A large, multi-line chargeability anomaly was delineated beneath the basaltic cover in the valley between La Preciosa Ridge and Zona Oriente.

A geochemical sampling programme was conducted over the southern part of La Preciosa in 2008, with the grid starting on Cerro El Venado and heading southward on east-west lines. On the ridges the line spacing was 100 m and the sampling spacing was 25 m along lines, and in flat-lying areas, the sample spacing was 50 m. The entire grid runs 5 km north-south and averages 2 km wide east-west. 1,167 soil samples were collected from the B soil horizon. Strong multi-element geochemical anomalies were found for the Veta Nueva, El Orito, El Orito Norte, and Nancy veins.

Diamond drilling by Orko commenced in March 2005 on La Preciosa Ridge, targeting the Abundancia and La Gloria veins and intersecting the Luz Elena vein at depth. Other holes drilled during 2005 targeted the Zona Oriente area located 1 km northeast of the La Preciosa Ridge historic mine areas. The 2006 drilling campaign demonstrated continuity of vein mineralisation at La Preciosa Ridge from the south at Abundancia and to the north at Esperancita. Drilling also intersected the up dip portions of the Luz Elena vein to the east. The Martha vein was discovered in drillhole BP06-77 during late 2006 and later drilling in 2007 targeted both the shallow La Preciosa veins and the deeper Martha vein. The Martha vein was the primary target for the mid to late 2007 drill campaign, which extended the limits of the vein to the south and further up dip to the east. Also during 2007, Orko re-entered and extended several holes drilled in 2006 which were completed just above the projected Martha vein depth. By this time, the Martha vein had been expanded to cover an area of 2,000 m in the north-south direction and 1,600 m in the east-west direction, remaining open to the south and west. The northern extension has not been found and the eastern end outcrops or else abuts the recent basalt. In 2008 the principal drilling target was the up dip eastward side of Martha as well as along strike to the south-southeast.

After compiling the drilling data from each drilling campaign, Orko prepared a series of five mineral resource estimates from 2006 to 2008. The results of these mineral resource estimates have been superseded by the updated mineral resource estimates presented in Section 14.

9.3 Exploration by Pan American from 2009 to 2010

Since Pan American commenced operating the La Preciosa Project, exploration efforts concentrated on drilling 331 drillholes, mainly as infill drillholes to the north of the deposit. Details of this drilling programme are given in Section 10. The database as of 25 October 2010 comprised 726 drillholes for 238,864 m and was used to prepare the updated mineral resource estimates given in Section 14.

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10 Drilling

Information in this section has been updated from MDA (2009).

10.1 Drilling summary and database

All drill holes on the Property have been diamond core, of varying diameters, and drilled by Luismin, Orko, or Pan American. The database as of 25 October 2010 comprised 726 drillholes for 238,864 m. Most holes are oriented from west to east at varying dips, depending on the target vein orientation, to optimise the drillhole intersection with the vein, and therefore the down hole length of the drill intersection is close to the true thickness of the vein. The mineral resource estimation methodology is not sensitive to intersection angle.

Holes have been drilled over targets for a combined strike distance of over 9 km, and the majority of the drillholes on the Property have been used for geological interpretation and estimation of Inferred and Indicated mineral resources at La Preciosa, covering an area of 3 km by 2 km.

Table 10.1 shows details of the drillhole database (as well as the channel samples) by operator and by prospect as at 25 October 2010. A location plan of the drillholes by operator available in the database as at 25 October 2010 relative to the lease boundaries and the mineral resource area is shown in Figure 10.1. A representative cross section at 2701980mN showing the orientation of drillholes relative to the dip of the veins is given in Figure 10.2.

Table 10.1 La Preciosa Property drillhole and channel database

Operator	Prospect	Years	# holes /channels	Metres	Hole number prefix
Luismin	La Preciosa	1981 1982 and 1994	7	1,319	BP
Luismin	La Preciosa channel samples	1981 - 1982	406	867	CG, CGV, CA, CAV
Orko	Orito	2006	7	2,326	BO
Orko	San Juan	2007	8	3,556	SJ
Orko	La Preciosa	2006	1	451	BC
Orko	La Preciosa	2005	6	1,910	BB
Orko	La Preciosa	2005 to 2008	366	144,125	BP05 to BP08
Pan American	La Preciosa	2009 to 2010	331	85,177	BP09 to BP10
Total			1,132	239,733	

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Figure 10.1 Location map of drillholes relative to mineral resource area

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Figure 10.2 Cross section at 2701980 showing drillhole orientation relative to vein orientations

10.2 Drilling by Luismin from 1981 to 1982 and 1994

There are currently 7 drillholes for 1,319 m in the La Preciosa database. Two holes were drilled from underground and five were drilled from the surface. The primary targets were the Abundancia and La Gloria veins at 50 m to 75 m below the primary underground workings on the 2065 level. A final hole 313 m in length was drilled in 1994 in the eastern vein breccia system, but the data for the drillhole is not in the database. The hole reportedly intersected a series of variably silicified zones and veinlet stock work with elevated silver, gold, lead, zinc, and mercury values, but did not completely intersect the structure. There are no available details on the drilling procedures, except that the drill core was either of BQ or AX size. The remaining half core is stored in the original core boxes on site.

10.3 Drilling by Orko from 2005 to 2008

Orko began drilling on the Property in March 2005, ultimately completing 388 diamond drillholes for 152,368 m, spaced on roughly 100 m centres and with all but 16 of the holes targeting various veins at La Preciosa. Orko contracted Major Drilling International for all of the drilling completed on the Project, using Longyear 44, 38A, and 38B machines. Drill core diameter started at HQ size with reduction to NQ size at around 260 m down the hole. Between rod runs, the drillers inserted a wooden block marked with the down hole depth in both feet and metres. Downhole surveys were taken approximately every 50 m down the hole with a Reflex survey instrument, and the results of the surveys indicate moderate deviation in bearing and dip down the hole. No down hole survey is available at the collar of the drillhole.

Drill core was collected on a daily basis from the drill rig by Orko technicians, who taped the boxes shut prior to transport to the site core shed. Once at the shed, the technicians cleaned the boxes and core, and marked the boxes with the hole number, box number, and the depth intervals, and reconciled them with the depths marked on the driller s depth blocks.

After the drillhole was completed, a PVC pipe was placed in the hole and a cement block was installed on the collar. The cement block was clearly inscribed with the name of the drillhole, the final hole length, and the bearing and dip of the hole. An independent surveyor was contracted to survey the coordinates of the collar on a regular basis.

10.4 Drilling by Pan American from 2009 to 2010

Pan American began drilling on the Property in June, 2009 and as at 25 October 2010 had completed 331 diamond drillholes for 85,177 m. The drilling focussed on infilling the 100 m centres completed by Orko to 50 m centres over an approximately 800 square metre (m2) area located to the north and northwest of the deposit. Additionally, confirmatory infill holes were drilled on section elsewhere over the mineral resource, as well as two 15 m to 20 m close-spaced drill crosses to assess the short range continuity of geology and mineralisation. The same drilling contractor employed by Orko, Major Drilling International, was engaged by Pan American to carry out the drilling programme and similar drilling and down hole surveying procedures were followed, although greater capacity drill rigs were employed which resulted in fewer NQ sized drillholes. From early 2010 selected drillholes were surveyed using a Reflex ACT/QPQ orientation tool to obtain oriented drill core for geotechnical purposes. The preservation of the drill collar and survey of the collar coordinates follow the same procedure established by Orko.

10.5 Exploration targets

In addition to extending the mineralisation limits of the known veins, there are a number of other smaller and lower priority exploration targets located outside of the area of the deposits comprising the mineral resource. Three of these targets, including the Nancy, El Orito Norte, and Mina El Orito veins, have been tested by diamond drillholes while the remaining targets have been identified by surface mapping and anomalous soil geochemistry results. A plan of these target areas is shown in Figure 10.3 and a summary of the drilling results is given in Table 10.2.

The Nancy vein is a structure located in the Western Structural Trend located about 1.4 km to the southwest of the Martha vein. The initial discovery of the surface outcrop of a 3 m wide northern extension of the vein was made by Orko in 2006. The target was followed up with soil and trench sampling which yielded anomalously high silver values. In 2010 Pan American drilled 12 holes spaced on a 100 m by 100 m grid. Eleven of these holes intersected a continuous flat lying vein with a strike length of 300 m and a width of 200 m at between 50 m and 100 m below the surface.

El Orito Norte is a continuation of a vein interpreted to connect Mina El Orito in the south to Veta Nueva in the north, and was discovered by Orko in 2004. The vein is located about 250 m to the southwest of the Martha vein. In 2011 Pan American drilled 11 holes on a roughly 100 m by 100 m grid spacing and intersected a moderately dipping vein (approximately 60°) located between 175 m and 275 m below surface in ten of the drillholes.

Mina El Orito is located at the southern end of the main structural trend and dates from before the Mexican Revolution. The old workings are not accessible but rock piles and a series of shafts can be traced over a strike length of approximately 500 m. In 2006 Orko drilled six holes on a 100 m by 100 m grid spacing, but no spatially continuous anomalous metal values have yet been intersected. The best drillhole, BO06-01, intersected anomalous metal values of 43 ppm Ag and 63.7 ppm Au over a down hole interval of 0.60 m.

Other targets identified by surface mapping and soil geochemistry include Veta Nueva, Nancy Sur, La Plomosa Sur, and Dany. Veta Nueva, located south of Cerro El Venado on La Preciosa Ridge in the Zona Sur area, was discovered by Luismin in the 1980s and is exposed in the north wall of a steep valley cut which separates Zona Sur from El Orito Norte. Nancy Sur is located 350 m to the southeast of the Nancy vein and is interpreted to be a southern extension of the Nancy vein. La Plomosa Sur is located to the west on the ridge above the Nancy vein in the Western Structural Trend. A very old open pit is present at the top of the ridge. Dany is located south of the La Plomosa Sur area.

Figure 10.3 Plan of exploration targets

Table 10.2 Downhole intersection results from exploration targets

Target	Hole number	Downhole length (m)(1)	Ag (ppm)	Au (ppm)
Nancy	BP10-588	0.90	91	0.12
	BP10-590	0.35	254	0.03
	BP10-592	6.70	67	0.08
	BP10-593	3.30	42	0.18
	BP10-595	2.50	157	1.12
	BP10-597	3.40	34	0.06
	BP10-601	0.80	104	1.46
	BP10-602	1.40	34	0.10

Target	Hole number	Downhole length (m)(1)	Ag (ppm)	Au (ppm)
	BP10-604	0.95	100	0.14
	BP10-606	1.50	41	0.17
	BP10-608	1.10	36	0.08
El Orito Norte	BP10-562	1.30	20	0.03
	BP10-568	2.05	2	0.02
	BP10-576	2.30	142	0.02
	BP10-584	14.30	69	0.02
	BP10-591	0.30	35	0.04
	BP10-594	0.40	12	0.05
	BP10-598	6.35	168	0.12
	BP10-603	1.15	149	0.09
	BP10-612	1.70	29	0.06
	BP10-614	8.20	84	0.11

Note(1): The down hole length is not equal to the true width of the intersection, which varies depending upon the orientation of the drillhole as it intersects the mineralised zone. Holes are planned to intersect the zones as close to perpendicular as possible, so the true width may be approximately 5% narrower than the down hole length.

10.6 Material impact on accuracy and reliability of drilling results

MDA (2009) examined core recovery of Orko drillholes in detail and QG examined all drillholes available as at 25 October 2010. Both MDA and QG determined that core recovery is generally poorer in mineralised lithologies, but that within mineralised material, there is no obvious or strong correlation between recovery and silver grades.

Both Orko and Pan American use the same methodology for recording core recovery. Between each drill rod run, the diamond drillers insert a block in the core tray indicating the down hole depth of the drill rods. The difference in the down hole depths between two consecutive depth blocks gives the length of each drill run and depending on ground conditions, the length of each drill run may not always be equal to the full length of the drill rod. Geological technicians determine core recovery by measuring the total length of recovered core in the core tray between the drill runs and dividing by the length of each drill run.

Measuring drill core recovery between drill runs is problematic when sample intervals do not coincide with the drill run depths. Usually sample intervals are selected with respect to geological boundaries and are usually selected at intervals less than the length of the drill run. This results in a repetition of the core recovery measurement between consecutive samples and prevents knowing exactly over which assay interval any core loss occurred. The number of these repetitions is relatively small, but to eliminate the problem entirely, measuring drill core recovery on a sample by sample basis is recommended for future drilling campaigns to better assess whether there is any correlation between sample recovery and silver grade.

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The lowest sample recovery of 86% occurs in vein lithologies. There is no way to determine how the core loss occurred for each drill run, but it can occur due to broken ground conditions, the presence of voids within the rock mass, or poor drilling practices.

10.7 Conclusions and recommendations

No special measures were taken to improve drill core recovery except changing the amount of additives such as polymers and drilling mud and occasionally drilling shorter runs when the geologist was present at the rigs. Considering that drill core recovery averages approximately 86% in vein material, and that this 14% loss of material is one of the major factors affecting confidence in the mineral resource estimates, more care should be placed on improving drill core recovery within the vein systems, perhaps with the use of triple tube drilling methods.

11 Sample preparation, analyses, and security

Information in this section is excerpted and updated from MDA (2009) and Snowden (2011).

11.1 Sampling by Luismin from 1981 to 1982 and 1994

There are no documented details on the sampling methodology, approach, and security measures employed by Luismin. The only detail known from visual inspection of the drill core is that samples were split with a screw-wedge core splitter and that the drill core was clearly marked and neatly stored. The number of samples selected and assayed by Luismin, approximately 130 samples, comprises a very small proportion of the database. The lengths of these samples are variable and were probably selected according to geological features, with the majority of the samples ranging from 0.5 to 2.0 m in length. The only known detail about the laboratory sample preparation and analytical methods undertaken by Luismin is that the samples were sent to the Luismin laboratory in Durango.

11.2 Sampling by Orko from 2005 to 2008

11.2.1 Sample preparation and security

Orko technicians collected the core trays from the drill rig on a daily basis. Once at the shed, the technicians cleaned the boxes and core, and marked the boxes with the hole number, box number, and the depth intervals, and reconciled them with the depths marked on the drillers depth blocks. For each drill rod run, the technicians recorded core recovery as a percentage between 0 and 100%. The technicians also recorded a variation of the rock quality designation (RQD which is a measure of the unbroken drill core segments in lengths of 10 centimetres (cm) or greater, recorded as a percentage between 0 and 100%) for each drill rod run, by measuring the unbroken drill core segments in lengths of 15 cm and 20 cm. Each core tray was then photographed.

Once the geologist was ready to log the hole, the drill core was laid out on racks for logging and the geologist recorded the lithological description, a graphic lithology/structural log, alteration and mineralisation type and strength, structural notes, sample numbers, sample depth intervals, oxidation and sulphide percentages, and codes for rock type, structure, and vein code by hand on paper logging sheets.

Once the logging was complete, the geologist marked the sample intervals on the drill core with respect to geological features and marked a cutting line on the long axis of the drill core. The sample interval and the corresponding sample number were also marked on the ribs of the core box in permanent marker for later reference. The majority of sample lengths are between 20 cm and 3 m and within the higher grade zones, sample lengths are typically between 20 cm and 2 m. The core was sawn in half by an Orko technician using a diamond bladed saw, following the line marked on the core by the geologist. After cutting, one half of the sample was placed in a plastic sample bag with a sample tag printed with the corresponding sample number and the remaining half was placed back in the drill core tray.

After sampling, the boxes with the remaining half drill core were stored on numbered racks in a large, well lit, and secure core shed where they currently remain. The samples in the plastic sample bags were then allowed to air dry, then measured for density,

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returned to the plastic sample bag, and placed in a large rice bag. The weight of each rice bag was recorded and the bag was driven by Orko personnel to the sample preparation facility.

From 2005 to 2007, most of the first approximately 100 drillholes (up to BP07-093) were prepared by the primary commercial laboratory, SGS Minerals Services (SGS) in Durango, or else at the secondary commercial laboratory, Inspectorate America Corporation (Inspectorate), also in Durango. After the first 100 drillholes, the primary laboratory was switched to Inspectorate in an attempt to increase sample assay turn-around times. The majority of the samples were prepared by Inspectorate. Check assays were sent to Inspectorate, SGS, and ALS Chemex in North Vancouver, BC, Canada, or in Reno, Nevada, USA.

Neither the Inspectorate or SGS sample preparation laboratories in Durango have been certified by any standards association, but SGS s analytical laboratory in Toronto, Canada has been accredited with ISO/IEC 17025 and Inspectorate s analytical laboratory in Reno, Nevada, USA, is ISO 9001:2008 certified.

11.2.2 Analytical methods

At the preparation laboratory, samples were numerically ordered, crushed, and a sub-sample was taken for pulverisation. The remaining crushed sample was returned to its plastic bag and placed in its sack. This portion, known as the coarse reject, was picked up from the preparation laboratory and returned to the Property site for safe-keeping. The pulp from the pulverisation process was placed in a paper packet and boxed for shipment to the SGS analytical laboratory in Toronto or else to Inspectorate in Reno. A summary of the analytical techniques used to assay Orko s samples is given in Table 11.1. The differences in the assay techniques and the resulting lower and upper limit thresholds shown in Table 11.1 may result in differences in grades obtained by the various laboratories. For example, a four acid digest may result in higher assay values at lower grade ranges when compared to a three acid digest method, but may not show any marked difference at higher grades (such as above 300 ppm Ag).

Table 11.1 Summary of analytical techniques used to assay Orko s samples

Laboratory	Element	Analytical method	Lower upper limit (ppm)
SGS	Au < 10 ppm	FA-AAS	0.005 10
	Au > 10 ppm	FA-GRAV	>3
	Ag <300 ppm	3 acid digest with AAS finish	0.3 - 300
	Ag > 300 ppm	FA-GRAV	>300
Inspectorate	Au	FA-GRAV	> 3
	Ag < 200 ppm	4 acid digest with ICP finish	0.1 200
	Ag > 200 ppm	FA-GRAV	> 200
ALS Chemex	Au < 10 ppm	FA-AAS	0.005 10

	Au > 10 ppm	FA-GRAV	0.05	1,000
	Ag	FA-GRAV	5	10,000

Note: FA is fire assay, AAS is atomic absorption spectrometry, GRAV is gravimetric finish, ICP is inductively coupled plasma.

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All samples were also run through an inductively coupled plasma (ICP) analysis for 40 element spectral determination after a strong acid digest. This method reports values well for elements in minerals which are digestible in the acid, however, mineral resistant to the acid may give only partial values. Oxides, sulphides, and carbonates yield full digestion and thus the base metal values are accurate. Silver has a 10 ppm upper detection limit and is thus not used for the Project where significant numbers of sample grades exceed this value. The base metals Pb and Zn have an upper threshold of greater than 10,000 ppm (1%) and only those exceeding 1% require analysis by alternate methods.

11.2.3 QAQC

Orko systematically inserted QAQC samples into the sample stream throughout their drilling programmes, including blanks and standards. Generally every tenth samples comprised a standard or a blank sample. Duplicates were not included in the sample stream but were submitted in separate batches to an alternate laboratory as a check on laboratory bias.

Blanks

Standard industry practice is to submit blank samples which comprise a full volume material known to be free of mineralisation relevant to the deposit at grade levels well below the detection limit of the analytical machine. Blanks are used to detect sample switching and contamination during crushing, pulverising, and analysis, and should be submitted at a frequency of one for every 20 geological samples, with preference placed within the mineralised zones. A blank sample is considered to have failed and the sample batch may be considered contaminated if the assay result is greater than ten times the detection limit of the analytical machine.

Orko manufactured a number of blank samples from basalt drill core and from basalt boulders found in nearby fields. The blanks used on the Project were named Orko-2, Orko-4, Orko-5, Orko-7, and Orko-9. Silver data from 1,030 Orko-2 samples submitted to SGS and silver and gold data for 97 blank basalt drill core submitted to Inspectorate are present in the database. One blank was submitted as every 20th sample.

4% of the Orko-2 blank samples and 10% of the basalt drill core samples exceeded the practical detection limit for silver. None of the basalt drill core samples exceeded the practical detection limit for gold. Smith (2010) notes that the blank samples were not considered completely blank. As such it is not possible to determine whether the failed samples are a result of contamination or background silver grade in the blank samples, and therefore the blanks are unreliable for assessing whether contamination is an issue in the sample preparation and analytical laboratories. However, the magnitude of the failures indicates that there is no significant concern with sample contamination.

Standards

Standard industry practice is to submit certified standards, which comprise material collected either from site or purchased from a commercial laboratory. The material is prepared by a laboratory and sent to an appropriate number of round robin laboratories who conduct a sufficient number of analyses, using consistent sample preparation and analytical techniques (and consistent with the methods intended for processing the site samples) to certify the sample grade within known error limits. Standards are used to assess the accuracy of sample grades and should be submitted at a frequency of one for every 20 geological samples, with preference placed within the mineralised zones. Standards should be submitted at all relevant grades, for all economic elements reported in the mineral resource, including the cut-off, low, average, and high grades. Standard samples should be monitored on a real time basis, plotted on a chart over time, and assessed for bias and failure. A standard sample is considered to have failed if the assay value is greater than or less than three standard deviations of the certified value.

Orko primarily used standards prepared by SGS from material obtained at the Project site, named Orko-1, Orko-3, Orko-6, and Orko-8. The accepted values and standard deviations for the Orko standards for gold and silver are shown in Table 11.2.

The standards were problematic because they were not certified, record keeping is reported to have been inadequate and therefore the identity of the standards was not always certain (MDA, 2009), the standard deviation of some of the standards are unusually high (such as silver for Orko-1 and gold for all standards), and some of the accepted values are very similar to other standards. Therefore it is unclear whether any observed inaccuracies are due to inconsistencies in the standard itself, mislabelling of the standard during sample submission, or errors in the assay procedure.

A plot of the standard results on the same set of axes colour coded according to the assigned standard name shown in Figure 11.1 suggests that there may be some incorrect standard labelling, especially for some of the Orko-3 silver results and possibly for some of the Orko-8 silver results. Mislabelling of gold standards appears to be evident however the failed results are not mirrored in the silver results. As a result of the similarity between the acceptable limits for Orko-6 and Orko-3, it is not clear if sample mislabelling has occurred.

A total of 3,994 Orko standards were inserted into the sampling stream for a 4.5% insertion rate.

Table 11.2 Details of the standards used by Orko

	Au		Ag	
Standard name	**Accepted value (ppm)**	**Standard deviation (ppm)**	**Accepted value (ppm)**	**Standard deviation (ppm)**
Orko-1	0.210	0.0217	293.40	11.31
Orko-3	0.068	0.010	112.00	6.60
Orko-6	0.072	0.009	146.10	11.10
Orko-8	0.134	0.030	237.90	14.98

Figure 11.1 Graphs of all Orko standards for silver and gold

Orko-1 standard

A total of 564 Orko-1 standards were assayed during the Orko drilling programme and the results are shown in Figure 11.2. 99% of the gold values are within acceptable limits and bias is insignificant. The mean obtained for silver from the standard results is

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278 ppm Ag which is significantly lower than the expected mean value of 293 ppm Ag, however, 94% of the silver values are within the acceptable limits. If the accepted silver value for this standard has been correctly defined, then the assay results associated with this standard may be considered conservatively low.

Figure 11.2 Graphs of the Orko-1 standard results for silver and gold

Orko-3 standard

Orko-3 standards were submitted for analysis to both Inspectorate and SGS. The results from Inspectorate are shown in Figure 11.3 and from SGS in Figure 11.4. 851 Orko-3 standards were submitted to Inspectorate while 95 were submitted to SGS. 99% of the values from both laboratories plot within the acceptable limit of the certified value for gold.

96% of the silver values by Inspectorate and 99% of the silver values by SGS are within the acceptable limits. No bias is noted in the Inspectorate data while a consistent bias in the SGS data is noted towards lower values than those expected from the certified data.

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Figure 11.3 Graphs of the Orko-3 standard results for silver and gold by Inspectorate

Figure 11.4 Graphs of the Orko-3 standard results for silver and gold by SGS

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Orko-6 standard

1,382 Orko-6 standards were assayed during the Orko drilling programme and the results are shown in Figure 11.5. All of the gold values plot within the acceptable limits of the certified value for gold and 99% of the values plot within the acceptable limits of the certified value for silver.

Figure 11.5 Graphs of the Orko-6 standard results for silver and gold

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Orko-8 standard

1,102 Orko-8 standards were used in the Orko drilling programme and the results are shown in Figure 11.6. Nearly all of the gold data falls within the acceptable limits of the certified value and no bias in the results is evident.

Nearly all of the silver data falls within acceptable limits of the certified value. A significant bias is evident in the silver results with results from early batches plotting above the acceptable limits of the certified value and later batches plotting below the certified value.

Figure 11.6 Graphs of the Orko-8 standard results for silver and gold

Duplicates

Submitting duplicate samples, comprising either field duplicates (half drill core), coarse rejects, or pulp duplicates to the primary laboratory allows for assessment of the errors introduced during sample preparation and analysis and also provides an indication of the nugget effect or inherent variability in sample grades at very close distances. Sampling duplicate samples to an alternate laboratory allows for assessment of error and bias between laboratories, but it is impossible to know whether any differences in duplicate grades are attributable to error or bias. In order to make meaningful comparisons of original and the duplicate sample grades, both the primary and check laboratory must use consistent sample preparation and analytical techniques. Standard industry practice is to submit duplicate samples on the order of one for every 20 geological samples, or 5%.

Given the fine grain size and relatively small reduction in volume during sampling, pulp duplicates are expected to have good precision, while the coarser grain size of coarse rejects is expected to result in less precise values. Plotting the duplicate data on a quantile-quantile (Q-Q) can be used to demonstrate sample grade precision and any biases in the sample grades.

Several stages of duplicate analysis were undertaken by Orko and these are tabulated in Table 11.3. A total of 1,104 duplicates or 1% of total samples has been submitted for the Orko drillholes. The duplicates comprise both coarse reject and pulp duplicates. The duplicate sampling programmes are problematic in that the analytical technique (and therefore the upper and lower grade limit thresholds) used by the check laboratory was not always consistent with the analytical technique used by the primary laboratory, and therefore differences in the grades will be expected. For duplicates submitted to the same laboratory as the primary laboratory for both Inspectorate and SGS samples, good precision and no bias is noted.

Table 11.3 Details of duplicate analyses of Orko drill core samples

Description	Number of duplicates reported	Number of duplicates in database	Type	Primary Lab	Secondary Lab	Comments
Orko duplicates	331	317	Pulp	Inspectorate	ALS Chemex	Samples from five drillholes. Orko holes only.
3 Lab comparison	134 (per lab)	120 (per lab)	Coarse Reject (SGS) Pulp (ALS Chemex)	Inspectorate	SGS, ALS Chemex	All samples from drillhole BP07-102.
MDA pulps	240	236	Pulp	SGS, Inspectorate	ALS Chemex	Suite of samples from each mineralised vein Orko holes only.
MDA coarse rejects	267	267	Coarse Reject	SGS, Inspectorate	IAC	As above using the same sample numbers as the MDA pulps Orko holes only.
Martha		146	Pulp	SGS, Inspectorate	SGS	All samples from Martha vein only. Pan American and Orko holes. Done because of problems in correlating mineralisation over short distances between Orko and Pan American holes.

Orko duplicates

Orko submitted 331 pulp duplicates to ALS Chemex in North Vancouver, BC, Canada as a check on the original assay results provided by Inspectorate. Results for 317 of the pulp duplicates are available in the database. Of these assays, 79 samples were analysed for silver using the same technique at both laboratories (fire assay with gravimetric finish for samples greater than 200 ppm Ag). Samples less than 200 ppm Ag were analysed by a four acid digest with ICP finish at Inspectorate and samples less than 300 ppm Ag were analysed by three acid digest with atomic absorption spectrometry (AAS) finish at SGS. Assessment of the silver results on a Q-Q plot show a bias towards higher grades for the Inspectorate results for grades less than approximately 30 ppm Ag and a good correlation greater than 30 ppm Ag. Gold was analysed by fire assay with gravimetric finish at Inspectorate and by fire assay with AAS finish at SGS. Analysis of the 317 gold data on the Q-Q plot shows the results to be within acceptable limits with a bias towards higher values in the Inspectorate results. The Q-Q plots for silver and gold assays of the pulp duplicates are shown in Figure 11.7.

Figure 11.7 Q-Q plots comparing silver and gold assays of pulp duplicates from Inspectorate and ALS Chemex

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Three laboratory comparison

A comparison of the assay results obtained from the three laboratories used throughout the Orko drilling programme was carried out. 134 samples from drillhole BP07-102 initially analysed by Inspectorate were submitted to SGS and ALS Chemex of North Vancouver, BC, Canada. SGS analysed a coarse reject duplicate while ALS Chemex analysed a pulp duplicate from the same sample. Results from 120 duplicates are available for the two laboratories.

Q-Q plots of the results between Inspectorate and SGS show a very good correlation between the duplicate and the primary analyses with no bias noted between the results for either silver or gold.

Analysis of the results between Inspectorate and ALS Chemex on a Q-Q plots for silver and gold show the data to be within acceptable limits for both silver and gold with no significant bias for gold and a trend towards higher values in the Inspectorate data at higher silver grades.

MDA pulp duplicates

MDA (2009) completed duplicate analysis of samples from each of the mineralised vein intercepts to increase confidence in the Orko analytical data. Samples were originally analysed at SGS and Inspectorate and were sent to ALS Chemex in Reno for analysis. A total of 240 pulp samples, 179 from Inspectorate and 61 from SGS, were submitted for re-analysis. MDA inserted blanks and standards obtained from WCM Minerals in British Columbia into the sample stream. Two of the ten blanks reported failures for gold but none failed for silver, and all of the standards performed within the accepted limits.

Analysis of the silver results between Inspectorate and ALS Chemex on a Q-Q plot shows a good correlation between the duplicate and the primary analyses with little bias between the two laboratories. The Q-Q plot for gold shows a good correlation for values greater than 0.20 ppm Au and a bias to higher values from ALS Chemex for grades less than 0.20 ppm Au.

Analysis of the silver results between SGS and ALS Chemex on a Q-Q plot shows a reasonable correlation between the duplicate and primary silver and gold analyses. No significant bias is present in either the silver or assay results.

MDA coarse reject duplicates

MDA (2009) submitted 267 coarse reject duplicates from the same list of samples as used for the three laboratories to Inspectorate to ascertain if there were any errors introduced during crushing and pulverising. Of the samples submitted, 192 were originally assayed at Inspectorate and 75 were originally assayed at SGS.

Q-Q plots of the original and duplicate Inspectorate results show a very good correlation between the duplicates and the primary analysis with no significant bias between the results for either silver or gold.

A Q-Q plot of the original SGS silver results and the Inspectorate duplicates shows a slight bias towards higher values obtained in the Inspectorate results between grades of approximately 50 ppm Ag and 200 ppm Ag. The bias could be due to the fact that SGS used a three acid digest for samples less than 200 ppm Ag while Inspectorate used a four acid digest for samples less than 200 ppm Ag. The Q-Q plot for gold shows a good correlation and insignificant bias between the results.

Martha duplicates

The results of the duplicate sampling of Orko s original samples by Pan American are discussed in the following section.

11.3 Sampling by Pan American from 2009 to 2010

11.3.1 Sample preparation and security

The drillhole logging and sampling methodology by Pan American follows much the same protocol developed by Orko. From drillhole BP10-458, RQD was measured on 10 cm intervals only.

The geologists determined the sample positions on the diamond drill core and marked the positions on the core and the core tray. The core was taken to the core cutting area and the trays were stacked until they were ready to be cut. The core was cut along the line defined by the geologist with a water cooled diamond bladed saw. After each piece of core was cut in half, both pieces were replaced in the core boxes.

Once the core was sawn in half, the boxes were taken to the sampling area where sample bags and sample tags were labelled and consecutive sample numbers were assigned to the sample intervals. Every tenth sample number is assigned to a standard or blank sample. Every 50th samples is duplicated. Duplicate splits from pulverised samples are taken from every sample number ending with 49 or 99. The duplicate pulp sample is separately bagged and given the next sample identification number (i.e., duplicate sample numbers end with 50 or 00). The pieces of half core to be assayed were placed in labelled sample bags along with the corresponding sample ticket, and then into labelled rice bags along with the labelled standard and blank samples. Ten samples were placed in each rice bag. The filled rice bags were stored on site until a few drillholes were ready to be sent out as one consignment to the SGS laboratory in Durango. The core samples are driven directly to the SGS laboratory in Durango by a Pan American employee.

11.3.2 Analytical methods

All Pan American samples, with the exception of the pulp duplicates, are prepared and assayed by SGS Laboratories in Durango, México. Pulp duplicate samples are analysed at Inspectorate in Sparks, Nevada.

At the SGS laboratory, the samples are set out in numerical order, individually crushed, then riffle split to provide a sub-sample for pulverising. The crushed material left over after the sub-sample has been removed (the coarse reject) is returned to the labelled plastic bag from which is was taken, sealed, and eventually returned to Pan American for storage at the La Preciosa site. The sub-sample is pulverised and approximately 200 g is placed in a small labelled paper packet. After the required assay aliquots have been removed, the residual material remaining in the packet is also returned to Pan American for storage on site at La Preciosa.

