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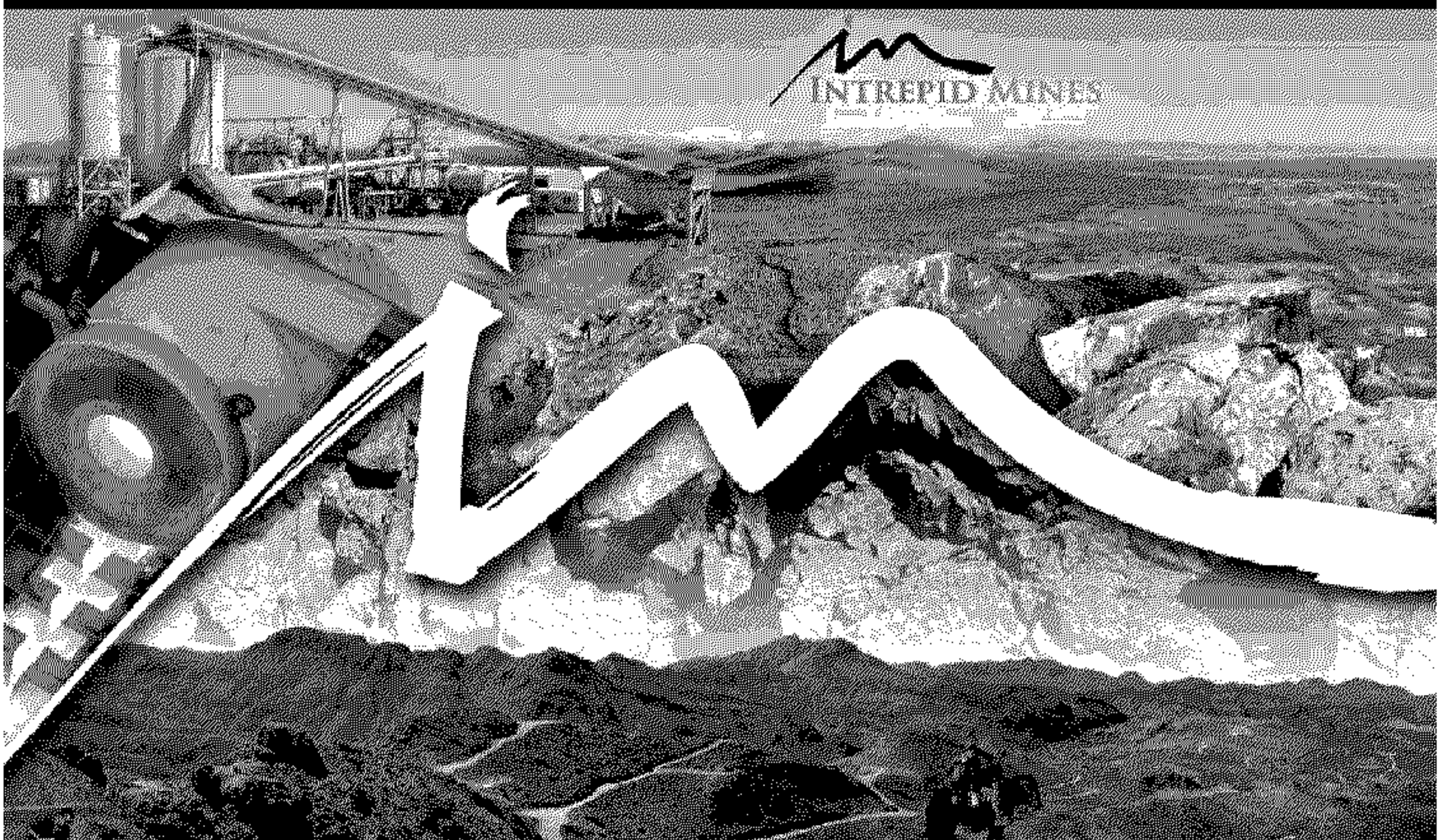
INTREPID MINES

Casposo Project - San Juan, Argentina
Technical Report on Feasibility Study



Prepared for:
Intrepid Mines Limited

By:
William Colquhoun, FSAIMM
Gary Taylor, P. Eng.
Rodrigo Marinho, CPG, AIPG
Armando Simon, R. P. Geo. MAIG

March 30, 2007

150164

CERTIFICATE OF QUALIFIED PERSON

William Colquhoun (FSAIMM)
AMEC (Perú) S.A.
Calle Las Begonias 441, Piso 8, San Isidro, Lima, Perú
Tel: (1) 221 3130
Fax: (1) 221 3143

I, William Colquhoun, FSAIMM, am employed as a Project Manager and Principal Metallurgist with AMEC (Perú) S.A., a division of AMEC Americas Limited.

This certificate applies to the Technical Report entitled "Casposo Project, San Juan, Argentina, Technical Report on Feasibility Study" dated 30 March 2007.

I am a member of the Engineering Council of South Africa (Registration 96003) and a member of the Southern African Institute of Mining and Metallurgy (SAIMM, #). I graduated from the University of Strathclyde with a Bachelor of Science degree in Chemical and Process Engineering in 1982.

I have practiced my profession for 25 years. Since 1982 I have continually been involved in mineral processing projects for precious and base metals and industrial minerals in South Africa, Ethiopia, Canada, the United States, Australia, Chile, Perú, Argentina, Ecuador, Brazil, Ukraine, Mongolia, Russia and the Middle East. I have been directly involved in the preparation of feasibility studies relating to gold and silver projects and metallurgical investigations supporting these.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43–101 *Standards of Disclosure for Mineral Projects* (NI 43–101).

I visited the Casposo Site between 21 to 23 March 2005, 11 to 13 January 2006 and 5 to 7 April 2006.

I am responsible for Sections 1, 2, 3, 4, 16, 18.2 to 18.7, 19, 20, 21 and 22 of the Technical Report titled "Casposo Project, San Juan, Argentina, Technical Report on Feasibility Study" dated 30 March 2007, and for the portions of the summary, conclusions and interpretations, and recommendations that are based on these Sections.

I am independent of Intrepid Mines Limited as independence is described by Section 1.4 of NI 43-101.

I have been involved with Casposo Project from December 2005 and March 2007 as manager of the Feasibility Study, metallurgical design and financial analysis.

I have read NI 43–101 and this report has been prepared in compliance with that Instrument.

As of the date of this certificate, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

"signed and sealed"

William Colquhoun Pr Eng
Project Manager and Principal Metallurgist

Dated: 25 May 2007

CERTIFICATE OF QUALIFIED PERSON

Gary W. Taylor, P.Eng
AMEC Americas Limited
200 – 421 Downey Road, Saskatoon,
Saskatchewan, Canada S7N 4L8
Tel: (306) 477-1155; Fax: (306) 477-1161
gary.taylor@amec.com

I, Gary W. Taylor, P. Eng., am employed as a Manager, Mining with AMEC Americas Limited.

This certificate applies to the Technical Report entitled "Casposo Project, San Juan, Argentina, Technical Report on Feasibility Study" dated 30 March 2007.

I am a member of the Association of Professional Engineers of Saskatchewan. I graduated from McGill University with a M. Eng. (Mining) in 1973.

I have practiced my profession for 34 years. I have been directly involved in mine operations of gold mines, mine design and management of Feasibility Studies for gold and other mining projects throughout my career.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43–101 *Standards of Disclosure for Mineral Projects* (NI 43–101).

I visited the Casposo Project property from April 5 to 7, 2006.

I am responsible for Sections 17.2, 17.3 and 18.1 of the Technical Report titled "Casposo Project, San Juan, Argentina, Technical Report on Feasibility Study" dated 30 March 2007, and for the portions of the summary, conclusions and interpretations, and recommendations that are based on these Sections.

I am independent of Intrepid Mines Limited as independence is described by Section 1.4 of NI 43-101.

I have been involved with the Casposo Project from April 2006 to May 2007 in the mine design and estimation of the Reserves for the Feasibility Study.

I have read NI 43–101 and this report has been prepared in compliance with that Instrument.

As of the date of this certificate, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

"signed and sealed"

Gary W. Taylor, P.Eng.

Dated: May 25, 2007

CERTIFICATE OF QUALIFIED PERSON

Armando Simón, Ph.D., Principal Geologist, R.P.Geo, (MAIG)
Américo Vespucio 100 Sur, Oficina 203
Las Condes, Santiago, Chile.
Tel. 56-2-210-9500; Fax 56-2-210-9510
armando.simon@amec.com

I, Armando Simón, Ph.D., R.P.Geo (MAIG), am a Principal Geologist for AMEC International (Chile) S.A., a division of AMEC Americas Limited.

This certificate applies to the Technical Report entitled "Casposo Project, San Juan, Argentina, Technical Report on Feasibility Study" dated 30 March 2007.

I am a Registered Professional Geoscientist (R.P.Geo) with, and a Member of, the Australian Institute of Geoscientists (MAIG # 3003). I graduated from the University of Bucharest with a Bachelor of Engineering degree in Geology and Geophysics in 1974, and a Doctorate of Engineering with mention in Economic Geology in 1985.

I have practiced my profession for 32 years. Since 1974 I have continually been involved in mineral exploration projects for precious and base metals and industrial minerals in Argentina, Brazil, Colombia, Cuba, Chile, Eritrea, Ethiopia, Guyana, Jamaica, Madagascar, Nicaragua, Peru, Portugal, Romania and Vietnam.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43–101 *Standards of Disclosure for Mineral Projects* (NI 43-101).

I visited the Casposo Property on the following dates: April 17 to 19, 2005, December 5 to 9, 2005, and August 28 to 30, 2006.

I am responsible for the preparation of Sections 5 to 15 of the Technical Report entitled "Casposo Project, San Juan, Argentina, Technical Report on Feasibility Study" dated 30 March 2007, and for the portions of the summary, conclusions and interpretations, and recommendations that are based on these Sections.

I am independent of Intrepid Mines Limited, as independence is described by Section 1.4 of NI 43-101.

I have been involved with the Casposo Project from April 2006 to May 2007 in the areas of geology and QA/QC for the Feasibility Study.

I have read NI 43-101, and this report has been prepared in compliance with that Instrument.

As of the date of this certificate, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Dated 25th May 2007.

"signed and sealed"

Armando Simón, Ph.D.,
Principal Geologist
R.P.Geo. (MAIG # 3003)

Dated: 25 May, 2007

CERTIFICATE OF QUALIFIED PERSON

Rodrigo Alves Marinho, Senior Geologist, CPG, (AIPG)
Américo Vespucio 100 Sur, Oficina 203
Las Condes, Santiago, Chile.
Tel. 56-2-210-9500; Fax 56-2-210-9510
rodrigo.marinho@amec.com

I, Rodrigo Alves Marinho, CPG (AIPG) am employed as a Senior Geologist with AMEC International (Chile) S.A., a division of AMEC Americas Limited.

This certificate applies to the Technical Report entitled "Casposo Project, San Juan, Argentina, Technical Report on Feasibility Study" dated 30 March 2007.

I am a member of the American Institute of Professional Geologists (CPG-10971). I graduated from University of Sao Paulo State with a Bachelor of Engineering degree in Geology in 1993.

I have practiced my profession for 14 years. I have been directly involved in mineral exploration and mining projects for precious and base metals and industrial minerals in Argentina, Australia, Brazil, Burkina Faso, Colombia, Chile, Peru, Portugal, South Africa, United States and Venezuela.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43–101 *Standards of Disclosure for Mineral Projects* (NI 43–101).

I am responsible for the preparation of Section 17.1 Casposo Project, San Juan, Argentina, Technical Report on Feasibility Study" and for the portions of the of the summary, conclusions and interpretations, and recommendations that are based on this Section

I am independent of Intrepid Mines Limited as independence is described by Section 1.4 of NI 43-101.

I have been involved with the Casposo Project from Nov 2005 to April 2006 as responsible for auditing the resource estimate of this property.

I have read NI 43–101 and this report has been prepared in compliance with that Instrument.

As of the date of this certificate, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

"signed and sealed"

Rodrigo Alves Marinho.
Senior Geologist
CPG-AIPG (CPG-10971)

Dated: 25 May, 2007

IMPORTANT NOTICE

This report was prepared as a National Instrument 43-101 Technical Report, in accordance with Form 43-101F1, for Intrepid Mines Limited (Intrepid) by AMEC (Perú) S.A. a division of Americas Limited (AMEC). The quality of information, conclusions, and estimates contained herein is consistent with the level of effort involved in AMEC´s services, based on: i) information available at the time of preparation, ii) data supplied by outside sources, and iii) the assumptions, conditions, and qualifications set forth in this report. This report is intended for use by Intrepid subject to the terms and conditions of its contract with AMEC. This contract permits Intrepid to file this report as a Technical Report with Canadian Securities Regulatory Authorities pursuant to National Instrument 43-101, Standards of Disclosure for Mineral Projects. Any other use of this report by any third party is at that party's sole risk.

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1.0 SUMMARY

In December 2005, Intrepid Mines Limited (Intrepid) commissioned AMEC (Peru) S.A. (AMEC) to prepare a bankable feasibility study for the Casposo gold–silver (Au–Ag) Project. The Project is located in the San Juan Province of northwest Argentina in the Department of Calingasta and is approximately 150 kilometres west, or a three hour drive, from the city of San Juan. It is centered on approximately latitude 31°12' S and longitude 69°36' W, and has an average elevation of about 2,400 masl.

The site is accessible year round by road. Access to the Project north from the international airport in Mendoza follows National Route 40 via the city of San Juan to the town of Talacasto, then along Provincial Routes 436, 414 and 12 to the village of Calingasta (the population centre nearest to the Project), and finally along Provincial Route 412 to the main Project access road. The region can also be accessed from the city of Mendoza via a separate southern route. The nearest town is Calingasta, a small village of 2,000 people, 20 km to the southeast. Paved roads lead to within 15 km of the property boundary, followed by 17 km of gravel roads, which lead to the Project area.

The Project is located at the base of rugged terrain, characterized by low mountains with steep slopes and ravines associated with dry drainage systems. The property is arid and sparsely vegetated. The median annual rainfall is 75 mm and the median temperature is 14.5 °C.

From 1993 to 1999, Battle Mountain Gold (BMG) conducted regional exploration programs in San Juan Province, Central Argentina, and on the basis of Landsat interpretation and the analysis of favourable mineralization criteria identified possible precious metal systems in the region including Casposo. Following this, BMG carried out 8,626 m of core drilling in 46 diamond drill holes at Casposo, as well as about 2,400 m of trenching in 37 trenches.

Intrepid has carried out exploration on the Casposo site since July 2002. This work has comprised mapping, channel and pit sampling, geophysics and diamond and reverse circulation drilling. In addition to exploration, Intrepid has undertaken metallurgical testing of the mineralized vein systems, resource and reserve estimation, and preliminary assessment studies, as well as initiated environmental baseline monitoring programs. The work has included 165 holes (153 diamond holes and 12 RC holes) for 27,274 m.

The Kamila Zone and associated prospects form a mineralized corridor approximately six km long that is characterised by an Au–Ag-rich low- to intermediate-sulphidation epithermal vein complex which outcrops principally at the Kamila and Mercado deposits (collectively the Kamila Zone). Au–Ag mineralization is structurally controlled, and occurs in crustiform-colloform quartz veins and stockworks, hosted in andesites and rhyolites.

The 2007 Feasibility Study was based on the proposed mining of two separate deposits-Kamila and Mercado. The Kamila deposit is much larger than Mercado and will be mined first. The top part of Kamila is proposed to be mined by open pit methods and the deeper portion by underground mining methods. The Kamila open pit will consist of a large pit (Kamila Main pit) and a small satellite pit (Kamila SE pit) located 100 m to the southeast.

Total Indicated Resources (effective date 10 October 2006) at the Kamila and Mercado deposits comprise 2.18 Mt grading 4.46 g/t Au and 116 g/t Ag, at a cut-off grade of 1.40 g/t gold equivalent (AuEq). Using the same AuEq cut-off grade, Inferred Resources for the deposits total 0.14 Mt at 2.12 g/t Au and 83.3 g/t Ag. The cut-off grade is based on a gold price of US$450/oz, and takes into consideration assumed process and administrative costs, recoveries, and gold:silver ratios, but does not consider mining costs as these values are an incremental cut-off that would be applied to loaded haulage trucks as they exit from the open pit.

Probable Open Pit Mineral Reserves (effective date 30 March 2007) for the planned Kamila, Kamila SE and Mercado open pits total 1.27 Mt at 5.4 g/t Au and 92.7 g/t Ag, using a cut-off grade of 1.98 AuEq. A further Probable Stockpile Mineral Reserve of 0.08 Mt at 1.25 g/t Au and 24.9 g/t Au at a cut-off grade of 1.70 g/t AuEq is contained within lower-grade stockpiles.

Underground Probable Mineral Reserves for the planned Kamila underground operation are estimated at 0.41 Mt at 3.22 g/t Au and 198.3 g/t Ag, using a cut-off grade of 3.50 g/t AuEq. A further Probable Mineral Reserve of 0.03 Mt at 4.38 g/t Au and 112.3 g/t Ag, using the same cut-off grade, is contained within remnants in the planned underground. Combined underground and remnant Probable Reserves total 0.44 Mt at 3.31 g/t Au and 191.7 g/t Ag at an AuEq cut-off grade of 3.5 g/t Au.

The higher cut-off AuEq g/t values used in open pit and underground mineral reserve estimation, as opposed to resource estimation, are due to inclusion of appropriate mining costs in the derivation of the gold equivalency equation.

The Casposo Project will generate an estimated 10 Mt of waste rock. About 2 Mt of the mine waste rock is currently planned for use in road construction, in-pit disposal and underground backfill. On this basis the waste rock dump is designed to store about 8 Mt.

The mine plan is based on a contract mining company providing ore to a conventional gold and silver whole leach recovery plant to support an average milling rate of 1,000 t/d (365,000 t/a). All facilities are designed for this throughput and operate on a continuous basis, 24 h/d, 365 d/a. For the average life of mine head grades the overall gold and silver recoveries are projected to be 93.7% and 80.6% respectively.

Average annual production is estimated to be about 50,478 oz of gold and 1.1 Moz of silver or 68,483 oz of gold equivalent annually (using base case financial analysis gold and silver pricing of US$500 and US$8.50/oz respectively) over the mine life of five years.

The process design and operational plan is based on existing technology. The process flowsheet will use conventional primary jaw and secondary cone crushing, ball milling, gravity concentration for coarse gold and silver, cyanide leach, counter current decantation and washing and dewatering of tails by belt filtration. Gold and silver will be recovered by standard Merrill-Crowe zinc precipitation and smelted to produce dorè bars. The leached tailings, following filtration to recover precious metals, will be washed and rinsed on the same belt filter to remove cyanide. The cyanide wash solution will be collected for the destruction of cyanide using a conventional SO_2/air process. The detoxified solution is recycled to the belt filter as wash solution. This will minimize the fresh water requirements for the process. The filtered tailings will be trucked to a lined tailings management facility and stacked in compacted lifts.

The planned Casposo processing plant will generate 1,000 t/d of tailings over its estimated five year operating life, resulting in some 1.8 Mt of tailings solids to be permanently stored in a lined dry-stack tailings management facility.

Other than the gravel access road and a weather station, no other infrastructure currently exists in the Project area. The major infrastructure required to develop the property includes site roads, power generation diesel plant, water supply, fuel storage, mining contractor area and administration facilities.

AMEC have estimated the capital cost to build the facilities to first gold pour to be US$45.5 M including working capital and 12% contingency. Not included are estimated sustaining costs of US$2.2 M, which are included directly in the Project cash flow.

Operating costs include all costs required to process 1,000 t/d of ore including normal surface and underground mine development and extension of principal underground mine ramps and raises. The life of mine Total Operating Costs, including mining, processing, and general and administration, are projected to be US$47.34/t of ore milled or US$248/oz of gold equivalent and US$168/oz of gold net of silver credits.

Base gold and silver prices used in the financial analysis were US$500/oz and US$8.50/oz. Silver accounts for about 25% of total revenue in the Project cash flow. Results of the base case financial analysis indicate that the Project has a potential after-tax internal rate of return of 15 % and net present value of US$10.8 M at a discount rate of 6%. The base case scenario has a projected payback period of approximately three years, from a five year mine life.

The results of the economic analysis represent forward-looking information that are subject to a number of known and unknown risks, uncertainties and other factors that may cause actual results to differ materially from those presented here.

The financial analysis indicates the Project is more sensitive to changes in metal price and grade than either capital or operating costs and is slightly more sensitive to operating costs than capital costs. The IRR rises to 33% if long term gold and silver prices are raised to US$600/oz and US$11.50/oz, respectively.

The results of the economic analysis represent forward-looking information as defined under Canadian securities law. The results depend on inputs that are subject to a number of known and unknown risks, uncertainties and other factors that may cause actual results to differ materially from those presented here. Factors that could cause such differences include, but are not limited to: changes in commodity prices, costs and supply of materials relevant to the mining industry, the actual extent of the mineral resources compared to those that were estimated, actual mining and metallurgical recoveries that may be achieved, technological change in the mining, processing and waste disposal, changes in operation. Forward-looking information in this analysis includes statements regarding future mining and mineral processing plans, rates and amounts of metal production, tax and royalty terms, smelter and refinery terms, the ability to finance the project, and metal price forecasts.

2.0 INTRODUCTION

AMEC Americas Limited (AMEC) was commissioned by Intrepid Mines Limited (Intrepid), to provide an independent Qualified Person's Review and Technical Report for the Casposo gold–silver Project located in the Department of Calingasta, San Juan Province, Argentina. The Project is operated by Intrepid's wholly-owned subsidiary company, Intrepid Minerals Corporation (IML). In this report, the names IML and Intrepid are used interchangeably.

The Report has been prepared in compliance with National Instrument 43–101, Standards of Disclosure for Mineral Projects (NI 43–101) and documents the results of a feasibility study on the Casposo Project completed by AMEC in March 2007 (the Feasibility Study). As part of the Feasibility Study, AMEC also updated the mineral resources and completed a mineral reserve estimate for the Casposo Project in conformance with the CIM Mineral Resource and Mineral Reserve definitions referred to in NI 43–101.

The Qualified Persons (QPs), as defined in NI 43–101 and in compliance with Form 43-101F1 (the Technical Report), responsible for the preparation of the technical report include:

- William Colquhoun (FSAIMM), Principal Metallurgist and Project Manager, Casposo Feasibility Study (AMEC, Lima), responsible for the metallurgy and financial evaluation sections.
- Gary Taylor (P. Eng.), Manager Mining (AMEC, Saskatoon), responsible for the mining sections and reserves.
- Rodrigo Marinho (CPG, AIPG), Senior Geologist, (AMEC, Santiago), responsible for the resources.
- Armando Simon (R.P. Geo MAIG), Principal Geologist (AMEC, Santiago), responsible for the geology and data analysis sections.

AMEC QPs conducted site visits to the Casposo Project between 2005 and 2007 as shown below:

- William Colquhoun visited from 21 to 23 March 2005, 11 to 13 January 2006 and 5 to 7 April 2006.
- Gary Taylor visited from 5 to 7 April 2006.
- Armando Simon visited from 17 to 19 April 2005, 5 to 9 December 2005 and 28 to 30 August 2006.
- Rodrigo Marinho did not complete a site visit.

The AMEC Feasibility Study team visited the Project Site during 2005–2007 to determine sites for the process plant and on-site infrastructure, to participate in collecting samples for metallurgical test work and to investigate available support facilities and contractors for the Project. Meetings were held with key Intrepid personnel during the Feasibility Study regarding sample collection, preparation and analysis, estimation of mineral resources and mineral reserves, mining plans, the plant and process-related issues, tailings, permitting, and capital and operating costs.

The personnel and areas of responsibility for the Feasibility Study are shown in Table 2-1.

Table 2-1
Site Visit Feasibility Study

Person	Company	Title	QP for 30 March 2007 NI 43-101 Technical Report	Site Visit Date	Area of Work
William Colquhoun	AMEC	Principal Metallurgist and Project Manager, Casposo Feasibility Study	QP	21 to 23 March 2005, 11 to 13 January 2006 and 5 to 7 April 2006.	Managed preparation of Feasibility Study. QP for Metallurgical, Engineering, Environmental and Financial Sections of November 2006 43-101 Report.
Rodrigo Marinho	AMEC	Senior Geologist	QP	No visit	Mineral Resource Estimate Section of Feasibility Study, performed under supervision of Mr Armando Simon
Armando Simon	AMEC	Principal Geologist	QP	17 to 19 April 2005, 5 to 9 December 2005 and 28 to 30 August 2006	QP for Geology and Resources Sections of May 2007 43-101 Report.
Gary Taylor	AMEC	Manager Mining	QP	5 to 7 April 2006	QP for Mining Section of Feasibility Study and May 2007 43-101 Report
Raimudo Almenara	Formerly AMEC	Manager Geotechnical		21 to 23 March 2005 and 5 to 7 April 2006	Geotechnical Section of Feasibility Study
Michael Shelbourn	AMEC	Senior Geotechnical Engineer		7 to 8 December 2005 and 2 to 4 October 2006	Waste Rock Dump and Tailings Management of Feasibility Study
Hernan Andrade	KP	Project Leader		25 to 28 January, 2006	Environmental and Social Sections of Feasibility Study
Katia Guanira	KP	Mining Engineer		28 to 29 March 2007	Managed the preparation of the Environmental, Social and Closure Plan Sections of Feasibility Study.
Carlos Espinoza	KP	Hydrogeologist		30 to 31 August 2006	Hydrology Section of Feasibility Study

The scope of work for the feasibility study included the following:

- overall study compilation and coordination
- review of project geology
- review of sample QA/QC program
- audit of mineral resources model and estimate
- reserve estimate and mine planning

- specification and management of metallurgical test work program
- flowsheet development
- design of the process facilities
- design of mine roads and access road upgrade
- capital cost estimating
- development of project implementation plan and schedule
- operating cost estimates for mining, process and administration
- financial analysis
- geotechnical engineering to support mine, waste dump and site facilities design
- design of mine site infrastructure and water supply
- design of mine waste management facility
- design of tailings management facility
- acid mine drainage analysis

AMEC understands that this Technical Report will be used in support of scientific and technical disclosure on the Casposo Project in the Intrepid Mines Limited Annual Information Form dated March 30, 2007.

2.1 Terms of Reference

AMEC is not an associate or affiliate of Intrepid, or of any associated company, and is independent of Intrepid. AMEC's fee for this Technical Report is not dependent in whole or in part on any prior or future engagement or understanding resulting from the conclusions of this Report. This fee is in accordance with standard industry fees for work of this nature, and AMEC's previously provided estimate is based solely on the approximate time needed to assess the various data and reach the appropriate conclusions.

In preparing this report, AMEC relied on reports and maps, miscellaneous technical papers listed in the References section at the conclusion of this report and AMEC's experience on similar deposit types.

Technical Reports have previously been completed for the Project by Intrepid staff, most recently in March, 2007. The reports are on file on the Sedar website (www.sedar.com) and the three most recent are:

- McGuinty, W., 2007: An Updated Report of Exploration Activities for the Casposo Property, Department of Calingasta, San Juan Province, Argentina, prepared for Intrepid Mines Limited, March 29, 2007.
- McGuinty, W., and Puritch, E., 2006: An Updated Resource Estimate and Report of Exploration Activities for the Casposo Property, Department of Calingasta, San Juan Province, Argentina, prepared for Intrepid Mines Limited November 9, 2006).
- McGuinty, W., 2006: An Updated Report of Exploration Activities for the Casposo Property Department of Calingasta, San Juan Province, Argentina, prepared for Intrepid Minerals Corporation, February 20, 2006.

AMEC completed a feasibility study on the Casposo Project during March 2007, entitled:

- AMEC, 2007. Casposo Feasibility Study Report, Project No. 150164, prepared for Intrepid Mines Limited, March, 2007.

A portion of the background information and technical data for AMEC´s current review of the Project were obtained from the aforementioned reports and additional data were provided by Intrepid.

The Effective Date of this report is March 30, 2007.

All measurement units used in this report are metric. Currency is expressed in US dollars unless stated otherwise. The currency used in Argentina is the Argentine Peso ($). The exchange rate on March 30, 2007 was US$1.00 equal to approximately $3.12.

3.0 RELIANCE ON OTHER EXPERTS

The AMEC QPs, authors of this Technical Report, state that they are qualified persons for those areas as identified in the "Certificate of Qualified Person" attached to this report. The authors have relied, and believe there is a reasonable basis for this reliance, upon the following reports, which provided information regarding mineral rights, surface rights, permitting, and environmental issues in sections of this Technical Report as noted below.

3.1 Mineral Tenure

AMEC QPs have not reviewed the mineral tenure, nor independently verified the legal status or ownership of the Project area or underlying property agreements. AMEC has relied upon Intrepid experts in Sections 4.3.2 and 4.3.3 of this report for this information through the following documents.

- Bosque, Hugo Arturo, 2006: Proyecto Casposo: Legal due diligence summary, Hugo Arturo Bosque to Goodman and Carr LLP, internal report prepared for Intrepid Mines Limited, November 13, 2006.
- Certificate of Title Document, 2006: Certificado de Titularidad: certified copy of certificate of title to the Kamila Mina, internal document supplied to Intrepid Mines Limited by the Corte de Justicia, San Juan, April 18, 2006.
- McGuinty, W., 2007: An Updated Report of Exploration Activities for the Casposo Property, Department of Calingasta, San Juan Province, Argentina, prepared for Intrepid Mines Limited, March 29, 2007.
- Emails and discussion meetings with Intrepid during the completion of the feasibility study.

3.2 Surface Rights, Access and Permitting

AMEC QPs have relied on information regarding Surface Rights, Road Access and Permits, including the status of the granting of surface rights by the Argentine Government for land designated for mining, milling, dumps and tailings impoundments and have relied on opinions and data supplied by Intrepid representatives for Sections 4.4 and 4.5 of this report as follows:

- R y J Lopez Aragon, 2006: Tax Review: Contadores Públicos review or Argentine Taxation regime, internal document prepared for Intrepid Minerals Ltd., December 2006.

- McGuinty, W., 2007: An Updated Report of Exploration Activities for the Casposo Property, Department of Calingasta, San Juan Province, Argentina, prepared for Intrepid Mines Limited, March 29, 2007.

3.3 Environmental and Socio-Economic

AMEC QPs have relied upon the environmental status and mine closure plan for the Properties documented in Sections 4.6, 4.7 and 18.5 of this report as prepared by opinions of experts retained by Intrepid, through the following documents:

- Knight Piésold, 2007: Sections 8, 9 and 10, Casposo Feasibility Study.
- Emails and discussion meetings with Intrepid and Knight Piésold personnel during the completion of the feasibility study.

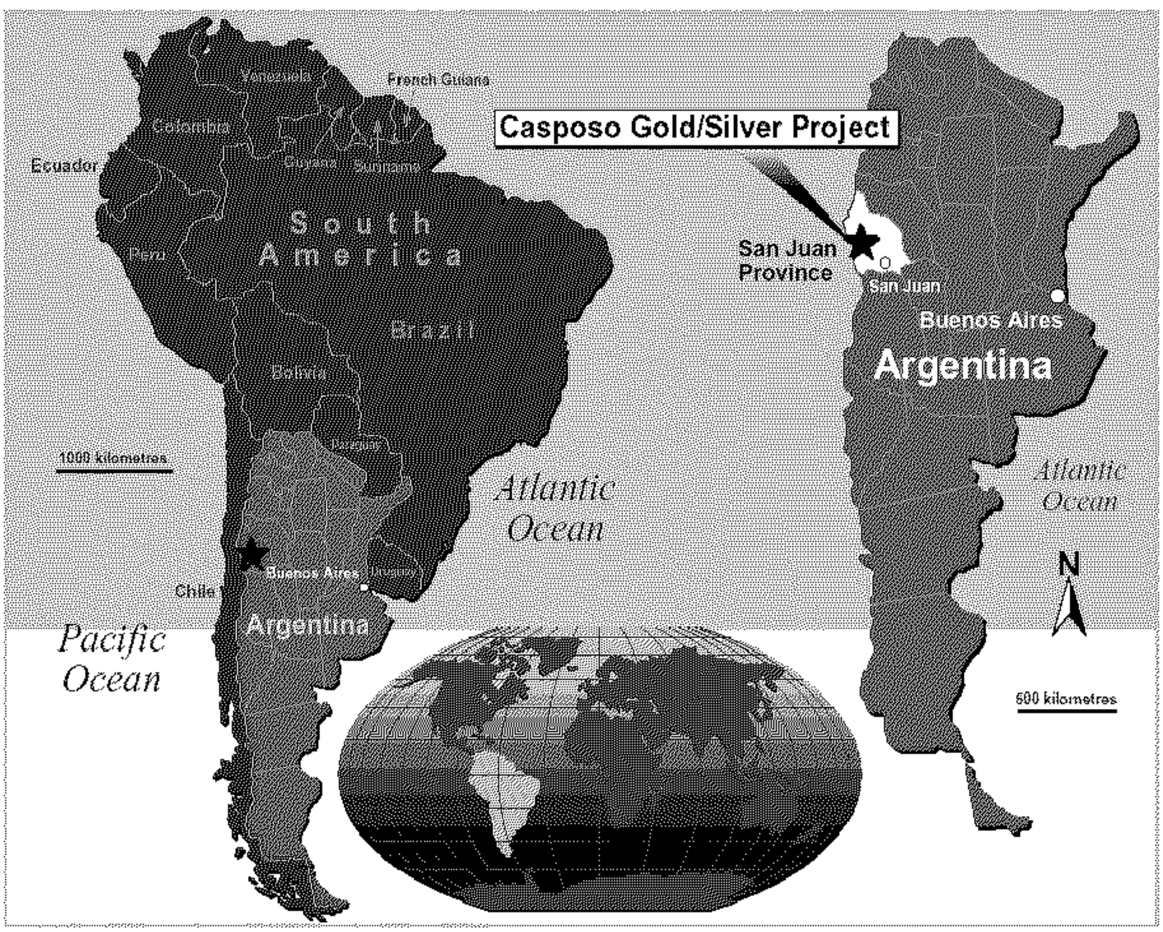
4.0 PROPERTY DESCRIPTION AND LOCATION

4.1 Location

The Casposo Project is situated about 150 km northwest of the city of San Juan, in the Department of Calingasta, San Juan Province, Argentina (Figure 4-1). The Property is at approximate latitude 31°12' S and longitude 69°36' W and centred on coordinates 6548000 north, 2438000 east (Gauss Kruger, Datum Campo Inchauspe 1969 Zone 2).

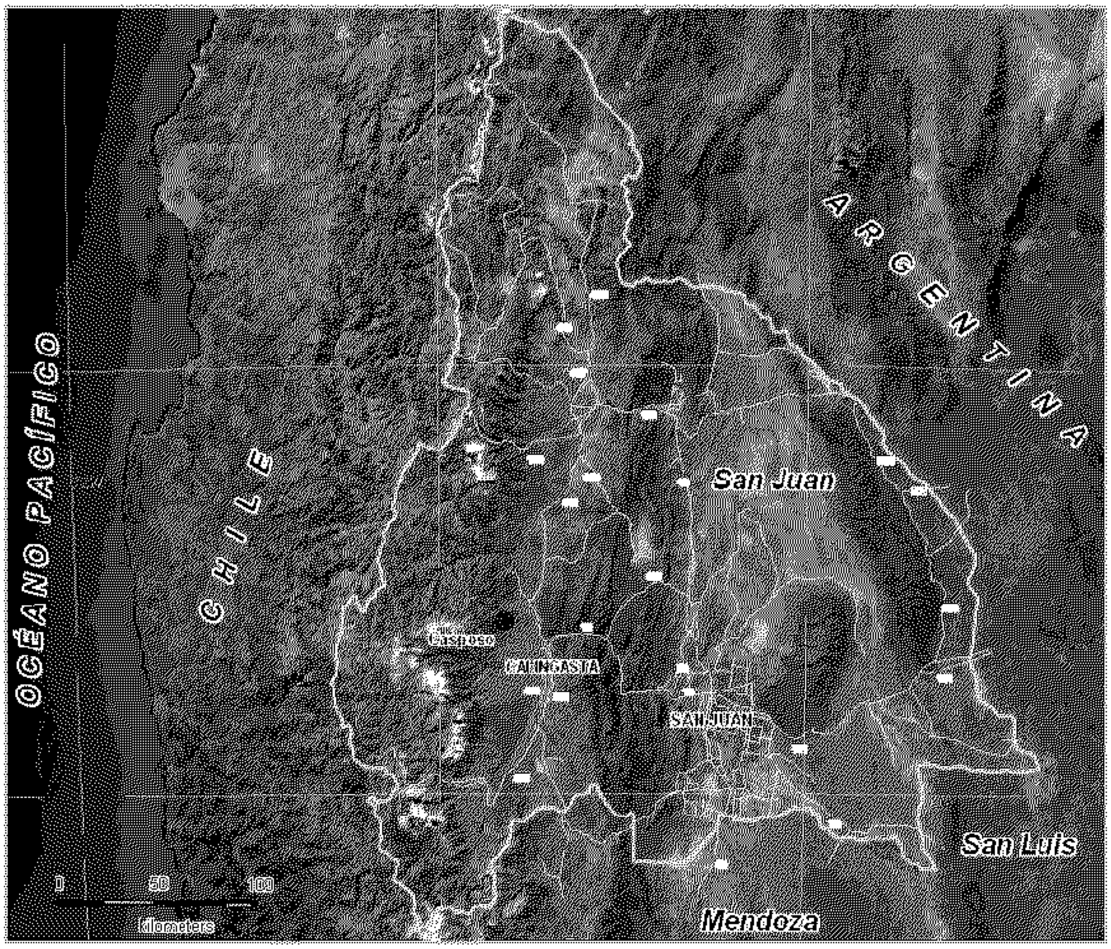
Mineralization defined to date is hosted in the Kamila and Mercado deposits, together referred to as the Kamila Zone (Figure 4-2).

Figure 4-1
Property Location Plan in Relation to Country and Province Boundaries



Note: Figure supplied by Intrepid Mines Limited.

Figure 4-2
Property Location Plan in Relation to Regional Centres



Note: Figure supplied by Intrepid Mines Limited.

4.2 Overview of Argentina

Argentina is the second-largest country in South America after Brazil and the eighth largest in the world. The population of the country is about 39.5 M, approximately just under half, or 16 M, live in the capital region of Buenos Aires.

Argentina can be separated into four major physiographic regions: the Andes Mountains in the west, the fertile lowland and subtropical rainforests in the north, the mix of humid and dry, flat expanses of the Pampas in the centre of the country, and the pastoral steppes and glacial zones of Patagonia in the extreme south. The principal watercourses are the Paraná, Uruguay and Paraguay rivers, which drain into the Río de la Plata Basin in northern Argentina. In the south, the Colorado and Negro rivers, which originate in the Andes, flow to the Atlantic Ocean.

Argentina's major cities include Buenos Aires, Cordoba, Rosario, and La Plata. The country has some 216,000 km of highways, 62,000 km of which are paved. Other transportation infrastructure includes about 38,000 km of railway line and 11,000 km of navigable waterways. Argentina's main ports and harbours include Bahía Blanca, Buenos Aires, Comodoro Rivadavia, Concepción del Uruguay, La Plata, Mar del Plata, Necochea, Río Gallegos, Rosario, Santa Fe, and Ushuaia. There are about 1,200 airports in Argentina, some 32 of which receive regular commercial flights. Ezeiza International Airport, located 34 km from Buenos Aires, serves many international airlines as well as the national airlines. The other international airports in Argentina are generally used for flights to neighbouring countries only.

4.2.1 Demographics and Government

The following summary is cited directly from the CIA Factbook (CIA, 2005).

Approximately 85% of Argentina's population is of European descent, with the remaining 15% comprised of mestizos (persons of mixed Spanish and indigenous background), indigenous, and other minorities. Spanish is the official language of Argentina, and is used for government and commerce. There are 17 indigenous languages, which are not widely encountered outside of the small towns and settlements.

The National Constitution affirms that Argentina is a federal country with three levels of state: federal, provincial, and local. The Constitution is based on the republican doctrine of the division of powers into three different areas: executive, legislative, and judicial. The federal state is ruled by a presidential system.

4.2.2 Business Investment Climate in Argentina

The following summary on Argentina's economy is cited directly from the CIA Factbook (CIA, 2005).

Argentina benefits from rich natural resources, a highly literate population, an export-oriented agricultural sector, and a diversified industrial base.

Mining

The value of Argentina's mineral production has increased significantly in the last decade, mainly due to the implementation of large scale mining. In 2003, the value of mineral production totalled US$1.1 billion, of which roughly two-thirds came from large-scale mining companies. The remainder was from the small- and medium-sized mining companies. About 95% of the production came from 12 provinces. The leading producing provinces were, in decreasing order of value, Catamarca (copper, gold, and lithium), Buenos Aires (crushed stone and limestone), and Santa Cruz (gold and silver). These three provinces represented 77% of Argentina's mineral production. About 70% of the production was exported.

4.2.3 Metal Mining in Argentina

Historically metal mining has not played a dominant role in Argentina's economy, but this situation has changed during the last five years. While industrial minerals and building materials accounted in the past for nearly two thirds of the total mining production, Argentina's gold production raised to 1.1 Moz of gold in 2003, becoming the 14th-largest world producer (fourth in Latin America, with 7% of the gold output of the region; Torres, 2004).

Argentina is one of three producers of primary aluminium in Latin America, accounting approximately for 12% of this total. The country was Latin America's third leading producer of lead (after Perú and Mexico) and steel (after Brazil and Mexico). It was the fourth leading producer of copper (after Chile, Perú, and Mexico) and primary iron and pig iron (after Brazil, Mexico, and Venezuela). Argentina was the fifth leading producer of silver.

Argentina and Chile signed an agreement in 1997 regulating the mining activities on shared border zones (Tratado de Integración y Complementación Minera Chileno-Argentina). Under this agreement, exploration and mining have dramatically increased (Torres, 2004). The most recent development in this sense corresponds to a protocol signed in January 2006 to facilitate the exploration of the Amos–Andrés gold-copper-molybdenum and the Vicuña gold projects, both located in border area between Central Chile and Argentina.

4.2.4 Mining Industry and Legislation

Information in this section is taken from Godoy (2007) and Torres (2004).

The Argentine Mining Code which dates back to 1886 is the legislation which deals with mining in the country. Special regimes exist for hydrocarbons and nuclear minerals. In the case of most minerals, the Mining Code dictates that the owner of the surface is not the owner of the mineral rights; these are held by the State. The State is also bound by the Code to grant to whoever discovers a new mine the rights to obtain a "mining concession".

Owners must comply with three conditions; payment of an annual fee, investment of a minimum amount of capital, and the carrying out of a reasonable level of exploitation. Failure to do so could lead to forfeiture of the property back to the State.

The administrative organization for mining-specific regulation is the Federal Ministry of Planning, Public Works and Investment which has a Mining Department headed by the Secretary of Mines. The Argentine Mining Law is a federally drafted law implemented through bi-lateral accords with the provinces who have jurisdiction over mineral rights. In recent years several provinces have made changes to the federal law as it applies in their jurisdictions in response to local initiatives. San Juan Province, where the Casposo Project is located has made very few changes to the current federal statute.

In 1980, an amendment recognized the need for modernizing the production and classification of minerals, size of individual mining areas (to encourage development of low grade deposits) and elimination of miners' rights National governments from exploiting mineral fields and to encourage foreign companies to engage in mining operations through public tenders.

In 1993, Argentina implemented a new Mining Investment Law (No 24,196), a Mining Reorganization Law (No. 24,224), a Mining Modernization Law (No 24,498), a Mining Federal Agreement (No. 24,228), and a Financing and Devolution of IVA Law (No 24,402). Amendments were also made to update the Mining Law (Decree 456/97). These amendments offered attractive economic incentives for exploration and mining to foreigners, and include both financial and tax guarantees. This group of laws also creates the basis for federal-provincial harmonization of mining rules such as import duty exemptions, unrestricted repatriation of capital and profits and a 3% cap on Provincial royalties.

In 2001, Law 25.429 "Update of the Mining Investment Law" was passed and in March 2004 approval was reached for a key provision of the Law allowing refund of the IVA (or value added tax) for exploration related expenses incurred by companies registered under the Mining Investment Law.

In 1995, Law N° 24.585 Environmental Protection (Mining Code) was passed and provides regulation for operations and environmental reporting at the exploration and exploitation levels.

In summary, the major changes to the mining code encompass:

- Exploration areas have been increased to a maximum of 100,000 ha per company and per province.
- Exclusive aerial prospecting areas of 20,000 km^2 are also permitted.
- A guarantee of tax stability for 30 years.

- Expenditures made in prospecting, exploring and construction of mining installations are tax deductible and value added taxes are recoverable.
- Imports of capital goods, equipment and raw material are exempt from import duties.
- Royalties will not exceed 3% of the ex-mine value of the extracted mineral.
- Environmental funds to correct damage are required and are deductible from income taxes; a National system of permanent mining environmental monitoring is set up. Implementation at the provincial level has been variable and in 2004-05 San Juan province began to increase staffing for monitoring purposes.
- Municipal taxes on mining were eliminated.
- Systemization and digital conversion of mining property registers has been implemented to varying degrees of success in each province and the definition by geographic co-ordinates now establishes mining rights.

In addition to the legal changes previously referred to the Argentine government has taken political steps to raise the profile of mining within the domestic industrial regime. A position of Secretary of Mines has been established, giving a seat at federal cabinet to the government's mining portfolio. This change was also made at the provincial level in San Juan Province in early 2004. The Secretariats are also commissioned to foster mining investment, participate in cooperation between international and inter-jurisdictional departments, and to oversee environmental, labour and hygiene issues related to mining. They respond to and govern initiatives of the National Mining Commission (which supervises the country's mining policy) and oversees the National Geological Service Board (SEGEMAR, which functions as a national Geological Survey).

4.2.5 Mineral Property Title

A *Cateo* (exploration right) is an area of land staked during the early stage of exploration. In Argentina, this is called the "Prospecting Stage". Cateos may be contiguous or separate and are subject to certain restrictions on size. A Cateo is sub-divided into 500 ha units with a defined exploration term determined by the cumulative number of units comprised. The maximum possible term is 1,100 days for the maximum lease size of 10,000 ha commencing from the grant date. Prior to its expiry, the holder of a Cateo may apply at any time for conversion to one or more 'Manifestación de Descubrimiento' (Application period for a Mining Lease) or 'Mina' (Mining Lease) rights within the perimeter of the Cateo up to its full area. Minas and Manifestaciones can also be established as the result of a discovery in open ground. A mining lease is subdivided into a minimum of two pertenencias, which are generally 6 ha for small deposits and 100 ha for larger, disseminated deposits.

To apply for a Manifestación or Mina, the applicant must present a representative sample of the outcrop as the discovery and indicate its co-ordinates and the surrounding area to be covered by the title. After about a 6-month period the Manifestación will be registered and convert to a 'Mina' or Mining Lease. Conversions and applications are administratively dependant and not date-dependant and are therefore not automatic. Processing times from one provincial jurisdiction to another may vary.

4.2.6 Royalties

Provincial Mining Royalty

According to the Mining Investment Law (Law No. 24.196), mining royalties in adherent Provinces (San Juan included) are capped at 3% of the mineral's mouth-of-mine value. Generally the Provinces determine mining royalty benefits in order to promote added-value activities, according to the following:

- Mining ore only: royalty of 3%.
- Milling: royalty between 1% and 2%. This category applies to companies producing dorè bars and concentrates.
- Final elaboration: between 0% and 1%. This royalty rate range applies to companies making final products such as jewellery.

A 1% royalty rate should be applied to Intrepid since dorè bars will be produced and, although this issue is subject to negotiation with the Provincial Mining Bureau, it is understood that a 2% royalty rate should be applied to mining companies producing concentrates.

Production Royalty

Additional royalty payments are required to be made during production to other independent parties by Intrepid as a result of land purchase agreements (4.3.2).

4.2.7 Taxes and Duties

This section summarizes the current tax regulations applicable to the Project in Argentina, including those covered in the Mining Investment Law (Ley de Inversiones Mineras), and any special benefits given to Intrepid for being registered in the Law. The relevant laws and number of rules and resolutions are numerous.

There are five taxes at the federal level that will be applied to the Project: the corporate income tax, the value added tax (IVA using the Spanish acronym), the export tax, the debits and credits tax, and the "Bienes Personales" (Personal Assets) tax. The Mining Investments Law exempts mining companies from paying two additional duties/taxes: import duty and assets tax. The various taxes and duties are described below.

At the provincial level an additional gross income tax is levied on gross sales at a rate of 3%, payable monthly. Gross sales of the mine are taken as total sales less cost of sales.

Income Tax

According to R y J López Aragón, an Argentine taxation consultant retained by Intrepid to complete a tax review, the general Argentine federal tax rate applicable to the Project will be 35% (R y J López Aragón *in* AMEC, 2007).

Value Added Tax (IVA using the Spanish Acronym)

The general IVA rate in Argentina is 21%, although other rates are also applicable for certain services such as telecommunications, electricity, water and gas (27%), and the purchase or definitive import of certain capital goods (10.5%).

In Argentina, mining companies that export 100% of their production are entitled to the following:

Exploration IVA Recovery:

Mining companies can ask for the reimbursement of the IVA paid as a result of the purchase of goods and hiring of services applied to mining exploration. IVA fiscal credits can be asked for after 12 mo of their inclusion in the tax affidavits. This stage of IVA recovery finishes with the completion of the feasibility study.

Early IVA Recovery or IVA Financing:

Mining companies are eligible to recover within 60 days the IVA paid as a result of the purchase and/or imports of equipment and infrastructure. Alternatively, mining companies can finance the payment of IVA derived from the purchase and/or imports of equipment and infrastructure with a local financial institution (up to 12% interest is paid by the Argentine government). These benefits, which cannot be combined, apply during the construction period.

IVA Reimbursement to Exporters:

Exporters can recover almost immediately the IVA they pay for the purchase of goods and services applied to the export activity.

Export Tax

In Argentina, exports of dorè bars are subject to a 5% Export Tax.

Debits and Credit Tax

This tax applies to bank transactions, and the rate is 0.6% for debits and 0.6% for credits. Up to 34% of the tax paid can be computed as a tax credit in the income tax and can be carried forward.

There are some exemptions such as:

- transfers between current accounts of the same company;
- credits in current accounts originated in bank loans regulated by Law No. 21.526;
- credits in current accounts as a result of exports.

The Debits and Credits tax was provisional and, according to Law No. 25.988, was applicable only up to December 31, 2005. However, this tax has since been prorogued and applies to the project.

"Bienes Personales" (Personal Assets) Tax

This tax is applied to the shareholders' equity of Intrepid at a 0.5% rate and is payable by Intrepid's shareholders. However, if shareholders are non-residents in Argentina, Intrepid has the obligation to pay the corresponding tax on behalf of the non-resident shareholder.

Import Duty

Article No. 21 of the Mining Investments Law grants the exemption of import duties, including the "Tasa de Estadística", for the import of capital goods, special equipment, spare parts, and certain supplies included in a list prepared by the Federal Mining Secretariat.

Assets Tax

Mining companies are exempt from paying this tax based on Article 17 of the Mining Investment Law.

Municipal Tax

Town council do not charge taxes, they charge rates, called municipals, to services (cleaning, street lighting, public roads maintenance and inspection laws).

4.2.8 Surface and Private Property Rights

Access over surface property rights in Argentina is obtained through the Ministry of Mines, who are required to communicate with the surface owners and ensure that they cooperate with the activities of the exploration/mining companies. Notice can be difficult due to delayed filing of personal property title changes and registry as well as limited staffing and mobility of the relevant authorities.

Private property rights are secure rights in Argentina, and the likelihood of expropriation is considered low. The Argentine legal and constitutional system grants mining properties all the guarantees conferred on property rights, which are absolute, exclusive and perpetual. Mining property may be freely transferred and purchased by foreign companies.

4.2.9 Environmental Regulations

Under Argentine Mining Law, the State Mining Secretary (SMS) of the San Juan Province manages the environmental approval system for new mining projects. The applicable evaluation process of the Environmental Impact Assessment (EIA) is defined by the Escribanía de Minas (EDM) according to the size of the mining project.

The new Decree of the Provincial Law 1679 SMS, dated October 2006, states that for small and medium mining projects in San Juan Province, the EIA must be presented together with the Feasibility Study. This allows the SMS to determine the size of the deposit in order to set up the members of the Evaluation Commission, as well as the corresponding terms of reference.

The environmental approval process is summarized as follows:

- The applicant completes the Environmental Baseline Studies and EIA and this is presented to the SMS. The SMS will forward this to the EDM within 24 h to determine if the project is classified as a small (<8,000 t/d) or medium scale (>8,000 t/d <25,000 t/d) project.

- The Mine Environmental Authority (AAM) will deliver the EIA to the Comisión Evaluadora Multidisciplinaria Ambiental Minera (CEMAM) and the Provincial Mining Consulting Council.
- The CENAM will consist of several institutions and within no more than 5 working days, will designate a representative and assistant that will evaluate the EIA. Within 30 working days for small projects, and 45 for medium projects, CENAM will issue a technical assessment on the acceptance, or rejection of the project's EIA. After this period the SEM will request the applicant to respond to any comments and/or questions about the Project.
- The CENAM may request presentations or expositions of the EIA, as well as meetings with the representatives of the applicant and their consultants, inspection visits, or demand opinions to professionals and other organizations.
- The applicant has a 30-day period to submit their replies to CENAM's comments and questions.
- The CEMAM will then take a further 30-day period to review the documentation and announce its decision on the approval for the EIA.
- The applicant holds community meetings according to the previous Community Participation Plan.
- The AAM will continue with a consulting process issuing an announcement in the Provincial Bulletin over a 3 working day period and allowing anyone interested to have access to the EIA (for 5 working days for small projects and 10 working days for medium projects).
- Comments or objections should be presented in writing to the Secretary of Mines within 3 working days for small projects and 5 working days for medium projects, counted from the day after public consultation's due date. The EIA will incorporate community inputs.
- Once the EIA is approved, proceedings towards granting of the Environmental License starts. It is estimated that another 30 day period is needed to prepare and grant the Environmental certificate, which is valid for 2 years.

After obtaining the EIA, the applicant must apply and obtain various permits and authorizations from the Province of San Juan to proceed with Project development. The permits and authorizations demonstrate compliance with current legislation for the construction and operation of mining operations.

4.3 Project Tenure

4.3.1 Company Registration

Intrepid's Argentine branch office, Intrepid Minerals Corporation Sucursal (IMLS) is duly registered under Argentine law having registration number 449 as of January 29, 2004. IMLS applied for its first refund in 2005.

4.3.2 Tenure History

In 2000, Battle Mountain Gold Corporation (BMG) was issued the Kamila Mina (Mining Lease) #520-0438-M-98. This mining lease covers the former Casposo Cateo1 546094-B-94 (Prospecting License) and the subsequent Manifestación de Descubrimiento (Declaration of Discovery) #520-0438-M-98. The conditions of the Mining Lease are defined under Ministerial Resolution No. 52-DM-2000, a copy of which can be found in the offices of the Gobierno de la Provincia de San Juan, Secretaria de Minería, Mesa de Entradas y Salidas.

The Kamila Mina Lease was subsequently held by Newmont Mining Corporation after the merging of BMG and Newmont on January 10, 2001. Ownership of the Kamila Mining Lease was assigned to Newmont Mining on October 17, 2001 and then by private agreement to a former employee, Mr. Eduardo Antonio Machuca on May 14, 2002.

On May 17, 2002, the Ministry registered the transfer of ownership to Eduardo Antonio Machuca, who at that time held clear and valid title to the Kamila mining lease. However, by November 25, 2002, the Kamila mining lease was registered to three partners: Eduardo Antonio Machuca, Hugo Arturo Bosque and Luis Alfonso Vega.

On May 22, 2002 IML signed a "Letter of Intent" with two of the vendors (Eduardo Machuca and Hugo Bosque) owning the Kamila Lease. On July 1, 2002 IML signed a "Rental Agreement with Option to Purchase" (the Agreement) with the three owners of the Kamila Lease, Eduardo Antonio Machuca, Hugo Arturo Bosque and Luis Alfonso Vega for a 100% interest in the "Kamila Mine Property" subject to "Option Payments" totalling US$300,000 over two years (US$50,000 payable on signing) and to a "Reserve Royalty" of US$1/oz of gold equivalent (up to a maximum of US$450,000).