A summary of the analytical techniques and tolerances used by SGS and Inspectorate is shown in Table 11.4.

Table 11.4 Details of the analytical techniques used to assay Pan American s samples

Laboratory	Element	Analytical method	Lower - upper limit (ppm)
SGS	Ag < 300 ppm	Three acid digest with AAS finish	0.3 - 300
	Ag > 300 ppm	Fire assay with gravimetric finish	> 5
	Au < 10 ppm	Fire assay with AAS finish	0.005 10
	Au > 10 ppm	Fire assay with gravimetric finish	> 3
	Ag and 33 other elements	Two acid digest with ICP finish	2 10
Inspectorate	Ag < 200 ppm	Four acid digest with ICP finish	0.1 200
	Ag > 200 ppm	Fire assay with gravimetric finish	5 5,000

Au	Fire assay with gravimetric finish	> 3

11.3.3 QAQC

QAQC samples have been systematically included in the sample stream throughout the sampling programme, including blanks, standards, and pulp duplicates. The intended

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protocol is based on sample number within batches of 50 samples. Sample numbers ending with 10 or 60 are Ag standards, 30 or 80 are Au standards, and 20, 40, 70, or 90 are blanks. Duplicate samples are collected at the pulp duplicate stage by the laboratory. Every sample ending with 49 or 99 is duplicated and the duplicate labelled 50 or 00 respectively. The duplicates are submitted in batches to an umpire laboratory. Pan American geologists regularly monitored the performance of standard and blank assays returned by SGS by plotting values on line graphs in Excel as soon as each batch of assays was reported by the laboratory. Whenever any of the standard results exceeded three standard deviations from the expected value, the entire batch of assays was re-submitted for re-analysis.

Blanks

Blank samples comprise full volume half core basalt. A total of 652 blank samples have been submitted with the Pan American drillhole samples which equates to a 4% insertion rate. Plots of the gold and silver assay results for the blank samples are shown in Figure 11.8. The plots in Figure 11.8 show two samples for gold and five samples for Ag to be above the detection limit. This equates to a failure rate of 0.3% for Au and 0.8% for Ag. These results show that contamination is not a major concern for these elements. The sample insertion rate should be increased to 5% from the current 4%, with the blank samples placed between suspected high grade samples in the veins.

Figure 11.8 Scatterplots of blank assay silver and gold results

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Standards

A total of 662 standards have been inserted into the Pan American sampling stream, equating to a 4% insertion rate. Three different standards have been used, including ORKO-10, GBM908-13, and G308-7 and details for each standard are shown in Table 11.5. All of the standards except for GBM908-13 have been depleted and are no longer in use at the time of this report.

Table 11.5 Details of the standards used by Pan American

	Ag		Au	
Standard	**Certified value (ppm)**	**Standard deviation**	**Certified value (ppm)**	**Standard deviation**
ORKO-10	145.47	4.23	0.057	0.005
GBM908-13	151.4	8.4	n/a	n/a
G308-7	n/a	n/a	0.27	0.02

Standard Orko-10 was prepared by SGS in Durango from material obtained from stockpiles at the Property site. This standard was not certified but round robin testing consisting of five laboratories has produced expected values and standard deviations for this standard. Only 21 of these standards were analysed with drillholes BP09-355 to BP09-364. For both Ag and Au, the assay results demonstrate a bias towards higher values than the expected value. Pan American commissioned SGS Peru to assess the Orko-10 (as well as Orko-9) standard material. SGS Peru concluded that the standard had unacceptably high variances, probably due to the presence of native silver. Pan American discontinued the use of Orko-10 from BP09-365 onwards.

Standard GBM908-13 and G308-7 are both certified standards obtained from Geostats (Pty) Ltd. (Geostats) and replaced Orko-10. GBM908-13 is a base metal standard which has certified values for silver, copper, and sulphur, while G308-7 is certified for gold.

391 GBM908-13 samples and 316 G308-7 samples were submitted with the Pan American drillhole samples. The performance of these standards for silver (GBM908-13) and gold (G308-7) are presented in Figure 11.9. Although analysed standards for GBM908-13 fall within acceptable limits, a consistent bias towards over-reporting of the silver grades is present. It is also noted that the standard deviations for silver grades are very wide and do not provide a tight constraint on the precision of the analysis. G308-7 shows a very slight bias towards under-reporting of the gold grades, however, all results are within two standard deviations of the certified value which indicates a good level of accuracy.

Figure 11.9 Graphs of the GBM908-13 and G308-7 standard results for silver and gold

Duplicates

Pulp duplicates submitted to an umpire laboratory allow for monitoring of grade bias by laboratory. Pan American submitted a total of 321 pulp duplicates (2% of the total number of samples) returned from the primary laboratory SGS to the umpire laboratory Inspectorate in Nevada for re-analysis. No field or coarse crush duplicates have been submitted to the primary laboratory for the Pan American drillholes to assess sample grade precision. Standards were inserted into the duplicate sample stream at a rate of one in ten (10%).

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Silver and gold results shown on the Q-Q plots given in Figure 11.10 indicate a bias towards higher silver grades obtained from the duplicate laboratory (Inspectorate) and a good correlation of gold grades with no bias in the results above the practical detection limit.

Figure 11.10 Q-Q plot comparing silver and gold assays of pulp duplicates from SGS and Inspectorate

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11.4 Density measurements

A total of 84,105 density measurements are available in the current database for La Preciosa. Of these, 65,318 were specific gravity measurements by Orko and the remainder by Pan American. The methodology followed by Orko was to select a half drill core sample weighing between 200 g and 300 g from the sample bag, record the sample weight on an electronic scale (with a precision of 0.01 g up to 500 g), then to place the drill core in a calibrated 2,000 millilitre (ml) capacity cylinder filled with 400 ml of water, and then to measure the displaced water volume. The density value is determined by dividing the sample weight by the volume of displaced water.

MDA (2009) were concerned that the samples were not dried prior to measuring density and that no information was available to demonstrate the dryness of the samples. The greater concern was that the methodology did not account for the presence of vugs (open spaces) in the samples, which will result in an artificially high density measurement. The measurement method followed by Orko does not provide in situ bulk density, which is required for the estimation of tonnes and contained metal, because it was not undertaken on dried samples nor did it take account for any open spaces.

To assess the reliability of the density measurements for mineral resource estimation, MDA directed Orko to undertake the measurement of dry bulk density of 92 pieces of whole drill core from vein material. NQ and HQ sized drill core samples were cut perpendicular to the core axis to form cylinders with lengths between 9.4 cm and 10.6 cm. The cylinders were oven dried in a domestic toaster oven and weighed every hour until the mass of the cylinder had stabilised, with a minimum drying time of 4 hours and a maximum drying time of 8 hours. The cylinders were then weighed on a bench top scale. After a dry weight was obtained, the length and diameter of the cylinders were measured by micrometer, taking into consideration any void space, to determine the volume of the sample. The dry bulk density was then calculated by dividing the dry sample weight by the volume of the sample. Next, the sample was cut in half parallel to the core axis and a specific gravity measurement was made on the sample using the water displacement method normally followed by Orko, in order to compare the results of the dry bulk density measurements and the specific gravity measurement.

The expectation was that the specific gravity value would be higher than the bulk density value, as the vug spaces should fill with water resulting in a lower volumetric displacement of the water in the cylinder, but this turned out to not be the case. All but three of the Orko specific gravity values were higher than MDAs bulk density values on exactly the same sample. Inspection of photographs taken of 72 of the 92 samples indicated that the samples contained very minor surficial cavities (estimated at less than 1%). These results led MDA to conclude that that the inaccuracy of the water displacement technique due to surficial cavities was not a major issue in the samples selected for the study. However, MDA had concerns about the precision with which each of the measurements were made, such as measuring the volume of the core by micrometer variably to the nearest 1 mm or to the nearest 0.01 mm, or by measuring the volume of displaced water to the nearest 1 ml, 5 ml, or 10 ml. MDA concluded that the Orko density values had a high variance but were not biased and were acceptable for the estimation of density as long as the high individual sample variance was considered.

Pan American undertook density testing on approximately 252 samples to determine the method which would result in the most reliable density values by taking four different measurements using different methodologies on exactly the same rock sample. One of the four measurement techniques included data for measurement of a void index which could be used to imply bulk density. 133 samples were measured from vein and near vein silicified material and 119 from un-silicified andesite, which is the most common wall rock to the mineralisation. Oxidation state was considered by assuming oxidation is depth related and each of the two rock types were taken from shallow, middle, and deep locations to reflect the potential highly, moderately, and weakly oxidation states, with approximately 40 samples selected in each of these zones. Samples were selected from intervals weighing between 400 g to 600 g to reduce measurement error, and were mostly selected from samples that had previous Orko specific gravity measurements. Each sample was geologically logged prior to testing and the measurements were made by a well-trained metallurgical technician at the Pan American La Colorado mine metallurgical laboratory. The results of the test work indicated that the use of the Orko specific gravity data is suitable for use in mineral resource estimation as long as a factor is applied to convert to bulk density. This factor is discussed in Section 14.8.

11.5 Conclusions and recommendations

In QG s opinion, the following conclusions and recommendations regarding sample preparation, analyses, and security can be made:

- Sample handling and preparation on site follow industry standard practices. Sample security is aided by the remote nature of the site and by the permanent presence of a caretaker to ensure unwanted intrusion. Once processed, samples are driven directly from site to the sample preparation facility in Durango. No specific anti-tampering precautions are taken between site and the laboratory because the samples remain in Pan American custody during this time. There is no reason to believe that any of the drill core or results have been tampered with.

- The sample preparation and analytical laboratories used are international and certified commercial laboratories. The sample preparation and analytical methods used to determine sample grades are suitable for the mineralisation style and tenor at La Preciosa and the results are reliable for use in mineral resource estimation with due consideration of the QAQC results during mineral resource classification.

- Contamination does not appear to be a major problem during sample preparation of Pan American s drill core, but the blank samples submitted with Orko s drill core shows minor contamination in the results. There is the possibility that the blank material used by Orko was not completely un-mineralised.

The standards included in Orko s sample stream were problematic because they were not certified, record keeping is reported to have been inadequate and therefore the identity of the standards was not always certain, the standard deviation of some of the standards are unusually high, and some of the accepted values are very similar to other standards. Therefore it is unclear whether any observed inaccuracies are due to inconsistencies in the standard itself, mislabelling of the standard during sample submission, or errors in the assay procedure. Standards included in Pan American s sample stream show all results are accurate within two standard deviations of the certified value for both silver and gold, with a slightly high silver bias and a slightly low gold bias.

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- Certified standards at low, medium, high, and cut-off grades with relatively narrow standard deviation ranges, for both silver and gold, should be purchased. The certified values of the various standards should not overlap in grade value to eliminate the possibility that mislabelled standards are the cause of any failed assay results. The certified standards should be sourced from an international certified standard provider or preferably prepared from material obtained from site and prepared by an accredited laboratory for full independent certification. The certified standards should be inserted in the sample stream at a submission frequency of one for every 20 geological samples.

- Duplicate samples from Orko s drilling campaigns show good precision and no bias when the comparison is made between the original and duplicate sample assays prepared and analysed by the same laboratory. When duplicate samples were sent to check laboratories however, the results were mixed and sometimes conflicting. In some cases there was no way to assess which laboratory is more accurate as no standard results for some batches are available. Duplicate samples submitted by Pan American to the check laboratory show that the original sample grades have a low silver bias and no bias is noted in gold.

- Duplicate samples comprising coarse reject samples as well as pulps should be submitted to the primary laboratory to monitor sample precision at the primary laboratory, at a submission frequency of one for every 20 geological samples. Duplicate samples should be focussed on sample intervals in the vein material. Whenever duplicate sample campaigns are undertaken, the check laboratory should use the same analytical method as the primary laboratory.

- During check sampling campaigns, both standards and blanks should be submitted to assess for sample contamination and sample accuracy at the check laboratory.

- All QAQC data should be assessed as assay results are received and any required corrective action should be undertaken immediately.

- Despite the lower performance of some of the QAQC data, it is QG s opinion that the sample data is reliable for use in mineral resource estimation with due consideration of the QAQC results during mineral resource classification. In future, the reliability of sample grades can be demonstrated by favourable results of QAQC samples submitted with infill drillhole samples as the Project advances.

- Bulk density measurements should continue to be made from spatially and lithologically representative areas of the mineral resource as drilling continues.

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12 Data verification

12.1 Data verification by MDA and Pan American

MDA (2009) assessed a re-assay campaign of coarse rejects undertaken by Orko and instigated a re-assaying programme of pulp samples and the results of this assessment are discussed in Section 11.2.3. During classification of the mineral resource estimate disclosed in 2009, MDA attributed as negative factors the grade biases noted between the original and re-assayed grade values and the lack of available Orko QC data to demonstrate the quality of sample data.

To eliminate any concerns about the quality of Orko data, Pan American undertook a specific testing programme of original data by re-assaying drillhole samples and by comparing recent Pan American drillhole sample grades with earlier Orko sample grades, which also showed grade biases.

While the check programmes were not entirely adequately undertaken and some of the results are inconclusive, it is QG s opinion that the check assaying performed by Pan American has not raised any major concerns with the quality of previous drillhole sample data and reinforces the suitability of the Orko data for inclusion into mineral resource estimates classified as either Inferred or Indicated. In future, the reliability of sample grades can be demonstrated by favourable results of QAQC samples submitted with infill drillhole samples as the Project advances.

12.2 Data verification by the current qualified persons

12.2.1 Site visit

Mr. Snider visited the site on 9 June 2011, accompanied by Hernán Dorado Smith, Senior Planning Engineer of Pan American. Mr. Snider reviewed the site and select process plant and tailings locations.

Mr. Finch and Mr. Stewart visited the site on 5 July 2011. Mr. Finch reviewed the potential pit locations, the portal and the dump locations, as well as the general infrastructure and access to the site.

Mr. Stewart reviewed representative drill core intersections of the veins and surrounding host rock located at the core storage facility on the Property site, confirmed the collar coordinates of selected drillholes, visited outcrops of the Martha, Gloria, Abundancia, and Transversal veins, reviewed paper and digital geological interpretations, and reviewed the geological database. At the time of inspection, no drill rigs were operating and no drill core handling processes were observed. Core handling procedures were discussed with Mr. Sergio Morfín, Exploration Manager México, including core mark-up, sample recovery and geotechnical logging, geological logging, sample delineation, sample cutting, insertion of QC samples, and sample dispatch.

The reviewed drill core intersections were chosen to represent the various structures and veins, different generations of drilling, and important high grade intersections. No discrepancies were noted between the information reviewed by Mr. Stewart and the data and information that has been provided by Pan American.

Because a current personal inspection has been conducted by other qualified persons responsible for the preparation of this technical report, and no additional beneficial information would have been derived from a site visit at this stage of the Project, Mr. Hawthorn and Mr. Drielick did not conduct a site visit. Mr. Hawthorn and Mr. Drielick reviewed the available data and consider that it is adequate for the purposes of this preliminary economic assessment.

12.2.2 Data reviews

Downhole surveys

A review of the down hole survey data indicates that a number of down hole surveys are incorrect, caused either by data transcription errors or by errors in the survey measurement. Additionally, no down hole surveys were taken at the collar of each drillhole, and the planned bearing and dip was entered in the database as the actual orientation at the collar. This practice has resulted in greater differences between the bearing and dip at the collar and the first actual down hole survey than is present between consecutive down hole surveys further down the hole. The reason for this discrepancy is that drillholes are rarely collared with exactly the same bearing and dip as is planned. Since nearly all of the drillhole collars are well preserved with a standpipe at the collar, it is recommended that a licensed surveyor take a bearing and dip measurement of all drillhole collars. As well, the down hole survey database should be reviewed for unrealistic differences between consecutive down hole surveys and checked against the values recorded on the driller s log. Any unresolved differences should be noted and a correction applied to the survey to ensure that the location of the drillhole trace is accurately plotted for geological interpretation and mineral resource estimation.

Drillhole collar coordinates

14 collar coordinates were independently confirmed by QG using a hand-held GPS. No discrepancies beyond the accuracy of the handheld GPS were noted between the GPS reading and the values in the database.

Assay database

Original assay certificates from SGS Laboratories in Durango, México and from Inspectorate America Corporation in Sparks, Nevada were emailed directly to QG for comparison against the values in the database. 441 assays from Pan American s drilling and 3,188 assays from Orko s drilling were reviewed. 44 errors were noted in Pan American s assay database, and 41 of these are related to a single assay batch which appears to have been re-assayed. It is not clear whether the values in the database relate to the final, accepted assay or whether the wrong certificate was provided. Two of the three remaining errors relate to inconsistent treatment of below detection limit assays, which has no effect on the estimation of mineral resources, and one assay was entered in error as the threshold of the AAS result rather than the fire assay result. Nine errors were noted in Orko s assay database, seven of which require follow up as it appears that repeat assay values have been entered in the database as the average of the two assays values, rather than the first assay value, as is best practice.

QG recommends that Pan American review all assays in the database for accuracy and ensure consistent treatment of below detection limit assays and repeat assay values. Standard practice is to store all raw assays in the database and to create columns which contain the final assay values for use in mineral resource estimates. Standard practice for entering assays below detection limit assays is to enter half the detection limit of the analytical machine. Standard practice for entering assays with repeat values is to enter the first assay value, unless an error has been identified with the first assay value, in which case best practice would be to use the second assay value.

Any assay certificates that have been superseded by a re-assay of the entire batch, such as in the case of failed QAQC results, should be identified in the database in order to facilitate data reviews.

Data should be stored in an industry standard database and data should be digitally captured using electronic geological logging and assay transfer software to ensure timely and accurate collection of geological data. This will provide robustness in the data capture process and security in the database.

QAQC reviews

Snowden were retained by Pan American to review all available QAQC data and to compile the information in a master database with the corresponding geological log data and with the three dimensional coordinate information, in order to assess the spatial distribution of the QAQC data and results. The results of Snowden s assessment (Snowden, 2011) are documented in Section 11 of this technical report.

12.3 Data adequacy

It is the opinion of the qualified persons responsible for the preparation of this report that the data used to support the conclusions presented in this technical report is adequate for the purposes of this preliminary economic assessment.

13 Mineral processing and metallurgical testing

Luismin initiated the first metallurgical test work on La Preciosa Project samples in 1988 in anticipation of processing mined material at their Avino mine located 20 km to the northeast of the Property. No other test work was undertaken until the modern series of testing from 2007 to 2010. Between 2007 and 2009, testing took place at the Process Research Associates (PRA, now Inspectorate) laboratory in Richmond, BC, and at Westcoast Mineral Testing Inc. (WMT) of North Vancouver, BC. Following the joint venture agreement between Orko and Pan American in 2009, the metallurgical composites were shipped to the SGS metallurgical laboratory in Durango, México, where they were used to prepare a new master composite that was used in the first of four programmes. Subsequently, a second master composite and variability composites were prepared for testing at SGS.

13.1 Historical testing

Whiting (2008) reports that historical Luismin reports have some brief descriptions of a preliminary metallurgical testing and bulk processing of mineralised rock extracted from the Abundancia and La Gloria veins during the slashing of the underground workings. Most of this testing was designed to determine the amenability of the material to serve as process feed to concentration in the Avino mill. In 1988, Comisión de Fomento Minero completed three cyanidation bottle roll tests on 9 kilograms (kg) samples ground to ¾, ½, and 3/8 inch size, as well as a finer 65% passing through a -200 mesh. A cyanide concentration of 0.2% NaCN, with incorporated lime to maintain a pH at 11, yielded the preliminary results shown in Table 13.1.

Table 13.1 Metallurgy test results by Comision de Fomento Minero

Sample Size	**Heads Au ppm**	**Heads Ag ppm**	**Tails Au ppm**	**Tails Ag ppm**	**Recovery % of Au**	**Recovery % of Ag**
-3/4	0.45	254	0.32	240	28.9	5.5
-1/2	0.45	254	0.26	233	42.2	8.3
-3/8	0.45	254	0.20	210	55.5	17.3
65% -200	0.45	254	0.12	41	73.3	83.9

13.2 Testing from 2007 and 2009

Bench scale testing during this period investigated the response to both flotation and agitation cyanidation.

13.2.1 Stage 1 flotation and agitation cyanidation tests

In December 2007, five 10 kg composites, all as nominally 6 mesh drill core assay coarse rejects were shipped to WMT. Summary details of the five composites are shown in Table 13.2. Portions of each of these five composites were used to prepare a single master composite grading 0.4 ppm Au and 300 ppm Ag. This master composite was subjected to a single flotation test by WMT plus three agitation cyanidation tests by PRA.

The remaining portions of the original five composites were sent in mid-2009 to Pan American in Durango, México, where they provided material for the first of four testing programmes undertaken by Pan American.

Table 13.2 Details of the five 10 kg Orko metallurgical composites

Composite name	No. of samples	Ag ppm	Au ppb	Pb ppm	Zn ppm	Cu ppm
Veta La Gloria	26	310	328	1887	2364	108
Veta Abundancia	18	250	379	3114	3694	125
Veta Martha	40	261	291	1092	2815	130
Veta Transversal	13	236	147	2610	2827	59
Veta Luz Elena	19	445	510	2220	4869	206

WMT flotation test W-07-37

A staged rougher-only flotation test (No. W-07-37) by WMT reported modest recoveries of both silver and gold (73.5% and 44.0%, respectively), with very limited potential to increase recoveries by any change in process variables. The results of the test work are shown in Table 13.3.

The following test conditions were used:

- Grind - 42% - 200 mesh (P_{80} = 200 microns).
- Pulp density of 20%.
- Frother - DF 250.

- Collector initially 9 ppm of Cytec 3418A then 30 ppm potassium amyl xanthate (PAX).
- pH natural at 8.8.
- Copper sulphate was added at the final stage of flotation, but it did not result in any flotation increment.
- Flotation to completion, requiring about six minutes.

The overall ratio of concentration was high with only 2.8% of the feed weight reporting to the overall rougher concentrate. The concentrate weight could be significantly decreased by cleaning, since the first rougher concentrate contained only about 35% sulphides, mainly as pyrite, with about 10% as sphalerite and 5% as galena.

Given the high ratio of concentration, if any future testing is undertaken, Cytec selective collector 3418A should be eliminated, retaining only the very strong and non-selective PAX as a bulk flotation collector.

Compared to the initial cyanidation results, the recoveries of silver and gold did not encourage the use of flotation for the global deposit. The use of a single master composite was a cost effective method to review the global deposit, but subsequent cyanidation testing reported that the gold in the Martha sulphide has a different response than after oxidation.

Table 13.3 Rougher only flotation test results

		Overall flotation				
Product	**Weight %**	**Ag ppm**	**Au ppm**	**Pb%**	**Fe%**	**Zn%**
1st rougher concentrate	1.4	14,360	8.26	4.41	9.4	6.63
2nd rougher concentrate	1.4	1,905	3.20	2.58	9.3	4.74
Overall rougher concentrate	2.8	8,001	5.68	3.48	9.3	5.67
Rougher tailing	97.2	84	0.21	0.21	2.7	0.26
Feed calculated	100.0	308	0.37	0.30	2.9	0.41
Feed from composite grades		300	0.33	0.21	2.9	0.33

		Distribution %			
Product	**Ag%**	**Au%**	**Pb%**	**Fe%**	**Zn%**
1st rougher concentrate	64.6	31.3	20.2	4.5	22.4
2nd rougher concentrate	8.9	12.7	12.3	4.6	16.7
Overall rougher concentrate	73.5	44.0	32.5	9.1	39.1
Rougher tailing	26.5	56.0	67.5	90.9	60.9
Feed	100.0	100.0	100.0	100.0	100.0

The tailing screen assay data, shown in Table 13.4, indicates a distribution of silver and gold for the flotation tailings product.

Table 13.4 Rougher tailing screen assay data

Mesh	**Weight %**	**Ag ppm**	**Au ppm**
	18.4	110	0.16
65			
	12.9	89	0.16
100			
	15.7	74	0.17
150			
	10.1	64	0.17
200			
	12.9	69	0.16
325			
	30.0	84	0.32
Weighted average	100.0	84	0.21
By direct assay		84	0.24

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PRA cyanidation tests Orko C1 to C3

Because of the low recovery results in the flotation test, the study shifted to the investigation of agitation cyanidation. The first cyanidation test (Orko C1) demonstrated the amenability of both gold and silver with cyanidation and two additional tests were performed to investigate indications of grind and cyanide concentration sensitivities. The three 96 hour kinetic tests, all at pH 10.0 10.5 and a pulp density of 40%, were undertaken by PRA and the results are summarised in Table 13.5.

Table 13.5 Orko cyanidation tests C1 to C3 results

Test	**NaCN (g/l)**	**Grind (P80 microns)**	**Ag head grade (ppm)**	**Ag extraction (%)**	**Au head grade (ppm)**	**Au extraction (%)**	**NaCN consumption (kg/t)**
Orko C1	2	227	335	75.8	0.44	81.6	2.14
Orko C2	1	80	300	64.9	0.49	78.5	0.72
Orko C3	4	97	327	89.1	0.85	88.0	2.56

This test series determined the following:

- Silver recovery is sensitive to both grind and cyanide concentration.
- Gold extraction does not appear to be particularly sensitive to grind but possibly sensitive to cyanide concentration.
- A retention time of 72 hours appears to reach a point of diminishing returns.
- Although test C3 reported the highest silver extraction, the extent to which grind and cyanide concentration each contributed to those results is uncertain given both parameters were varied. Future testing should investigate cyanide concentrations surrounding 2 g/l and at finer grinds.

- These tests suggest that silver extraction will be low unless the cyanide concentration is sufficiently high to incur cyanide consumption greater than 2 kg/t. That was not confirmed in subsequent testing at SGS in Durango.

- It would appear that the optimum silver recovery may require a grind that is at least as fine at P80 = 100 microns.

- There were no significantly elevated concentrations of any of the normal cyanicides in the pregnant leached solution (PLS), however, there could be potential to reduce cyanide consumption by increasing the pH.

The screen assay on the leached residue for test C3, shown Table 13.6, indicates that gold extraction is not grind sensitive but silver extraction could potentially benefit from finer grinding. This characteristic was later demonstrated by almost all of the composites for which this type of data was provided.

Table 13.6 Screen assays of Orko test C3 leached residue

Tyler mesh size fraction	Weight (%)	Au ppm	Ag ppm
+105	16.3	0.11	63.2
-105+74	15.6	0.10	39.7
-74+53	13.1	0.12	35.4
-53+37	10.3	0.12	35.6
-37	44.7	0.09	24.0
Calculated	100.0	0.10	35.5

The kinetic curve for test C3 suggests that three days of agitation leaching is required to achieve a point of diminishing returns. In future testing, four days of leaching is justifiable to increase silver extractions providing improved economic returns.

At this stage of the testing it appeared that agitation cyanidation is superior to flotation.

13.2.2 Stage 2 bottle roll leaching tests

In mid-2008, 12 composites from diamond drill core sample assay coarse rejects for agitation leaching testing were prepared for bottle roll leaching tests at PRA. Details of the 12 composites are shown in Table 13.7. The comparative assay data shows acceptable repeatability of the composite grades using several assay procedures.

These composites represented the major mineralised zones known at the time and included both oxide and sulphide composites from the Martha zone. Since the drill core geological logs and an examination of the drill core identified both oxide and sulphide mineralisation, it was useful to determine whether sulphide encapsulation of the gold and silver minerals has any role in cyanide leaching.

The Stage 2 tests were performed with the following details:

- Nominal P_{80} = 80 microns.

- NaCN concentration = 1 g/l.
- pH = 10.5.
- Pulp density of 40% solids.
- Leaching time: 72 hours for the first six composites, then 96 hours for the last six.
- All as kinetic tests.
- All tests were run in triplicate.
- Every third test (one on each composite) included tailing screen assaying for gold and silver.

The following findings were reported from the test work:

- Low extraction of silver, averaging 76%, with 5% difference in silver extraction between the oxide and sulphide composites with the oxides yielding lower silver extraction than the sulphide composites.

- Gold extraction from the three Martha sulphide composites of 45.1%, with a large variance between the 80.9% average results in the oxides and 45.1% average results in the sulphides.

- A somewhat elevated cyanide consumption ranging from 0.9 to 1.9 kg/t, averaging 1.4 kg/t.

Although the global gold feed grade is less than 0.3 ppm, it does provide an increment of economic recovery, so the impact of gold extraction cannot be ignored.

The following can be concluded for the test series:

- Many of the tests did not completely leach at either 72 or 96 hours. This is potentially attributable to a low (1 g/l) cyanide concentration, and not necessarily a result of insufficient leaching time as the combined effects of cyanide concentration and retention time is not defined.

- The gold is not grind sensitive, although it did appear to leach slowly.

- Based upon the previous Orko tests the cyanide concentration may need to be increased to a minimum of 2 g/l in the next test series.

- In the context of gold, the Martha sulphides reported significantly lower extractions than did the oxide composites.

The summarised results are shown in Table 13.8.

Table 13.7 Stage 2 agitation leaching composite summary data

PRA sample number	Composite description	Number of samples	Mean grade of individual samples Ag ppm	Composite grade		Composite screen assay grade		Back calculated grade from met testing	
				Ag ppm	Au ppm	Ag ppm	Au ppm	Ag ppm	Au ppm
1	Abundancia low grade composite	28	56	51	0.12	52	0.21	60	0.15
2	Abundancia medium grade composite	23	150	144	0.29	140	0.35	154	0.29
3	Abundancia high grade composite	25	375	367	0.36	371	0.41	364	0.50
4	Gloria low grade composite	16	49	52	0.12	54	0.11	57	0.14
5	Gloria medium grade composite	24	152	161	0.23	153	0.22	172	0.30
6	Gloria high grade composite	20	380	395	0.34	375	0.30	400	0.34
8A	Martha sulphide composite 1	53	130	126	0.24	124	0.33	125	0.29
8B	Martha sulphide composite 2	44	231	182	0.45	212	0.44	197	0.49
9	Martha sulphide composite 3	33	316	275	0.38	285	0.39	280	0.47
10	Martha mix composite 1	29	43	40	0.11	43	0.14	41	0.18
11	Martha mix composite 2	29	218	201	0.26	234	0.29	217	0.36
12	Martha oxide composite	45	195	195	0.34	196	0.35	195	0.41

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Table 13.8 Stage 2 agitation leaching testwork results

			Extraction		Residue		Consumption (kg/t)	
Sample number	**Description**	**P80 size (microns)**	**Au (%)**	**Ag (%)**	**Au (ppm)**	**Ag (ppm)**	**NaCN**	**Lime**
1	Abundancia - low grade	89	77.0	67.1	0.04	20	0.9	1.1
2	Abundancia medium grade	88	83.9	70.9	0.05	45	1.1	0.7
3	Abundancia high grade	79	81.0	80.1	0.09	73	0.8	0.9
4	Gloria low grade	52	77.9	72.3	0.03	16	1.0	1.3
5	Gloria medium grade	85	85.2	75.3	0.04	43	1.1	0.9
6	Gloria high grade	80	85.2	67.0	0.05	132	1.1	0.8
8A	Martha sulphide 1	52	39.7	83.5	0.18	20	1.9	1.1
8B	Martha sulphide 2	78	51.7	79.6	0.24	42	1.6	1.0
9	Martha sulphide 3	77	43.9	76.2	0.27	67	1.7	1.0
10	Martha mixed	78	72.7	81.3	0.05	8	1.5	1.1
11	Martha mixed	82	80.0	83.9	0.07	38	1.9	1.0
12	Martha oxide	82	85.4	75.6	0.06	49	1.8	1.0
Average of all			72.0	76.1	0.10	46	1.4	1.0
Average oxide		77	80.9	74.8	0.05	47	1.3	1.0
Average sulphide			45.1	79.8	0.23	43	1.7	1.1

13.2.3 Stage 3 cyanidation and flotation tests

PRA cyanidation tests C37 to C42

On reviewing the Stage 2 test results, the following important processing characteristics became relevant to subsequent cyanidation testing:

- A grind of P80 = 80 microns appeared to be more or less optimum at current silver prices.
- The cyanide concentration should be increased. Although 2 g/l was believed to be adequate, 4 g/l was initially used to assure technical success.
- The test time needs to be 96 hours to reasonably determine optimum leach retention time.

Cyanidation testing (PRA tests C37 to C42) was performed on six composites, which were made from re-compositing the Stage 2 composites in order to achieve silver grades of 200 ppm. Details of the composites are shown in Table 13.9. The composites were all subjected to tailing screen assaying and ICP analysis of the pregnant leaching solution.

The test conditions were as follows:

- Target grind P80 = 90 microns.
- Pulp density = 40% solids.
- Retention time = 96 hours.

- Cyanide concentration = 4 g/l.

- pH = 10.5

- Test method by kinetic bottle roll cyanidation.

- Tailing screen analysis was reported for gold and silver for every test.

The following observations are noteworthy:

- pH = 10.5

- The silver extractions were in the narrow range of 88.1% to 92.0%, averaging 90.1%, suggesting that the silver metallurgy is similar in both oxide and sulphide material.

- The consumption of cyanide was anomalously high at a typical 10 kg/t. This is unexplained by the analysis of the pregnant leaching solution and raises serious questions about the reliability of the results.

- The kinetic curves consistently show that silver leaching has reached completion in 48 hours, but gold is still leaching at 96 hours. This is also an anomalous feature. Irrespective, it appears both silver and gold extractions increased substantially in this Stage 3 testing compared to the results of the Stage 2 testing.

- Silver recoveries increased from averages of 77.9%, 82.6%, and 75.6% in the Stage 2 tests to 87.8%, 92.0%, and 92.0% for Stage 3 Martha sulphide, mixed, and oxide tests, respectively, under similar parameters. Gold recoveries increased from averages of 47.8%, 76.4%, and 85.4% in Stage 2 to 63.7%, 81.8%, and 89.6% for Stage 3 Martha sulphide, mixed, and oxide, respectively, under similar conditions. These results have to be viewed with caution given the anomalously high and unexplained cyanide consumptions in the Stage 3 tests.

- As had been reported in the Stage 2 testing, gold recovery is significantly lower in the Martha sulphide composite, although not nearly to the degree experienced in the Stage 2 work. The 60.2% gold extraction in the Orko # 3 (Martha) composite in this Stage 3 testing was significantly lower than the 88% result achieved in the Stage 1 test series.

A summary of the results from PRA cyanidation tests (C37 to 42) is shown in Table 13.10.

Table 13.9 Details of composites used in PRA cyanidation tests C37 to C47

Sample description	Zone	Calculated Head Grades Ag (ppm)	Au (ppm)	Composite preparation
Comp 2A	Abundancia	196	0.44	400 g of Stage 2 2 and 100 g of Stage 2 3
Comp 5A	Gloria	213	0.51	400 g of Stage 2 5 and 100 g of Stage 2 6
Comp 8B	Martha sulphide composite 2	217	0.52	original Stage 2 composite
Comp 11	Martha mix composite 2	231	0.31	original Stage 2 composite
Comp 12	Martha oxide composite	225	0.50	original Stage 2 composite
Orko 3	Veta Martha	269	0.41	original Veta Martha composite

Table 13.10 PRA cyanidation test C37 to C42 results

			Extraction		Residue		Consumption	
Test No	Sample ID	P80 size (microns)	Au (%)	Ag (%)	Au (ppm)	Ag (ppm)	NaCN (kg/t)	Lime (kg/t)
C37	2A	90	88.1	88.1	0.05	23	10.09	1.03
C38	5A	91	91.0	91.3	0.05	18	10.98	0.68
C39	8B	96	63.7	87.8	0.19	26	10.50	0.73
C40	11	90	81.8	92.0	0.06	18	9.85	1.11
C41	12	83	89.6	92.0	0.05	18	10.35	0.68
C42	Orko 3	101	60.2	89.5	0.17	28	10.82	0.87

The leaching kinetic curves for both silver and gold are shown in Figure 13.1. The shortfall in gold extraction with the two Martha sulphide composites is clearly demonstrated. The Orko Martha composites were not described as being sulphide but the metallurgical data in this Stage 3 study strongly suggests that that is the case.

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Figure 13.1 Silver and gold leaching kinetic curves

Flotation testing W-09-02

On reviewing the Stage 2 and 3 test results, the Martha sulphide composite 8B was subjected to flotation testing (WMT test W-09-02) followed by tailing cyanidation. Although the 84.3% silver recovery was not as high as the 87.8% reported in the comparable whole ore cyanidation test on the same composite, the grind in the flotation tests was appreciably coarser at about P_{80} = 130 microns. The gold recovery was almost identical to that of the comparable Stage 3 whole ore cyanidation test. The test work results are shown in Table 13.11.

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Table 13.11 WMT flotation test W-09-02 results

		Assays		Recovery distribution - %	
Product	**WT %**	**Ag (ppm)**	**Au (ppm)**	**Ag**	**Au**
Cleaner concentrate	4.7	3,493	7.95	80.7	64.3
Cleaner tailing	2.2	327	0.96	3.6	3.7
Rougher concentrate	6.9	2,478	5.71	84.3	67.9
Rougher tailing	93.1	34	0.20	15.7	32.1
Feed calculated	100.0	204	0.58	100.0	100.0
Feed - average calculated grade		206	0.49		
Feed - assay grade		197	0.45		

The tailing screen assay data, shown in Table 13.12, does not exhibit any potential to increase silver recovery with a finer grind, so an even coarser grind may be optimum.

Table 13.12 Rougher tailing screen assay data

Mesh	**WT%**	**Ag (ppm)**	**Au (ppm)**
	28.3	41	0.13
150			
	18.4	30	0.08
200			
	53.3	32	0.28
Total	100.0	34	0.20

Although flotation concentration produced a concentrate that is not a finished product, unlike the convenience of marketing doré from a cyanidation circuit, possible attractive features of flotation include:

- A possible coarser grind than with whole ore cyanidation, resulting in significant savings in capital and operating costs in the grinding circuit, although grind sensitivity testing is limited in both cases.

- Possibly sell the flotation concentrate, although it is improbable that this will be the most economical disposition of that product.

- Undertake very fine regrind (perhaps finer than P_{80} = 40 microns) on the estimated 5% of the feed weight that reports to flotation concentrate, then cyanide leach the product. The leaching recoveries have not been demonstrated on the flotation concentrate, but they can be inferred from the companion bottle roll cyanidation tests. That opinion is supported by the tailing screen assay data. The testing however suggests that a fine grind prior to cyanide leaching may decrease the silver grade of the tailing by 5 to 10 ppm (with respect to feed) representing a possible 2.5 to 5.0% increase in silver recovery.

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PRA cyanidation tests C43 to C47

In the following series of cyanidation tests, four of the previous composites from the C37 to C42 series were subjected to leaching at 2 g/l NaCN as the only procedural change in an attempt to decrease the anomalously high reported 10 kg/t cyanide consumption. The fifth test was on the rougher tailing from the W-09-02 flotation test on Martha sulphide composite 8B.

The results of this test work are shown in Table 13.13. Although the cyanide consumption decreased significantly, it was still anomalously high at 6 kg/t, even in the test on the #8A flotation tailing where depletion of cyanicides was anticipated. Note also that the previously reported elevated gold grades in two of the leached tailing were not replicated in these tests, suggesting an analytical error. Also, the overall silver grades in the leached residues were somewhat higher, perhaps due to slower kinetics at the lower cyanide concentration. Since no additional testing was performed on these composites, the effect of cyanide concentrations in the range of 1 to 2 g/l is unknown.

Table 13.13 PRA cyanidation test C43 to C47 results

Test No	Sample ID	P80 size (microns)	Extraction Au (%)	Extraction Ag (%)	Residue Au (ppm)	Residue Ag (ppm)	Consumption NaCN (kg/t)	Consumption Lime (kg/t)
C43	#2A	93	85.8	86.1	0.06	28	5.7	1.4
C44	#5A	91	90.3	88.1	0.04	27	6.1	1.3
C45	#8B Flotation Tails	99	79.8	79.8	0.05	6	5.0	0.8
C46	#11	102	87.6	89.0	0.05	29	6.1	1.0
C47	#12	89	88.5	86.4	0.05	29	5.9	0.9
Average	excluding C45	94	88.0	87.4	0.05	28	6.0	1.1

The graphs shown in Figure 13.2 compare cyanide consumption and silver extraction. Although there were some unintended differences in the test variables, the curve reflects comparisons using a nominal grind of P_{80} = 80 to 90 microns, with feed grading 200 ppm Ag, and with 96 hours retention time.

However, note that in many of the tests the leaching of both gold and silver was not complete in 72 hours (Stage 2 tests 1 to 18) or in 96 hours in tests C19 to C47. Additional extraction is therefore technically feasible, but the incremental cyanide consumption will discourage that because of a never diminishing consumption of cyanide, as shown in Figure 13.2 for typical test C43, albeit reporting anomalously high and unexplained cyanide consumptions that raise doubt as to the validity of the conclusions.

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Figure 13.2 Cyanide consumption and silver extraction graphs

13.2.4 Mineralogical considerations

The deportment of gold and silver in the various flotation and cyanidation tests suggests the following:

- The vast majority of the silver is present as argentite (acanthite) that is mainly relatively fine, but sufficiently exposed for cyanide leaching to be effective.
- The float and float tail leach reported significantly higher silver extractions and equivalent gold extractions to whole ore leach.
- In the case of the Martha sulphide mineralisation, there is increased gold encapsulation compared to oxide mineralisation.

13.2.5 Cyanidation

At this stage of the Project, without any detailed optimisation of grind size, cyanide concentration, pH, and retention time, whole ore agitation cyanidation has been demonstrated to achieve essentially 85% to 90% silver extraction on feed grading above 200 ppm Ag.

Disregarding the unexplained anomalously high cyanide consumptions in the Series 3 testing, the cyanide consumptions range between 1.4 to 2.1 kg/t in the State 1 and 2 results. Later testing at SGS Minerals Services in Durango reported a typical 2.5 to 2.8 kg/t consumption on a master composite that was prepared from the Orko composites. Decreasing the cyanide concentration below 2 g/l will decrease both the cyanide consumption and silver extraction.

At 2 g/l cyanide concentration, gold extraction will be approximately 45% from the Martha sulphide and approximately 81% from oxide ores. Increased leaching time beyond 96 hours may increase both gold and silver extraction at a cyanide concentration of 2 g/l. Because the composites consistently demonstrated a sustaining appetite for cyanide, the economics of incremental retention time versus incremental metal extractions may be modest.

Finer grading beyond the laboratory nominal P_{80} = 80 to 90 microns will improve silver but not gold extraction.

Throughout this section, the term extraction is used to represent the portion of the metal that reports to pregnant solution. In operating plants, it is not possible to recover all of that extracted metal into doré, and typically about 1% is lost in the plant tailing solution in the normal counter-current decantation (CCD) circuit. Filtration of the plant tailing could potentially decrease the loss of both metals as well as that of cyanide and lime.

13.2.6 Cyanide consumption

Typically, an elevated consumption of cyanide, as was reported in these tests, can be explained by ICP analysis of the final pregnant solution. Normally that is demonstrated by elevated concentrations of some or all of Cu, Zn, and Fe. That did not occur in any of the tests that included ICP analyses. In all cases, the cyanide consumption did not follow the normal pattern with an initial consumption within the first two hours, followed by much smaller increments. In all tests, the rate of cyanide consumption was more or less constant with time.

Note that the stoichiometric requirement of sodium cyanide on feed grading 200 ppm of Ag as argentite (Ag_2S) is only 0.35 kg/t, far less than the suggested minimum of 2 kg/t that is required to achieve 90% extraction.

Test C43 on the Stage 3 8A flotation tailing revealed the same pattern of sustained cyanide consumption as in the whole ore bottle tests. The Stage 3 8A flotation tailing had been washed and dried, so the elevated cyanide consumption could not be attributed to soluble salts.

13.3 Testing from 2009 to 2010

During 2009 and 2010, Pan American commissioned a series of four test programmes at SGS Minerals Services in Durango, México (SGS 2009, SGS 2010a, SGS 2010b, and SGS 2010c) which proved to be problematic with regard to the comminution processes followed, the role of grind size on cyanidation extractions, and the lack of screen analyses to support the reported grind sizes. Although the absolute grind sizes are not clearly defined, the role of grind versus extraction in the test work is believed to be valid.

Cyanidation testing investigates the main process variables including:

- Optimum grind This is studied at progressively finer grinds until no further decrease in the tailing grade is observed. That relationship will later be input into a grind cost/benefit study to determine the optimum grind. The optimum grind will change with changing commodity prices and power costs. It is independent of the other process variables.

- The optimum relationship between retention time and cyanide concentration. Typically as the retention time is increased the circuit will tolerate a lower cyanide concentration, so the purpose of this phase of testing is to provide data to determine the optimum plant design criteria in the financial model. Typically where silver is an important contributor to revenue, the retention time will range between three and five days. Typically any gold will have been leached to technical completion within a maximum of three days.

13.3.1 SGS test 18-09

This testing programme was performed on a single master composite from the Orko composites that were shipped in mid-2009 from PRA to Pan American s offices in Durango. The details of the material used to prepare the master composite are given in Table 13.14.

This composite was subjected to mineralogy, whole ore cyanidation, flotation and gravity concentration. Work Index (WI) and Abrasion Index (AI) tests were performed on samples of Martha sulphide and oxide as well as Abundancia.

Table 13.14 Details of master composite used in SGS test 18-09

		Grade - ppm	
Description	**Weight (kg)**	**Au**	**Ag**
Veta Martha	7.40	0.328	310
Veta Transversal	2.72	0.379	250
Veta la Abundancia	2.62	0.291	261
Veta Luz Elena	2.32	0.147	236
Veta La Gloria	2.72	0.510	445
No. 1 Head	1.62	0.170	52

No. 2 Head	1.72	0.320	142
No. 3 Head	1.72	0.385	369
No. 4 Head	1.82	0.115	53
No. 5 Head	1.72	0.225	157
No. 6 Head	1.72	0.320	385
No. 8B Head	3.04	0.445	197
No. 9 Head	5.70	0.385	280
No. 10 Head	5.70	0.125	41

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		Grade - ppm	
Description	**Weight (kg)**	**Au**	**Ag**
No. 11 Head	4.13	0.275	217
No. 12 Head	4.70	0.345	195
TOTAL	51.37	0.304	227

Mineralogy

The study examined three samples that are presumed to be the same ones that were subjected to WI and AI testing. The mineralogy work identified the various contained minerals including the metallic minerals, but was of limited use since there was no quantitative data. Gold was not identified, as expected, because of the low concentration of that metal.

The silver minerals identified included:

- Martha Oxide: argentite (Ag_2S) and stromeyerite ($Ag_2S.CuS$), the latter of which was the sole copper mineral.
- Martha Sulphide: argentite.
- Abundancia composite: bromo-argentite (AgBr), argentite, and anglesite (a lead sulphate mineral with less than 9% Ag in solid solution).

The various silver minerals range in size from 15 to 135 microns.

Cyanidation concentrations

The cyanidation testing was undertaken in three test series to investigate the following:

- Grind in the range of P_{80} = 80   120 microns (60   80% passing 200 mesh).

- Cyanide concentrations of 1.5 to 4.0 g/l.
- Kinetic leaching for 96 hours.
- Lead salts.
- 11.0 11.5 pH (increased from 10.0 10.5 in previous tests).
- Extraction of Cu, Zn, and Fe.

The pH may be important, and will be discussed later in the following sections of this report.

The grind sensitivity tests clearly reported a significant technical improvement in silver extraction at finer grinds, but not so for gold. Screen assaying of the cyanidation tailing, shown in Table 13.15, , is consistent with the earlier data obtained at PRA, showing a decrease in the silver grade in the leached tailing from 28 to 19 ppm Ag at finer grinds. Figure 13.3 shows a graph of the grind size versus the silver tailing grade, which demonstrates excellent linearity.

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Table 13.15 Cyanidation tailing screen assay results

	P80 = 120 µm 60 % passing 200 mesh			P80 = 100 µm 70 % passing 200 mesh			P80 = 80 µm 80 % passing 200 mesh		
	Weight	Grade		Weight	Grade		Weight	Grade	
Microns	%	Au ppm	Ag ppm	%	Au ppm	Ag ppm	%	Au ppm	Ag ppm
106	26.0	0.09	46	16.6	0.08	43	9.2	0.08	34
75	13.8	0.11	34	14.6	0.10	32	12.7	0.09	28
-75	60.1	0.10	20	68.8	0.10	19	78.1	0.09	15
Leached Tailing	100.0	0.10	28	100.0	0.09	25	100.0	0.09	19

Figure 13.3 P80 grind size versus silver tailing grade

Because all of the size data throughout the four testing programmes at SGS were reported as % passing 200 mesh, the graph shown in Figure 13.3 is reproduced in Figure 13.4 to show the SGS reporting procedure.

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Figure 13.4 Grind % passing 200 mesh size versus silver tailing grades

Variations in the results shown in Figure 13.5 appear to be within the expected test variables, so it can be concluded that silver extraction is essentially independent of cyanide concentration within the range 1.5 to 4.0 g/l NaCN. Subsequent testing (SGS-18-10) however reported increased silver extraction at 2 g/l cyanide. Gold leaching was complete well within 24 hours.

The consumptions of NaCN and CaO were 2.4 kg/t and 4.8 kg/t, respectively. Cyanide consumption was essentially constant for the lowest three cyanide concentrations, increasing to 4.5 kg/t at 4 g/l NaCN.

Figure 13.5 Kinetic leaching time of silver extractions

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The third series of tests investigated the role of litharge (PbO) and lead nitrate ($PbNO3$). Lead salts are frequently used industrially when argentite (acanthite - $Ag2S$) is present, and added to the grinding circuit as a precipitant for solubilised sulphur. If that is not done, the consumption of cyanide will be higher due to the unwanted production of thiocyanate (CNS).

Litharge was an unexpected candidate for the test work since it is not soluble in cyanide solutions. However, like lead nitrate, it reported reduced cyanide consumption and modestly improved the silver recovery. The conclusion relative to the use of lead nitrate was however refuted in the subsequent SGS-18-10 test work (series 4). Overall the use of lead salts resulted in inconsistent conclusions. Test SGS-18-10 does not support the use of lead salts, but test SGS-40-10 on the variability composites does.

These tests were performed at 1.5 g/l NaCN for 96 hours at P_{80} = 74 microns. The kinetic leaching data for this test are shown in Figure 13.6. The pH remained at 11.0 to 11.5 throughout these tests.

The tailing assay data indicates that the use of lead salts decreased the tailing grade by as much as 3 ppm Ag. The data also reported a significant reduction in the consumption of NaCN from 2.4 kg/t in the 1500 test to 1.6 kg/t in the other tests. Subsequent testing investigated lower addition rates of lead nitrate at 50 ppm, and that became a standard test condition.

Figure 13.6 Kinetic leaching time of silver extraction at P_{80} = 74 microns

The series 2 tests included metal balances for copper, zinc and iron. From feed grading 100 ppm Cu, approximately 40% of the copper was leached in four tests, independent of cyanide concentration. Zinc extraction was less than 4%, and iron less than 1%, with the single reported 7% extraction almost certainly based upon erroneous data.

Flotation concentration

Several rougher flotation tests were performed on the master composite all using Cytec 3418A collector which is considered to be a highly effective silver mineral collector, typically with very good selectivity against inactivated pyrite. Since the feed sample contained only 1.23% sulphur, and therefore a maximum of 2.5% sulphide minerals, a more aggressive collector could have been considered.

Despite this issue, the silver recovery into the rougher concentrate was a respectable 85%, and 67% for gold. These results were an improvement over the earlier results on the Orko master composite that reported only 73.5% silver and 44% gold recoveries into rougher concentrate. The gold recovery was similar to the extraction in the whole ore cyanidation tests and the silver recovery was about 6% less than reported in the best cyanidation tests, similar to the results of the Orko testing.

Cyanidation of the flotation concentrate reported 60% gold and 94% silver extractions. This suggests that the gold is significantly locked in sulphides. The mineral relationships were more thoroughly examined in test SGS-18-10.

Gravity concentration

Both shaking table and Knelson centrifugal concentrator tests were performed. The tests should have, but did not, include a final panning stage, so the resultant products represent an excessive weight percent recovery. The better of the two Knelson tests reported 42% and 40% gold and silver recoveries into 16% of the feed weight. The shaking table tests products contained 17.5% and 20.8%, of the gold and silver, in 6.0% of the feed weight.

Work Index and Abrasion Index

Bond Work Index and Abrasion Index data on three composites is shown in Table 13.16. The values represent relatively high AI.

Table 13.16 Work Index and Abrasion Index results

Sample	**F80 (µ)**	**P80 (µ)**	**WI (kWh/tonne)**	**AI g**
Martha Oxide	2810	111	16.1	0.760
Martha Sulphide	2138	113	14.1	0.721
Abundancia Composite	2128	113	12.2	0.764

13.3.2 SGS test 18-10

This test series was performed on a single 50 kg composite probably prepared from coarse rejects from diamond drill core assays, all from the Martha zone. A total of 108 intervals were included at an average interval length of 1.0 m. The weighted average grade of the composite is given in Table 13.17.

Table 13.17 Grade of composite used in SGS-18-10 testwork

Ag ppm	Au ppm	S %	Cu %	Pb %	Zn %	As ppm
223	0.42	0.56	0.01	0.16	0.27	256

The tests series evaluated the following parameters:

- Whole ore kinetic cyanidation.
- Diagnostic leaching.

- Flotation including cyanidation of the flotation concentrate and tailing.
- Gravity concentration.
- Settling and filtration.

Whole ore cyanidation

The variables that were investigated in bottle roll tests included:

- Grind in the range 65 90% minus 200 mesh.
- Leaching times of 48 and 72 hours.
- Slurry density in the range of 35 to 45% solids.
- Cyanide concentrations in the range of 0.5 to 2.0 g/l.
- Oxidants: lead nitrate and kerosene.

The effect of these variables on gold extraction is not discernable since the gold grade in the leached tailing was reported with a single significant figure, reporting gold extractions ranging 78 85%. The silver data was much more emphatic, with tailing grades ranging 16 to 42 ppm Ag and extractions ranging 83 to 93%. The highest extraction was reported at the finest grind (90% less than 75 microns) and at the highest cyanide concentration of 2 g/l.

The strong relationship between cyanide concentration and silver tailing grade was implied in test series 3, as shown in Figure 13.7. What is certain, however, is that low cyanide concentrations will incur low leaching rates, but not necessarily low metal extractions if sufficient leaching time beyond the 72 hour test conditions is provided. The cyanide consumption remains low until a threshold occurs at 1.5 g/l. Although that observation is not so important in the context of this test series, it may prove to be quite important with the earlier composites in which the maximum silver extraction incurred an incremental cyanide cost that exceeded the incremental sales revenue.

Figure 13.7 Cyanide concentration and consumption with silver tailing grade

Cyanide consumption ranged from 0.07 to 1.5 kg/t, with CaO consumption ranging from 1.3 to 1.7 kg/t.

The silver extraction improved by increasing the cyanidation time from 48 to 72 hours, reporting a decrease in the tailing grade by 2 to 3 ppm Ag. An economic study will be required to determine if this increase is economically supportable.

When studying cyanide concentrations ranging 0.5 to 2.0 g/l, at both 40 and 45% solids, improved silver extraction was reported at the highest cyanide concentration. The kinetic data was somewhat more favourable at 40% solids, as shown in Figure 13.8.

Figure 13.8 Kinetic silver leaching time at 40% solids

Data from the same tests is shown in Figure 13.9 to demonstrate the relationship between the leached residue silver grade and the cyanide concentration. The data clearly shows a benefit of increased cyanide concentration.

Figure 13.9 Cyanide concentration versus leached tailing silver grade at 40% solids

The results from both test SGS-18-10 and SGS-18-09 were combined to create the graph shown in Figure 13.10. This data clearly demonstrates a 3 to 5 ppm reduction in the silver tailing grade as a function of fine grinding and increased cyanide concentrations higher than the typical industrial practice at 2 and 3 g/l.

From a metal extraction perspective, there is no apparent advantage in using cyanide concentrations greater than 2 g/l. Fortunately, unlike the earlier Orko testing series C37-47, increased cyanide concentrations did not incur an increase in cyanide consumption, perhaps due to the higher pH used in the testing.

Figure 13.10 Grind versus silver tailing grade from tests SGS-18-09 and SGS-18-10

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Figure 13.11 shows the relationship between the cyanide concentration and cyanide consumption. Although the data is scattered, it shows increased consumption with increased concentration. However, the benefit in terms of increased silver extraction is significantly greater than the incremental cost of cyanide.

Figure 13.11 Cyanide concentration versus cyanide consumption

Diagnostic leaching

Diagnostic leaching was performed on feed sample and flotation tailing products. The study determined that of the metals that are not recovered in whole ore cyanidation, the majority of the gold is encapsulated in sulphides and the majority of the silver is in carbonates. This is supportive of the observation that gold is not significantly grind sensitive, but silver benefits from fine grinding. In the case of the flotation tailing, the majority of the gold and silver are amenable to cyanidation and the majority of the refractory gold and silver is encapsulated in sulphides.

Flotation concentration

Flotation tests were not ideal, as too much emphasis was placed on producing an unnecessary low concentrate weight and too little emphasis on maximising the metal recoveries. The highest gold and silver recoveries were reported in test # 14 of 22 tests, at 62% and 86%, respectively.

As was to be expected, the flotation performance improved at the finest grind of 75% minus 75 microns, and with aggressive rougher flotation in which 8% of the feed weight was recovered into rougher concentrate. Since the only possible destination for any flotation concentrate is to a cyanidation circuit, a high ratio of concentration, at the expense of metal recovery, is neither desirable nor required.

When comparing the flotation results, including possible cyanidation of the flotation concentrate and tailing with whole ore cyanidation, and without any economic evaluation, it appears that whole ore cyanidation will provide improved economics compared to flotation. This opinion is based upon the following observations:

- The lowest rougher tailing grade of 31 ppm Ag is 12 ppm Ag higher than in whole ore cyanidation. The grade of the flotation concentrate is too low for sale to a smelter, so it still needs to be cyanide leached to produce a marketable product.

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- Some of the gold and silver that reports to the flotation concentrate is encapsulated in sulphides and cannot be recovered by cyanidation at P80 = 74 microns. The role of fine re-grinding of the flotation concentrate has not been investigated.

Gravity concentration and cyanidation

These gravity and concentrate cyanidation tests had no potential to demonstrate a viable processing option due to low gold and silver recoveries into the gravity concentrates, at 24% and 40%, for gold and silver, respectively.

Sedimentation and filtration

In WMT s experience, drying of samples can produce agglomerates that resist subsequent slurrying, leading to more favourable results than can be achieved in the plant. In the laboratory, there is no need to pre-dry samples, so in future, all settling and filtration tests should be performed on fresh slurries.

13.3.3 SGS test 40-10

This third phase testing programme was performed on 28 5 kg variability composites. Details of the variability composites are given in Table 13.18, these same composites were used in the subsequent SGS-60-10 study.

Table 13.18 Details of the variability composites used in tests SGS-40-10 and SGS-60-10

			Expected Composite Grade						
Sample	**Vein**	**Location**	**Ag ppm**	**Au ppm**	**Cu ppm**	**S %**	**Ca %**	**Pb ppm**	**Zn ppm**
1	M	Shallow	119	0.08	71	0.1	8.7	1664	3872
2	M	Shallow	107	0.14	42	0.1	2.9	553	1322
3	M	Shallow	180	0.16	43	0.1	0.2	606	704
4	M	Mid	375	0.31	111	0.1	2.3	1283	2211
5	M	Mid	94	0.15	32	0.1	0.1	310	231
5	M	Mid	138	0.21	113	0.1	3.4	4601	7049
6	M	Deep	185	0.19	33	0.3	9.5	355	484

7	M	Deep	119	0.21	173	1.2	5.3	2021	4003
8	M	Deep	334	0.55	174	1.9	4.4	2501	6301
9	M	Shallow	149	0.14	82	0.1	0.2	2214	2097
9	M	Shallow	152	0.24	37	0.1	0.1	514	596
10	M	Shallow	313	0.35	177	0.1	0.1	4147	3396
11	M	Mid	123	0.40	30	0.2	1.5	499	923
12	M	Mid	171	0.45	69	0.1	0.5	1534	1897
13	M	Mid	217	0.48	47	0.1	0.2	628	1269
14	M	Deep	65	0.21	115	0.5	14.2	1474	2867
15	M	Deep	57	0.19	61	1.6	2.1	995	1433
16	M	Deep	224	0.43	932	5.7	10.3	15321	27249

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			Expected Composite Grade						
Sample	Vein	Location	Ag ppm	Au ppm	Cu ppm	S %	Ca %	Pb ppm	Zn ppm
17	A	Shallow	109	0.32	42	0.1	0.1	354	421
18	A	Mid	202	0.24	97	0.1	0.4	1833	5097
19	A	Deep	110	0.27	120	0.1	0.2	3700	4643
20	A	Shallow	157	0.13	40	0.1	0.3	303	734
21	A	Mid	418	0.47	104	0.1	1.1	3050	6174
22	A	Deep	116	0.83	416	0.0	0.9	3054	1086
23	G	Shallow	209	0.19	46	0.1	0.1	426	1656
24	G	Mid	223	0.22	84	0.1	0.1	802	1417
25	G	Deep	213	0.32	168	0.2	0.7	1500	5692
26	G	Shallow	187	0.06	27	0.1	0.1	129	326
27	G	Mid	254	0.21	86	0.4	0.1	1557	948
28	G	Deep	286	0.66	414	4.2	0.1	4950	8631
Average			187	0.29	133	0.8	2.3	2096	3491

Vein code legend: M = Martha, A = Abundancia, and G = Gloria

The composites were all processed in kinetic cyanidation tests in which cyanide and lime were added to the grinding mill. This was an unnecessarily complicated method to perform kinetic testing, and resulted in an unnecessarily complicated presentation and analysis of the data.

The only variable that was intentionally studied was the effect of 50 ppm of lead nitrate. The grinds were all reported at 80% passing 200 mesh, but there are no screen analyses to support that claim.

The test constants were:

- Cyanide concentration of 1.5 g/l.
- pH 11.0 to 11.5.
- 72 hours kinetic, but poorly reported.

- Target grinds of 80% passing 200 mesh.

- 500 g samples consumed in each test.

- 45% solids.

The results of the test work, shown in Figure 13.12, are very inconsistent, as expected for variability testing on widely spaced composite samples. However, for the average of the 28 composites, the addition of 50 ppm of lead nitrate resulted in a 6% increase in silver extraction equivalent to a decrease in the tailing grade of about 11 ppm Ag. There was no significant change in the average gold extraction, and there would have been no expectation for that to occur since lead salt has no chemical role in the cyanide leaching of electrum.

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Figure 13.12 Silver extraction with and without lead nitrate

The same data points shown in Figure 13.12 are colour coded in Figure 13.13 according to vein type to assess whether any variability in extraction existed between the different vein types. Although there is a grouping of data in the plus 80% extraction range, all three veins contained material that reported less than 70% silver extractions without lead nitrate. In five of six cases, the silver extraction improved with the addition of 50 ppm of lead nitrate, but in four of those cases it was still less than 80%.

A perhaps important observation is that Gloria composites 24 to 26 showed a 20% increase in silver extraction with the addition of lead nitrate. Care should be taken when comparing this data with the results reported in SGS-60-10, since the latter did not include testing of Gloria 23 to 26.

Figure 13.13 Silver extraction with and without lead nitrate by vein type

As in the SGS-18-09 study, the extractions of copper and zinc were reported. The 40% copper extraction from feed grading 100 ppm Cu was consistent with the earlier results. The 3,700 ppm Zn grade reported 3% zinc extraction, also similar to the SGS-18-09 results. Although the reported zinc extraction was quite low, in several tests, the zinc grade of the pregnant solution is significantly higher than the silver grade.

13.3.4 SGS test 60-10

The same composites that were studied in the SGS 40-10 test were used in the SGS-60-10 studies except for four of the six Gloria composites that had been consumed in the SGS-40-10 test work. For this work, 24 standard bottle roll (pulverise/botella) tests were undertaken and compared with the molienda/botella (grind/bottle roll) results from test SGS 40-10. The test work used the following parameters:

- Cyanide concentration of 1.5 g/l (no change).
- pH 11.0 to 11.5 (no change).
- 96 hours kinetic (increased from 72 hours in the previous molienda/botella tests).
- 80% passing 200 mesh (no change).
- 500 g samples were consumed in each test (no change).
- 45% solids (no change).
- 50 ppm lead nitrate (no change).

The more favourable silver extractions in the botella tests is probably attributable to an unintentional change in the size distribution and not because of a longer leaching time.

Between the two testing procedures, there was no significant difference in the consumptions of NaCN (1.29 versus 1.23 kg/t) or CaO (2.05 to 2.22 kg/t). However, it should be noted that the reported cyanide consumption is the actual quantity that was consumed in the laboratory testing, and does not include any additional amount that is inevitably discarded in the plant tailing slurry. The actual plant consumption of NaCN will depend upon the plant flow sheet that, in this case, includes several stages of counter current decantation (CCD) washing of the leached tailing with fresh water addition to the final CCD thickener.

The relationship between the silver grade in the feed and the silver recovery/silver tailing grades were derived. The graphs shown in Figure 13.14 demonstrate quite variable results, but the relationship between feed grade and cyanidation extraction is quite apparent and as expected.

Figure 13.14 Comparison of silver feed grade vs silver tailing grade and silver extraction

Figure 13.15 shows the relationships between the sulphur content in the Martha composites and the gold and silver extractions. Although the data points are based upon total sulphur grades, they clearly show that the gold metallurgy is severely impacted in the presence of sulphur. With a single exception (Composite No. 1) silver extraction was not impacted at elevated sulphur content. These results are consistent with the data from the Stage 2 test work programme in which the composites included identified both Martha oxide and sulphides. The Abundancia composites contained virtually no sulphur.

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Figure 13.15 Comparison of sulphur grade versus gold and silver extraction in botella tests on Martha composites

13.3.5 Summary of SGS test work

From a technical perspective (but not necessarily an economic perspective), the near optimum cyanidation processing parameters are tabulated below. In the absence of screen analyses the optimum grind is speculative but otherwise the optimum conditions are reasonably well demonstrated in the testing reports.