The royalty is based on mineable and recoverable reserves defined by a feasibility study and was to be paid on or before July 1, 2005. The Agreement was then re-structured so that beginning in 2006, annual payments of US$150,000 were scheduled to be paid by Intrepid to the vendors each July, until the royalty total of US$450,000 is paid, or the property attains commercial production. A payment of US$150,000 was made in June 2006, a further US$150,000 is due in July 2007. On production, a "Production Royalty" of US$6/oz of gold equivalent will be paid to the vendors, net of any advanced royalties. Gold equivalent ounces mined surpassing 450,000 oz will be subject to reduced royalty payments of US$5/oz of gold equivalent. As of December 31, 2006 Intrepid reports that they have completed all its prescribed cash commitments.

In addition to the Option and Royalty payments, Intrepid committed to a total of US$600,000 in exploration expenditures over the first two years of the Option period on the property, of which a minimum of US$300,000 would be spent in each year. Intrepid met this first year minimum investment obligation by spending in excess of US$493,000 by June 30, 2003. By June 30, 2004 Intrepid had completed expenditures of US$838,133, exceeding the US$600,000 requirement.

The Agreement is also subject to a 5 km "Area of Influence" surrounding the Kamila mine, so that any new land within this area would be subject to the same terms as those set out in the Agreement. Since 2002, Intrepid Minerals has been applying for contiguous land areas as Cateos to cover prospective ground adjacent to the Kamila Lease.

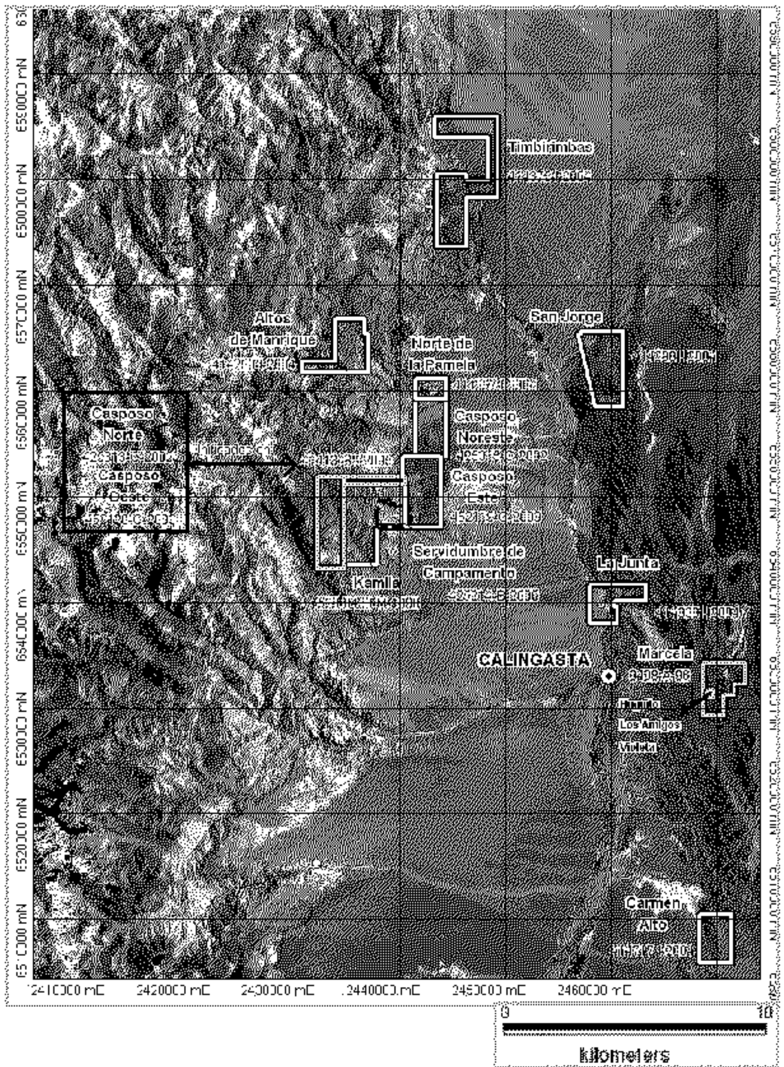
On February 8, 2006, the three original vendors (Eduardo Machuca Woodbridge, Hugo Arturo Bosque and Luis Alfonso Vega) transferred the title to the Kamila Mine property to Intrepid Minerals Corporation Sucursal. On July 4, 2006, Intrepid Mines Limited announced the completion of the merger between Intrepid Minerals Corporation and NuStar Mining Corporation Limited. As a result of the merger, Intrepid Minerals Corporation became a wholly-owned subsidiary of Intrepid Mines Limited. Intrepid Minerals Corporation Sucursal remains a branch office of Intrepid Minerals Corporation, and the holder of the Casposo property mineral titles.

4.3.3 Tenure Details

The Casposo Project covers an area of 35 km^2, and comprises a Mining Lease, five exploration Cateos (Exploration Concessions) and one Manifestación (application stage for a Mining Lease). The current tenure is shown in Figure 4-3 and summarised in Table 4-1.

Although still active, Cateos 313 and 120 reported by Intrepid in 2006 (McGuinty, 2006 and McGuinty and Puritch, 2006) have been covered by Manifestación 414-1348-I-05, named Julieta. Until granted, there are no expiry dates for Cateos. The two granted Cateos have been superseded by this Manifestación, which is awaiting survey prior to conversion to a Mining Lease.

Figure 4-3
Project Tenure



Note: Figure supplied by Intrepid Mines Limited. The figure is orientated with north to the top of the plan. Tenements Carmen Alto, Marcela, La Junta, San Jorge, Altos de la Manrique and Timbirimbas are not discussed in this report.

Table 4-1
Project Tenure Summary

Property Name	Property Type	Date Initially Granted	Name	Area (km2)	Province	Notes
Casposo	Mine Lease 530-0438-M-98	May 30, 2000	Kamila	35.00	San Juan	Title received and transferred to Intrepid
Property Name	**Property Type**	**Date of Last Title Activity**	**Name**	**Area (km2)**	**Province**	**Notes**
Casposo	Exploration Cateo 425 313 C 2002	August 15, 2003	N/A	397.80	San Juan	Published
Casposo	Exploration Cateo 425 120 C 2003	June 20, 2003	N/A	2211.20	San Juan	Charted pending publication
Casposo	Exploration Cateo 425 119 C 2003	June 20, 2003	N/A	2326.13	San Juan	Charted pending publication
Casposo	Exploration Cateo 425 315 C 2002	August 15, 2003	N/A	1591.58	San Juan	Published
Casposo	Mina 414 1348 I 2005	December 18, 2006	Julieta	2624.88	San Juan	Survey order requested and legal field report filed

4.4 Project Surface Rights

Intrepid conducted a complete study of current surface ownership related to the Mina and Cateos, which comprise the Casposo Project in 2003–2004. Surface rights in Argentina are not associated with title to either a mining lease or a claim and must be negotiated with the landowner(s).

In 2004, Intrepid negotiated with a group of property holders who held non-subdivided (condominium) interests for the surface rights over the Project area. As at December 31, 2004 Intrepid had secured 92% of the condominium rights to the property. There has been no change in the percentage of condominium rights held since that date. Intrepid hold sufficient surface rights to allow the planned mine development to go ahead.

In order to access the northern portion of the Kamila mine a new access road was constructed via Quebrada La Puerta in 2005. A new Right of Way was requested for this road in October 2005 and approval of the application is still pending. The Right of Way request has the following reference: 414.1349-V-05 Dpto. Calingasta - Pampa de la Puerta - DM No. 6 Servidumbre de Camino. The access request is not required to be finalized for project construction. The equity and the currently held right-of-way permit secure future access for exploration and development purposes.

The property purchase has been approved by the Border Guard, the agency which reviews foreign ownership along national border areas in Argentina.

4.4.1 Camp Easement

In 2000, a request to establish a camp easement was filed with the San Juan government. This is a standard step in the title process so as to identify the location of operations. The area submitted was preliminary, as economic deposits and infrastructure sites had not been delineated. In late 2004, an amended Mining Camp easement plan was submitted to more correctly identify the area of potential operations at Kamila. To date, this right has not been granted; however there is a reasonable expectation that Intrepid will be granted the easement rights.

4.4.2 Capital Investment Report

A Capital Investment Report was completed by Intrepid during 2000 to 2001, which outlined the development plan and expenditures to be completed on the property over a 5-year period.

The minimum required capital investment under law is 300 times the mining canon (fee) set at $800 per pertenencia (refer to Subsection 4.2.6). The minimum investment required on the Casposo property is therefore 4.8 M Argentine pesos (at current exchange rates, about US$1.55 M). Payment of the annual canon by the titleholder must commence within three years of the date of inscription of the mining lease. Payments of $28,000 are due to be paid by Intrepid annually, on the August 31 anniversary date.

As of December 31, 2006 Intrepid reports that all mining canons have been paid.

4.4.3 Surveys

On June 17, 2003, a survey edict for the Kamila mining lease was published. This edict provided the guidelines for legal survey of the Mina. The submission of the completed survey was the final step in the lease title process. Upon filing and publication of the legal survey, the property has full and transferable exploitation rights subject to permitting. A standard mine survey requires a 2 m-high, steel post monument with a cement base to be constructed at the corners of each pertenencia. These pertenencias are laid out in a best-fit pattern related to the Mina boundaries. The Kamila Mina comprises 35 pertenencias and would have required 56 surveyed monuments to be in accordance with the mining law. In 2004, Intrepid applied to modify the survey by reducing the number of internal cement markers by only surveying the perimeter and the area of the delineated mineral deposit. The modified survey was accepted by the San Juan government and the survey was completed in November 2004 and inspected in December 2004.

On December 19, 2005 Council of the mining department of San Juan approved the final resolution of ownership for Casposo.

4.4.4 Permits

Intrepid will require various permits and authorizations from the Province of San Juan to proceed with the development of the Casposo Mine Project. The permits and authorizations demonstrate compliance with current legislation for the construction and operation of the mine.

A project permit is required in order to obtain all of the other permits and authorizations. This permit is referred to as the "Environmental Impact Declaration (EID) Resolution" and is granted by the Subsecretaría de Estado de Minería (Undersecretary of Mining) in accordance with the requirements of National Law 25.485, Environmental Protection for Mining Activity. The requirements are incorporated in the Mining Code as Title XIII, Section 2, to which the Province of San Juan adheres through Provincial Decree 1426/96.

The following subsections describe the other main permits that Intrepid must apply for and be granted subsequent to obtaining the EIA (see Section 4.2.10).

Use of Surface and Groundwater Resources

These rights are granted by the Undersecretary of Water Resources through its Hydraulics Department, and must adhere to the following criteria included in the Water Code:

- demonstrate that the requested resource exists, which in the case of groundwater means demonstrating its discovery and flow,
- demonstrate that the granting of rights does not damage or reduce the rights of others.

To grant water rights, the Hydraulics Department has added another requirement: prove that the use of the requested resource will not adversely affect the respective aquifer.

Discharge of Liquid Effluents

Different authorities are responsible for approving this application, depending on the type of system that will collect the industrial and/or domestic effluent and the quality of the effluent to be discharged into the ground or waterways.

- Authorization for a Private Sewer System: Public Health Secretariat Law 2.553 (Sanitary Code).
- Certificate of Discharge Feasibility: Provincial Law 5.824 Regulation Decree 683.
- Certificate of Discharge Authorization: Provincial Law 5.824 Regulation Decree 683.

Cultural and Natural Patrimony

This permit is required by the following three laws, which are designed to protect archaeological and paleontological resources as a part of the nation's cultural heritage:

- Provincial Law 6.801
- Regulatory Decree 1134/01
- National Law 25.743, Regulatory Decree 1.022/2004

Permits are authorized by the Human Development Ministry of the Province of San Juan, represented by the Undersecretary of Culture of the Province of San Juan. As part of the application, the following documents must be submitted:

- A report on "Property that is Part of the Natural and Cultural Patrimony".
- Filing of "Fortuitous Finding of Properties" documentation if any items identified on the property may significant to the Cultural and Natural Heritage of the Province.
- Request for "Releasing the Area".

Disposal of Solid Non-Dangerous Industrial and Domestic Waste

This permit is related to Provincial Law 7.375, as modified by Provincial Law 7.396, on Solid and Urban Waste, Controlled Disposal Sites and provides the legal requirements for the control of industrial and domestic waste, along with other disposal requirements. In terms of controlled dumps or landfills, the Law establishes that siting and construction must take place in an appropriate and approved site, in compliance with municipal regulations and must meet the requirements established by Provincial Law 6.571 of the Environmental Impact Assessment, modified by Provincial Law 6.800.

Transport and Disposal of Hazardous Waste Permit

This permit applies to the transport and disposal of materials such as oil and grease.

Storage of Chemical and Toxic Substances

Decree 1095/96, modified by Decree 1161/2000, establishes that persons or entities, and in general all those involved in the producing, making, preparing, elaborating, repackaging, distributing, commercializing, retailing, wholesaling, storing, importing, exporting, transporting, transferring, and/or performing any other national or international transaction involving substances included in Lists I, II and III of Annex I of the Decree must be registerd with and inscribed in the special registry kept by the Registro Nacional De Precursores Químicos (National Registry of Chemical Substances) before beginning any operation (SEDRONAR).

Fuel Storage

National Law 13.660 and its Regulatory Decree require that the construction, enlargement or modification of fuel deposits (liquids, gases or mineral solids) must be authorized by the Executive Branch, and in conformance with the law. The Law also stipulates registration with and inscription in the Registro de Bocas de Expendio de Combustibles Líquidos, Consumo Propio according to Resolución Nacional 404/94 of the Secretaría de Energía (Registry of Liquid Fuel Outlets for Own Use according to National Resolution 404/94 of the Secretary of Energy). The designated authority for enforcement of this regulation and the resolutions cited is the Secretaría de Energía de la Nación (National Secretary of Energy), which reports to the Ministerio de Economía y Obras y Servicios Públicos (Ministry of Economy and Works and Public Services).

Authorization for the Installation of a Food Factory and Dining Room Permit, Permit for Water Treatment Plant, Potable Water Quality Certificate, Authorization for Private Potable Water System

Based on the provisions of Article 2 of Provincial Law 2.553, known as the Sanitary Code, the Province has the responsibility to safeguard health throughout its territory in accordance with national and municipal requirements, in order to ensure the adequate health level of its inhabitants.

This requirement is satisfied by health protection, promotion, and restoration activities. These include activities related to environmental health; promotion and control of drinking water; disposal of sewage; control of insects, rodents, garbage, and animals; control of foodstuffs; and the sanitary improvement of dwellings, industries, and premises, among others.

According to Provincial Law 3.705-72 and Resolution 1112-MEyAS-94, is administered by the Código Alimentario Argentino (Argentine Food Code) in the Province of San Juan is the Secretaría de Salud Pública (Public Health Secretary), through its División Bromatología (Bromatology Division), through the authority delegated by national health authorities.

Boiler Systems National Law 19.587 (Sanitary Code)

According to Article 18 of the Sanitary Code, the installation and functioning of boilers in private buildings must be controlled by the sanitary authorities. Such facilities must fulfill the requirements of the Ley de Higiene y Seguridad (Hygiene and Safety Law), National Law 19.587.

Camps and Civil Constructions

For the construction of camps and general civil construction the applicant must obtain a Building Permit from the Dirección de Planeamiento y Desarrollo Urbano (DPDU) de la Provincia de San Juan (Planning and Urban Development Department of the Province of San Juan).

The Feasibility Certificate or building site location, is the approved location of the Project granted by the governing provincial authority (the DPDU), who administer the Código de Edificación de la Provincia (Provincial Building Code) and are responsible for the control of all construction activity in the Province.

Awarding of this permit has eight required steps:

- Feasibility Certificate of building location.
- Approval of building plans suitable for construction.
- Certificado Final de Obra Resolución 5580 [Código de Edificación de San Juan], Dirección de Planeamiento y Desarrollo Urbano (Final Building Certificate Resolution 5580 [San Juan Building Code], Planning and Human Development Department).
- Certificado de Normas de Higiene y Seguridad en el Trabajo Ley Nacional 19587, DPDU (Certificate of Work Hygiene and Safety National Law 19.587, Planning and Urban Development Department).
- Fire Department Certificate, National Law 19.588, Dirección de Bomberos (Fire Department).
- Habilitación de Obras Civiles de Construcción [Tasa Municipal] de la Municipalidad de Calingasta (Building Permit [Municipal Tax] from the Municipality of Calingasta).
- Indoor Electric Installation Permit from the Municipality of Calingasta.
- Potable Water Quality Certificate: Provincial Law 2.553.

Explosives

Everything related to "The acquisition, use, possession, bearing, transmission under any title, transport, introduction into the country, and importation of firearms and hand thrown weapons or any type of device, chemical aggressor, and other material classified as weapons of war, gunpowder and similar explosives, and civilian arms, ammunition, and other materials are subject to the limitations of this law throughout Argentina" (Article 1, National Law 20.429) for which the National Arms Registry is the designated authority.

Tailings Dam or Tailing Storage Facilities

Article 2 of Provincial Decree 1544/02, Department of Energy Resources applies to hydroelectric applications, dams, dykes and auxiliary works of the Provincial State, be they in operation, design stage, construction stage and/or out of service. Also under its jurisdiction are all private dams and dykes and others. As well, any embankment or structure for any purpose that involves or could involve retaining any fluid will be subject to review by the Application Authority if it considers it subject to article specifications.

The decree names as the Application Authority the Dirección de Recursos Energéticos (Department of Energy Resources) a sub-department of the Subsecretaría de Recursos Hídricos y Energéticos (Undersecretary of Water and Energy Resources) which is administered by the Ministerio de Infraestructura, Tecnología y Medio Ambiente (Ministry of Infrastructure, Technology and Environment) of the Government of San Juan.

Operación Minera Ley Provincial 6.531 y Reglamento Policía Minera 40/48 (Mining Operations Provincial Law 6.531 and Mining Police Rules 40/48

The Provincial Mining Authority keeps a registry of mining producers, merchants and industrialists, in which all categories of mineral producers, mineral merchants and mining industrialists that perform milling, concentration, calcination, cutting, smelting procedures or any other activity that involves mineral transformation within the province of San Juan, must register.

Registration must be made within three months of starting mining related activities and must include:

- Registration as a Mining Producer.
- Production sheet.
- Mineral Transit Permit.
- Communication of the beginning of the mining activity.

Water Concession for Industrial Mining Use and for Population Use Hydraulics Department

Title IV of the Código de Aguas (Water Code) (art. 110 to 116) legislates on mining use and determines that "the concession of waters, riverbeds and public beaches for mining use, are granted for the purposes of using the water in mining extraction activities; for the extraction of mineral resources from the water, or for secondary recuperation of oil or natural gas, as well as to allow industrial, loading and sales tasks in mining establishments."

The nature and volume of the water use must be clearly defined, according to Article 20 of the Water Code which states: "No concessionary or user may use public water for a purpose different to that for which it was authorized. He also cannot use a greater volume or proportion to that granted".

Registration in the National Registry of Dangerous Waste Generators

Registration with this organization is mandatory when there is dangerous waste that cannot be treated in the Province of San Juan, according to National Laws 24.051 and 25.612.

Telecommunications

National Law 19.798 establishes that telecommunications matters administered under the jurisdiction of the Commission Nacional de Comunicaciones CNC (National Communications Commission), under the Executive Branch. The Law states that the following authorizations must be obtained:

- Authorization for the Installation of Radioelectric Stations and the Installation, Use, Expansion, Modification and Moving of the Different Media or Communication Systems and Frequency Changes.
- Certification and Authorization of Communication Equipment.

Quarries of Construction Aggregates Mining Code: "Subsecretaría de Estado de Minería" (Undersecretary of Mining)

The Mining Code defines Third Category mines as those that involve the production of minerals of stony or earthy nature, and in general all those that serve as building materials and ornaments. This group consists of quarries, including borrows, belong exclusively to the surface land owner and a third party cannot exploit these without the surface land owners consent. A concession from provincial authorities is required if a quarry is located on federally owned land.

A quarry to extract materials frequently used in the construction of different mine facilities will occur in one of three situations:

- Quarry on Provincial Fiscal Land.
- Quarry on Third Party Private Land.
- Quarry on own Private Land.

Authorization for construction and operation of a sewage treatment plant

This authorization applies for onsite sewage treatment.

4.5 Environmental Issues

Prior to initiation of exploration work in Argentina, an environmental certificate must be obtained. BMG initiated the filing process for this report in 1999 and made various revisions as they conducted their work programs. In 2001, BMG filed a report and a ministerial inspection took place but final approval was not obtained as Newmont, subsequent to their take-over of BMG, dropped the exploration and mining rights to the property.

Intrepid has been allowed to continue the process for the Kamila mining lease. In March 2004, an inspection of the Casposo site was undertaken by the Ministry of Mines and in April final approval was obtained for the environmental study including updates for each work program undertaken by the Company. In order to maintain the mining lease in good standing, additional reports still must be filed detailing the work completed over the previous two years and describing upcoming work in subsequent exploration programs. Intrepid has maintained its environmental reporting of exploration programs up to date and receives regular inspections from the San Juan Department of Mines. The last inspection was on October 13, 2006.

In addition to environmental reporting, Intrepid has been applied for and been granted water use permits from the appropriate provincial water authority to service its exploration drilling programs. A projection of daily water usage is provided by Intrepid as well as the time frame in which the water will be drawn. The applicable fee for the water use is charged to Intrepid. These permits are requested prior to each drilling campaign, and a permit was current for drilling operations as at March 30, 2007.

The company also files annual exploration work reports at the Department of Mines at both provincial and federal levels of government.

Intrepid's Casposo exploration site was inspected by the San Juan Ministry of Mines in 2004 and in 2005 and found to comply with the applicable standards under the law.

Intrepid's updated environmental report of its exploration activities was also accepted in April 2004. A further update, filed December 2005, is pending approval, and may be approved in April, 2007.

4.5.1 Protected Natural Areas

Six protected Natural Areas lie outside the direct area of influence of the Project. Three of the protected areas should be noted because of their proximity to the existing access routes that connect the localities within Project's socioeconomic area of influence. These are the Don Carmelo Protected Natural Area, the Cerro Alcázar Protected Natural Area and el Leoncito National Park.

The Don Carmelo Protected Natural Area features geological formations that are about 3 km away from the access route to the Project. National Route 149 does not cross the Don Carmelo Protected Natural Area or the Cerro Alcázar Protected Natural Area.

The Leoncito National Park is traversed by a major access road, National Route 412, which may be used as an access route for some project supplies from the port of Valparaiso in Chile when the mine operation is underway.

4.5.2 Social and Cultural Heritage

Archaeological field surveys have been completed by Intrepid for the Project Access Road Area and the Project Development Area. To date, the surveying and protection of archaeological sites at the Project has been carried out under the supervision of a qualified archaeologist retained by Intrepid and authorized by the Secretary of Culture.

In the Mercado deposit area, an arrowhead was found; the area is postulated to have been occupied sporadically, possibly for hunting. Near La Cantera, historical small quarries, paths and embankments that are inferred to have been used to load and transport a type of red, pink and grey soft rock were noted. However, there was no material available near the quarries that could be used to date the usage period.

A paleontological survey was carried out to identify any bedrock outcrops within the Project's Direct Area of influence that may contain invertebrate, vertebrate or plant fossils of potential value. Volcanic rock such as rhyolitic porphyry and andesite was identified. No sedimentary rocks were encountered.

4.6 Socio-Economics

Politically the Project site lies within the Department of Calingasta, in the Province of San Juan. The population base is characterized as a dispersed rural population with concentration in certain localities, namely Villa Calingasta and Barreal. Based on a 2005 survey, approximately 75% of the 8,456 residents of the Department lived in these two towns. The remaining smaller population settlements are considered hamlets: Villa Nueva, Puchuzún, Villa Corral, La Capilla, La Isla, Hilario and Sorocayense.

The locality of Villa Calingasta and the hamlets of Villa Corral and Puchuzún are the closest communities to the Project site, (approximately 20 and 30 km east, respectively). As the Project develops these two population bases will be potentially see some socioeconomic impacts due to traffic, transportation, the use of public facilities, goods and services, and other factors.

During the development and operation of the Casposo Project an increase in labour demand and goods and services is expected, possibly leading to increased migration to the Calingasta area. It is estimated that about 60 to 80 unskilled and semi-skilled jobs generated during the mine operation will be filled primarily by local inhabitants. Additional unskilled work will be created by the mining contractor who will establish a camp in Calingasta. Skilled workers will likely be recruited in San Juan city and commute weekly to site. Construction requirements will peak at about 250 but while providing opportunities for local unskilled labour, this will be temporary.

The Department of Calingasta has a basic or limited economic structure. In general, there is a lack of employment and personal income is low. The main economic activity in the zone is agriculture.

Fruit orchards and forestry together utilize 77% of the cultivated surface area of the department. These agricultural activities create little formal employment and economic benefits from these are mostly restricted to owners and value added markets outside the area.

The irrigated portions of the Calingasta valley are suitable for forestry (mostly poplar) production, which supplies the local wood mills, a related industrial activity. These mills process wood to supply posts, planks and slats for packing crates, homebuilding products and/or furniture in the Tullum Valley carpentries. Bread products, spring water and water are also produced for the Department's internal consumption.

From an industrial point of view, the department of Calingasta can be considered to contain a small industry sector made up of small and medium-sized companies. Industrial production includes food and beverages, wood and manufacturing of wood and cork products. The most significant products include apple cider produced by the Champagne method (unique to the country) and the production of pressed apple cider by the Los Tambillos cooperative.

Commercial activities include group retail commerce, vehicle repair, and fuel sales. Among service establishments, those directed to leisure predominate, like restaurants, bars, canteens, and sporting clubs. Other establishments include hotels, lodgings and cabins, medical and dentistry service establishments, radio stations, telephone booths.

In Calingasta there are a total of 2,162 active employees. The sectors that produce most employment are: agriculture with 669 workers (31%); public administration, defence and obligatory social security with 308 jobs (14.2%); teaching with 226 (10.5%); domestic service 120 (5.6%); construction 122 (5.6%); mines and quarries 55 (2.5%). Other minor employment sectors account for the remainder.

Within the Department of Calingasta, 55% of the population (56% of males and 54% of females) do not belong to a social services plan and/or lack private or group health coverage. The development and operation of the Project is indirectly expected to increase demand for health services in Calingasta because the average increase in income of the population income should result in people being more able and willing to pay for better private health care services.

The Department of Calingasta has 26 educational institutions with a total of 2,837 students. Calingasta has 1.5% of total students in the province. Educational services include the following levels: initial, primary, secondary and non-university higher level studies. The illiteracy rate is 5%, which is higher than the provincial average.

Total households are distributed in 1,531 houses, 246 huts, 12 cabins, 13 in rooms or boarding houses and 1 in lodgings. Of the 1,531 housing units, 582 (35% of the population) have floors made of earth, loose brick or other material or they do not have internal running water and toilets with water discharge. The remaining 949 housing units (50% of the population) have floors made of material (ceramic tile, tiles, mosaics, marble, or wood), internal running water and toilets. The remainder of the population live in huts (13%) or boarding houses or lodgings (2%).

The Project is not expected to add significant demand on the existing community infrastructure, since the need for imported labour resources will not be significant. As a result significant infrastructure improvements are not foreseen and the main opportunity for the community is expected to be the creation of local employment.

5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Accessibility

The property is located in department of Calingasta, San Juan Province, Argentina approximately 150 km from, or a two hour drive west from, the city of San Juan, travelling on paved roads. The region can also be accessed from the city of Mendoza via a separate southern route (Figure 5-1).

Access to the Project north from the international airport in Mendoza follows National Route 40 via the city of San Juan to the town of Talacasto, then along Provincial Routes 436, 414 and 12 to the village of Calingasta (the population centre nearest to the Project), and finally along Provincial Route 412 to the main site access road.

Alternatively, the site can be reached from Mendoza by two other road combinations: by following National Route 40 to National Route 7 and continuing northwest to the town of Uspallata, then along Provincial Routes 39 and 412 via Calingasta to the main site access road; or from the city of San Juan to the town of Talacasto on National Route 40 then along Provincial Route 436 to Cerro Puntudo and south along Provincial Routes 425 and 412 to the main site access road. The southern route via Uspallata passes through El Leoncito National Park to the town of Barreal. Barreal serves as the base for the El Pachón Copper project, currently held by Xstrata.

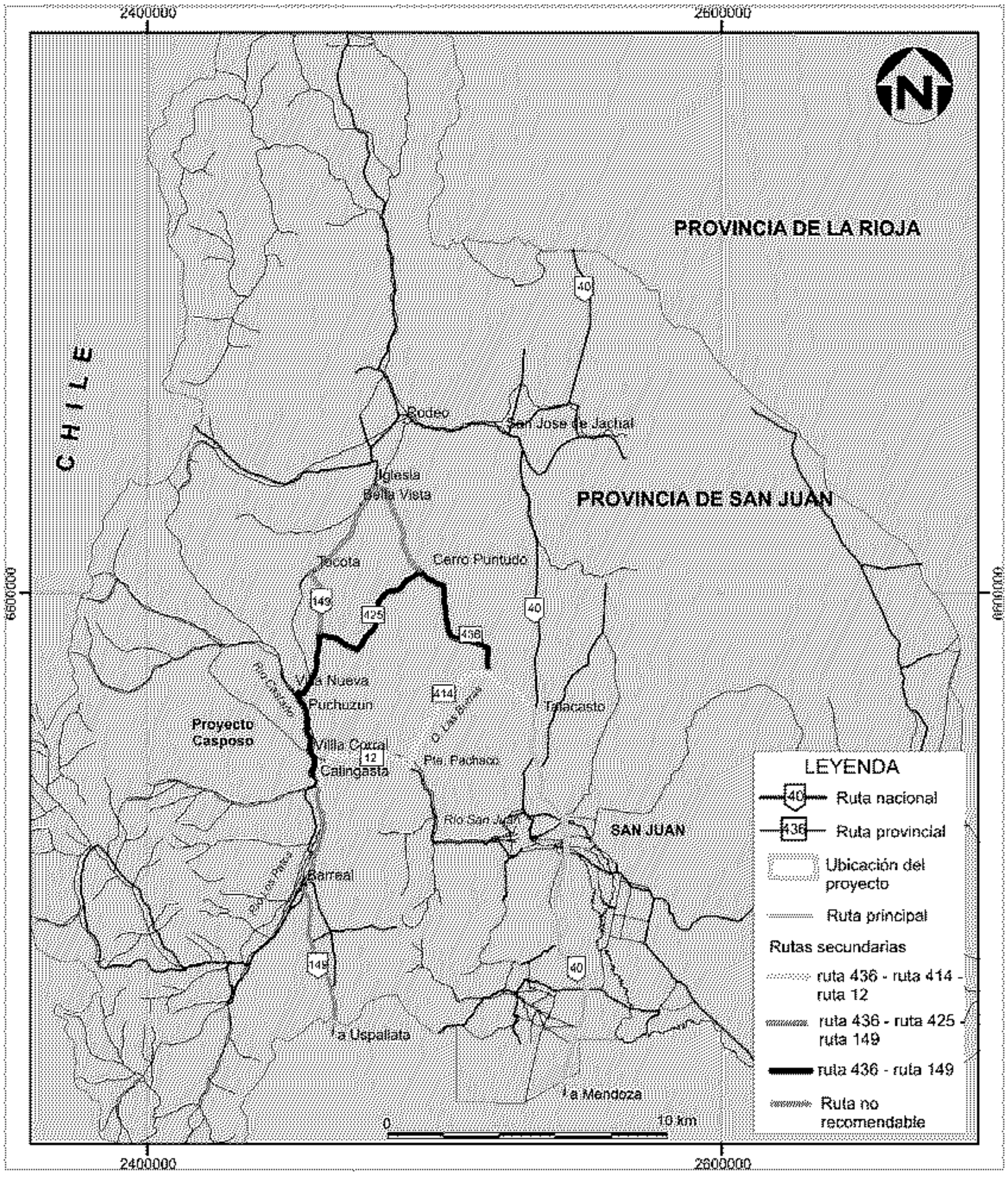
There is no rail or air access to the Project. The closest airport is in the city of San Juan which is suitable for 737-series jet services.

5.2 Climate

The climate in the Project Area is classified "desert dry", with a median annual rainfall of 75 mm and a temperature range accentuated by the altitude, both seasonally and daily. The median temperature is 14.5°C. Summers are generally warm (highs of 30°C) and winters dry and cold (lows of -2°C).

The area is generally arid with a short summer rainy season (January–March). Rains can be very strong and the lack of vegetative cover contributes to localized flash flooding. Net evaporation rates are high, and exceed annual rainfall by a significant margin. Of the total annual rainfall, 80% occurs between October to March. Rainfall in the high mountains is common during other months of the year as well.

Figure 5-1
Property Access



Note: Figure from Intrepid Mines Limited.

During the winter months (June to September), snow falls at the site, occasionally with up to several centimetres accumulating. Even so, snowfall melts almost immediately if exposed to a full day of sun.

The area can be very windy during the whole year. The area is subject to short-lived, yet very strong, easterly winds locally referred to as "Zonda" Winds. This phenomenon brings dry winds of up to 100 km/h and can cause severe drops in humidity.

5.3 Local Resources

San Juan city is a major centre with full hospital service and education to university level with two universities. The Universidad Nacional de San Juan has a century-old mining engineering and geology faculty as well as diverse science and humanities programs and a medical school.

The nearest town to the Project is Calingasta, a village of approximately 2,000 people, located approximately 20 km to the southeast. The nearest settlement is at the hamlet of Villa Corral, about 20 km to the east.

During the development and operation of the Casposo Project an increase in labour demand and goods and services is expected for the local communities, possibly leading to some migration to the Calingasta area. It is estimated that about half the mine operations manpower generated (103 jobs) will be filled primarily by local inhabitants. It is assumed that all of the low skilled labour (40) and half the medium-skilled (24) workforce will be recruited from the Calingasta area and other nearby towns and environs. The rest of the medium-skilled workforce (24) and on-mine management and supervisory staff (15) will be drawn from San Juan and it has been assumed that they will continue to reside there. It is planned that these employees will obtain unaccompanied status board and lodging during the week in the Calingasta area.

The planned Intrepid mine supply chain for most materials is from the city of San Juan, the local hub for supplies to the Barrick Gold Corp. - owned Veladero Mine and Pascua-Lama Project. San Juan is approximately 150 km from Casposo. Many major equipment providers and several international engineering firms have warehouses and offices in the city and a number of new mine service and supply businesses have been opened. Major items such as fuel and reagents are planned to be sourced from San Juan or from Buenos Aires and hauled by transport contractors.

5.4 Infrastructure

5.4.1 Local Infrastructure

Paved access to the Project ends at Villa Corral (Figure 5-1). In 1998, an easement was requested by the previous property holder for a 22-km road access to the Kamila zone from Villa Corral. After the easement was granted by the Survey Department of the San Juan government, gravel road access to the Kamila Zone was constructed in 1999.

A power line carrying hydro-generated electricity passes within 15 km of the property. However, this has limited capacity and power sources evaluated for the feasibility study determined that power for the Project will be generated on site using diesel generators.

It has been estimated that the power requirements of the project will need five power generators. The first phase of mining at the open pit will require the use of three 900 rpm generating sets with a site rating of 1,000 kW each at constant load and an extra standby generator of the same characteristics. For the planned underground phase an additional generator will be required to supply the power requirements of the Kamila underground mine.

Fresh water will be obtained from a well located in the Vallecito dry river bed, 13 km from the plant, at an elevation 100 m above the plant site. A 1200 m^3 capacity tank located at the plant site will provide both fire and fresh water storage.

Process water will be sourced from the process water pond, as reclaim from the processing plant.

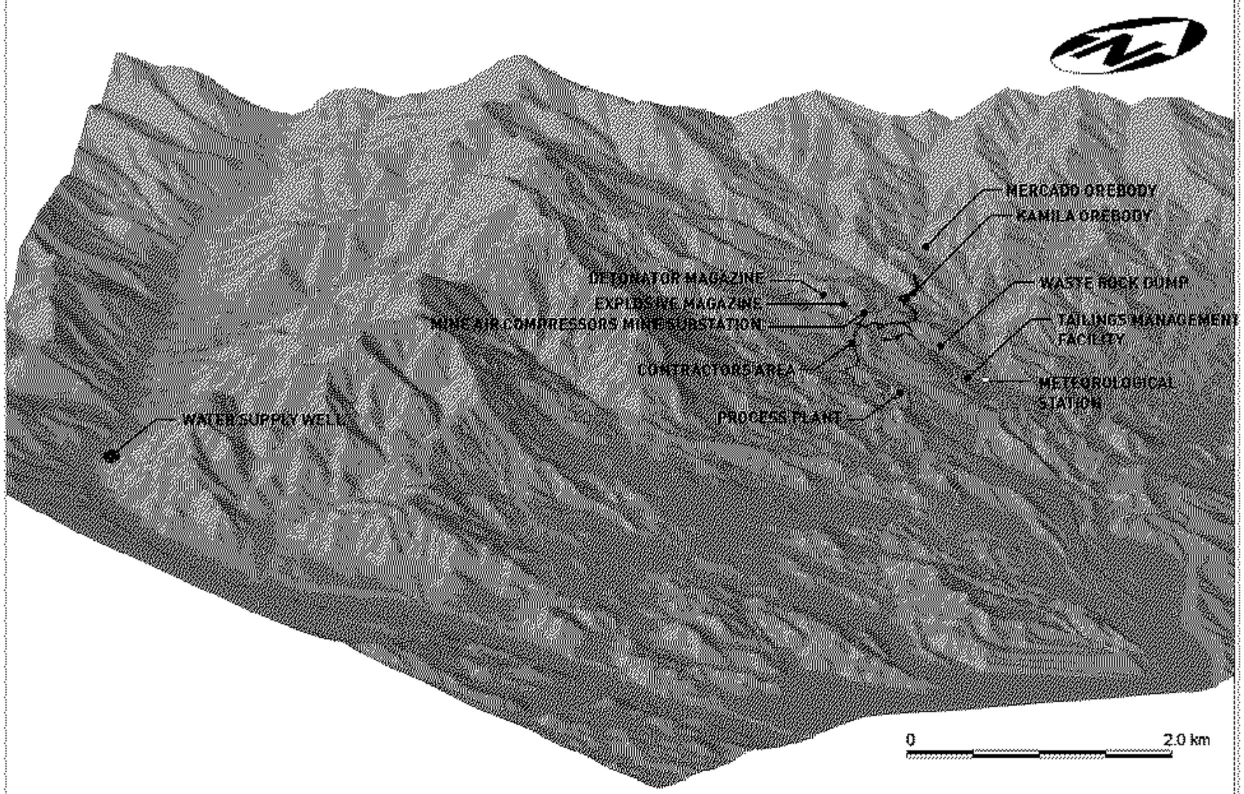
5.4.2 Site Infrastructure

Current site infrastructure comprises a gravel access road and a weather station. Camp operations during the exploration and early development work are serviced from Intrepid-owned property in Villa Calingasta and Villa Corral

The planned site infrastructure consists of: haul and access roads, roads around the site facilities, open pit and underground mines, a process plant, waste rock dumps, a tailings dump facility and administration and services (including water and power supply and communications).

The planned infrastructure layout is shown in Figure 5-2.

Figure 5-2
Planned Site Infrastructure Layout



5.5 Physiography

The Casposo property lies on the western side of the Calingasta Valley near the western edge of San Juan Province at the base of the Cordillera Frontal. The average elevation is roughly 2,400 masl. The Project site is located at the base of rugged terrain, characterized by low mountains with steep slopes, and ravines associated with dry drainage systems.

5.5.1 Soils

According to the Aptitud Agrícola Potencial de los Suelos, the Project site is located within a Class VIII soil zone, for which "the limitations are of such magnitude that it is impossible to use them for commercial agricultural or animal production". Soil use will therefore be restricted to recreation, the extraction of natural resources and/or protection of the basins.

Extensive areas of soil are limited within the Project Area due to the predominance of bedrock and rocky outcrops that prevent soil development. The small areas of soil constitute isolated spots that support the small amount of existing vegetation at site. From a socioeconomic point of view, mining activity at the site is unlikely to compete with other potential activities for land and soil use, and may be the only productive alternative.

In regard to current land use, the soils of the Project site are virgin, with no evidence of tilling or other agricultural practice. Existing soil use in the Project Site Area has been restricted to low-yield grazing and for conservation of local wildlife and (limited) native vegetation.

5.5.2 Surface Water

There is no current surface water use in the general Project Area, and specifically in the upper reaches of Quebrada Vallecito, because all of the water infiltrates the ground surface before reaching the Río Castaño. The irrigation channels that have been constructed in the other creeks descending from the Andes are diverted for agricultural activities in the arid Calingasta and Villa Corral area. Groundwater use in the area is also associated with agriculture and domestic water requirements.

The potential use of water resources in the area should not be significantly affected by the Casposo Project development, with the exception of limited groundwater extraction in the upper reaches of a suitable creek such as in Quebrada Vallecito that would follow from mining development. Given the extremely arid conditions in the general area, the application of technically advanced irrigation procedures would be required to develop vineyard and orchard production.

5.5.3 Flora

The dominant plant formation is a shrub steppe (>1 m tall) and sub shrub (<1 m tall) with a dominance of perennial grasses in the herbaceous stratum. Larger size shrubs characterize the lower foothill areas (e.g. *retamo Bulnesia retama*) and the bottom of valleys and ravines (e.g. *Larrea spp*), while subarbustive scrubs dominate on the slopes (e.g. *Gochnatia glutinosa*, *Artemisa mendozana*). On the high-elevation slopes, there are populations of *Denmoza rodacantha* columnar cacti, which are replaced by *Lobvivia formosa* at yet higher altitudes. In valley bottoms, there are small patches of wetlands commonly known as vegas which are dominated by gramineae and herbs that form a continuous grass covering. In low areas, the vegas are characterized by the presence of *Cortaderia rudiuscula* grass, while *Juncus balticus* and some Cyperoids characterize the vegas at higher altitudes. There are no vegas or endemic plant species in the Project Area.

5.5.4 Fauna

Two faunal surveys have been undertaken on the Project Area during 2006 by Knight Piésold.

Forty animal species were identified during the summer campaign: one reptile, 33 birds and six mammals. During the winter campaign, 35 species were identified in the study area: two reptiles, 23 birds and ten mammals. Both the median density and average abundance of species tend to steadily decrease with a decrease in altitude. Based on the survey, the bushy habitat of the mining road area has with the lowest average density and lowest species abundance of all areas analysed.

Within the Project Area, the only endemic species is the pale basketweaver or *Asthenes steinbachi*, but this is not expected to be irreversibly negatively affected by mine operations.

Two limnological campaigns were carried out in 2005–2006 to determine the structure of the aquatic communities through direct field observations in the Quebrada Vallecito, Rio Castaño, Río de Los Patos, Río San Juan and Vertiente de Araya water sources. The analysis of the Phytoplankton community identified diatoms, chlorophytes and cyanophytes. Most of the identified species corresponded to benthonic habits; these are detached from the bottom by the current, and therefore are found in the water column. The benthonic community was represented by diatoms, chlorophytes and cyanophytes alga species.

5.5.5 Seismicity

The region of San Juan, including the area of the Project, is in an active tectonic area, having experienced two large-scale earthquakes of magnitude (M_s 7.0 or greater, over the last sixty years. In particular, this region had been struck by a M_s 7.4 earthquake in 1944, causing nearly 10,000 casualties and leaving half the province homeless. Similarly, a magnitude 7.0 earthquake occurred in 1977, resulting in seventy people killed and up to 40,000 left homeless in Western Argentina. Records indicate that large-scale earthquakes occur in the region every forty to fifty years. Better building construction techniques and codes accounted for major improvement in death toll statistics. All facilities must now be built to withstand Richter 7 earthquakes to Argentine codes equivalent to UBC4 or better.

Despite the severe seismic classification of the Province, records indicate the Project is located in an area of relative low seismic density.

5.5.6 Other Potential Hazards and Risk

The potential impact of the Project and risks associated with soils, current land use and agriculture potential are regarded as low.

Rockfall

The Project Area contains relatively steep, tall bedrock slopes that represent a potential rockfall hazard, particularly in zones of fractured and/or weathered bedrock from which loose material may be more readily derived. Rockfall activity may be triggered by intense mechanical weathering processes (such as freeze-thaw cycles), raised secondary pore water pressures during significant rainfall events, or seismic activity. The frequency of fissures or cracks and the fracture pattern will influence the probability and severity of rockfall hazards. Additionally, the slope gradient and its material characteristics will influence the distance over which the rockfall material will travel. Soft colluvial ground conditions at the toe of a bedrock slope will tend to reduce the travel distance of rockfall material by slowing down or stopping the rolling or bouncing of large, mobilized rock particles.

Observations made during site investigation work of the extent and characteristics of rockfall accumulations were used to derive estimates of the magnitude of the morphogenic processes. This information was then used to generate qualitative estimates of the rockfall hazards present in the Project Area. In general, a significant percentage of the site presents rockfall hazards, as witnessed by the quantity of debris cones and slopes with associated rock outcrops above. These specific areas will present a hazard for personnel, equipment and structures. This was considered during site selection and the risk exposure to the facilities proposed in the feasibility study is considered to be low. Care should be exercised during the initial pit development with respect to potential rockfall hazards to personnel and equipment on the Kamila pit outcrop slopes.

Fluvial (Flood) Events

In general, the ephemeral watercourses within the Project Area support well-established vegetation, indicating relatively infrequent and non-intensive fluvial activity. Intense soil erosion associated with large storms has been reported, but these precipitation events are not frequent and the extent of erosion is generally localized. The overall fluvial hazard at the Project Site is considered low, with only some few, readily identified, areas presenting a hazard for personnel, equipment and structures. The water well system proposed in the Quebrada Vallecito is positioned far downstream of the active stream bed.

6.0 HISTORY

There is no recorded exploration on the Project prior to 1998. Deposit and prospect locations referred to in this section are shown in Figure 6-1.

6.1 Battle Mountain Gold

From 1993 to 1999 BMG conducted regional exploration programs in the San Juan Province, Central Argentina, on the basis of Landsat interpretation and the analysis of favourable mineralization criteria. In 1998, BMG focused its efforts on the Casposo Project, located in the Cordillera Frontal, where Landsat colour anomalies were interpreted to indicate the presence of intense alteration processes. Several outcropping veins yielding values as high as 5.65 g/t Au over 63 m were identified.

BMG undertook a program of surface sampling and geological mapping. A total of 22 trenches for 1,620 m of trenching were completed on the Kamila Zone, and 8,626 m of drilling in 46 holes were completed on the Kamila and Mercado deposits (collectively referred to as the Kamila Zone) during 1998. BMG's exploration also included an airborne magnetic and resistivity survey across an area measuring 15 x 25 km. A number of targets were delineated however only limited follow-up was carried out over areas outside the Kamila Zone.

A resource estimate was completed in 1998, but is not reported here as it was estimated prior to the introduction of NI 43–101 guidelines.

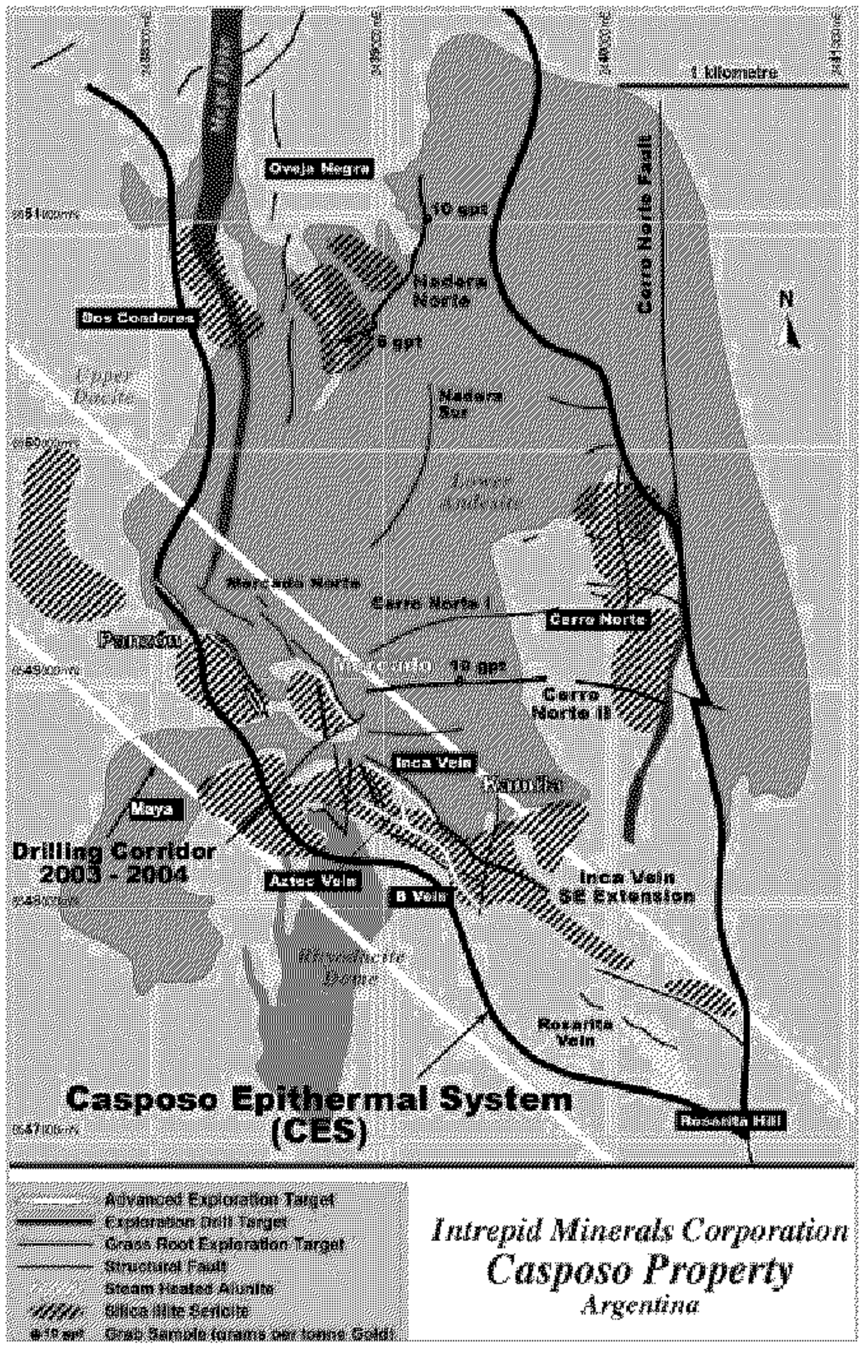
6.2 Intrepid Mines Limited

6.2.1 2002

In July 2002, Intrepid Mines Limited (Intrepid) acquired the Casposo Property, and conducted regional reconnaissance studies, detailed trench sampling of the vein systems, and logging and bulk sampling for metallurgical studies.

Four areas on the Kamila vein system and one area above the Mercado vein were sampled to provide material for first-stage metallurgical test work. The veins were sampled in a grid across the true width of each vein and over a strike length of 10 to 20 m. The distance between the various zones sampled at Kamila was approximately 500 m. Approximately 20 kg of material from each of the five zones was submitted for metallurgical testing.

Figure 6-1
Deposit and Prospect Location Plan, 2006



Note: Figure from Intrepid Mines Limited.

Mapping the northwest strike extension of the Casposo structure led to the identification of mineralized structures along the Casposo corridor over a total strike length of 1.6 km. A 1,678 m diamond drill program (16 holes, Intrepid Phase 1 drilling) was completed in early 2003, comprising twin hole drilling of selected BMG holes, and infill drilling over the Kamila Zone.

6.2.2 2003

A resource estimate, based on open pit and underground mining methods, was prepared in 2003 (Pitman and Puritch, 2003) using data from 50 drill holes, 46 trenches and four pits. Domain boundaries were determined from lithology, structure and grade boundary interpretation, for eight rock type domains, including the Aztec, Inca and B vein systems, dykes and waste. The Aztec Vein was interpreted from 16 north-facing vertical sections set on 25 m spacings, whereas the Inca and B Veins were interpreted from 13 northwest-facing vertical sections set on 50 m spacings. Each domain was interpreted for both open pit and underground resource estimations. Open pit and underground resources were developed within the same resource space and therefore cannot be summed (Pitman and Puritch, 2003).

The domains were physically created by computer screen digitizing on drill hole sections in Gemcom. The outlines were influenced by selection of mineralized material (+0.4 g/t gold equivalent (AuEq) for open pit and 5.0 g/t AuEq for underground) that had a reasonable expectation of being profitably mined as well as by lithological and structural controls. The AuEq grade was determined with a Ag:Au ratio of 90:1. This ratio is based on metal prices of US$350/oz Au and US$4.50/oz Ag combined with recoveries of 94% Au and 81% Ag as determined by metallurgical testing.

On each section, polyline interpretations were digitized from drill hole to drill hole but never extended more than 25 m into untested territory. The polylines from each section and domain were wireframed in Gemcom into 3-dimensional solids. Length weighted composites were generated for each drill hole, trench and pit sample that fell within the constraints of the domains. The composites were compiled over 1.5 m lengths. Any calculated composites that were less than 0.5 m in length were discarded. Grades were capped for the Aztec vein at 60 g/t Au and 1,200 g/t Ag, for the Inca vein at 90 g/t Au and 1,000 g/t Ag and for the B vein at 50 g/t Au and 900 g/t Ag.

The specific gravity used was 2.3 t/m^3 for all domains and non-domain material. Block sizes for the model were 2.5 m x 2.5 m x 2.5 m (X, Y, Z). The interpolation method was Inverse Distance Cubed (ID3). Two interpolation passes were used to determine Indicated, and Inferred classifications to meet CIM guidelines (CIM, 2005).

The preliminary open-pit resource, estimated at an internal cut-off grade of 0.5 g/t and 1 g/t AuEq is shown in Table 6-1. A preliminary resource was also estimated for the deposit as an underground mine at a cut-off grade of 5.0 g/t gold AuEq (Table 6-2). The metal prices used were US$350/oz gold and US$4.50/oz silver.

The two resource estimates are for the same deposit volume; however they have been constrained differently. Therefore the underground and open pit two data sets cannot be added together and each represents a stand-alone preliminary resource estimate.

Table 6-1
Open Pit Resource, 2003 (Pitman and Puritch, 2003)

Tonnes			Grade Gold (g/t)	Silver (g/t)	Gold Equivalent (1) (g/t)	Contained Gold (oz)	Contained Silver (oz)	Contained Gold Equiv. (1) (oz)
Indicated	1.4 g/t AuEq cut off	1,879,000	5.04	127	6.69	304,500	7,665,800	404,000
Inferred	1.4 g/t AuEq cut off	439,000	1.90	91	3.08	26,800	1,287,000	43,500

Note: One oz of recoverable gold is equivalent to 90 oz recoverable silver.

Table 6-2
Underground Resource, 2003 (Pitman and Puritch, 2003)

Tonnes			Grade Gold (g/t)	Silver (g/t)	Gold Equivalent (1) (g/t)	Contained Gold (oz)	Contained Silver (oz)	Contained Gold Equiv. (1) (oz)
Indicated	1.4 g/t AuEq cut off	1,534,000	5.75	127	7.40	283,700	6,237,100	364,700
Inferred	1.4 g/t AuEq cut off	145,000	2.40	87	3.54	11,200	404,900	16,500

Notes: (1) One oz of recoverable gold is equivalent to 90 oz recoverable silver.

6.2.3 2004

From October 2003 to April 2004, Intrepid conducted a second phase of diamond drilling and surface exploration at Casposo. This Phase II program consisted of 3,158 m (24 holes) of drilling on the Kamila target and 1,804 m (13 holes) of drilling on the Mercado and Panzón targets. Intrepid also drilled 2,185 m using reverse circulation (RC; 12 holes) methods, although seven of these holes were not on the Kamila deposit per se. The main focus of drilling at Kamila was to upgrade and expand the resources identified in the 2003 estimate. An updated resource estimate was prepared in April 2004 for the Kamila-Mercado deposits.