- Retention time 96 hours.
- Cyanide concentration 2.0 g/l.
- Cyanide consumption 1.25 kg/t + plant allowance.
- pH probably around 11, but not emphatically determined.
- Grind P_{80} = 80 microns.
- CaO consumption 2.2 kg/t.

- 50 ppm lead nitrate.

- Pulp density = 40 to 45% solids.

The variability composites should have provided data on the relationship between feed grade, metal extractions and tailing grades. Unfortunately, the test procedures did not adequately address or correctly report the very important benefits of both fine grinding and cyanide concentration greater 1.5 g/l, as were clearly demonstrated in the first two testing programmes undertaken at SGS.

At this stage of the Project, the range in tailing grades in the successful cyanidation tests was only about 0.02 ppm Au and 5 ppm Ag. Although these values are important in the Project financials, it requires another stage of laboratory testing to fine tune the processing parameters for the feasibility study.

13.4 Sample representativity

The metallurgical programmes to date have evaluated several metallurgical composites in a total of seven programmes. A total of 865 individual samples, all from diamond drill core, have been used in the preparation of composites.

All of the metallurgical samples are shown in the two horizontal projections of the deposit given in Figure 13.16. Note that the samples are almost entirely from the portions of the mineral resource classified as Indicated. Although there is very little representation from the Inferred mineral resource, that material is mainly of a lower grade, and frequently at less than the cut-off grades used in this preliminary economic assessment. However, since additional laboratory testing is recommended and since new variability composites will in any case be required, it is recommended that a larger sample footprint be included when preparing those composites.

The typical 180 to 230 ppm Ag feed grades of most of the metallurgical composites to date significantly exceeds the grades of the majority of the drillhole sample intersections and the mineralised material likely to be encountered during mining.

Figure 13.17 shows the variability composites samples versus forecast silver grades. The implication is that although those samples were suitable to identify the important processing variables, the silver extractions will be overstated by as much as about 3% when replacing the metallurgical composite grades with the mineral resource grades.

When comparing the anticipated composite grades from the drill hole samples to the back-calculated composite grades, they were generally in agreement. A single notable exception occurred with the Abundancia composite 22, in which the calculated grades for both gold and silver were only about 20% of the anticipated grades. Discounting composite 22, the average gold difference for the variability composites was 10%, and the average silver difference was less than 2%.

The silver grades of the variability composites that were provided for the SGS-40-10 and SGS-60-10 studies, with the exception of all six of the very high grade Gloria composites, envelop the current lower estimated average grades and at the preliminary economic assessment stage are suitable for the determination of recovered values by mining blocks.

Figure 13.16 Metallurgical sample locations plotted on a horizontal projection of the deposit coloured by mineral resource grade and classification

Figure 13.17 Variability composite samples versus forecast silver grades

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13.5 Material issues and deleterious elements

Detailed chemical analyses were reported for all of the composites that were included in the SGS studies, and those composites included portions of virtually every one of the 865 individual samples. Table 13.19, from test SGS-18-09, is representative of the western portion of the deposit since it contains portions of all of the Orko composites that collectively represent more than 50% of all of the metallurgical samples. The data is typical of the analyses for almost all of the SGS composites. Exceptions occurred with the sulphide composites in which sulphur grades of more than 4% were reported (variability composites 16 and 28) with associated elevated base metal analyses.

The deposit contains small quantities of copper, lead, and zinc. Of these, typically 40% of the copper and less than 2% of the zinc report to the cyanidation pregnant solution as metal cyanides. They will be destroyed in the cyanide destruction circuit, then report to the solid tail product as metal hydroxides.

Although the reported mercury content was quite low, the extent to which it reports to the pregnant solution, then to the precipitate is not known. This needs to be checked in the next phase of testing. If elevated mercury is reported in the pregnant solution, the inclusion of a mercury distillation stage in the refinery will be required to eliminate a potential health hazard from volatilised mercury.

At this stage of the Project the potential role of acid generation and metal leaching from waste rock, low grade feed, and plant tailing is unknown.

Table 13.19 Representative element concentrations

Element	Grade	Element	Grade	Element	Grade	Element	Grade
Au	0.29 ppm	Bi	<5 ppm	La	7.1 ppm	Sb	31 ppm
Ag	235 ppm	Ca	4.31%	Li	12 ppm	Sc	0.9 ppm
Pb	0.23%	Cd	36 ppm	Mg	0.34%	Sn	<10 ppm
Zn	0.42%	Co	4 ppm	Mn	4590 ppm	Sr	80.7 ppm
Al	0.39%	Cr	15 ppm	Mo	22 ppm	Ti	<0.01%
As	379 ppm	Cu	148 ppm	Na	<0.01%	V	63 ppm
B	18 ppm	Fe	3.35%	Ni	6 ppm	W	30 ppm
Ba	380 ppm	Hg	1 ppm	P	0.02%	Y	4 ppm
Be	0.8 ppm	K	0.07%	S	1.23%	Zr	1 ppm

13.6 Conclusions

13.6.1 Metallurgical composites

All of the master composites were of a much higher grade than the average grade of drillhole sample intersections and the material that is likely to be encountered during mining. The 28 variability composites from 2010 were almost entirely selected from within the portion of the mineral resource classified as Indicated, while providing very few samples from the lower grade Inferred portion of the mineral resource. It is noted that Inferred mineral resources comprise about 20% of the total mineral resources in the economic analysis. The implication of the sample representativity is summarised below:

- A possible over-representation of high grade intersections, and an under-representation at grades adjacent to and below the average grades.

- Under-representation at the 35 and 85 ppm Ag cut-off grades for open pit and underground mining, respectively. The lowest grade in the series was 27 ppm Ag and only three of the composites graded less than 100 ppm Ag.

- The grade range for the six Gloria composites, at 187 to 285 ppm Ag, is well in excess of the estimated Gloria mineral resource grade of 149 ppm Ag. With no samples grading close to either the average grade or the cut-off grade, the metallurgical forecasts for Gloria will be uncertain based upon the need to use the lower grade Martha composites results to forecast the Gloria metallurgy. Fortunately, the three Orko Stage 2 Gloria composites were well graded and the metallurgical results are supportive of the use of global variability composite grade versus extraction versus tailing grade relationship.

- It is unknown whether the sulphide zones are under- or over-represented.

SGS reported that of the original 28 composites, almost all have been entirely consumed, so any additional testing will require a new set of variability composites.

13.6.2 Gold and silver metallurgy

The average silver extractions in the successful cyanidation tests are sufficiently consistent to support a global silver recovery based upon a leached tailing grade of 20 ppm Ag. This forecast is based upon data from master composites nominally grading 220 ppm Ag, as reported in the SGS-18-09 test comprising the Orko composites and the SGS-18-10 study on a new master composite prepared by Pan American. That global silver forecast can be applied to both Martha and Abundancia vein material for feed grades ranging a nominal 180 to 230 ppm Ag.

The La Preciosa mineral resource estimate comprises 40 million tonnes of resource, of which almost 33 million tonnes are attributable to the three major veins at Martha (approximately 24.5 Mt at 128 ppm Ag), Abundancia (approximately 5.5 Mt at 136 ppm Ag), and Gloria (approximately 2.9 Mt at 149 ppm Ag). Note however, that the silver grades encountered in drillhole vein intersections and estimated in the mineral resources are appreciably lower than the metallurgical composite silver grades.

The variability composites from Martha and Abundancia and used for the SGS-40-10 and SGS60-10 test work span the above average silver grades. The Gloria composites were all quite high grade, ranging from 191 to 357 ppm Ag. Only four of the variability composites graded less than 100 ppm Ag.

Even though the silver feed versus tailing grades versus extraction data shown in Figure 13.18 demonstrates poor regression, and the Gloria metallurgical forecasts are compromised by excessively high composite grades, this data will be used at this stage of financial analysis. Fortunately the data is supported by the results on three Orko Stage 2 Gloria composites covering the range of 60 to 400 ppm Ag. The data shown in Figure 13.18 demonstrates lower silver extractions at lower feed grades. However, even at low feed grades, with only a single exception, the silver extraction remained above 80%. The SGS-40-10 study (molienda/botella tests) consumed four of the six Gloria composites, so the SGS-60-10 report, and the data shown in Figure 13.18, includes botella testing data from only two of the Gloria composites.

The SGS-40-10 variability data for Gloria, which comprises approximately 7% of the mineral resource tonnage, reported a very high average leached tailing grade of 38 ppm Ag, ranging from 25 to 57 ppm Ag. That has resulted in uncertainty in the forecasting of the Gloria process metallurgy in the absence of both average grade and cut-off grade composites.

Based upon the Pan American variability composites for which sulphur grades were reported, gold metallurgical forecasting is somewhat more complicated. The Martha elevated sulphur composites report much lower gold extractions ranging 62 to 80% and averaging 70%, increasing to essentially 90% for low sulphur composites. The six Abundancia composites all contained virtually no sulphur, so gold extractions can be based upon tailing grading 0.05 ppm Au.

As was the case with Martha, the one Gloria composite (No. 28) with elevated sulphur content reported gold tailing grades at 0.12 and 0.28 ppm Au in the three variability tests. The differences in the tailing grades defy explanation and are therefore likely due to variable metallurgical and/or mineralogical characteristics. That composite graded a particularly high 0.65 ppm Au, reporting 59 and 77% gold extraction.

The remaining five Gloria composites reported very low tailing grades ranging from 0.01 to 0.05 ppm Au (extractions of 89 to 94%) on feed grading 0.10 to 0.33 ppm Au). The global tailing grade for the six Gloria composites was 0.07 ppm Au.

Figure 13.18 Silver feed versus tailing grades and extraction during botella tests

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13.6.3 Processing parameters

From a technical perspective (but not necessarily an economic perspective), the near optimum cyanidation parameters are summarised below:

- Retention time 96 hours (possibly 72 hours).
- Cyanide concentration 2.0 g/l.
- Cyanide consumption 1.2 to perhaps 2.5 kg/t plus the plant allowance for solution losses in the tailing slurry.
- CaO consumption 2.2 kg/t.
- pH not determined, but will be either 10.0 10.5 or 11.0 11.5.

- Grind P_{80} = 60 to 70 microns.

- 50 ppm lead nitrate.

13.6.4 Cyanide consumption

The reported consumption of sodium cyanide has been quite variable, ranging from less than 1 kg/t in several of the SGS tests subsequent to SGS-18-09 to a very high 10 kg/t in the second Orko test series in which very high cyanide concentrations were evaluated. The 10 kg/t test results, although technically valid, cannot be justified industrially based upon a comparison of incremental silver extraction versus incremental cyanide costs. However, when the cyanide consumption was decreased to 2 g/l in Orko tests C43-47, consistent with that which was subsequently determined to be the near optimum in the SGS tests, the cyanide consumption remained high at 6 kg/t. At this time, there is insufficient data to determine what would have occurred at a concentration of 1.5 g/l, for example, on that set of samples.

Typical data is shown in Table 13.20, in which silver extractions were satisfactory with the exception of the Stage 2 test work series in which the 1 g/l cyanide concentration was excessively low. No supportable explanation can be provided to discard the very elevated cyanide consumption in the test series C37 to 47. It is highly improbable that that test data was in error, since the series was performed in two batches spanning more than one month, but it still raises concerns with the reported cyanide consumptions.

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Table 13.20 Typical NaCN consumption, silver extraction, and cyanide concentrations

Test Series	NaCN consumption (kg/t)	Average Ag Extraction (%)	Cyanide concentration (g/l)
Orko original	2.6	89.1	4
Stage 2	1.4	76.1	1
Orko tests C37-42	10.4	90.1	4
Orko tests C43-47	6.0	87.4	2
SGS-18-09	2.8	91.3	3
SGS-18-09	4.5	90.5	4
SGS-18-10	1.0	92.2	2
SGS-40/60-10	1.3	83.3	1.5

All of the tests in which greater than 87% silver extraction was reported occurred with cyanide concentration equal to or exceeding 2 g/l. However, in balance, it can be concluded that high cyanide concentrations must be avoided in the plant due to the increased costs associated with excess cyanide consumption.

In all tests the initial addition of cyanide needed to be augmented during the test, and in most of the tests, the target cyanide concentrations were not maintained throughout the tests.

Inevitably, there were variations in the feed grades, with the averages ranging 180 310 ppm Ag.

13.6.5 Lime addition point

There is nothing in the SGS data to suggest that the addition of lime to the grinding stage in test SGS-40-10 is superior to the addition at the beginning of the bottle roll stage in test SGS-60-10. Although there were differences in the silver extractions between the two procedures, it was unrelated to the lime addition point. The issue is unimportant in the context of the operating plant, since recycled plant solution (semi-pregnant solution) with a lime pH at a minimum of 10.0 will be used for dilution in the grinding circuit. The addition point for make-up lime will be dictated by the preference for direct addition of pebble lime to the grinding circuit, or as slurry potentially fed throughout the circuit.

13.7 Recommendations

13.7.1 Laboratory testing

WMT considers that the data to date indicates that the optimum global processing parameters have been reasonably established in the context of the preliminary economic assessment. To advance the Project further, a two staged laboratory programme is recommended in which the first stage will determine the role of pH. That is best done on the SGS-18-09 and SGS-18-10 master composites in which the SGS target pH was 11.0 11.5. Note that SGS did not identify the cyanidation tests in these two reports by number, so tests have to be identified by their location in the test series. Since the plant will inevitably be operated using recycled process solution at a minimum of pH 10, the test procedure should include lime addition to the grinding mill.

- On the SGS-18-09 composite, replicate the series 1 at 80% passing 200 mesh test, but at a reduced 10.0 to 10.5 pH.
- On the SGS-18-10 composite, repeat the series 4 50 ppm lead nitrate test, but at a reduced 10.0 to 10.5 pH.

Depending upon the procedure that SGS used to report the grind in the original tests, it may be necessary to perform two pairs of tests using identical grind (not pulverising) times to evaluate the two pH ranges. These tests should include screen assaying directly on the tailing slurry and not on dried solids, to avoid possible agglomeration. Assaying can be for silver only.

Another variability testing programme is recommended as the second stage. Some of the existing variability composites may be used for this programme but approximately 10 additional composites will be required for between 10 and 26 additional tests. The variability samples should be selected from representative locations of the veins and should include some that are at less than the cut-off grades.

This programme has the potential to report a modest global decrease in the silver and gold tailing grades of 5 ppm Ag and 0.01 ppm Au, respectively. More importantly, the proposed testing may alter the feasibility stage design criteria sufficiently that it will impact the capital and operating costs, particularly so in the grinding circuit.

The optimisation parameters can be investigated by evaluating:

- Finer grinding at nominally P_{80} = 60 - 70 microns.
- Increased cyanide concentration to 2 g/l.
- pH at the better of the 10.0 to 10.5 or 11.0 to 11.5.

The following should be included in the proposed future tests in addition to what is recommended above:

- Silver and gold assaying only.
- Screen assaying of the leached tailing.
- 96 hour kinetic tests, in 24 hour increments.

It is recommended that a larger sample footprint be included when preparing any new composites, from a range of silver grades spanning the 35 ppm and 85 ppm Ag cut-off grades, up to 200 ppm Ag, so that a suitable recovery algorithm can be developed.

13.7.2 Sulphur data

To the extent to which sulphur assay data is provided in the data base, that data should be noted, since whenever sulphur grades greater than 1% were reported in the variability composites, gold extraction was adversely affected. Although the assay data includes only total sulphur analyses, the relationship between lower gold extractions and elevated sulphur grades suggest that the sulphur is present as sulphides. The implication is that the sulphur data will not benefit from sulphur speciation assaying.

The six Abundancia variability composites contain virtually no sulphur, but the population is too small to eliminate a possible role of sulphides unless it is supported in the drill core logs.

14 Mineral resource estimates

14.1 Disclosure

Mineral resources reported in Section 14 were prepared by Mr. Michael Stewart, F.AusIMM, Principal Consultant of Quantitative Geoscience Pty. Ltd. Mr. Stewart is a qualified person as defined by NI 43-101 and is independent of Pan American and Orko. The mineral resource estimate has an effective date of 30 June 2011.

Mineral resources that are not mineral reserves do not have demonstrated economic viability. CIM (2010) defines a mineral reserve as the economically mineable part of a Measured or Indicated mineral resource demonstrated by at least a preliminary feasibility study. No mineral reserves have been estimated for the Property at this time. The results of the preliminary economic assessment disclosed in this technical report forms the basis for the likely cut-off grades used for reporting mineral resources.

There are no known issues that may materially affect the mineral resources. These conclusions are based on the following:

- There are no known environmental issues. Pan American holds the required exploration permits from SEMARNAT and they are in good standing.
- The Project has the required permits to undertake exploration and other activities that are current and in good standing. Mineral and surface rights have secure title.
- There are no known legal, title, taxation, socio-economic, marketing, political, infrastructure, mining, metallurgical, or other issues.

14.2 Assumptions, methods and parameters 2010 mineral resource estimates

The mineral resource estimate was prepared in the following steps:

- Data preparation and import into various software packages.

- Data validation was undertaken by Pan American and reviewed by QG.

- Analysis of the sample assay QAQC data.

- Assessment of wireframe interpretations provided by Pan American against the drillhole sample data.

- Coding of drillhole samples for vein number using the wireframe models of the 18 discrete veins.

- Compositing of sample assay intervals.

- Exploratory data analysis of sample grades within each vein system.

- Variogram analysis and modelling.

- Creation of block models.

- Derivation of two-dimensional estimation methodology for vein material and three-dimensional estimation methodology for non-vein material.

- For vein material, estimation of vein thickness, accumulation (sample grade multiplied by vein thickness) and bulk density. Estimated grade variables include

Ag, Au, Pb, Zn, Cu, S, As, Ca, and Fe. For non-vein material, the same grade and density variables were estimated.

- Translation of the two dimensional estimates into three dimensions.
- Treatment for minimum mining widths and planned mine dilution.
- Validation of estimated block grades against input sample composite grades.
- Confidence classification of estimates with respect to CIM guidelines.
- Mineral resource tabulation and mineral resource reporting.

14.3 Supplied data, data preparation, data transformations, and data validation

14.3.1 Supplied data

Pan American provided QG with a data package of drillhole data, surface topography, and geological wireframe interpretations in the form of a Minesight mining software project file on 25 October 2010. The geological wireframes include interpretations of 19 veins, the base of the modern-day soil profile, the base of recent extrusive basalt flows, the base of the paleosoil profile buried beneath the basalt flows, the base of andesite unit, and the base of conglomerate unit. During the course of the mineral resource estimate, Pan American changed the grouping of the veins by combining Olin with Luz Elena, and the total number of veins changed from 19 to 18.

Table 10.1 shows details of the drillholes available for assessment during the mineral resource estimate and Figure 10.1 shows a plan of the drillholes relative to the mineral resource outlines. Of these available drillholes, two Luismin holes for 382 m, 366 Orko holes for 144,127 m, and 309 Pan American holes for 79,648 m lie within the mineral resource model and were used for geological interpretation and mineral resource estimation. Given the age and the lack of information supporting the Luismin drillhole data, the intention was to exclude the Luismin drillholes from the estimate. However, two Luismin drillholes re-assayed by Orko at Inspectorate Laboratories, BP-3 and BP-6, were included in error. QG reviewed the likely impact of these drillholes on the estimate and concluded that the effect on the estimates and potential mineral resources and mineral reserves will be negligible. Any future estimates should exclude these drillholes. None of the channel samples collected by Luismin were used in the mineral resource estimate.

In general the available drillholes are arranged on an initial grid of 100 m by 100 m over almost the entire deposit, infilled in areas to 100 m by 50 m and in areas to 50 m by 50 m. Two crosses of closely spaced drilling are also present.

14.3.2 Data preparation

The La Preciosa drillhole database was imported into Isatis geostatistical software for processing and estimation. No allowance was made for sample recovery, for example by using sample recovery as a weighting item in analysis and estimation. This approach assumes that any missing core (less than 100% recovery) has the same grade as the recovered material in the sample. This approach was selected because there is no obvious relationship between sample recovery and sample grade.

14.3.3 Data transformations

A two dimensional estimation approach was adopted in preference to a three dimensional approach to eliminate issues with large estimation weights on the boundary of the veins, to reduce local and global estimation bias, and to obtain a higher quality variogram with improved structure and greater continuity. Estimation in two dimensions requires that drillhole intercepts be projected onto a common plane. This plane can be at any orientation and for simplicity one of the main orthogonal directions is usually chosen. At La Preciosa, the veins are variably oriented, which has required projection onto all three of the main orthogonal planes. The modelled veins can be grouped into three types of orientations:

- The shallow dipping Martha vein group (generally -16° to 235°).
- The steeper dipping Abundancia and Gloria veins above Martha (generally -21° to 270°).
- The steeply dipping Transversal vein group (generally -60° to 190°).

The Martha veins and any other veins with a dip less than 45° were projected onto the XY (horizontal) plane, the remaining steeper dipping veins were projected onto the XZ plane, and the transverse veins were projected onto the YZ plane. Figure 14.1 shows an oblique view from the SSW of the vein projections.

Figure 14.1 Oblique view from the SSW of vein projections

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The vein projection was accomplished by transforming the coordinates of the veins onto the projection plane prior to loading into Isatis geostatistical software for analysis and estimation.

14.3.4 Data validation

QG assessed the wireframe interpretations of 18 veins provided by Pan American for changes in the interpretation between previous Orko drilling and recent Pan American drilling. QG noted a strong continuity of sample grades and mineralisation thickness and a good consistency between the two generations of drilling. QG determined that a drill pattern of 50 m by 50 m should be sufficient to identify the presence or absence of continuous zones of mineralisation and should be of sufficient local accuracy to allow for reliable mine planning.

QG assessed the assaying performance of the recent Pan American drillhole samples and also assessed the samples for any potential grade biases due to sampling, assaying, or other causes in the Orko sample database. No serious concerns about the quality of previous or recent drillhole data were raised during the assessment and the data was considered suitable for inclusion in mineral resource estimates.

The sample interval information contains a code for sample reliability and any intervals coded as unknown or unreliable origin were excluded from the subsequent analysis. Samples were coded as unreliable if there were no multi-element assays available for the sample interval, or when adjacent intervals had exactly the same assay values for the entire suite of assays. This treatment resulted in the removal of 24 assays within the veins, and of these assays, only three had significant silver grades.

QG assessed the available bulk density measurements collected by Orko and Pan American and found the work to be of high quality.

14.4 Geological interpretation, modeling, and domaining

14.4.1 Geological interpretation and modelling

Sectional interpretation polygons and wireframe solid models of surfaces and the 18 veins or vein packages were prepared by Pan American geologists and provided to QG. The interpretation polygons were prepared on variably spaced sections depending on the drillhole spacing along strike, which ranges from 15 m to 100 m. The geological interpretation on each section was snapped to the drillhole intersection in three dimensions and the wireframes were prepared by linking the polygon sections. Vein packages were defined primarily on logged geology (veining, alteration, and mineralisation), with reference to grade.

Surface interpretations did not extend to the limits of the block model, and using the interpretations to code the model for lithology resulted in some marginal areas being assigned a default schist code from immediately below the surface. However these areas are well outside of the limit of any potential open pit designs and are not relevant to the mineral resources.

QG reviewed the wireframes, made some minor adjustments to improve the volume constraint, and clipped the vein solids against the surface topography. QG believes that the interpretations are a reasonable representation of the likely geology. In some locations, interpretations have been made through drillhole intersections where no vein has been logged or sampled, but these are in general towards the peripheral, lower grade margins of the veins.

Vein widths vary between 0.2 m and 23 m, and the average width of drillhole intercepts across all veins is 4.7 m. The majority of the intercepts are less than 6 m thick. Some of the 18 separate vein packages contain multiple veins and the veins vary in geometrical continuity. Martha Alta is the most continuous vein with a strike of 3,100 m and a dip extent of greater than 1,100 m defined by 451 drillhole intercepts. The smallest veins modelled are Nieto and Carmen, which are both defined on eight drillhole intercepts, and with dimensions of around 300 m by 150 m. Details of the vein volumes and orientations are given in Table 14.1.

Close spaced drilling carried out in a narrow, near surface portion of the Martha Alta vein demonstrates strong continuity of vein structure and reveals that, within the structure, the zones above the probable underground mining cut-off grades occur in shoots separated by lower grade mineralisation. In this particular location, the material above cut-off grade has dimensions of around 75 m by 100 m. Assuming that this observation is typical of the deposit as a whole, then a drill pattern of 50 m by 50 m should be of sufficient local accuracy to allow preliminary mine planning for both open pit and underground. Definition of vein geometry and the limits of economic mineralisation within veins will require a phase of infill drilling.

Table 14.1 Vein volumes and orientations

Vein	Volume (m3)	Strike length (m)	Down dip width (m)	Thickness (m)	Dip (°)	Strike (°)	Depth below surface (m)
Carmen (2 lenses)	274,400	450	50 - 125	5	80	290	30 and 165
Gloria	743,600	900	200	4	80	350	0 to 50
Gloria Rama	162,300	380	160	3	60	350	0 to 50
Nieta	137,700	280	100	4	60	340	0
Pica (2 lenses)	543,600	900	200 - 400	2	60	350	50 and 100
Martha Alta	15,217,800	3100	600 1400	5-15 (average 5 m)	20	320	0
Martha Baja	2,045,800	1400	200 1000	3	20	330	0
Martha Media	1,030,000	580	200	9	40	360	185
Martha Media Alta	187,700	290	120 180	5-10 (average 5 m)	20	360	200
Martha Ramas (7 lenses)	978,100	2150	100 220	2	10 30	360	30 to 300
Transversal Norte	242,600	500	150	2.5	45	75	0
Transversal Sur	423,700	650	150 400	3	50	90	0
Abundancia (2 lenses)	2,402,100	1700	300 500	4	40	360	0
Alacran	244,000	500	250	2	30	350	60
Esperancita	165,600	450	200	2	45 70	330	0
Luz Elena (2 lenses)	237,300	550	300	2	30 and 65	350	50 and 150
Nueva	292,900	530	400	1.5	5	350	35
Olin (combined with Luz Elena for the estimate)	71,900	125	200	3	30	350	160
Sur	218,500	530	175 250	2	20	350	10

14.4.2 Definition of grade estimation domains

Grade estimation domains are sub-divisions of the geological model and are represented by subsets of the sample data. The creation of grade estimation domains ensures that samples used for estimating a block grade are from the same sample population as the point of estimation. A grade population may be defined by attributes such as spatial location, lithology, mineralisation style, and structural boundaries.

The 18 discrete vein interpretations were used to select the drillhole sample intersections used for grade estimation and to constrain the volume used in the estimation. Non-vein (host rock) lithologies were domained on the basis of lithology. Models representing the andesite, schist, and conglomerate were prepared as separate domains. Further domaining within these lithological units for estimation of dilution grade is made difficult by incomplete sampling down the hole. In many drillholes, only the vein and a small zone adjacent to the vein were sampled and assayed.

14.5 Sample statistics

14.5.1 Summary statistics

Statistics and analyses of sample grade populations within a domain may be biased by clustering of sample data in particular areas of the domain. Clustering of sample data is present at the La Preciosa Project because infill drilling on the original 100 m by 100 m drilling grid has focussed on the higher grade portions of the deposit. There are a number of different methods to decluster data, all of which give different results depending on the declustering parameters. QG used moving window declustering to generate declustered sample statistics and for calculation of variograms. Summary statistics of the vein data are presented in Table 14.2, including the vein orientation, number of intercepts, total metres of sampling, approximate vein dimensions, and core recovery.

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Table 14.2 Summary vein statistics

Vein name	Code	Dip direction (°)	Dip (°)	Projection plane	# intercepts	Total length (m)	Average DH length (m)	Strike length (m)	Dip width (m)	% core recovery	Ag ppm
Carmen	7	201	-81	XZ	8	82.3	10.3	350	180	93.4	169
Gloria	8	255	-83	YZ	45	329.4	7.3	340	180	90.4	197
Gloria Rama	9	264	-61	YZ	23	105.3	4.6	370	170	87.6	114
Nieto	10	250	-60	YZ	8	23.7	3.0	280	150	94.4	87
Pica	11	263	-57, -50	YZ	71	188.5	2.7	580	250	92.2	130
Martha Alta	12	234	-22	XY	450	2950.4	6.5	2400	1100	82.4	131
Martha Baja	13	238	-17	XY	95	182.7	1.9	1300	750	93.7	81
Martha Media	14	268	-30	XY	60	674.1	11.2	500	170	80.9	176
Martha Media Alta	15	275	-24	XY	15	67.9	4.5	290	150	95.9	171
Martha Ramas	16	235	-19	XY	59	169.8	2.9	400	200	90.6	90
Transversal Norte	17	170	-47	XZ	20	73.6	3.7	500	250	90.7	91
Transversal Sur	18	178	-49	XZ	32	92.0	2.9	600	200	89.1	122
Abundancia	19	263, 282	-32, -38	XY	156	761.6	4.9	950	500	88.4	161
Alacran	20	281	-37	XY	37	73.0	2.0	400	150	93.1	79
Esperancita	21	255	-60	YZ	28	57.7	2.1	250	150	90.9	150
Luz Elena	22	265, 280	-30, -29	XY	51	110.9	2.2	300	150	90.9	160
Nueva	23	315	-10	XY	19	32.2	1.7	500	300	93.2	68
Sur	25	270	-15	XY	18	32.8	1.8	530	200	87.2	132
Total					1195	6007.4	5.0			85.4	140

14.5.2 Compositing of sample assay intervals

When estimating mineral resources in two dimensions, sample assay intervals are composited to correctly derive the average grade of a vein intercept, the thickness of the intercept in the direction perpendicular to the projection plane, and the metal accumulation represented by the drill intercept. The average intercept grade is calculated by the length weighted average of all sample intervals in the intercept. This results in a single composite for each vein intersection.

The down hole intercept length cannot be used to calculate the metal accumulation in two dimensions because the angle between the drillhole and the vein is not constant. To correctly represent thickness and accumulation, it is necessary to use the thickness in the direction of the projection. There are two methods commonly used to calculate this thickness for an individual intercept, either by trigonometric calculation assuming a constant orientation of the vein, or by using the actual thickness derived from the wireframe interpretation of the vein. QG adopted the latter approach.

In practice, this is achieved by gridding the top and bottom surfaces of the vein in the projection plane. At each grid node, the thickness of the wireframe can be calculated as the difference in elevation (or easting or nothing, depending on the projection direction) between the top and bottom surfaces. The wireframe thickness at the position of the drillhole intercept centre is then calculated by averaging the four nearest grid nodes. This process ensures that any irregularities due to wireframe triangulation are reduced. Metal accumulation is then calculated as the product of the wireframe thickness and the average grade.

No recovery weighting was used in the calculation of composite grade, which assumes that the lost or missing material has the same grade as the sampled and assayed material, and that no systematic grade bias is present due to core loss. It should be noted that in any individual intercept there may have been preferential loss of high or low grade material, and thus the grade of the recovered material may be biased with respect to the true grade.

A summary of the vein thickness and accumulation statistics is shown in Table 14.3.

Samples in the host rock lithologies were composited at 5 m lengths, in consideration of the block height employed for the deposit wide three dimensional block model, and respecting lithology contacts. Compositing of sample grades in the host rock material is complicated by irregular sampling. Few drillholes have been routinely sampled in host rock material as well as in the vein material, since sampling is usually driven by the presence of veining or mineralisation. This results in a considerable bias in mean sample grade.

Characterisation of the grade of the host rock is most important in open pit mining scenarios where all material must be moved for either processing or transport to and storage in waste dumps.

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Table 14.3 Summary of thickness and accumulation by vein

		Thickness					Ag grade (ppm) × thickness (m)					Au grade (ppm) × thickness (m)				
Vein code	Count	Min. (m)	Max. (m)	Mean (m)	Std Dev.	CV	Min. (m)	Max. (m)	Mean (m)	Std Dev.	CV	Min. (m)	Max. (m)	Mean (m)	Std Dev.	CV
7	8	1.29	4.52	2.07	0.79	0.38	59.2	680	342	166	0.49	0.1	3.04	0.73	0.87	1.19
8	45	1.22	14.36	4.8	3.26	0.68	0.56	3806	866	849	0.98		3.82	1.2	0.99	0.83
9	23	0.87	10.88	3.97	2.87	0.72	65.81	1614	432	390	0.90	0.05	4.78	0.83	1.17	1.41
10	8	0.21	10.84	3.11	3.35	1.08	2.47	1367	269	441	1.64	0.01	2.21	0.43	0.71	1.65
11	71	0.21	18.97	3.67	3.72	1.01	0.96	4375	385	542	1.41		2.33	0.4	0.53	1.33
12	450	0.66	23.45	5.34	4.19	0.78	0.08	7090	536	836	1.56		14.28	1.1	1.89	1.72
13	95	0.23	7.99	2.0	1.45	0.73	0.14	1605	166	329	1.98		4.33	0.28	0.51	1.82
14	60	1.74	31.31	8.21	6.6	0.80	72.71	8218	1177	1,480	1.26	0.21	12.28	3.45	3.63	1.05
15	15	0.7	13.87	4.26	3.28	0.77	41.39	2807	667	833	1.25	0.11	3.47	1	0.96	0.96
16	59	0.32	8.02	2.63	1.88	0.71	1.61	1129	215	286	1.33		2.7	0.37	0.56	1.51
17	20	0.45	8.59	3.51	2.12	0.60	7.55	2798	357	654	1.83	0.02	2.52	0.53	0.61	1.15
18	32	0.37	9.72	3.05	2.08	0.68	19.32	2566	335	400	1.19		1.62	0.43	0.41	0.95
19	156	0.49	19.48	4.68	3.74	0.80	4.84	5895	651	843	1.30	0.03	15.29	1.06	1.76	1.66
20	37	1.18	6.28	2.47	1.09	0.44	3.49	1199	202	233	1.15	0.01	1.59	0.37	0.38	1.03
21	28	0.45	7.79	2.45	1.79	0.73	0.11	1318	376	363	0.96		4.67	0.83	1.02	1.23
22	51	0.22	7.76	2.24	1.72	0.77	7.74	2524	354	479	1.35	0.01	6.12	0.54	0.91	1.69
23	19	0.5	3.26	1.5	0.75	0.50	3.77	459	109	147	1.35	0.01	0.2	0.07	0.06	0.86
25	18	0.52	3.8	1.63	0.87	0.53	31.89	897	221	194	0.88	0.01	2.24	0.39	0.47	1.21
Total	1195			4.30					496					0.95		

14.5.3 Extreme grade value treatment

Extreme grade values are considered those which lie outside the range of values expected based on the distribution of the other samples in each grade estimation domain. Top cuts (also known as capping) of extreme values, by re-setting outlier grades to a lower grade, are done to prevent over-estimation in grade estimation domains when using ordinary kriging as the interpolation technique. When estimating mineral resources in two dimensions, the averaging of grades across the width of the vein reduces the variability of metal grades and reduces the impact of individual high grade assays. However, the accumulation distributions at La Preciosa are still strongly skewed and the possibility of over-estimation is still present.

QG applied the option in Isatis that uses a distance and grade threshold to impose a top cut. Samples lying at a greater distance than the threshold distance from the point of estimation are top cut to the threshold value if the grade exceeds the grade threshold. QG selected thresholds for grades based on the shape of the histogram and the distance threshold based on drill spacing and sample grade continuity. Overall, the restrictions applied are weak and have little impact on the global metal content.

Extreme grades are more of an issue in the host rock lithological units where the sample grade rather than the thickness and accumulation variable have been estimated. There are numerous high grade values present in the host rock lithologies which are not continuous enough to be interpreted as an individual vein structure. Because of the limited geometric distribution of the high grade values, it is important to restrict the influence of these grades. Therefore more restrictive outlier grade thresholds and distance thresholds were defined for samples in the host rock lithologies.

The grade and distances thresholds for both vein and host rock lithologies are given in Table 14.4.

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Table 14.4 Grade and distance thresholds applied to extreme grade values

	Lithology code(1)				**Vein code**																		
Variable	**4**	**5**	**6**	**Variable(2)**	**7**	**8**	**9**	**10**	**11**	**12**	**13**	**14**	**15**	**16**	**17**	**18**	**19**	**20**	**21**	**22**	**23**	**25**
Ag grade	50	50	50	Ag accum.						4000	1000				1000	1000	2000	1000				
Ag distance	25	25	25	Ag distance						25	40				40	40	40	40				
Cu grade	500	500	500	Cu accum.					5000	5000	5000					1500	1500					
Cu distance	100	100	100	Cu distance					50	50	50					50	50					
Pb grade	2500	2500	2500	Pb accum.					50000	50000	50000	50000					50000					
Pb distance	100	100	100	Pb distance					50	50	50	50					50					
Zn grade	2500	2500	2500	Zn accum.					80000	80000	80000	80000		80000			80000					
Zn distance	100	100	100	Zn distance					100	100	100	100		100			100					
As grade	2000	2000	2000	As accum.		2000	2000	2000	2000	2000	2000	2000	2000	2000	2000	2000	2000	2000		2000		
As distance	50	50	50	As distance		35	35	35	35	35	35	35	35	35	35	35	35	35		35		
Au grade	0.5	0.5	0.5	Au accum.	No restriction applied																	
Au distance	100	100	100	Au distance	No restriction applied																	
Ca grade	100	100	100	Ca accum.	No restriction applied																	
Ca distance	100	100	100	Ca distance	No restriction applied																	
Fe grade	100	100	100	Fe accum.	No restriction applied																	
Fe distance	100	100	100	Fe distance	No restriction applied																	
S grade	100	100	100	S accum.	No restriction applied																	
S distance	100	100	100	S distance	No restriction applied																	

Note(1): Lithology code 4 = andesite, 5 = conglomerate and 6 = schist

Note(2): accum. is accumulation (thickness × grade)

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14.6 Variogram analysis and variogram modelling

14.6.1 Variography of veins

Continuity analysis of the spatial correlation of thickness and accumulation between sample pairs was undertaken to determine the major axis of spatial continuity of those variables. Continuity analysis can be undertaken by the calculation of experimental variograms which measure the correlation of variables between sample pairs by direction and by distance.

Two dimensional grade continuity analyses were conducted in Isatis software on thickness and accumulation (grade × thickness) for each grade variable. The variables were visually examined and directional experimental variograms were generated to assess whether any preferred orientation of accumulation or thickness continuity could be identified in two dimensions. A first pass analysis was conducted at the Martha Alta domain, which has the greatest number of intercepts (450).

In plan view there is a slight trend of higher accumulation to the NNE-SSW, but this was not confirmed by the variography or variogram maps. In the absence of any compelling geological reason for a preferred orientation of thicker intercepts of higher metal accumulations, variograms of all variables were modelled as isotropic (a variogram that has the same continuity distance in all directions) within the two dimensional plane.

Two patterns of close spaced holes were drilled in Martha Alta to assess short range geological and grade variability. This close spaced data was used to check the modelling of short range continuity of thickness and Ag, Pb, and Zn accumulation variables. For the thickness variable, variograms calculated from the short range data are very similar between the east-west and north-south directions and closely agree with the omni-direction variogram calculated on the regular 100 m spaced data. The variogram is well structured.

Variograms on the silver metal accumulation variable were poorly structured as expected considering the highly skewed nature of the accumulation variability. A normal scores transform was applied to the metal accumulation variable to improve the quality of the variogram.

Experimental variograms were then calculated on the less well informed Martha Media and Abundancia domains and checked against the models fitted to Martha Alta. The experimental variograms calculated on these less well informed domains are moderately well structured, robust to changes in calculation parameters, and are consistent with those derived from the well informed Martha Alta.

Attempts to derive structured variograms in the other domains were hampered by low data numbers. It was considered reasonable to adopt the variogram models defined on Martha Alta, Martha Media, and Abundancia for these domains, with the model to adopt assigned on the basis of mean silver grade and similarity to the other variables. Table 14.5 show the assignment of variogram models by vein code and by accumulation variable and Table 14.6 shows the variogram parameters of each variable and the vein code the variograms were calculated from. All variograms were isotropic, which means that the continuity is the same in the two dimensions.

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Table 14.5 Assignment of variogram models

Vein name	Vein code	Ag	Au	Cu	Pb	Zn	As	S	Fe	Ca
Carmen	7	12	12	12	12	12	12	12	12	12
Gloria	8	14	14	14	14	14	14	12	12	12
Gloria Rama	9	12	12	12	12	12	12	12	12	12
Nieto	10	12	12	12	12	12	12	12	12	12
Pica	11	14	14	14	14	14	14	12	12	12
Martha Alta	12	12	12	12	12	12	12	12	12	12
Martha Baja	13	12	12	12	12	12	12	12	12	12
Martha Media	14	14	14	14	14	14	14	12	12	12
Martha Media Alta	15	12	12	12	12	12	12	12	12	12
Martha Ramas	16	12	12	12	12	12	12	12	12	12
Transversal Norte	17	12	12	12	12	12	12	12	12	12
Transversal Sur	18	19	19	19	19	19	19	12	12	12
Abundancia	19	19	19	19	19	19	19	12	12	12
Alacran	20	19	19	19	19	19	19	12	12	12
Esperancita	21	12	12	12	12	12	12	12	12	12
Luz Elena	22	12	12	12	12	12	12	12	12	12
Nueva	23	14	14	14	14	14	14	12	12	12
Sur	25	12	12	12	12	12	12	12	12	12

Table 14.6 Variogram parameters applied to vein estimates

Variable	Vein code	C0(1)	Nugget % sill	C1(2)	Range 1 (m)	C2(3)	Range 2 (m)
Length (m)	12	3.2	20	3.3	40	9.7	230
	14	5.0	15	11.0	40	17.0	180
	19	1.0	21	1.1	45	2.8	240
Ag accumulation	12	300	39	74	100	396	350
	14	876	30	637	55	1380	145
	19	60	26	44	35	126	190
Au accumulation	12	0.9	23	0.76	100	2.3	450
	14	2.7	21	10.3	105		
	19	0.05	9	0.20	55	0.31	180
Cu accumulation	12	50	6	720	250		
	14	33	25	100	135		

Variable	Vein code	C0(1)	Nugget % sill	C1(2)	Range 1 (m)	C2(3)	Range 2 (m)
	19	46	45	55	180		
Pb accumulation	12	30	20	30	35	93	290
	14	154	36	272	130		
	19	15	36	10	150	17	150
Zn accumulation	12	130	21	147	95	343	348
	14	140	15	772	110		
	19	20	17	50	20	50	120
As accumulation	12	1.2	37	0.52	60	1.52	330
	14	2.4	37	1.05	60	3.05	120
S accumulation	12	10	19	42	160		
Fe accumulation	12	8	14	27	150	21	420
Ca accumulation	12	10	14	35	150	27	420

14.6.2 Variography of host rock

Variography for the host rock lithologies was carried out on the 5 m composites. Omni-directional variograms were calculated for Ag, Au, Cu, Pb, and Zn. For As, Ca, Fe, and S, the down hole variogram was used to establish ranges perpendicular to the general structural trend, while the omni-directional (between drillhole) variogram was used to establish continuity parallel to the general structural trend. Extreme values were dampened to restrict the influence of outliers and to prevent them from destabilising the variogram, to better reveal the average continuity. The majority of variograms were moderately well structured. Variogram parameters used in estimating grades in the host rock lithologies are given in Table 14.7.

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Table 14.7 Variogram parameters applied to host rock estimates

		Nugget			Structure 1				Structure 2			
Lithology	**Variable**	**C0**	**C1**	**Model Type(1)**	**Major**	**Intermediate**	**Minor**	**C2**	**Model Type**	**Major**	**Intermediate**	**Minor**
Andesite	Ag	128	350	Sph	71	71	71	0.001	Sph	71	71	71
	Au	0.00025	0.00053	Sph	75	75	75	0.001	Sph	75	75	75
	Cu	80	15	Sph	100	100	65	85	Sph	140	140	100
	Pb	12,934	31,012	Sph	65	65	65	0.001	Sph	65	65	65
	Zn	35,363	168,385	Sph	70	70	70	0.001	Sph	70	70	75
	Ca	0.3	1.14	Exp	80	80	80	0.4	Sph	800	800	200
	Fe	0.08	0.29	Exp	50	50	33	0.7	Sph	180	180	150
	As	20500	3800	Exp	50	50	33	4500	Sph	180	180	150
	S	0.28	0.14	Exp	120	120	50	0.12	Sph	250	250	175
Conglomerate	Ag	150	245	Sph	71	71	71	0.001	Sph	71	71	71
	Au	0.0005	0.00105	Sph	75	75	75	0.001	Sph	75	75	75
	Cu	1000	650	Sph	100	100	65	700	Sph	140	140	100
	Pb	44,238	56,622	Sph	65	65	65	0.001	Sph	65	65	65
	Zn	122,119	305,766	Sph	70	70	70	0.001	Sph	70	70	75
	Ca	1.39	5.27	Exp	80	80	80	1.85	Sph	200	200	200
	Fe	0.08	0.29	Exp	40	40	20	0.7	Sph	150	150	120
	As	7500	14,000	Exp	25	25	25	16500	Sph	150	150	150
	S	0.25	0.22	Exp	120	120	50	0.1	Sph	250	250	175
Schist	Ag	5.36	7	Sph	71	71	71	0.001	Sph	71	71	71
	Au	0.0002	0.00023	Sph	75	75	75	0.001	Sph	75	75	75
	Cu	2570	2820	Sph	100	100	65	0.001	Sph	140	140	100
	Pb	17,300	55,600	Sph	65	65	65	0.001	Sph	65	65	65

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		Nugget			Structure 1					Structure 2		
Lithology	**Variable**	**C0**	**C1**	**Model Type(1)**	**Major**	**Intermediate**	**Minor**	**C2**	**Model Type**	**Major**	**Intermediate**	**Minor**
	Zn	50,000	50,000	Sph	70	70	70	31000	Sph	70	70	75
	Ca	0.367	1.2	Exp	20	20	20	1.09	Sph	150	150	60
	Fe	0.8	1.2	Exp	80	80	50	0.6	Sph	300	300	150
	As	14,800	27,500	Exp	20	20	15	23500	Sph	120	120	50
	S	0.25	0.3	Exp	120	120	50	0.0001	Sph	250	250	175

Note(1): Sph = spherical model, Exp = exponential model.

14.7 Estimation parameters

Kriging neighbourhood analysis (KNA) was performed to determine the optimum kriging parameters. KNA is the process of undertaking multiple ordinary kriged estimates using a variety of block sizes and search neighbourhood parameters (such as minimum and maximum sample numbers) and selecting the configuration that optimises estimation parameters that minimise conditional bias, or the degree of over smoothing of an estimate compared to the true grade of a deposit.

14.7.1 Block size selection

The full three dimensional volume model required for mine planning uses a cell size of 12.5 m in the easting, 12.5 m in the northing, and 5 m in elevation. This cell size was chosen in collaboration with Pan American, considering the likely mining scenarios and the accuracy of representing vein geometries. Vein coding and vein proportions were transferred from the three dimensional block model to two dimensional grids established in the XY, XZ, and YZ planes, with dimensions that match the three dimensional model.

Drill spacing at La Preciosa is dominated by two main patterns, a regular 100 m by 100 m grid and infill drilling on 50 m by 50 m or better. At the dimensions of the chosen block size, a very small proportion of cells within the 100 m by 100 m drill pattern (roughly one in every 65) will actually contain a drillhole intersection. Given the drill spacing and spatial continuity of the variables, these block sizes are considered too small for reliable direct estimation using ordinary kriging.

Consequently larger parent cell grids, in which the grade estimation is performed, was established over the underlying model. In the XY plane, two grids were established, one at 50 m by 50 m for use in areas of broad spaced drilling and one at 25 m by 25 m where drilling is close spaced. This choice of parent cell grid size appropriately considers drillhole spacing, the dimensions of the underlying grid, minimisation of conditional bias, and results in improvement in estimation quality.

14.7.2 Estimation and sample search parameters

Variogram parameters from the modelled variograms shown in Table 14.6 and Table 14.7 were applied to the estimates of veins and host rock lithologies, respectively. Because of the isotropic variogram models fitted to the veins as well as the visual confirmation of a lack of any anisotropy in grade and thickness continuity, isotropic sample searches of 300 m by 300 m were used in the estimation of grades within the veins.

The minimum and maximum sample numbers required to return an estimate in blocks within the vein are shown in Table 14.8. The relatively low number of samples is considered to be at the lower end of the range of acceptable neighbourhoods but is considered acceptable because the estimates are made using full vein width metal accumulation composites and effectively each intercept is an average of multiple samples across a vein width that embodies a greater support than is provided by a typical composite length. This low number of maximum sample number effectively reduces the search ellipse to one or two sections of drillholes away from the block cell being estimated.

For host rock lithologies, grade estimation within the 50 m thick skin of rock surrounding the veins took place within an anisotropic search ellipsoid oriented parallel to the average orientation of the vein set and was restricted to samples lying within the skin. The ratios of anisotropy were designed to ensure that samples were preferentially selected from locations parallel to the contact. Grades estimated outside of the 50 m skin used an isotropic search. Search parameters for estimations made outside the 50 m skin are tabulated in Table 14.9.

All searches were visually confirmed in three dimensions in Isatis software to ensure that the rotations were correctly applied and to ensure that the samples used for estimation of a block were as intended (e.g., that the domain boundaries were honoured during the sample search).

Table 14.8 Vein search parameters

		Vein code																	
Variable	**Sample numbers**	**7**	**8**	**9**	**10**	**11**	**12**	**13**	**14**	**15**	**16**	**17**	**18**	**19**	**20**	**21**	**22**	**23**	**25**
Length	Minimum	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4
	Maximum	8	8	8	8	8	8	8	8	8	8	8	10	10	8	10	8	8	8
Ag accum.	Minimum	2	2	2	2	2	2	2	2	2	2	3	2	2	3	2	2	2	2
	Maximum	16	14	16	16	14	16	16	14	16	16	16	12	12	16	12	16	16	16
Au accum.	Minimum	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
	Maximum	12	10	12	12	10	12	12	10	22	12	12	14	14	12	14	12	12	12
Cu accum.	Minimum	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
	Maximum	8	8	8	8	8	8	8	8	8	8	8	8	9	8	8	10	8	8
Pb accum.	Minimum	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
	Maximum	12	6	12	12	6	12	12	6	12	12	12	12	12	12	12	12	12	12
Zn accum.	Minimum	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
	Maximum	9	9	12	12	9	12	12	10	10	12	12	8	8	10	8	12	10	10
Ca accum.	Minimum	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
	Maximum	9	9	12	12	9	12	12	10	10	12	12	8	8	10	8	12	10	100
Fe accum.	Minimum	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
	Maximum	9	9	12	12	9	12	12	10	10	12	12	8	8	10	8	12	10	10
As accum.	Minimum	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3
	Maximum	10	10	10	16	10	8	16	10	10	16	10	16	10	10	10	12	10	10
S accum.	Minimum	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4
	Maximum	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8

Accum. is the metal accumulation, grade × thickness.

Table 14.9 Host rock search parameters

		Vein skin parameters									Outside vein skin parameters								
		Rotation(1)			Search						Rotation			Search					
Variable	**Domain**	**Maj**	**Int**	**Min**	**1**	**2**	**3**	**Sectors(2)**	**Min(3)**	**Max**	**Maj**	**Int**	**Min**	**1**	**2**	**3**	**Sectors**	**Min**	**Max**
Ag	Andesite	140	28	90	500	500	50	1	2	12	0	0	0	500	300	30	1	2	12
	Conglomerate	166	45	90	500	500	50	1	2	12	0	0	0	500	300	30	1	2	12
	Schist	78	48	90	500	500	50	1	2	12	0	0	0	300	300	30	1	2	12
Au	Andesite	140	28	90	500	500	50	1	2	12	0	0	0	300	300	30	1	2	12
	Conglomerate	166	45	90	500	500	50	1	2	12	0	0	0	300	300	30	1	2	12
	Schist	78	48	90	500	500	50	1	2	12	0	0	0	300	300	30	1	2	12
Cu	Andesite	140	28	90	500	500	50	1	2	12	0	0	0	300	300	30	1	2	12
	Conglomerate	166	45	90	500	500	50	1	2	12	0	0	0	300	300	30	1	2	12
	Schist	78	48	90	500	500	50	1	2	12	0	0	0	300	300	30	1	2	12
Pb	Andesite	140	28	90	500	500	50	1	2	12	0	0	0	300	300	30	1	2	12
	Conglomerate	166	45	90	500	500	50	1	2	12	0	0	0	300	300	30	1	2	12
	Schist	78	48	90	500	500	50	1	2	12	0	0	0	300	300	30	1	2	12
Zn	Andesite	140	28	90	500	500	50	1	2	12	0	0	0	300	300	30	1	2	12
	Conglomerate	166	45	90	500	500	50	1	2	12	0	0	0	300	300	30	1	2	12
	Schist	78	48	90	500	500	50	1	2	12	0	0	0	300	300	30	1	2	12
Ca	Andesite	140	28	90	250	250	50	4	4	6	0	0	0	250	250	100	4	4	6
	Conglomerate	166	45	90	250	250	50	4	4	6	0	0	0	250	250	100	4	4	6
	Schist	78	48	90	250	250	50	4	4	6	0	0	0	250	250	100	4	4	6
Fe	Andesite	140	28	90	250	250	50	4	4	5	0	0	0	250	250	50	4	4	5
	Conglomerate	166	45	90	250	250	50	4	4	5	0	0	0	250	250	50	4	4	5
	Schist	78	48	90	250	250	50	4	4	6	0	0	0	250	250	50	4	4	6
As	Andesite	140	28	90	250	250	50	4	4	7	0	0	0	250	250	100	4	4	7

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		Vein skin parameters									Outside vein skin parameters								
		Rotation(1)			Search						Rotation			Search					
Variable	Domain	Maj	Int	Min	1	2	3	Sectors(2)	Min(3)	Max	Maj	Int	Min	1	2	3	Sectors	Min	Max
	Conglomerate	166	45	90	250	250	50	4	4	7	0	0	0	250	250	100	4	4	7
	Schist	78	48	90	250	250	50	4	4	7	0	0	0	250	250	100	4	4	7
S	Andesite	140	28	90	250	250	50	4	4	8	0	0	0	250	250	100	4	4	8
	Conglomerate	166	45	90	250	250	50	4	4	9	0	0	0	250	250	100	4	4	9
	Schist	78	48	90	250	250	50	4	4	8	0	0	0	250	250	100	4	4	8

Note(1): Rotations are in the major, intermediate, and minor directions

Note(2): Sectors =1 no sectorial search, Sector = 4 search ellipse divided into four quadrants

Note(3): Min and Max = the minimum and maximum samples used to return an estimate in the block cell.

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14.7.3 Grade interpolation and boundary conditions

Because of the narrow tabular planar geometries of the veins a two dimensional estimation approach was adopted in preference to a three dimensional approach to eliminate problems with high kriging weights on the margin of the structures, to reduce local and global estimation bias, and to obtain a higher quality variogram with improved structure and greater continuity. In a two dimensional estimation approach both thickness and accumulation (thickness multiplied by grade) are estimated and grade is calculated by dividing accumulation by thickness. If a strong relationship exists between vein width and grade, a co-estimation technique is required to honour the relationship. At La Preciosa, there is no strong or consistent relationship between vein width and grade, and therefore thickness and accumulation were estimated independently.

Once grade estimation within the veins was complete, the grades were transferred back to the block models in three dimensions. The use of parent cells means that certain areas of the block model will have identical grades and thus the grade variability in the three dimensional model will be lower than the true grade variability in volumes of this size. This is an important consideration in mine planning, which should be carried out on suitably sized aggregates of blocks.

In host rock lithologies, estimation of grade for mining dilution was undertaken by ordinary kriging in the 12.5 m by 12.5 m by 5 m blocks in three dimensions. The main purpose of the host rock estimate is to characterise the grades of material that may need to be moved to access economic material by open pit mining. The ability to do the estimate reliably is hampered by selective sampling in the host rock lithologies. Consequently the risk of conditional bias and smoothing in performing direct estimation of small blocks is considered acceptable. Within the host rock volume, estimates should not be used as a basis for local prediction in mine planning, but should reliably predict gross characteristics in sufficiently large volumes. This uncertainty has been reflected in the classifications applied to host rocks.

The association of alteration haloes with mineralisation and the selective sampling practice of sampling intercepts in veins and vein margins only have resulted in strongly biased and spatially clustered data sets in the host rock lithologies. To overcome the clustering of samples, a two-step estimation process was developed:

- Firstly veins were divided into three general trends corresponding to the three main vein trends, and blocks and samples in host rock material within 50 m of a vein were flagged with the vein code and the distance to the closest vein. Blocks within 50 m of a vein were estimated using an anisotropic search ellipse oriented parallel to the general orientation of the vein set that the vein belonged to, and only using samples from within that 50 m skin. The intention of this search strategy was to minimise the effect of selective sampling on vein margins and to prevent smearing of higher grades on vein margins away from the veins.

- Blocks further than 50 m from a vein were estimated using an isotropic search, reflecting the lack of a preferred orientation of grade continuity, and only using samples located more than 50 m from a vein. To reduce the number of samples used from any one drillhole in the estimate, a flat lying anisotropic search combined with a limit on the maximum number of samples taken from a single hole was employed. Sectorial searching was used for some variables to ensure sample selection was spatially distributed, by dividing the search ellipse into four quadrants and limiting the number of samples located in any quadrant during estimation.

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Figure 14.2 shows an oblique view of the estimation zones of non-vein material within 50 m of the vein.

Figure 14.2 Oblique view of host rock estimation zones within 50 m of veins

14.8 Density

A total of 83,733 density measurements are available for the La Preciosa Project, 65,318 made by Orko and the remainder by Pan American. QG reviewed a detailed analysis of a testing programme undertaken by Pan American on bulk density determination to address uncertainties about the quality of the Orko density measurements. QG considers the density analysis work to be of high quality and supports the conclusions of the analysis.

The conclusions state that the Orko specific gravity data can be used for estimation of specific gravity provided the estimation method incorporates the high variability of the data, likely through a large sample search for estimation. The resulting estimation may be slightly underestimated by around 1%. After kriging the estimation of specific gravity, the estimate of bulk density for vein material should include the formula

BDv = SGv × (1-v), where v = (6.9011   0.012 × depth)/100, and where v cannot be less than 0.005.

The bulk density of non-vein material should be 0.99 × SG. Where no data exists, bulk density will be assigned at 0.99 × the average specific gravity for the respective rock types.

QG implemented these recommendations for the estimation of specific gravity and bulk density. Estimation of specific gravity in the veins was carried out using ordinary kriging in two dimensions using vein composites. Estimates used an anisotropic variogram with a 30% nugget and ranges of 90 m at 70% of total variability and 1,100 m

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at 100% of total variability. A minimum of two composites and a maximum of 20 composites were required to return an estimate. Blocks not estimated in this process were assigned a default specific gravity of 2.51. Bulk density for the veins was then calculated from the estimated or assigned specific gravity values using the depth based regression relationship described above.

Specific gravity of the andesite, conglomerate, and schist lithologies was estimated by ordinary kriging in three dimensions and then converted to bulk density using a fixed multiplier of 0.99.

The variogram parameters for the specific gravity estimate are shown in Table 14.10 and the search parameters are shown in Table 14.11.

Table 14.10 Specific gravity variogram parameters

		Nugget		**Structure 1 (spherical)**				**Structure 2 (spherical)**		
Lithology	**Projection**	**C0**	**C1**	**Maj(1)**	**Int**	**Min**	**C2**	**Maj**	**Int**	**Min**
Veins	2D	0.005	0.0065	90	90		0.005	1100	1100	
Andesite	3D	0.008	0.003	90	90	90	0.0045	1100	1100	1100
Conglomerate	3D	0.00896	0.00926	230	230	230				
Schist	3D	0.007	0.0031	170	170	170	0.002	1100	1100	1100

Note(1): Maj, Int., and Min are the major, intermediate, and minor directions.

Table 14.11 Specific gravity search parameters

		Search distances (m)			**Sample numbers**	
Lithology	**Projection**	**Maj(1)**	**Int**	**Min**	**Minimum**	**Maximum**
Veins	2D	300	300		2	20
Andesite	3D	500	500	100	2	24
Conglomerate	3D	500	500	100	2	24
Schist	3D	500	500	100	2	24

Note(1): Maj, Int., and Min are the searches in the major, intermediate, and minor directions.

14.9 Estimation validation

Estimation validation was undertaken to detect errors in the implementation of the estimation methodology and to demonstrate that the estimates are statistically representative of the input data and that the estimation methodology choice was appropriate. The estimates were visually reviewed in three dimensions to assess whether all intended blocks had returned an estimate, whether the block values made sense with respect to input sample values, and that expected grade trends had been reproduced.

Global declustered mean input sample statistics were compared with the mean estimated value statistics for each vein and averaged for all veins. Table 14.12 shows the global comparison of estimated and declustered input values averaged for all veins. There are some large differences between estimated and input data mean values.

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Because of low composite numbers in some domains, the mean value is not well characterised by the data, and the kriged estimate can deliver a domain-wide average value that differs to the mean of the input data. There are also differences due to the treatment of extreme values. In particular, estimated As, Cu, and Pb grades are substantially lower than the input data, which is due to the restriction placed on extreme grades during estimation. The grade of both Pb and Cu is low, and the mean is strongly affected by a few extreme sample grades.

Table 14.12 Comparison of estimated and input values

Variable	# composites	Composite mean value	# estimates	Estimated mean value	% difference
Ag ppm	1,195	110	125,537	106	-4
As ppm	1,181	391	125,537	356	-9
Au ppm	1,195	0.19	125,537	0.19	0
Ca%	1,180	1.99	125.537	1.90	-5
Cu ppm	1,187	167	125,537	136	-18
Fe%	1,178	3.14	125,537	2.96	-6
Pb ppm	1,194	2107	125,537	1847	-12
S%	553	0.88	92,736	0.76	-15
Zn ppm	1,194	3502	125,537	3280	-6
Thickness	1,196	4.31	125,537	4.91	14
Specific gravity	622	2.52	13,198	2.52	0

The estimates were also validated by comparison of input and estimated grade trends in swath or slice plots. No problems with the comparison between grades were observed and the degree of smoothing appears reasonable. An example swath plot showing sample input and estimated silver grades for the Martha Alta vein is shown in Figure 14.3.

A global grade tonnage distribution for Ag grades was derived from a discrete Gaussian change of support model of the raw Ag variable, and compared with the back-calculated Ag grade estimates. Overall the change of support model indicates that tonnage and grade estimates of silver mineral resources derived from ordinary kriging are reasonably close to those derived from the theoretical model of change of support, particularly at higher cut-off grades.

Figure 14.3 Swath plot showing sample input and estimated silver grades for the Martha Alta vein

14.10 Estimation post-processing

The estimates were loaded back into the Minesight three dimensional block model from Isatis software and a number of model post-processing steps were performed, including infilling non-estimated blocks with default values.

The estimates were then treated for minimum mining widths and planned mine dilution in Minesight mining software, based on parameters provided by Pan American. The following steps were followed to create diluted grade estimates:

- A minimum of 1.0 m dilution was applied to the vein volume as a skin of 0.5 m thickness of the host rock lithologies on both the footwall and hanging wall side of the vein wireframe solid.

- Any area of the steeply dipping vein with a thickness less than 2 m was expanded to 2 m and any area of the flat lying veins with a thickness less than 3 m was expanded to 3 m.

- The expanded wireframe solid was used to code the vein as diluted (VDIL) and to code the proportion of the block inside the expanded solid (VDIL%).

- Diluted grade calculations were then performed on steeply dipping veins as follows:

- For blocks with a thickness greater than 2 m, the diluted vein grade was calculated as (wireframe thickness × undiluted vein grade + 1 × dilution grade)/(wireframe thickness + 1).

- For blocks with a thickness less than 2 m, the diluted vein grade was calculated as (wireframe thickness × undiluted vein grade + (2 wireframe thickness) × dilution grade).

- Diluted grade calculations were performed on flat dipping veins as follows:

- For blocks with a thickness of greater than 3 m, the diluted vein grade was calculated as (wireframe thickness × undiluted vein grade + 1 × dilution grade)/(wireframe thickness + 1).

- For blocks with a thickness of less than 3 m, the diluted vein grade was calculated as (wireframe thickness × undiluted vein grade + (3 wireframe thickness) × dilution grade).

Dilution grade is derived from the average grade of the waste lithology grade estimates in the blocks surrounding the veins.

14.11 Comparison with previous estimates

A number of significant changes to the understanding of the La Preciosa deposit have taken place since the previous mineral resource estimates reported in March, 2009 (MDA, 2009). These changes include:

- Nearly a 50% increase in the number of available drillholes, mainly as infill drilling towards the north of the deposit, which has contributed to the confidence in the mineral resource estimate.

- Updated geological interpretations building on the new drillhole intersections and growing understanding of the controls on mineralisation.

- Consideration for likely minimum mining width requirements based on the results of the preliminary economic assessment disclosed in this technical report as well as the application of planned dilution.

- Updated methodology for estimating bulk density.

- Alternate mineral resource estimation methodology which better reduces bias and error in the estimate.

- Alternate cut-off grades for mineral resource reporting based on changing metal prices and likely mining methods.

Considering these changes, above a 100 ppm Ag cut-off grade, the 2010 mineral resource estimate presents a 45% increase in tonnes, a 17% decrease in silver grade, a 20% increase in contained silver ounces, a 14% increase in gold grade, and a 67% increase in contained gold ounces.

14.12 Depletion for historical mining

Historical mine-workings are known to be present in the Gloria and Abundancia veins. Best known are two historical drifts that were expanded by Luismin in 1981 and 1982. Luismin reportedly mined around 11,000 t from these underground openings, which was never processed and remains on surface at this time. The only known diagrams of these openings are in the form of paper long section outlines of the drifts and apparently hand-worked stopes above these. The accuracy of these mined outlines is not known. QG understands that Luismin did not undertake any stoping and that the stopes were part of the older undocumented historical workings.