The April 2004 resource (Puritch, 2004) was based on 93 drill holes, 108 trenches and five pits, of which 74 drill holes, 74 trenches and 5 pits were used to define the constraining domain boundaries. Model construction methodologies were similar to those described above for the 2003 resource estimate; however the rock type domains modelled were modified from 2003 and included the Aztec, B, Inca and Mercado veins, and waste. The composites were compiled over 1 m lengths. Any calculated composites that were less than 0.4 m in length were discarded. Gold and silver grades were capped with different caps for each domain.

The 2004 resource estimate used a bulk specific gravity of 2.50 t/m^3 for all domains and non-domain material. Block sizes for the model were 3 m x 3 m x 3 m (X, Y, Z). The interpolation method was ID3. Two interpolation passes were used to determine Indicated and Inferred classifications.

The resource estimate, reported to a cut-off grade of 1.4 g/t AuEq (Au:Ag ratio of 89:1), and using a gold price of US$375/oz, assumed processing and administrative costs of US$16.30/t, and a process recovery of 94% is shown in Table 6-3. Open pit mining methods were assumed.

Table 6-3
Open Pit Resource, 2004 (Puritch, 2004)

Classification	Tonnes	Au (g/t)	Ag (g/t)	Au (oz)	Ag (oz)	Au Equivalent (oz)
Indicated	775,000	5.57	87.5	138,800	2,177,900	163,200
Inferred	1,968,000	3.57	110.7	225,900	7,004,100	304,600

In June 2004, Intrepid commissioned gradient-array induced polarization (IP) and pole dipole IP surveys at the Kamila property. The surveys were conducted by Quantec Geoscience Argentina S.A. and comprised 18.5 km of gradient array IP and 2.75 km of pole–dipole survey. The focus of the survey was the southeastern extension of the Kamila vein system.

Lines were spaced at 100 m centres, with peg markers along the lines at 50 m intervals.

Scoping Study

The resource estimate was used as the basis for a scoping study in mid-2004 (Buck et al., 2004a; 2004b). The scoping study evaluated heap leach and open pit mining methodologies. Key items from the scoping study included:

- The Casposo mineralization would be mined by open pit mining using backhoe style excavators and diesel haul trucks.

- All ore and waste rock requires drilling and blasting. The mineralization would be mined at a rate of 438,000 t of ore per year (1,200 t/d).
- Waste rock would be placed in a stockpile located approximately 2.5 km from the open pit operations.
- No detailed geotechnical work or review were performed, however, observations of diamond drill core and site in-situ rock properties indicated that an overall pit slope of 50° could be supported for use in open pit planning.
- Total mine capital expenditures for equipment and facilities was estimated at US$7.9 M without project contingency.
- The mineralization and waste mining costs would be approximately US$1.59 and US$0.98/t respectively.
- The mining operations envisaged would require a total of 77 personnel in management, supervision, operations and maintenance
- The process flowsheet proposed a primary jaw crusher; grinding of ore in cyanide using a semi-autogenous grinding (SAG) mill and a ball mill; a centrifugal gravity concentrator; gravity concentrate subjected to intensive leaching with gold recovery by electrowinning; leaching of pulp in a series of agitated tanks; a Merrill Crowe circuit and refinery.
- Gold and silver bullion would be the products of this operation.
- Preliminary test work results suggested that plant recovery estimates would be 94% for gold and 74 % for silver.
- Total estimated processing plant (1,200 t/d) capital expenditures, before a 20% contingency was applied, was US$20.3 M.
- Operating costs were estimated at approximately US$13.80/t of mineralization.
- The processing plant was estimated to require 57 employees in management, supervision, operations and maintenance.
- A heap leach option would include a crushing plant, heap leach pad and a Merrill Crowe plant for gold recovery from cyanide solution from the heap leach.
- There were no heap leach tests performed. It was assumed that heap leach recoveries could be 75% for gold and 60% for silver.
- Total estimated heap leach (1,200 t/d) capital expenditures, before a 20% contingency was applied, were US$11.3 M.
- The potential operating cost was estimated to be of the order of US$7.61/t of mineralization.

- The heap leach operation was estimated to need approximately 44 employees in management, supervision, operations and maintenance.

The main infrastructure required to be constructed and/or upgraded for the potential project included access and site roads, power generating plant, maintenance facilities for the mine, warehouse and yard, other services facilities, mine services office building, waste stockpile, tailings storage area, potable water supply, sewage treatment, sanitary landfill, Villa Corral administration building and telecommunications and computer networking.

The total infrastructure expenditure for a conventional processing plant option was estimated at US$5.9 M of which approximately US$3 M would be required for the power plant construction. The heap leaching option infrastructure capital cost would be approximately US$4.9 M of which approximately US$2 M would be required for power plant construction. Total general services and administration (G&A) manpower was approximately 21 employees in management, supervision and support positions. The G&A total yearly operating costs were estimated at US$1.2 M or US$2.80/t of mineralization processed.

The total Project capital cost for the conventional processing plan option was estimated to be US$40.6 M in constant 2004 US dollars. This included an average 18% contingency. The capital expenditure estimate for a heap leach operation including contingency was US$21.5 M. The total average operating cost over the mine life is US$27.80/t of mineralization for the conventional processing plant operation and US$20.89/t of mineralization for the heap leaching operation.

The Project investment and returns were estimated to provide internal rates of return (IRRs) in the range of 14 to 17% at a US$350 gold price. Increases in capital and operating costs in the order of 10 to 15% reduced IRRs to ranges of 12–15%.

6.2.4 2005

From October 2004 to February 2005, 3,492 m of HQ diamond core drilling in 25 holes (Intrepid Phase III drilling) were completed. Diamond drilling focused on enhancing geological and grade continuity within the Kamila zone by increasing drill density within the major vein domains to a nominal 25 m spacing. Diamond drilling also included evaluation of the Kamila Zone proximal vein systems such as Mercado Norte, Panzón and Maya, and satellite vein systems of the Oveja Negra prospect.

From April to December 2005 Intrepid completed a further 3,980 m of diamond drilling in 20 holes (part of the Intrepid Phase III and Phase IV drilling programs). Drilling continued to evaluate the Kamila zone at depth, but also tested the Panzón, Oveja Negra and Inca SE extension prospects.

A resource update was completed by Eugene Puritch, with an October 2005 effective date, and validated by external consultants RSG Global Inc (McGuinty, 2007). Drill holes after CA-06-174 were not included in the resource model. The resource, at a 1.4 g/t AuEq cut off, and using metal costs of US$400 for gold and US$6.50 for silver, comprised 1.534 Mt at 5.75 g/t Au and 127 g/t Ag in the Indicated category and 0.145 Mt at 2.40 g/t Au and 87 g/t Ag in the Inferred category. McGuinty (2007) reported that the "estimate incorporated an additional 3,240 m of diamond drilling" over that included in the 2004 resource (Puritch, 2004). McGuinty (2007) also noted that the "new calculation used updated parameters for gold and silver price (US$400 and US$6.50 respectively) as well as revised costs for processing and G&A at US$13.80 and US$2.92 respectively". No further information is available on the estimate, and no technical report was prepared as there was a limited change in the global resource figures (McGuinty, 2007). The "global resource" and "potentially-mineable portion" of the resource are shown in Tables 6-4 and 6-5, as presented by McGuinty, 2007. The "potentially-mineable portion" is based on an optimized pit.

Table 6-4
"Global" Resource, 2005 (McGuinty, 2007)

Tonnes			Grade Gold Silver		Gold Equivalent	Contained Gold	Contained Silver	Contained Gold Equivalent
			(g/t)	(g/t)	(1) (g/t)	(oz)	(oz)	(oz)
Indicated	1.4 g/t AuEq cut off	1,879,000	5.04	127	6.69	304,500	7,665,800	404,000
Inferred	1.4 g/t AuEq cut off	439,000	1.90	91	3.08	26,800	1,287,000	43,500

Table 6-5
"Potentially Mineable Resource", 2005 (McGuinty, 2007)

Tonnes			Grade Gold Silver		Gold Equivalent	Contained Gold	Contained Silver	Contained Gold Equivalent
			(g/t)	(g/t)	(g/t)	(oz)	(oz)	(oz)
Indicated	1.4 g/t AuEq cut off	1,534,000	5.75	127	7.40	283,700	6,237,100	364,700
Inferred	1.4 g/t AuEq cut off	145,000	2.40	87	3.54	11,200	404,900	16,500

Note: One oz of recoverable gold is equivalent to 77 oz recoverable silver based on recoveries of 94% for gold and 74% for silver at a gold price of US$400/oz and silver price of US$6.50/oz. Current (2005) viability of the resource is established using a 1.4 g/t gold equivalent cut off grade based on stated recoveries and densities, processing and G&A costs of US$13.80 and US$2.92 respectively

Cerro Norte

Channel sampling and mapping were also undertaken at the Cerro Norte prospect 1 km east of Kamila.

6.2.5 2006

Kamila Area

In 2006, 9,944 m in 85 holes was drilled, primarily over the Kamila, and Mercado deposits, and the Kamila SEXT and Julieta prospects. Drilling was part of the Intrepid Phase V, VI and Vii programs.

A new resource estimate for Kamila with an effective date of November 2006 was completed by Intrepid (McGuinty and Puritch, 2006). The estimate used an updated drill and trench database which included drill holes up to hole CA-06-172. The database comprised 172 drill holes, 112 trenches and 5 pits, from which 133 drill holes, 71 trenches and 5 pits were used to define the constraining domain boundaries.

Domain boundaries were determined from lithology, structure and grade boundary interpretation from visual inspection of drill hole sections. Five domains were developed and referred to as High Grade, Aztec, Inca, B and Mercado vein systems. The High Grade domain was developed from drill hole intercepts above a cut-off of 10 g/t AuEq within the Aztec, Inca and B vein systems that demonstrated good down dip and section to section continuity. The domains were physically created by computer screen digitizing on drill hole sections in Gemcom.

The outlines were influenced by selection of mineralized material (+0.5 g/t AuEq) that had a reasonable expectation of being profitably mined, along with an interpretation of lithology and structure. On each section, polyline interpretations were digitized from drill hole to drill hole but never extrapolated more than 25 m from the closest drill hole.

Seven rock type domains were coded. The composites were compiled over 1 m lengths. Any calculated composites that were less than 0.4 m in length were discarded. Gold and silver grades were capped with different caps for each domain.

The estimate used a bulk density of 2.50 t/m^3 for all domain and non-domain materials. Block sizes for the model were 4 m x 4 m x 6 m (X, Y, Z). The interpolation method was ID3. Two interpolation passes were used to determine Indicated and Inferred classifications.

The estimate was derived by applying an AuEq (g/t) cut-off grade to the Au and Ag block models and reporting their respective tonnages, grades and classification. Table 6-6 reports the "global" resource estimate (McGuinty, 2007) at a 1.40 g/t AuEq (Au:Ag ratio of 77:1) cut-off that is considered to be mineralization with the reasonable potential to become economically extractable. The following parameters were utilized to develop the 1.40 g/t AuEq cut-off grade equation:

- (US$15.70 + US$3.35/t) / [(US$450/oz/31.1035) x (94%)] = 1.40 g/t AuEq:
- Au price: US$450/oz
- Process cost: US$15.70/t
- General and administrative cost: US$3.35/t
- Au process recovery: 94%
- Ag process recovery: 80%

Table 6-6
"Global" Resource, 2006 (Puritch, 2006 in McGuinty and Puritch, 2006)

Classification	Tonnes	Au g/t	Ag (g/t)	Au (oz)	Ag (oz)	Au Equivalent (oz)
Indicated	2,208,139	4.45	114.7	315,700	8,139,400	421,400
Inferred	137,730	2.12	83.2	9,400	368,600	14,200

A first-pass Whittle 4X pit optimization was completed to determine the potentially extractable portions of the constrained mineralization in the Casposo block model. The optimization utilized the following parameters:

- Mineralization mining cost per tonne: US$1.62
- Waste mining cost per tonne: US$1.12
- Process cost per tonne: US$15.70
- General and administration cost per processed tonne: US$3.35
- Process production rate (ore t/a): 438,000
- Pit slopes (overall ramp angle): 50 deg
- Oxide bulk density: 2.50 t/m^3
- Waste rock bulk density 2.50 t/m^3

The resulting "potentially mineable" resource (McGuinty, 2007) is summarized in Table 6-7.

Table 6-7
"Potentially Mineable Portion of the Resource", 2006
(Puritch, 2006 in McGuinty and Puritch, 2006)

Classification	Tonnes	Au (g/t)	Ag (g/t)	Au (oz)	Ag (oz)	Au Equivalent (oz)
Indicated	1,691,000	5.29	115.4	287,300	6,270,900	368,800
Inferred	13,000	3.35	55.9	1,400	23,800	1,700

One oz of recoverable gold is equivalent to 77 oz recoverable silver based on recoveries of 94% for gold and 74% for silver at a gold price of US$400/oz and silver price of US$6.50/oz.

Julieta

In January 2006 road construction was completed by Intrepid to accommodate drill testing at the Julieta prospect. During February 2006, an initial 960 m of core in 10 holes was completed to a maximum 40 m depth. This was followed up in July 2006 by 21 shallow diamond holes for 1,675 m, resulting in nominal 25 m drill spacings across the prospect. Fluid inclusion studies were also completed.

6.2.6 2007

Feasibility Study

A Feasibility Study, commissioned in 2005, was competed in March, 2007. AMEC provided the initial metallurgy, geotechnical investigations, mining method selection, locations of proposed facilities, and financial modeling.

As part of the Feasibility Study, AMEC undertook an audit of the deposit's mineral resources as estimated by Intrepid as at October 2006 and reported in full in Section 17 of this report. The October 2006 Feasibility Study resource is modified from the resource estimated by Puritch in 2006, and reported by McGuinty and Puritch (2006) and McGuinty (2007) in that the Feasibility Study resource estimate uses data from 130 drill holes, 70 trenches and 5 pits. The Puritch 2006 estimate used 133 holes, 71 trenches and 5 pits.

AMEC also estimated mineral reserves for the Kamila and Mercado deposits. Environmental and social baseline and impact studies were completed by Knight Piésold and incorporated into the feasibility study by AMEC.

Results of the feasibility study are discussed in more detail in Sections 11 to 19 of this report.

The resource and reserve estimates presented in Section 17 of this report are those used as the basis of the feasibility study (AMEC, 2007), and are the figures presented in the Intrepid Mines Ltd annual information form (Intrepid Mines Ltd, 2007). The minor differences between the feasibility study resources and those "global resources" estimated by Puritch in 2006 (and reported in McGuinty and Puritch 2006, McGuinty, 2007) are due to changes in supporting data and rounding. Three fewer drill holes and one less pit are utilised in the feasibility study.

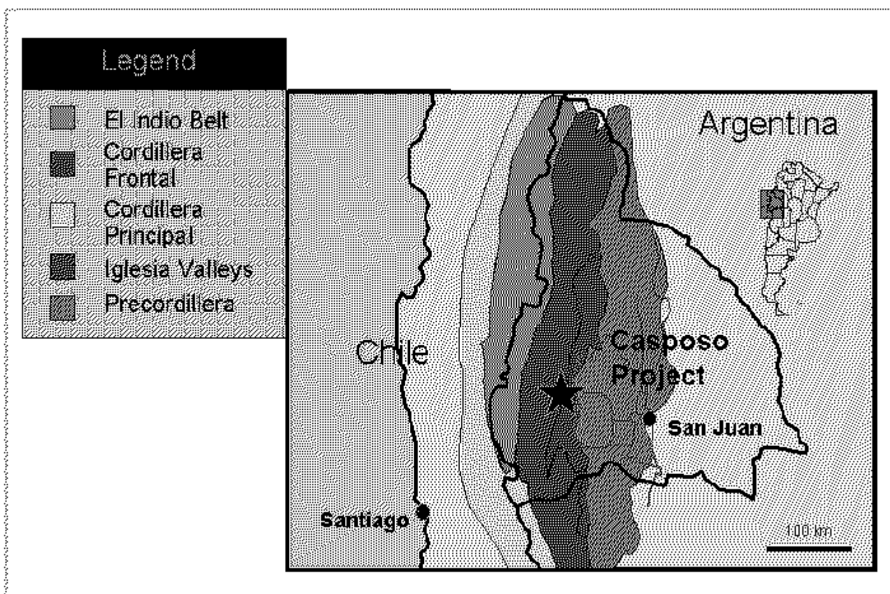
7.0 GEOLOGICAL SETTING

7.1 Regional Geology

The regional geological summary is adapted from McGuinty (2007).

San Juan Province straddles three major north–south-trending ranges, the Cordillera Principal, Cordillera Frontal and Precordillera as well as part of the Pampean range (Sierras Pampeanas range). The Project is located on the eastern border of the Cordillera Frontal, separated from the Precordillera to the east by the Rodeo–Calingasta–Uspallata Valley (Figure 7-1).

Figure 7-1
Regional Geology, Casposo Project



Note: Figure from McGuinty, 2007

The Cordillera Principal runs along the Chile-Argentine border for some 1,500 km. It is a volcanically and seismically active zone formed by subduction of the Nazca plate beneath the South American continent. This convergent plate margin has been active since the Cretaceous. The main basement is formed by Permian–Triassic intrusive and volcanic rocks, of calc-alkaline affinity and andesitic to rhyolite composition, regionally known as the Choiyoi Group. These and younger sediments of Jurassic and Cretaceous age have been thickened by compression and thrusting principally since the Late Cretaceous in a thin-skinned fold thrust belt.

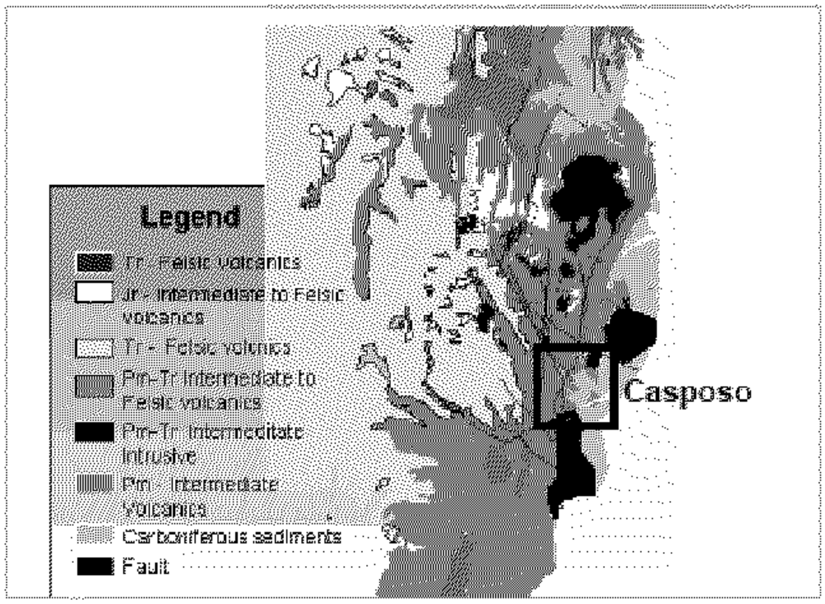
Mineralization associated with this range includes the Mio-Pliocene porphyry Cu–Mo deposits (El Teniente, Los Bronces–Disputada and Pelambres–Pachon), the El Indio Belt high sulphidation Au systems (El Indio–Tambo, Pascua–Lama and Veladero) and the Maricunga Belt porphyry Au–Cu deposits (Refugio, Marte–Lobo, Cerro Casale and Volcán). Outliers of the Palaeocene–Eocene porphyry Cu–Mo belt also occur in this region (Fortuna, Vicuña–La Batea). In this last group and in the El Indio Belt, 160 km north of Casposo, the wall-rocks include the Permo–Triassic volcanics (Figure 7-2).

Figure 7-2
Major Mineral Belts, Argentina–Chile

Note: Figure from Puritch, 2004

The Cordillera Frontal comprises a basement of Carboniferous clastic sediments to the west, intruded and overlain by Permian–Triassic volcanic and intrusive complex to the east (Figure 7-3). This complex consists of the same rock units as those in the Cordillera Principal, and was also uplifted with the Cordillera Principal. The Choiyoi Group hosts coeval mineralization, mainly porphyry Cu–Mo and Cu–Au deposits such as San Jorge and El Salado and low-sulphidation Au systems such as Casposo, La Cabeza and Castaño Nuevo. Tertiary mineralization occurs at Poposa (high-sulphidation Au) and at Paramillos (porphyry Cu–Mo) prospects.

Figure 7-3
District Geology, Casposo Project



Note: Figure from McGuinty, 2007

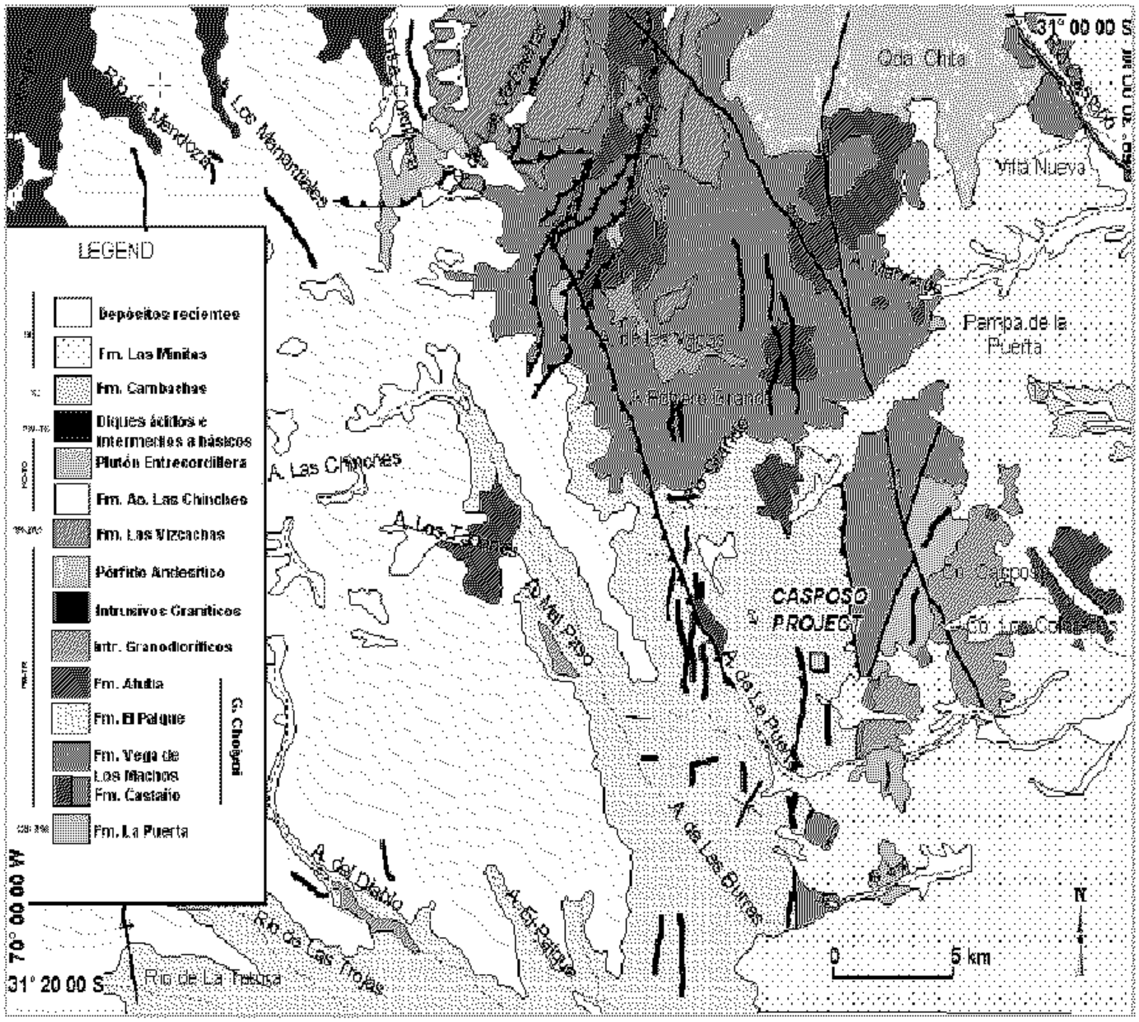
The Precordillera comprises a series of north–south ranges, covering about 1,000 km north–south and 100 km east–west. It is the product of large-scale tectonic compression since the Jurassic and culminating in the Miocene, and is still seismically active. The ranges in San Juan Province comprise Palaeozoic limestones and clastic sediments separated by plains reminiscent of the Basin and Range extensional terrain of the western United States.

East of the Precordillera, the Pampean and Transpampean Ranges (Sierras Pampeanas) are composed of Precambrian and Palaeozoic granitic and metamorphic rocks. Uplift occurred along Tertiary Laramide-style high angle reverse faults. These ranges host minor Precambrian mineralization and, within the Precordillera, some Tertiary-aged deposits, associated with calc-alkaline to alkaline volcanic and sub-volcanic centers of Miocene Pliocene age (for example Famatina and Gualcamayo).

7.2 Casposo Project Geology

In San Juan Province, the Cordillera Frontal is underlain by marine sediments (shales, sandstones and conglomerates) of the Carboniferous Cerro Agua Negra Formation. These sedimentary sequences are overlain by a thick intrusive and volcanic sequence assigned to the Permian–Triassic Choiyoi Group (refer to Figure 7-4).

Figure 7-4
Detail Geology, Casposo District



Note: Figure from Intrepid Mines Ltd.

Basal andesitic volcanic flows, tuffs and breccias are the main sub-surface unit in the Casposo Property and are overlain by rhyolite breccias, rhyolite-dacite flows and dacitic ignimbrite flows. The volcanic units dip gently to the east at 15–20° and are cross-cut by north–south, east–west and northwest–southeast-trending structures. Rhyolite and andesite dykes that trend north–south transect older rock units. Table 7-1 presents a stratigraphic column through the Project Area.

Table 7-1
Project Stratigraphic Column

Formation	Unit	Age	Estimated Thickness (where known)	Description	Comments	Petrographic Descriptions
Recent Deposits		Qt				
Las Minitas Formation		Qt				
Cambachas Formation		Tc				
Ao Las Chinches Formation		Kc-Tc				
	Andesite–Basalt Dykes	>Tr			Very late post mineralization. May be pre-main faulting	Latest magmatic event
Colanguil Batholith	Granite-Rhyolite Dykes			Colanguil granite phase: Includes the Tocota and Fraguita Plutons (tonalities and granodiorites) and Red Granite Suite in the same area. Cerro Colorado	Late post mineralization. Pre-main faulting. FeOx stained after pyrite – some white quartz veins	2
	Casposo Granodiorite	250Ma		Cerro Casposo and Rosarita area		1 (alteration M. Antonia prospect)
	Vallecito Pluton	264Ma				
El Palque Formation	Welded Rhyolites		>200m	Massive, strongly spherulitically devitrified welded rhyolite unit. Poorly consolidated 1m base surge deposit at Julieta	Post mineralization? W area of Ao La Puente and Julieta. Andesites similar to Casposo area but thick welded rhyolite sequence around Don Bosco prospect is certainly Palque Fm. And same andesite seen locally below rhyolite at Julieta. Possibly younger than this vein but not likely.	None
Within Palque Fm	High Andesita Unit		<100m	Massively bedded andesite ash-flows and flow breccias with rare cm "basement" clasts. Weak argillic-propylitic alteration in the two areas known.	Occurs at Julieta (thin) and La Puerta S (thick) below the welded rhyolite unit.	None
El Palque Formation	Dacite Ash Flow		150 m	Single, major welded crystal dacite ashflow unit. Never mapped out. A similar unit occurs apparently within the Upper andesite unit near Pascual.	Generally only weakly altered, may contain megaveins at south Rosarita.	1
Co-Vega de Los Machos Intrusions	Trachyte Dykes				Early post mineralization	
	Megadyke-Dome				Early post (late syn) mineralization	1
	Casposo Vein System	280Ma			Brecciated-Banded Qtz-(Adularia) Veins	Many
	Oldest Trachyte Dykes				Pre-mineralization. Contain veins. Cut by EW Qtz-Carbonate veins.	2

The Casposo gold–silver mineralization occurs in both the rhyolite breccias and underlying andesite, where it is associated with banded quartz–chalcedony veins, typical of low sulphidation epithermal environments. Adularia in the main veins gives an age date of 280± 0.8Ma (K/Ar), very close to the published age dates for the andesite unit.

Mineralization at Casposo occurs along a 10 km long west–northwest (N120°E) regional structural corridor, with the main Kamila vein system forming a sigmoidal set 500 m-long near the centre (see Figure 6-1). The Mercado vein system is the northwesterly continuation of Kamila, separated by an east–west fault. A series of east–west veins (Cerro Norte and Oveja Negra systems) appear to splay off these major sets to the east and northeast. Together with the recently identified mineralization southeast of Kamila (Kamila SEXT) and the Julieta veins (5 km to the west–northwest of Kamila), the Casposo vein district identified to date covers an area of about 5 by 7 km, or 35 km^2.

Post-mineralization dykes, of rhyolitic (Kamila), aphanitic-felsic and trachytic (Mercado) composition often cut the vein systems. These dykes, sometimes reaching up to 30 m thickness, are usually steeply-dipping and north–south-oriented.

7.3 Deposit Geology

7.3.1 Kamila Deposit

The Kamila deposit is developed within a northwest-trending regional-scale structural corridor of N140E sinistral faults characterised by a pair of west–northwest-trending bounding vein systems, the Inca and B-veins. These veins appear to dip towards each other, converging to the southwest at depth and to the southeast along strike into a 'keel-shaped' envelope.

The envelope encloses a series of 'ribs' or dilational veins, which trend north and dip steeply west. The most important of these veins are the Aztec and Aztec Footwall vein structures. The majority of the exploration to date by Intrepid has been directed towards the Aztec, B-vein and Inca structures.

The vein system extends for over 650 m along strike and over 260 m in depth, with a general dip of 60° to 70° to SW. At surface, the individual veins attain 12 m maximum thickness, which decreases with depth to less than 4 m.

Geological mapping in 2003 indicated that veins at Kamila–Mercado were oriented along three dominant structural trends; N140E, north–south and east–west. Significant precious metal mineralization at Kamila and Mercado appeared to be related to the intersection of N140E and north–south quartz-bearing structures.

A geological plan for the Kamila Zone and typical geological section are shown in Figures 7-5 and 7-6 respectively.

7.3.2 Mercado Deposit

The Mercado vein system is exposed 200 m north of Kamila deposit and is separated from it by the east–west-trending south-dipping Mercado fault.

This northwest trending hydrothermal quartz vein zone extends for over 500 m along strike, and over 150 m in depth, dipping 45° to 50° to the southwest. The Mercado system is variably composed of a compact vein (Main Mercado Vein or MV-1) or various thinner parallel veins, from which the north–south-trending MV-1 vein splits. At surface, the Mercado veins reach 8–10 m in thickness (including over 4 m for the MV-1 vein), but widths generally decrease with depth to less than 4 m.

Mercado exhibits bounding fault and dilational vein characteristics similar to the Kamila deposit. At surface the vein system crops out as an apparent forked vein system opening to the south towards the Kamila deposit and dipping to the west. Intrepid considers Mercado to be part of a larger Kamila structure. The Kamila zone appears to be dropped down relative to Mercado, possibly along the late-stage east–west fault.

A geological plan for the Mercado Zone and typical geological section are shown in Figures 7-7 and 7-8.

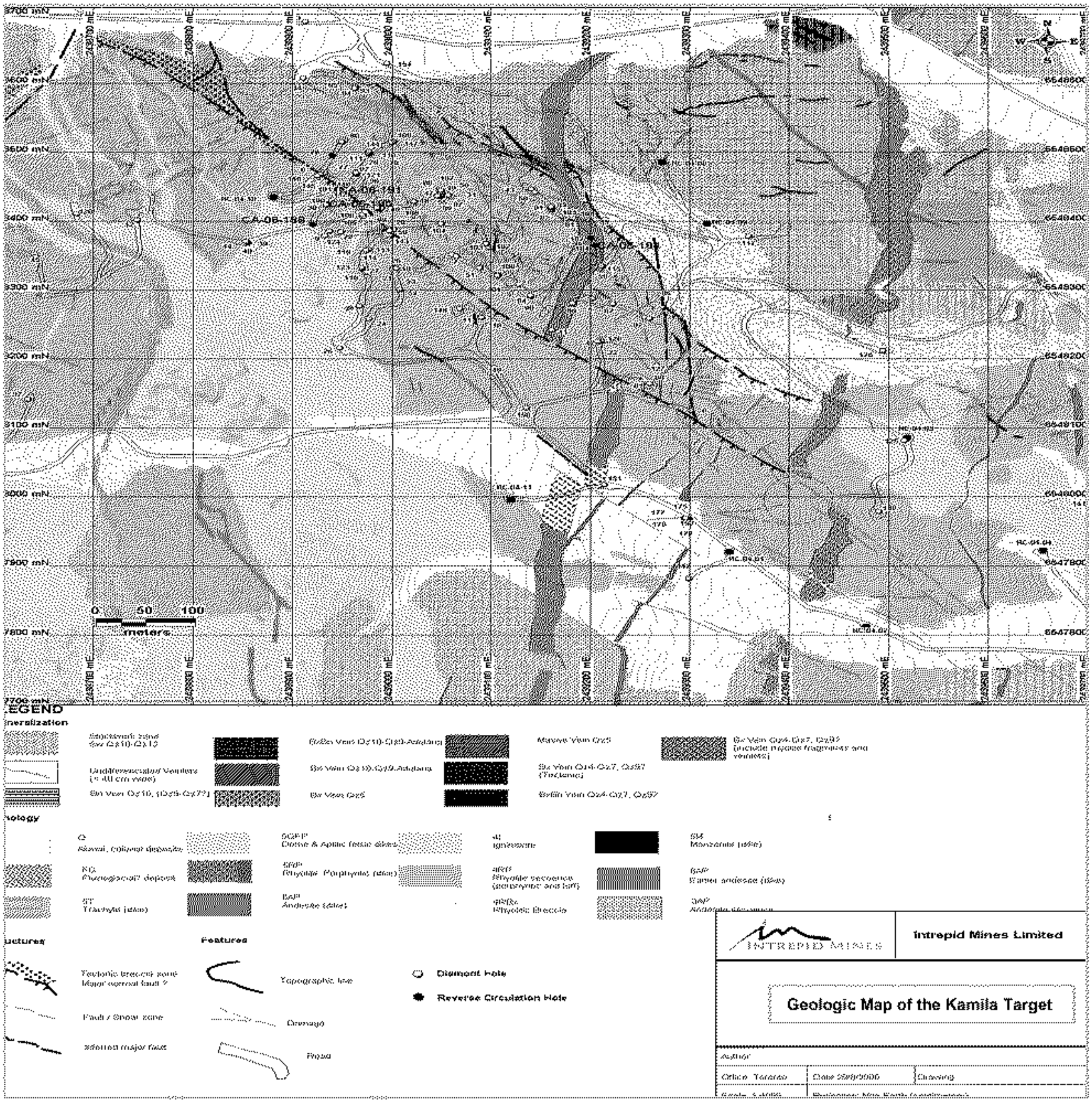
7.3.3 Mercado Norte Prospect

No information was available to AMEC for the Mercado Norte vein complex.

7.3.4 Panzón Prospect

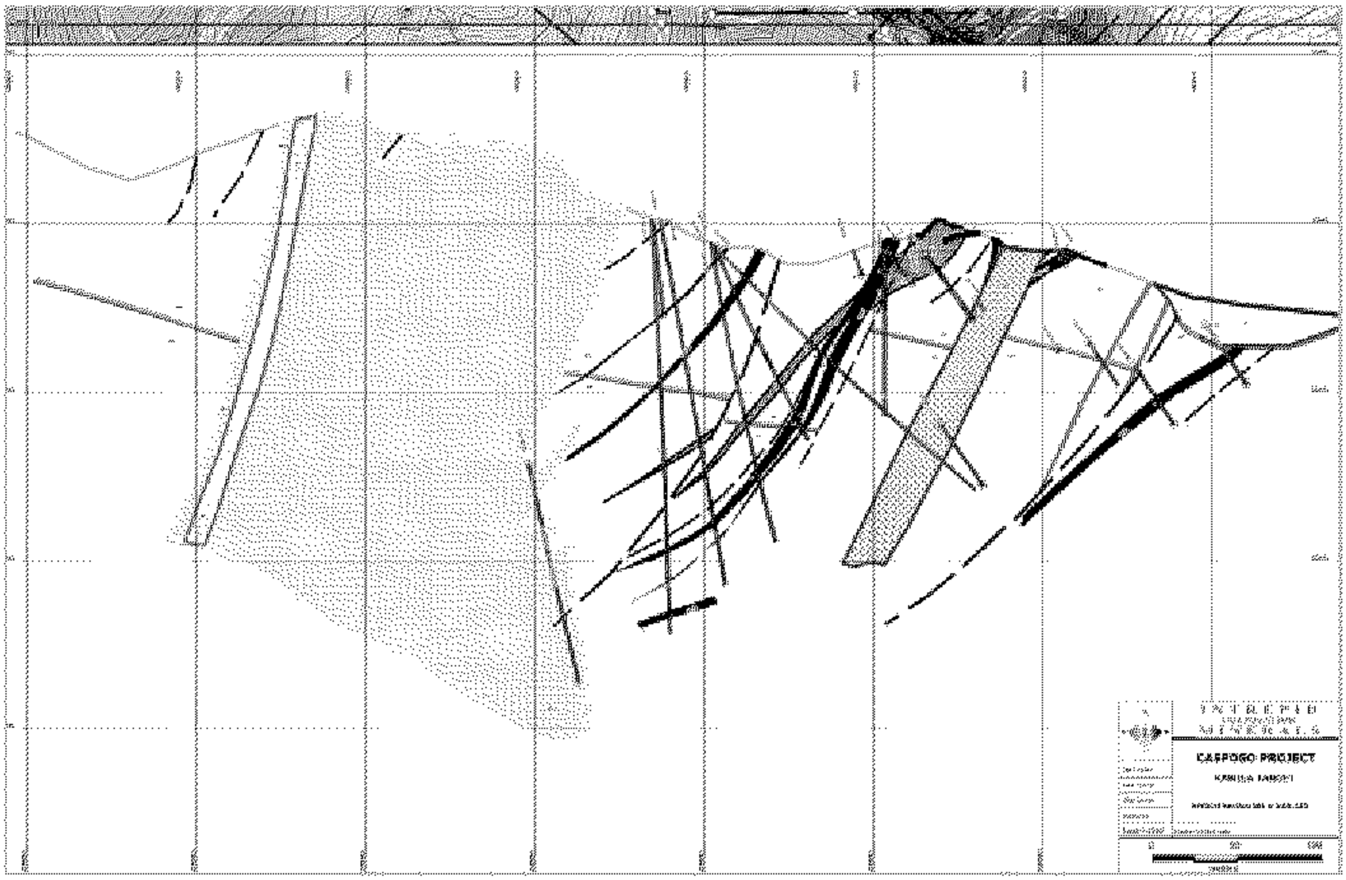
The Panzón prospect is located approximately 600 m west–northwest of the Mercado Zone. The Panzón vein system trends east–southeast, and is parallel to, but south of, the Mercado and Kamila vein structures. True horizontal separation of the two vein structures appears to be 250 m. Projection of this structure suggests this feature may be an extension of the structure controlling the B-vein system or a separate more southerly vein structure related to the Maya vein.

Figure 7-5
Geology Plan, Kamila Zone



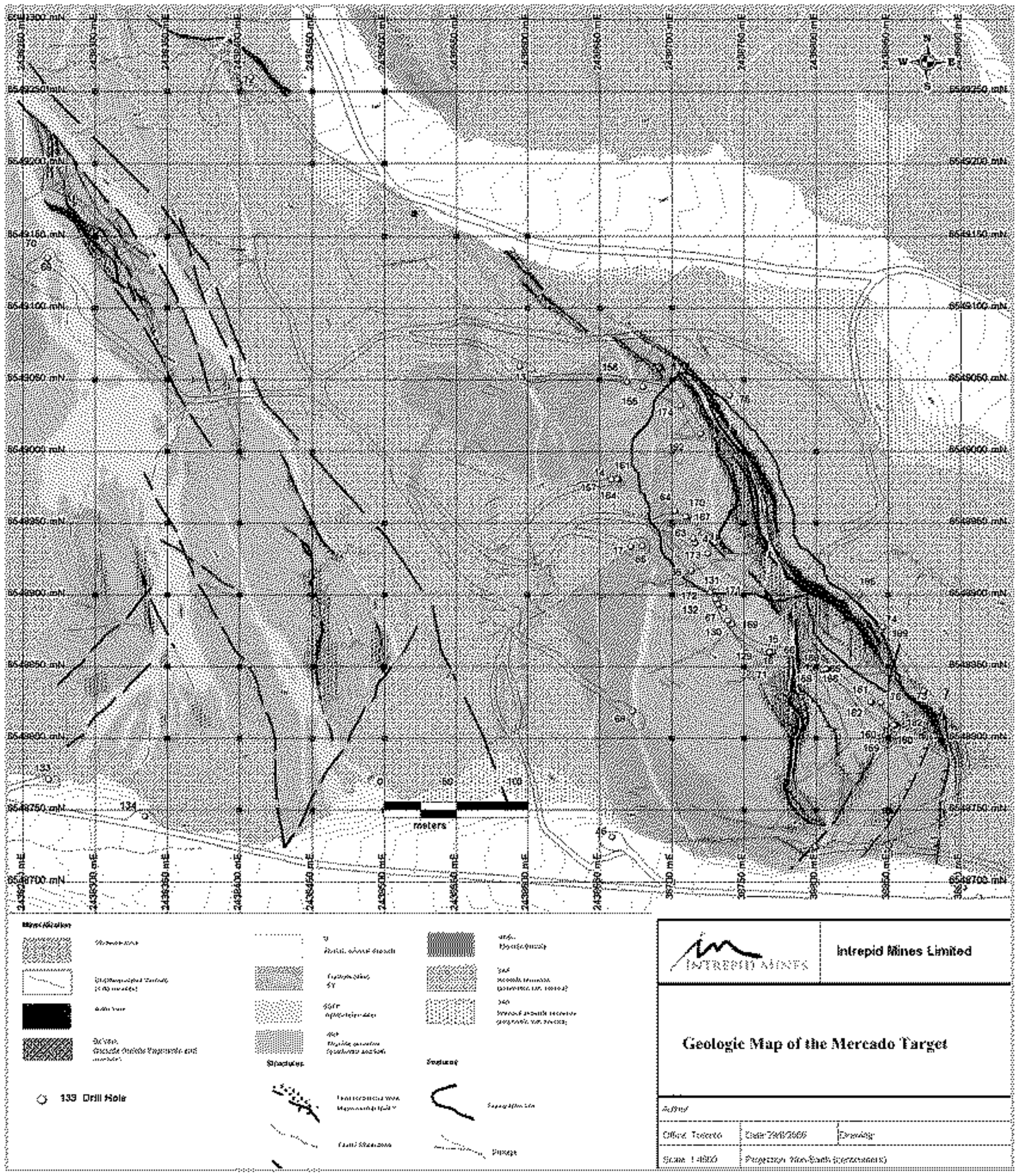
Note: Figure from Intrepid Mines Limited.

Figure 7-6
Typical Section, Kamila Deposit



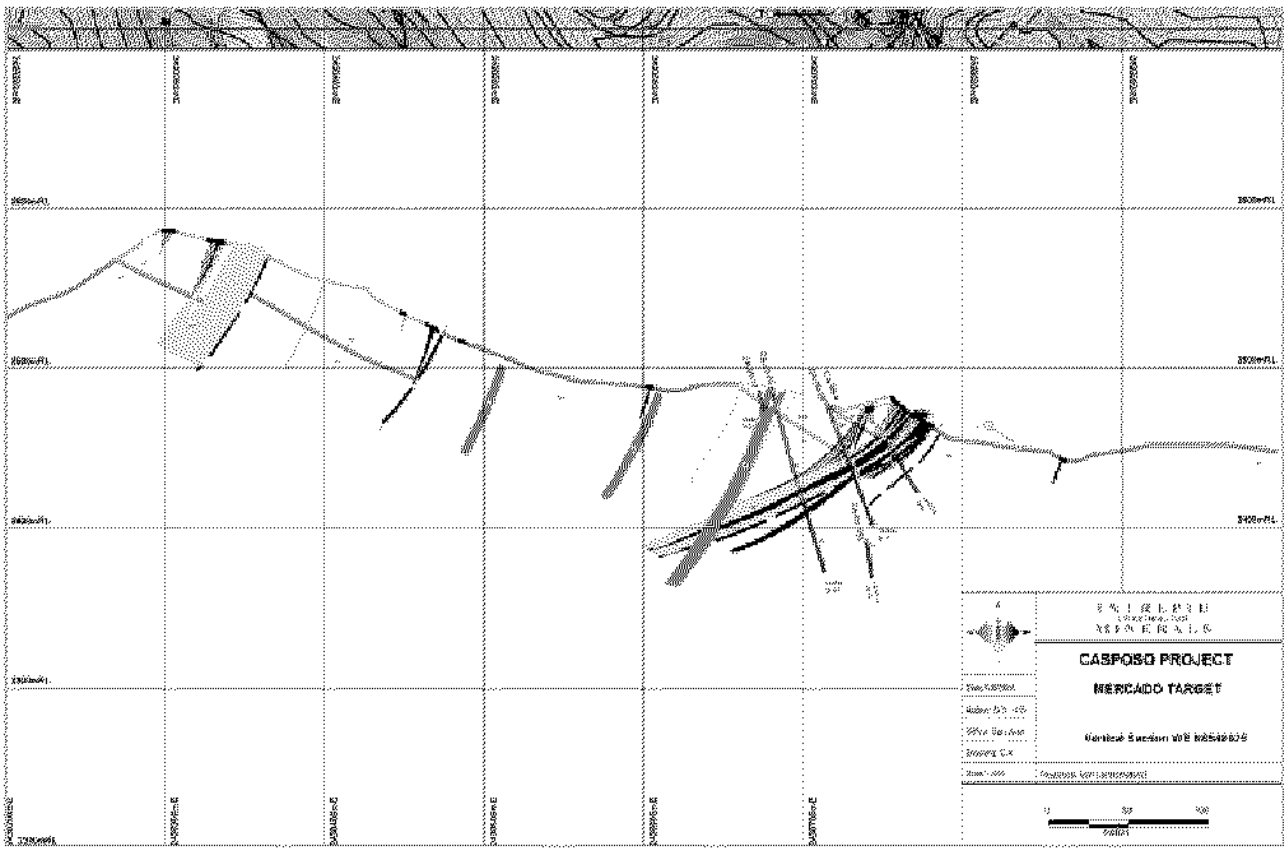
Note: Section looking North. Section is oriented west to east, along section line 6548325, and was drawn at 1:2,000 scale.
Figure from Intrepid Mines Limited.

Figure 7-7
Geological Plan, Mercado Deposit



Note: Figure from Intrepid Mines Limited.

Figure 7-8
Typical Section, Mercado Deposit



Note: Section looking north. Section is oriented west to east, along section line 6548925, and was drawn at 1:2,000 scale. Figure from Intrepid Mines Ltd.

At Panzón limited low-grade veining was intercepted in a volcanic sequence intruded by numerous dykes. Felsic dykes in this area trend northerly and occupy north-westerly structures previously mineralized by quartz veins and have stopped out the vein material making consistent targeting of mineralized zones difficult.

No plans or sections were available to AMEC for this prospect.

7.3.5 Oveja Negra Prospect

The Oveja Negra veins occur in unaltered andesite volcanic rocks and range in thickness from 30 cm to 1.5 m, generally striking east–northeast and dipping north. Several veins have sigmoidal patterns which generally retain the same axial planar strike and dip as the main veins. The main vein set is intersected by a series of smaller southerly-trending veins.

AMEC did not view any plans or sections for the Oveja Negra vein system.

7.3.6 Inca SE Prospect

No information was available to AMEC for the Inca SE zone.

7.3.7 Kamila SEXT (Kamila Southeast Extension) Prospect

The Kamila SEXT zone was discovered in February 2006 by exploration hole CA-06-152 which intersected a brecciated quartz vein assaying 4.51 g/t Au and 2,274 g/t Ag over 1.60 m at a depth of 90 m downhole. This zone is not exposed at surface, being truncated by a low angle, south-dipping southeasterly-trending fault.

Holes intersecting the Kamila SEXT structure laterally did not produce significant mineral intercepts although the vein structure was present over core intervals of 6 to 9 metres. It appears the vein in this area is situated in a normal fault plane (west side down) which offsets the relatively flat lying contact between the andesites and overlying rhyolite strata seen at Kamila.

A typical section for Kamila SEXT is presented in Figure 7-9.

7.3.8 Maya Prospect

No information was available to AMEC for the Maya zone.

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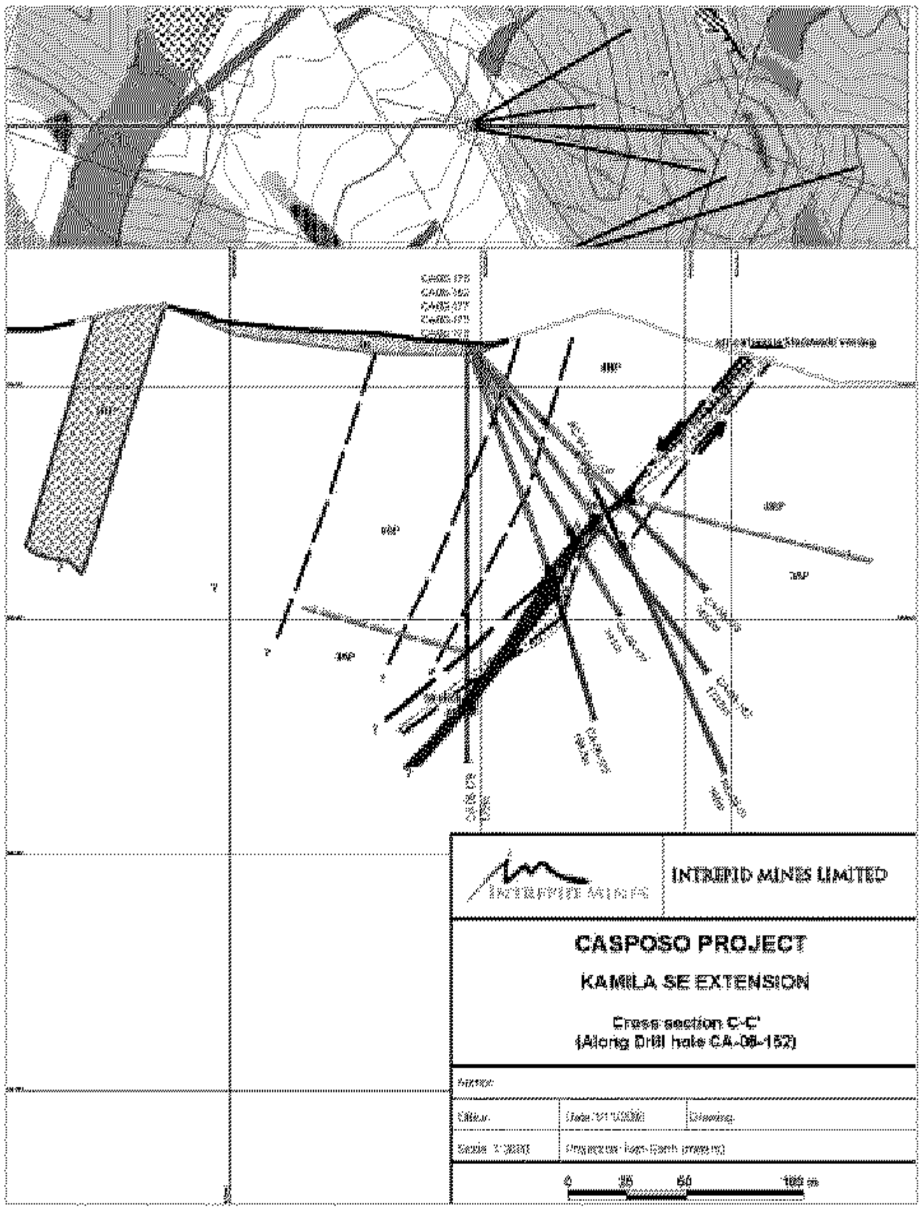
Figure 7-9
Drill Hole Section CA-06-152, Kamila SEXT



Note: Cross section shows vein and intercepts as grade thickness (g/t x m).
Figure from Intrepid Mines Limited.

7.3.9 Cerro Norte Prospect

Cerro Norte is a steep hill underlain by rhyolite volcanics. These volcanics are cut by a persistent set of east trending sub-vertically dipping quartz and quartz carbonate veins. The prospect comprises three main parallel vein sets and separated by a distance of more than 100 m. The aerial distribution of the vein system is approximately 1 km by 1 km. These veins are generally banded and locally brecciated and may locally exceed 4 m in width, although the average width is 2 m. Individual veins may be traced for over 100 m. Each braided vein complex extends over widths of between 2 to 10 m with strike lengths ranging from 650–950 m.

The host rhyolites appear to be locally silicified and brecciated with quartz fill. From a prospect-scale perspective the veins appear to have an anatomizing pattern. However at outcrop-scale the veins are heavily controlled by a north-trending joint set which causes sharp, right angled, step-like deviations in vein trends. Veins traverse rhyolitic rocks in the central, higher portion of the Cerro Norte prospect and stratigraphically-underlying andesite in the western and eastern extremities. This is consistent with the flat-dipping Permo Triassic lithologies visible at Kamila. Intrepid have interpreted the Cerro Norte vein complex as a large-scale extensional system which may be rooted in the basement below the Mesozoic andesite–rhyolite complex.

No plans or sections were available to AMEC for this prospect.

7.3.10 Julieta Prospect

The Julieta vein structure was discovered by Intrepid staff in 2005. The vein system is parallel to and roughly on-strike with the Kamila–Mercado vein system, 4 km to the southeast.