To deplete the mineral resources for previous mining, QG used the long section outlines to remove the mined volume of Gloria and Abundancia vein material from within the shapes. The vein percentage code in the model was adjusted to account for the mined material. This resulted in removing 62,000 m3 (around 168,000 t) from Gloria and 112,000 m3 (around 300,000 t) from Abundancia. This volume is an order of magnitude higher than the volume reportedly mined by Luismin. Based on this, and the lack of significant tailings or mine waste on surface, it is QGs opinion that the depletion volume to account for historic workings is probably highly conservative.

14.13 Mineral resource classification

Recent drilling by Pan American has confirmed that the deposit scale geological interpretation is now well understood. In QG s opinion, close spaced drilling has confirmed that vein geometry and grade continuity within veins is sufficiently predictable, at a regular spacing of 50 m by 50 m or better, to allow reasonably local mine design and planning to be carried out. QG is thus satisfied that drilling at a regular spacing of 50 m by 50 m or better is sufficient to allow definition of Indicated mineral resources. At this stage, QG consider that uncertainty about core sample losses and in situ bulk density preclude classifying any mineral resources at La Preciosa as Measured. Within the remainder of veins with wider spaced drillhole spacing, mineral resource estimates are considered sufficiently robust to support classification as Inferred mineral resources.

The presence and results of the QAQC data collected to assess the accuracy and precision of the assay data used to estimate mineral resources, the accuracy of down hole surveys to appropriately define the position of the drillhole trace in three dimensions, the continuity of mineralisation demonstrated through the geological interpretation and by the variograms, and the quality of the estimation have all been considered when assigning classification to the mineral resources.

A series of polygons were constructed to encompass those areas of veins defined as Indicated. The remaining vein estimates were classified as Inferred mineral resources. Host rock material was not classified. Only a minor amount of discontinuous mineralisation is present in the host rock lithologies.

14.14 Mineral resource tabulation

The mineral resource estimate for La Preciosa effective 30 June 2011 is shown in Table 14.13. The mineral resource has been reported at cut-off grades that are likely to be applicable to both underground and open pit mining operations, based on the findings of this preliminary economic assessment, using metal prices of $25/oz silver and $1,250/oz gold. These cut-off grades were determined on 30 June 2011. The elevation for the division of open pit and underground mineral resources is roughly at 1920 m elevation, and this elevation was used for reporting mineral resources. Grade tonnage curves are shown in Figure 14.4 and present the tonnes and grade of the mineral resource estimate above a range of cut-off grades.

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Table 14.13 La Preciosa Project mineral resource estimate effective 30 June 2011

Mining method	Classification	Cut-off grade Ag ppm	Tonnes (millions)	Ag ppm	Silver million ounces	Au ppm	Gold thousand ounces	Silver equivalent ppm	Silver equivalent million ounces
Open pit	Indicated	35	10.9	129	45	0.19	66	139	49
Open pit	Inferred	35	7.6	74	18	0.13	31	81	20
Underground	Indicated	85	13.9	152	68	0.35	156	170	76
Underground	Inferred	85	7.6	117	28	0.21	52	128	31
Total	Indicated		24.8	142	113	0.28	222	156	124
	Inferred		15.2	96	46	0.17	83	105	51

Notes:

Mineral resources that are not mineral reserves do not have demonstrated economic viability. CIM (2010) defines a mineral reserve as the economically mineable part of a Measured or Indicated mineral resource demonstrated by at least a preliminary feasibility study. No mineral reserves have been estimated. Mineral resources have accounted for minimum mining width and planned mining dilution.

The estimate of mineral resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues, but no such issues have been identified at this time.

Tonnes, grades, and ounces have been rounded and this may have resulted in minor discrepancies in the totals. Grades are expressed in parts per million (ppm) which is equivalent to grams per tonne (g/t).

Cut-off grades are based on operating cost estimates and metal prices of $25 per ounce silver (Ag) and $1,250 per ounce gold (Au). Metal prices are based on a weighted average of historical three year average daily silver prices and a two year future price forecast.

The division between open pit and underground mineral resources is set on a horizontal level at the 1920 m elevation, which is considered close to optimum at the metal prices and operating costs assumed in this preliminary economic assessment.

Silver equivalent grade values assume a gold to silver ratio of 50 to 1 based on the assumed metal prices. The metallurgical recoveries and refining charges are assumed to be the same for silver and gold for the purposes of the equivalence calculation only.

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Figure 14.4 Grade tonnage curves by mining method and by classification

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14.15 Recommendations

The geological interpretation at La Preciosa is the key driver of mineral resource and future mineral reserve estimates. Pan American should continue to develop their geological understanding of the deposit and translate this into geological interpretations of mineralisation and prepare wireframe representations of the interpretations. In particular, the investigation should focus on the relationship between mineralisation and local structural controls.

Recommendations with respect to QAQC data discussed in section 11.2.3 should be followed. Future drilling campaigns should continue to include the submission of certified standards, blanks, and duplicate samples to ensure the reliability of sample assays for future mineral resource estimates.

Given the large lateral extent and limit width of the majority of veins, the two dimensional mineral resource estimation methodology is considered to be an appropriate method for estimation of mineral resources at La Preciosa. In the zone of thickened veining in the Martha system, consideration should be given to the development of a local three dimensional mineral resource model.

QG considers that there remains significant exploration potential at La Preciosa, including the northern extension of the system.

Within the rock mass at La Preciosa, numerous veins exist that cannot be confidently interpreted from section to section. In open pit mining, it is likely that there is considerable upside potential from un-modelled veins. The ability to realise this upside depends on whether potential metal gained outweighs the additional cost and time in defining and mining these veins. A programme of reverse circulation grade control drilling in advance of mining may be required to define the potential gains.

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15 Mineral reserve estimates

No mineral reserves have been estimated at this time.

16 Mining methods

16.1 Geotechnical parameters

The geotechnical characterisation of the deposit and the surrounding host rock was conducted by Golder Associates and was based primarily on detailed geotechnical data collected from diamond drill hole core during a field programme conducted in 2010 and by observation during two site visits (Golder, 2010). Where insufficient geotechnical data exists, information obtained from core photographs and geotechnical data collected from the 2009 drilling campaign, including RQD, field strength rating, and core recovery was used.

The quality of the rock mass was classified from the drillhole data using the rock mass rating (RMR) system, and in general, the rock is classified as fair to good rock quality. Zones of weak, altered rock are very rare and the rock mass quality is governed primarily by fracture intensity. The rock quality of the veins is observed to be highly variable in the drill core. In general, the veins are the strongest unit and are moderately to highly fractured. The rock quality of the Martha veins is better, in terms of both RQD and RMR values, than that of the other veins. Highly fractured zones are more commonly encountered within the veins than within the units above or below with the exception of the volcanic rocks in the immediate hanging wall of the Martha vein.

For the portion of the deposit that is to be mined using underground methods, Golder have made estimates of the mine excavation dimensions and mining method, and have made ground support recommendations based on empirical formulae and their experience of rock of similar quality. Those recommendations are specified by Golder as being suitable for pre-feasibility level only and will require more detailed analysis for further studies and assessment to the level of confidence required of feasibility studies.

It is the opinion of Snowden that the recommendations put forth by Golder are appropriate for the mine plan and operating cost estimates presented in this preliminary economic assessment. It is recommended that further geotechnical assessments be conducted in order to better understand the likely behaviour of the rock as mining advances in the highly fractured zones in the veins and the immediate hanging wall of the veins. The greater certainty required as the project advances to feasibility level may necessitate the selection of an underground mining method with cemented backfill in order to provide additional support to allow safe mining in the highly fractured zones. The selection of a higher cost underground mining method is likely to negatively impact the cut-off grade and the mine plan, and ultimately on the project s economic returns.

16.1.1 Data collection

In January 2010, Pan American embarked on a field drilling campaign to collect additional geological and geotechnical data. Golder established geotechnical logging procedures and provided training to Pan American staff at the outset of this programme. Geotechnical parameters for the estimation of RMR were collected for 56 of the 2010 holes. In addition, oriented core data were collected for select structures only, including faults, shears, and any altered/infilled joints. One structure from each joint set was identified within each run.

As part of the 2010 drilling programme, Golder engineers undertook two site visits to provide training and to conduct quality checks on the geotechnical logging.

Geotechnical mapping of available rock exposures was carried out by Golder engineers during the site visits. The collected data were used to verify and augment the structural data collected from the diamond drill core.

In May 2010, Pan American provided data on groundwater elevations based on an IP survey and groundwater level measurements taken in open boreholes.

16.1.2 Open pit mining parameters

Golder (2010) makes recommendations for open pit walls based on 18 m high benches (double 9 m high benches) as shown in Table 16.1. The same slopes were recommended for all mining areas. As part of this preliminary economic assessment, the bench height has changed to 20 m (double 10 m high benches) and the berm width has been scaled accordingly to preserve the inter-ramp angle.

Table 16.1 Open pit bench parameters

Item	Units	Golder (2010)	PEA Study
Bench height	m	18	20
Face angle	degrees	70	70
Berm width	m	8	9
Inter-ramp angle	degrees	51	51

16.1.3 Underground mining parameters

Underground production dimensions

The dimensions for underground production voids used in this preliminary economic assessment have been developed based on the recommendations made by Golder in 2010.

For room and pillar mining (with or without fill):

- Maximum room dimensions will be 5.0 m wide by 5.0 m high.

- Pillar dimensions will be 5.0 m wide with a square profile.

- Where the vertical height of the panel is greater than 5 m, fill will be used for the lower portion and the upper portion will be mined using overhand techniques.

Ground support

As the rock mass is highly fractured in many areas ground support will need to accommodate a variety of conditions. Golder (2010) recommends the following ground support for production areas:

- 1.8 m bolts on a 1.2 m pattern in backs and walls to within 1 m of the floor.

- No. 6 welded mesh installed in backs and walls to within 2 m of the floor.

- 5 cm of shotcrete applied in-cycle where the rock is highly fractured (approximately 50 to 75% of the production openings).

In waste development, the ground support recommendations are:

- 2.4 m bolts on a 1.2 m pattern in backs.

- 1.8 m bolts on a 1.2 m pattern on the walls as near as possible to the floor.

- No. 6 welded mesh installed in backs and walls as near as possible to the floor.

- 5 cm of shotcrete applied in-cycle.

16.2 Dilution modelling and factors

The mineral resource models developed by QG were diluted into expanded vein wireframes according to the criteria in Table 16.2. Veins were first expanded by the dilution width and then further dilution was applied where the minimum mining width was not met, i.e. the dilution width is not applied over the minimum mining width. An example of the dilution is shown in the section in Figure 16.1.

Expanded wireframes were only created for underground targets where the undiluted grade was greater than 100 ppm Ag. Open pit targets were defined using the results of a preliminary pit optimisation using a fixed dilution rate, and diluted wireframes were only prepared for those areas within the economic pit shell.

Within the context of planned and unplanned dilution terminology used in mining engineering, the skin dilution (0.5 m either side of the vein) as applied to the vein mineralisation interpretation wireframes, is considered unplanned as it is not planned to extract this material during mining, however, it is prudent to assume that it will be taken. Planned dilution is the dilution that is incurred as a consequence of the minimum mining width constraint that was applied to the vein mineralisation interpretation wireframes.

Table 16.2 Mining dilution and minimum widths

Method	**Dilution width per side (m)**	**Minimum mining width (m)**
Open pit	0.50	2.0
Underground - room and pillar	0.50	3.0
Underground - shrinkage	0.25	1.5

Figure 16.1 Example section showing undiluted and diluted vein

The dilutant grades used for each vein were provided by QG and were determined using the average grade of samples falling within a close range of the vein (subject to some capping and rounding). The dilutant grades are included in

Table 16.3. Note that

the vein numbers and names in Table 16.3 have been used throughout the mining study.

The undiluted mineral resources and diluted inventories, each above a cut-off grade of 100 ppm Ag, are shown in Table 16.4.

Table 16.3 Dilutant grades

		Dilution grades applied to vein estimates	
Vein Number	**Name**	**Ag (ppm)**	**Au (ppm)**
7	Carmen	5	0.016
8	Gloria	10	0.030
9	Gloria Rama	9	0.016
10	Nieto	13	0.102
11	Pica	7	0.018
12	Martha Alta	5	0.018
13	Martha Baja	4	0.018
14	Martha Media	7	0.030
15	Martha Media Alta	10	0.040
16	Martha Ramas	4	0.016
17	Transversal Norte	9	0.020
18	Transversal Sur	11	0.027
19	Abundancia	11	0.034
20	Alacran	12	0.038
21	Esperancita	7	0.024
22	Luz Elena	11	0.042
23	Nueva	7	0.013
24	Olin	17	0.055
25	Sur	8	0.021

Table 16.4 Undiluted and diluted inventory above a 100 ppm Ag cut-off grade

			Undiluted			Diluted	
Vein		**Mass (kt)**	**Ag (ppm)**	**Au (ppm)**	**Mass (kt)**	**Ag (ppm)**	**Au (ppm)**
7	Carmen	142	178	0.32	178	129	0.23
8	Gloria	1,941	202	0.27	2,053	182	0.24
9	Gloria Rama	409	127	0.15	356	112	0.13
10	Nieto	18	102	0.07	0	107	0.09
11	Pica	1,092	160	0.16	842	156	0.15
12	Martha Alta	15,080	162	0.35	13,221	155	0.33
13	Martha Baja	1,195	135	0.23	351	121	0.27
14	Martha Media	2,121	167	0.40	2,032	161	0.35
15	Martha Media Alta	450	173	0.24	528	146	0.21

16	Martha Ramas	583	136	0.23	402	122	0.19
17	Transversal Norte	80	170	0.29	43	195	0.25
18	Transversal Sur	482	127	0.14	209	125	0.11
19	Abundancia	4,574	165	0.27	4,630	148	0.24
20	Alacran	61	119	0.19	18	115	0.17
21	Esperancita	405	147	0.32	283	119	0.27

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		Undiluted			Diluted		
Vein		**Mass (kt)**	**Ag (ppm)**	**Au (ppm)**	**Mass (kt)**	**Ag (ppm)**	**Au (ppm)**
22	Luz Elena	935	164	0.23	814	140	0.19
23	Nueva	144	135	0.05	8	119	0.07
25	Sur	333	148	0.22	177	127	0.16
	Total	30,045	163	0.30	26,144	153	0.29

16.3 Open pit mining

16.3.1 Pit optimisation

To identify the economic extents for open pit mining a pit optimisation was performed using Whittle software, which uses the industry standard Lerchs-Grossman algorithm. The parameters used for the pit optimisation for 4,000 tonne per day (tpd) feed are shown in Table 16.5 and Table 16.6. Some of these parameters, including the production rate, silver recovery, and metal prices, were subsequently changed in the financial model. This includes an increase in the production rate to 5,000 tpd, a reduction of the silver recovery to 86%, an increased silver price of $25/oz, and an increased gold price of $1,250/oz.

The mining costs for 4,000 tpd feed presented in Table 16.6 are based on a reference cost at the elevation of the predicted exit to the pit ramp, and increase with depth below this elevation. Since the topography is different for the main west and smaller east pit identified in previous studies, the crest or reference elevation is varied either side of 555700 mE.

Table 16.5 Pit optimisation parameters

Parameter	**Units**	**Value**
Mining recovery	%	95
Mining dilution		see Section 16.1
Overall pit slopes	degrees	45
Mining cost		see Table 16.6
Process recovery		
Silver	%	88
Gold	%	78
Process cost	$/t	17.25
General and administration cost	$/t feed	4.35
Payable metal		
Silver	%	99.75
Gold	%	99.50
Metal prices		
Silver	$/oz	18

Gold	$/oz	1,100

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Table 16.6 Mining costs used for pit optimisation

		Pit 1 (West)		Pit 2 (East)	
Bench	**Units**	**Waste**	**Feed**	**Waste**	**Feed**
Reference elevation	m RL	2070	2070	2020	2020
Reference cost	$/t	1.10	1.45	1.10	1.45
Cost increase per 10m bench	$/t	0.052	0.052	0.052	0.052

The resulting pit shells are detailed in Table 16.7 and displayed in Figure 16.2 and Figure 16.3.

Table 16.7 Pit optimisation results

Shell	Revenue factor	Mined (Mt)	Strip ratio	Waste (Mt)	Feed (Mt)	Ag (ppm)	Au (ppm)
10	0.48	38.6	10.0	35.1	3.5	159	0.25
11	0.50	42.0	10.0	38.2	3.8	157	0.25
12	0.52	48.3	10.5	44.1	4.2	157	0.25
13	0.54	51.4	10.7	47.0	4.4	156	0.24
14	0.56	58.6	11.2	53.8	4.8	155	0.24
15	0.58	63.1	11.5	58.0	5.1	155	0.24
16	0.60	68.2	11.6	62.8	5.4	153	0.23
17	0.62	70.6	11.6	65.0	5.6	152	0.23
18	0.64	73.7	11.8	67.9	5.8	151	0.23
19	0.66	79.0	12.1	73.0	6.1	150	0.23
20	0.68	89.3	12.6	82.7	6.6	148	0.22
21	0.70	92.2	12.6	85.4	6.8	147	0.22
22	0.72	96.3	12.8	89.3	7.0	146	0.22
23	0.74	99.7	13.0	92.6	7.1	146	0.22
24	0.76	102.7	13.1	95.3	7.3	145	0.22
25	0.78	245.6	19.5	233.6	12.0	151	0.26
26	0.80	254.1	19.5	241.7	12.4	150	0.26
27	0.82	283.4	19.6	269.6	13.7	147	0.25
28	0.84	321.5	20.2	306.3	15.2	148	0.26
29	0.86	332.7	20.3	317.1	15.6	147	0.26
30	0.88	339.1	20.3	323.2	15.9	146	0.26
31	0.90	346.9	20.3	330.6	16.3	146	0.25
32	0.92	365.5	20.6	348.6	16.9	145	0.25
33	0.94	390.1	20.7	372.1	18.0	143	0.25
34	0.96	397.2	20.7	378.9	18.3	142	0.25

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Shell	Revenue factor	Mined (Mt)	Strip ratio	Waste (Mt)	Feed (Mt)	Ag (ppm)	Au (ppm)
35	0.98	425.6	20.6	405.9	19.7	139	0.24
36	1.00	445.9	20.7	425.3	20.6	137	0.24
37	1.02	451.7	20.7	430.9	20.8	137	0.24
38	1.04	465.3	20.8	444.0	21.3	136	0.24
39	1.06	476.2	20.8	454.4	21.8	135	0.24
40	1.08	481.5	20.8	459.4	22.1	135	0.24
41	1.10	506.6	21.3	483.8	22.7	134	0.24
42	1.12	513.5	21.2	490.4	23.1	133	0.24
43	1.14	527.2	21.2	503.5	23.7	132	0.24
44	1.16	537.6	21.4	513.6	24.0	132	0.24
45	1.18	544.0	21.4	519.7	24.3	131	0.24
46	1.20	700.6	23.4	672.0	28.7	131	0.25
47	1.22	706.4	23.4	677.5	28.9	131	0.25
48	1.24	771.6	24.0	740.8	30.9	129	0.25
49	1.26	777.9	24.0	746.8	31.2	129	0.25
50	1.28	783.4	23.9	751.9	31.4	128	0.25

Figure 16.2 Pit shell tonnes and grade

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Figure 16.3 Plan view of pit shells

16.3.2 Pit design

The parameters used for the open pit design for 4,000 tonnes per day feed are included in Table 16.8.

Table 16.8 Pit design parameters

Parameter	Units	Value

Face height	m	20
Face angle	degrees	70
Berm width	m	9
Two lane ramp (pit 1)	m	25
Two lane ramp (pits 2/3)	m	21
One lane ramp	m	16

The designs were based on the generated pit shells to maximise the open pit value whilst allowing appropriate access to the higher value underground mineral resources. The dimensions and contained material in the pits are included in Table 16.9. The designed pits are shown in Figure 16.4 and a cross section through Pit 1 is included in Figure 16.4.

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Table 16.9 Summary of pit physicals

Parameter	Units	Pit 1	Pit 2	Pit 3	Total
Waste	Mt	104.3	33.7	17.8	155.8
Potential feed	Mt	7.4	1.7	1.3	10.4
	ppm Ag	135	124	70	125
	ppm Au	0.20	0.21	0.15	0.19
Approximate max. depth	m	190	110	120	

An interim pushback was defined for Pit 1 using a smaller pit shell. Due to their small size, no intermediate pushbacks have been used in Pits 2 or 3.

Figure 16.4 Plan view of pit designs

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Figure 16.5 Cross section through Pit 1 at 2702250 mN

16.4 Underground mining

16.4.1 Underground mining methods

The veins at La Preciosa vary widely in both width and dip to an extent that the choice of a single underground mining technique will not be appropriate for all veins. Vein widths vary from less than 1 m to greater than 15 m whilst the dip can vary from near flat to vertical, with the majority of the mineral resource contained in veins dipping less than 45°. The underground mining methods available for hard rock deposits dipping between 20° and 45° typically have lower mining recovery than more steeply dipping deposits because of the need to leave pillars to support the roof. They also require more ramps to provide the necessary access for mechanised equipment than flat deposits.

According to Golder (2010), the mining methods proposed for each stoping block incorporate the following considerations:

- The back span should be limited to 5 m. This recommendation is based on Golder s experience with rock of similar quality.

- Where the veins are steeply dipping (greater than 30°) and have a horizontal width less than or equal to 5 m, conventional cut and fill mining is proposed.

- Where the veins have a vertical thickness of less than 5 m, conventional room and pillar is proposed.

- Where the veins have a vertical thickness of between 5 m and 12 m, room and pillar mining with fill is proposed. In this case the fill is required to facilitate mining between 5 m and 12 m height. The fill will however only surround the lower part of the pillar (up to 7 m) and will therefore have minimal impact on the behaviour and stability of the pillar. Pillars are designed to carry full tributary area loading as in conventional room and pillar mining.

- Where veins have a vertical thickness greater than 12 m, post pillar cut and fill is proposed. In this method, mining takes place in 5 m vertical cuts that are progressively backfilled. The pillars become progressively more slender as mining progresses upwards. The pillars tend to yield within the backfill (the upper exposed portion of the pillar remains stable) and shed the load to the abutments.

Based on Golder s recommendation, Snowden applied the criteria shown in Table 16.10 to classify veins by mining method. Snowden elected to assume a variant of cut and fill known as shrinkage stoping, a common mining method in the region, for the narrow and steep veins. Shrinkage stoping is a method that Pan American uses at its other operations and its risks and costs are well understood. Note than in Table 16.10, room and pillar refers to all forms of room and pillar mining described by Golder.

Table 16.10 Underground mining methods

Method	Dip	Minimum mining width (m)	Cut-off grade (ppm Ag)
Room and pillar	< 35º	3.0	120
Cut and fill (shrinkage)	> 40º	1.5	150

In the room and pillar areas, Snowden further subdivided the deposit by vein dip. For vein dips between 9° and 22°, it was assumed that an apparent dip (or stepped) approach be used for room and pillar mining where the drives are placed at an angle to the vein strike to achieve apparent drive gradients of less than 15% so that trackless equipment can be used. For vein dips greater than 22°, it was assumed a room and pillar with backfill method (which includes post pillar cut and fill where the vertical thickness exceeds 12 m) is used. The distribution of underground mining feed for each dip category is presented in Table 16.11.

Table 16.11 Underground feed mass and grade by panel dip

Dip of panel	Mining method	Mass (kt)	Ag (ppm)	Au (ppm)
<9 º	Room and pillar	2,413	152	0.32
9 º to 22 º	Apparent dip (stepped) room and pillar	4,429	152	0.35
22 º to 33 º	Backfill room and pillar	4,422	140	0.29
>40 º	Cut and fill (shrinkage)	582	154	0.24
Total		11,845	148	0.32

Note: Includes mining/pillar losses and unplanned dilution.

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16.4.2 Identification of production areas

Areas of the veins suited to economic extraction were selected if there was sufficient quantity of contiguous material above the underground panel selection cut-off grade to justify the development of the vein or zone. The underground cut-off grades used for panel selection are shown in Table 16.10.

The panel selection cut-off grade was used to assist in selecting underground mining panels. These panels typically included cells of material below the panel selection cut-off grade. Actual final scheduling and cash flows were generated using the commodity prices stated in the cash flow statements. No specific cut-off grades were used in scheduling as each cell was evaluated on its own merits (such as recoverable metal and cost to develop and to extract).

Smaller zones located well away from other stoping areas were subsequently removed from the design. The remaining production areas were divided where necessary into smaller blocks to improve flexibility in scheduling. Following this, 29 production zones remained, containing the underground inventory (before mining and pillar losses and unplanned dilution) detailed in Table 16.12. A plan view of the panels is shown in Figure 16.6.

Table 16.12 Underground panel mass and grade

Panel	Mass (kt)	Ag (ppm)	Au (ppm)
1	643	146	0.35
2	15	112	0.27
3	660	121	0.34
4	33	110	0.39
5	62	112	0.27
6	510	153	0.27
7	800	146	0.29
8	1,581	151	0.35
9	933	130	0.45
10	64	114	0.19
11	2,332	189	0.49
12	21	166	0.10
13	15	120	0.14
14	1,918	152	0.23
15	300	122	0.30
16	301	127	0.26
17	57	117	0.21
18	924	138	0.32
19	1,760	174	0.33
20	542	152	0.40

21	141	149	0.26
22	173	153	0.31
23	31	140	0.28
24	845	156	0.26
25	157	166	0.41

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Panel	Mass (kt)	Ag (ppm)	Au (ppm)
26	124	151	0.21
27	96	200	0.13
28	511	147	0.20
29	225	160	0.23
Total	**15,776**	**155**	**0.33**

Note: Before mining/pillar losses and dilution have been applied.

Figure 16.6 Underground panels

16.4.3 Development and accesses

The conceptual design for underground development was prepared using a maximum decline grade of 15%. The design is preliminary in nature but appropriate for this preliminary economic assessment and serves principally to evaluate probable development requirements and as a consequence determine likely development costs.

The portal has been located away from the open pits to maintain the flexibility of scheduling as underground development and production will be required very early in the Project life.

Ventilation raises for fresh air intakes and used air exhausting have been included in the design, however their placement is only conceptual and the appropriate dimensions and fan requirements have not been identified as part of this preliminary study. The final conceptual underground design showing both development and stoping areas viewed from the southwest and the southeast is shown in Figure 16.7.

Figure 16.7 Conceptual underground development with stoping panels

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16.5 Mining schedule

16.5.1 Targets and methodology

The preliminary mine plan was initially developed at 4,000 tonnes per day and optimised based on an $18/oz silver price. Later, the silver price was increased to $25/oz which caused a decrease in the cut-off grade and a resultant increase in the amount of mineral resources used in the plan. That in turn led to the plan being re-scheduled with an aim of achieving a combined mill feed of 5,000 tpd from the open pit and underground mines. Scheduling was performed with Snowden s in-house Evaluator software package. Evaluator is a scheduling package based on a Mixed Integer Programming formulation. The software enables multiple sources (both open pit and underground production and underground development) to be modelled together and optimized simultaneously for schedule and cut-off grade, for a net present value (NPV) objective. The modelling of each location incorporates the input of economic and technical parameters. The economic parameters include price, operating cost, capital cost, and discount rate. The technical parameters include mining recovery and dilution as well as metallurgical recovery. The optimization honours sequencing and physical capacity limit constraints to ensure a feasible solution.

16.5.2 Limits

Mining capacity in each area was constrained using limits relating to tonnage and advance rates that were considered appropriate for this environment and equipment. The open pit mining capacity limits are included in Table 16.13. The bench limit assumes waste is mined in 10 m high benches and potential feed in 5 m high benches.

Table 16.13 Open pit mining capacity limits

Parameter	Units	Value
Mining rate	Mtpa	15
Bench limit	# per year	10

The underground schedule contains capacity limits on single heading advance rates, production from individual panels, and production from separate ventilation zones. The capacity limits for ventilation zones are shown in Table 16.14. Production capacity limits on individual panels vary between 250 and 1,000 tpd for room and pillar panels depending on the dip, width and extent of the panel. A limit of 400 tpd was applied to total production from the five shrinkage stoping panels. Single heading advance rates were capped at 90 m per month.

Table 16.14 Underground mining capacity limits

Zone	# Panels	Limit (ktpa)
North	12	400
East	3	300
Southeast	2	200
South	6	300
West	5	200

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Zone	# Panels	Limit (ktpa)
Zone 28	1	100

16.5.3 Schedule

The resulting mining schedule is presented in Table 16.15 and Figure 16.8 through Figure 16.11. The potential mill feed breakdown by mineral resource classification is shown in Table 16.16.

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Table 16.15 Mining schedule summary

		Units	Life of Mine	-1	1	2	3	4	5	6	7	8	9	10	11	12	13
Under-ground	Feed	kt	11,845		508	984	948	1064	1040	969	960	1072	964	846	1100	992	398
		Ag ppm	148		133	178	174	154	149	151	148	147	144	137	135	123	126
		Au ppm	0.32		0.38	0.39	0.35	0.33	0.33	0.34	0.29	0.29	0.31	0.27	0.31	0.28	0.26
	Waste	kt	1975	113	546	116	152	36	60	131	140	28	136	240		108	170
		Feed km	204.6		8.6	17.3	15.3	18.0	19.0	17.2	17.3	19.7	17.0	14.0	18.4	16.4	6.5
	Development	Capital km	13.0	1.9	6.4	0.6	0.8	0.3	0.2	0.6		0.2	1.0	0.7			0.4
		Waste km	21.5		3.3	1.3	2.3	0.9	1.2	1.4	2.2	0.2	1.1	3.1		1.8	2.7
	Raise boring	m	2089	550	550	483	73			205			229				
Open pit	To process	kt	8583	10	1172	841	877	284	785	423	853	405	586	979	605	713	49
		Ag ppm	131	154	128	162	197	92	153	129	110	106	116	165	70	75	81
		Au ppm	0.20	0.11	0.24	0.21	0.27	0.15	0.25	0.14	0.16	0.16	0.14	0.22	0.21	0.12	0.13
	To stockpile	kt	1796	220	67	282	143	154	471	178	32			250			
		Ag ppm	95	121	95	104	90	57	115	76	58			68			
		Au ppm	0.16	0.17	0.26	0.20	0.17	0.09	0.19	0.11	0.05			0.13			
	Waste	Mt	155.8	3.5	13.0	13.2	13.3	13.4	13.1	14.0	14.1	14.2	14.3	13.4	13.9	2.3	0.0
Stockpile	Off	kt	1796		145			477		433	11	349	131		120	120	10
		Ag ppm	95		132			103		118	107	74	54		81	56	52
		Au ppm	0.16		0.21			0.20		0.20	0.18	0.11	0.08		0.17	0.09	0.09
	Balance	kt		220	142	424	567	244	715	459	480	131		250	130	10	
		Ag ppm		121	98	102	99	65	98	71	69	54		68	56	52	
		Au ppm		0.17	0.17	0.19	0.19	0.10	0.16	0.11	0.10	0.08		0.13	0.09	0.09	
Processing	All feed	kt	22,224	10	1825	1825	1825	1825	1825	1825	1825	1825	1682	1825	1825	1825	457
		Ag ppm	137	154	130	170	185	131	151	138	130	124	127	152	110	100	120
		Au ppm	0.26	0.11	0.28	0.31	0.31	0.27	0.30	0.26	0.23	0.23	0.23	0.25	0.27	0.20	0.24
	Indicated	kt	17,195	10	1450	1636	1683	1712	1689	1427	1213	1410	1477	1478	1040	755	217
		Ag ppm	145	154	131	174	187	130	153	144	141	129	126	155	126	110	133
		Au ppm	0.27	0.11	0.26	0.32	0.32	0.27	0.31	0.28	0.25	0.25	0.24	0.26	0.29	0.24	0.27

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	Units	Life of Mine	-1	1	2	3	4	5	6	7	8	9	10	11	12	13
Inferred	kt	5029		375	189	142	113	136	398	612	415	205	347	785	1070	240
	Ag ppm	111		125	136	163	140	129	115	109	109	133	138	88	93	108
	Au ppm	0.21		0.36	0.23	0.24	0.25	0.17	0.18	0.18	0.16	0.20	0.21	0.25	0.18	0.22
Metal	Ag Moz	84.2	0.0	6.6	8.6	9.3	6.6	7.6	7.0	6.6	6.3	5.9	7.7	5.5	5.0	1.5
output	Au koz	145.0	0	12.7	14.2	14.2	12.2	13.6	11.8	10.4	10.4	9.8	11.3	12.3	9.3	2.7

Figure 16.8 Mill feed by source

Figure 16.9 Open pit mining physicals

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Figure 16.10 Underground development

Figure 16.11 Underground mining tonnages

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Table 16.16 Potential mill feed breakdown by mineral resource classification

Mineral resource classification	Mass (kt)	Ag (ppm)	Au (ppm)	Mass (%)	Ag metal (%)	Au metal (%)
Indicated	17,195	145	0.27	77.4	81.7	81.7
Inferred	5,029	111	0.21	22.6	18.3	18.3

16.6 Waste rock dumps

Three conceptual waste rock dumps have been designed for the waste from the open pits as shown in Figure 16.12. The North waste rock dumps includes the backfilling of Pit 2 (15.4 Mm3) and only a portion of its capacity will be available prior to the completion of Pit 2 in Year 5. The storage volumes available in the waste rock dumps are shown in Table 16.17.

Figure 16.12 Open pits and waste rock dumps

Table 16.17 Waste rock dump storage volumes

Waste rock dump	Volume (Mm3)
North	56.7
East	15.0
West	10.4
Total	**82.1**

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16.7 Mining fleet

The major mobile fleet items are shown in Table 16.18 with the quantity of units contained in the peak fleet size.

Table 16.18 Fleet requirements for major mining equipment

Equipment		Max. units
Open pit	Production drill rig	2
	Pre-split drill	2
	Wheel loader	2
	Haul truck 100 t	8
	Hydraulic excavator	1
	Water truck	2
	Track dozer	2
	Grader	2
Underground	Articulated truck 20 t	13
	Scoop tram 6 yard3	8
	Scoop tram 3.5 yard3	3
	Jumbo twin boom	8
	Jumbo single boom	2
	Roof bolters	8
	Shotcrete sprayer	1
	Agitator trucks	2
	ANFO loader	2
	Rammer jammer	2

16.8 Workforce

The peak workforce for the mining operations is estimated at 390 employees, comprising:

- 185 underground operators.
- 82 surface operators.

- 81 maintenance personnel.
- 27 in technical services.
- 15 in mining management and administration.

16.9 Alternative mine concept

An alternative mine concept was investigated for a much larger open pit with no underground mining, the aim being to maximise recovery of the mineral resource and reduce capital expenses. Although this plan does maximize mineral resource recovery it also carries high risk with respect to large strip ratios in some years (the strip ratio is greater than 40 for years 5 to 7) and generates a lower discounted Project value due to the higher operating cost resulting from additional overburden stripping. As such it was agreed in consultation with Pan American and Orko that this option should be set aside for this preliminary economic assessment.

17 Recovery methods

The results of metallurgical testing have indicated that conventional cyanide leaching technology can be used to recover silver and gold from the La Preciosa feed material. The design basis for the feed processing facility is 5,000 dry metric tonnes per day (dtpd) or 1.8 million dry metric tonnes per year (dtpy). Feed will be transported from the mine to the concentrator facility by off-highway haulage trucks. The feed will be processed to produce silver and gold doré that will be loaded onto highway haul trucks and transported to a metal refinery.

The process operations are summarised as follows:

- Crushing of the feed by primary gyratory crusher to reduce the feed size from run of mine to minus 200 mm.
- Stockpiling primary crushed feed in a coarse feed stockpile and then reclaiming by feeders and conveyor belt.
- Grinding feed in a semi-autogenous grinding (SAG) mill / ball mill circuit prior to processing in a cyanide leach circuit. The SAG mill will operate in closed circuit with a vibrating screen and pebble crusher. Crushed pebbles will be returned to the SAG mill. The ball mill will operate in closed circuit with a hydro-cyclone to produce the desired grinding product size distribution of 80% (P80) passing 74 microns.
- Thickening the slurry to 45% solids by weight in a pre-leach thickener prior to cyanide leaching. Reclaim solution will be recovered from the pre-leach thickener for use in the process plant.
- Leaching of the slurry with cyanide solution in agitated leach tanks to dissolve the silver and gold contained in the slurry.
- Recovering soluble silver and gold using multi-stage counter current decantation technology (CCD). Barren solution will be added in the final CCD thickener as the washing solution.
- Clarifying of the pregnant solution followed by adding zinc dust to the solution to obtain a precipitate containing the recovered precious metals.

- Filtering and batch smelting of the precipitate to produce a silver and gold doré.

- Thickening of the leached tailing slurry and the recovery of the cyanide solution before detoxifying the thickened tailing slurry using oxygen and sulphur, with copper sulphate as a catalyst, prior to disposal in a tailings pond.

- Recycling water from the tailing pond for re-use in the process plant. Plant water stream types include barren solution, reclaim solution, fresh water, and potable water.

- Storing, preparing, and distributing reagents to be used in the process.

The overall process flow sheet is shown in Figure 17.1. Reagents to be used in the process and other main process consumable items are listed in Table 17.1 and the general design criteria used for equipment selection are identified in Table 17.2.

Table 17.1 Process consumables

Item	kg/tonne feed
Reagents	
Anti-scalant	0.05
Copper sulphate	0.015
Diatomaceous earth	0.05
Flocculant	0.25
Flux	0.28
Lead nitrate	0.05
Lime (CaO)	2.35
Sodium cyanide	1.6
Sodium meta-bisulphite	0.87
Zinc dust	0.13
Grinding media	
Grinding balls, SAG mill	0.72
Grinding balls, ball mill	1.10
Utilities	
Fresh water	35.4 litres per second
Power	12.2 megawatts

Table 17.2 Process design criteria

Item and unit	Value
General	
Feed assay	
Ag (ppm)	137
Au (ppm)	0.26
Feed Abrasion Index (AI)	0.7645
Feed Work Index (kWh/t)	
Ball mill (Bwi)	12.3
Ore moisture content (%)	
Design	3
Minimum	1
Maximum	5
Production Schedule	
Average (dmtpy)	1,825,000
Average (dmtpd)	5,000

Item and unit	Value
Metal production schedule	
Silver recovery (%)	86
Gold recovery (%)	78
Primary crushing	
Days per year	365
Hours per day	12
% availability	75
Ore crushing rate (dmtph)	555.6
Crusher feed (F80, mm)	500
Crusher product (P80, mm)	204
Grinding	
Days per year	365
Hours per day	24
% availability	92
Milling rate (dmtph)	226.4
Primary grinding	
Mode of operation	Closed circuit with vibrating screen
Feed size (microns)	204,000
Transfer size (microns)	3,000
Pebble circulating load (%)	15
Secondary grinding	
Mode of operation	Closed circuit with hydrocyclones
Product size (microns)	74
Recirculating load (%)	300
Pre-aeration/CCD/cyanide recovery	
Days per year	365
Hours per day	24
% availability	92
Thickener unit area (t/d-m2)	11 27
Cyanide leach	
Days per year	365
Hours per day	24
% availability	92
Tank feed rate (m3/h)	369.1
Slurry (% solids w/w, design)	45
Residence time (hr, total)	72

Item and unit	Value
Mode of operations	Series
Silver and gold precipitation	
Days per year	365
Hours per day	24
% availability	92
Feed rate (m3/h)	1,030
Metal recovery from solution (%)	
Gold	99
Silver	99

Figure 17.1 La Preciosa process flow sheet

18 Project infrastructure

18.1 Facility layout

The main processing facilities for the Project are located in a compact arrangement to limit the amount of required earthworks. These facilities are shown in Figure 18.1 and include primary crushing, grinding, leaching, CCD thickeners, Merrill Crowe processing, and refining facilities.

In addition to the main process facilities, there will be several surface buildings constructed to support the mining and process operations. These facilities include administration, guard house, truck shop, warehouse, change house, and explosives storage buildings and truck wash and mill maintenance facilities.

Figure 18.1 La Preciosa processing facility plan

18.2 Tailings facility

Thickened tailing material will be pumped approximately 2 km from the plant site tailing thickener to the planned lined tailings facility. A plan of the tailings facility layout is shown in Figure 18.2. Tailing material will be discharged into the dam from multiple spit points and supernatant water will be pumped and recycled back into the process facility.

The starter dam will primarily be constructed of placed and compacted mining overburden material and liners will be placed on top of the engineered fill. The majority of the ultimate tailings impoundment is planned to be constructed during pre-production mining activities. A single downstream dam raise will be constructed near the end of the Project life.

Figure 18.2 Tailings facility layout

18.3 Transportation

The Project is located 84 km by road northeast of Durango and can be accessed by vehicle in approximately 90 minutes. To reduce the travelling time between Durango and the Project to approximately 45 minutes, a new, 9.7 km long unpaved road will be constructed to access the site. The road has been planned to avoid passing through Francisco I Madero, Lázaro Cardenas, Francisco Serrano, and Javier Mina and to reduce interference with the local community. Once complete, site will be accessed from Durango travelling to the northeast on the paved Federal Highway 40 connecting Durango to Torreon, then by the new access road leading from a turnoff from Highway

40 located to the southwest of the town of Francisco I. Madero, and then heading to the northwest on the new access road to the Project site. Deliveries to and shipments from the site will be primarily via this route. A plan of the proposed access route is shown in Figure 18.3.

Figure 18.3 Plan of the proposed access route from Durango to the Project site

18.4 Power

The Comisión Federal de Electricidad (CFE) has proposed that 115 kV, 25 MVA power is available for the Project from the Canatlán II substation located northwest of the Project site in the city of Canatlán, Durango. The length of the proposed power line route is 41 km, generally following the shortest possible distance to the Project. A plan of the proposed power line route is shown in Figure 18.4. CFE provided an estimated cost of the power service and distribution system to supply power to the mine and plant facilities.

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The on-site power distribution will include lower voltage transmission to the main process facilities as well as ancillary facilities, the well field, mining equipment and the tailings facility. The anticipated cost for the power line and associated improvements is $10.1 million. The power rate calculated from CFE is $0.091/kWh.

Figure 18.4 Plan of the proposed power line route

18.5 Water

Limited groundwater has been encountered on the Property. Local farmers irrigate crops with groundwater sourced from thick gravel beds in the surrounding plains and these same gravels are a potential source of production water. Water for exploration activities is currently obtained from the dam in nearby Francisco Javier Mina.

During a site visit, David B. Hawkins, a consulting hydrogeologist with Barranca Group LLC, of Tucson, Arizona, USA, identified little potential for ground water supply in the Project area, but did identify three target areas for Project water supply in nearby surface gravel aquifers. For this study, a potential well field located 8.6 km east of the Property was investigated by Pan American and is assumed will adequately supply the estimated 30 to 50 litres per second that will be required for the Project. The well field location is shown in Figure 18.5.

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Figure 18.5 La Preciosa Project proposed well field location

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19 Market studies and contracts

19.1 Marketing

19.1.1 Product specification and selling costs

On-site processing will produce doré that is transported from the site to a refinery that produces fine silver and gold with a purity of 99.99%. The refined metal is then sold on the open market. The study assumptions for marketing of doré are presented in Table 19.1 and are derived from Snowden s database of similar studies in this market.

Table 19.1 Marketing parameters

Item	**Units**	**Value**
Doré purity	% Ag / Au	98
Transport cost	$/oz doré	0.12
Refining charge	$/oz doré	0.19
Silver payable	%	99.75
Gold payable	%	99.50
Minimum gold deduction	%	0.01
Gold refining charge	$/oz Au	0.50

19.1.2 Metal prices

The metal prices used in this technical report are based on a weighted average calculation performed by M3 of the historical prices (weight 60%) and future forecast prices (weight 40%) as of the end of June, 2011. Pan American rounded these weighted averages to the nearest US$1 per ounce in the case of silver and to the nearest US$50 per ounce in the case of gold.

For the calculation of the silver price used in this technical report, the historical price of US$19.44 is the average of the London Bullion Market daily silver prices for the 36 months prior to the end of June, 2011. The future price of $33.70 is the 24 month future price forecast on the Chicago Mercantile Exchange (previously the Comex futures division of NYMEX). The weighted average price of $25.14 was rounded down to $25.00.

For the calculation of the gold price used in this technical report, the historical price of US$1,110.70 is the average of the London Bullion Market daily PM gold prices for the 36 months prior to the end of June, 2011. The future price of $1,491.38 is the 24 month future price forecast on the Chicago Mercantile Exchange. The weighted average price of $1,262.98 was rounded down to $1,250.

19.1.3 Assumptions

Mr. Finch and Mr. Snider have reviewed the marketing parameters and metal prices cited in this section and believe that they are reasonable to support the assumptions in this preliminary economic assessment.

19.2 Contracts

At the time of this technical report no contracts for the transportation, refining of the doré products, or sale of the precious metals to be produced at the La Preciosa Project have been signed or are under negotiation.

20 Environmental studies, permitting, and social or community impact

20.1 Environmental summary

Environmental baseline studies were carried out in 2010 and 2011 for the Property site and adjacent land. An Environmental Impact Statement, (EIA), known in México as a Manifestacion Impacto Ambiental (MIA), forest land use modification (ETJ) document and Risk Analysis (ER) document will be prepared and submitted to the Secretaría de Medio Ambiente y Recursos Naturales (SEMARNAT) in 2011 for environmental permitting. The only other known environmental information is sourced from the mapping by the Instituto Nacional de Estadística, Geografía e Informática (INEGI) and of the Comisión Nacional para el Conocimiento y Uso de la Biodiversidad de México (CONABIO) at scales of 1:50,000 and 1:250,000.

The environmental baseline characterisation for the site and surrounding land (the local environmental system) has been completed. One year of data was collected during the dry and rainy season during 2010 and 2011. The information collected will be used for the preparation of the MIA and for other environmental documents to be submitted to the authorities for environmental permitting. The information will also be used for the formulation of design criteria and construction plans.

The environmental baseline is composed of data collected on a regional level as well as physical and biological data collected on a local scale. The environmental baseline was scaled to ensure suitable coverage by collecting regional and site specific data, including the definition of the local environmental system, based mostly on nano- and micro-basins directly influenced by the La Preciosa Project. The total area considered for environmental baseline is 5,880 ha. A total of 49 nano-basins have been considered for hydrological baseline and environmental system. Surface runoff is mostly to the south and east and all the surface runoff is seasonal.

The main soil types in the area are phaeozems and regosols. The soils are mainly shallow and only the flat alluvial areas have a potential for deeper soils. The texture type is mostly medium. The main soil conservation problem in the area is erosion; approximately 54% of the environmental system is considered moderately susceptible while 37% is considered as highly susceptible to erosive processes.

The water quality data as of April 2011 do not indicate fatal flaws for the Project. The Property is located in the vicinity of four different aquifers. Water availability is negative and the securing of water rights will be an important component of Project development.

Grasses and small shrubs along with several varieties of cacti make up most of the vegetation on the steep hillsides, with larger bushes and mesquite trees near springs and streams. One cactus species defined as protected by the Mexican government has been detected in the area.

A total of 90 animal species have been identified, detected and/or estimated in and around the La Preciosa area. From these, nine species including four reptiles, four birds, and one mammal have been identified as under a protection category by NOM-059-SEMARNAT-2010.

The major environmental issue in the area is the current cattle, goat and agricultural regime that tends to exceed the carrying capacity of the area.

20.2 Expected material environmental issues

The main environmental liabilities expected from the development of the Project will be associated with the tailings impoundment facility, open pit mining, cyanide storage and handling, and water management.

The likely environmental issues of concern with development of the La Preciosa Project include:

- Long-term water management, particularly due to the arid nature of the area; over exploited aquifers, securing water rights and limited water potential.
- The length of time associated with obtaining regulatory approvals and permits.
- Long-term management of metal leaching/acid rock drainage (MLARD). Limited data suggest neutral to minor concerns, though further testing is required.
- Costs associated with Project closure and reclamation.
- Construction and operation of the access road, particularly with respect to social issues and land acquisition.
- Management of the cyanide process plant.
- Management and liner considerations for the tailings impoundment facility.

20.3 Waste and tailings disposal

A stockpile of approximately 10,000 tonnes of material produced from the historical workings is located towards the north end of La Preciosa Ridge and there are small historical workings over some of the veins. These do not present a significant environmental liability and will be processed as plant feed.

The design and maintenance of any tailings facilities will include control and monitoring of surface and ground water and airborne dust. The exact prevention and mitigation initiatives will depend on many variables, including the location of the facilities, ambient conditions, background water quality and quantity, mineral processing methods, etc. With the correct prevention and mitigation controls in place it is expected that none or minimal offsite impact will be observed. Waste rock areas primarily contribute to surface and groundwater contamination. The exact prevention and mitigation measures will depend on many variables, however, it is not anticipated that the waste rock will generate any appreciable offsite impact.

20.4 Site monitoring

Anticipated monitoring activities in the future may include:

- Installation of a weather station for meteorological data collection.

- Installation of monitoring wells and piezometers for open pit areas, upstream and downstream from the tailings dam and process plant to monitor ground water characteristics.

- Design and implementation of an environmental monitoring programme.

- Dust monitoring.

- Complete static and kinetic testing for MLARD which will be coupled with an ongoing monitoring programme for waste rock and tailings.

20.5 Water management

The acquisition of water rights is anticipated to be challenging. Therefore, water conservation and recycling has been identified as key design criteria for the Project.

20.6 Permitting

Development of the La Preciosa Project will require a number of permits that comply with the laws and regulations established for México on the Federal level and the Durango State level. The main documents to be prepared and submitted are:

- An Environmental Impact Statement (MIA) that describes the potential impacts to the environment that may occur as a result of the preparation, construction and operation of the La Preciosa Project, as well as the proposed measures to prevent, control, mitigate or compensate for environmental impacts.

- A Forest land use modification (ETJ) document in order to technically justify the change of forest land use to mining and industrial land use.

- A Risk Analysis (ER) document, which is a detailed risk analysis based on feasibility level designs, substance quantities and potentially affected areas. A strong emphasis is placed on the storage and handling of hazardous materials such as cyanide, fuel, and tailings. Risk control and mitigation measures are included.

- An Accident Prevention Programme (PPA), which is a compendium of general and specific protocols tailored to the La Preciosa Project, aimed at prevention and response. The PPA can be developed during the initial phases of Project development and is mostly based on the ER report.

- An archaeological survey and feasibility study using archaeologists from the Instituto Nacional de Antropología e Historia (INAH), which is the governmental archaeological authority, in order to acquire a Permiso de Liberación de Tierras (land liberation permit). This must be completed prior to Project development.

20.6.1 Applicable federal laws and regulations

In addition to the collection of baseline environmental and social information and the preparation of the MIA, ETJ and ER, compliance with various laws and regulations will be required in order to obtain the necessary construction and operations permits for the La Preciosa Project. Federal laws and regulations that are likely applicable include:

- General Law of Ecological Balance and Environmental Protection (Ley General de Equilibrio Ecológico y Protección del Ambiente).
- Environmental Impact Regulation (Reglamento en Materia de Evaluación de Impacto Ambiental).
- Prevention and Control of Atmospheric Contamination Regulation (Reglamento en Materia de Prevención y Control de la Contaminación de la Atmósfera).
- General Law for Sustainable Forest Development (Ley General de Desarrollo Forestal Sustentable).
- General Law of Wildlife (Ley General de Vida Silvestre).
- National Waters Law (Ley de Aguas Nacionales).
- General Law for the Prevention and Management of Waste (Ley General para la Prevención y Gestión de los Residuos).

- Permit for the purchase, storage, and use of explosives (Secretary of National Defense).

20.6.2 Applicable state laws and regulations

The Project will be required to comply with the Law of Ecological Balance and Environmental Protection for the State of Durango (Ley del Equilibrio Ecológico y la Protección del Ambiente para el Estado de Durango).

20.6.3 Generally applicable Mexican standards

Several Mexican Official standards have been established to protect the environment and will likely be applicable to the development of the La Preciosa Project. Specific areas under legislation include:

- Control of atmospheric contamination through the identification of permissible levels of various emissions that may be produced by industrial operations, administered by SEMARNAT and the Secretaría de Salud (SSA), the Secretariat of Health.

- Control of noise emissions, administered by SEMARNAT.

- Environmental impacts, administered by SEMARNAT.

- Protection of flora and fauna, administered by SEMARNAT.

- Water conservation and responsible use, administered by SEMARNAT and the Comisión Nacional del Agua (CNA), the National Water Commission.

- Handling of hazardous and urban waste, administered by SEMARNAT.

- Soil Protection, administered by SEMARNAT.

20.7 Social and community requirements

The closest communities to the Property include the small community of Vicente Suarez to the south, Ricardo Flores Magón to the west, Javier Mina to the north-northeast, Flor de Mayo (a small one family ranch in close proximity to the Property) to the east, and Lázaro Cardenas to the east. The closest urban center in the region is Francisco I. Madero located 14 km to the east of the Property. The valleys to the east and north are also fertile agricultural areas.

Likely social concerns surrounding the development of the La Preciosa Project include:

- Acquisition of surface land.
- Water use (securing rights, lobbying, management, and recycling).
- Perceptions of the cyanide facility and use of cyanide.
- Operation of the access road, particularly with respect to potential use by non-Project personnel and land acquisition negotiations.
- Potential imposition of access restrictions to the area.
- Expectations that the Project development may generate in surrounding communities with respect to employment and quality of life.

An archaeological survey and feasibility study will be conducted, using INAH archaeologists in order to acquire a land liberation permit. This needs to be completed prior to Project development.

20.8 Project closure

The main components that will require closure and reclamation actions include:

- Process plant (cyanide).
- Open pit.
- Underground stabilisation and sealing.
- Tailings impoundment.
- Waste rock piles.
- Project roads that will not be integrated into the local road network or used for post-closure maintenance and monitoring.
- Buildings and foundations.
- Other areas considered for environmental enhancement and ecological integration.
- Implementation of a financial instrument used to ensure Project closure expenses.
- Periodic review and updating of the Project Closure and Reclamation plan, based on progress and new alternatives.

Suitable areas will be re-vegetated as part of the reclamation strategy, both actively during operation, as a progressive reclamation strategy, and upon closure. The largest surface area to be restored by means of re-vegetation is the buffer zone yet to be defined around the Property, as an effort to actually improve the ecosystem quality of the area in a manner that is reasonable and feasible from a cost/benefit perspective.

An allowance of $8.9 million for closure costs have been made in this preliminary economic assessment with the expenditure allocated to the final year of the Project life.

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21 Capital and operating costs

21.1 Capital costs

Project estimated capital costs are summarised in Table 21.1. These costs are in 2011 US$ with an assumed escalation during the construction period of $6.4 million or 2.37%.

Table 21.1 La Preciosa Project preliminary initial capital cost estimates

Type	**$M**
Open pit mining equipment	25.9
Underground mining equipment	15.8
Open pit pre-strip	4.4
Underground capital development	3.7
Plant (including tailings and infrastructure)	143.4
EPCM(1)	22.1
Owner s costs	47.9
Escalation of plant and EPCM costs	6.4
Total estimated capital(2)	**269.7**

Note(1): EPCM is engineering, procurement, and construction management.

Note(2): Excludes sustaining capital.

21.1.1 Mining

Estimated mining capital costs including the equipment fleets detailed in Table 16.18 are shown in Table 21.2. The fleet costs are inclusive of 15% contingency.

Table 21.2 La Preciosa Project preliminary mining capital cost estimates

Item	Build ($M)	Sustaining ($M)
Open pit fleet	25.9	17.7
Pre-strip	4.4	
Underground fleet	15.8	53.4
Underground development	3.7	19.2
Total(1)	49.8	90.2

Note(1): Costs have been rounded and this has resulted in minor discrepancies in the totals.

21.1.2 Processing and infrastructure

The capital costs for processing and infrastructure were developed based on prefeasibility study level engineering effort. Estimates (material take offs) for civil, concrete, structural steel, and mechanical were made to quantify the amount of materials required. Unit rates for materials and installation were based on similar projects in Mexico. Equipment costs were based on recent vendor quotations for the specific equipment planned for this plant. If vendor quotes were not received, historic data for similar equipment was used. Costs for electrical and instrumentation were factored based on historical data. The estimated processing and infrastructure capital is shown in Table 21.3. The capital cost estimate for the plant includes a 20% contingency.

Table 21.3 La Preciosa preliminary processing and infrastructure capital cost estimates

Item	Description	Value ($M)
General site	Access road and mass civil works for the process plant	3.95
Primary crushing	Gyratory crushing facility and stockpile feed conveying	7.17
Reclaim stockpile	Reclaim tunnel and feeders and SAG feed conveyor	2.98
Grinding	SAG/ball mill facility with hydro-cyclone classification	19.86
Leaching & CCD	Outdoor leach tanks and CCD thickening facilities	18.54
Merrill Crowe	Merrill Crowe installed equipment	7.86
Refinery	Refinery building and equipment	1.42
Tailings	Detoxification, tailing thickeners, and tailings dam	10.65
Water systems	Water well, water tanks, fresh and fire water systems, and process water systems	4.49
Main substation	CFE transmission line and main power substation	10.10
Reagents plant	Reagent tanks and distribution system and dry storage facilities	3.96
Ancillaries	Truck wash, fuelling station, truck shop, laboratory, administration building, guardhouse, warehouse, change house, and mill maintenance	11.08
Freight		14.03
IMMEX(1)		2.52
Sub-total plant direct costs		118.62
Indirect field costs		0.59
Bussing/site transportation		0.41
20% contingency on plant		23.72
Total(2)		143.35

Note(1): IMMEX is the Industria Manufacturera, Maquiladora y de Servicios de Exportación programme in Mexico that reduces import taxes.

Note(2): costs have been rounded and this has resulted in minor discrepancies in the totals.

21.1.3 Other initial capital

Other initial capital includes the owner s costs, EPCM costs, as well as escalation. These are detailed in Table 21.4.

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Table 21.4 Other initial capital cost estimates

Item	Cost ($M)
Owners Project team in Vancouver	1.85
Owner s Project team in Mexico	6.22
Non-plant engineering and construction management	3.84
MIA activities, studies, permits	2.31
Community upgrades, local donations	0.60
Road construction and maintenance	0.45
First fills	1.50
Warehouse supplies	2.24
Capital spares	2.76
Working capital for operating costs	2.88
Construction insurance	1.04
Other insurances	0.35
Temporary and build power	1.00
Receiving and inspections	0.35
Transportation costs	0.12
Land purchase / transfer	8.23
Training, operating manuals, maintenance manuals	0.29
Miscellaneous and administrative	0.29
Communications and security	1.15
Customs and duties	0.58
Packaging for freight	0.23
Diamond drilling for mineral resource definition	4.95
Salary manpower from operations list	3.00
Hourly manpower	1.73
EPCM	22.1
Escalation of plant and EPCM	6.14
Total(1)	**76.4**

Note(1): Costs have been rounded and this has resulted in minor discrepancies in the totals.

21.1.4 Sustaining capital

Sustaining capital includes equipment replacement and owner s costs. Equipment replacement schedules are based on expected equipment life cycles. Owner s costs include items such as non-plant engineering, MIA activities (including reclamation), communications and security, insurances, water rights purchases, land purchase/leases, etc. The sustaining capital schedule is presented in Table 21.5.

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Table 21.5 Sustaining capital schedule

Year	Cost ($M)
Year -3	
Year -2	
Year -1	
Year 1	34.2
Year 2	7.6
Year 3	5.4
Year 4	4.6
Year 5	5.7
Year 6	6.7
Year 7	7.1
Year 8	8.1
Year 9	7.1
Year 10	7.7
Year 11	6.4
Year 12	6.3
Year 13(1)	
Year 14(1)	
Total(2)	**106.8**

Note(1): Includes the allowance for reclamation. Note(2): Costs have been rounded and this has resulted in minor discrepancies in the totals.

21.2 Operating costs

Operating cash costs have been developed from first principles for each area. The average life-of-mine cash costs were estimated at $11.84 per ounce of silver, net of gold by-product credits. The estimated costs are detailed in Table 21.6.

Table 21.6 Summary of La Preciosa Project preliminary operating cost estimates

Cost type	$/Tonne	$/Tonne milled
Open pit reference mining ($ per tonne of waste)(1)	1.11	
Open pit reference mining (($ per tonne of feed)	1.45	
Underground mining ($ per tonne of feed)	31.49	
Total mining		26.60
Processing ($ per tonne of feed)		16.64
General and administration ($ per tonne of feed)		2.57

Average total cost(2) ($ per tonne of feed)	**45.81**

Note(1): Reference costs are at pit crest. Mining cost increases with depth.

Note(2): Excludes taxes and royalties.

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21.2.1 Mining

The mining operating cash costs have been estimated from first principles and benchmarked against other Mexican operations, including other mines owned by Pan American. This section includes a summary of the operating costs.

Open pit

The components of the estimated open pit operating costs are included in Table 21.7. The haulage cost for waste and plant feed increases with the depth of the pits at a rate of $0.052/t per 10 m bench.

Table 21.7 La Preciosa Project preliminary open pit operating cost estimates

Activity	**Waste ($/t)**	**Feed ($/t)**
Drilling	0.15	0.34
Blasting	0.25	0.31
Loading	0.24	0.27
Haulage (initial)	0.22	0.22
Project support	0.16	0.22
Project G&A	0.09	0.09
Total	1.11	1.45

Underground

The components of the underground operating costs are given in Table 21.8.

Table 21.8 La Preciosa Project preliminary underground operating cost estimates

Activity	**Cost ($/t feed)**
Stoping	12.47
Waste backfilling	2.81
Haulage	3.35

Power	3.47
Operational development ($1,650/m)	2.56
Maintenance	1.78
Services installation	0.52
Project administration and technical services	3.69
Diamond drilling	0.85
Total	31.49

21.2.2 Processing and general and administrative costs

Processing and G&A cash costs are detailed in Table 21.9.

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Table 21.9 La Preciosa Project preliminary processing and G&A cost estimates

Item	**Annual cost ($)**	**Cost ($/t feed)**
Operating and maintenance labour	1,700,000	0.93
Power	7,564,000	4.14
Liners and grinding media	4,190,000	2.30
Reagents	12,460,000	6.83
Maintenance parts and services	2,711,000	1.49
Water charges	265,000	0.14
Supplies & services	1,488,000	0.82
Total processing costs	**30,378,000**	**16.65**
Labour and fringe benefits	1,089,000	0.597
Accounting (excluding labour)	50,000	0.027
Safety and environmental (excluding labour)	50,000	0.027
Human resources (excluding labour)	50,000	0.027
Security (excluding labour)	450,000	0.247
Office operating supplies and postage	100,000	0.055
Maintenance supplies	25,000	0.014
Maintenance labour, fringes benefits, and allocations	16,600	0.009
Laboratory costs	392,000	0.215
Power allocation 15% process ancillary facilities	99,300	0.054
Water charges	5,400	0.003
Propane/fuel	75,000	0.041
Communications	100,000	0.055
Small vehicles	75,000	0.041
Legal and audit	350,000	0.192
Consultants	200,000	0.110
Community relations	500,000	0.274
Janitorial services	50,000	0.027
Insurances	400,000	0.219
Meals	365,000	0.200
Subs, dues, public relations, and donations	50,000	0.027
Travel, lodging, and meals	100,000	0.055
Recruiting/relocation	100,000	0.055
Total G&A costs	**4,692,300**	**2.57**

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22 Economic analysis

22.1 General

This preliminary economic assessment is preliminary in nature. It includes Inferred mineral resources that are considered too speculative geologically to have the economic conditions applied to them that would enable them to be categorised as mineral reserves, and there is no certainty that the preliminary economic assessment will be realised.

Certain of the statements and information in this Technical Report constitute forward-looking statements within the meaning of the United States Private Securities Litigation Reform Act of 1995 and forward-looking information within the meaning of applicable Canadian Provincial securities laws. All statements, other than statements of historical fact, are forward-looking statements. When used in this Technical Report the words estimates , expects , projects , plans , contemplates , calculates , objective , potential , and other similar words and expressions, id forward-looking statements or information. These forward-looking statements or information relate to, among other things: the future successful development of the Project; the estimates of expected or anticipated economic returns, as reflected in the preliminary economic assessment; the timing for completion of a feasibility study and environmental impact assessment on the Project; future production of silver and gold and mine-life of the Project; future cash costs per ounce of silver; the price of silver and gold; the sufficiency of Pan American s current working capital, anticipated operating cash flow or its ability to raise necessary funds; the capital necessary to construct a mine at the Project and the time-line for such construction; the accuracy of mineral resource estimates; estimated production rates for silver and other payable metals produced at the Project; timing of production and the cash and total costs of production; the estimate of metallurgical recoveries for silver and gold; the estimate for mining dilution; the estimated cost of and availability of funding necessary for sustaining capital; and ongoing or future development plans and capital replacement, improvement or remediation programmes.

These statements reflect current views with respect to future events and are necessarily based upon a number of assumptions and estimates that, while considered reasonable, are inherently subject to significant business, economic, competitive, political and social uncertainties and contingencies. Many factors, both known and unknown, could cause actual results, performance or achievements to be materially different from the results, performance or achievements that are or may be expressed or implied by such forward-looking statements contained in this Technical Report and assumptions and estimates have been made based on or related to many of these factors. Such factors include, without limitation: fluctuations in spot and forward markets for silver, gold, base metals and certain other commodities (such as natural gas, fuel oil and electricity); fluctuations in currency markets (such as the Mexican Peso versus the United States Dollar); changes in national and local government, legislation, taxation, controls or regulations and political or economic developments, particularly in Mexico and in Canada; risks and hazards associated with the business of mineral exploration, development and mining (including environmental hazards, industrial accidents, unusual or unexpected geological or structural formations, pressures, cave-ins and flooding); employee relations; relationships with and claims by local communities and indigenous populations; availability and increasing costs associated with mining inputs and labour; the

speculative nature of mineral exploration and development, including the risks of obtaining necessary licenses and permits and the presence of laws and regulations that may impose restrictions on mining; diminishing quantities of grades of mineral reserves as properties are mined; global financial conditions; challenges to, or difficulty in maintaining, title to properties and continued ownership thereof; the actual results of current exploration activities, conclusions of economic evaluations, and changes in Project parameters to deal with unanticipated economic or other factors; increased competition in the mining industry for properties, equipment, qualified personnel, and their costs; and, with respect to Pan American, those factors identified under the caption Risks related to Pan American s business in Pan American s most recent Form 40F and annual information form filed with the United States Securities and Exchange Commission and Canadian provincial securities regulatory authorities. Investors are cautioned against attributing undue certainty or reliance on forward-looking statements. Although Pan American and Orko have attempted to identify important factors that could cause actual results to differ materially, there may be other factors that cause results not to be as anticipated, estimated, described, or intended. The companies do not intend, and do not assume any obligation, to update these forward-looking statements or information to reflect changes in assumptions or changes in circumstances or any other events affecting such statements or information, other than as required by applicable law.