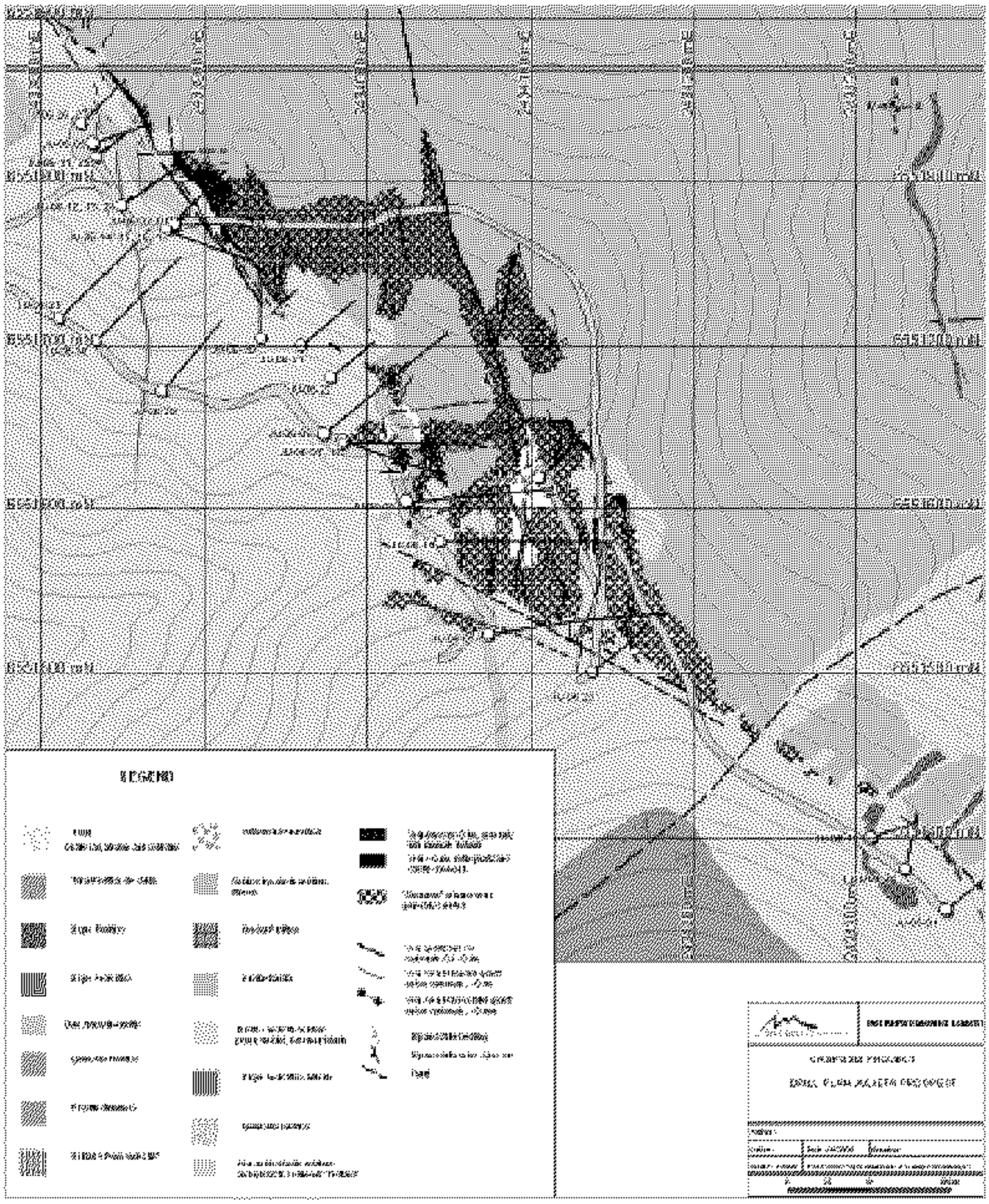
Host rocks to the Julieta vein set are vitreous, rhyolitic, welded ashflows which are weakly pyritized and, in the vicinity of Julieta veins, cut by numerous, often thick (10 cm), veinlets of quartz–calcite ("stockwork"). This unit is the only significant mineralization host. It is shallow dipping (30°) and in faulted contact with an interpreted younger) non-veined and less welded dacite ash flow which crops out west of the vein.

Both units are interpreted as being high in the Permo-Triassic Choiyoi Group volcanics (Palque Formation) and are probably 300 m stratigraphically above the Casposo flow dome and its accompanying andesitic ashflows. Overlying the poorly welded ashflows and possibly the veins are epiclastic sediments of the Cerro Pelado Formation, though no continuity of veining below these has been noted to date.

The vein system is emplaced along a northwest–trending, southwest–dipping fault with dips of 50–75°. A slightly later trachyte dyke of varying width (but generally less than 10 m), lies on the hanging wall of the vein, and locally cuts the vein. The dyke is propylitized and argillically altered. A single 10 m-wide north–south-trending rhyolite porphyry dyke cuts the vein in its southern part of the central area, though the veins that occur to the south and north appear to be similar, reflecting little change as a result of displacement. A set of steeply dipping north–south-oriented and east–west-trending veinlets and faults cut and sinistrally offset the main vein.

A geological plan for the Julieta system is included as Figure 7-10.

Figure 7-10
Geological Plan, Julieta



Note: Figure from Intrepid Mines Limited.

8.0 DEPOSIT TYPES

The mineralization identified at the Kamila and Mercado deposits, and other prospects within the Casposo Property are examples of low-sulphidation epithermal deposition of gold and silver.

8.1 Deposit Model

The type description for low-sulphidation epithermal deposits below is abstracted from Panteleyev (1996).

Low-sulphidation epithermal deposits are high-level hydrothermal systems, which vary in crustal depths from depths of about 1 km to surficial hot spring settings. Host rocks are extremely variable, ranging from volcanic rocks to sediments. Calcalkaline andesitic compositions predominate as volcanic rock hosts, but deposits can also occur in areas with bimodal volcanism and extensive subaerial ashflow deposits. A third, less common association is with alkalic intrusive rocks and shoshonitic volcanics. Clastic and epiclastic sediments in intra-volcanic basins and structural depressions are the primary non-volcanic host rocks.

Mineralization in the near surface environment takes place in hot spring systems, or the slightly deeper underlying hydrothermal conduits. At greater crustal depth, mineralization can occur above, or peripheral to, porphyry (and possibly skarn) mineralization. Normal faults, margins of grabens, coarse clastic caldera moat-fill units, radial and ring dyke fracture sets and hydrothermal and tectonic breccias can act as mineralized-fluid channelling structures. Through-going, branching, bifurcating, anastomozing and intersecting fracture systems are commonly mineralized. Mineralization forms where dilatational openings and cymoid loops develop, typically where the strike or dip of veins change. Hangingwall fractures in mineralized structures are particularly favourable for high-grade mineralization.

Deposits are typically zoned vertically over about a 250 to 350 m interval, from a base metal poor, Au–Ag-rich top to a relatively Ag-rich base metal zone and an underlying base metal rich zone grading at depth into a sparse base metal, pyritic zone. From surface to depth, metal zones grade from Au–Ag–As–Sb–Hg-rich zones to Au–Ag–Pb–Zn–Cu-rich zones, to basal Ag– Pb–Zn-rich zones.

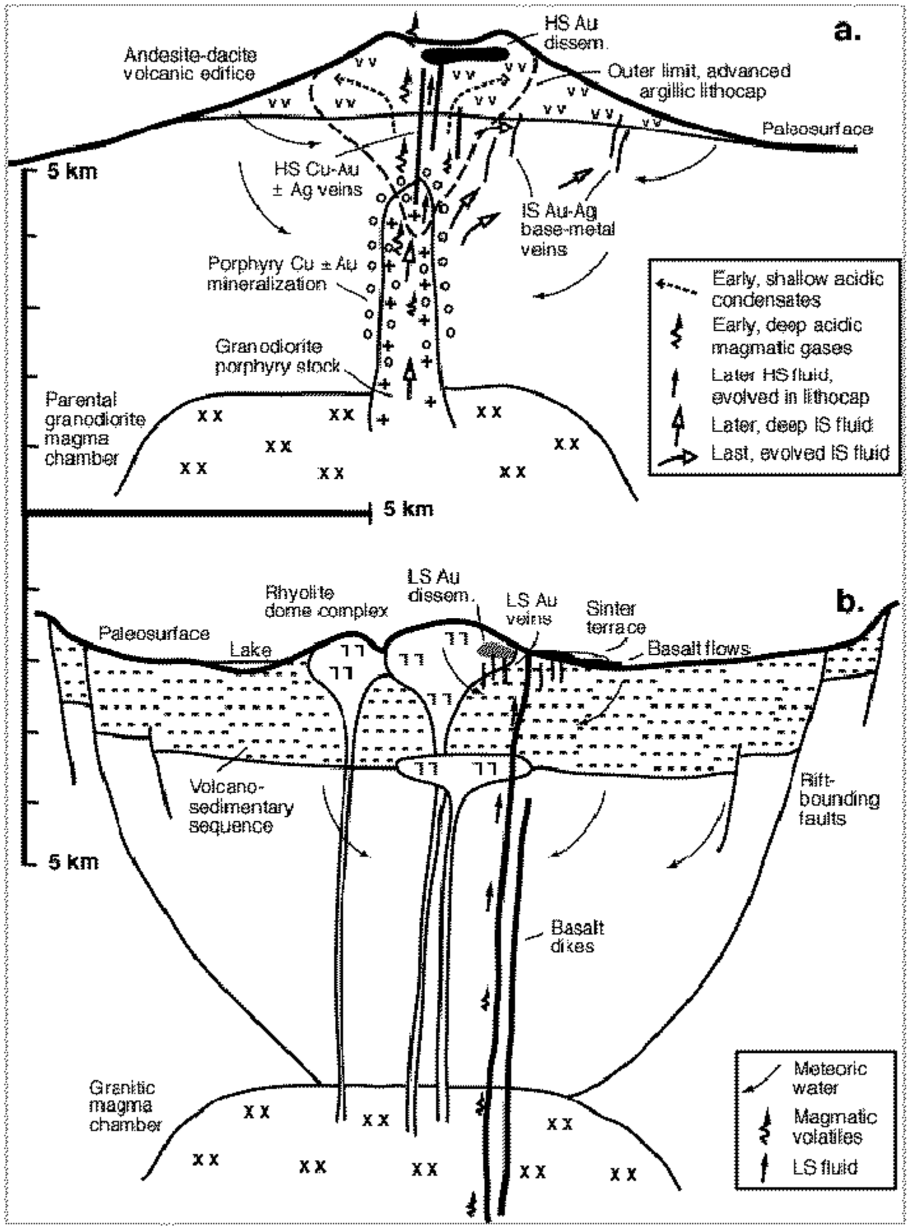
Silicification is the most common alteration type with multiple generations of quartz and chalcedony, which are typically accompanied by adularia and calcite. Pervasive silicification in vein envelopes is flanked by sericite–illite–kaolinite assemblages. Kaolinite illite–montmorillonite±smectite (intermediate argillic alteration) can form adjacent to veins; kaolinite–alunite (advanced argillic alteration) may form along the tops of mineralized zones. Propylitic alteration dominates at depth and along the deposit margins.

Mineralization characteristically comprises pyrite, electrum, gold, silver, and argentite. Other minerals can include chalcopyrite, sphalerite, galena, tetrahedrite, and silver sulphosalt and/or selenide minerals. In alkalic host rocks, tellurides, roscoelite and fluorite may be abundant, with lesser molybdenite as an accessory mineral.

Post the summary by Panteleyev, a number of workers have revisited the epithermal deposit classifications. Corbett (2001) introduced subcategories of arc-related low-sulphidation and rift-related low sulphidation to the traditional epithermal classifications in an attempt to better categorize features of epithermal deposits in the Chile–Argentina area. In 2000, Hedenquist et al., (2000) identified transitional features in a number of the South American epithermal deposits, and termed the transitional members "intermediate sulphidation" deposits.

Figure 8-1 shows a model for formation of epithermal-style mineralization. There can be a continuous gradation within a mineral district from low-sulphidation through intermediate- to high-sulphidation (pyrite-enargite-luzonite-covellite) assemblages.

Figure 8-1
Schematic Deposit Model, Epithermal-style Deposits



Note: Figure from Sillitoe and Hedenquist, 2003.

9.0 MINERALIZATION

9.1 Kamila Deposit

The gold–silver mineralization at the Kamila deposit is structurally controlled and occurs in crustiform-colloform quartz veins and stockworks in both andesites and rhyolites. The veining extends for about 2 km in a northwesterly direction, attaining widths of up to 500 m. Arsenopyrite and stibnite occur in the stockwork zones that are developed adjacent to the gold-bearing veins. Vein alteration is characterized by strong to pervasive silicification. Wallrock alteration varies from argillic to propylitic. Banded quartz–calcite veins with lattice bladed textures are common in the andesites.

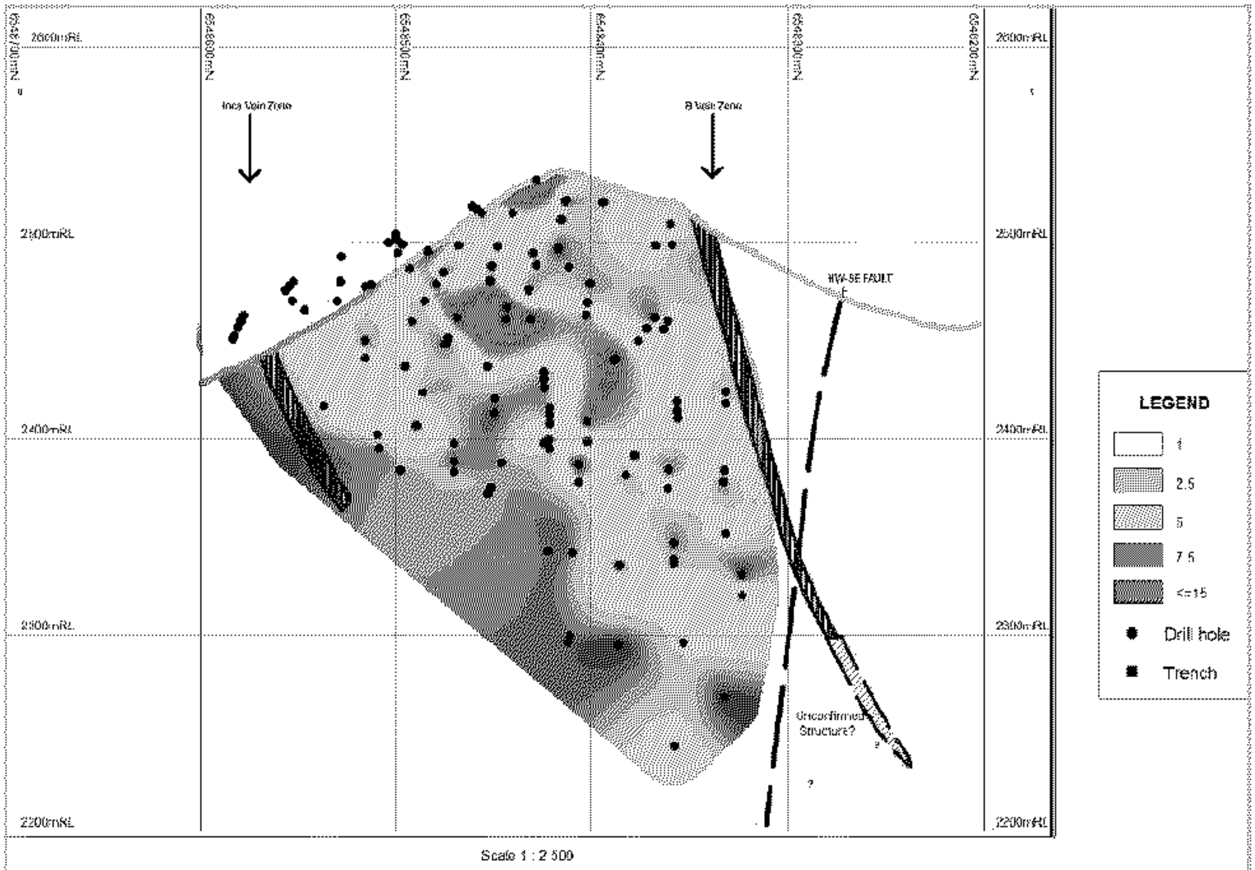
Interpretations of the drill core show that mineralization is vertically zoned, as follows:

- At 2,400 masl, crustiform textures are dominant;
- Between 2,300–2,400 masl quartz crystalline textures are dominant; and
- At <2,300 masl coarse crystalline quartz–carbonate textures dominate.

Quartz vein textural mapping from drill core and surface exposures indicate that the Aztec vein is dominated mainly by brecciated and banded textures, with minor bladed and massive patches. The B veins are texturally similar to the Aztec vein, and the Inca vein shows a balance between the dominant banded +/-brecciated textures along with areas of crustiform and colloform textures.

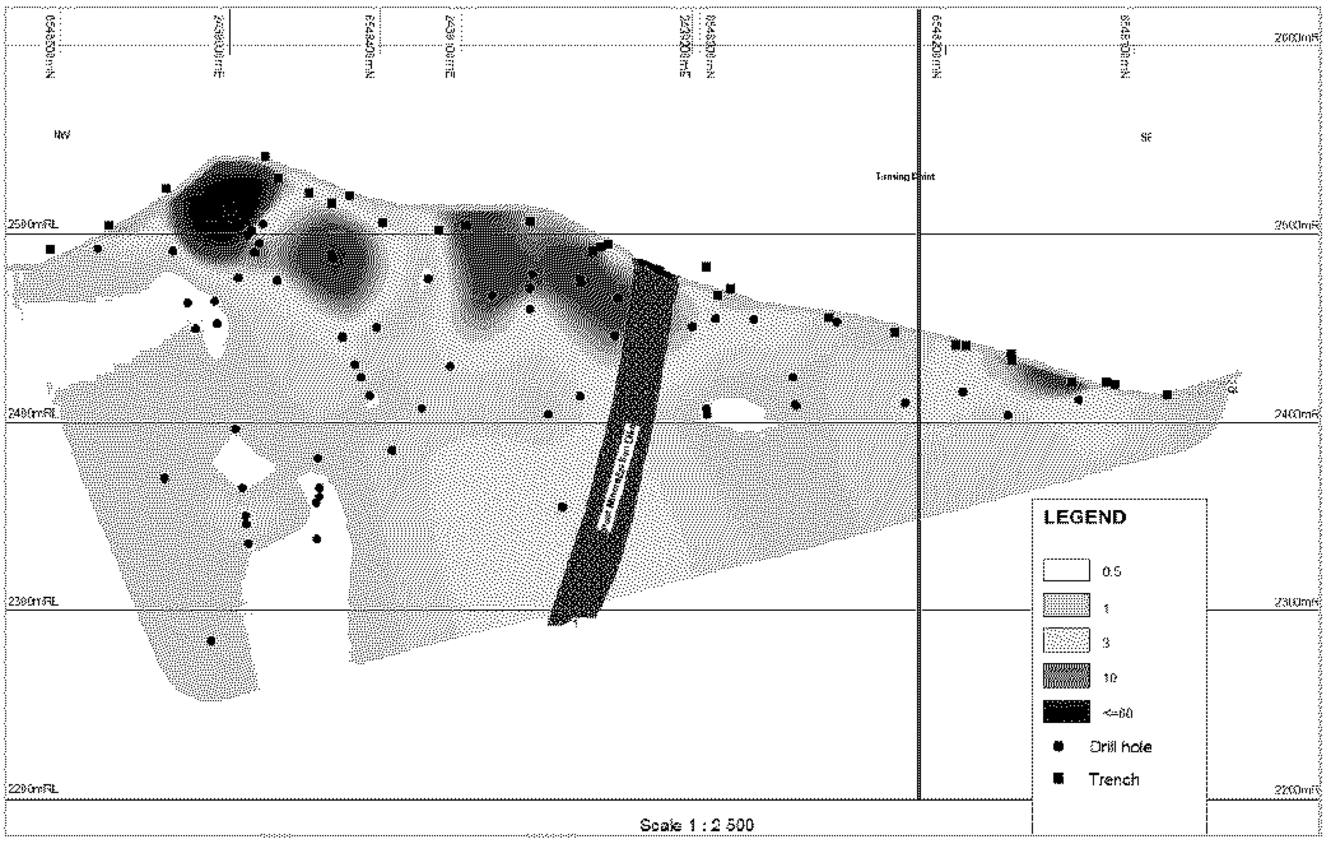
Longitudinal sections through the mineralization showing gold grades are presented for the Aztec (Figure 9-1), B (Figure 9-2), and Inca (Figure 9-3) vein systems.

Figure 9-1
Longitudinal Section Showing Au Grades, Aztec Vein



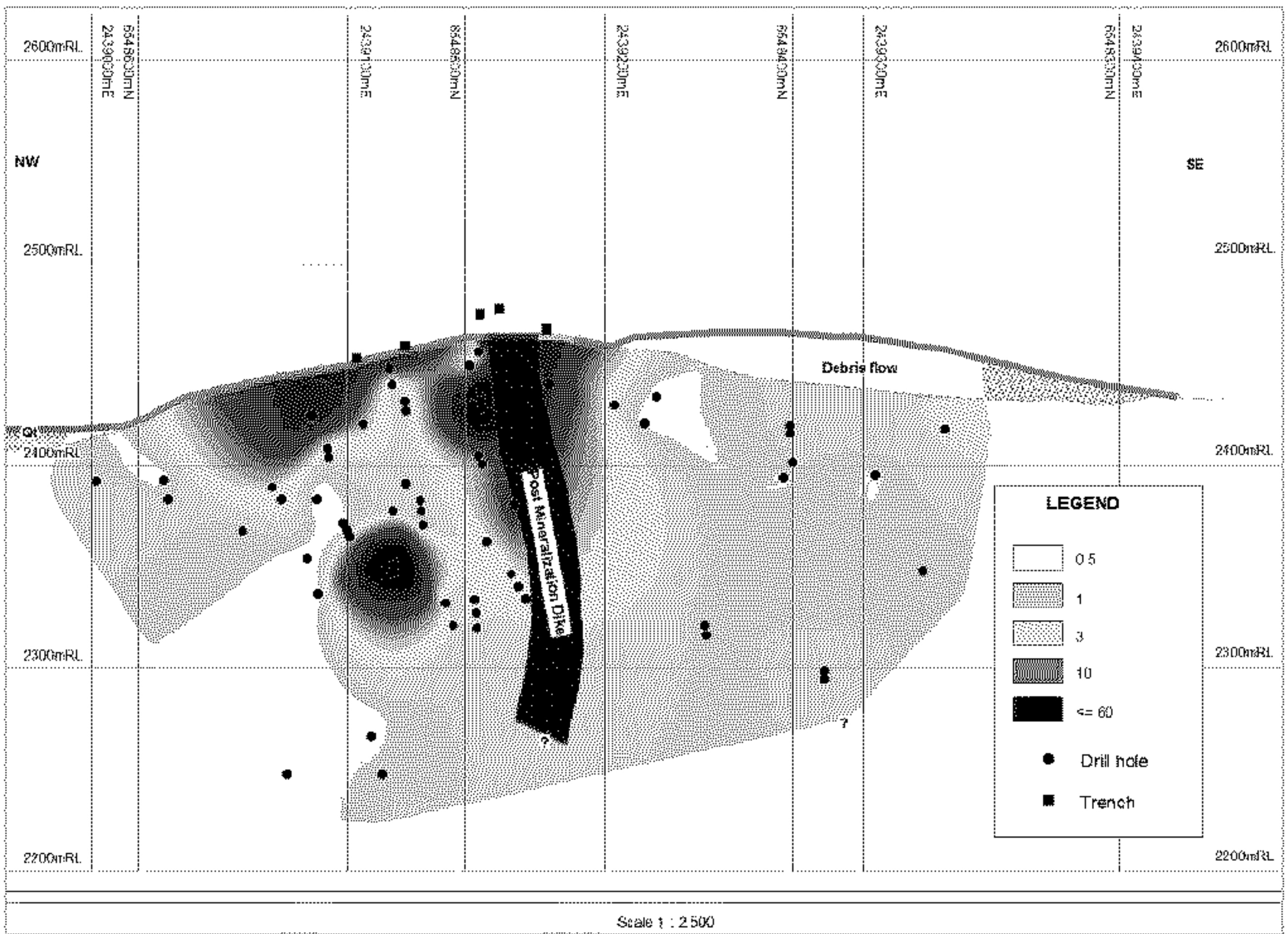
Note: Au grades in g/t Au; figure from Intrepid Mines Limited.

Figure 9-2
Longitudinal Section Showing Au Grades, B Vein



Note: Au grades in g/t Au; figure from Intrepid Mines Limited.

Figure 9-3
Longitudinal Section Showing Au Grades, Inca Vein



Note: Au grades in g/t Au; figure from Intrepid Mines Limited.

9.2 Mercado Deposit

Mineralization within the Mercado vein system contains moderately higher base metal values, as well as increased amounts of iron and arsenic sulphides, as compared to the veins at the Kamila deposit. Mineralization and differences in mineralogy associated with this vein system suggest the relative position of Mercado within an epithermal system would be lower than Kamila.

Two main quartz vein texture types were noted from drill core and surface exposures: banded–brecciated and banded with some areas also showing minor colloform textures.

A longitudinal section through the mineralization at Mercado showing grade thicknesses is shown, together with Kamila and SEXT in Figure 9-4; gold grades are presented in Figure 9-5.

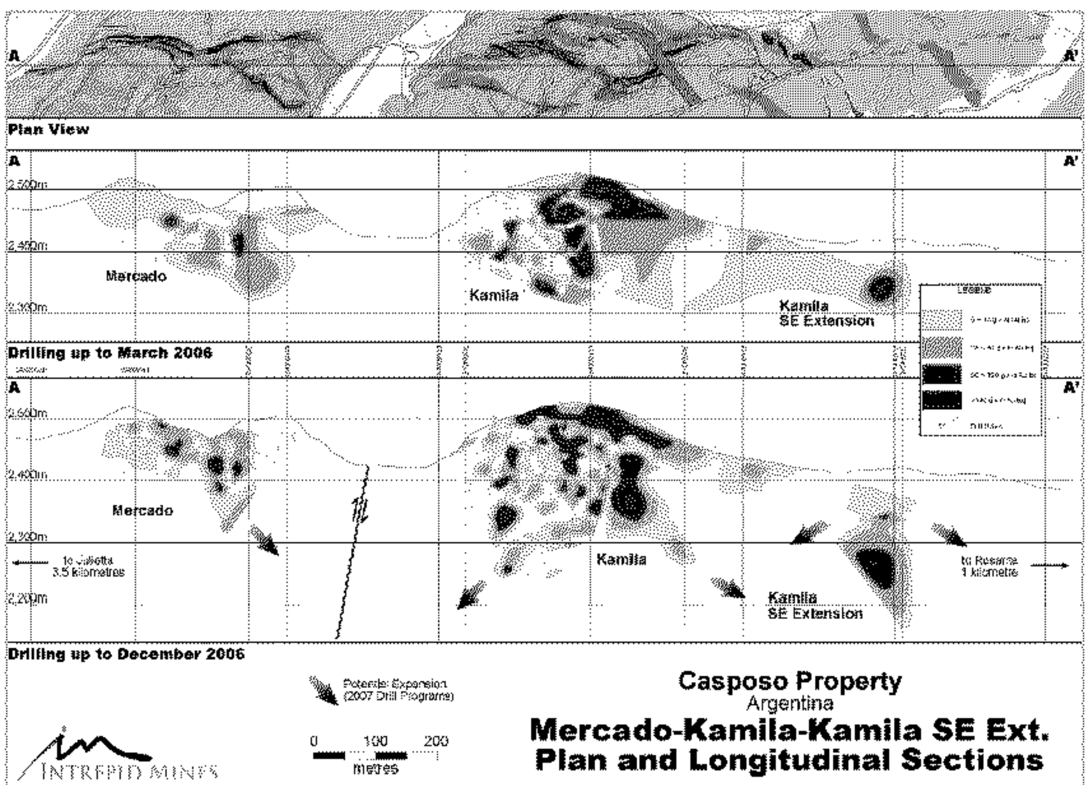
9.3 Mercado Norte, Panzón, Oveja Negra, Inca SE, and Maya Prospect

No information on the mineralization styles for these prospects was available to AMEC. All are reported by Intrepid (McGuinty, 2007) to be low-sulphidation quartz–adularia type veins that carry gold and silver values.

9.4 Kamila SEXT (Kamila Southeast Extension) Prospect

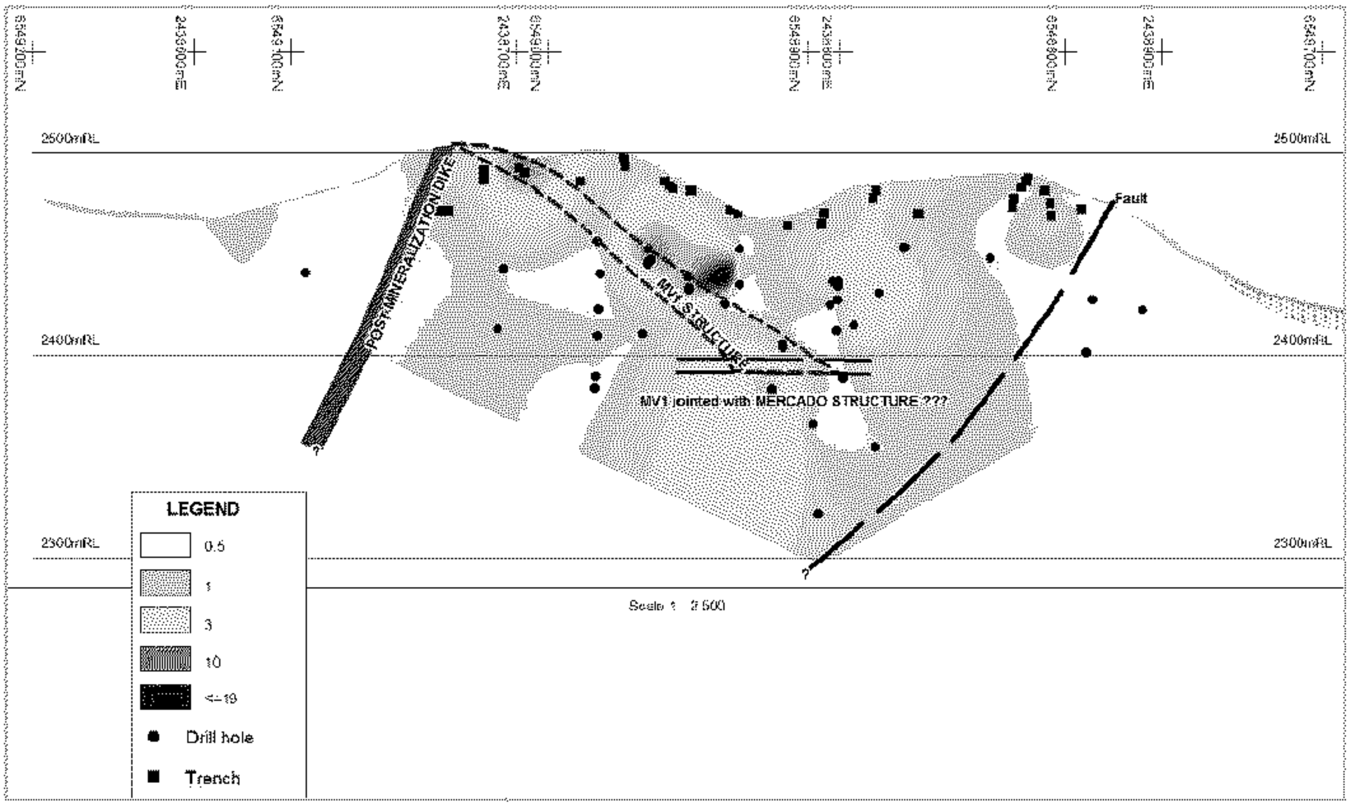
A longitudinal section showing the Kamila SEXT, Kamila and Mercado mineralization is given in Figure 9-4. No other information was available on the prospect.

Figure 9-4
Grade Thickness Contours, Kamila, Mercado and Kamila SEXT



Note: Figure from Intrepid Mines Limited.

Figure 9-5
Longitudinal Section Showing Au Grades, Mercado Vein



Note: Figure from Intrepid Mines Limited.

9.5 Cerro Norte Prospect

Vein textures vary from calcite–quartz in the east part of the Cerro Norte Prospect to lattice bladed–banded breccias in the west. Both of these features are indicative of high-level mineralizing episodes within a typical epithermal system.

Wallrock and host rock (andesite and rhyolite) captured within the anastomozing vein systems is silicified and contains numerous micro-veinlets of quartz and calcite commonly containing values of 100–300 ppb gold over widths of 2–10 m. These anomalous intervals are found in each vein set and across the entire (east to west) extent of the veins system. Silver values are low, generally in a 1:1 ratio with gold.

9.6 Julieta

Two gold mineralized veining events are recognized at Julieta; an early main vein with a central sigmoidally-shaped loop, and a north–south-trending vein swarm that constitutes a "stockwork" zone. The main vein is cut and displaced left-laterally (<5 m) by the vein swarm.

The main vein at Julieta consists of well banded quartz–chalcedony–calcite zones to the north and south of the central "Loop Zone" where banded, white, microcrystalline quartz and chalcedony form on average 50% of the vein system. The remainder is filled by calcite. Locally, the proportion of silica relative to calcite is extremely variable, ranging from 0 to 100%. Calcite deposition is typically synchronous with silica, forming parallel bands and radiating clusters of fine bladed crystals. Calcite replacement textures by silica are ubiquitous although not pervasive. No adularia was identified from surface mapping.

In the central "Loop Zone", the main vein splits, is segmented, and is affected by a later set of narrower, but continuous, north–south-oriented veins. The "Loop Zone" covers an area of 100 m x 150 m, and coincides roughly with a zone of increasing brecciation in the main vein. The veins forming the "Loop Zone" and most of the north south oriented veins are banded, with brecciation occurring mainly in the northwestern corner of the loop.

A trachyte dyke, which may be late-stage, cross-cuts the vein through the breccia zone.

The vein-breccias in the "Loop Zone" appear synchronous with vein emplacement as indicated by a 10–20 cm banded zone mapped in the footwall of the vein. The vein-breccias pinch-and-swell reaching maximum widths of 5 m, however these thicker pods are generally less than 20 m in length. Brecciation is interpreted as a combined tectonic-hydrothermal feature, with significant and locally dominant rock-flour cemented by either chalcedony or calcite. This contrasts with the cockade textures locally developed in the banded veins.

The system of north–south-oriented veins ("stockwork zone") in the centre of the loop appears to have formed later than the main vein and breccias. Faulting, associated with the late veins sinistrally displaces the main vein up to 5 m. Like the vein-breccias, the late veins have variable dips, generally steeper than the main vein and often vertical.

A good exploration target exists where the vein-breccias, along with the southerly-oriented veins and breccias within the loop converge on a low angle, northwesterly-trending vein of the loop. However, a late trachyte dyke that intrudes along the vein structure will make exploration, and establishment of grade continuity in this area a difficult task.

The Julieta vein system shows similar geochemical patterns to the Kamila area especially with regard to very low base metal values and very low As, Hg and Ba. The two also share moderately anomalous Sb (>10 to <200 ppm), which is strongly correlative with Au values, and low iron levels, corresponding to a low pyrite concentration.

Fluid inclusion studies on five samples of the Julieta vein (Townley, 2003) indicate that the relative depth of exposure is significantly higher than Casposo, as suggested by its stratigraphic setting. This would suggest possible continuity of mineralization at depth. The calcite content and high Au:Ag ratio would seem to confirm a higher level.

9.7 Mineralization Codes

A total of 55 lithology codes are used in geological logging of drill core at Casposo, as shown in Table 9-1.

Table 9-1
Logging Codes, Casposo Project

Code	Explanation	Litho Num
3ABx	Andesite breccia	1
3ABxqcv	Andesite breccia quartz–calcite veinlets	2
3ABxqv	Andesite breccia quartz veinlets	3
3ABxqvsw	Andesite breccia quartz vein stockwork	4
3ABxsw	Andesite breccia stockwork	5
3AP	Porphyritic andesite	6
3APcpyv	Andesite calcite–pyrite veinlets	7
3APqcsw	Andesite quartz–calcite stockwork	8
3APqcv	Andesite quartz–calcite veinlets	9
3APqv	Andesite quartz veinlets	10
3APqvsw	Andesite quartz stockwork	11
3APsw	Andesite stockwork	12
4I	Ignimbrite	13
4PC	Pyroclastic (crystal tuff)	14
4PCBx	Pyroclastic breccia	15
4PCqcv	Pyroclastic breccia quartz–calcite veinlets	16
4PCqv	Pyroclastic breccia quartz veinlets	17
4PCqvsw	Pyroclastic breccia quartz stockwork	18
4RBx	Rhyolite breccia	19
4RBxqv	Rhyolite breccia quartz veinlets	20
4RBxqvsw	Rhyolite breccia quartz stockwork	21
4RBxsw	Rhyolite breccia stockwork	22
4RDa	Rhyolite-dacite	23
4RDasw	Rhyolite–dacite stockwork	24
4RFbx	Rhyolite flow brecciated	25
4RFbxqvsw	Rhyolite flow brecciated quartz stockwork	26
4RP	Porphyritic rhyolite	27
4RPqv	Porphyritic rhyolite quartz veinlets	28
4RPqvsw	Porphyritic rhyolite quartz stockwork	29
4RPsw	Porphyritic rhyolite stockwork	30
5GR	Granite	31
5M	Monzonite	32
5QFP	Dome and aplitic felsic dike	33
5RP	Rhyolite dike	34
5T	Trachyte	35
6AP	Andesite dike	36
BnQv	Banded quartz vein	37
BX	Breccia	38
BxBnQv	Brecciated banded quartz vein	39
Bxqcv	Minor brecciated quartz calcite vein	40
BxQv	Brecciated quartz vein	41
Bxtc	Tectonic breccia	42
BxU	Undifferentiated hydrothermal breccia	43
cv	Minor calcite vein	44
DF	Debris flow	45
fr	Fracture zone	46
FZ	Fault zone	47
fz	Minor fault zone	48
NR	No recovery	49
QCV	Quartz–calcite vein	50
qcv	Minor quartz–calcite vein	51
Qt	Quaternary	52
QV	Massive quartz vein	53
qv	Minor quartz vein	54
sz	Shear zone	55

10.0 EXPLORATION

Two main phases of exploration have been undertaken on the Project: Battle Mountain Gold (BMG), from 1993–1999, and Intrepid Mines Ltd (Intrepid) from 2002 to the present. The exploration history is summarised in Section 6 of this report. In general, there is little documentation available on the methodologies of the BMG programs. Intrepid is in possession of all remaining physical core samples and core and channel sample assays certificates for sampling completed by BMG prior to the 2002 acquisition by Intrepid of the Property

10.1 Grids and Surveys

10.1.1 Battle Mountain

As a base for geological mapping, BMG used a 1:10,000 topographic map prepared by the Instituto de Fotogrametría of the San Juan University through photogrammetric restitution of 1:40,000 scale aerial photos. Successive restitutions to more detailed scales (up to 1:1,000) were later conducted.

Mapping was also supported by a 50 m x 50 m grid (locally 100 m x 100 m), with a northeast southwest orientation. Alfredo Herrada, an external contractor, connected the local grid to the National Geodesic Grid through three base points (Table 10-1). These measurements were referenced to the National Point 14-093, with POSGAR ´94 coordinates (Campo Inchauspe 1969 datum) as follows: Latitude = -31° 12' 19.8062"; Longitude = -69° 27' 40.6477"; Ellipsoidal Elevation = 1429.496 m.

Table 10-1
Coordinates of Topographic Base Points at the Casposo Project

Point	x (CAI '69)	y (CAI '69)	x (Posgar '94)	y (Posgar '94)
PB 01	6,547,715.64	2,439,623.26	6,547,508.88	2,439,534.07
PB 02	6,547,974.29	2,439,353.55	6,547,767.53	2,439,264.36
PB 03	6,548,285.55	2,438,754.96	6,548,078.79	2,438,665.77

10.1.2 Intrepid Mines

In 2003, Intrepid contracted Eagle Mapping Sudamérica (2003) to complete a low altitude flight over the Casposo area to take air photos, using two double-frequency Topcon GPS ground receivers as control. Based on the photos, a 1:1,000 topographic map with 2 m contour lines was prepared.

Coordinates in the Casposo Project correspond to the Argentine Gauss Krüger Campo Inchauspe 1969 system, with elevations established from USO 91. Eagle identified major differences between these coordinates and the Posgar 04 system, as follows: north, -6.252 m; east, +7.701 m; and altitude, +23.276 m. For this reason, the Gauss Krüger system was not adopted for use on this project.

Alfredo Herrada (see Section 10.2) continues to conduct topographic surveys at Casposo, when requested. Herrada uses two Ashtech-Magellan mono-frequency (L1) Monocode (C/A) receivers, model Pro-Mark X-CM, with 10 parallel channels, by the differential static method. The resulting data are processed with the MSTAR 2.03 software.

10.2 Geological Mapping

Intrepid has undertaken prospect-scale mapping at 1:1,000 scale in areas that were not covered by the initial BMG mapping.

10.3 Geophysics

In 2004, Quantec undertook a Time Domain Induced Polarization and Resistivity (IP/resistivity) survey on a portion of the Casposo Project, with the aim of detecting and delineating silicified structures and related disseminated sulphides associated with gold mineralization. The grid consisted of 100 m-spaced northeast–southwest lines staked with wooden pegs at 50 m intervals along the lines. The survey used an Iris Elrec-6 six channel time domain receiver and a 5 kW Iris VIP transmitter. All the survey data resulting from the gradient IP/resistivity and Pole-Dipole surveys were considered by Intrepid to be of high quality. No instances of poor contacts or grounding wires were observed by Quantec during the course of the survey. In virtually all areas, signal strength and repeatability were ample for the survey requirements (Quantec, 2004).

Quantec (2004) summarized the survey results as follows:

On lines 250W, 200W and 100W the position of known mineralized veins coincides with elevated resistivity (between 700 and 1000 ohm-m) as well as increase chargeability (7mV/V to 10mV/V).

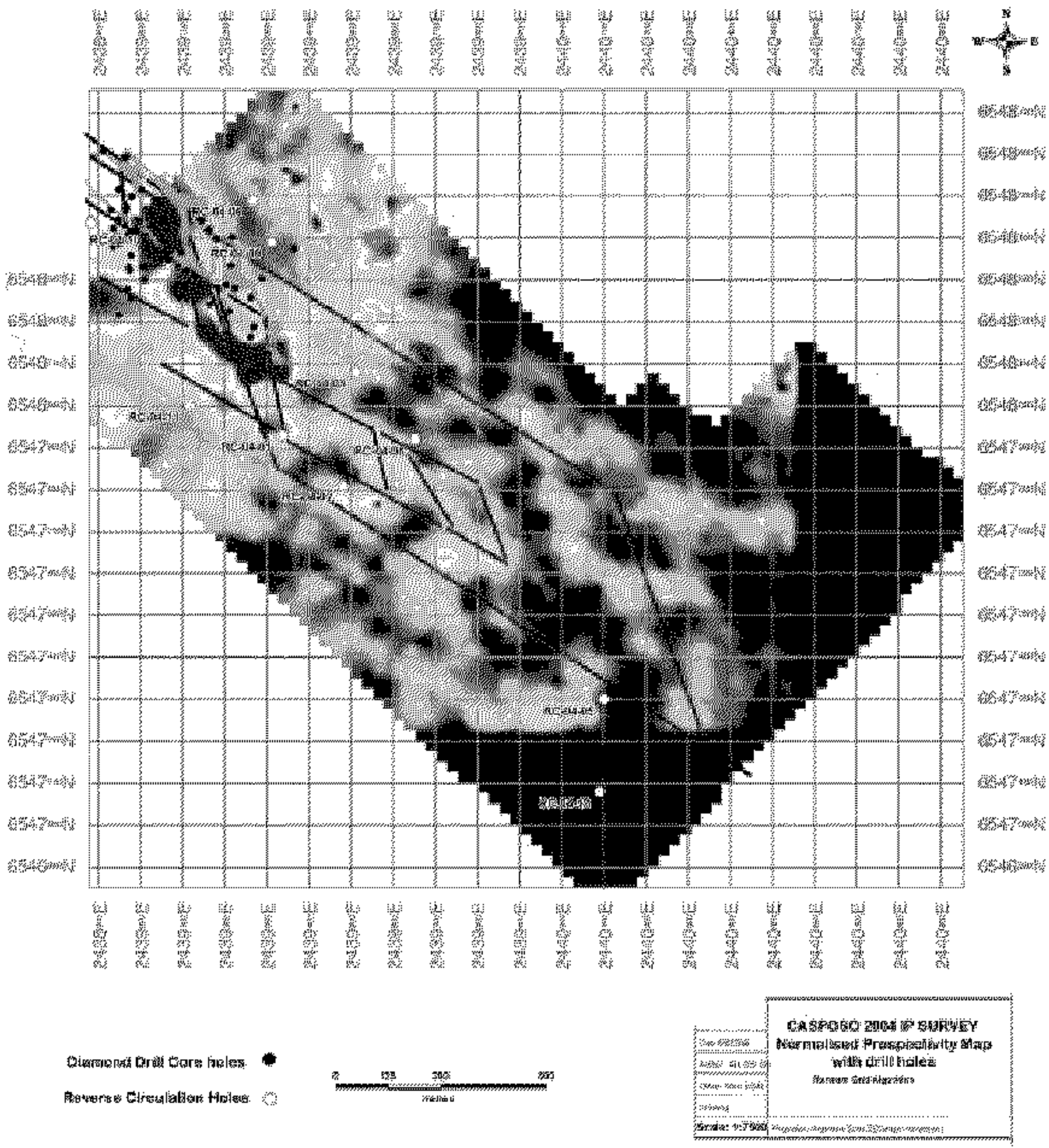
These results agree with expectations for the property. Decreased porosity resulting from silicification increases electrical resistivity. Minor quantities of sulphides increase IP chargeability. It is likely though that these properties are more related to silicified and pyritized host rock than the actual veining, hence we observe a broader anomaly than would be expected for the veins themselves. These anomalous areas extend toward the southeast. In the vicinity of Line 600E, the trend of the anomalous features changes. It appears be cross-cut by predominantly north-south trending structures.

From the IP and resistivity data one can identify a number of lineaments suggestive of structural features. These are mostly northwest-southeast trending, but some north-south and northeast-southwest trending features also occur. Further correlation of these features with the geology may show if these features are related to faulting or lithology and can play an important part in the interpretation of the control of mineralization in the area.

The identification of drill targets based on these geophysical data should follow the process of correlation of geophysics and geology. One might anticipate that relatively narrow resistive lineaments, particularly in zones of increased chargeability, are potentially mineralized structures. In covered areas, additional geological evidence might not be forthcoming without test drilling.

An IP/resistivity prospectivity plan is shown in Figure 10-1. Drill holes on the plan were current as at plan drafting in mid-2004.

Figure 10-1
IP/Resistivity Plan



Note: Figure from Intrepid Mines Limited.

10.4 Surface Sampling

While there is mention of stream sediment and rock chip sampling being undertaken by Intrepid (McGuinty, 2007), these early-stage sampling techniques have been superseded by the trenching, pitting and drilling programs and are therefore not discussed in any detail in this report.

10.5 Trenching

A summary of the trenching undertaken on the Project is given in Table 10-2. Assay results from 70 of these trenches are incorporated into the mineral resource estimate reported in Section 17. As the Intrepid database does not record which years sampling was undertaken, it is not possible to break out the data on an annual basis. BMG sampling was confined to 1998–2000.

Table 10-2
Trench Summary

Target	Area	Number of Trenches	Total Length (m)	Comment
Cerro Norte		72	1598.85	Intrepid
Kamila		22	1620	BMG
Kamila		57	915.2	Intrepid
Kamila	Mayan NW	20	79.8	Intrepid
Kamila	Inca Footwall	6	20.2	Intrepid
Mercado		15	791.27	BMG
Mercado		15	406	Intrepid
Mercado NW		10	154.3	Intrepid
Oveja Negra		26	277.1	Intrepid
Panzón		11	106.5	Intrepid
subtotal BMG		37	2411.27	BMG
subtotal Intrepid		217	3557.95	Intrepid
Total Trenches		**254**	**5969.22**	**Total**

The Intrepid Casposo database includes 217 trenches, totalling 3,558 m, of which 115 were completed at the Kamila deposit, 40 at the Mercado deposit, 11 at the Panzón prospect, and 26 in the Oveja Negra prospect. A total 22 trenches were completed by BMG at Kamila, and a further 15 at Mercado. The remaining 25 trenches were completed on prospects outside the main deposits, such as Cerro Norte.

The trenches were designed to cut the quartz veins and silicified zones. The most common orientations are southwest–northeast to southeast–northwest, although some of the trenches have general north–south orientations.

The trenches were excavated or cleaned with pick and shovel, without using explosives and sampled by chisel and hand cobbing. Trench lengths ranged from less than 1 m to 224 m, averaging 23.3 m. Individual samples within the trenches ranged from 0.1 m to 21.6 m in length and averaged 2.3 m.

10.6 Pits

No information was made available to AMEC on the pitting programs. Samples collected from all five completed pits are incorporated into the current mineral resource estimate (Section 17).

10.7 Drilling

Seven phases of diamond drilling and one phase of reverse circulation (RC) drilling have been completed on the deposit. Details for these programs are presented in Section 11 of this report.

10.8 Other Studies

10.8.1 Mineralogy (Kishar Research Inc., 2003)

Six samples in polished section from the Kamila Zone (Inca and AF veins) were examined under scanning electron microscope by personnel from Kishar Research Inc. Gangue minerals identified included ultra fine-grained–cryptocrystalline to very fine-grained quartz, fine anhedral to subhedral grains of adularia, and ultra fine-grained matrix carbonate. No bladed carbonate textures were noted.

The opaque mineral assemblage identified in the suite of six samples included translucent (low Fe) sphalerite, chalcopyrite, pyrite, lead sulphide and selenide, several sulphosalt minerals, native metal alloys that range from silver-rich electrum to native silver, silver selenide and sulphide and rare native arsenic. The native metal alloys are typically zoned with gold-rich (gold + silver) cores and mantled by more silver-rich margins.

Based on intra-mineral textures, the opaque minerals were divisible into three sub-groups or assemblages. These assemblages are:

- an early sulphide assemblage followed by
- an assemblage dominated by sulphosalts with native metal alloys and minor base metal sulphides and selenide and
- a poorly represented late stage sulphosalt + silver selenide and silver sulphide + native silver that are hosted in micro-veinlets within either of the previous two assemblages.

10.8.2 Quartz Textural Mapping (Fernandez and Belvedieri, 2005)

Vein textural mapping was undertaken on all drill core and surface vein exposures during 2003–2005 by C. Gustavo Fernandez and Irma L. Belvideri.

The study was based on the core logging database, from which the vein intervals were extracted along with their textural descriptions. The study also relied upon field observations of surface vein textures. This data was standardized and fitted to a specific quartz textural model (Morrison et al., undated), and then plotted as points on the longitudinal sections. After that, an empirical and experience based set of polygons were drawn representing the textural zone distribution. Based on this textural model, Fernandez and Belvedieri concluded that there is no indication that the middle to lower parts of the mineralizing system has been reached by the drill holes. The abundance of banded, brecciated, colloform-crustiform and bladed textures dominating the drill core to date suggested to the study authors that the middle to upper and upper sections of the mineralization system were present. This idea, they considered, was supported by the presence of stockwork at depth in drill core.

10.8.3 Fluid Inclusion Study (Townley, 2003, 2005)

Fluid inclusions contained within 25 samples of drill core from the Casposo area were evaluated for salinity and estimated depth of formation during 2003 and 2005.

The 2003 study investigated two areas, namely Kamila (B Vein, Aztec Vein, Inca Vein, SE Extension and Rosarita), and Mercado (Mercado Vein, Maya Vein, Panzón Vein and Cerro Norte). Conclusions from the report are as follows:

The results described for all studied samples evidence two phase fluid inclusion homogenization temperatures that range between 190º and 360ºC, and salinity values below 5% NaCl eq., mostly below 2% NaCl eq. The broad temperature range is in part compatible with temperatures described for epithermal precious metal systems, yet higher temperatures observed in many cases (Mercado Vein, B Vein, Aztec Vein, Inca Vein, Panzón Vein and Cerro Norte) are above or within the expected upper limit of the epithermal range. Higher temperatures could be explained in part due to over pressuring conditions associated with processes such as crack and seal flow. Such an observation is compatible with banded texture of veins, a feature common to practically all samples. It is also compatible with brecciation, and fragments of rock with banded veinlets within, indicating recurrent processes of sealing and brecciation. In some cases a lower temperature overprint associated with quartz, on higher temperature associated with calcite, is observed. This feature, coupled with banded textures, would indicate recurrent boiling conditions, telescoping and vertical migration of boiling zones, and most likely, a large vertical extent of boiling zones. Estimated depth of emplacement, based on boiling conditions under hydrostatic pressure indicate a range between 1800 m up to 400 m.

Low salinity values suggest a strong meteoric water source, and presence of adularia and moderate to low sulphidation ore minerals would indicate the Casposo Vein system as a low sulphidation epithermal system or as a Quartz-Adularia epithermal system. Very low salinity values better fit the second indicated model.

Considering the main vein system (Mercado - Kamila), the highest temperatures and deepest estimated levels of emplacement are indicated for the Mercado Vein and the B Vein. The average depth of emplacement for the Mercado Vein is 1325 m, and for the B Vein, 1250 m. In the Kamila sector, veins immediately NW (Aztec and Inca) and SE (SE Extension, Rosarita) of the B Vein (Kamila), indicate slightly lower temperatures and shallower levels of emplacement, suggesting a zonation around the core of the Kamila sector. This may be interpreted as a lateral zonation around a core zone centered on the B Vein, or as an irregular paleoisotherm with its apical point centered on the core. In the cases of the B, Inca and Aztec veins, the deeper drill core samples show lower temperatures, suggesting that current surface represents telescoped levels of the system and that most likely processes of crack and seal flow were associated with the generation of this ore zone. These observations indicate ore potential may yet remain at depth within this core zone. Zonation towards the SE Extension to lower temperature and estimated depth is carried SE towards Rosarita, where lower temperatures and shallower levels of emplacement are estimated. This could either represent a lateral vein zonation from the Kamila core zone towards the SE, or that Rosarita represents higher levels of the system, with an unknown potential at depth.

Results of the 2003 work are summarized in Table 10-3.

Table 10-3
Fluid Inclusion Data Summary, 2003

Vein System or Area	Vein	Sample Id	Th [°C] Range (Avg.)	% NaCl eq. Avg.	Estimated Depth [m]	Location
Kamila	B Vein	1960	260-370 (304)	0-4 (1.1)	1,100	Surface
		1961	320-360 (343)	0-1.5 (0.6)	1,800	Surface
		1976	240-350 (282)	0-1.5 (0.5)	850	DDH
	Aztec Vein	1958	260-300 (277)	0-2 (1.6)	750	DDH
		1962	270-340 (304)	0-2 (0.5)	1,100	Surface
	Inca vein	1959	270-320 (298)	0-2 (1.3)	1,050	DDH
		1963	190-340 (252)	0-2 (0.6)	450	Surface
	SE Extension	1977	260-320 (286)	0-4.5 (1.3)	900	Surface
	Rosarita	1970	240-360 (280)	0-2 (0.7)	800	Surface
		1540	240-290 (256)	0-1.5 (0.4)	500	Surface
Mercado	Mercado Vein	1964	260-320 (297)	0-3 (1)	1,050	Surface
		1965b	290-370 (334)	1-5.5 (2.7)	1,750	Surface
		1968	290-360 (316)	0-1.5 (0.4)	1,400	Surface
		1975	260-350 (302)	0-2 (0.7)	1,100	DDH
	Maya Vein	1972	260-320 (289)	0-1.5 (0.5)	900	Surface
	Panzón Vein	1967	270-340 (308)	0-1.5 (0.7)	1,250	Surface
	Cerro Norte	1971	260-350 (300)	0-3.5 (1)	1,100	Surface
		11971 Cal*	270-330 (316)	0-1.5 (0.6)	1,350	Surface
		11971 Qz*	230-320 (265)	0-2 (1.5)	650	
		14650*	250-290 (269)	0-3.5 (1.9)	650	Surface
		AC-44-1*	220-290 (251)	0-2 (0.8)	450	Surface

The Julieta vein system was analysed for fluid inclusion data during 2005 (Table 10-4). Fluid inclusion studies indicate that the relative depth of exposure is significantly higher than the main Casposo vein systems reviewed to date.

Table 10-4
Fluid Inclusion Data Summary Julieta Vein, 2005

Sample	Vein	Mineral	Th (°C)	NaCl% eq.	Estimated depth (m)*	Au ppm	Ag ppm
10062	Julieta	Qz(n) Qz(vf)	258 259	0.6 0.4	550 550	0.132	8.9
10065A	Julieta	Qz Cal	243 177	1.4 1.0	400 100	3.56	20.3
10065B	Julieta	Qz(n) Cal(n)	263 254	0.5 0.9	600 550	3.56	20.3
5425	Julieta	Qz(bv) Qz(mas)	312 251	0.9 0.5	1400 550	22.1	324
10052	Julieta	Qz(vg) Cal (vg) Qz(c)	306 307 296	0.6 0.7 0.7	1200 1200 1100	2.15	12.2

Note: Qz = quartz; Cal = calcite; (n) = needle or bladed texture; (vf) = veinlet filling; (bv) = banded vein; (ms) = massive sugary vein; (vg) = coarse granular grained vein; (c) = comb texture. * Estimated depth of emplacement calculated after Haas (1971).

11.0 DRILLING

11.1 Introduction

Intrepid Mines and precursor company BMG have conducted exploration activities on the Casposo Property since 1993. Core drilling (HQ diameter) and a small amount of RC drilling were completed on several zones and prospects during the period 1993–2006. A drill hole summary is included as Table 11-1 by year, and by drilling phase in Table 11-2.