22.2 Taxes, royalties, levies, and interests

22.2.1 Taxation

The principal taxes of México that affect the La Preciosa Project are income tax, flat tax (IETU), annual fees for holding mineral properties, various payroll and social security taxes, and refundable value added tax. A 28% income tax rate that will become effective in México in 2013 was applied in the economic model. Also included in the model is the IETU, which is defined as a minimum tax in respect to income tax, but with a wider taxable base as many of the tax deductions authorized for income tax are not permitted for IETU. The IETU rate applied in the economic model is 17%.

22.2.2 Royalties

Net smelter return royalties

A Net Smelter Return Royalty Agreement dated June 19, 2002 among Minas Luismin S.A. de C.V., Minas Sanluis, S.A. de C.V. and Corporación Turística Sanluis, S.A. de C.V (CTS) runs with the Property and grants a 3% net smelter returns royalty to CTS on minerals derived on the La Preciosa, Lupita, Fracción La Preciosa, San Patricio, El Choque Tres, and La B.

El Choque Cuatro and El Choque Seis are also included in economic model but they are not subject to any royalties. Therefore, based on the estimated feed grade tonnes to be extracted from the various claims, a weighted average net smelter royalty of 2.5% is applied in the economic model.

22.3 Economic model

The life of Project outcomes are summarised in Table 22.1.

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Table 22.1 La Preciosa Project preliminary financial metric estimates

Measure	Units	Value(1)
Undiscounted present value	$M	497.0
Net present value at 5% discount p.a.	$M	314.6
Net present value at 10% discount p.a.	$M	191.2
Internal rate of return	%	24.3
Payback period	Years	3.3

Note(1): The economic assessment is preliminary in nature. It includes Inferred mineral resources that are considered too speculative geologically to have the economic conditions applied to them that would enable them to be categorised as mineral reserves, and there is no certainty that the preliminary economic assessment will be realised.

Note(2): Based on metal prices of $25/oz silver and $1,250/oz gold. Inclusive of taxes, royalties, and a management fee of 5% of operating costs ($50.9 million) and 5% of initial capital costs ($13.5 million) paid to the operator.

Table 22.2 shows the life of Project cash flow from pre-production to closure. As can be seen on the table a pre-production period of three years has been allowed. The cumulative cash flow is shown in Figure 22.1.

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Table 22.2 La Preciosa Project preliminary economic summary (figures shown in $M)

		Total	**-3**	**-2**	**-1**	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**	**11**	**12**	**13**
Revenue		2243.7			1.1	176.3	228.1	246.6	177.1	203.7	185.3	173.7	166.5	156.9	202.2	150.8	134.9	40.5
	Capital costs																	
	Surface fleet	43.6		14.3	11.7	0.0	0.2	2.2	1.5	1.5	1.5	1.2	3.8	1.4	0.9	0.7	2.7	
	UG fleet	69.1			15.8	21.8	4.7	0.4	1.4	2.6	2.7	4.3	2.7	2.5	4.3	4.1	1.7	
	Plant	143.3		71.7	71.7													
	EPCM	22.1		11.0	11.0													
	Owners costs	64.4	8.8	17.1	22.1	1.3	1.2	1.5	1.2	1.3	1.2	1.5	1.2	1.3	1.2	1.5	1.9	
	Escalation	6.4		1.9	4.5													
	UG development	22.9			3.7	11.1	1.5	1.3	0.4	0.2	1.2		0.3	1.9	1.2			
	Pre-stripping	4.4			4.4													
Reclamation expenses		9.0																9.0
Capital costs	**Pre-production**	269.7	8.8	116.0	144.9													
	Sustaining	106.8				34.2	7.6	5.4	4.6	5.7	6.7	7.1	8.1	7.1	7.7	6.4	6.3	
	Operating costs																	
	OP mining	216.5				16.0	17.0	19.6	20.2	22.5	16.2	16.7	16.8	20.0	23.4	22.2	5.6	0.2
	UG mining	373.0				19.6	30.7	29.0	31.1	33.2	30.7	32.1	32.9	29.8	28.3	30.4	30.0	15.1
	SP re-handle	1.8				0.1			0.5		0.4		0.3	0.1		0.1	0.1	
	Processing	369.6				30.4	30.4	30.4	30.4	30.4	30.4	30.4	30.4	28.0	30.4	30.4	30.4	7.6
	Admin	57.1				4.7	4.7	4.7	4.7	4.7	4.7	4.7	4.7	4.3	4.7	4.7	4.7	1.2
	Total	1018.1				70.9	82.7	83.7	86.9	90.8	82.4	83.9	85.1	82.3	86.7	87.8	70.8	24.1
	Management fee	64.4				17.0	4.1	4.2	4.3	4.5	4.1	4.2	4.3	4.1	4.3	4.4	3.5	1.2
	Royalties	56.1			0.0	4.4	5.7	6.2	4.4	5.1	4.6	4.3	4.2	3.9	5.1	3.8	3.4	1.0
	Taxes	222.6			0.3	14.1	27.3	31.8	14.5	18.2	16.4	14.1	12.5	15.9	27.9	13.5	14.1	2.0
Cash flow	**Net**	497.0	-8.8	-116.0	-144.1	35.8	100.7	115.3	62.3	79.3	71.1	60.1	52.4	43.6	70.6	35.0	36.7	3.1

Figure 22.1 Cumulative cash flow graph

22.4 Sensitivity analysis

Table 22.3 and Figure 22.2 show the sensitivity of the Project to changes in various parameters, including capital and operating costs, silver price, and the proportion of pillars left behind in underground room and pillar mining areas. This final sensitivity is included to assess the Project risk should geotechnical conditions affect the mining dimensions. A sensitivity analysis was also run by mineral resource classification to assess the contribution of Inferred and Indicated mineral resources on the potential mill feed. The removal of Inferred mineral resources (comprising 23% of the feed tonnes and 18% of the contained metal) from the schedule reduces the Project NPV by $117.1 million or 37%.

As expected the Project is highly sensitive to changes in the silver price. The Project is more sensitive to variations in operating costs than capital costs, and altering the proportion of pillars does not translate to a large variation in the Project value.

Table 22.3 Sensitivity analyses

Sensitivity		PV	NPV	IRR (%)	Payback (years)
Base case		497.0	314.6	24.3	3.3
Indicated mineral resources only (no Inferred material)		316.2	197.6	20.1	3.7
Capital costs	-10%	530.5	346.8	28.1	2.9
	+10%	463.1	282.1	21.0	3.8

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Sensitivity		**PV**	**NPV**	**IRR (%)**	**Payback (years)**
	+20%	429.1	249.6	18.2	4.2
Operating costs	-10%	578.7	374.6	27.1	3.0
	+10%	414.7	254.3	21.2	3.6
	+20%	332.4	193.9	18.0	4.1
Silver price ($/oz)	$12	-359.7	-327.2	No return	No payback
	$18	52.9	-18.7	3.5	8.7
	$30	802.9	543.4	35.5	2.5
	$40	1,393.3	985.2	55.5	1.9
Underground R&P (pillars as proportion of panels)	30%	464.7	295.3	23.8	3.2
	20%	529.6	330.7	24.4	3.2
	15%	562.7	347.1	24.5	3.2

Notes: PV is Project value. NPV is at a 5% discount. R&P is room and pillar mining method. IRR is internal rate of return.

Figure 22.2 Sensitivity analysis spider graphs

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23 Adjacent properties

There is no relevant information on adjacent properties to report.

24 Other relevant data and information

There is no additional information to report.

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25 Interpretation and conclusions

25.1 Mineral resources

The Property comprises a block of mineral exploitation concessions covering an area of approximately 1,134 ha located on the eastern flank of the Sierra Madre Occidental mountain range. Conglomerate and andesitic rocks are the main hosts of epithermal quartz veins containing economic levels of silver and gold mineralisation, as well as barite and lesser quantities of base metals, primarily zinc and lead. Two major vein and vein breccia systems are exposed on hills and ridges on either side of an approximately 800 m wide valley. The dominant geological feature on the Property is the northwest-trending La Preciosa Ridge which hosts the dominantly north-striking and westward-dipping main vein system, which includes the Martha, Abundancia, Gloria, Pica, Luz Elena, Sur, and Nueva veins. These veins are cross-cut by east-striking, south-dipping Transversal veins. The major vein breccia system to the east of La Preciosa Ridge on the eastern side of the valley floor includes the northwest striking Zona Oriente and Zona Oriente Extension, which is believed to be the surface expression of the Martha vein.

The most recently estimated mineral resources effective 30 June 2011 are shown in Table 25.1. No mineral reserves have been estimated for the Project.

Table 25.1 La Preciosa Project mineral resource estimate effective 30 June 2011

Mining method	Classification	Cut-off grade Ag ppm	Tonnes (millions)	Ag ppm	Silver million ounces	Au ppm	Gold thousand ounces	Silver equivalent ppm	Silver equivalent million ounces
Open pit	Indicated	35	10.9	129	45	0.19	66	139	49
Open pit	Inferred	35	7.6	74	18	0.13	31	81	20
Underground	Indicated	85	13.9	152	68	0.35	156	170	76
Underground	Inferred	85	7.6	117	28	0.21	52	128	31
Total	Indicated		24.8	142	113	0.28	222	156	124
	Inferred		15.2	96	46	0.17	83	105	51

Notes:

Mineral resources that are not mineral reserves do not have demonstrated economic viability. CIM (2010) defines a mineral reserve as the economically mineable part of a Measured or Indicated mineral resource demonstrated by at least a preliminary feasibility study. No mineral reserves have been estimated. Mineral resources have accounted for minimum mining width and planned mining dilution.

The estimate of mineral resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues, but no such issues have been identified at this time.

Tonnes, grades, and ounces have been rounded and this may have resulted in minor discrepancies in the totals. Grades are expressed in parts per million (ppm) which is equivalent to grams per tonne (g/t).

Cut-off grades are based on operating cost estimates and metal prices of $25 per ounce silver (Ag) and $1,250 per ounce gold (Au). Metal prices are based on a weighted average of historical three year average daily silver prices and a two year future price forecast.

The division between open pit and underground mineral resources is set on a horizontal level at the 1920 m elevation, which is considered close to optimum at the metal prices and operating costs assumed in this preliminary economic assessment.

Silver equivalent grade values assume a gold to silver ratio of 50 to 1 based on the assumed metal prices. The metallurgical recoveries and refining charges are assumed to be the same for silver and gold for the purposes of the equivalence calculation only.

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Within the rock mass at La Preciosa, numerous veins exist that cannot be confidently interpreted from section to section. In open pit mining, it is likely that there is considerable upside potential from un-modelled veins. The ability to realise this upside depends on whether potential metal gained outweighs the additional cost and time in defining and mining these veins. A programme of reverse circulation grade control drilling in advance of mining may be required to define the potential gains.

25.2 Mineral processing, metallurgical testing, and recovery methods

Whole ore agitation cyanidation has been determined to be the most favourable processing option. At the currently estimated global plant feed grade of 137 ppm Ag, the data forecasts a silver extraction of 86%. Gold extraction is affected by oxidation, with a nominal 70% extraction in the sulphide zones increasing to 90% in some of the oxidised zones. An average silver extraction of 86% and an average gold extraction of 78% have been used in this preliminary economic assessment. Further metallurgical test work is planned in order to verify and potentially improve these recoveries.

The consumption of cyanide has been quite variable. Although the majority of the test work reported consumption of less than 1.3 kg/t, some tests were as high as 6 kg/t (which is considered to be a high rate of cyanide consumption) where favourable metallurgical results were reported. Additional testing will be required to better understand the cyanide consumption. It is anticipated that to the extent that elevated cyanide consumption is encountered in operation, the cyanide concentration in the plant solutions will be permitted to decrease below the optimum of 2 g/l, incurring a few percent loss in silver extraction but a net gain as a result of lower cyanide costs.

The deposit benefits from fine grinding to at least P_{80} = 74 microns (80% passing 200 mesh) with a leaching time that is expected to be in the range of three to four days. Small quantities of copper and zinc are leached in the process and report to the pregnant solution as cyanides. They will be destroyed in the operating plant cyanide destruction circuit.

The available metallurgical data and test work was used to develop the Project flow sheets and design criteria. No unproven technologies are planned for the La Preciosa Project and many other process plants within this size range have been constructed in the past.

The plant facilities for La Preciosa have been designed for a feed throughput of 5,000 dry metric tonnes per day. Feed grade material will be processed by crushing and grinding prior to cyanide leaching. Recovery of soluble metals will be accomplished by multi-stage counter current decantation followed by zinc precipitation. The precipitate will be further refined through smelting to yield a doré product for sale that will contain the silver and gold produced at the Project. Leach tailings will go through cyanide destruction prior to discharge at the tailings facility. The tailings facility will be of conventional construction, utilising multiple spigot discharge points and a reclaim barge. A tailing facility containment dam will be constructed from mined waste or surface quarry.

25.3 Mining and financial

Exploiting the entire mineral resource by open pit mining is desirable but limited by a cost prohibitive strip ratio. The optimum mine plan includes three open pits producing a nominal 2,000 tonnes per day of feed material and an underground mine producing a nominal 3,000 tonnes per day of feed. Underground mining and underground mineral resource recovery is dependent on the actual geotechnical conditions encountered in the underground development and in the room and pillar zones. The study shows that using conservative geotechnical parameters, the deposit can be exploited effectively using well known mining techniques. As a consequence of increased knowledge of the deposit as mining progresses, actual mining may result in reduced costs and dilution and improvements in mining recoveries in excess of those that have been used in this study.

The preliminary financial analysis presented in this study shows that based on the commodity price assumptions used, the Project is strongly economic and most sensitive to changes in commodity price.

25.4 Environmental and community

Reporting of environmental baseline data collected for a full year between 2010 and 2011 has been completed for the Project. The baseline data will be used to compile the environmental impact statement (MIA), which will be submitted to the Mexican government for approval prior to the issuing of construction and operation permits. The most likely significant environmental issues that may be related to the permitting of the Project include long-term water quality and quantity management, securing water rights, protracted approval and permitting processes, long-term management of metal leaching and acid rock drainage, construction and operation of the access road, social issues, securing surface rights, and management and liner considerations for the tailings impoundment facility. The main documents to be prepared and submitted to obtain construction and operation permits are the MIA, a forest land use modification, a risk analysis, and an archaeological study report.

Likely social concerns surrounding the development of the Project include acquisition of surface land, water use, perceptions of the cyanide facility and the use of cyanide, operation of the access road, potential imposition of access restrictions to the area, and the expectations that the Project development may generate in surrounding communities with respect to employment and quality of life.

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26 Recommendations

26.1 Project advancement

Based on the positive results of this preliminary economic assessment, the recommendation is made to proceed to a feasibility study. The robust economics of the Project at the assumed commodity prices is demonstrated in the preliminary financial metric estimates summarised in Table 26.1. Inferred mineral resources will be excluded from the estimates forming the basis of the recommended feasibility study, as required by CIM (2010), which will impact the feasibility study financial outcomes.

Table 26.1 La Preciosa Project preliminary financial metric estimates(1)

Measure	Units	Value(2)
Undiscounted present value	$M	497.0
Net present value at 5% discount per annum	$M	314.6
Net present value at 10% discount per annum	$M	191.2
Internal rate of return	%	24.3
Payback period	Years	3.3

Note(1): based on metal prices of $25/oz silver and $1,250/oz gold. Inclusive of taxes, royalties, and a management fee of 5% of operating costs ($50.9 million) and 5% of initial capital costs ($13.5 million) paid to the operator.

The key components of the feasibility study will include:

- Refine the mineral resource estimate as appropriate.
- Further geotechnical assessments to better understand the likely behaviour of the rock when exposed underground to advance the proposed mining methods in this preliminary economic assessment to feasibility-level.
- Collection of additional data on rock structures to finalise pit wall designs.

- A geotechnical programme to evaluate the suitability of the selected tailings storage facility.

- Further metallurgical test work to provide a higher level of confidence in silver and gold recovery and to further test the material properties of the tailings.

- Commence negotiations with land owners in the area.

- Potentially acquire the land and commence drilling the proposed water well for the process plant water supply.

- Potentially complete the preparation of the Project MIA.

- Engineer and perform cost estimates for the plant and associated infrastructure, the open pit mine, and the underground mine to feasibility study level standards.

- Continue with the community relations work programme.

The expected cost of completing the feasibility study, from the issuing of the preliminary economic assessment forward, has been estimated at $8 million (excluding the acquisition costs of land and water rights).

26.2 Mineral resources

Additional infill drilling is recommended to upgrade the currently estimated economic portions of the Inferred mineral resources to Indicated mineral resources for later conversion to mineral reserves as the Project progresses. A number of continuous improvement recommendations have been made in the text of this document to increase confidence in the data used to estimate mineral resources. The estimated costs to implement a drilling programme and the continuous improvement items are approximately $5 million. This drilling programme will likely occur following the completion of the feasibility study and prior to commencement of operations.

A new model of the mineralised zones based on continuing geological interpretations and an updated mineral resource estimate is recommended to reflect the new interpretation as well as a small number of drillholes that were completed since the effective date of this Technical Report.

26.3 Mineral processing, metallurgical testing, and recovery methods

The test work to date indicates that the optimum global processing parameters have been reasonably established in the context of the preliminary economic assessment. To further advance the confidence level, a two staged laboratory programme is recommended in which the first stage will determine the role of pH. The test procedure should include lime addition to the grinding mill. It may be necessary to perform two pairs of tests using identical grind times to evaluate the pH ranges. These tests should include screen assaying directly on the tailing slurry.

Another variability testing programme is recommended as the second stage. Some of the existing variability composites may be used for this programme but approximately 10 additional composites will be required for between 10 and 26 additional tests. The variability samples should be selected from representative locations of the veins and should include some that are at less than the cut-off grades.

This programme has the potential to report a modest global decrease in the silver and gold tailing grades on the order of 5 ppm Ag and 0.01 ppm Au. The proposed testing may alter the feasibility study stage design criteria sufficiently that it will impact the capital and operating costs, particularly so in the grinding circuit.

The optimisation parameters can be investigated by evaluating:

- Finer grinding at nominally P_{80} = 60 - 70 microns.
- Increased cyanide concentration to 2 g/l.
- pH at the better of the 10.0 to 10.5 or 11.0 to 11.5.

The following should be undertaken in these recommended tests:

- Silver and gold assaying only.
- Screen assaying of the leached tailing.
- 96 hour kinetic tests in 24 hour increments.

It is recommended that a larger sample footprint be considered when preparing any new composites, from a range of silver grades spanning 35 ppm and 200 ppm Ag so that a robust recovery algorithm can be developed.

The second phase of the test programme should also include establishing the correlation between head grade and recovery for each feed type and the correlation between feed type and processing equipment sizing. Tests should be undertaken to establish ball work, rod mill, and crushing work indices, and Julius Kruttschnitt (JK) drop weight for the expected feed types. Testing for tailing filterability in vacuum and pressure filters to evaluate a filtered tailing option should take place if this option is considered likely. Cyanide destruction testing should be undertaken to establish the optimum reaction conditions in terms of percent solids, reagent consumption, and residence time.

The anticipated cost for these test programmes is $300,000 to $550,000, not including any allowance for any costs for composite selection and preparation. This budget is included in the anticipated costs for the feasibility study.

26.4 Mining and financial

It is recommended that the Project be progressed to a feasibility study where more detailed mining engineering can be undertaken, including written quotations for the supply of equipment, consumables, and services. Such a feasibility study will develop costs and cash flow estimates to an accuracy of ±15% and provide the La Preciosa JV Company with the required confidence to seek the necessary finance to build the Project. The anticipated cost for the mining engineering component of such a study is between $600,000 and $1 million and has been included in the anticipated costs for the feasibility study.

Inferred mineral resources will be excluded from the estimates forming the basis of the recommended feasibility study, as required by CIM (2010), which will impact the feasibility study financial outcomes.

Further geotechnical assessments should be conducted in order to better understand the likely behaviour of the rock as mining advances in the highly fractured zones in the veins and the immediate hanging wall of the veins. The greater certainty required as the project advances to feasibility study may necessitate the selection of an underground mining method with cemented backfill in order to provide additional support to allow safe mining in the highly fractured zones. The selection of a higher cost underground mining method is likely to negatively impact the cut-off grade and the mine plan, and ultimately on the project s economic returns.

26.5 Environmental and community

It is recommended that the Project commence the process to acquire the necessary rights to the land in the area of the Project which will allow the infrastructure to be commissioned on the Property. The access rights will include the area for the well, transmission corridors, mine and mill areas, tailings storage facility, buffer zone, etc. As water supply is critical to the success of the Project, it is recommended that water rights required for the planned facilities also be acquired. The costs of the land and water rights will be negotiated. Annual water costs will likely be approximately $6.5 million Mexican Pesos once the Project enters into production. Continued community engagement is expected to cost $1 million Mexican Pesos annually.

Collection of environmental data which may be important for monitoring and comparisons during future operations should continue. These activities include:

- The installation of a weather station for meteorological data collection.
- The installation of monitoring wells and piezometers for open pit areas, upstream and downstream from the tailings storage facilities and process plant, to monitor ground water characteristics.
- Design and implementation of an environmental monitoring programme.
- Dust monitoring.
- Static and kinetic testing for MLARD which will be coupled with an ongoing monitoring programme for waste rock and tailings.

These activities considered are anticipated to cost approximately $240,000. Approximately one half of this cost is included in the $8 million estimated for the feasibility study.

27 References

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Whiting, 2008	Technical Report Mineral Resource Estimate V on La Preciosa Silver-Gold Deposit Durango State, México. Technical Report prepared by Ben Whiting for Orko Silver Corporation, March 2008.

28 Date, signatures, and certificates

CERTIFICATE of QUALIFIED PERSON

I, Mr. Anthony Finch, P.Eng., M.AusIMM, B.Eng., B.Econ., Divisional Manager of Mining with Snowden Mining Industry Consultants Inc., 600-1090 W. Pender St., Vancouver, BC., V6E 2N7, do hereby certify that:

1. I am one of the authors of the technical report for the La Preciosa Property entitled Pan American Silver Corp. and Orko Silver Corp.: La Preciosa Silver Property, Durango, México, Preliminary Economic Assessment Technical Report with an effective date of 30 June 2011 (the Technical Report).

2. I graduated with a degree in Mining Engineering from the University of Queensland in Australia in 1986. I am a Professional Engineer in the Province of British Columbia, number 164687, and a Member of the Australasian Institute of Mining and Metallurgy, number 103583. Since graduation I have had 25 years continuous experience in the mining industry, in both operations and consulting, in various roles of increasing seniority. I have worked in hard rock underground mining, including precious metals, for over ten years, and in open pit mining for over ten years. By reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be considered a qualified person , as described in Section 1.1 of National Instrument 43-101, Standards of Disclosure for Mineral Projects (NI 43-101).

3. I visited the La Preciosa Property on 5 July 2011.

4. I am responsible for sub-sections 1: Summary; 2: Introduction; 3: Reliance on Other Experts; 4: Property Description and Location; 5: Accessibility, Climate, Local Resources, Infrastructure and Physiography; 12: Data Verification; 15: Mineral Reserve Estimates; 16: Mining Methods; 19: Market Studies and Contracts; 20: Environmental Studies, Permitting and Social or Community Impact; 21: Capital and Operating Costs; 22: Economic Analysis; 24: Other relevant data and information; 25: Interpretation and Conclusions; 26: Recommendations; and 27: References of the Technical Report.

5. I am independent of both Pan American Silver Corp. and Orko Silver Corp. as described in Section 1.5 of NI 43-101.

6. I have had no prior involvement with the La Preciosa Property.

7. I have read NI 43-101 and the portions of the technical report for which I am responsible have been prepared in accordance with NI 43-101.

8. As of the effective date of the technical report and to the best of my knowledge, information, and belief, the portions of the Technical Report for which I am responsible contains all the scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated the 21st day of September, 2011

[signed]

Mr. Anthony Finch, P.Eng, M.AusIMM, B.Eng.(Min), B.Econ.

September 2011

CERTIFICATE of QUALIFIED PERSON

I, Mr. Michael Stewart, MAIG, MSc., CFSG, Principal Consultant with Quantitative Geoscience Pty Ltd., Level 2, 25 Cantonment Street, Fremantle, Western Australia, 6160 Australia, do hereby certify that:

1. I am one of the authors of the technical report for the La Preciosa Property entitled Pan American Silver Corp. and Orko Silver Corp.: La Preciosa Silver Property, Durango, México, Preliminary Economic Assessment Technical Report with an effective date of 30 June 2011 (the Technical Report).

2. I graduated with an MSc in Geology from the University of Canterbury, Christchurch, New Zealand in 1988 and earned a CFSG post graduate Diploma in Geostatistics from Ecole des Mines de Paris, Fontainebleau, France, in 2003. I am a Member of the Australian Institute of Geoscientists, number 3119. Since graduation I have had 23 years continuous employment as a geologist, including 19 years within the metalliferous mining industry. For the past six years I have been employed as a consultant engaged in mineral resource and mining related consulting across a wide range of commodities, including precious and base metals. By reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be considered a qualified person , as described in Section 1.1 of National Instrument 43-101, Standards of Disclosure for Mineral Projects (NI 43-101).

3. I visited the La Preciosa Property on 5 July 2011.

4. I am responsible for sub-sections 1: Summary; 2: Introduction; 6: History; 7: Geological Setting and Mineralisation; 8: Deposit Types, 9: Exploration; 10: Drilling; 11: Sample Preparation, Analyses and Security; 12: Data Verification: 14: Mineral Resource Estimates; 23: Adjacent Properties; 25: Interpretation and Conclusions; and 26: Recommendations of the Technical Report.

5. I am independent of both Pan American Silver Corp. and Orko Silver Corp. as described in Section 1.5 of NI 43-101.

6. I have had no prior involvement with the La Preciosa Property.

7. I have read NI 43-101 and the portions of the technical report for which I am responsible have been prepared in accordance with NI 43-101.

8. As of the effective date of the technical report and to the best of my knowledge, information, and belief, the portions of the Technical Report for which I am responsible contains all the scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated the 21st day of September, 2011

[signed]

Mr. Michael Stewart, MAIG, MSc.

September 2011

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CERTIFICATE of QUALIFIED PERSON

I, Mr. Joshua Snider, P.E., B.S., Engineer and Project Manager with M3 Engineering & Technology Corp., 2051 W. Sunset Road, Suite 101, Tucson, Arizona 85704 USA, do hereby certify that:

1. I am one of the authors of the technical report for the La Preciosa Property entitled Pan American Silver Corp. and Orko Silver Corp.: La Preciosa Silver Property, Durango, México, Preliminary Economic Assessment Technical Report with an effective date of 30 June 2011 (the Technical Report).

2. I graduated with a B.Sc. degree in Civil Engineering from the University of Arizona in 1996. I am a Professional Engineer in the State of Arizona, number 41971. I have worked as a consulting engineer since 1996. By reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be considered a qualified person , as described in Section 1.1 of National Instrument 43-101, Standards of Disclosure for Mineral Projects (NI 43-101).

3. I visited the La Preciosa Property on 9 June 2010.

4. I am responsible for sub-sections 1: Summary; 2: Introduction; 12: Data Verification; 18: Project Infrastructure; 21: Capital and Operating Costs; 25: Interpretation and Conclusions; and 26: Recommendations of the Technical Report.

5. I am independent of both Pan American Silver Corp. and Orko Silver Corp. as described in Section 1.5 of NI 43-101.

6. I have had no prior involvement with the La Preciosa Property.

7. I have read NI 43-101 and the portions of the technical report for which I am responsible have been prepared in accordance with NI 43-101.

8. As of the effective date of the technical report and to the best of my knowledge, information, and belief, the portions of the Technical Report for which I am responsible contains all the scientific and technical information that is required to be disclosed to make the Technical

Report not misleading.

Dated the 21st day of September, 2011

[signed]

Mr. Joshua Snider, P.E., B.Sc.

September 2011

CERTIFICATE of QUALIFIED PERSON

I, Mr. Thomas L. Drielick, P.E., B.Sc., MBA, Senior Vice President with M3 Engineering & Technology Corp., 2051 W. Sunset Road, Suite 101, Tucson, Arizona 85704 USA, do hereby certify that:

1. I am one of the authors of the technical report for the La Preciosa Property entitled Pan American Silver Corp. and Orko Silver Corp.: La Preciosa Silver Property, Durango, México, Preliminary Economic Assessment Technical Report with an effective date of 30 June 2011 (the Technical Report).

2. I graduated with a B.Sc. degree in Metallurgical Engineering from Michigan Technology University in 1970 and a M.BA from Southern Illinois University in 1973. I am a Professional Engineer in the State of Arizona, number 22958. I have practiced metallurgical engineering for 40 years. I worked for the US Army as a metallurgical engineer; for Kennecott Corporation as a process and plant metallurgical engineer and as operations foreman; for Newmont Mining Corporation as a project manager and project engineer; and for M3 as a consulting project manager and metallurgist. By reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be considered a qualified person , as described in Section 1.1 of National Instrument 43-101, Standards of Disclosure for Mineral Projects (NI 43-101).

3. I have not visited the La Preciosa Property and there was no requirement for me to do so because a current personal inspection has been conducted by other qualified persons responsible for the preparation of this technical report, and no additional beneficial information would have been derived from a site visit at this stage of the Project.

4. I am responsible for sub-sections 1: Summary; 2: Introduction; 12: Data Verification; 17: Recovery Methods; 25: Interpretation and Conclusions; and 26: Recommendations of the Technical Report.

5. I am independent of both Pan American Silver Corp. and Orko Silver Corp. as described in Section 1.5 of NI 43-101.

6. I have had no prior involvement with the La Preciosa Property.

7. I have read NI 43-101 and the portions of the technical report for which I am responsible have been prepared in accordance with NI 43-101.

8. As of the effective date of the technical report and to the best of my knowledge, information, and belief, the portions of the Technical Report for which I am responsible contains all the scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated the 21st day of September, 2011

[signed]

Mr. Thomas L. Drielick, P.E., B.Sc. MBA

September 2011

CERTIFICATE of QUALIFIED PERSON

I, Mr. Gary Hawthorn, P.Eng., B.Sc., President of Westcoast Mineral Testing Inc., of 2806 Thorncliffe Drive, North Vancouver, BC., Canada, do hereby certify that:

1. I am one of the authors of the technical report for the La Preciosa Property entitled Pan American Silver Corp. and Orko Silver Corp.: La Preciosa Silver Property, Durango, México, Preliminary Economic Assessment Technical Report with an effective date of 30 June 2011 (the Technical Report).

2. I graduated with a B.Sc. in Mining Engineering from Queen s University, Kingston, Ontario, Canada, in 1964. I have been registered as professional engineer (#8084) in the Province of British Columbia since 1972. I have worked as a mineral processing engineer for 46 years and as a professional engineer for 38 years. My experience includes 18 years, mainly as a mill superintendent for Cominco Ltd. And Placer Development Ltd. Starting in 1982 I have been self employed as a consulting Mineral Processing Engineer and since 1988 have operated as Westcoast Mineral Testing Inc. By reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be considered a qualified person , as described in Section 1.1 of National Instrument 43-101, Standards of Disclosure for Mineral Projects (NI 43-101).

3. I have not visited the La Preciosa Property and there was no requirement for me to do so because a current personal inspection has been conducted by other qualified persons responsible for the preparation of this technical report, and no additional beneficial information would have been derived from a site visit at this stage of the Project.

4. I am responsible for sub-sections 1: Summary; 2: Introduction; 12: Data Verification; 13: Mineral Processing and Metallurgical Testing; 25: Interpretation and Conclusions; and 26: Recommendations of the Technical Report.

5. I am independent of both Pan American Silver Corp. and Orko Silver Corp. as described in Section 1.5 of NI 43-101.

6. My prior involvement with the La Preciosa Property was to oversee limited metallurgical testing in 2007 and 2008 and to act as one of the co-authors of the Technical Report entitled Technical Report on the La Preciosa Project, Durango State, Mexico dated 31 March 2009.

7. I have read NI 43-101 and the portions of the technical report for which I am responsible have been prepared in accordance with NI 43-101.

8. As of the effective date of the technical report and to the best of my knowledge, information, and belief, the portions of the Technical Report for which I am responsible contains all the scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated the 21st day of September, 2011

[signed]

Mr. Gary Hawthorn, P.Eng., B.Sc.

September 2011

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SIGNATURES

Pursuant to the requirements of the Securities Exchange Act of 1934, the registrant has duly caused this report to be signed on its behalf by the undersigned, thereunto duly authorized.

			PAN AMERICAN SILVER CORP (Registrant)
Date:	September 21, 2011	By:	*/S/ ROBERT PIROOZ*
			Name: Robert Pirooz
			Title: General Counsel and Director