Table 11-1
Drilling By Year at the Casposo Deposit (to end September 2006)

Year	Company	Drilling Type	Deposits and Prospects Drilled	No. Holes	Meterage (m)
1999	BMG	Diamond	Kamila, Mercado	26	4,337.17
2000	BMG	Diamond	Kamila, Mercado, Cerro Norte	20	4,288.97
2003	Intrepid	Diamond	Kamila, Mercado	33	4,104.30
2004	Intrepid	Diamond	Kamila	37	4,817.55
2004	Intrepid	RC	Kamila, BEXSE*, Rosarita Sur*	12	2,185
2005	Intrepid	Diamond	Kamila, Mercado, Panzon, Oveja Negra	29	5,850.90
2006	Intrepid	Diamond	Kamila, Mercado, Kamila SEXT, Mercado SE	54	7,315.15
	Totals			**211**	**32,899.04**

Note: * indicates areas drilled are outside the main Kamila deposit.

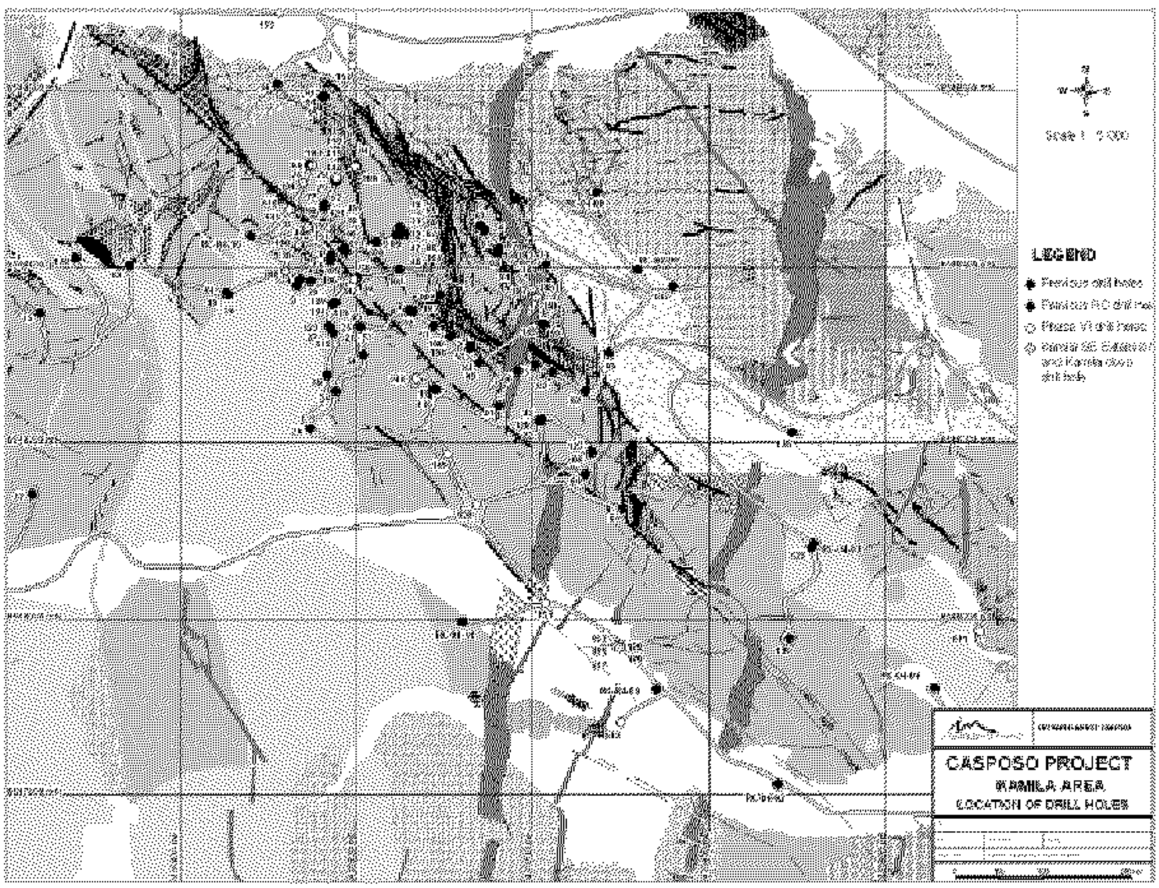
Table 11-2
Drilling Campaigns by Phase at the Casposo Deposit (to end September 2006)

Company	Phase	Drilling Type	Period	Range of Hole Numbers	No. Holes	Meterage (m)
BMG			1993–2000		46	8,626
Intrepid	I	Diamond	2002– early 2003		16	1,678
Intrepid	II	Diamond	Late 2003–early 2004		37	4,962
Intrepid	III	Diamond	Late 2004– middle 2005		39	5,972
Intrepid	IV	Diamond	Late 2005–early 2006		16	3,921
Intrepid	V	Diamond	Up to July 2006	to hole CA-06-166	12	1,053
Intrepid	VI	Diamond	July–August 2006	to hole CA-06-190	24	2,992
Intrepid	VII	Diamond	September 2006	to hole CA-06-199	9	1,511
Intrepid		RC	2004		12	2,185
Totals			**1993–2006**		**211**	**32,900**

Drilling to end September 2006 on the Project comprises 199 diamond holes (30,715 m) and 12 RC holes (2,185 m) for a combined RC and diamond total of 211 holes for 32,899,04 m. A total of 46 of these holes for 8,626 m were drilled by BMG, and 165 holes (24,274 m), including the RC, by Intrepid Mines.

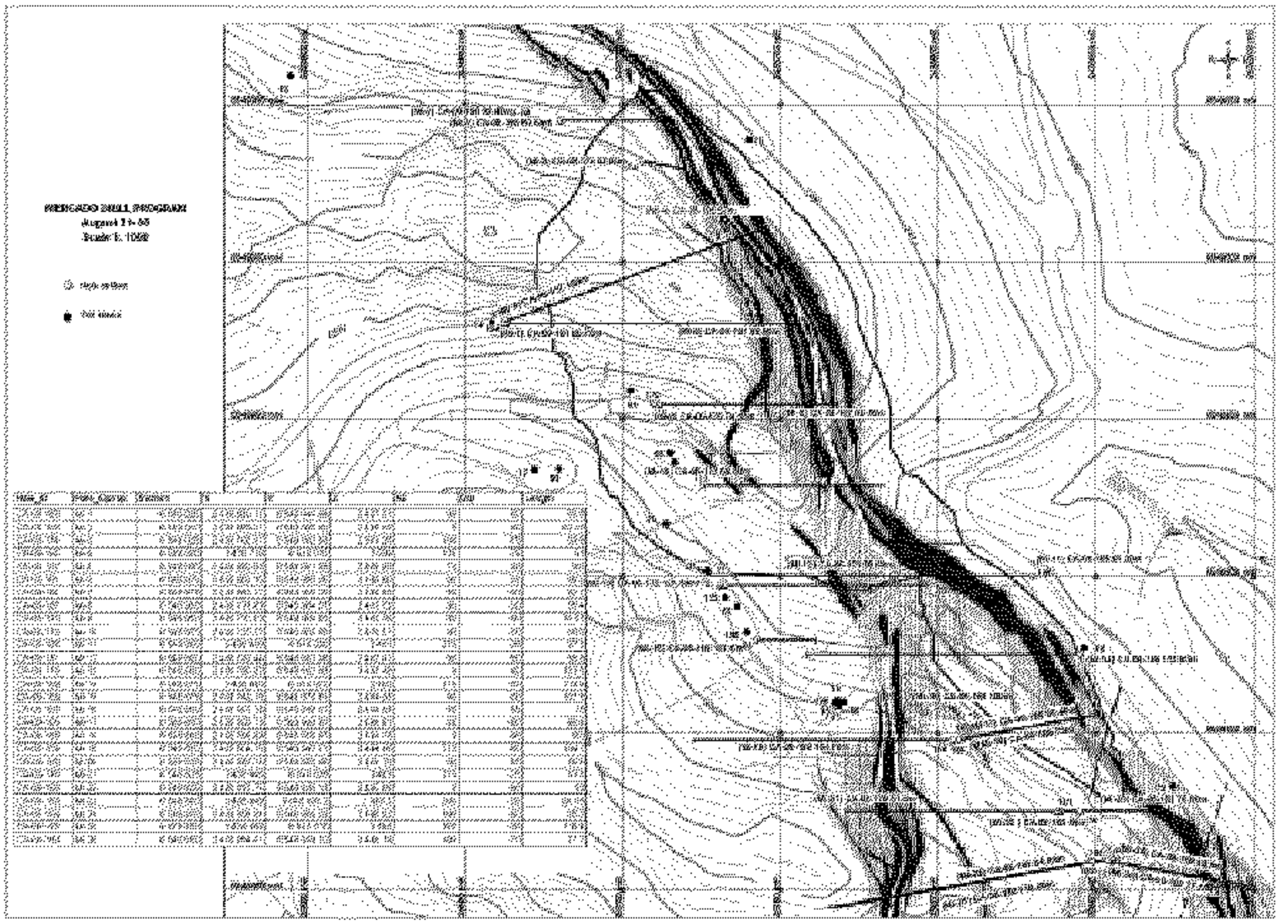
A drill hole location plan showing all holes completed to the end of the 2006 drilling program for the Kamila Zone is included as Figure 11-1, for Mercado in Figure 11-2, and for Kamila SEXT in Figure 11-3.

Figure 11-1
Drill Hole Location Plan, 2006 Kamila Drilling, Phases VI and VII



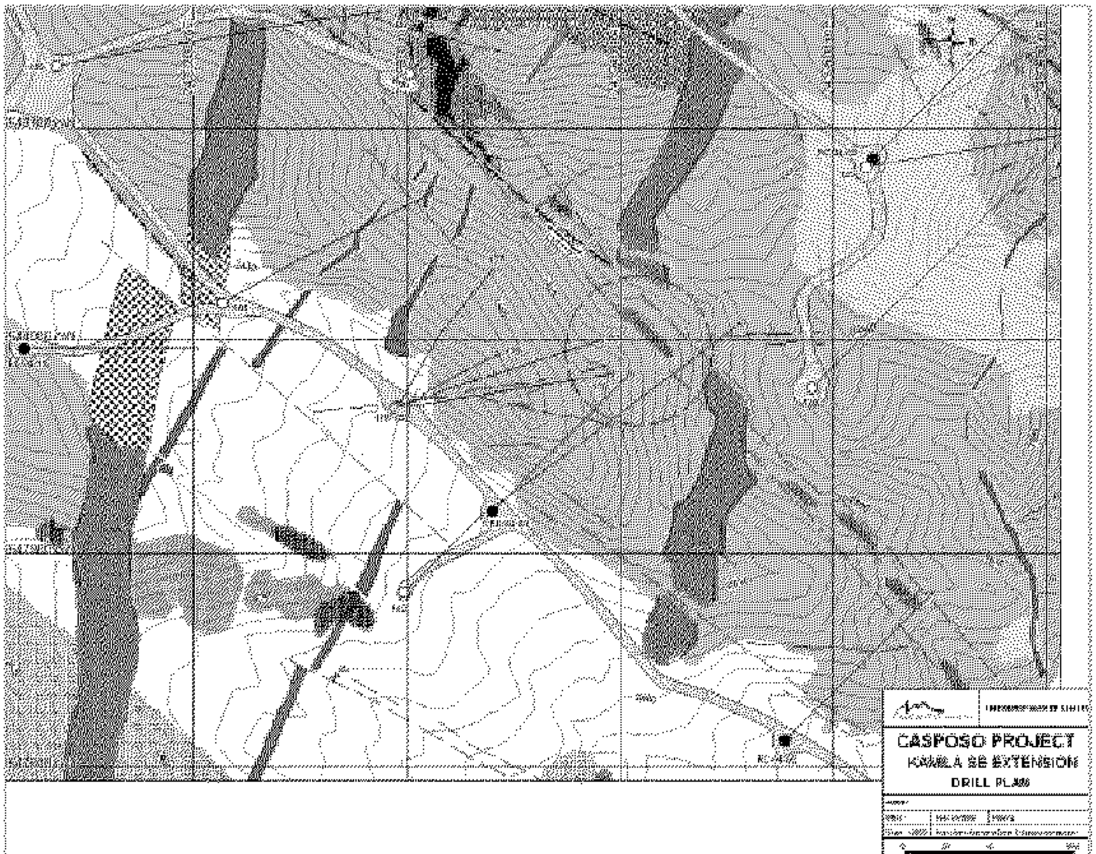
Note: Figure from Intrepid Mines Limited.

Figure 11-2
Drill Hole Location Plan, 2006 Mercado Drilling, Phase VI



Note: Figure from Intrepid Mines Limited.

Figure 11-3
Drill Hole Location Plan, 2006 Kamila SEXT Drilling



Note: Figure from Intrepid Mines Limited.

Only drill holes up to CA-06-174 were included in the database (close-off date October 10, 2006) that was incorporated into the mineral resource estimate in Section 17. Data from drill holes CA-06-175 to CA-06-199 had not been received at the time when the database was closed for estimation purposes.

The holes that were not included in the current estimation database were primarily drilled to test the Inca vein system at Kamila and Mercado. In addition, nine additional holes (CA-06-191 to 199) were drilled during September to test the Kamila SEXT prospect.

11.2 RC Drilling

Intrepid used a sole contractor, Major Drilling (Major), for RC drilling. Holes were oriented to 45°, 90°, 135°, 300°, 310°, 350° and had inclinations from -50° to -60°. Prospects and deposits tested by the RC program included Rosarita Sur, BEXSE and Kamila.

The RC hole depths ranged from 123 m to 207 m, averaging 182 m. All RC holes were drilled with a 139.7 mm (5.5") diameter drill bit.

No logging or sampling procedures were available to AMEC. The RC drilling does not inform the resource model.

11.3 Diamond Drilling

11.3.1 Battle Mountain Drilling

Between 1993 and 1999, BMG completed 8,626 m of diamond drilling in 46 holes, to approximate a 50 m x 50 m drilling grid (Puritch, 2004). Drill hole depths ranged from 75.3 m to 437.4 m, averaging 187.5 m. Drilling was completed primarily on the Kamila and Mercado deposits, but two holes tested the Cerro Norte prospect.

BMG used two contractors: Major and Connors Drilling (Connors) who completed 11 holes (1,732 m) and 35 holes (6,894 m) respectively. Most holes were east or northeast-oriented, generally normal to the strike of the silicified units, although four holes were also oriented to the north. All drill holes had 45º to 80º inclinations. The drill hole diameter was primarily NQ (47.6 mm nominal core diameter), although some holes were collared with HQ (63.5 mm nominal core diameter), and reduced to NQ for the deeper sections.

Recovery

The average core recovery was 93.7%, but the drill hole database only includes recovery data for 2,346 m, representing 27% of the total meterage.

Collar Surveys

No information was provided to AMEC for the locations of the collars for the BMG drilling. AMEC validated survey data as discussed in Sections 14.2.1 and 14.2.2.

Down Hole Surveys

Acid tests (27 holes) and the Tropari system (19 holes) were used to measure the down hole deviations. AMEC verified the survey data, as discussed in Section 14.2.2.

Logging Procedures

BMG used the following core logging procedures:

- Core was placed in well-identified, 1 m long, labelled wooden core boxes, from left to right, with the start and finish of each drill run labelled with a meterage marker.
- Core boxes were closed and regularly transported to core logging facility, and laid out in order of increasing hole depth.
- Core box labels and meterage were checked for accuracy and core was photographed and printed in paper format.
- Specially designed forms including general data, as location, date drilled, diameter, down hole deviation, etc., were used for logging.
- Geological data recorded included lithology (usually succinct), alteration, veining, mineralization, structures, and references about the oxide/sulphide boundary, all numerically or alphanumerically coded.
- Initially, the geotechnical data were not systematically recorded. Later, recovery and RQD were documented for all BMG holes from CA-00-30 forward.
- Information from the drill logs was hand entered into BorSurv, a special logging program, using a single-entry procedure.

Drill hole geological data from BMG are available on site as descriptive logs recorded on A4 paper sheets. BMG core is currently stored at site, along with the Intrepid core.

11.3.2 Intrepid Mines Drilling

Between the acquisition date of the property in 2002 and July 2006, Intrepid completed 17,586 m of diamond drilling in 120 holes (up to drill hole CA-06-166), in addition to 2,185 m of RC drilling in 12 holes. Intrepid used Connors for the majority of the diamond drilling programs in that period. Between July and September 2006, Intrepid drilled 33 additional drill holes (drill holes CA-06-167 to CA-06-199). Connors and another contractor, Bolland Drilling, were used for drilling during the July September 2006 campaign.

Drill holes after CA-06-174 are not included in the resource estimation supporting the Feasibility Study (drill holes CA-06-175 to CA 06 199), as the assays for those drill holes had not been received at the time the database was closed off for resource estimation purposes on October 10, 2006.

Most diamond drill holes were oriented to the east (ranging from northeast to southeast), generally normal to the strike of the silicified units, although six holes were also oriented to the west and one hole to the north. All holes used in the Feasibility Study resource estimate have 45° to 80° inclinations. Diamond drill hole depths ranged from 50 m to 409.8 m, averaging 146.6 m. The diamond drill hole diameter was primarily HQ (the NQ diameter was used only in three holes, namely CA-03-38, CA-05-133 and CA-05-134). All Intrepid core has been drilled with the HQ-3 triple tube method to ensure minimum core rotation and maximum sample recovery, with the exception of two holes, at Panzón, in 2005, which were drilled as NQ-size.

Recovery

The drill hole database includes recovery data for 10,126 m, representing 57% of the total Intrepid diamond drilling meterage. The average core recovery is 93.1%.

Collar Surveys

No information was provided to AMEC regarding collar survey methodologies. AMEC validated collar locations as discussed in Sections 14.2.1 and 14.2.2.

Down Hole Surveys

The Tropari system was used to measure the down hole deviations in 13 holes, whereas the Sperry Sun method was used for the remainder of the holes. The values were read on paper and then introduced into the logging forms and the database. AMEC verified the survey data as discussed later in Section 14.2.2.

Logging Procedures

Intrepid used the following core logging procedures:

- Core was placed in well identified, 1 m long, wooden core boxes, from left to right, with the start and finish of each drill run labelled with a meterage marker;
- Core boxes were closed and regularly transported to core logging facility, and laid out in order of increasing drill hole depth;
- Core box labels and meterage were checked for accuracy and core was photographed in digital format by a company geologist;
- Specially designed forms were used for logging. These included general header data, such as location, date drilled, core diameter, down hole deviation, etc;

- Geological data recorded included lithology, structures, alteration, and mineralization. A set of alphanumeric codes synthesize the geological data;
- Geotechnical data recorded included RQD and recovery, as well as coded hardness, weathering and various fracture data;
- Information from the drill logs was hand-entered into Excel files using a single-entry procedure.

Intrepid drill hole geological data are available on site as descriptive logs recorded on paper sheets.

Once core was logged and sampled, it was stacked by drill hole in an enclosed warehouse or tarped area outside of the warehouse. All boxes are stored with lids.

12.0 SAMPLING METHOD AND APPROACH

12.1 Introduction

Sampling programs at the Casposo Project have included drill core samples, RC samples and various geochemical samples comprising surface rock chip, soil and stream sediment sampling (surface sampling), trench and pit sampling (channel sampling), The surface sampling has been superseded by the drilling, trenching and pitting samples and are therefore not discussed in detail in this report.

No detailed information was available for AMEC to review for the logging, sample collection and sample preparation protocols for the BMG exploration programs.

Intrepid established detailed logging, sample collection, and sample preparation protocols for core and RC sampling, and implemented procedures for the collection of geotechnical data.

AMEC was on site during a portion of the core and RC drilling and was able to observe the procedures for those activities.

Table 12-1 summarizes the sample database by sample type (for those samples used for resource estimation). Samples from drill holes CA-06-175 to 199, which were not included in the resource estimation (refer to Section 11.0), are not reported in the table.

Table 12-1
Sample Types at the Casposo Deposit

Type	Number	Total Length (m)	Percentage	Average Length (m)
Surface	1,734	----		
Channel	1,730	4,054	22%	2.34
Core BMG	5,333	6,463	35%	1.21
Core Intrepid	5,233	6,671	36%	1.27
RC Intrepid	816	1,190	6%	1.46

12.2 Surface Sampling Procedures

Soil and stream sediment samples were dried and screened with an 80# sieve, after which a 100 g sub-sample was split, bagged and sent to the lab. The rock chip samples were collected with hammer and chisel, and consisted of approximately 1 cm diameter fragments, taken from an area of influence of about 2 m in diameter. The average weight was 3 kg.

12.3 Channel Sampling

Continuous channel sampling was conducted in trenches with chisel and hammer, usually at the bottom of the excavations. The average length was 2.38 m, but about 85% of them were 1 m to 4 m long, and nearly 12% exceeded 5 m, sometimes reaching 10 m (some BMG trenches). The average weight was 3 kg to 5 kg.

12.4 RC Sampling Procedures

RC samples were collected from the cyclone every 1 m, then homogenized and split twice, to obtain a 3 kg to 5 kg sample. Another split of the sample was stored as backup. The remaining reject was discarded.

12.5 Diamond Sampling Procedures

Core was split in half with a mechanical splitter (BMG) or diamond saw (Intrepid). One half of the core was sent for analysis and the remaining half returned to the core box in its original orientation as a permanent record. Normally, the entire hole was sampled. The sample interval was usually 1 m to 2 m for BMG, and 0.5 m to 2 m for Intrepid (maximum 1.5 m in mineralized zones). Highly fragmented core was bound with adhesive tape before cutting, but currently all core is taped, even when compact.

13.0 SAMPLE PREPARATION, ANALYSES, AND SECURITY

13.1 Sample Preparation

Sample preparation methods for rock chip and stream sediment sampling are not discussed in this report, as the data are not used in resource estimation.

13.1.1 BMG

BMG used ALS Geolab in Mendoza as the primary lab. However, there were no detailed references in the BMG data available to AMEC in regards to preparation and assay procedures.

13.1.2 Intrepid

Intrepid used ALS Chemex (in La Serena, Chile) as primary laboratory for most of the sampling programs, and Alex Stewart (in Mendoza, Argentina) as the secondary laboratory. The samples were bagged in large sacks holding 10 samples each. ALS Chemex collected the samples at the camp, and transported them to Mendoza with their own vehicle

Samples were prepared at the preparation facility in Mendoza. The preparation protocol consisted of:

- drying,
- crushing to 85%-10#,
- splitting and pulverization of 1,000 g to 85% passing 200 mesh, and
- separation of two bags of pulp with approximately 200 g each
- pulps were forwarded directly by ALS Chemex to their main laboratory in Chile.

Starting from drill hole 148 (February 2005), Intrepid switched to Alex Stewart (in Mendoza) as the primary laboratory. During this period, no check samples were sent to a secondary laboratory. The sample preparation protocol at Alex Stewart was similar to the protocol used by ALS Chemex.

13.2 Analyses

13.2.1 BMG

The samples were assayed for Au (FA, method PM209), Ag, Pb, Zn, Mo, Cu, As, Sb (AAS, method G105) and occasionally for Hg (method G008).

13.2.2 Intrepid

A brief description of ALS Chemex assay methods is presented in Tables 13-1 and 13-2. A brief description of Alex Stewart assay methods is presented in Tables 13-3 and 13-4. All Intrepid pulps are stored on site.

Table 13-1
ALS Chemex Assay Methods used for the Casposo Project

Element	Method Code	Brief Description	Range (ppm)
Au	Au-AA24	Au by FA and AAS finish 50 g nominal sample weight	0.005 – 10.000
	Au-GRA22 (for samples with Au>10 ppm)	Au by FA and gravimetric finish 50 g nominal sample weight	0.05 – 1,000.00
Ag, Al, As, Ba, Be, Bi, Ca, Cd, Co, Cr, Cu, Fe, K, Mg, Mn, Mo, Na, Ni, P, Pb, S, Sb, Sr, Ti, V, W, Zn	ME-ICP61	27 elements by four acid digestion and ICP-AES	(Table 13-2)
Ag	Ag-AA63	Ore grade Ag – four acid digestion and AAS	1 – 100
	Ag-GRA22 (for samples with Ag>100 ppm)	Ag by FA and gravimetric finish	5 – 3,500
Hg	Hg-CV41	Trace Hg – cold vapour/AAS	0.01 – 100.00

Table 13-2
Analytical Ranges of the ME-ICP61 Method (ALS Chemex)

Elements and Ranges (ppm, %)							
Ag	(0.5 - 100)	Cd	(0.5 - 500)	Mn	(5 - 10,000)	Sb	(5 - 10,000)
Al	(0.01% - 25%)	Co	(1 - 10,000)	Mo	(1 - 10,000)	Sr	(1 - 10,000)
As	(5 – 10,000)	Cr	(1 - 10,000)	Na	(0.01% - 10%)	Ti	(0.01% - 10%)
Ba	(10 - 10,000)	Cu	(1 - 10,000)	Ni	(1 - 10,000)	V	(1 - 10,000)
Be	(0.5 - 1000)	Fe	(0.01% - 25%)	P	(10 - 10,000)	W	(10 - 10,000)
Bi	(2 - 10,000)	K	(0.01% - 10%)	Pb	(2 - 10,000)	Zn	(2 - 10,000)
Ca	(0.01% - 25%)	Mg	(0.01% - 15%)	S	(0.01% - 10%)		

Table 13-3
Alex Stewart Au and Ag Assay Methods used for the Casposo Project

Element	Method Code	Brief Description	Range (ppm)
Au	Au4-50	Au by FA and AAS finish 50 g nominal sample weight, final volume 10 ml	0.01 – 10.00
	Au4A-50 (for samples with Au>10 ppm)	Au by FA and gravimetric finish 50 g nominal sample weight	0.1 – 1,000.0
Ag	GMA	Four-acid digestion, AAS reading (for check samples up to February 2006)	0.5 – 200.0
	ICP-AR-39	Aqua regia digestion, ICP-OES reading (for ordinary samples after February 2006)	0.5 – 200.0
	Ag4A-50 (for samples with Ag>200 ppm up to February 2006; for all samples in mineralized intersections after February 2006)	Ag by FA and gravimetric finish	1 – 5,000

Table 13-4
Analytical Ranges of the ICP-AR-39 Method (Alex Stewart)

Elements and Ranges (ppm, %)							
Ag	(0.5 – 200.0)		(0.01% - 10.00%)		(1 – 10,000)	Ta	(10 – 1,000)
Al	(0.01% - 10.00%)	Ga	(2 – 2,000)	Ni	(1 - 10,000)		(10 – 2,000)
As	(5 – 10,000)	Hg	(2 – 500)	P	(10 - 10,000)	Ti	(0.01% - 5.00%)
Ba	(2 - 2,000)	K	(0.01% - 10.00%)	Pb	(2 - 10,000)	Tl	(5 – 1,000)
Bi	(5 - 2,000)	La	(1 – 2,000)	S	(0.01% - 10.00%)	V	(1 - 10,000)
Ca	(0.01% - 10.00%)	Li	(1 – 10,000)	Sb	(5 - 2,000)	W	(20 - 2,000)
Cd	(1 – 2,000)	Mg	(0.01% - 10.00%)	Sc	(5 – 2,000)	Y	(1 – 2,000)
Co	(1 - 10,000)	Mn	(1 - 20,000)	Se	(10 – 2,000)	Zn	(2 - 10,000)
Cr	(1 - 10,000)	Mo	(1 - 10,000)	Sn	(20 – 2,000)	Zr	(1 – 5,000)
Cu	(1 - 10,000)	Na	(0.01% - 5.00%)	Sr	(1 – 2,000)		

13.3 Bulk Density Measurements

Intrepid conducted a limited bulk density sampling program on the Casposo Project, which produced 94 samples from 36 holes. The bulk density samples consisted of half core pieces, 10 cm to 15 cm long, which were taken from quartz veins and silicified breccias (47), andesites (28) and rhyolites (19). The bulk density statistics for Casposo are presented in Table 13-5.

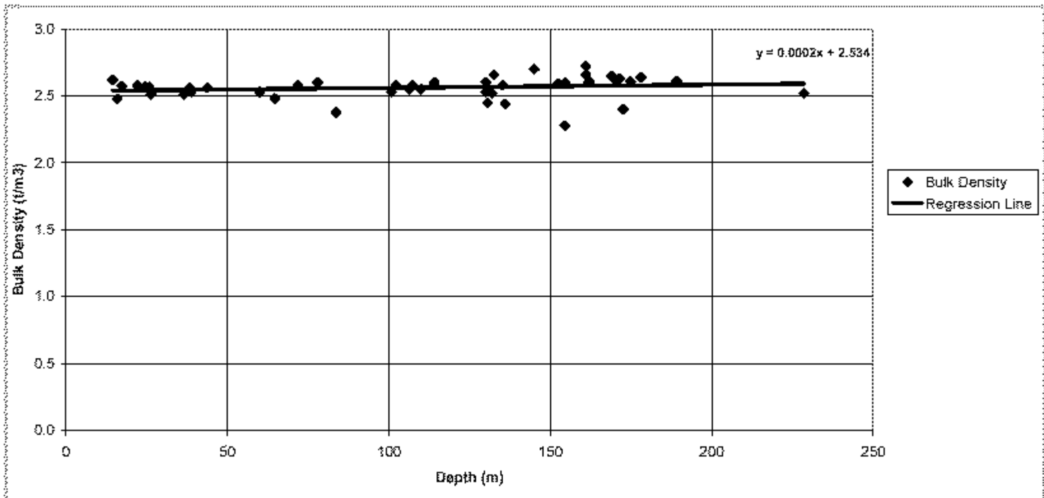
Table 13-5
Bulk Density Statistics at Casposo (from drill hole samples)

Parameter	Quartz Veins	Andesites	Rhyolites
Ave (t/m^3)	2.56	2.69	2.59
SD (t/m^3)	0.08	0.05	0.08
CV	3.2%	1.7%	3.1%
Median (t/m^3)	2.58	2.69	2.57
Mode (t/m^3)	2.53	2.69	2.58
Count	47	28	19
Min (t/m^3)	2.28	2.58	2.48
Max (t/m^3)	2.72	2.79	2.73

Drill hole bulk density samples were sent to ALS Chemex facilities in Mendoza. AMEC visited the ALS Chemex Mendoza preparation facilities on February 21 and 22, 2005, and reviewed the bulk density determination procedure. Bulk density determinations at ALS Chemex are conducted with an SB16001, Delta Range, Mettler Toledo balance, adapted for this kind of measurement. This balance handles weights from 5 g to 16,100 g, with 0.1 to 1.0 g precision. The samples are dried and covered by paraffin before being tested. The standard water displacement method is used, by measuring the sample mass in air (Ma) and submerged in water (Mw).

As observed in Table 13-5, bulk density at Casposo exhibits a low coefficient of variation (CV), calculated as the percent ratio between the standard deviation and the average bulk density (3.4%). Bulk density of quartz veins ranges within a relatively narrow dispersion interval, from 2.28 t/m3 to 2.72 t/m^3, and similar trends can be observed for andesites and rhyolites. The depth of the quartz vein bulk density samples ranged from 14.5 m to 228.5 m, but no significant variation with depth was observed (Figure 13-1).

Figure 13-1
Quartz Vein Bulk Density versus Depth



13.4 Quality Assurance and Quality Control (QA/QC)

13.4.1 BMG QA/QC Program

BMG only had a very limited QA/QC program in place during their drill program, consisting of the insertion of 16 standards over the duration of the sampling campaign. BMG did not insert twin samples, coarse duplicates or blanks in the sample batches. Occasionally, some pulp duplicates appear to have been assayed (Irma Belvedieri, technical communication, December 2005), but the results are not identified in the database. However, there are no other references in the data made available to AMEC for the BMG QA/QC program.

13.4.2 Intrepid QA/QC Program

The QA/QC program implemented by Intrepid for the Casposo Project from 2003 to 2005, included the insertion of control samples to monitor assay accuracy (standards) and contamination (coarse blanks). Sampling, sub-sampling and assay precisions were not assessed, since the QA/QC program lacked the insertion of twin samples, coarse duplicates and pulp duplicates.

13.5 Sample Security

The chain-of-custody for core samples collected and being shipped from site is as follows:

- Core is transported to the camp either by the company geologists, company technicians or alternatively by the drill contractors and placed in the core logging area.
- Core boxes are brought to the village of Calingasta to a Company-owned compound where the sample intervals are rechecked, recoveries are noted and core is photographed. Sampling takes place in a fenced compound using a diamond saw.
- Prepared samples are placed in plastic sample bags with a sample tag; the tops of the bags are folded over several times and stapled shut. Bags are then combined in numerical sequence in rice bags; the number of samples varies with total weight ranging to a maximum of about 25 kg.
- Each rice bag has the company name and has a list of samples written on the outside
- A sample submission form accompanies each shipment, which are transported to the assay laboratory in trucks operated by laboratory employees or contractors. Intrepid notifies the laboratory prior to each shipment going out.
- Assays and analyses are sent electronically by the lab to a preset list of recipients with final paper and electronic certificates sent to Intrepid's San Juan office.

14.0 DATA VERIFICATION

14.1 QA/QC

14.1.1 Intrepid QA/QC Program

AMEC had access to the results of 619 control samples (standards, coarse blanks and check samples), which accounted for approximately 10.6% of all core samples collected and assayed by Intrepid. The insertion frequency was as follows: 3–4 coarse blanks and 5–6 standard samples per one hundred samples included in the submission batch. The actual frequency of the check samples is not clear, since such samples were randomly chosen. Up to August 2005, the check samples were re-submitted for assaying to Alex Stewart Mendoza, the secondary laboratory. However, after that date no check samples were submitted. A summary of Intrepid's control samples available to AMEC is presented in Table 14-1.

Table 14-1
Summary of Intrepid QAQC Program

Type of Sample	Description	Number of Samples
Sampling	Total number of samples	5,821
	comprising	
	Control samples	619 (10.6%)
	of which:	
	Coarse blanks	192 (3.3%)
	Standards	295 (5.1%)
	Check samples	128 (2.2%)

14.1.2 AMEC Evaluation Procedure

The standard procedure followed by AMEC for evaluation of the Intrepid QA/QC results involved the following steps:

Duplicate Samples

The duplicate samples were evaluated according to the Hyperbolic Method. For evaluating the twin and duplicate samples, Min–Max plots were constructed for all studied elements, where the maximum and minimum values of the sample pairs were plotted in the y and x axis, respectively. This way, all the points were plotted above the x = y line. The failure line was plotted according to the formula y2 = m2x2 + b2. Sample pairs plotting above this line were considered failures. An acceptable level of precision is achieved if the failure rate does not exceed 10% of all pairs.

Standard Samples

For evaluating the standard samples, control charts were constructed for each standard and for each documented element. The values reported for the inserted standard samples were plotted in a pseudo-time sequence (by drill hole number, since the certificate dates were not reported). Lines corresponding to *BV, 1.05*BV+CI, 0.95*BV-CI* and *AV±2*SD* were also plotted (*BV, CI*: Best Value and Confidence Interval at the 95% confidence level, respectively, calculated as a result of round-robin tests; *AV, SD*: average value and standard deviation, respectively, calculated from the actual assay values of the inserted standards).

In principle, the standard values should lie within the *AV±2*SD* boundaries to be accepted. Otherwise, these values are qualified as outliers. However, isolated values within the *AV±3*SD* limits were also accepted. The analytical bias was calculated as:

Bias (%) = (AV_{eo} / BV) – 1

where AV_{eo} represents the average recalculated after the exclusion of the outliers. The bias values are assessed according to the following ranges: good: between -5% and +5%; reasonable, with care: from -5% to -10% or from +5 to +10%; unacceptable: below -10% or above 10%.

Accuracy plots (Mean versus Best Value) were constructed, in addition to the control charts, for all the standards and studied elements. Through the accuracy plots, the overall element bias was calculated, taking into consideration the results of all the standards used for each element during the extent of the program. In this case, the overall bias (*OABías*) of ALS Chemex for each element was calculated as:

OABias (%) = RLS – 1

where RLS is the slope of the linear regression line of the Mean versus Best Value for each standard and element.

Coarse Blank Samples

Coarse blank versus preceding sample plots were prepared, which allow the identification of possible incidents of cross-contamination during preparation.

Check Samples

For evaluating the check samples, Reduction-to-Major-Axis (RMA) plots were constructed for the studied elements. The RMA method offers an unbiased fit for two sets of pair values (original samples and check samples) that are considered independent from each other. In this case, the bias of the primary laboratory for each element as compared to the secondary laboratory is calculated as:

Bias (%) = 1- RMAS

where *RMAS* is the slope of the RMA regression line of the secondary laboratory values versus the primary laboratory values for each element.

14.1.3 Analysis of Detailed Results

Practical Detection Limits

Practical detection limits for the studied elements could not be independently estimated, since no pulp duplicate assays were available. Therefore, the practical detection limits were assumed to be as indicated by the laboratory.

Standard Samples (SS)

BMG Exploration

BMG used two standards (40410 and Bor-3) at the end of the exploration program, both of them obtained from an unknown source. The certified values of the standards are presented in Table 14-2.

Table 14-2
Statistics of Standard Samples (ALS Chemex)

Company	Standard	Au		Ag		No. of Samples (outliers)	Mean (ppm)		Bias (%)	
		BV (ppm)	CI (ppm)	BV (ppm)	CI (ppm)		Au	Ag	Au	Ag
BMG	40410	2.25				5	**2.19**		**-2.8**	
	Bor-3	0.75				11	**0.75**		**-0.6**	
Intrepid	OREAS 6Ca	1.48	0.06			10	**1.34**		**-9.3**	
	OREAS 6Pa	1.65	0.04			5	**1.61**		**-2.2**	
	OREAS 6Pb	1.425	0.026			20	**1.46**		**2.7**	
	OREAS 7Ca	2.54	0.08			10	**2.34**		**-7.8**	
	OREAS 7Pa	3.00	0.06			15	**3.12**		**3.9**	
	OREAS 17Pb	2.56	0.06			63	**2.69**		**4.9**	
	OREAS 50P	0.727	0.021			21	**0.75**		**2.5**	
	OREAS 61Pb	4.75	0.06			50	**4.98**		**4.8**	
	OREAS 61Pb			9.0	0.3	47		**8.98**		**-0.3**
	OREAS 62Pb	11.33	0.17			37 (1)	**10.70**		**-5.6**	
	OREAS 62Pb			21.5	0.537	39		**19.54**		**-9.1**
	40410	2.25				26	**2.22**		**-1.4**	
	Bor-3	0.75				9	**0.72**		**-3.9**	

In total, 16 standard samples were assayed by ALS Geolab. AMEC prepared control charts for Au and for the two standards. No outliers were identified.

The Au accuracy was good for both standards: -2.8% bias for standard 40410, and -0.6% bias for Bor-3. Unfortunately, the standards were introduced only at the end of the program, for which the laboratory accuracy remains insufficiently assessed.

Intrepid Exploration (ALS Chemex)

Intrepid used 11 certified standards until early 2006 in the sample batches submitted to ALS Chemex. Nine standards were acquired from Ore Research & Exploration Pty Ltd., whereas two of them (40410 and Bor-3) are from an unknown source. The certified values of the standards are presented earlier in Table 14-2.

In total, 268 standard samples were assayed by ALS Chemex. AMEC prepared control charts for all the standards, and Ag control charts for two standards, OREAS 61Pb and OREAS 62Pb. Only one outlier was identified for Au for standard OREAS 62Pb, which represents a very low rate of failing samples.

The Au accuracy was generally good for most standards (Table 14-2), with the exception of standards OREAS 6Ca (-9.3% bias) and OREAS 7Ca (-7.8% bias), which were accepted with care, since other standards corresponding approximately to similar values had acceptable accuracies. The Ag accuracy for standard OREAS 62Pb was very poor (negative), close to the unacceptable boundary (-9.1% bias); however, considering the low value of the standard (21.9 ppm Ag), AMEC does not consider this to be a definitive assessment of the Ag accuracy.

Accuracy plots, which plot the Average Values versus the Best Values, were also prepared for Au. Through the accuracy plots, the overall element bias is calculated, taking into consideration the results of all the standards used for Au during the extension of the program. The overall Au accuracy was acceptable (-3.0% bias). On the other hand, Ag exhibited an overall poor accuracy (8.3% bias), close to the unacceptable limit of 10%. However, considering that the two Ag standards have very low values, AMEC does not judge this as a definite assessment of the overall Ag accuracy.

On the basis of these results, AMEC concludes that the Au accuracy at ALS Chemex during the Intrepid exploration campaign was acceptable. However, AMEC could not properly assess the Ag accuracy, due the low Ag values of the inserted standards.

Intrepid Exploration (Alex Stewart)

Intrepid used five certified standards in 2006 in the sample batches submitted to Alex Stewart, all of them acquired from Ore Research & Exploration Pty Ltd. The certified values of the standards are presented in Table 14-3.

Table 14-3
Statistics of Standard Samples (Alex Stewart)

Company	Standard	Au		Ag		No. of Samples (outliers)	Mean (ppm)		Bias (%)	
		BV (ppm)	CI (ppm)	BV (ppm)	CI (ppm)		Au	Ag	Au	Ag
Intrepid	OREAS 6Pb	1.425	0.026	6	**1.37**	6	**1.37**		**-4.2**	
	OREAS 7Pa	3.00	0.06	2	**3.07**	2	**3.07**		**2.3**	
	OREAS 7Pb	-	-	4	**2.58**	4	**2.58**		**?**	
	OREAS 50P	0.727	0.021	6	**0.72**	6	**0.72**		**0.8**	
	OREAS 61Pb	4.75	0.006	9	**4.73**	9	**4.73**		**-0.4**	

In total, 27 standard samples were assayed by Alex Stewart. AMEC prepared Au control charts for all the standards. No outliers were identified. The Au accuracy was acceptable for all standards (Table 14-3).

Accuracy plots were also prepared for Au. The overall Au accuracy was acceptable (0.1% bias). On the basis of these results, AMEC concludes that the Au accuracy at Alex Stewart during the Intrepid exploration campaigns was acceptable.

Coarse Blanks (PB)

Intrepid used coarse blank material to evaluate assay contamination during preparation. Coarse Blank versus Previous Sample plots were prepared for the studied elements for assays produced at ALS Chemex. The analysis of 165 coarse blank samples showed some minor cross-contamination for Au in two samples, although none of them exceeded 20 times the detection limit.

Another 27 coarse blank samples were assayed at Alex Stewart in 2006. All of them yielded values below the detection limits. Therefore, no obvious Au and Ag cross contamination was identified during sample preparation at ALS Chemex and Alex Stewart.

Check Samples (CS)

In total until mid-2005, Intrepid sent 128 check samples to Alex Stewart in Mendoza for external control. The samples were assayed for Au and Ag. To the best knowledge of AMEC, the check sample batches did not include internal control samples.

AMEC prepared RMA plots for Au and Ag. The RMA statistics are presented in Table 14-4. After excluding a few outliers (two for Au, and one for Ag), both plots indicate a good fit between the check assays and the original assays, reflected in the high values of the coefficient of determination R^2 (ranging between 1.000 and 0.999), as well as acceptable Au and Ag bias values of ALS Chemex as compared to Alex Stewart (0.5% and -0.4%, respectively).

Table 14-4
Accuracy of ALS Chemex Relative to Alex Stewart for Au and Ag on the Basis of Check Assays

Casposo Exploration Project - RMA Parameters - All Samples								
Element	R^2	N (total)	Pairs	m	Error (m)	b	Error (b)	Bias*
Au (ppm)	0.996	128	128	1.004	0.009	0.007	1.083	**-0.4%**
Ag (ppm)	0.999	128	128	1.002	0.005	-3.033	11.669	**-0.2%**

Casposo Exploration Project – RMA Parameters - No Outliers								
Element	R^2	Accepted	Outliers	M	Error (m)	b	Error (b)	Bias*
Au (ppm)	1.000	126	2	0.995	0.002	-0.010	0.264	**0.5%**
Ag (ppm)	0.999	127	1	1.004	0.004	-1.573	7.725	**-0.4%**

* Bias of ALS Chemex relative to Alex Stewart.

Granulometric Checks

There are no references in the data made available to AMEC as to whether any granulometric (sieve) checks were conducted by BMG or Intrepid on the Casposo Project on reject or pulp samples.

14.1.4 AMEC Independent Sampling

During the two site visits to the Casposo Project, AMEC conducted independent sampling programs, which consisted of collecting twin core check samples from the BMG and the Intrepid programs, and pulp check samples from the Intrepid program. BMG twin check samples were only taken during the 2005 site visit, and the pulps from the BMG program were not available. The twin and pulp check samples are equivalents of the above defined twin and check samples, respectively, re submitted in this case to a third certified laboratory. These samples are used to obtain an independent estimation of the overall accuracy.

These check samples were resubmitted by AMEC to ACME laboratory in Santiago. The 2005 program included samples assayed until July 2005 at ALS Geolab for BGM and at ALS Chemex for Intrepid. The 2006 program included Intrepid samples assayed between July 2005 and August 2006 at ALS Chemex and Alex Stewart.

The check sample batches also included reasonable proportions of pulp duplicates of the samples included in the batch, as well as standard samples and pulp blanks. These samples are necessary to assess the precision, accuracy and possible assay contamination, respectively, at the third laboratory. A summary of AMEC´s re-sampling programs is presented in Table 14-5. A brief description of ACME assay methods is presented in Table 14-6.

Table 14-5
Summary of AMEC´s Re-sampling Programs

Date of Site Visit	Type of Sample	Campaign	Primary Laboratory	Number of Samples (% of Samples in Mineral Intersections)
December 2005	Twin Check Samples	BMG Intrepid		12 (7.5%) 24 (4.5%)
	Pulp Check Samples	Intrepid	ALS Geolab ALS Chemex	39 (7.4%)
	Pulp Duplicates Standards Pulp Blanks		ALS Chemex	4 5 2
August 2006	Twin Check Samples	Intrepid		9 (9.1%) 11 (8.9%)
	Pulp Check Samples	Intrepid	ALS Chemex Alex Stewart	10 (10.2%) 10 (8.9)
	Coarse Duplicates Pulp Duplicates Standards Pulp Blanks		ALS Chemex Alex Stewart	1 2 1 2

Table 14-6
ACME Assay Methods used for the Casposo Project

Element	Method Code	Brief Description	Range (ppm)
Au	Group 6-Au	Au by FA and AAS finish 29.2 g nominal sample weight	0.01 – 10.00
	Group 6-Au (for samples with Au>10) ppm)	Au by FA and gravimetric finish	0.1 – 1,000.0
Ag	Group 7TD	Four-acid digestion, AAS reading	0.5 – 200.0
	Group 6-Ag (for samples with Ag>100 ppm)	Ag by FA and gravimetric finish	1 – 5,000

Twin Check Samples (2005 Re-sampling Program)

BMG Exploration

AMEC prepared RMA plots for Au and Ag. The RMA statistics are presented in Table 14-7.

Table 14-7
RMA Parameters - AMEC Re-sampling Program (2005)

RMA Plot-Twin Check Samples / BGM Exploration – RMA Parameters - All Samples								
Element	**R^2**	**N (total)**	**Pairs**	**m**	**Error (m)**	**b**	**Error (b)**	**Bias**
Au (ppm)	0.881	12	12	0.942	0.129	-0.077	0.464	**5.8%**
Ag (ppm)	0.906	12	12	1.729	0.211	-28.707	24.353	**-72.9%**
RMA Plot-Twin Check Samples / BGM Exploration – No Outliers								
Element	**R^2**	**Accepted**	**Outliers**	***m***	**Error (*m*)**	***b***	**Error (*b*)**	**Bias**
Au (ppm)	0.979	10	2	1.015	0.023	-0.037	0.058	**-1.5%**
Ag (ppm)	0.975	9	3	0.961	0.024	-1.133	1.286	**3.9%**
RMA Plot-Twin Check Samples / Intrepid Exploration (2003-2005) - All Samples								
Element	**R^2**	**N (total)**	**Pairs**	***m***	**Error (*m*)**	***b***	**Error (*b*)**	**Bias**
Au (ppm)	0.966	20	20	0.756	0.044	0.355	0.690	**24.4%**
Ag (ppm)	0.966	20	20	0.907	0.052	-7.045	23.504	**9.3%**
RMA Plot-Twin Check Samples / Intrepid Exploration (2003-2005) - No Outliers								
Element	**R^2**	**Accepted**	**Outliers**	***m***	**Error (*m*)**	***b***	**Error (*b*)**	**Bias**
Au (ppm)	0.985	17	3	0.982	0.018	0.013	0.165	**1.8%**
Ag (ppm)	0.972	15	5	1.018	0.026	-7.193	4.857	**-1.8%**
RMA Plot-Pulp Check Samples / Intrepid Exploration (2003-2005) / All Samples								
Element	**R^2**	**N (total)**	**Pairs**	***m***	**Error (*m*)**	***b***	**Error (*b*)**	**Bias**
Au (ppm)	0.993	39	39	1.009	0.019	0.049	0.474	**-0.9%**
Ag (ppm)	0.978	39	39	1.001	0.034	-9.668	17.905	**-0.1%**
RMA Plot-Pulp Check Samples / Intrepid Exploration (2003-2005) / No Outliers								
Element	**R^2**	**Accepted**	**Outliers**	***m***	**Error (*m*)**	***b***	**Error (*b*)**	**Bias**
Au (ppm)	0.997	37	2	0.984	0.009	0.117	0.219	**1.6%**
Ag (ppm)	0.998	34	5	0.964	0.007	0.278	1.950	**3.6%**

After excluding some outliers (two for Au, and three for Ag), both plots indicate good fits between the check assays and the original assays, reflected in the high values of the coefficient of determination R^2 (ranging between 0.979 and 0.975), as well as acceptable Au and Ag bias values of the original ALS Geolab assays as compared to the new ACME assays (-1.5% and 3.9%, respectively).

AMEC also tried to evaluate, albeit indirectly, the possible significance of the sampling error, on the consideration that the difference in analytical precision between the initial assays and the ACME assays was probably minor, if compared to the usually larger sampling error. With this purpose, AMEC prepared Max–Min plots for Au and Ag, and processed the twin check samples as ordinary twin samples. This test resulted in two failures for Au (16.7%) and one failure for Ag (8.3%).

However, one of the Au failures is practically at the failure line. Therefore, despite the small sample size, AMEC infers that the sampling error during the BMG campaign was probably within or close to the acceptable limits. The data were accepted for use in resource estimation.

Intrepid Exploration

AMEC prepared RMA plots for Au and Ag. The RMA statistics are presented in Table 14-7. After excluding some outliers (three for Au, and five for Ag), both plots indicate good fits between the check assays and the original assays, reflected in the high values of the coefficient of determination R^2 (ranging between 0.985 and 0.972), as well as acceptable Au and Ag bias values of the original ALS Chemex assays as compared to the new ACME assays (1.8% and -1.8%, respectively).

AMEC also tried to evaluate, albeit indirectly, the possible significance of the sampling error, based on the assumption that the difference in analytical precision between the initial assays and the ACME assays were probably minor, if compared to the usually larger sampling error. With this purpose, AMEC prepared Max–Min plots for Au and Ag, and processed the twin check samples as ordinary twin samples. This test resulted in eight failures for Au (40.0%) and one failure for Ag (8.3%), so only the Au figure was beyond the acceptable range (less than 10% failure). However, most of the failures were actually very close to the failure lines. Therefore, AMEC infers that the sampling error during the Intrepid campaign until July 2005 was probably within or close to the acceptable limits.

Pulp Check Samples (2005 Re-sampling Program)

AMEC prepared a random selection of pulp check samples, chosen from the mineralized zones identified during the Intrepid drilling campaigns. However, AMEC did not apply a selection cut-off, thus avoiding the introduction of a selection bias. The samples consisted of duplicates of the backup pulp samples. No pulps were available from the BMG exploration campaign.

The selection included 39 samples from the Intrepid exploration samples (7.4% of 528 samples). The samples were submitted to ACME in Santiago de Chile.

AMEC prepared RMA plots for Au and Ag. The RMA statistics are presented in Table 14-7. After excluding some outliers (two for Au, and three for Ag), both plots indicate good fits between the check assays and the original assays, reflected in the high values of the coefficient of determination R^2 (ranging between 0.997 and 0.998), as well as acceptable Au and Ag bias values of the original ALS Chemex assays relative to the new ACME assays (1.6% and 3.6%, respectively).

Four pulp duplicates, five certified standards (three Oreas 61Pb and two Oreas 6Pb), and two blank samples were included in the check sample batch, in order to obtain an independent assessment of the accuracy, precision and possible contamination at ACME.

AMEC prepared Max–Min plots for the pulp duplicates. None of the duplicates presented failures for Au and Ag. AMEC also prepared control charts for Au for both standards. Both standards yielded acceptable accuracies (Au: 3.9% bias for Oreas 61Pb and -2.8% bias for Oreas 6Pb; Ag: 0% bias for Oreas 61Pb). Therefore, it can be concluded that the precision and accuracy of ACME during the assay of AMEC's re-sampling batch were appropriate. No contamination was observed during assaying.

Although Intrepid did not assess the analytical precision during its exploration campaign, considering the very high values of the coefficient of determination R^2 obtained between ACME's new assays and ALS Chemex's original assays, and the fact that the analytical precision at ACME was within the acceptable limits, AMEC infers that the analytical precision at ALS Chemex during the Intrepid exploration campaign was probably within or close to the acceptable limits.

Twin Check Samples (2006 Re-sampling Program)

Samples Assayed at ALS Chemex

AMEC prepared RMA plots for Au and Ag. The RMA statistics are presented in Table 14-8. After excluding three outliers for Au, and two outliers for Ag, both plots indicate relatively good fits between the check assays and the original assays, reflected in the values of the coefficient of determination R^2 (0.997 for Au and 0.932 for Ag), although the Au bias value of the original ALS Chemex assays as compared to the new ACME assays is -8.3%.

Table 14-8
RMA Parameters - AMEC Re-sampling Program (2006)

Element	R^2	Accepted	Outliers	m	Error (m)	b	Error (b)	Bias
RMA Plot-Twin Check Samples / Intrepid Exploration Program (2006-ALS Chemex) / All Samples								
Au (ppm)	0.997	6	3	1.083	0.007	0.032	0.038	**-8.3%**
Ag (ppm)	0.932	7	2	1.047	0.030	0.356	1.534	**-4.7%**

Element	R^2	N (total)	Pairs	m	Error (m)	b	Error (b)	Bias
RMA Plot-Twin Check Samples / Intrepid Exploration Program (2006-Alex Stewart) / All Samples								
Au (ppm)	0.799	10	10	0.859	0.122	0.012	0.113	**14.1%**
Ag (ppm)	0.865	10	10	1.110	0.129	-0.302	1.484	**-11.0%**

Element	R^2	Accepted	Outliers	m	Error (m)	b	Error (b)	Bias
RMA Plot-Pulp Check Samples / Intrepid Exploration Program (2006-ALS Chemex) / All Samples								
Au (ppm)	0.998	10	0	0.992	0.005	-0.142	0.066	**0.8%**
Ag (ppm)	0.995	10	0	1.057	0.008	2.609	2.175	**-5.7%**

Element	R^2	N (total)	Pairs	m	Error (m)	b	Error (b)	Bias
RMA Plot-Pulp Check Samples / Intrepid Exploration Program (2006-Alex Stewart) / All Samples								
Au (ppm)	0.992	8	0	0.990	0.014	-0.003	0.013	**1.0%**
Ag (ppm)	0.998	8	0	1.097	0.008	0.303	0.103	**-9.7%**

Samples Assayed at Alex Stewart

AMEC prepared RMA plots for Au and Ag. The RMA statistics are presented in Table 14-8 A high dispersion of the Au and Ag values is apparent, probably due to the fact that in both cases the values are very low (below 1 g/t Au and 20 g/t Ag).

Pulp Check Samples (2006 Re-sampling Program)

Samples Assayed at ALS Chemex

AMEC prepared RMA plots for Au and Ag. The RMA statistics are presented in Table 14-8. No outliers were identified. Both plots indicate very good fits between the check assays and the original assays, reflected in the high values of the coefficient of determination R^2 (0.998 for Au and 0.995 for Ag). The bias values of the original ALS Chemex assays as compared to the new ACME assays were acceptable (0.8% for Au and -5.7% for Ag).

Samples Assayed at Alex Stewart

AMEC prepared RMA plots for Au and Ag. The RMA statistics are presented in Table 14-8. No outliers were identified. Both plots indicate very good fits between the check assays and the original assays, reflected in the high values of the coefficient of determination R^2 (0.992 for Au and 0.998 for Ag). The Au bias value of the original Alex Stewart assays as compared to the new ACME assays was acceptable (1.0%). The Ag assays at Alex Stewart exhibit a high negative bias as compared to ACME assays (-9.7%), but considering that the Ag values are very low (below 20 g/t Ag) this fact is not relevant.

Two pulp duplicates, one certified standard (Oreas 7Pb), and two blank samples were included in the check sample batch, in order to obtain an independent assessment of the accuracy, precision and possible contamination at ACME. None of the duplicates presented failures for Au and Ag, and the Au bias of the standard was 0.0 %. Therefore, it can be concluded that the precision and accuracy of ACME during the assay of AMEC´s re sampling batch were appropriate. No contamination was observed during assaying.

Although Intrepid did not assess the analytical precision during its exploration 2006 exploration campaign, considering the very high values of the coefficient of determination R^2 obtained between ACME's new assays, on one hand, and ALS Chemex's and Alex Stewart's original assays, on the other hand, and the fact that the analytical precision at ACME was within the acceptable limits, AMEC infers that the analytical precision at ALS Chemex and at Alex Stewart during the 2006 exploration campaign was probably within or close to the acceptable limits.

Discussion on the QA/QC Evaluation

As a result of the analysis of the QA/QC programs established by BMG and Intrepid during the exploration campaigns at the Casposo deposit, complemented by AMEC´s independent sampling, AMEC concludes that:

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- Although AMEC could not directly assess the Au and Ag sampling errors during the BMG and Intrepid exploration programs, on the basis of independent re-sampling AMEC infers that the Au and Ag sampling errors were probably within or close to acceptable limits, and thus the samples can be used in resource estimation.
- Although AMEC could not directly assess the Au and Ag analytical precisions during the Intrepid exploration program, on the basis of independent re-sampling AMEC infers that the Au and Ag analytical precisions were probably within or close to acceptable limits, and thus the samples can be used in resource estimation.
- The Au and Ag sub-sampling variances during the BMG and Intrepid programs could not be evaluated.
- On the basis of independent re-sampling, AMEC established that the assay accuracy for Au and Ag at ALS Geolab during the BMG exploration program was satisfactory.
- Assay accuracy for Au and Ag at ALS Chemex during the Intrepid exploration program was satisfactory, which was confirmed by AMEC´s re-sampling program.
- Assay accuracy for Au and Ag at Alex Stewart during the Intrepid exploration program was satisfactory, which was confirmed by AMEC´s re-sampling program.

- BMG did not evaluate the possible contamination during preparation and assaying at ALS Geolab.
- No significant cross-contamination was detected during preparation at ALS Chemex and Alex Stewart during the Intrepid exploration program.
- Analytical precision and accuracy at ACME during AMEC's re-sampling programs were satisfactory; no significant cross-contamination was detected during assaying at ACME.

AMEC considers that the Au and Ag assays of the BMG and Intrepid diamond drill hole exploration campaigns at Casposo can be used to support resource and reserve estimation.

Other Samples

QA/QC data for the remaining sample types, trenches and pits, used in the resource estimation were not reviewed. Thus, AMEC has not commented on the precision and accuracy of that portion of the database or its suitability for use in resource estimation.

14.2 Data Verification by AMEC

14.2.1 Database Integrity Checks

Intrepid has maintained a database in GEMS® software that contains drill holes, trenches and pits completed during several campaigns, and covers a number of exploration targets such as Julieta, Panzón and Oveja Negra, as well as the Kamila and Mercado deposits.

The database integrity checks focused on data available for Kamila and Mercado.

From the existing GEMS database, 130 drill holes, 70 trenches and 5 pits were used to define the constraining domains boundaries.

The database is structured with the following tables and fields:

- Header: Hole-ID, Location X, Location Y, Location Z, Length, Drill_Area, Unique ID
- Survey: Hole-ID, Distance, Azimuth, Dip
- Lithology: Hole-ID, From, To, Lith+MV, Vein+Min, Litho+Vein, Vein
- Structure: Hole-ID, From, To, Code
- Assays: Hole-ID, From, To, Length, Sample, Au_Final, Ag1, As, Cu, Pb, Zn, Sb, Hg, Mo, Au-Eq, Domain, AuCut-g/t, AgCut-g/t
- LG_Fr_To: Hole-ID, From, To, Length, Domain

- HG_Fr_To: Hole-ID, From, To, Length, Domain.
- LG_Cmp_Cut: Hole-ID, From, To, Length, AuCut (g/t), AgCut (g/t), Domain
- HG_ Cmp_Cut: Hole-ID, From, To, Length, AuCut (g/t), AgCut (g/t).

AMEC conducted various validation and verification tests, which are described in the following subsections.

14.2.2 Database Validation

AMEC conducted a database validation using GEMS® software tools that check each table for missing intervals and intervals that are out of sequence or overlapping, and also compare the total drill hole length against the interval lengths defined in the secondary tables. AMEC verified drill holes, trenches and pits effectively used for resource estimation. The validation resulted in some errors from tables "Assays" and "Lithology" as listed:

- Table Assays: Ten drill holes have overlapping problems, but most of these occur areas outside of the resource model; however, hole CA-05-143 has an error that could cause problems in compositing.
- Table Lithology: AMEC found 69 drill holes with overlapping intervals and one drill hole that has an interval longer than the drill hole length.
- Table Survey: AMEC searched the database for abrupt changes in azimuth and dip, and found nine drill holes with possible minor downhole measurements problems. The intervals selected belong to the following holes: CA-00-30, CA-00-34, CA-00-35, CA-00-39, CA 00 46, CA-03-49, CA-03-53, CA-03-58, and CA-06-159. AMEC plotted those holes in the tri-dimensional space with the solids that were modeled for each vein, and confirmed that the eventual deviation problems will not impact in the resource estimation.

Although the problems identified are not critical and will not materially impact resource estimation for Kamila and Mercado, AMEC recommends correcting them in order to keep a consistent and clean database.

14.2.3 Drill Hole Collars

In December 2005, AMEC reviewed the collars of 21 diamond drill holes from the BMG and Intrepid campaigns, and measured the coordinates with a Vista e-Trex GPS, set for the Campo Inchauspe 1969 datum and a user-defined UTM projection. The measured coordinates were compared with the database coordinates, and the maximum planar difference found was 24 m (Table 14-9).

Table 14-9
Review of Collar Coordinates (December 2005)

Hole ID	Database Coordinates		AMEC Coordinates		Differences	
	X (m)	Y (m)	X (m)	Y (m)	X (m)	Y (m)
CA-99-20	2,439,004	6,548,331	2,438,983	6,548,314	-21	-17
CA-99-21	2,439,004	6,548,331	2,438,983	6,548,314	-21	-17
CA-99-25	2,439,004	6,548,331	2,438,983	6,548,314	-21	-17
CA-99-8	2,438,928	6,548,468	2,438,907	6,548,447	-21	-21
CA-00-28	2,438,967	6,548,277	2,438,945	6,548,260	-22	-17
CA-00-41	2,439,162	6,548,243	2,439,144	6,548,227	-18	-16
CA-03-53	2,439,004	6,548,331	2,438,983	6,548,314	-21	-17
CA-03-62	2,439,223	6,548,280	2,439,203	6,548,260	-20	-20
CA-03-76	2,438,947	6,548,516	2,438,927	6,548,498	-20	-18
CA-04-102	2,439,095	6,548,367	2,439,075	6,548,348	-20	-19
CA-04-103	2,439,095	6,548,367	2,439,075	6,548,348	-20	-19
CA-04-108	2,439,022	6,548,428	2,439,005	6,548,414	-17	-14
CA-04-113	2,438,976	6,548,359	2,438,952	6,548,377	-24	18
CA-04-114	2,438,975	6,548,359	2,438,952	6,548,377	-23	18
CA-05-119	2,438,974	6,548,358	2,438,952	6,548,377	-22	19
CA-04-80	2,438,947	6,548,516	2,438,927	6,548,498	-20	-18
CA-04-84	2,439,140	6,548,291	2,439,121	6,548,271	-19	-20
CA-04-87	2,439,057	6,548,438	2,439,046	6,548,424	-11	-14
CA-04-90	2,439,140	6,548,291	2,439,121	6,548,271	-19	-20
CA-05-122	2,438,984	6,548,420	2,438,966	6,548,402	-18	-18
RC-04-10	2,438,881	6,548,435	2,438,860	6,548,434	-21	-1

In August 2006, AMEC checked the GPS coordinates of two topographic reference points (PB 01 and PB 02; discussed earlier in Section 10.1.1 and 10.1.2). The readings showed large differences with the official recorded coordinates of the points, similar to those differences documents by AMEC in Table 14-9.

Then, AMEC reset the UTM projection to the following parameters, indicated by the Project surveyors:

- Longitude origin: W 069°00.000'
- Scale: +1.0002800
- False Easting: +2,500,024.2
- False Northing: +10,003,250.8

With this correction, the maximum planar differences in X and Y are adjusted to 3.7 m and 2.4 m, respectively. Subsequent to this adjustment, AMEC re-surveyed the two reference points and the collars of 12 representative drill holes from the 2006 Intrepid campaign, using a Vista e-Trex GPS. The measured coordinates were compared with the original coordinates, and the maximum planar differences in X and Y coordinates were 6.6 m and 9.2 m, respectively, all within the acceptable GPS error (Table 14-10).

Table 14-10
Review of Collar Coordinates (August 2006)

Hole ID/Point	Database Coordinates		AMEC Coordinates		Differences	
	X (m)	Y (m)	X (m)	Y (m)	X (m)	Y (m)
PB 01	6,547,715.64	2,439,623.26	6,547,717	2,439,624	1.4	0.7
PB 02	6,547,974.29	2,439,353.55	6,547,978	2,439,356	3.7	2.4
CA-05-180	6,548,810.00	2,438,851.00	6,548,810.00	2,438,855.00	0.00	-4.00
CA-05-159	6,548,810.00	2,438,853.00	6548809.23	2438854.84	0.77	-1.84
CA-05-162	6,548,819.00	2,438,845.00	6548825.06	2438838.74	-6.06	6.26
CA-05-166	6,548,845.00	2,438,815.00	6548848.37	2438807.73	-3.37	7.27
CA-05-168	6,548,861.00	2,438,792.00	6548856.97	2438795.89	4.03	-3.89
CA-05-169	6,548,874.00	2,438,742.00	6548879.61	2438742.18	-5.61	-0.18
CA-05-171	6,548,896.00	2,438,733.00	6,548,897.06	2,438,732.44	-1.06	0.56
CA-05-173	6,548,930.00	2,438,716.00	6,548,928.88	2,438,725.22	1.12	-9.22
CA-05-170	6,548,948.00	2,438,710.00	6548954.61	2438710.02	-6.61	-0.02
CA-05-164	6,548,984.00	2,438,664.00	6548980.35	2438662.31	3.65	1.69
CA-05-155	6,549,048.00	2,438,682.00	6549044.98	2438680.17	3.02	1.83
CA-05-156	6,549,050.00	2,438,670.00	6549048.02	2438669.31	1.98	0.69

As part of the collar review, AMEC also plotted all the Casposo drill holes on the final version of the topographic map and compared the collar altitude with the location altitude according to the contour lines in the topographic map. None of the holes presented significant differences in elevation, and all appear to be correctly plotted.

14.2.4 Down Hole Surveys

Down hole surveys were directly entered into the logs. Therefore, no paper records exist for these measurements but Sperry-sun films were available on site. In addition to checks described in Section 14.1, AMEC visually reviewed the down hole survey table, cross checked the information from various cells through simple formulas, and did not find significant discrepancies between collar survey data and down hole survey data, or between the down hole survey data for consecutive intervals. For Casposo, the average deviation in azimuth was 4.2º/100 m, and the average deviation in dip was 3.3º/100 m, which are considered reasonable.

14.2.5 Twin Holes

AMEC reviewed four pairs of twin holes drilled by Intrepid: CA-99-2 and CA-03-57; CA 99 19 and CA-03-47 and CA-99-20 and CA-03-50, at Kamila, and CA-99-4 and CA 03 63, at Mercado. The Au and Ag grade versus depth distributions were plotted on mirror graphics. Although the grades usually differed, in all cases there was a good match between the positions of the mineralized intervals in the original and the twin holes.

14.2.6 Original Logs versus Geological Sections

All original drill hole logs are properly filed on site and can be easily reviewed. The original logs include very detailed hand-written descriptions of the lithology, mineralization, structure, alteration, and other features. AMEC visually cross-checked the original logs and the information included in the digital files for the drill holes listed in Table 14-11, and found only few minor changes in the spelling. The information included in the digital files corresponds accurately to the hand-written descriptions.

Table 14-11
List of Reviewed Drill Hole Logs

Drill Hole Id	Drill Hole Id
CA-99-10	CA-04-110
CA-00-30	CA-05-130
CA-02-50	CA-06-161
CA-03-71	CA-06-163
CA-04-90	CA-06-167

14.2.7 Drill Core versus Original Logs

During the two site visits, AMEC reviewed selected core intervals and compared the core with the original logs from the drill holes listed in Table 14-12.

Table 14-12
List of Reviewed Drill Hole Core

Drill Hole Id	From – To
CA-99-21	75 m to 115 m
CA-00-31	70 m to 110 m
CA-03-51	10 m to 40 m
CA-03-73	0 m to 272 m
CA-04-90	25 m to 50 m
CA-04-98	135 m to 170 m
CA-04-113	120 m to 151 m
CA-05-119	160 m to 185 m
CA-06-161	40 m to 70 m
CA-06-163	55 m to 90 m
CA-06-167	30 m to 60 m

AMEC checked the main lithologic contacts, alteration features, structures and sample intervals, which were adequately recorded. The core intervals were adequately marked in the boxes. Sample intervals (from-to) were usually in agreement with major lithology changes on drill logs, but occasionally sampling did not respect these boundaries, and included up to 20 cm of the wall rocks in the “mineralized” sample.

14.2.8 Original Assay Certificates versus Database Entries

The original assay certificates were available at the San Juan office. Intrepid provided AMEC with copies of 16 certificates from ALS Geolab (BMG program), as well as eleven certificates from ALS Chemex and two certificates from Alex Stewart (Intrepid program). The reviewed assay certificates are listed in Table 14-13.

Table 14-13
List of Assay Certificates in the Double Data Entry Check

Laboratory	Certificate No.	Program	Laboratory	Certificate No.	Program
ALS Geolab	ME1808	BMG	ALS Geolab	ME2958	BMG
ALS Geolab	ME1831	BMG	ALS Chemex	ME04023257	Intrepid
ALS Geolab	ME1855	BMG	ALS Chemex	ME04078935	Intrepid
ALS Geolab	ME1874	BMG	ALS Chemex	ME04078936	Intrepid
ALS Geolab	ME1887	BMG	ALS Chemex	ME04079589	Intrepid
ALS Geolab	ME2122	BMG	ALS Chemex	ME04085505	Intrepid
ALS Geolab	ME2131	BMG	ALS Chemex	ME04087660	Intrepid
ALS Geolab	ME2143	BMG	ALS Chemex	ME04090164	Intrepid
ALS Geolab	ME2153	BMG	ALS Chemex	ME05010452	Intrepid
ALS Geolab	ME2521	BMG	ALS Chemex	ME05012153	Intrepid
ALS Geolab	ME2537	BMG	ALS Chemex	ME05033724	Intrepid
ALS Geolab	ME2548	BMG	ALS Chemex	ME05041614	Intrepid
ALS Geolab	ME2564	BMG	Alex Stewart	M060867	Intrepid
ALS Geolab	ME2573	BMG	Alex Stewart	M060885	Intrepid
ALS Geolab	ME2942	BMG			

The Au and Ag assay results from the reviewed certificates were manually re-entered by AMEC. The certificates included 2,490 samples from the BMG program, representing 47% of the BMG Au and Ag assay data, and 1,270 samples from the Intrepid program, representing 24% of the Intrepid Au and Ag assay data to hole CA-06-166. All reviewed samples were included in the database. The double data entry was checked against the database, and the results are presented in Table 14-14.

Table 14-14
Summary of Assay Double Data Entry Check

BMG					
Au			Ag		
Total Assays	2,847	Error Rate	Total Assays	2,847	Error Rate
Difference > 0.02 g/t	65	2.6%	Difference > 0.5 g/t	13	0.5%
Difference > 0.3 g/t	17	0.0%	Difference > 20 g/t	2	0.0%
ALS Chemex					
Au			Ag		
Total Assays	1,158	Error Rate	Total Assays	1,158	Error Rate
Difference > 0.02 g/t	5	0.2%	Difference > 0.5 g/t	1	0.0%
Difference > 0.3 g/t	0	0.0%	Difference > 20 g/t	1	0.0%
Alex Stewart					
Total Assays	122	Error Rate	Total Assays	122	Error Rate
Difference > 0.02 g/t	0	0.0%	Difference > 0.5 g/t	0	0.0%
Difference > 0.3 g/t	0	0.0%	Difference > 20 g/t	0	0.0%

The error rate of the BMG Au assay data in the database is 2.6%, which is above the 1% acceptable figure, although only 16 of these values (0.007% of the entries) had differences exceeding 0.3 g/t. The error rates of the BGM Ag assay data and the Intrepid Au and Ag assay data are within an acceptable range (Table 14-14). These data can be used in the resource estimation, although AMEC recommends that Intrepid re-digitize the BMG Au assay data.

15.0 ADJACENT PROPERTIES

There are no adjacent properties that are at an advanced exploration, development or production stage adjacent to the Casposo Project.

16.0 MINERAL PROCESSING AND METALLURGICAL TESTING

16.1 Historical Metallurgical Test Work

In 2000, BMG carried out two internal metallurgical studies on Kamila core samples at BMG's Kori Kollo mine site in Bolivia. The results have been superseded by later work.

Instituto de Investigaciones Mineras (IIM)–Kappes Cassidy (2002–2004)

Between 2002 and 2004 Intrepid sampled additional surface areas and core from the Kamila vein system and one area above the Mercado vein for additional metallurgical work. Initial metallurgical bench-scale testing conducted in 2002 was carried out on five composite samples (#1595 to #1599) at the Instituto de Investigaciones Mineras (IIM) in San Juan Faculty of Engineering, University of San Juan. The work included Bond Work Index determination, gravity concentration and cyanidation bottle rolls.

Selected splits of the 2002 samples were also forwarded to Kappes, Cassidy & Associates (KCA) of Reno, Nevada for independent verification. This work basically confirmed the IIM metallurgical testing.

In 2003 Intrepid submitted 110 m of newly drilled half core samples from Kamila to IIM for additional bottle roll leach testing at P80 grinds of 180 µm and 105 µm. This comprehensive test program was conducted on nineteen new metallurgical composite samples (#6083 to #6101) created from composite intervals of mineralization of the individual Aztec, Inca and B vein structures intersected in thirteen drill holes. Some coarser bottle roll leach extractions were also conducted to investigate heap leaching potential. Mineralogical investigations were also conducted on selected samples of Kamila core and included Electron Microprobe analysis by Kishar Research and Scanning electron microscope (SEM) investigations by Miller and Associates.

Numerous leach tests were conducted on surface and core samples. The work indicated the Casposo samples tested were reasonably amenable to direct cyanidation and did not require intensive grinding. In 2003 IIM conducted a more comprehensive leach program at two relatively coarse P80 grinds of 180 µm and 105 µm (80 and 150 # mesh). The average head grade of the samples tested were 8.58 g/t Au and 157 g/t Ag which is higher than the current reserve grade. Therefore the average recoveries reported at these grinds should not be applied directly to the current reserves because this would tend to overstate metal recovery projections as a result of the higher grade. However, the data provides useful relevant information about grind sensitivity, extraction amenability and variability, and reagent consumption.

At a P_{80} grinds of 180 µm (80 mesh #) gold and silver extractions of about 90.2% (70.6% to 96.0%) and 75% (55% to 87%) were achieved respectively in 48 hours. At a P_{80} grind of 105 µm (150 mesh #) gold and silver extractions of about 91.3% (97.4% to 81.4%) and 79% (60% to 87%) were achieved respectively in 48 hours. This indicates an extraction benefit of about 1% and 4% for gold and silver respectively with finer grinding.

This much more extensive program used composites from individual veins across the deposit and again did not indicate any marked decrease in gold or silver leach extraction with the elevation or vein structure from which the sample was taken. This is consistent with IIM's earlier 2002 observations. AMEC resampled selected intervals of these same core samples to create new composites for the basis of the feasibility study program. The objectives were to conduct independent verification of the IIM results, as well as assess finer grinding at 200 mesh and optimization of leach extraction conditions. AMEC's analysis of the IIM results suggested low cyanide concentrations could be limiting the ultimate extractions achieved, particularly for gold.

Based on very poor coarse bottle roll leach extractions reported by IIM in their September 2003 report, heap leaching was not considered an option for Casposo ore, and no additional work was conducted, or recommended on this.

Sodium cyanide and lime consumption were studied more extensively in the IIM 2003 program and they reported about 0.8 kg/t (0.5-1.2 kg/t) and 2 kg/t respectively. In the IIM report lime addition is reported, actual consumption was about 0.5 kg/t and this is reasonably consistent with that expected from the composition of the ore and more recent testing at SGS.

The Bond ball mill work index (BWI) and the abrasion index (AI) were measured on seven samples of Casposo ore. These data indicate the ore is hard with BWI indices in the range of 17.0–18.5 Kw/h/t (Imperial) and very abrasive with an Ai in the range of 0.825–1.036. This is consistent with the highly silicified quartz vein composition of the ore.

In addition some of these samples showed reasonable amenability to gravity concentration. Tests conducted by IIM resulted in up to 25% of the gold being recovered by gravity.

Mineralogical reports on some drill core samples taken from below 2,400 meter elevation showed that the gold and silver values are present as fine grained (generally <25 µm) gold, electrum, native silver, acanthite, and silver-bearing tetrahedrite-tennantite.

Resource Development Incorporated (2005)

In 2005 Resource Development Incorporated (RDI) carried out flotation metallurgical studies on a 50 kg composite sample prepared from 48 individual samples of Kamila analytical rejects, drill core rejects or reverse core cuttings.

The composite assay was 3.16 g/t gold and 165 g/t silver. This is lower than the average of samples previously tested for cyanide extraction by IIM. The calculated feed grade from test work indicated gold and silver values to be slightly higher than the assayed values.

RDI initially treated the sample using gravity methods. Four different P_{80} grind sizes were tested: 300, 212, 150 and 105 µm.

Reasonable gravity extractions were achieved, especially within the coarse fractions, as well as very good concentrate grades. The gravity tail was used as the flotation feed to determine the flotation extraction as well as the combined extraction.

Four scoping flotation tests at 4 size fractions were completed. With finer grinding the gold recovery increased. Silver extraction was best at P_{80} grind of 150 µm. The highest gold combined gravity and flotation concentrate grade was obtained at the finest grind of 150 µm.

The following conclusions were made:

- Gravity recovered 8.5% to 21.1% of gold and 8% to 20.2% of silver in the feed.
- Flotation recovered 62% to 72% of gold and 75% to 83% of silver in the flotation feed.
- Combined gravity and flotation process recoveries were 66.1% to 73.3% for gold and 79.1% to 86.2% for silver.
- The finer the primary grind, the higher the precious metal recoveries generally and the higher the concentrate grade.

These concentration studies indicated that combined gravity and flotation process may be an alternative, though second best in terms of recovery, to the cyanidation process. The combined gravity and flotation process precious metals recoveries were lower than those obtained in the direct cyanidation process. However, the gravity and flotation tests were only preliminary in nature and additional testing was suggested by Moritz to optimize the process parameters. Gold recovery in particular was significantly lower (18% less). It was suggested that the old and fine analytical reject and reverse core cuttings samples used, if not properly stored, would have oxidized and be detrimental to flotation recovery results. Additional flotation test work was recommended to validate this using coarser core samples rather than analytical rejects. Reagent optimization and finer grinding were also suggested.

SGS Lakefield Feasibility Testwork (January–March 2006)

The objective of this program was to confirm earlier IIM metallurgical test work in 2004, and to provide the metallurgical data necessary to support a feasibility process flowsheet selection trade-off study. AMEC resampled core intervals representing the splits from the IIM samples in 2003 and collected new 2006 drill core, and forwarded these to SGS Lakefield Research for independent verification testing. The previous RDI flotation work had been conducted on old assay rejects and the use of samples freshly prepared from split core was intended to assess any potential improvement in flotation metallurgy using material with less potential oxidation problems. Additional surface bulk samples were taken from existing trenches for the Bond Work and Abrasion tests. These tests require larger pieces than the half core available.

The SGS program compared whole ore cyanidation to a gravity + flotation + concentrate cyanidation flowsheet. In addition, general ore characterization tests, including head analyses, quantitative mineralogy, Bond Index determinations, acid base accounting and settling and filtration testing were completed.

Ten sub-composites (seven from split core and three from coarse trench samples) were prepared. Samples of each composite 1 to 5 and 7 (at -3") were split out and combined as the Kamila Master Composite for flowsheet testing. Composite 6 consisted of split core samples from Mercado, which had a much higher relative silver content and was tested separately to avoid biasing the silver grade of the Kamila Master Composite. The other three composites were prepared from coarse surface samples and used for additional comminution testing.

Ore characterization of the samples indicated the following key characteristics:

- The Master Composite gold head grade calculated by averaging the heads from the individual sub-composite samples was 6.61 g/t Au. This agrees very well with the average 6.45 g/t calculated gold head grade from the individual metallurgical tests. Direct gold assay of the Master Composite (from 30 g fire assays) was lower, possibly due to the influence of gravity gold in the combined sample. The Master Composite silver head grade was consistently in the range of 110 to 115 g/t Ag.
- Analyses of the individual sub-composites indicated no significant elements typically detrimental to cyanidation, or the production of a saleable flotation concentrate.
- Sulphur levels were relatively low from 0.02% to 0.74% and whole rock analyses indicated generally high silica levels >90%. These analyses are consistent with the very low sulphidation and highly siliceous nature of the mineralized quartz vein system.

- A sample of Master Composite ore was submitted for modified acid-base accounting (ABA) test work. This indicated the Master Composite is unlikely to be acid generating. However, it was recommended that longer term kinetic tests be completed in order to confirm and better quantify this potential.
- Comminution test work on four samples indicated they are relatively hard and abrasive with respect to ball milling. The average rock density was 2.59 and the metric Bond ball mill work index ranged from 19 to 21 kW/h/t. The abrasive index ranged from 0.894-0.986 g. These data are consistent with earlier IIM test work and confirm the hard and siliceous nature of the mineralization host quartz vein rock.

Test work on the Master Composite indicated the following metallurgical characteristics:

- Overall the SGS test work basically verified the previous 2004 IIM gravity-leach and RDI flotation metallurgical testing.
- At a primary P80 grind of 160 µm, 33% of gold was recovered into a 0.042% wt gravity concentrate indicating its amenability to gravity processing. A simple, low mass yield, centrifugal gravity circuit (Knelson or Falcon) will likely yield gold recoveries in the 20 to 30% range. Additional grind-recovery gravity optimization work was recommended in conjunction with, or without, leaching to assess the overall potential recovery benefits of gravity recovery. Only 6% of silver was recovered in the gravity concentrate.
- Rougher flotation, at P80 grind sizes ranging from 150 to 100 µm, gave relatively poor gold and silver recoveries in the two tests conducted. Gold recovery by gravity separation + rougher flotation ranged from 72% to 80% while silver recovery was between 68% and 74%. Based on these low recoveries using freshly prepared split core flotation does not appear to be a viable economic option in the treatment of the Casposo ore. This confirms the poor flotation response reported by RDI in their 2004 test work using assay rejects. No additional work was recommended in this area.
- Whole ore cyanidation test work, on both the Master Composite and the Mercado (Composite 6) sample, indicated gold is readily extractable at a relatively coarse primary grind size of 100 µm. Gold extractions of about 95-96% were achieved in 48 hours. Increasing the pulp density to 45% solids (w/w) from the standard 40%, while lowering the cyanide concentration from the standard 2 g/L NaCN (to 1.5 g/L), had no negative impact on gold metallurgy for either sample. Cyanide and lime consumption were relatively low at about 0.6 kg/t and 0.4 kg/t respectively. The good metallurgical leach response observed is consistent with the elemental and rock composition characteristics of the samples.
- Silver extraction was reduced from 84% to about 81% at the higher density and lower cyanide concentration. It is likely this was due to the reduced cyanide concentration.

- Silver extraction from the Mercado (Composite 6) sample was much lower (at 73.4%) in the one test completed with standard leach conditions. However the high silver grade possibly required a higher cyanide concentration. Additional work was recommended to confirm this in the future.

It was envisaged a Counter Current Decantation (CCD), or a filtration wash system, or hybrid of both, would be employed in the plant design for precious metal solution recovery and tailings dewatering. Preliminary scoping static thickening tests were conducted on leached slurry using various flocculants. The cyanide leached pulps also filtered very well at the end of leaching. Additional thickening and filtration test work was recommended to assist in a technical-economic trade-off study to select the most appropriate dewatering and washing technology.

SGS Lakefield Feasibility Testwork (June–November 2006)

The objective of this test program was to optimize the metallurgical parameters for the gravity separation and gravity tailings cyanidation flowsheet chosen, develop equipment design selection criteria and conduct recovery variability mapping to develop an overall geometallurgical grade-recovery model. Additional thickening and filtration test work was conducted by Pocock International to assist in a technical-economic trade-off study to select the most appropriate dewatering and washing technology. General ore characterization tests were also completed, including additional head analysis and ore hardness BWI mapping.

Two basic gold-silver recovery flowsheets were examined in this program:

- Gravity separation + gravity tailing cyanidation, and
- Whole ore cyanidation.

In addition to metallurgical testing, humidity cell testing was conducted by SGS to characterize tailings acid mine drainage (AMD) potential on a composite leach sample created in the metallurgical program.

The recovery variability test work was conducted on portions of the ten individual sub-composites retained from the first test work phase and an additional master composite sample was created from these for the remaining work.

Seven samples were submitted for Bond Abrasion Index testing and the values ranged between 0.7334 and 0.9863 g indicating very abrasive ore.

A single sample was submitted for (each of) Bond Crushing and Rod Mill Work Index testing. The results were 6.6 Kwh/t and 17.6 Kwh/t respectively.

Eight samples were submitted for Bond Ball Mill Work Index testing and the values ranged between 18.0 and 21.0 Kwh/t.

Gold recovery by gravity separation averaged approximately 25% while silver recovery was about 11%. The significant differences between gold and silver recovery indicates a fundamental difference in the character and mineralogical association of the two metals.

Cyanidation test work was completed on whole ore representing each of the ten composites and on gravity tailings representing the same samples. This test series was intended to evaluate the two principle flowsheet options head to head.

The data indicates no difference in gold or silver recovery between gravity separation and gravity tailing cyanidation compared to direct whole ore cyanidation.

Gold extractions were consistently good. The lower silver leach extractions of composite 4 and 10 are a function of the relatively low silver head grade of those samples, of 21 to 53 g/t. Composite 6 indicated a relatively low silver extraction of 79% than might be expected relative to the high head grade of 563 g/t. Rather than a mineralogical problem the very high silver grade ore extraction was most likely limited by insufficient cyanide and required a higher cyanide concentration than the standard.

These tests were completed using oxygen sparging to eliminate possible experimental variations in the oxygen concentration parameter in the closed bottle rolls tests conducted. In this instance there is no indication from earlier work of any significant improvement in leach rate kinetics, or recovery, in boosting oxygen concentrations beyond air saturation, nor does the ore mineralogical composition suggest the ore will have an abnormally high oxygen demand. The plant is located at a moderate elevation of about 2,400 masl, therefore no leach oxygen supply limitations are expected.

Subsequent to the initial series of variability tests and based on those results, several parameters were identified as requiring further evaluation for design or optimisation for leach extraction. The parameters identified and tested were:

- Pulp Density = 40% solids (w/w) Cyanide Concentration and Consumption; which appeared to be, in several cases, significantly higher than the original test work completed on Master Comp had indicated. Cyanide concentrations ranging from 0.5 to 5 g/L were examined.
- Grind Size; A feed P80 size range of ~80 – ~150 µm was evaluated.
- Dissolved Oxygen Concentration; Tests were completed with sparged air (6-8 mg/L O_2) and oxygen (>15 mg/L O_2).

- Zinc Concentration; A series of tests was completed for the purpose of determining the impact of zinc build-up from the Merrill-Crowe circuit recycle stream in the plant leach circuit.

Silver recovery/extraction in composite 4 showed a remarkable improvement at the higher cyanide concentrations. Clearly, silver extraction from the high grade Comp 6 (~560 g/t Ag) had been limited by the concentration of cyanide is solution in the previous tests. In plant operation such high average transient ROM feed grades will be blended out in the mill feed.

Overall gold extraction benefited from finer grinding, increasing from 92.6% at 120 µm (CN 39) to 94.3% at P_{80} of 84 µm (CN-37). Gold recovery at a grind of P_{80} of 102 µm (CN 38) was 93.7%. Silver extraction was essentially identical in all three tests ranging from 82.5% to 82.8% while residue grades ranged from 31 to 33 g/t Ag.

Tests were conducted on Master Comp 3 for the purpose of evaluating the impact of dissolved oxygen level on both gold-silver extraction and cyanide consumption. Based on the results from these tests, higher oxygen level did seem to have a minor positive impact on both gold and silver extraction. Overall gold recovery, including gravity recovery in the test completed with oxygen @~20 mg/L O_2 was 94.3% compared to 92.3% with air ~7 8 mg/L O_2. It is recommended that this parameter is investigated further during operation using a portable modular pressure swing adsorption oxygen generation plant. This initially can be leased, but will be a relatively insignificant capital and operating cost item to add if small incremental recovery benefits can be demonstrated.

A final series of tests was conducted on Master Composite 3 whole ore for the purpose of estimating the potential impact of soluble zinc (from the planned Merrill-Crowe gold-silver recovery plant) on the cyanidation circuit in the Casposo mill. The Merrill-Crowe barren solution would be recycled to the leach plant. The tests were conducted at ~ P_{80} of 100 µm with 1.5 g/L NaCN maintained. Zinc was added to each pulp as zinc sulphate ($ZnSO_4$). Soluble zinc levels of 0, 0.5, 1.0 and 1.5 g/L were tested. The impact of 0.5 g/L Zn was imperceptible in terms of its effect on gold extraction.

Gold extraction dropped by approximately 3% when the zinc level was increased to 1 g/L. Silver extraction however was significantly impacted by solution zinc content. At 0.5 g/L Zn in solution, silver extraction at 83.6% decreased by ~2.5% (from 86.1%) compared to the baseline (0 g/L Zn) test. Further increasing the zinc content to 1 g/L and 1.5 g/L had a significant effect on silver extraction. Total silver extraction was 68.3% at 1 g/L Zn and 60.4% at 1.5 g/L Zn. This indicates the Casposo design criteria should incorporate a bleed to maintain zinc concentrations in the circulating solution below 0.5 g/L.

Thickening, Filtration and Pulp Rheology Studies

Thickening, filtration and pulp rheology investigations were conducted by Pocock International in order to provide:

- flowsheet development data for use in a solid liquid separation and tailings disposal method trade-off study AMEC prepared
- process design criteria for settling aid (flocculant) selection and consumption, and equipment sizing

The test work was conducted on-site at SGS Lakefield on the CN-7 leach residue sample

The bleed volume requirement is calculated using a typical zinc addition of 0.65 kg Zn/kg of precious metal precipitate produced and precipitate production based on the head grade of the ore. These fix the barren bleed volume requirement to be about 7.5 m^3/h. This is equivalent to about 6.8% of the nominal MC feed flowrate of 118 m^3/h. The barren solution bleed stream from the MC and filtrate solution from the cyanide wash zone of the belt filter will be pumped to a conventional SO_2-Air cyanide tank destruction system circuit to destroy cyanide and to control (precipitate) zinc values in the MC circuit. The process water produced will be conditioned in the plant process solution pond and recycled to the process.

The results of flocculant screening tests indicated standard polyacrylamide flocculants provided the best performance and a dose of 40 to 50 g/t was required to produce reasonable settling rates and good supernatant clarity. A design unit flux area of 0.125 $m^2/t/d$ was recommended for conventional thickener sizing based on properly flocculated material. Rheological data indicated no problems with thickener underflow pumping should be expected up to a threshold solids concentration of 72%, which is significantly higher than the normal operating range of about 55% in this high rate thickener application.

Vacuum filtration tests examined the effect of cake thickness and air dry duration on production rate and filter cake moisture. Washing material balances for selected wash ratios for predicting soluble recovery were developed. Accordingly filter sizes for horizontal belt vacuum filters were modelled at the various wash ratios and combinations of prewash thickeners.

At a wash ratio of 3.0 and without prewash thickeners, a solute recovery of 96% was estimated with a filter production rate of 623 $Kg/h/m^2$ giving a cake of 17% moisture. Using two prewash thickeners as planned, the solute recovery is estimated to be 99.7%.

16.2 Process Mineralization Types

The ore consists of relatively compact and siliceous quartz vein rock of low sulphide content. Dilution by country andesite footwall and rhyolite hanging wall rock is expected to be less than 15%, very little of which carries much grade. Gangue constituents include some interstitial alteration clay from wall alteration cavities, but this is not expected to exceed five percent in the ore. The primary vein structures forming the basis of the ore reserves that will be processed are: Aztec, Inca and B veins and the smaller adjacent Mercado zone of lower grade. Extensive sampling of the vein structures across the deposit and at various depths shows they have very similar geometallurgical characteristics and mineralogical compositions. Oxidation is developed in the surface expression of the veins to depth in some places, but detailed recovery mapping test work has not indicated any marked variability in gold or silver leach extraction with the plan, elevation or vein structure from which the sample was taken. Ore hardness testing across the deposit has also indicated it is consistently hard and abrasive with respect to ball milling, with relatively little variation in the Bond Ball Mill Work (BWI) index.

The projected Casposo milling schedule and head grades are given in Table 16-1.

Table 16-1
Milling Schedule and Head Grades

Reserve Schedule					
Mined Zone	**Reserves kt**	**Au Grade g/t**	**Ag Grade g/t**	**Au kOz**	**Ag kOz**
Kamila Open Pit	1,188	5.61	92.6	214.5	3,537
Kamila Underground	439	3.31	191.7	46.6	2,705
Mercado Open Pit	80	2.19	93.5	5.6	241
Low Grade Ore	84	1.25	24.9	3.4	67
Total	**1,791**	**4.69**	**113.8**	**270.1**	**6,550**

Milling Schedule							
Mill Production	**%Au+Ag**	**Year 1**	**Year 2**	**Year 3**	**Year 4**	**Year 5**	**Total**
Mill feed, t/y		355,875	365,000	364,925	364,597	340,917	1,791,313
Mill feed, t/d		975	1,000	1,000	1,000	1,000	
Mill Feed Grades							
Silver, g/t		55.4	99.8	150.3	161.7	100.5	113.8
Gold, g/t		4.77	6.72	5.15	3.81	2.82	4.69
Silver, oz/a		633,573	1,171,311	1,763,497	1,897,803	1,099,885	6,566,069
Gold, oz/a		54,613	78,888	60,433	44,727	30,838	269,500
Mill Recoveries							
Silver, %		76.8%	79.8%	81.7%	82.0%	79.8%	80.7%
Gold, %		93.6%	93.8%	93.6%	93.5%	93.5%	93.7%
Metals Production							
Silver, oz/a		486,788	934,525	1,440,022	1,556,330	877,881	5,295,547
Gold, oz/a		51,117	74,032	56,590	41,825	28,825	252,389
Precipitate							
Precipitate, kg/a	75%	22,305	41,821	62,060	66,270	37,598	230,054
Dorè							
Dorè, kg/a	93%	17,609	33,017	48,994	52,319	29,683	181,622

16.3 Trade-off Studies

AMEC completed an economic trade-off study using the SGS test work data to support the selection of a process flowsheet for the Casposo Feasibility Study. Four processing and project configurations were investigated.

- Case 1. Gravity, direct cyanidation, Counter Current Decantation (CCD) and recovery by MC to produce dorè.
- Case 2. Gravity, direct cyanidation, filtration and recovery by MC to produce dorè.
- Case 3. Gravity and flotation and concentrate cyanidation and recovery by MC to produce dorè.
- Case 4. Gravity and flotation to produce a concentrate for sale.

The direct ore cyanidation and MC precious metal recovery routes considered in Cases 1 and 2 were identified as the most economic processes.

Additional dewatering test work and trade-off studies were subsequently conducted on the direct leaching cases to support the selection of one of the following three flowsheet tailings disposal and dewatering options. These were:

- Conventional slurry tailings storage confined in a double lined facility behind an earthfill dam using CCD solution recovery, whole tailings cyanide destruction and process water reclamation.
- Paste or high density tailings (70% wt) confined in a double lined facility behind a smaller earthfill dam and using CCD solution recovery, whole tailings cyanide destruction, high density tailings thickening and pumping.
- Dry stacked tailings on a single liner, using filtration, or a hybrid combination of CCD and filtration, and cyanide recovery and destruction via the filtration process.

The trade-off study identified the dry stack alternative using a hybrid combination of CCD and filtration as the most economic. The dry stack is a simple less environmentally invasive sediment control structure than other alternatives. The unsaturated state of the tailings, together with the increased placement density, also eliminates liquefaction and flowsliding potential under even the most severe seismic loading conditions such as occur in the Province of San Juan. Transpiration of the remaining interstitial water in the solids to the surface of the tailings facility, and subsequent evaporation, results in the continued progressive drying of the solids as well as the complete destruction of any minor residual cyanide in that solution by exposure to natural sunlight. Finally, dry stacking provides the ability to carry out progressive reclamation during operations, and facilitates rapid final reclamation upon closure.

The potential for gold, and to a lesser extent silver, recovery by gravity separation had been established in test work. There was no obvious gold or silver recovery synergy between gravity separation and gravity tailing cyanidation compared to direct whole ore cyanidation. This synergy is generally difficult to demonstrate statistically in bench-scale test work and gravity concentration processing is included in the feasibility flowsheet mainly because of industry experience. This type of ore can contain transient coarse gold which can be lost to tails because it is slow to leach especially when associated with silver as electrum. Some coarse metallic electrum has been observed in some metallurgical samples and reported in historical mineralogy. The gold rich gravity concentrate will be intensively leached and recycled to the ball mill for additional grinding because of this.

AMEC completed a trade-off study on the use of a portable versus fixed crusher installations and the former was selected.

16.4 Process Flowsheet

The feasibility process flowsheet selected to process Casposo ore will use conventional primary jaw and secondary cone crushing, ball milling, gravity concentration for coarse gold and silver, cyanide leach, counter current decantation and washing and dewatering of tails by belt filtration. Gold and silver will be recovered by standard Merrill Crowe zinc precipitation and smelted to produce dorè bars. The leached tailings, following filtration to recover precious metals, will be washed and rinsed on the same belt filter to remove cyanide. The cyanide wash solution will be collected for the destruction of cyanide using the conventional SO_2/air process. The detoxified solution is recycled to the belt filter as wash solution. This will minimize the fresh water requirements for the process.

Filtered tailings will be trucked to a lined tailings management facility and stacked in compacted lifts.

The average gold and silver recoveries are expected to be approximately 93% and 82% respectively. The milling circuit will nominally process 1,000 t/d, over three eight-hour shifts, at a target grind size P_{80} of 105 µm. However, the ball mill will be designed to grind to 74 µm, and this will provide flexibility with respect to optimizing recovery, or throughput. Operational flexibility is further enhanced by a mobile crushing plant that is designed to operate on two shifts and has a capacity about double that of the mill.

Average annual precious metal production is projected to be about 50,500 oz of gold and 1.1 Moz of silver. Forecast gold production increases from 55,000 oz in Year 1 to a peak of 74,000 oz in Year 2 and then gradually decreases to approximately 29,000 oz in Year 5. Silver production is forecast to gradually increase from 0.60 Moz in Year 1 to a peak of approximately 1.60 Moz in Year 4 and then declines in Year 5 to 0.90 Moz. Low grade ore produced during the mine life will be stockpiled adjacent to the ROM ore stockpile and processed for about three months after mining operations are completed during Year 5.

16.5 Design Criteria

The plant has a nominal throughput of 1,000 t/d at a 90% availability. Table 16-2 summarizes the major design criteria for the Casposo Project.

Table 16-2
Summary of Major Process Design Criteria

Criteria	Units	Value
Annual throughput	t/a	365,000
Annual operating days	days	365
Daily throughput	t/d	1000
Crusher availability	%	75
Mill availability	%	90
Crushing method		Jaw-Cone-Screen
Comminution method		Crush-Ball Mill
Fine ore storage capacity	h	24
Bond ball mill work index average	kWh/t	19.0 metric
Primary Grind P80 for operating cost	microns	105
Primary Grind P80 for design	microns	74
Ore abrasion index		0.80
Gravity concentrate recovery method		Centrifugal
Intensive Cyanidation Batch Reactor	kg	800
Leach Residence Time	h	60
Pregnant solution recovery method		Wash Thickener + Belt Filter
CCD Settling Rate	t/m2/d	20
Effective Filtration Rate	Kg/m2/h	400
Precious metal recovery method		Leach-Merrill Crowe
Merrill Crowe feed grade Design	m^3/h	150
Cyanide wash recovery method		Belt Filter
MC Design Bleed	%	15
Cyanide destruction method		SO_2/air
Precipitate dewatering method		Mercury Retort
Overall gold recovery	%	93.7
Overall silver recovery		80.6
Tailings disposal method		Dry Stacking

16.6 Plant Equipment

The major equipment in the plant is summarized in Table 16-3. In general the flowsheet is based on industry standard processes and equipment. The equipment is priced as new and based on budget quotes.

Table 16-3
Plant Major Equipment Summary

Components	Dimension	No. of Units
Primary Crusher	Jaw, 530 mm x 800 mm , 75kW	1
Secondary Crusher	Cone, 910 mm , 100 kW	1
Ball mill	3.96 m x 5.18 m, 1125 kW	1
Gravity Concentrator	500 mm, centrifugal	1
ICU Batch Reactor	1 t/d	1
Leach tank	9.8 m diameter x 11.0 m high	6
Leach agitators	3.3 m diameter, 37.5 kW	6
CCD thickeners	10 m diameter	2
CCD clarifier	10 m diameter	1
Horizontal Belt Filters	80 square meters	2
Pregnant solution tank	12.0 m diameter x 12.0 m high	1
Barren solution tank	12.0 m diameter x 12.0 m high	1
Solution clarifying filters	1.67 m diameter – 31 plate	2
Crowe tower	1.5 m diameter x 6.7 m high	1
Zinc filter presses	1.21 m x 1,21 m – 44 plate	2
Precipitate retort	500 kg capacity	1
Furnace	3 cubic ft capacity, 200 kW	1
Cyanide destruction tank	4.27 m diameter x 4.27 m high	1

16.7 Metallurgical Recovery

An overall a life-of-mine gold and silver recovery of 93.7% and 80.6% is projected in Table 16-4. The annual mill production schedule recoveries are calculated based on the above correlations.

Table 16-4
Overall Life of Mine Recovery

	Head	Tail Calc	Extraction	Sol Loss	Misc Loss	Overall Recovery
	g/t	g/t	%	%	%	%
Gold	4.68	0.27	94.4%	0.5%	0.2%	93.7%
Silver	114	21	82.1%	0.5%	1.0%	80.6%

16.8 Run of Mine Ore Storage, Blending and Crushing

Run of mine ore (ROM) will be hauled to the crushing plant in 36 t road trucks. About half the ore will be directly dumped into the crusher feed hopper. The rest of the ore will be dumped at elevation into an excavated ROM ore receiving bunker. Individual ROM ore grade control piles will be maintained in this depository and blended to control the plant feed grade during reclaiming. A front-end loader will reclaim the ROM stockpiled ore and dump into the crushing plant feed bin. The bin will have a capacity of twelve cubic metres and will be protected by a fixed grizzly with 400 mm apertures.

The crushing plant will be completely portable and will operate in two stages in closed circuit producing a product that will be 80% -13 mm. The conveyors, crushers and screen will all be on-board the chassis of the crushing plant, which will have a short wheel base and tight turning radius to allow transport on highways and movement to other possible crushing sites in the future. The portable crushing plant will be placed on a simple concrete pad against a retention wall and hydraulic legs will allow levelling and stabilization. The excavated retaining rock wall will be constructed to support a loading ramp. This arrangement allows direct or front end loader feeding of the crusher feed bin. All electrical motors, instrumentation and automation controls will be contained on-board. Audible alarms and equipment interlocking will provide operational safety measures.

Ore will be withdrawn from the surge pocket by a variable speed vibrating grizzly feeder. The feeder will have a long grizzly section of 1.25 m with 75 mm apertures for efficient fines removal. Grizzly oversize will be fed into a 530 mm x 800 mm jaw crusher with a closed side setting of about 60 mm. A magnet suspended at the discharge end of the feeder will keep oversize tramp iron out of the crushing circuit. Jaw crusher product will fall onto a conveyor belt that will transport it to a 1.8 m x 4.9 m double deck vibrating screen. The apertures of the upper and lower screen decks will be 50 mm and 13 mm respectively. This conveyor belt will also receive the grizzly undersize and the cone crusher product. The on board conveyors handling crushed ore will be 610 mm wide and are reversible. A short heavy duty discharge belt conveyor conveys the screen undersize to a chute for transfer to a fine ore bin feed conveyor, or to a separate emergency stacking conveyor.

Screen undersize will be conveyed to a 1,000 t live fine ore bin. Screen oversize will be fed to a 910 mm secondary short head cone crusher installed with a coarse cavity feed opening of 100 mm and a 13 mm closed side setting. The cone crusher will be equipped with automatic tramp release and hydraulic bowl rotation for fine control setting adjustments and to simplify liner change. Cone crusher product will report to the vibrating screen in closed circuit.

Cranes with hand hoists for 2 t lifts will be provided for jaw and cone crusher maintenance. Dust control in the crushing circuit will be achieved using a water dust suppression system at critical locations.

16.9 Fine Ore Storage & Reclaim

The fine ore bin will have a nominal live capacity of 1,000 t, or the equivalent of one day's operation at the design throughput rate. The fine ore bin will be closed and its discharge cone lined with 20 mm ASTM 242 plates. A vent fan and baghouse will be installed on the bin to control dust emissions. Crushed ore will be reclaimed from the bin by 1,220 mm wide belt feeder and transferred to a 610 mm wide ball mill feed conveyor to the process plant.

A four-idler weightometer on the main feed conveyor will measure the feed tonnage for metallurgical accounting and provide input for grinding circuit control.

Fine ore can also be reclaimed from the emergency stockpile by front-end loader and dumped into a mobile feeder. This provides operational reclaim flexibility to feed onto the fine ore bin conveyor, or onto the tail of the fine ore bin reclaim belt feeder, in the event the bin is empty or its outlet is blocked.

16.10 Grinding and Gravity Concentration

Fine ore will be ground to 80% passing 105 µm in a single stage ball mill grinding circuit to produce a product suitable for leaching. The grinding circuit will consist of a 3.96 m diameter x 5.18 m ball mill driven by a 1,125 kW wound rotor induction motor.

The ball mill will operate in closed circuit with a cyclo-pac consisting of four 250 mm diameter cyclones (three operating, one standby). The cyclo-pac will be fed by two 200 mm x 150 mm variable speed pumps (one operating, one standby). Oversize tramp material from a trommel screen installed on the mill discharge will dump into a concrete collection bunker for subsequent recycling.

A centrifugal concentrator gravity recovery circuit installed on a bleed stream of the grinding circuit cyclone underflow will recover about 15 to 25% of the available gold. About 25% of the total cyclone underflow product will be treated in the gravity circuit, while the remainder will return to the ball mill. The feed to the gravity circuit will initially pass through a 10 mesh scalping screen to remove any oversized material. The screen oversize will be combined with the gravity concentrator tailings and directed to the mill discharge pumpbox. The screen undersize will flow by gravity to a 500 mm centrifugal gravity concentrator to recover a gold and silver gravity concentrate. Fluidization water for the concentrator is provided from a booster tank and pump. To control scaling of the concentrator fluidization ports an antiscalant is added to the water. The operation of the gravity concentrator fill and flush cycles is completely automated.

Periodically, the concentrator will be flushed to the concentrate feed cone of an intensive cyanidation unit (ICU). Excess water will be decanted using a decant valve. The concentrate will discharge from the feed cone by gravity and be leached in a batch ICU with a capacity of 1 t/d. The ICU will be a modular skid mounted structure and will be supplied complete for ease of installation. The operation will be manual. The precious metal leach solution produced by the ICU will be treated in the MC circuit described below and the solid tailings will be recycled to the ball mill.

In normal operation the cyclone overflow product, at 45% WT solids and a P_{80} of 105 µm will gravitate to a mill product pumpbox and be pumped to the leach circuit. Operational flexibility is further enhanced by providing a diverter box that is designed to allow the first thickening stage of countercurrent decantation to operate as a grind thickener in the event leaching at higher densities is desirable.

Barren solution from the MC circuit is pumped to the grinding circuit as grind water. Protective alkalinity is maintained in the mill circuit by quick lime addition to the mill feed conveyor. Washdown and spills in the grinding area will be collected by floor sumps and pumped to the cyclone feed pumpbox.

The ball mill is installed in the open and a marine style hoist is installed on the cyclone platform structure adjacent to the mill for cyclone maintenance and ball loading. Automatic lubrication systems will be provided for the ball mill girth gear and trunnions. The ball mill, retractable feed spout and discharge chutes will be equipped with rubber liners. The gravity concentrator is installed on the operating platform directly beneath the cyclones and immediately above the ICU that is installed in a secured and fenced area on the basement floor.

16.11 Leaching

The ground ore will be leached in a traditional agitated tank leaching process for the recovery of cyanide soluble gold and silver. Mill product will be pumped to the first of six 9.8 m diameter x 11.0 m high mechanically agitated leaching vessels with a capacity of 750 m^3 providing a total residence time of 60 hours. The pH of the pulp will be adjusted if necessary to 11 by adding hydrated lime slurry and cyanide solution (maintained at a concentration of 0.8 to 1.0 g/L) and compressed process air will be metered into the tanks to promote gold and silver dissolution. Tailings from the last leach tank can also be drawn from a mid level position in the tank to allow five to ten hours of surge time between the leach and CCD and filtration plant sections. This allows operational flexibility by providing for full or partial planned maintenance shutdowns in these sections while the mill is still running. Prior to this maintenance the level in the last leach tank will be drawn down to accommodate the surge volume required.

The leach tanks are of steel plate construction and installed on pedestals in three rows of two, with a one meter drop between each row to promote gravity flow. The leach agitators and shafts are rubber lined and can be split to facilitate maintenance. Beams and manual hoists are provided for agitator removal and maintenance.

Washdown and spills in the leach area will be collected by a trench and floor sump and pumped back into the leach. The bund area will accommodate the entire contents of a single leach tank in the event a tank has to be emptied in an emergency. The emergency overflow from the leach area directs spillage into a trench that discharges into the plant process solution pond.

A prefabricated laboratory cabin on the operating leach platform is provided for conducting cyanide concentration control titrations. Access to the leach is provided by stair cases at the north and south end of the leach, the latter connects to the cyclone platform of the grinding mill.

16.12 Precious Metal Solution Recovery

Pregnant precious metal solution will be recovered from the leached tailings in a hybrid CCD thickener circuit and horizontal belt filter arrangement.

Leached slurry from the last vessel will gravitate to the first of two stages of 10 m diameter high-capacity thickeners arranged for counter-current decant (CCD) washing. Flocculant solution will be added to the CCD thickeners as a settling aid. This circuit will wash and thicken the slurry and separate the dissolved precious metals from the leach residue prior to final dewatering and washing on a belt filter. Belt filter filtrate from the first wash zone and MC barren solution will be recycled to the wash thickeners as wash water. Pregnant wash solution overflow from first wash thickener will overflow to a solution clarifying thickener. The clarifier will be equipped as an identical high capacity thickener to provide additional operational flexibility as a standby CCD thickener when one normal stage of CCD is either being maintained, or used as a grind thickener. The clarifier overflow will be pumped to a standard Merrill-Crowe zinc precipitation process. The clarifier underflow with settled solids will intermittently be pumped out to the first CCD stage. Washdown and spills in the CCD area will be collected by a floor sump and pumped back into the CCD. Two underflow pumps (one operating variable, one fixed standby) are provided for maintenance redundancy and operating flexibility.

Underflow slurry from the second CCD stage will be pumped to one of two 80 m^2 horizontal vacuum belt filters, dewatered, washed in two stages and finally rinsed. The first wash will be with barren solution to recover most of the residual pregnant solution precious metal values. This filtrate is recycled to the CCD wash thickeners as wash.

The leached tailings, following filtration to recover most of the precious metals, will be washed and rinsed on the same belt filter to remove most of the cyanide. The cyanide wash and rinse solution from this stage will be collected separately for the destruction of cyanide using the conventional SO_2/air process. The detoxified solution will be recycled to the belt filter as wash solution. This will minimize the fresh water requirements for the process. The rinse stage at the end of the belt filter will provide fresh water make-up to the circuit and will wash additional residual cyanide from the filter cake. Filtered tailings at 82% solids will discharge and the filter cake will be collected by a gathering conveyor and stacked on a concrete slab pad adjacent to the filter plant. It is not expected that the filtered tailings solids will contain residual free mobile moisture. However any bleed solution from the tailings solid that occurs on the pad will be collected in a solution trench and directed to the process water pond and recycled to the process. During day shift the tailings will be trucked to a lined tailings management facility and stacked in compacted lifts.

Each belt filter will have a dedicated vacuum pump. The vacuum pumps will be cooled by fresh water which is in closed circuit with a packed forced air cooling water tower. This is done to avoid excess dilution in the circuit water balance and minimize fresh water usage.

Washdown and spills in the filter area will be collected by a floor sump and pumped back to the CCD area. Filtrate receivers are provided with dual pumps for maintenance redundancy. The open design allows all the wearing parts to be easily accessible for routine maintenance purposes. Operational flexibility is further enhanced by sizing each filter for a design capacity about fifty percent more than that of the nominal mill throughput. During breakdowns, or planned major filter maintenance of about eight hours, the plant can continue to operate at 75% of capacity with only one filter operating. Alternatively in a planned maintenance shutdown of one filter, the mill can continue to operate at 100% capacity for eight hours until the leach surge capacity is utilized.

16.13 Merrill Crowe

Clarifier overflow containing most of the recovered precious metals will be pumped to a Merrill Crowe (MC) circuit for recovery. The overall precious metal solution recovery from the CCD and belt filter is expected to be about 99.5% based on an overall nominal wash ratio of about 2.5 and results in a MC solution feed volume of 125 m^3/hour. The plant is designed for a feed of about 150 m^3/hour which allows operational flexibility for catch-up after MC clean-ups, or to use a higher wash ratio of about 3.0 when higher ore feed grades are being processed or if one of the thickeners is placed on pre-leach duty. The pregnant solution will be further clarified, de-aerated and gold and silver will be recovered using zinc precipitation. The precipitated precious metal sludge will be retorted and smelted to produce dorè bars.

A 12.0 m x 12.0 m pregnant solution holding tank provides about 8 hours storage capacity at the MC design feed rate. From there the solution will be pumped to two 1.67 m diameter-31 plate horizontal pressure-leaf clarifying filters (one operating, one standby) to remove fine residual solids from about 100 ppm to below 10 ppm. Diatomaceous earth will be used as the filtering medium. The filter will require a pre-coat of diatomaceous earth as a filter aid. The filters will be back-washed periodically to remove solids, with the backwash solution pumped to the wash CCD thickener. When the settling clarifier is being used temporarily as a CCD wash thickener the suspended solids content of the clarifier feed will be higher and this will necessitate a shorter operating time between backwash cycles.

Clarified pregnant solution will be pumped to a packed bed de-aeration tower of 1.5 m diameter x 4.3 m packed bed, 6.7 m tower height, after which zinc dust will be added to precipitate precious metals. A dissolved oxygen meter monitors the efficiency of de-aeration. Two vacuum pumps (1 operating, 1 standby) provide the vacuum required.

Cyanide solution, zinc dust and lead nitrate will be added to the suction side of the precipitation filter press feed pump along with de-aerated pregnant solution. Residual lead nitrate in barren solution also assists in the leaching process. The zinc-gold-silver precipitate will be collected in two 1.21 m x 1.21 m - 44 plate-and-frame filter presses in the refinery. Filtrate from the presses will be collected in the 12.0 m x 12.0 m barren solution tank and recycled by two barren solution pumps (1 operating and 1 standby) to the CCD wash thickener and circuits as wash. A bleed of barren solution is also directed to cyanide destruction for management of zinc levels in the MC and to provide a detoxified solution for cyanide washing in the filter plant. Two barren solution gland service water pumps are also installed on the barren tank for process slurry pump gland seal duty.

The MC plant is installed in a pre engineered building with steel frame and cladding to be assembled on site. The building floor will be concrete slab on grade on well-compacted soils. All the MC equipment will be supplied in containerized modular structures and installed on skids for ease of installation on the concrete building floor. Washdown and spills in the MC area will be collected by a floor sump and pumped back to the CCD wash thickener area. Offices and a dry laydown area are provided in the building for equipment maintenance and reagent pallet storage. The MC feed and barren tanks and de-aeration tower are installed outdoors on a curbed slab adjacent to the refinery.

16.14 Refinery

Precipitate will be manually "discharged" from the refinery filter presses into wheeled collection trays and placed into a drying retort. Approximately 1,000 kg of dry precipitate will be produced each week.

Average concentrations of deleterious mercury in the ore and its zinc precipitate are expected to be below trace. However, a mercury retort is included in the refinery design to manage risk exposure associated with any possible peak concentrations. This will dry the precipitate as well as drive off any mercury present and collect it in a condenser and filter system ahead of smelting.

A car tray will place the precious metal precipitate into a resistance-heated furnace equipped with tray-handling rollers. Any mercury fumes generated from the precipitate over a 24-hour period, while the precipitate heats to 750°C under a slight vacuum, will be condensed in a water-jacketed condenser. Any residual by-product mercury collected will be stored in flasks for sale. Annual mercury production is expected to be less than 50 Kg. Final trace amounts of mercury will be stripped from the gas stream by a carbon filter prior to discharge.

The dried retort precipitate will be mixed with fluxes (borax, soda ash and silica) and charged into a 200 kW electric induction furnace of three cubic feet (454 kg steel) crucible capacity. The furnace is nose tilting with hydraulic activation. The mixture will be melted to allow the gold and silver to be separated from the other metals, which will report to the slag phase. The gold and silver will be poured into dorè mould bars and cooled. The dorè bars will be shot blasted clean, weighed, stamped, packed and stored in a vault for secure shipping by armoured truck to an off-site refinery for final processing. Approximately 25,000 oz (750 kg) of dorè will be produced each week using a five d/wk melt schedule.

A fume control manifold on the furnace will collect furnace off-gases and precious metal bearing dusts and particulates. Furnace off-gases will be cleaned through a bag filter. The treated gas will be discharged to the atmosphere. The bags will be emptied of reclaim back to the flux addition operation prior to recharging the induction furnace.

Slag from the induction furnace will be collected, crushed and tabled in the refinery. The crushed slag will be passed over a gravity table to recover any residual dorè. Any precious metal collected will be returned to the induction furnace for remelt. The slag will then be recycled to the grinding circuit.

The refinery is installed in a concrete block building adjacent to the MC building. Washdown and spill in the refinery area will be collected by a floor sump and pumped back to the CCD wash thickener area through an in-line filter. Offices and change house wash facilities for the refinery staff are provided. Personnel access to the refiner is controlled by security at the change house entrance. A truck access door to a laydown area will be provided for maintenance and loading and shipping of dorè. Access will be controlled by a secured fence installed on the truck entrance on the outside the building.

16.15 Cyanide Destruction

A barren solution bleed stream from the MC and filtrate solution from the cyanide wash zone of the belt filter will be pumped to a conventional SO_2-Air cyanide tank destruction system circuit to destroy cyanide and to control zinc values in the MC circuit. Process air will be blown into the agitated reactor and sodium metabisulphite, copper sulphate and lime will be metered to maintain the pH at about 8.0 to 9.0.

At the design throughput of 30 m^3/h the cyanide destruction reactor tank of 4.27 m x 4.27 m cyanide will provide ninety minutes retention time to oxidize cyanide to cyanate. Some cyanate will further decompose to ammonia and other carbon and nitrogen products. The cyanide destruction process will also oxidize any trace metal complexes for subsequent precipitation and fixation as metal oxyhydroxides in the tailings filter cake when it is washed with this solution. The detoxified solution is expected to contain less than 5 ppm (CN_{WAD}). Some of this will be pumped to the horizontal belt filter cyanide wash zone as wash. No cyanide test work was conducted for the Feasibility Study. However full metal and cyanide speciation of all leach solutions was conducted and the analyses results indicate no problems should be expected processing these by the conventional SO_2-Air process to destroy cyanide.

Excess detoxified solution will be directed to a lined process water pond for evaporation and conditioning. Any residual CN_{WAD} and cyanate remaining in the solution stream after treatment will decompose in the process water pond by exposure to natural sunlight to ammonia and other carbon and nitrogen products. The average free cyanide concentration in the process water pond during normal operation is expected to be less than 1 ppm. Some of this solution will be evaporated to maintain the plant water balance and the rest after conditioning will be reclaimed as process water.

The cyanide destruction reactor tank will be installed in a curbed concrete slab in the open, closed and vented to the atmosphere. Area washdown and spill will be contained and collected by a floor sump and pumped back into the reactor.

16.16 Tailings Management Facility

During day shift the dewatered tailings will be hauled to a lined dry tailings management facility and stacked by a bulldozer in compacted lifts. At least a portion of the facility will be formally compacted. This is a simpler, less environmentally-invasive, sediment control structure than other alternatives. The unsaturated state of the tailings, together with the increased placement density, also eliminates liquefaction and flowsliding potential under even the most severe seismic loading. Transpiration of the remaining interstitial water in the solids to the surface of the facility, and subsequent evaporation, results in the continued progressive drying of the solids as well as the complete destruction of any minor residual cyanide in that solution by exposure to natural sunlight. Finally, dry stacking provides the ability to carry out progressive reclamation during operations, and facilitates rapid final reclamation upon closure. The design, operating placement strategy, compaction and works for closure and reclamation of this facility are described in more detail in Section 18.2.1 and 18.2.5.

16.17 Plant Services

16.17.1 Water System

The process plant will use five kinds of water: barren process solution, process water, fresh water, fire water and potable water.

Barren process solution is supplied from the MC barren solution tank and is used for grinding, CCD and filter wash and process slurry pump gland seal.

Process water is produced by detoxifying barren solution and cyanide wash filtrate using an SO_2-Air destruction plant and reclaimed from the process water pond after conditioning to a process water tank. Process water is used as gravity concentrator fluidization water, general area wash down and belt filter wash.

Fresh water will be used for reagent mixing, cooling water and eye/safety showers. Fresh make up water will be obtained from fresh water wells at Vallecito and pumped to a fresh water tank at the plant site. The bottom portion of the fresh water tank is utilized for the firewater system. A water treatment plant is provided to produce potable water.

16.17.2 Process Air

Two (1 operating, 1 standby) 9,000 m^3/h, 750 KPa process air compressors will supply low-pressure air to the leaching and cyanide destruction vessels. Only one unit will be needed at any one time, with the other serving as a spare.

16.17.3 Plant Air

Two (1 operating, 1 standby) 3,000 m^3/h, 1500 KPa plant air compressors will supply air to various outlets for general service throughout the plant including hose stations and service utility. Service use areas will include dust baghouse shakers, pneumatic maintenance tools and a coalescer that can supply instrument air as needed. The instrument air will be dried and stored in satellite receivers in various areas of the plant. Compressors will be equipped with inlet filters to keep out ambient dust.

16.17.4 Metal Accounting and Samplers

To ensure proper metallurgical control of the plant, the following shift and metallurgical metal balance sample points will be established:

- mill feed (manual belt sample cuts)
- mill product (pressure sampler and vezin cutter)
- intensive cyanidation pregnant solution (solution wire sampler)
- leach tailings (sample pump and vezin cutter)
- MC pregnant solution (solution wire sampler)
- MC barren solution (solution wire sampler)
- filter tailings cake (manual truck top sample)
- zinc precipitate (manual sample)
- dorè prills (manual sample)
- slag (manual sample)

Slurry samples will be collected from the critical streams on a continuous basis by a combination of primary pressurized pipe or pump samplers and delivered to secondary vezin samplers. Sample flows exiting the vezins will be directed back into the process.

Daily mill feed tonnage will be obtained from a weightometer installed on the mill feed belt and this will be reconciled on a daily basis with the tailings weight determined by tailings trucks on the weighbridge, both adjusted for moisture content. Solution flows will be measured by flowmeters. Scales will be provided in the refinery for weighing zinc precipitate, dorè bars and slag.

16.17.5 Reagents

The process will use conventional suite of gold and silver processing reagents. Their point of use and expected consumption are shown in Table 16-5.

Table 16-5
Process Reagent Usage

Reagent	Use	Approx. Daily Max. Usage	Annual Consumption	Solid/Liquid	Normal Delivery Format	On-Site Storage Volume
Grinding Balls 2.5"	Grinding	1.5 t	550	Solid	Drums	Pallet drums
Quick Lime	pH control	0.5 t	180 t	Solid	Bulk pebble lime	40 t
Sodium cyanide	Leaching	1.1 t	400 t	Solid	1 t box bags	Pallet
Sulphuric acid	Filter aid	5 kg	1.5 t	Liquid	20 gal drums	Pallet drums
Zinc (granular, powder)	Precipitation	75 kg	30 t	Solid	25 kg drums	Pallets
Sodium metabisulphite	CN Destruction	1.5 t	550 t	Solid	1 t supersacs	Pallet
Copper sulphate	CN Destruction	5 kg	2 t	Solid	25 kg bags	Pallet
Hydrated Lime	CN Destruction	0.8 t	300 t	Solid	1 t supersacs	Pallet
Anti-scalant	Water Treatment	0.03 m^3	10 m^3	Liquid	20 gal drums	Pallet drums
Flocculant	Flocculation	120 kg	40 t	Solid	25 kg bags	Pallet
Diatomaceous earth	Filter aid	60 kg	20 t	Solid	25 kg bags	Pallet
Lead nitrate	Leaching/Precip.	30 kg	10 t	Solid	25 kg bags	Pallet
Borax/Sodium Nitrate	Refinery Flux	90 kg	30 t	Solid	25 kg bags	Pallet

Procedures will be established for the storage and handling of reagents to ensure personnel safety and control of potential impacts to the environment. The reagent mixing building will be a pre-engineered steel shell. The building floor will be concrete slab on grade on well-compacted soils. Generally, separate mix tanks and day tanks are provided in curbed areas on one level with pump transfer between tanks. Safety shower, sump pumps and vent fans are provided.

Cyanide

Sodium cyanide in solid pellet form will be received in sealed plastic bags in plywood boxes of approximately 1,000 kg capacity. The solid cyanide will be dissolved with process water to prepare a 20% (w/w) solution for storage and distribution. Merrill-Crowe tails solution will be added to the mix tank ahead of cyanide along with sufficient quicklime to ensure that the protective alkalinity of the cyanide solution is maintained. The mixed solution will be pumped to the leach circuit through a ring main system.

Quick Lime

Quick lime will be supplied in bulk into a 4.3 m x 11.2 m, 60 deg, 80 t capacity lime silo. It will be reclaimed by a screw feeder directly to the mill feed conveyor.

Flocculant (Settling Aid)

Dry flocculant will be received in 25 kg capacity HDPE bags. An automatic mixing arrangement will be provided to facilitate hydration of the powder. A 0.5% flocculant solution will be prepared and stored; this solution will be further diluted to 0.05% prior to use in the CCD thickener and filter circuits.

Copper Sulphate

Copper sulphate pentahydrate ($CuSO_4.5H_2O$) will be received in 25 kg bags lined with polythene and mixed in batches to 15% concentration in a mix tank. The mixed batch will be transferred to a holding tank and pumped from there to the cyanide destruction tank.

Sodium Metabisulphite

Sodium metabisulphite will be received in 1,000 kg HDPE supersacs lined with polythene and mixed with fresh water in a mixing tank. The resulting solution will be metered from there to the cyanide destruction tank.

16.18 Process Plant Layout, Buildings and Structures

The site is divided into three zones: Zone 1 is the crushing and ROM stockpile, Zone 2 is the process plant area and Zone 3 is the site services and administration complex. A central pipe rack and ground trench separates the dirty process facility Zone 2 from the clean site services Zone 3. All run off or emergency spillage from Zone 2 is directed into the process water pond via the ground trench for evaporation or recycling within the process. Run off from Zone 3 is directed to a separate site drainage pond.

The process plant and service buildings and structures in the plant area are:

- Electrical rooms. These will be pre-fabricated as modular structures with all equipment installed prior to delivery to site.
- Plant Control room. This will be installed above the mill area electrical room. The control room will also be pre-fabricated complete with all controls and operator stations before delivery to site. It includes a slurry sample preparation area in a separate room.
- Plant Compressors. These will be installed in a concrete block building adjacent to the reagent storage building.
- Reagent Mix and Storage. These facilities are installed in a pre engineered ventilated building on slab with steel frame and cladding to be assembled on site. A dry laydown area is provided for daily pallet storage.

- Cyclone Structure. This structure is installed adjacent to the grinding mill and houses the cyclones, maintenance crane and gravity plant equipment. The ball mill is open and the mill foundations will be constructed of concrete extending to bedrock. The grinding area floor will be concrete slab on grade on well-compacted soils.
- Pipe Rack. A central walkway and distribution piping (supported on low level pipe racks) will connect the grinding, leach, CCD thickeners and filters with the upper operating platform of the associated equipment. Beneath this a trench will collect run-off and spillage and direct it to the process water pond.
- Merrill Crowe and Refinery Building. The MC plant is installed in a pre engineered building with steel frame and cladding to be assembled on site. The building floor will be concrete slab on grade on well-compacted soils. The refinery structure adjacent to this will have concrete block walls.
- Power Plant. The generators will be installed in the open on a concrete slab on grade on well-compacted soils. The control room will be pre-fabricated as a modular structure with all equipment installed prior to delivery to site.

A 3-dimensional rendering of the overall plant site is shown in Figure 16-1 and a simplified depiction of the overall process flowsheet is shown in Figure 16-2.

Figure 16-1
Casposo Plant Site

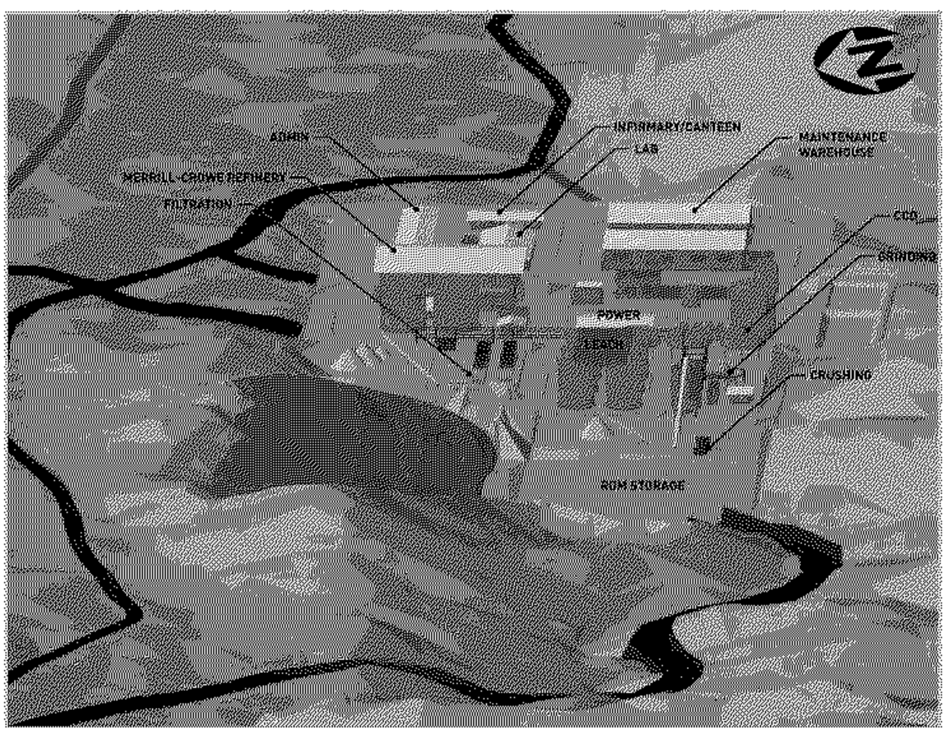
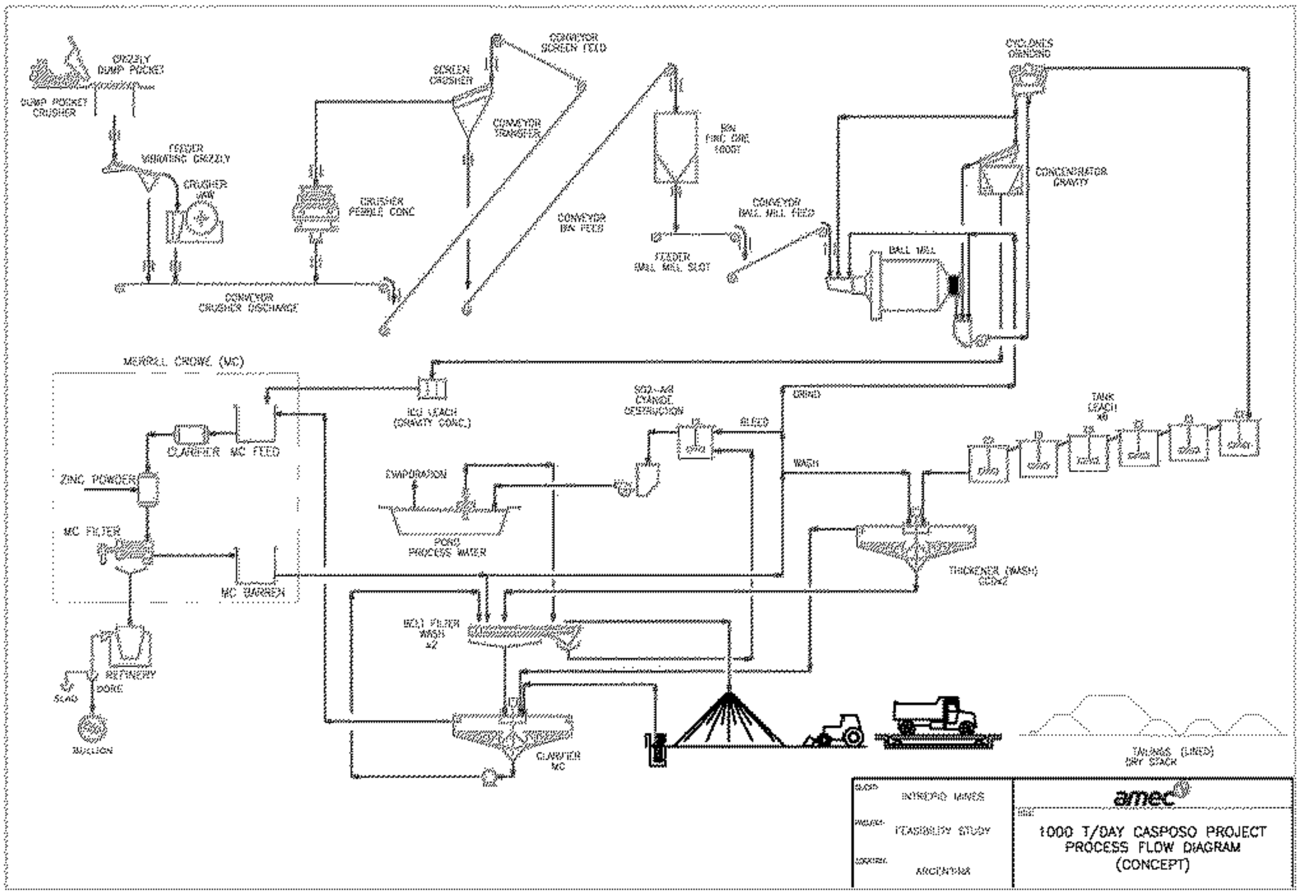


Figure 16-2
Casposo Simplified Process Flowsheet



17.0 MINERAL RESOURCE AND MINERAL RESERVE ESTIMATES

17.1 Mineral Resources

17.1.1 Introduction

The resource models for the Casposo Project Feasibility Study (Kamila and Mercado Zones) were prepared by Intrepid and its independent consultants. Rodrigo Marinho, CPG Senior Geologist, (AMEC, Santiago), audited and reviewed these models and prepared the tables for the mineral resource estimates dated October 10, 2006. Mr Marinho is the Qualified Person (QP) for this resource.

The supporting drillhole, pit and trench databases were constructed by Intrepid and composites and 3-dimensional solid models were constructed by Intrepid utilizing commercial mine modelling software (Gems®). All material below the Kamila and Mercado Zone topographic surface was considered. For the purposes of the model review, interpolation of gold and silver was done by AMEC with ordinary kriging methods using Gems® modelling software.

The resource models for each of the zones mentioned above contain blocks estimated for gold and silver in four mineralogical domains (veins) at Kamila and one at Mercado. The blocks have been classified as Indicated or Inferred based on the relative confidence in the supporting data for each block. Grades have not been estimated beyond the limits of these classifications. The classified mineral resources and reserves for the Casposo Project were prepared in conformance with the CIM Mineral Resource and Mineral Reserve definitions referred to in National Instrument (NI) 43–101, Standards of Disclosure for Mineral Projects.

17.1.2 Estimation Database

Drill holes (130), pits (5) and trenches (70) were used to support the geological modeling and estimation of grades. AMEC created a drill hole selection file based on information provided by Intrepid (Table 17-1). The supporting data includes drill holes, up to drill hole CA-06-174 even though an additional 25 holes (up to CA-06-199), have been drilled but these were considered to be past the cut-off date for the estimate, since the analytical data was pending at the time the database was considered final.

Table 17-1
List of Drill holes, Pits and Trenches used for Resource Estimation at Casposo

Pit		Trenches		Drill Holes	
BLKBSE	T-01	TK5403	CA0027	CA04102	CA05144
BULK-A	T-02	TK5503	CA0029	CA04103	CA05145
BULK-B	T-03	TK5603	CA0030	CA04104	CA06146
BULK-I	T-04	TK5703	CA0031	CA04105	CA06147
BULKM	T-05	TK5803	CA0032	CA04106	CA06148
	T-06	TK5903	CA0033	CA04107	CA06155
	T-07	TK6103	CA0035	CA04108	CA06156
	T-09	TK6303	CA0037	CA04109	CA06157
	T-10	TK6403	CA0040	CA04110	CA06158
	T-11	TK6503	CA0041	CA04111	CA06159
	T-12	TK6603	CA0042	CA04112	CA06160
	T-19	TK6703	CA0043	CA04113	CA06161
	T-20	TK6803	CA0044	CA04114	CA06162
	T-22	TK6804	CA0347	CA04115	CA06163
	T2502A	TK6903	CA0348	CA04116	CA06164
	T2502B	TM-02	CA0349	CA0480	CA06167
	T2502C	TM-03	CA0350	CA0481	CA06168
	T26-02	TM1303	CA0351	CA0482	CA06170
	T27-02	TM1403	CA0352	CA0484	CA06171
	T28-02	TM1703	CA0353	CA0485	CA06172
	T3002A	TM1803	CA0354	CA0486	CA06173
	T3202A	TM2003	CA0355	CA0487	CA06174
	T3202B	TM2103	CA0356	CA0488	CA991
	T33-02	TM2203	CA0357	CA0489	CA9910
	T36-02	TM2303	CA0358	CA0490	CA9911
	T39-02	TM2403	CA0359	CA0491	CA9914
	T40-02	TM2503	CA0360	CA0492	CA9915
	T4102A	TM-A	CA0361	CA0493	CA9916
	T4102B	TM-B1	CA0362	CA0494	CA9917
	T4102C	TM-C	CA0363	CA0496	CA9919
	T42-02	TM-D1	CA0364	CA0498	CA992
	TK4203	TM-D2	CA0365	CA05118	CA9920
	TK4303	TM-E	CA0366	CA05119	CA9921
	TK4403	TRK-95-05	CA0367	CA05121	CA9923
	TK5303	TRK-96-05	CA0368	CA05122	CA9924
			CA0371	CA05123	CA9925
			CA0373	CA05124	CA993
			CA0374	CA05128	CA994
			CA0376	CA05129	CA996
			CA0377	CA05130	CA997
			CA0378	CA05131	CA998
			CA0379	CA05132	CA999
			CA04100	CA05140	
			CA04101	CA05143	

17.1.3 Geological Interpretation

AMEC reviewed in detail all the existing geological sections that were constructed by Intrepid from Kamila (18) and Mercado (17), and at least two drill hole logs in each section. The original logs were visually cross checked against the information displayed in the sections. AMEC reviewed the geometry of the interpreted geological shapes in the geological sections, and found them to be an acceptable representation of the deposit.

AMEC recognizes that small individual drill hole segments may be misclassified and may fall into neighbouring geologic zones, and that minor inconsistencies in logging over time may have led to other minor interpretive conflicts, but taken as a whole, the interpretation respects the data recorded in the logs and the sections, as well as the interpretation from adjoining sections, and is consistent with the known characteristics of the deposit type.

The lithologic model has been appropriately constructed and is in conformance with standard industry practices.

17.1.4 Domain Interpretation and Modeling

Intrepid defined five domain zones to represent the different veins systems at Kamila and Mercado. Domains were defined based on lithology, structure and grade boundaries.

At the Kamila zone, domains are referred to as Aztec vein, B vein, Inca vein and High Grade. The High Grade (HG) domain was determined using drill hole intercepts that are above 10.00 g/t Au equivalent within the other domains to separate the high-grade population from the lower grades. At Mercado only one domain is defined.

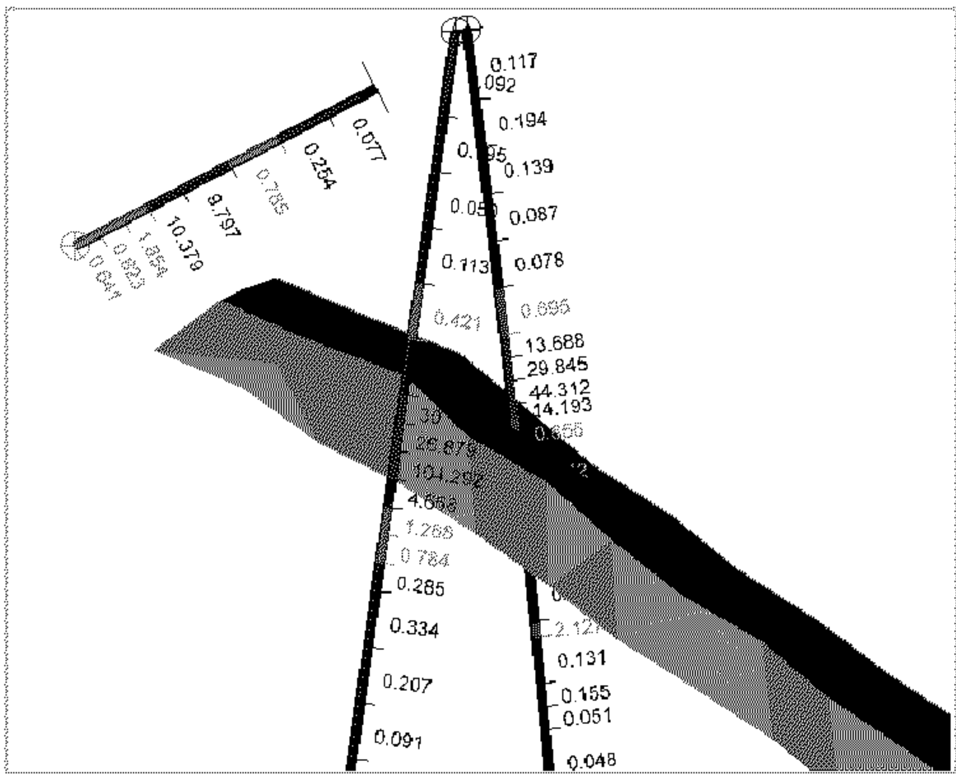
Intrepid used only one set of vertical sections, east–west direction spacing of 25 m, to interpret and model Aztec, B and Inca veins. Interpreted polygons honour the drill hole intersections and guarantee that ore intervals will not be left outside the solids. Interpretation of the orebodies using two sets of orthogonal sections produces an enhanced three dimensional representation and AMEC recommends interpreting a second set of sections, vertical or by levels, for future studies.

For the interpretation of the HG domain, Intrepid personnel used east–west-oriented vertical sections spaced every 25 m to generate bench interpretation every 3 m. Only the bench interpretation was used to generate solids for the High Grade domain and although the solids fit well with the bench polygons, the vertical sections generally do not reconcile well at the solids terminations. AMEC recommends that two interpretations, both vertical and bench polygons, are necessary for modeling, to produce solids that represent the current drilling information.

AMEC verified the HG domain solid against the composites inside the shell and found 91 composites out of 272 that have AuEq grades lower than 10.00 g/t. The interpretation of the HG zone is continuous and this explains the presence of composites with grades lower than the defined limit. AMEC considers this internal dilution acceptable and reflects the reality of the deposit.

Some continuous high-grade intervals (AuEq >10.00 g/t) were found outside of the interpreted boundaries of the High Grade domain. Figure 17-1 shows an example close to section 6548350N, where intersections of holes CA04103 and CA04102 were flagged as Aztec vein low grade mineralization. This could cause some dilution since high grades are not confined and adjacent low-grade intervals are used for estimating grades at surrounding blocks. AMEC points to this problem as having locally the potential for upside in grade estimates.

Figure 17-1
Vertical Section 6548350N Showing AuEq High Grades
Intersections not considered in the Interpretation of High Grade Solid



AMEC also reviewed all of the solids that represent the vein domains. The interpretations are primarily based on grade values from drill holes. The drill coverage is not extensive, however sufficient to interpret the solids. AMEC is of opinion that the solids are reasonable however for future models infill drilling is warranted to improve the continuity of the interpreted solids. In addition, modelling of lithology units is recommended to assist with future interpretations.

17.1.5 Exploratory Data Analysis

The database provided to AMEC contains both raw assay data and capped data. AMEC performed basic statistics on this information. Table 17-2 summarizes the statistics of raw data for Kamila and Mercado zones according to the domains defined.

Table 17-2
Drill Hole Raw Data Statistics

Zone	Element	No. Samples	Min.	Max.	Mean	Standard Deviation	Coefficient of Variation
Aztec	Au (ppm)	786	0.009	178.500	2.664	8.623	3.237
	Ag (ppm)	786	0.899	3070.000	66.635	177.881	2.669
B	Au (ppm)	770	0.009	151.000	2.267	7.275	3.209
	Ag (ppm)	770	0.099	1700.000	33.734	94.250	2.794
Inca	Au (ppm)	303	0.020	207.000	3.629	15.266	4.206
	Ag (ppm)	303	0.899	2090.000	115.889	264.769	2.285
Mercado	Au (ppm)	645	0.000	36.900	1.123	2.930	2.608
	Ag (ppm)	644	0.000	2787.000	68.597	213.014	3.105
High Grade	Au (ppm)	223	0.045	346.000	20.381	40.059	1.965
	Ag (ppm)	223	0.999	3400.000	387.342	546.468	1.411

17.1.6 Capping

Intrepid investigated raw assay values to identify high erratic values for gold and silver in the database. Log-normal histograms were used to help defining capping strategies.

Table 17-3 shows the capping values for gold and silver as calculated by Intrepid for each vein and reported in an Updated Mineral Resource Estimate from August 2006.

Table 17-3
Capping Values Defined by Intrepid

Domain	Au (g/t)	Number of Au Assays Capped	Ag (g/t)	Number of Ag Assays Capped
High Grade	120	5	2,000	7
Aztec Vein	35	7	700	13
Inca vein	15	13	500	13
B Vein	30	6	300	11
Mercado	10	8	1,000	8

AMEC verified lognormal probability plots of raw data assays to support the capping values and compared the values defined by Intrepid with frequency distributions of the database. AMEC used the GEMS® database and the selection of drill holes considered by Intrepid to estimate mineral resources.

The High Grade (HG) domain capping value of 120 g/t Au represents 98% of the population and effectively cuts five samples out of a total of 223 assays. AMEC observed a natural break on the lognormal probability curve at around 100 g/t Au, but reducing the capping value to 100 g/t Au would only cut one additional sample. For silver, AMEC agrees with a capping value of 2,000 g/t Ag, which effectively cuts six samples out of the database (instead of seven as interpreted by Intrepid).

For Aztec, the gold capping value of 35 g/t represents 99.11% of the total assays distribution and cuts out seven samples. From the analysis of lognormal probability plots, AMEC is of opinion that a capping value of 20 g/t Au is more appropriate. Only 30 blocks from the Aztec domain have gold grades higher than 20 g/t Au and if these blocks are weighted by the average percentage of the blocks inside the vein solid this will result in only 3,600 t being affected. Therefore, the net impact of those high erratic assay values on the global resources is considered low. Nevertheless AMEC recommends Intrepid review the gold capping strategies applied for the Aztec vein prior to completing the next model update. AMEC considers the silver capping value of 700 g/t Ag to be adequate.

For the Inca vein, 13 samples were capped at a gold value of 15 g/t Au, which represents 89.11% of the total population. AMEC verified the lognormal probability plot and confirms a break in the line at this value. For silver, the capping value of 500 g/t Ag is accepted by displaying a natural break in the lognormal probability plot at 95% of the frequency distribution.

AMEC accepts the capping value of 30 g/t Au used for the B vein data, based on the lognormal probability curve. A total of 8 samples were capped for gold. The capping value for silver is 300 g/t Ag and this removes 11 of 770 samples.

Mercado gold capping of 10 g/t Au represents 98.75% of the data distribution and caps a total of eight samples. For silver, a capping value of 1,000 g/t Ag was used to cap eight samples, or 98.76% of the distribution. AMEC accepts the capping values defined.

Table 17-4 summarizes the statistics for capped raw data.

Table 17-4
Capped Drill Hole Raw Data Statistics

Zone	Element	No. Samples	Min.	Max.	Mean	Standard Deviation	Coefficient of Variation
Aztec	Au (ppm)	786	0.009	35.000	2.384	4.956	2.078
	Ag (ppm)	786	0.899	700.000	60.190	120.859	2.008
B	Au (ppm)	770	0.009	30.000	2.024	4.335	2.141
	Ag (ppm)	770	0.099	300.000	29.189	53.493	1.833
Inca	Au (ppm)	303	0.020	15.000	1.991	3.511	1.764
	Ag (ppm)	303	0.899	500.000	86.806	128.112	1.476
Mercado	Au (ppm)	645	0.000	10.000	0.931	1.558	1.674
	Ag (ppm)	644	0.000	1000.000	38.664	104.831	2.711
High Grade	Au (ppm)	223	0.045	120.000	17.101	23.219	1.358
	Ag (ppm)	223	0.999	2000.000	349.647	452.894	1.295

17.1.7 Composites

Composites of 1 m length were created inside the geological solids. The compositing process starts at the first point on the intersection of drill holes and the solid hangingwall. Non assayed intervals were not utilized in the composite generation process. Intrepid discarded all composites that were shorter than 0.4 m (at the lower domain boundary) to avoid introducing bias from the very short composites. AMEC replicated the process of generating composites and calculated that only 7.5% of the low grade composites and 15% of the high grade composites are shorter than 1.0 m; AMEC considers the results acceptable.

Table 17-5 shows the basic statistics of composites within each domain. The number of composites per domain is greater than the number of raw assays because several original samples are longer than 1 m. Using a composite length shorter than intervals defined for the original sample length would result in several consecutive samples with the same grade. This will generate a larger amount of composites than samples and possibly lead to smoother variograms and lower coefficient of variation (CV) values. The impact on grade estimation could be significant when using a kriging interpolator, however Intrepid used inverse distance to estimate grades and AMEC is of opinion that the results will not be affected significantly.

Table 17-5
Capped Composites Statistics

Zone	Element	No. Samples	Min.	Max.	Mean	Standard Deviation	Coefficient of Variation
Aztec	Au (ppm)	928	0.028	35.000	2.219	4.126	1.859
	Ag (ppm)	928	0.999	700.000	53.505	103.169	1.928
B	Au (ppm)	985	0.009	30.000	2.176	4.248	1.952
	Ag (ppm)	985	0.159	300.000	28.515	47.732	1.673
Inca	Au (ppm)	330	0.019	15.000	1.794	2.904	1.618
	Ag (ppm)	330	0.899	500.000	76.338	105.501	1.382
Mercado	Au (ppm)	863	0.005	10.000	0.932	1.358	1.457
	Ag (ppm)	861	0.299	863.610	34.233	87.325	2.551
High Grade	Au (ppm)	272	0.044	120.000	17.439	21.956	1.259
	Ag (ppm)	272	0.899	2000.000	336.584	412.859	1.226

17.1.8 Variography

Intrepid created variograms for gold and silver for each vein domain, however variograms for composites from the HG domain were not provided to AMEC. Both omni-directional and directional variograms were constructed. Orientations of the variograms were defined by the experience and knowledge of the deposit from the Intrepid geologist. Then, directional variograms were built along mineralization strike, downdip and across dip. Experimental variograms presented by Intrepid for Aztec vein, Inca vein, B vein and Mercado are very well structured for gold values and show a good structure, in general, for silver grades.

Variograms were fitted using spherical models; however the model used for fitting directional variograms did not consider the nugget effect defined from the omni-directional variograms. The modeled sill (C1) values are not the same in each direction and reveal a zonal anisotropy. AMEC's evaluation of the data does not support this conclusion and AMEC is of opinion that the sill should be normalized to an equal value and thus ranges may differ from those interpreted by Intrepid. However, overall AMEC expects the impact of this on grade distribution would be small.

Additionally, the nugget effect should be defined from omni-directional or downhole variograms and the value obtained should be used to model all directional variograms. The nugget effect represents the global grade variance of the deposit regardless the grade continuity direction analysed and that is the reason only one value of nugget effect should be modelled.

Considering that variograms were created to help define search ellipse radii for grade estimation, AMEC examined the variogram ranges modeled and compared them with the search ellipses defined by the first estimation pass. Both, orientation and radii of the search ellipses are in accordance with the variogram parameters.

17.1.9 Block Model Setup

To cover the entire extension of Kamila and Mercado zones, Intrepid created a block model framework with 4,590,000 blocks each with dimensions of 4 m x 4 m x 6 m in the X, Y and Z directions, respectively. The framework is formed by 225 columns (X), 300 rows (Y) and 68 levels (Z) and is not rotated, with the X axis parallel to the Easting and Y axis parallel to Northing.

The block model is a partial percentage type and is designed to report the proper percentage of low grade and high grade material for each block within the geological solids for each domain. Low grade and high grade folders exist to store the following variables in the model:

- Rock Type: domain code according to geological solids
- Density: density value, a constant value of 2.5 is assigned to all blocks
- Percent: percentage of low or high grade material inside each block
- Au–ID: gold values estimated using inverse distance cubed methodology
- Ag–ID: silver values estimated inverse distance cubed methodology
- Au–Eq–ID3: gold equivalent values calculated using a script (inverse distance cubed methodology)
- Indicated: code for blocks that are categorized as Indicated resources

- Inferred: code for blocks that are categorized as Inferred resources
- Class: final resource classification code used for reporting

AMEC verified the percentages of low and high grade materials assigned using visual validation and scripts to check for values over 100 % and below zero. No problems were found and AMEC agrees with the methodology used.

AMEC selected blocks that are inside each solid domain and verified that the correct rock codes were assigned.

Blocks that are outside the domain solids and are above or below the original topographic surface have a rock code = 0 and density values of 0.0. AMEC recommends that waste blocks (outside the domain solids) that have a rock code and density value of 0 be changed with alternate default values so that additional steps are not required to manipulate the data to estimate waste tonnages. This should be done by Intrepid in the next model update.

17.1.10 Grade Estimation Parameters

Gold and silver grades are estimated using an inverse distance cubed (ID3) methodology. Two estimation passes are defined using incremental search ellipsoid radii. Table 17-6 shows the ellipsoid orientations and radii used for each domain:

Table 17-6
Search Ellipsoids Orientation

Domain	Element	Estimation Pass	Ellipsoid Rotation (°)			Ellipsoid Ranges (m)		
			Azim	Dip	Azim	X	Y	Z
Aztec	Au	1	260	-60	350	45	35	10
		2	260	-60	350	100	100	50
	Ag	1	260	-60	350	45	30	10
		2	260	-60	350	100	100	50
B	Au	1	220	-65	310	30	35	10
		2	220	-65	310	100	100	50
	Ag	1	220	-65	310	35	40	10
		2	220	-65	310	100	100	50
Inca	Au	1	220	-55	310	40	35	10
		2	220	-55	310	100	100	50
	Ag	1	220	-55	310	25	35	10
		2	220	-55	310	100	100	50
Mercado	Au	1	245	-55	335	115	40	10
		2	245	-55	335	150	150	150
	Ag	1	245	-55	335	125	60	10
		2	245	-55	335	150	150	150
High Grade	Au	1	260	-60	350	30	30	5
		2	260	-60	350	150	150	10
	Ag	1	260	-60	350	30	30	5
		2	260	-60	350	150	150	10

Radii and ellipsoid orientations are defined based on the analysis of variograms and geological interpretation. Search ellipses used for the first estimation pass have a radius that corresponds to 100% of the variogram range in each of the anisotropic orientations.

AMEC verified variograms and the orientation of domain solids and agrees with the parameters used.

Additional parameters for gold and silver grade estimations are listed in Table 17-7 as well as the number and percentage of blocks estimated in each pass compared to the total number of blocks by domain. The number of blocks estimated was obtained from the analysis of the GEMS estimation reports.

Table 17-7
Gold and Silver Grade Estimation Parameters by Domain According to Intrepid Estimation Profiles

Zone	Element	Kriging Pass	Composites			Blocks Estimated	% of Total Blocks	Total Blocks
			Min.	Max.	Max. Per Hole			
Aztec	Au	1	3	12	2	10,541	87%	12,089
		2	1	12	2	1,548	13%	
	Ag	1	3	12	2	10,193	84%	12,089
		2	1	12	2	1,896	16%	
B	Au	1	3	12	2	8,946	67%	13,397
		2	1	12	2	4,451	33%	
	Ag	1	3	12	2	10,014	75%	13,397
		2	1	12	2	3,383	25%	
Inca	Au	1	3	12	2	4,672	65%	7,157
		2	1	12	2	2,485	35%	
	Ag	1	3	12	2	3,801	53%	7,157
		2	1	12	2	3,356	47%	
Mercado	Au	1	3	12	2	6,856	97%	7,103
		2	1	12	2	247	3%	
	Ag	1	3	12	2	6,856	97%	7,103
		2	1	12	2	247	3%	
High Grade	Au	1	2	12	2	3,280	71%	4,652
		2	1	12	2	1,,372	29%	
	Ag	1	2	12	2	0	0%	4,652
		2	1	12	2	4652	100%	

The number of blocks estimated for Mercado was obtained from the Intrepid estimation completed in August, 2006.

AMEC reviewed the parameters reported in "Casposo Project - Updated Mineral Resource Estimate" document from August, 2006 (McGuinty and Puritch, 2006) and compared them to the kriging profiles defined in GEMS software and found no inconsistencies. Intrepid did not report a second estimation pass for the High Grade domain as was found in the kriging profiles. AMEC´s evaluation of the reported resources in 2006 shows that Intrepid has not considered blocks that were estimated during the second pass for the High Grade domain. AMEC replicated the estimation profiles and obtained a different number of estimated blocks, as seen in Table 17-8.

Table 17-8
Estimation Parameters by Domain According to AMEC´s Replication

Zone	Element	Kriging Pass	Composites Min.	Composites Max.	Composites Max. Per Hole	Blocks Estimated	% of Total Blocks	Total blocks
Aztec	Au	1	3	12	2	10,492	87%	12,040
		2	1	12	2	1,548	13%	
	Ag	1	3	12	2	10,145	84%	12,040
		2	1	12	2	1,895	16%	
B	Au	1	3	12	2	8,942	67%	13,393
		2	1	12	2	4,451	33%	
	Ag	1	3	12	2	10,010	75%	13,393
		2	1	12	2	3,383	25%	
Inca	Au	1	3	12	2	4,649	69%	6,769
		2	1	12	2	2,120	31%	
	Ag	1	3	12	2	3,778	56%	6,769
		2	1	12	2	2,991	44%	
Mercado	Au	1	3	12	2	7,165	92%	7,791
		2	1	12	2	626	8%	
	Ag	1	3	12	2	7,165	92%	7,791
		2	1	12	2	626	8%	
High Grade	Au	1	3	12	2	3,280	71%	4,652
		2	1	12	2	1,372	29%	
	Ag	1	3	12	2	3,280	71%	4,652
		2	1	12	2	1,372	29%	

The resource estimation replicated by AMEC is very similar to that provided by Intrepid. Although some differences were found in the number of blocks reported at each estimation pass, AMEC accepts the estimation parameters used since they reflect grade distribution and continuity according to the existing drill hole spacing and geological interpretation of domains.

The different vein domains have no direct contact between them; being totally independent, with the exception of a small zone where vein B intersects Aztec. Composites from each domain were used separately to estimate blocks for the corresponding domain. In other words, hard boundaries were used for grade estimation of all domains.

Using composite gold grades, AMEC performed a contact analysis to confirm the type of boundary (hard, soft, firm) that would be most appropriate to use between domains. The analyses of the graphs confirms that domains are independent and that the use of hard boundaries is the most appropriate approach.

17.1.11 Block Model Validation

In order to verify the ID3 grade estimation, AMEC performed a visual inspection of the vertical sections and horizontal plans of the block model. In addition, a review of statistics of estimated blocks was completed and AMEC constructed Swath Plots on both vertical sections and horizontal plans.

Visual Inspection

The visual inspection of the overall vertical section shows a good correlation of composites and block grades; however AMEC found some problems in the grade shell interpretation where some composites were not considered. This problem is not prevalent throughout the model, and its occurrence is mainly localized.

Swath Plots

AMEC constructed swath plots for both vertical sections and horizontal plans for gold and silver values in order to identify possible local bias in the grade estimation. For this purpose, AMEC created a nearest neighbour (NN) estimation and stored the results as an attribute in the block model. The NN estimation used the same composites provided by Intrepid, but only used the search ellipse corresponding to the second estimation pass. The swath plot considers the volume of influence or corridor of each section and calculates weighted grades for this volume, plotting both nearest neighbour and ID3 estimated values for each section or level in the model.

Only blocks that were classified as Indicated and Inferred resources were considered. Swath plots were created for Au and Ag for each vein domain. Below is a summary of the swath plot findings:

- AMEC did not identify any significant bias for gold estimates for the Aztec vein, but a small overestimation of the ID3 estimate was noted between levels 2300 and 2255 m.
- AMEC considers that the estimation of gold for the B vein presents no local bias. There is a local overestimation of ID3 grades for Inca vein below level 2306 m. This corresponds with a decrease in the number of composites used in the estimation. Although several composites were used to estimate the upper portion of the high grade domain, there is an overestimation of ID3 grades around the 2507 m level.
- For Mercado, AMEC observed local overestimation of ID3 grades between levels 2477 and 2441, 2405 and 2387, and 2369 and 2351 m and between the vertical sections 6548800N and 6548875N. Overall, there does not appear to be any global bias within this domain.
- AMEC noted that silver grades are locally overestimated by ID3 between levels 2300 and 2255 m within the Aztec domain. AMEC notes the continuous increase of silver grades with depth until level 2255 where silver grades drop abruptly. For gold the behaviour is the opposite, with higher grades close to the surface. Between levels 2375 and 2339 m of the B vein domain, AMEC observes an overestimation of NN grades compared to ID3.

- Similar to gold grades in the Inca vein, the ID3 estimate for silver shows higher grades than NN at lower levels of the deposit. This corresponds to a decrease in the number of composites used in the estimation. Overall, there appears to be a global high bias of the ID3 silver estimates in the Inca domain when compared to the NN estimate.
- Although a local high bias of the ID3 silver grades is observed at the upper levels (2507 2519 m) of the HG domain, there is no observable global bias on silver grades estimated by ID3 and NN. There is a significant underestimation of ID3 silver grades below level 2351 m for the Mercado vein set when compared to NN estimate.
- Overall, AMEC considers the local bias observed in the swath plots a consequence of the last composite in each drill hole populating several blocks during the NN estimation. While a high local bias exists with the ID3 estimated grades, AMEC is of opinion that this will not significantly impact the overall resource estimation at the Casposo deposit.

17.1.12 Statistics

AMEC selected the estimated blocks for each vein domain and calculated the primary statistics. Results are summarized in Table 17-9.

Table 17-9
Block Model Statistics by Vein

Zone	Element	No. Samples	Min.	Max.	Mean	Standard Deviation	Coefficient of Variation
Aztec	Au (ppm)	12,040	0.051	34.108	2.001	2.576	1.287
	Ag (ppm)	12,040	1.040	676.788	68.662	83.493	1.216
B	Au (ppm)	13,393	0.020	29.224	1.928	2.657	1.378
	Ag (ppm)	13,393	0.259	296.350	28.030	32.967	1.176
Inca	Au (ppm)	6,769	0.020	14.977	1.688	1.935	1.146
	Ag (ppm)	6,769	2.004	488.101	81.464	77.741	0.954
Mercado	Au (ppm)	7,103	0.011	7.986	0.858	0.775	0.903
	Ag (ppm)	7,103	0.320	835.630	32.421	50.996	1.573
High Grade	Au (ppm)	4,652	0.351	120.000	14.595	12.846	0.880
	Ag (ppm)	4,652	4.112	1906.359	328.808	297.358	0.904

A comparison of grade averages for composites and blocks estimated by ID3 and NN is shown in Table 17-10.

Table 17-10
Comparison of Composites and Block Grade Averages

Zone	Element	Composites Mean	Block Model (NN) Mean	Block Model (ID3) Mean	ID3/NN Difference
Aztec	Au (ppm)	2.219	2.020	2.001	99.1%
	Ag (ppm)	53.505	65.812	68.662	104.3%
B	Au (ppm)	2.176	1.921	1.928	100.4%
	Ag (ppm)	28.515	28.787	28.030	97.4%
Inca	Au (ppm)	1.794	1.665	1.688	101.4%
	Ag (ppm)	76.338	71.741	81.464	113.6%
Mercado	Au (ppm)	0.932	0.784	0.858	109.4%
	Ag (ppm)	34.233	29.409	32.421	110.2%
High Grade	Au (ppm)	17.439	14.067	14.595	103.8%
	Ag (ppm)	336.584	332.900	328.808	98.8%

Composites are not declustered and only composites inside of each geological solid were used for calculating average values. The nearest neighbour model (NN) reflects the declustering of composites and this is the best comparison to the ID3 estimation.

Gold grades estimated at the blocks present an average overestimation of 2.9% compared to NN estimate and AMEC considers acceptable, however Mercado gold grades are overestimated at almost 10%. Silver grades are overestimated by 5% overall but higher deviations are for Inca vein and Mercado, confirming the local bias observed in the swath plots. The differences are within the acceptable margin of 5% globally, however AMEC recommends Intrepid review search ellipses for grade estimation of Mercado, since high grades are being over-projected through the model.

Although local biases of minor expression can be found for gold estimation, AMEC considers the grade estimation acceptable.

17.1.13 Mineral Resource Classification

Intrepid defined the resource classification based on grade estimation passes. Blocks estimated for gold during the first estimation run are classified as Indicated resources and remaining blocks estimated during the second run are classified as Inferred. The search ellipse radii used at the first estimation pass represents 100% of the variogram ranges and AMEC is of opinion that this represents grade and geological continuity of the mineralization at Casposo.

After this initial classification, Intrepid examined block model sections and compared them to composites, and upgraded some blocks from the Inferred to the Indicated category. AMEC verified the upgraded blocks. They are mainly in a direction parallel to drill holes and within 5 to 10 m away from other Indicated blocks. AMEC accepts the classification approach used. In a few sections grade estimation for some blocks is supported by only one drill hole but the drill hole is one in which more than five mineralized intervals are available to inform the block.

AMEC verified the Intrepid resource estimate and found a negative difference of 30,000 t in the Inferred resources compared to the figures determined by Intrepid, but considers this difference not important since Inferred resources are not converted to reserves. Therefore AMEC accepts the mineral resources as defined by Intrepid. AMEC performed estimations using a fixed specific gravity value of 2.5 t/m^3 to determine tonnage for all blocks in the model.

The gold equivalent cut-off was determined by Intrepid according to the parameters below:

- Au/Ag ratio 77:1
- Au Price US$450/oz
- Grams/troy oz 31.1035
- Process Cost US$15.70/t
- G&A Cost US$3.35/t
- Au Process Recovery 94%
- Ag Process Recovery 80%

Therefore the equation to determine the cut-off is as follows:

(US$15.70 + US$3.35/t)/[(US$450/oz/31.1035) x (94%)] = 1.40 g/t Au

This cut-off grade was calculated using the above preliminary unit cost estimates prior to the completion of the 2007 Feasibility Study. To confirm that this value was still current a check was done using the Feasibility Study costs and methodology. The strategy of stockpiling low grade mineralization (below full cut-off) within the pit limits and processing it at the end of the mine life was adopted in the Feasibility Study. At the end of the mine life, mining activities will cease and the mine will operate with minimal operating and supervisory staff and consequently lower unit costs can be used: processing costs are based on 75% of full cost (US$19.43 x 0.75 = US$14.57/t) and G&A costs at 65% of full cost (US$6.39 x 0.65 = US$4.15/t). In addition, the cost of reclaim from the stockpile will be US$0.44/t.

Therefore the check on cut-off grade is:

(US$0.44 + US$14.57 + US$4.15/t) / [(US$450/oz/31.1035) x (94%)] = 1.41 g/t Au

This check is very close to the cut-off grade used in the resource estimate.

This cut-off value does not include the cost of mining, as this value is an incremental cut-off that would be applied to the loaded haulage trucks as they exit from the open pit. Trucks filled with mineralized rock above this cut-off would be sent to the process plant or end-of mine stockpile while trucks loaded with rock below that grade would be sent to the waste dump. This methodology is consistent with pit design using the Whittle optimizer which develops an economic pit shell considering all cost and price factors, then applies this incremental cut-off to determine which mineralization within that pit shell is to be included in the resource.

Based on the foregoing considerations, AMEC considers 1.40 g/t Au an acceptable base case gold equivalent cut-off for reporting the mineral resources.

Resources within the Kamila and Mercado zones have an effective date of October 10, 2006, and are summarised in Table 17-11, at a 1.4 g/t Au equivalent grade.

Table 17-11
Casposo Mineral Resources
(Effective Date 10 October 2006)

Resource Category	Deposit/Zone	Tonnes (Mt)	Au g/t	Ag g/t	Au contained oz	Ag contained oz	Au Eq contained oz
Indicated	Kamila	1.960	4.76	120.0	300,130	7,563,468	398,357
	Mercado	0.225	1.82	81.9	13,148	591,898	20,835
	Total Indicated	**2.185**	**4.46**	**116.1**	**313,278**	**8,155,366**	**419,192**
Inferred	Kamila	0.131	2.14	86.1	9,000	362,393	13,706
	Mercado	0.007	1.66	26.0	354	5,957	431
	Total Inferred	**0.138**	**2.12**	**83.3**	**9.354**	**368,350**	**14,137**

17.2 Mineral Reserves

A hybrid mining case was developed for the Casposo Project, consisting of mining the upper part of the deposit by open pit and the lower part by underground methods. The concept behind this option is that the shallower ore would be mined by low strip ratio open pit mining (i.e. low cost open pit mining) and the deeper and narrower ore that would require high strip ratios to mine by open pit would be mined by underground methods. The open pit and underground mineral reserves were estimated by Gary Taylor, P. Eng. from AMEC, in 2007.

The Reserves were estimated separately for the open pits and Kamila underground.

17.2.1 Open Pit Reserves

The Reserves for the open pit were calculated separately for the Kamila Main, Kamila SE and Mercado open pits.

Two types of Open Pit Reserves have been estimated:

- Mining Reserve – mineralized rock which is above a cut-off grade calculated from the full operating costs during mining.
- Stockpile Reserve – this is mineralized rock which exceeds the stockpile cut-off grade. This rock must be mined during mining operations to access the ore but does not exceed the full mining operations cut-off grade. A reduced cut-off grade has been calculated for this material on the basis that it will be hauled to a separate low grade stockpile and only processed through the mill in the last three months of operation, once the mining reserve has been depleted. Costs will be lower than during full mining operations with only stockpile recovery costs and reduced processing and G&A costs.

17.2.2 Open Pit Cut-off Grade

The open pit mining cut-off grade was calculated based on the parameters listed in Table 17-12.

Table 17-12
Open Pit Mining Cut-off Estimation Parameters

Operating Costs	US$/t	Comments
Mining	0.44	Additional Cost of Mining Ore than waste (Ore US$1.98/t, Waste US$1.54/t)
Processing	18.93	Phase 1 operating cost estimate
G&A	6.97	Phase 1 operating cost estimate
Total	26.34	
Metallurgical Recoveries		
Au	93.6%	Phase 1 estimate
Ag	81.2%	Phase 1 estimate
Metal Prices		
Au	450/oz	Selling Cost - US$8.96/oz, Net - US$441, (US$441/31.103)= US$14.18/g
Ag	7.00/oz	Selling Cost - US$0.54/oz, Net - US$6.46
AuEq Ratio	80	This is factor used to convert Ag value to an equivalent Au value

Note: Phase 1 estimates were preliminary estimates derived prior to the completion of the feasibility study.

Therefore:

- AuEq cut-off grade calculation = (26.34/14.18)/0.936 = 1.98 g/t AuEq.

The open pit stockpile cut-off grade was calculated based on the parameters listed in Table 17-13.

Table 17-13
Open Pit Stockpile Cut-off Estimation Parameters

Operating Costs	US$/t	Comments
Mining	0.44	Load from stockpile, haul to crusher
Processing	15.14	80% of US$18.93 due to reduced manpower and long term maintenance
G&A	4.88	70% of US$6.97 due to reduced manpower and long term maintenance
Total	20.46	
Metallurgical Recoveries		
Au	85%	Reduced from 93.6% due to lower grade
Ag	70%	Reduced from 81.2% due to lower grade
Metal Prices		
Au	450/oz	Selling Cost - US$8.96/oz, Net - US$441, (US$441/31.103)= US$14.18/g
Ag	7.00/oz	Selling Cost - US$0.54/oz, Net - US$6.46
AuEq Ratio	80	This is factor used to convert Ag value to an equivalent Au value

Therefore the equation to determine the cut-off is as follows:

- AuEq cut-off grade calculation = (20.46/14.18)/0.85 = 1.70 g/t AuEq.

17.2.3 Open Pit Mining Dilution

The open pit mining dilution was estimated by first expanding the mining solids a distance estimated by AMEC to be appropriate to represent the mining conditions for this deposit and then Gemcom was used to calculate the tonnes and grade of the ore within the expanded solid. This result yielded a diluted mining tonnage and grade and the actual dilution was then back-calculated by subtracting the unexpanded resources.

Two types of dilution were considered based on the boundary of the above cut-off grade shell. One type was when the above cut-off material was limited by the vein boundary or lithology. Beyond this boundary expected Au and Ag grades are quite low.

However, 40% to 50% of the time the mining limit is simply a cut-off grade boundary which will be more difficult to identify both during blast planning and during digging by the excavator. This dilution however is often only marginally below the cut-off grade and accordingly will carry significant grade with it. Both dilution types were addressed in the details of the reserve estimation, which are as follows:

- The lithology solids for Aztec, Inca, B and Mercado veins were expanded an additional 0.5 m resulting in half a meter 'skin' being added to the original lithological solids. The 0.5 m expansion on both the hanging wall and footwall of the zone was applied to reflect dilution at the lithology contact. Only the 0.5 m dilution was selected based on digging with a 2.3 m^3 excavator along the contact between the mineralized zone and barren material which can be visually established enhancing the miners' ability to dig selectively.

- A new percent block model (%new), containing the volumes updated from the expanded lithology solids was created. The original resource model (Gemcom) was supplied in the percent format meaning that if the particular block, of the block model, is not totally included in the mineralized zone then the percent of the portion belonging to the mineralized zone is known. The tonnage of the block was then calculated by multiplying the volume of the 'whole' block by the percent by the density. After adding the 0.5 m 'skin' to the lithological contact the percentages of the new, diluted model were recalculated by updating from the expanded solids, resulting in new percent model. The block grades were calculated in a similar way using constant dilution grade for the portion of the block which represents the difference in the block volume.
- The grade models, for ore (Grade Ore) and two stockpiles (Grade Pile High) and (Grade Pile Low), were created from the block model based on the cut-offs of >1.38 g/t AuEq, 1.70 g/t AuEq and >1.96 g/t AuEq respectively. These grade solids were expanded 1.0 m in every direction.
- Where the change in grade is gradual, and within the mineralized zone, the mining selectivity is expected to be considerably more difficult then at the lithology contact. For this reason it was decided to expand the grade solids 1.0 m, which is approximately half of the bucket width for the selected excavator.
- The new, diluted grades for the 'reserve block model' were calculated (New Au grade, New Ag grade) using the following formulas:
  - For gold: a grade of 0.15 g/t was applied for dilution outside of the zone (i.e. outside the lithology contact), as the block model did not extend outside the lithology contact. This grade was assigned based on a review of the drill hole data.
  - New Au grade = ((original grade*original %) + (0.15*(new%-original %)))/new%
  - For silver: a grade of 4.0 g/t was applied for dilution outside of the zone (i.e. outside the lithology contact), as the block model did not extend outside the lithology contact. This grade was assigned based on a review of the drill hole data.
  - New Ag grade = ((original grade*original %) + (4.0*(new%-original %)))/new%

These calculations took place only in case the original % of the block was in contact with waste (i.e. the block had <100% of ore).

  - For gold equivalent, the formula was applied after the individual grades for gold and silver grades were computed.
  - New AuEq=New Au+NewAg/80.

The final report of diluted tonnes and grades was estimated using the 'new percent' block model, new diluted grades and in the limit of the expanded grade solids 'Grade Ore', 'Grade Pile High' and 'Grade Pile Low'.

The dilution estimation procedure used followed to calculate the tonnes and grade of the diluted (expanded) mining shapes. These results were input into a spreadsheet and compared to the corresponding resources to determine the quantity and grade of the dilution that this procedure added to the resources.

Dilution was calculated as: dilution = waste (t) / resource (t). The results of this calculation are shown in Table 17-14. Overall open pit dilution was estimated at 18% with a grade of 0.69 g/t Au, 11 g/t Ag, and 0.82 g/t Au equivalent. Based on AMEC's experience with other similar projects this is a reasonable estimate and should be achievable with the application of good geology, engineering and mining practices with constant emphasis on grade control.

Table 17-14
Mining Dilution

Zone	Tonnage[1] (t x 1000)	Au (g/t)	Ag (g/t)	AuEq (g/t)	Dilution (%)
Kamila Main Pit	178	0.72	10.8	0.85	18.1
Kamila SE Pit	11	0.31	6.4	0.39	14.3
Mercado Pit	14	0.62	16.4	0.83	19.8
Total	**203**	**0.69**	**11.0**	**0.82**	**18.0**

Note: (1) tonnage before 95% recovery factor was applied.

Dilution percentages were not calculated for the stockpiles because of the difficulty of comparing stockpile resources to the diluted stockpile shapes. The problem is that the 1.0 m of ore dilution into the mineralized zones incorporates some of the stockpile resource. When the diluted stockpile results are compared to the theoretical stockpile resource the numbers are no longer comparable.

17.2.4 Open Pit Mining Recovery

For open pit a mining recovery of 95% was applied to all resources contained within the pit shell. This loss of 5% was incorporated into the Reserve calculation to provide for mining losses of resource due to a number of causes including:

- Mine planning due to irregular ore outlines that would result in some ore being excluded from the ore blast design.
- Digging losses by the excavator in the muck pile
- Spillage losses from trucks and loaders.
- Recovery losses during reclaim from stockpiles.

- Operator error whereby an ore load may be dumped on the waste dump.

This allowance is a factor based on AMEC experience with similar mining situations and industry-wide accepted practice. The 95% recovery factor was applied to the diluted resource as output from the Gemcom query.

17.2.5 Open Pit Reserves Statement

Open Pit Reserves are summarized in Table 17-15.

Table 17-15
Summary Open Pit Mineral Reserves, Inpit Reserves at an AuEq Cut-off of 1.98 g/t, and Stockpile Reserves at an AuEq Cut-off of 1.70 g/t Au
(G. Taylor P. Eng., QP, Effective Date 30 March, 2007)

Zone	Probable Reserve						
	Tonnage	Au	Ag	Au	Contained Metal		
	(t)	Grade (g/t)	Grade (g/t)	Grade (g/t)	Au (oz)	Ag (oz)	AuEq (oz)
Ore							
Kamila Main Pit	1,103,513	5.66	94.7	6.85	200,958	3,359,937	242,957
Kamila SE Pit	84,396	4.96	65.7	5.79	13,469	178,386	15,699
Mercado Pit	80,270	2.19	93.5	3.36	5,647	241,173	8,661
Total	**1,268,179**	**5.40**	**92.7**	**6.56**	**220,074**	**3,779,496**	**267,317**
Stockpile							
Kamila Main Pit	74,375	1.25	24.4	1.55	2,983	58,403	3,713
Kamila SE Pit	1,082	1.27	24.5	1.58	44	854	55
Mercado Pit	8,858	1.23	29.2	1.60	350	8,320	454
Total	**84,315**	**1.25**	**24.9**	**1.56**	**3,377**	**67,577**	**4,222**
Ore + Stockpile							
Kamila Main Pit	1,177,888	5.39	90.3	6.51	203,941	3,418,339	246,670
Kamila SE Pit	85,478	4.92	65.2	5.73	13,513	179,240	15,754
Mercado Pit	89,128	2.09	87.1	3.18	5,997	249,493	9,115
Combined Total	**1,352,494**	**5.14**	**88.5**	**6.24**	**223,451**	**3,847,073**	**271,539**

17.2.6 Open Pit Additional Potential

There is potential to add to the reserve base within the current planned open pit. AMEC reviewed the pit to determine if there was potential for upside within the designed pit with the use of higher gold and silver prices. If the gold and silver price parameters for the open pit are increased from the base case US$450 to US$550/oz for gold and base case US$7 to US$9/oz for silver, which equates to a grade cut-off of about 1.4 g/t AuEq, there is potential to add about 80,000 t of material at an average grade of about 6.7 g/t Au Eq in the Kamila Open Pit that can report to stockpiles for processing at the end of the mine life.

17.3 Underground Reserves

Underground Reserves have been estimated for the Kamila deposit only and have been separated into two groups; "Below Pit", which are located below the main Kamila pit and "Remnants", which are outside the final pit walls. The Remnants will be recovered only once the main Kamila pit has been completely mined out and access will be provided from inside the pit.

The following steps were used to calculate the underground Reserves:

- The mining method and stope design parameters were selected. A cut-off grade of 3.0 g/t AuEq was used to outline longhole stope boundaries.
- Stope outlines (shapes) were created on plan for each 6 m bench by manually outlining the resource above 3.0 g/t AuEq to create outlines that would be both mineable and practical for operations. Creation of these outlines involved judgement including provision for mining some below cut-off grade material within the outlines to create continuity within the stopes and isolating some above cut-off grade material that would not be economic to mine.
- The stope outlines were then examined in three dimensions (on longitudinal section and cross section) to evaluate the continuity of the stopes vertically and how they related to the planned 20 m vertical sublevel spacing. Further manual adjustments to the stope shapes were then made for practical mining considerations. Longitudinal sections of the Aztec and Inca veins were created to assist with this process.
- Dilution and recovery were estimated and applied to the in-stope resources.

17.3.1 Underground Cut-off Grades

The underground mining cut-off grade was calculated based on the parameters shown in Table 17-16.

Table 17-16
Underground Mine Cut-off Grade Parameters

Operating Costs	US/t	Comments
Mining	21.03	Phase 1 operating cost estimate
Processing	18.93	Phase 1 operating cost estimate
G&A	6.97	Phase 1 operating cost estimate
Total	46.93	
Metallurgical Recoveries		
Au	94.5%	Phase 1 estimate (higher than open pit due to higher grade)
Ag	83.6%	Phase 1 estimate (higher than open pit due to higher grade)
Metal Prices		
Au	450/oz	Selling Cost - US$8.96/oz, Net - US$441, (US$441/31.103)= US$14.18/g
Ag	7.00/oz	Selling Cost - US$0.54/oz, Net - US$6.46
AuEq Ratio	80	This factor is used to convert Ag value to an equivalent Au value

Note: Phase 1 estimates were preliminary estimates derived prior to the completion of the feasibility study.

Therefore the equation to determine the cut-off is as follows:

- AuEq Cut-off grade calculation = (46.93/14.18)/0.945 = 3.50 g/t AuEq

This is the full underground cut-off grade which will support all underground mining activities including access development. For outlining stope boundaries a lower incremental cut-off grade of 3.00 g/t AuEq was used to reflect the fact that once the access development is in place then incrementally the ore on the periphery of a stope need only support the mining costs less those for access development (US$14.00/t). The incremental AuEq cut-off grade calculation is as follows:

- Incremental AuEq Cut-off grade calculation = (39.90/14.18)/0.945 = 3.00 g/t AuEq.

In summary, the lower cut-off grade of 3.00 g/t AuEq was used to outline the stopes. However, once the overall grade of a particular stope was estimated, and if it did not exceed the full cut-off grade of 3.50 g/t AuEq, then that stope was excluded from the mine plan. During the mine design process, when a stope was estimated to be below the full cut-off grade, then the boundaries of that stope were reviewed and adjusted if possible to eliminate some lower grade material from the stope and thereby raise the overall grade of the stope.

17.3.2 Underground Dilution

The procedure used to calculate underground mining dilution is similar to that used to calculate the open pit mining dilution except that the amount of dilution on the grade contact was reduced to reflect more selective mining procedures underground. There was no change applied to the dilution applied at the lithology contact.

For the underground reserve calculation the lithology and grade solids were each expanded 0.5 m to calculate mining dilution which resulted in 0.5 m dilution being applied on both the hanging wall and footwall of the stopes. This is different to the open pit calculation, where the lithology solid was expanded 0.5 m but the grade solid was expanded 1.0 m. The dilution of the grade solid was reduced to 0.5 m for underground for two reasons. First, selective blasting and drilling practices will be used and second, there will be high quality geological information on which to base blasthole layouts derived from detailed mapping and back and face sampling of the undercut and overcut sublevel development. The calculation procedure was as follows:

The lithology solids for Aztec, Inca, B and Mercado veins were expanded 0.5 m resulting in half a metre 'skin' added to the original lithological solids. The grades for this dilution were applied at Au 0.15 g/t and Ag 4.0 g/t.

The dilution at the grade boundary was applied in an identical way. The main difference is that instead of grade model being expanded, as was the case for the open pit, the individual stope boundaries were expanded 0.5 m to obtain the actual dilution grade of dilution within the limits of the vein. Expanding the stope solids resulted in an overlap between the solids. The solids precedence was established such that the overlying stope has priority over the underlying stope so the common volume gets assigned to the stope above. This way the common volumes are not double counted in the reserve calculations.

Dilution was calculated as: Dilution = waste (t) / resource (t). Results are summarized in Table 17-17.

Table 17-17
Underground Dilution Summary

Zone	Tonnage[1] (t x 1000)	Au (g/t)	Ag (g/t)	AuEq (g/t)	Dilution (%)
Below Pit	49.7	0.41	20	0.66	13.2
Remnants	5.2	0.66	15	0.85	15.1
Total	**54.9**	**0.43**	**20**	**0.68**	**13.3**

Note (1): Before 95% recovery factor was applied.

The overall underground dilution was estimated at 13% with a grade of 0.43 g/t Au, 20 g/t Ag, or 0.68 g/t AuEq. Based on AMEC´s experience with other similar projects this is a reasonable estimate and should be achievable with the application of good geology, engineering and mining practices with constant emphasis on grade control.

17.3.3 Underground Mining Recovery

For underground mining a recovery of 95% was applied to all resources contained within the main stopes below the open pit. A lower recovery of 85% was applied to the mining of the Remnants. These mining losses were incorporated into the Reserve calculation to provide for mining losses of resources due to a number of causes including:

- Mine planning due to irregular ore outlines that would result in some ore being excluded from the ore blast design.
- Mucking losses by the scooptram in the stope. Since much of the stope mucking will be by radio remote operation the operators ability to completely clean the stope floor will be reduced.
- Ore hung up on the stope footwall in low dipping areas and abandoned.
- Blast misfires resulting in ore not being blasted.
- Spillage losses from trucks and loaders.
- Recovery losses during reclaim from stockpiles.

- Operator error whereby an ore load may be dumped on the waste dump.

The mining recovery was reduced a further 10% to 85% for the remnants for the following reasons:

- Access will be out of the pit and local pit wall conditions may restrict access to some parts of the remnants.
- Mining of the remnants will affect the stability of the pit walls. During a detailed geotechnical evaluation of the mining of these remnants during mine operations some support pillars in ore may be left to improve the pit wall stability.

These allowances are factors based on AMEC experience with similar mining situations and industry wide accepted practice. The 95% and 85% recovery factors were applied to the diluted resource as output from the Gemcom query.

17.3.4 Underground Reserves Statement

All the underground Reserves are in the Probable category. A summary of the Underground Reserves, reported to a cut-off grade of 3.5 g/t AuEq is shown in Table 17-18.

Table 17-18
Casposo Project Underground Mineral Reserves Summary Reported to a Cut-off of 3.5 g/t AuEq
(G. Taylor P.Eng., QP, Effective Date, 30 March 2007)

Zone	Reserve						
	Tonnage	Au	Ag	AuEQ	Contained Metal		
	(t)	Grade (g/t)	Grade (g/t)	Grade (g/t)	Au (oz)	Ag (oz)	AuEq (oz)
Below Pit							
2380 Sublevel	103,649	3.51	167.1	5.59	11,680	556,685	18,639
2370 Sublevel	25,210	5.56	386.1	10.38	4,503	312,957	8,415
2360 Sublevel	83,421	3.55	161.8	5.57	9,515	434,010	14,941
2350 Sublevel	33,019	3.12	294.5	6.80	3,308	312,636	7,216
2340 Sublevel	38,769	4.41	179.1	6.65	5,493	223,256	8,284
2320 Sublevel	27,758	2.23	157.4	4.19	1,986	140,468	3,742
2315 Sublevel	30,359	2.98	228.9	5.84	2,905	223,438	5,698
2300 Sublevel	25,102	1.54	178.2	3.77	1,245	143,844	3,043
2280 Sublevel	20,215	1.15	213.0	3.81	748	138,449	2,479
2258 Sublevel	17,650	0.91	171.8	3.06	519	97,466	1,737
Subtotal	**405,151**	**3.22**	**198.3**	**5.70**	**41,903**	**2,583,211**	**74,193**
Remnants							
2444 Remnant A	8,487	7.52	22.0	7.79	2,051	5,998	2,126
2426 Remnant D	3,196	4.13	56.9	4.84	425	5,851	498
2402 CP Remnant N	10,917	2.63	175.0	4.81	921	61,409	1,689
2402 CP Remnant S	11,040	3.77	135.7	5.47	1,338	48,162	1,940
Subtotal	**33,641**	**4.38**	**112.3**	**5.78**	**4,736**	**121,420**	**6,253**
Total	**438,792**	**3.31**	**191.7**	**5.70**	**46,638**	**2,704,631**	**80,446**

Note: * Included in Reserves based on incremental cut-off grade o f3.0 g/t AuEq.

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

18.1 Mining Operations

18.1.1 Open Pit Mining

The open pit mining plan comprises the mining of two separate orebodies - Kamila and Mercado - separated by a distance of approximately one km. The Kamila deposit will be mined by one large pit (Kamila Main Pit) and a smaller pit located 100 m to the south-east of the main pit (Kamila SE Pit). The Mercado deposit will be mined as a separate pit (Mercado pit). Figure 18-1 is a three dimensional representation of the deposits and open pits. Figure 18-2 is a general plan view showing all three pits.

Figure 18-1
Mineralized Zones and Planned Open Pit Representations

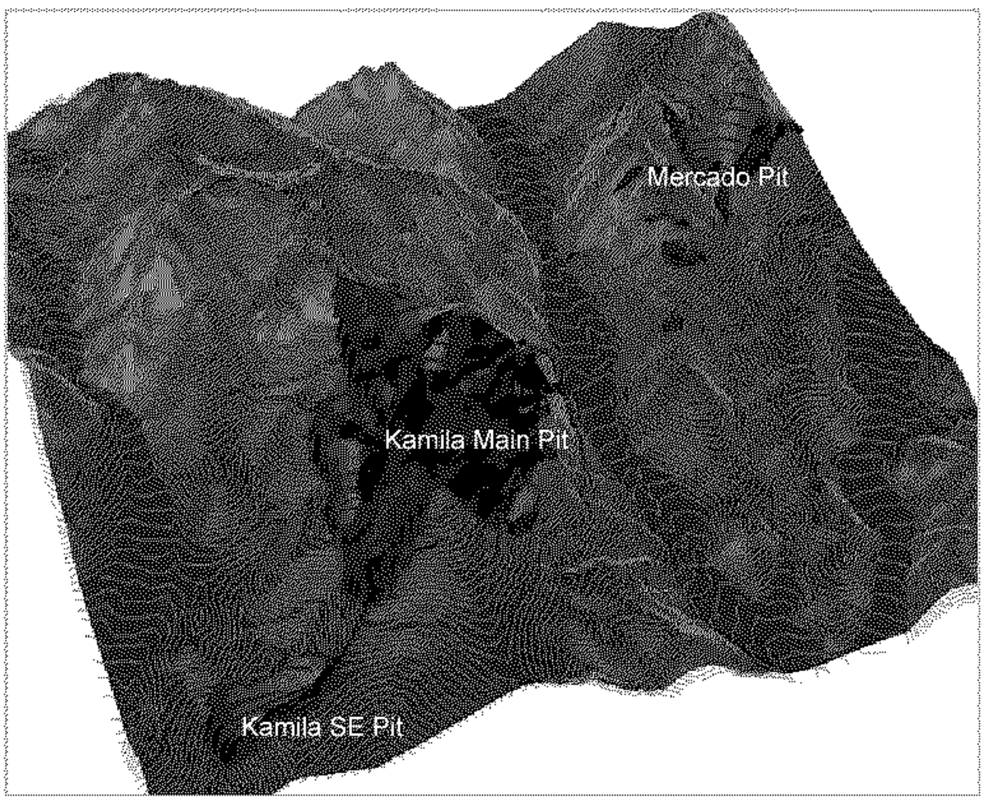
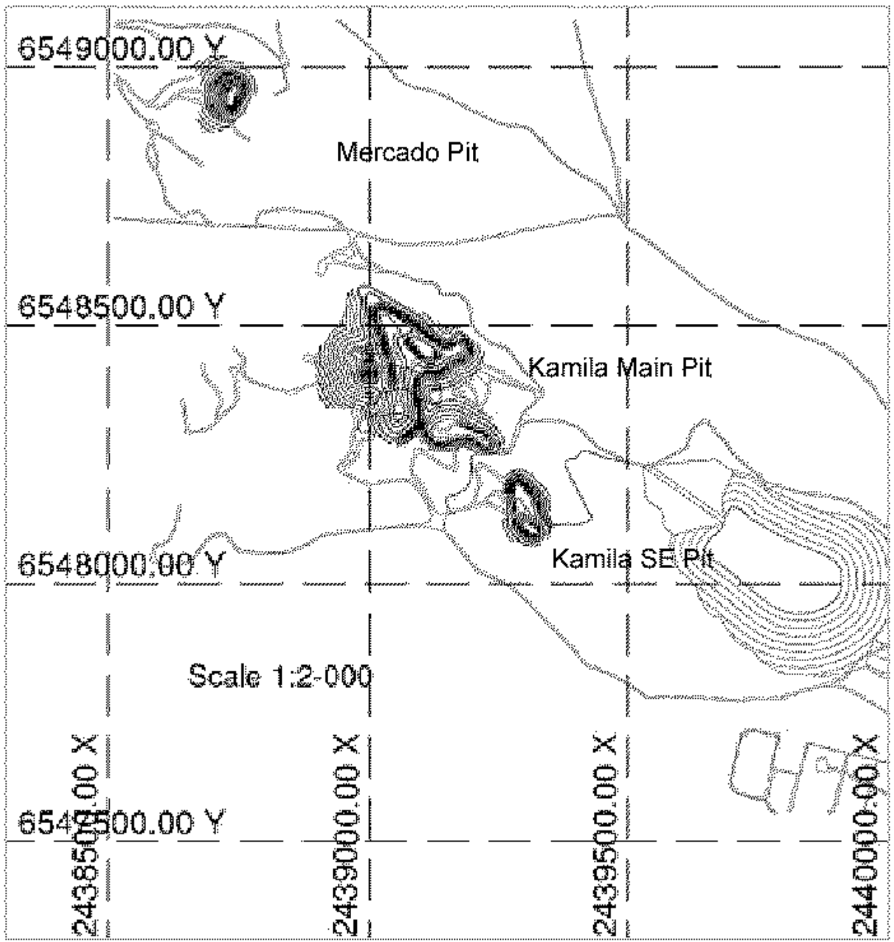


Figure 18-2
General View of Planned Open Pits showing Roads



Near surface, the Kamila deposit is located within a hill with steep slopes on all sides. The top of the hill within the pit perimeter is at 2,550 m elevation, while the base of the hill is at approximately 2,450 m elevation. The final pit bottom is at 2,402 m elevation, so essentially 100 vertical metres of the hill will be removed by mining before the pit extends another 50 m to the final pit bottom.

The mining strategy will be to start mining the main Kamila open pit, with the mining of the underground portions of Kamila and the Mercado open pit to follow later in the mine life. Prior to the start of mining a haul road from the plant to the Kamila pit and waste dump will be constructed and the initial pioneering access road to the top elevation of pit will be completed.

Open Pit Design

Detailed pit design was completed using Whittle software and was based on the optimum open pit shell. The detail pit design parameters are as follows:

- Bench face angle (BFA) of 85° degrees
- Berm width of 7m
- Bench height of 6 m
- Double benching for final waste walls
- Overall pit slope angle 54°
- Ramps
  - Main Kamila ramp – 10 m in width at 10% grade
  - The internal ramps accessing the two pit bottoms in the Main Kamila pit have been set at 5 m in width and either 10% or 13% grade. The 13% grade was used to obtain the access to the bottom-most benches.
  - The internal ramps to access Mercado and the 'stand alone' Kamila SE pit were set at 5 m in width and either 10% or 13% grade depending on the design

Pioneering Road

The start of mining operations at the main Kamila pit will require that first an access or pioneering road be constructed to the top of the hill encompassing the deposit. This road will be 10 m wide at maximum grade of 10% and will become the main haulage road for much of the tonnage in the main Kamila pit. The construction of this road will be a significant task due to the steepness of the hill and the fact that major drill and blast side cuts will be required.

Operations

Loading and hauling of ore and waste in the open pits will be done using backhoe style excavators with 2.3 m^3 buckets and 26–36 t fixed body conventional highway type haul trucks. All ore and waste rock will require drilling and blasting, loose overburden over the pit area is minimal. For the first two years the full production rate of 365,000 t of ore per year (1,000 t/d) will come from the Kamila pits but open pit production will drop to 232,380 t of ore per year (650 t/d) when underground production begins in Year 3. In addition to the 1,000 t/d of ore (above full cost cut-off grade), lower grade ore will be mined with the same methods as the ore but will be dumped on a low grade stockpile located near the plant and then processed at the end of the mine life. Mining of low grade stockpile ore will vary up to 100 t/d.

Waste quantities will peak at 3.7 Mt/a in the first year and decrease in future years. The waste stockpile will hold 8.9 Mt and will be located to the south east of the Main Kamila pit. The average haulage distance from the open pit to the waste stockpile will be 1.2 km. The stockpile pad will have final dimensions of 500 m by 400 m, with a height of 100 m.

A specialist mining contractor will be retained to perform all the mining activities. The contract will be all inclusive with the contractor supplying all the required manpower, mining equipment, support equipment and infrastructure facilities to meet the scheduled mining rates.

The open pit operating schedule will comprise two 12-hour shifts per day, seven days per week, for fifty working weeks per annum. The work schedule will have employees working four days on, four days off. Workers will be transported from Calingasta to the site at the start and end of each shift. All equipment will be diesel operated.

Both ore and waste will be drilled with a diesel hydraulic top hammer self-contained rig. Holes will be 127 mm diameter for both ore and waste with a hole pattern of 4 m x 4 m for ore and 5 m x 6 m for waste.

Ore mining benches will be 6 m in height, with blast holes drilled to 6.6 m in depth (0.6 m of sub-grade). Waste rock benches will be also be 6 m in height, although where practical double benches of 12 m will be used in waste with blast holes drilled 13 m in depth. The pit design was based on a 7 m wide berm being provided every 12 m vertically, so double benching can be used in waste for the final pit walls.

Total drilling requirements will be approximately 10,000 m/a for ore and 50,000 m/a for waste (at maximum of 3.7 Mt of waste stripping). One drill rig will be capable of handling all the drilling requirements for both ore and waste with a utilization factor of 35%. A second smaller more mobile drill is provided for initial pioneering requirements and as a backup unit for the primary drill.

Drill holes will be loaded with ANFO, which will be mixed at the blasting site. The site and rock is expected to be mostly dry, however if wet holes are encountered, the holes will be pumped dry, using a submersible pump, and ANFO loaded into a plastic sleeve and inserted into the hole. The powder factor has been calculated to be 0.27 kg/t in ore and 0.25 kg/t in waste.

Ore and waste will be loaded into trucks by a diesel operated excavator equipped with a 2.3 m^3 bucket. Two excavators will be required to meet the requirements. A front-end loader with a 2.3 m^3 bucket will also be employed, as required, for road and bench clean-up and occasional truck loading of waste.

The haulage trucks will be 36 t fixed-body conventional highway-type vehicles. A 36 t truck is preferred; however the mining contractor may choose the smaller size due to the fact that the 28 t model is a very common throughout Argentina. All haulage roads and ramps have been design with a maximum gradient of 10% and a width of 10 m which is suitable for these type of highway trucks.

Typical ore haulage distance to the process plant from the main Kamila pit will be 1.5 km and to the waste dump will be 1.2 km. A unique feature of the main Kamila pit is that approximately 65% of the ore and waste to be hauled within the pit will be transported "down grade" due to the fact that much of the deposit is located within a hill.

Support mining equipment will include a track dozer for ripping and levelling the waste dump. The open pit haul roads will be maintained with a grader and a water truck to control dust. Fuelling of equipment will utilize a fuel/lube/service truck, which will travel to equipment requiring fuelling anywhere in the open pit. Service equipment will include a service truck, utility loader, mobile crane and boom truck crane.

Small pickup trucks (similar to the Toyota Hi-Lux) will be used by engineering, geology, and management personnel, for traveling on the site, between the site, and other facilities.

The open pit mining equipment is summarized in Table 18-1.

Table 18-1
Open Pit Mining Equipment

Description	Typical Model	Year (Number Units) -1	1	2	3	4	5
Ore and Waste Schedule – Open Pit							
Ore (Kt)		34	389	373	194	191	172
Waste (Kt)		922	3,645	2,511	1,180	719	842
Total (Kt)		**956**	**4,034**	**2,884**	**1,374**	**910**	**1,014**
Production Equipment							
Blasthole Drill – Diesel Hydraulic Top Hammer	Atlas Copco Panterra 1500	1	1	1	1	1	1
Air track drill (for pioneering)	-	1	1	1	1	1	1
Haulage Trucks -36 t highway type	Mercedes Actros 4143	4	6	4	2	2	2
Excavator – 2.3 m^3	CAT 330DL	2	2	2	1	1	1
Front End loader	CAT 966H	1	1	1	1	1	1
Bulldozer	CAT D8T	1	1	1	1	1	1
Grader	CAT 14H	1	1	1	1	1	1
Support Equipment							
Water truck- 20,000 l	Mercedes Actros 4143	1	1	1	1	1	1
Fuel/Lube Truck	Scania 26 t	1	1	1	1	1	1
Mechanics Truck	-	1	1	1	1	1	1
Blasthole Loader truck	-	1	1	1	1	1	1
Truck flatbed 10 t c/w 25 t Boom Crane	-	1	1	1	1	1	1
Crew Van	-	2	2	2	2	2	2
Pick-up Truck	-	4	4	4	4	4	4
Lighting Plant	-	2	2	2	2	2	2

18.1.2 Underground Mining

The open pit will extend down to the 2,402 elevation, below which the ore will be extracted by underground methods.

A composite plan view of the underground mine is shown in Figure 18-3 and a 3 D representation in Figure 18-4.

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Figure 18-3
Composite Mine Plan – Underground

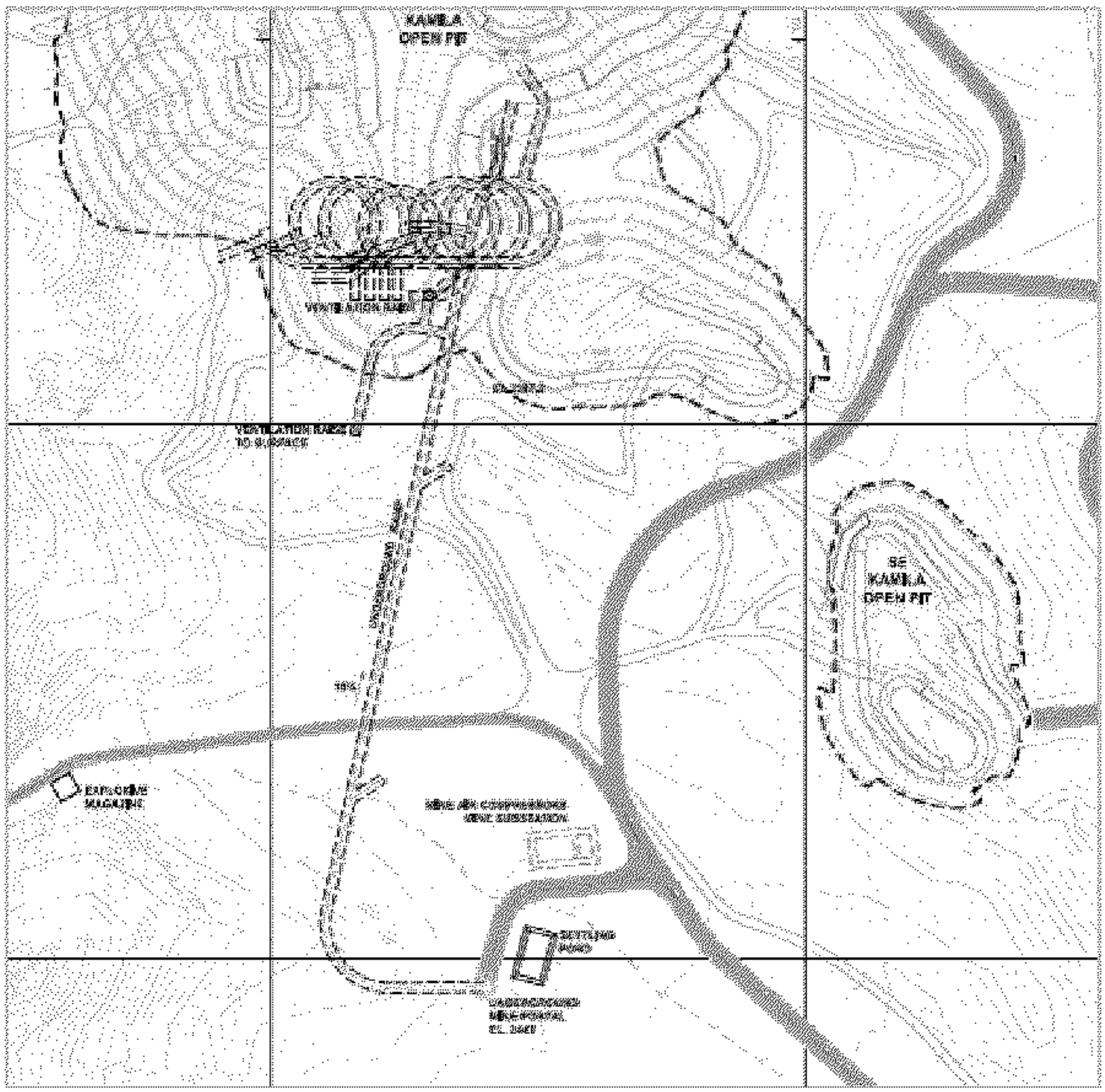
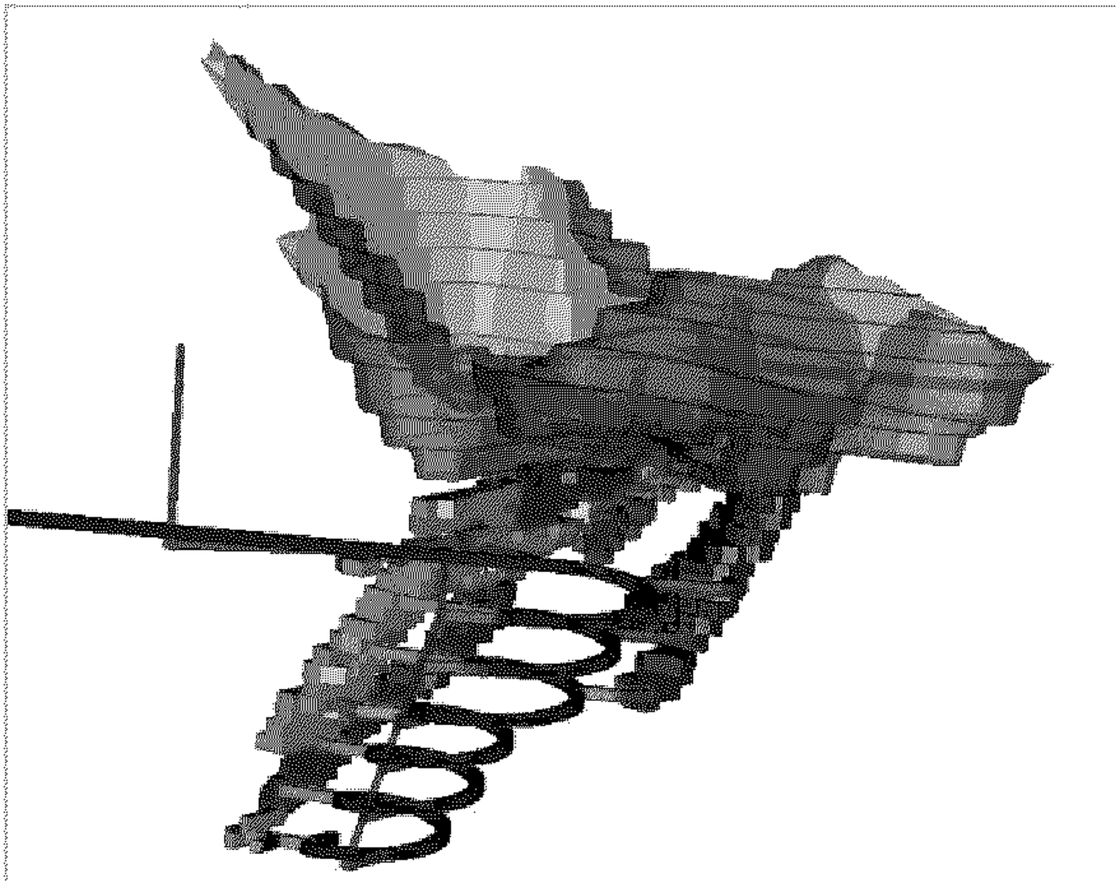


Figure 18-4
3-D Mine View – Underground Mine Plan Below Kamila Main Open Pit, Looking Northwest



Underground Design

The mining method will be Longhole Sublevel Retreat, a bulk mining method that will provide for moderate operating costs and good production productivities. Based on the geotechnical analysis, it has been determined that the ground conditions are sufficiently competent that open stoping can be employed without significant ground support requirements. With the selected mining method, waste development will be kept at minimum since no mucking levels are required. Access to the sublevels will be provided directly from the main ramp from surface.

The bulk of the underground reserves are located in the Aztec and Inca zones, which are both relatively wide and moderately dipping veins with widths of 5 m to 25 m thick, and continuous strike lengths of ore grade material up to 120 m long. The dip of the zones varies between 55° and 65°, with some local flattening down to 40°. There is significant grade variation within the overall vein boundaries, with only approximately 50% of the mineralized vein material being above the underground mining cut-off grade of 3.0 g/t AuEq, in the selected mining areas. Nevertheless, there is sufficient continuity in the ore grade material to create continuous longhole stopes of up to 120 m in length. These stopes typically have one side on the lithology contact while the other side is within the zone and is defined strictly by a grade cut-off with lower grade material bordering that stope wall.

Stope sublevels will be 4.1 m high and located every 20 m vertically within the ore zones. This sublevel spacing was selected to limit the blasthole lengths to approximately 17 m so as to avoid excessive blasthole deviation during drilling. These sublevels will be excavated to full ore width unless the ore width exceeds 12 m at which time temporary pillars will be left to support the back. The sublevels will be mapped and sampled providing detailed information on the vein boundaries and grade distribution at those elevations. This data will supplement the diamond drill information and will be used to provide accurate ore interpretations required for the longhole drill layouts. A greater sublevel spacing would reduce the accuracy of these interpretations.

The veins will be mined longitudinally using longhole retreat with the blasted ore being loaded using scooptrams equipped with radio remote control. The radio remote control will be required to permit loading of ore located out in the open stope beyond the safety of the brow.

The size of the stopes has been limited to a maximum of 120 m in length and 100 m in height for stope stability. Where required rib pillars have been provided to ensure the stopes don't exceed this length. In all cases, the rib pillars were left in low grade zones so only minimal ore grade material has been lost to these pillars.

For each of the lenses, a top-down stoping sequence with mucking of blasted ore from each sublevel will be employed.

Stoping Design

The stope limits were defined as described in Section 17.14.

The ore continues up through the base of the open pit and an 18-28 m thick temporary crown pillar has been provided so that mining of the open pit and underground can continue simultaneously. The temporary crown pillar will be left in place until mining of both the underground and Kamila Main open pit are complete. Underground mining will continue through to the bottom of the stopes before the 18 m crown pillar is to be recovered.

Underground mining will begin on the 2360 level and 2350 level stoping panels in Aztec and Inca respectively, and then progress downwards.

Recovery of the crown pillar will be achieved by first backfilling the underground stopes with waste rock. The backfill will serve two purposes; first, the top of the backfill will provide a platform just below the crown pillar for access by the scooptram and longhole drill to drill and muck the pillar following blasting. If the stopes were not backfilled, the blasted ore from the crown pillar could theoretically be mucked from the base of the open stopes but it is very likely that much of the ore would be caught up in the stope and not recovered. Second, the backfill will restrict any future caving that would complicate the long term closure plan.

The waste will be placed into the stopes from the bottom of the pit through short raises within the crown pillar and also by trucks hauling waste underground via the ramp. Waste from ongoing mine development will also be placed into select stopes as much as possible during regular mine operations. For example the Inca N Zone will be mined early in the underground production plan and would provide a location to place waste. The backfill will be levelled to just below the crown pillar leaving a 4.0 m high void to allow for access by underground drills and scooptrams.

The crown pillar will then be drilled and blasted from underground using the sublevel retreat method. Once the crown pillar has been mined the resulting void created at the bottom of the pit will be backfilled with waste to provide it with long term containment.

There are four remnant stopes above the pit bottom totalling 34 Kt which will be mined by underground methods. Two of these remnants are actually extensions of the Aztec crown pillar that extend above the pit bottom and into the pit walls. These two stopes will be mined as an integral part of the crown pillar with the ore being mucked from underground. The other two remnants are vein extensions into the pit walls and will be mined by underground sublevel retreat with access provided from the pit.

Initial construction and development of the underground mine will start in the third quarter of Year 2 with production beginning in Year 3. The first ore production will be from development of the first sublevel in ore (level 2380 in Aztec). The production rate for underground will be 500 t/d of longhole production plus development ore from the sublevels (sublevel development will contribute 20% of total ore). The underground could produce at a rate of up to 1000 t/d.

Access and Development

The Kamila underground mine will be accessed to a total vertical depth of 180 m from surface by a 1,248 m long ramp inclined at -15%. The ramp will be 4.6 m wide by 4.5 m high. The upper segment of the ramp connecting the ramp portal, located at 2,443 elevation, to the first sublevel access (SL 2380) will be 423 m long. The lower segment of the ramp will extend to the bottom of the mine at 2,258 elevation, connecting the remaining sublevels accessing the Aztec and Inca veins at regular 20 m vertical intervals. This segment will be 825 m long, designed as a 19 m radius spiral ramp.

The ramp has been located between the Aztec and Inca zones to minimize the length of the accesses to the sublevels. From this ramp location each stope access will be approximately 40 m long and including the accesses to the ventilation raises. Sublevel development within the stopes and veins has been estimated to be 1,646 m in ore and 260 m in waste low grade) to connect the stopes on the sublevels.

The portal will be excavated in a hillside with a solid outcrop minimizing the surface excavation and ground support required. The portal brow and first 25 m of ramp will be well supported including resin grouted rebar, weld mesh and shotcrete as required.

Remuck stations (4.6 m x 4.1 m x 15 m long), to provide for loading of trucks, will be excavated every 150 m along the ramp except where other development will be available to serve this purpose. Safety bays (2.0 m x 2.0 m) will be excavated every 50 m along the ramp. As the ramp progresses and the locations for the various access drifts to the sublevels are reached, these drifts will be advanced according to the development schedule, and will also serve as remuck stations and safety bays as appropriate.

A ventilation and escapeway raise system will be developed from surface through to the lowest sublevel (2258 SL). The raises will be 2.4 m wide by 1.8 m wide and vary in dip depending on the location. The raise system will be equipped with a steel ladderway to provide an emergency escape route from the mine. The raise system has been planned to parallel the south side of the spiral ramp except for the top segment where a lateral ventilation transfer drift was provided to extend the raise system outside the limit of the Kamila main pit. At each loop of the ramp there will be connection drift to the ventilation raise system. As the ramp is driven downwards and reaches the location of the next drift connection to the raise system, that connection and the segment of the raise from it to the connection above will be will be immediately excavated to provide for a through ventilation circuit for the ramp activity while it advances. A total of 176 m of raises will be required.

All ore and waste rock resulting from the development will be hauled up the ramp to surface and then transported directly on surface either 900 m to the ore stockpile at the process plant or 1,000 m to the waste stockpile.

A summary of the underground development requirements is shown in Table 18-2.

Table 18-2
Summary of Underground Development Requirements

Description	Grade %	Size (m)	Length (m)
Lateral and Ramps			
Upper Ramp (2443 el to 2380 el)	15.0	4.6 x 4.5	423
Lower Ramp (2380 el to 2258 el)	15.0	4.6 x 4.5	825
Stope and ventilation raise accesses	3.0	4.6 x 4.1	599
Infrastructure (substation, sump)	3.0	varies	55
Sublevels in waste (low grade)	3.0	4.6 x 4.1	260
Sub Total Waste			**2,162**
Sublevels in ore	3.0	4.6 x 4.1	1,646(1)
Total lateral and Ramps			**3,808**
Raises	50^0 -90^0	2.4 x 1.8	176
Total			**3,984**

Note (1) Ore development length is "equivalent length" based on converting slashing of the sublevels to full ore width to an equivalent length based on a 4.6 x 4.1 m size drift.

Table 18-3 shows the development schedule over the life of the underground mine.

Table 18-3
Underground Development Schedule

Description	Width (m)	Height (m)	Units	Year 1	2	3	4	5	Total
Ramp/access	4.6	4.5/4.1	m advance		458	1,024	680		**2,162**
Ore Drifts	4.6	4.1	m advance			1,015	631		**1,646**
Subtotal Lateral			**m advance**		**458**	**2,039**	**1,311**		**3,808**
Raises	2.4	1.8	m advance			113	63		176
Total			**m advance**		**458**	**2,152**	**1,374**		**3,984**
Tonnages									
Development Ore			t x 1000			54.5	32.6		**87.1**
Development Waste			t x 1000		26.3	58.7	40.3		**125.3**

Water Inflows and Pumping

According to hydrogeological analysis conducted by Knight Piésold (KP) water inflows to the mine will be minimal and dewatering requirements will not be significant. The majority of the pumping requirements will be to remove process water introduced into the mine for drilling, wash down of headings for geological mapping, and spraying of muck piles and roadways for dust control.

The mine plan includes a permanent collection sump and pump station at the bottom of the mine and a pair of submersible pumps (one as a spare) with a capacity of about 22 m^3/h. It is expected that this pump would only operate a maximum of 25% of the time.

While the ramp is being driven submersible pumps will be used to dewater the mine.

On surface, the water will be pumped to a settling pond located adjacent to the portal to remove sediments and then will be either reused in the process plant, sprayed on the roadways for dust control or otherwise will evaporate.

Underground Mining Equipment

Ore will be loaded into 26 t trucks by scooptrams and hauled up the ramp to the portal and then on surface 900 m to the processing plant. The type of truck selected is an articulated type surface construction truck such as the Volvo A30-D. This truck is available in a low profile underground version capable of travelling up 15% ramp grades at good speeds. This type of truck is used in a number of underground mines in South America and costs half that of typical underground mining truck. Two trucks will be required; one full time for production and one shared between development and production.

A 6.0 m^3 scooptram with a 14 t tramming capacity will be used for development and longhole production, and for standardization the same model be used for both functions. A radio remote controlled scooptram will be required for mucking in the open longhole stopes out beyond the safety of the brow. An electric hydraulic two boom jumbo will be used for mine development, but an electric hydraulic top hammer longhole drill rig will be used for production drilling in the stopes.

Support equipment will include a scissor lift truck for rockbolting and construction, a boom lift truck for materials handling and construction, a personnel carrier, a supervisor's vehicle and pick up trucks. Other non fixed equipment will include handheld drills, portable axivane fans, portable submersible pumps, and a shotcrete machine. The underground equipment is listed in Table 18-4.

Table 18-4
Underground Mobile (Non-fixed) Equipment

Equipment	Type	Typical Model	No. of Units
LHD (6 m3)	Diesel radio remote control scooptram – 14 5 tramming capacity	Sandvik Toro 1400	2
Haulage truck – 28 t	Articulated surface type Truck	Volvo A30D	3
Development drill jumbo	Electro-hydraulic two boom Jumbo	Sandvik Axera 5, 1F	1
Production drill jumbo	Electro-hydraulic longhole drill rig	Sandvik Solo 5C	1
Scissor lift truck	For rockbolting with jackleg/stopers	Normet Utilift 1430X	1
Personnel carrier	12 passenger	Toyota	1
Supervisors vehicle	5 passenger with cargo space	Toyota	1
Pick up trucks	3 passenger	Toyota	4
Handheld drills	Jackleg and stopers		12
Fans	Portable axivane – 50 hp		6
Fans	Portable axivane – 30 hp		6
Portable Pumps	Submersible electric – 30 hp		6
Shotcrete machine	Wet mix type		1

Mine Ventilation

During development the ramp will be force ventilated by fans through ventilation ducting (1.1 m diameter). However, raises will be provided at various points along the ramp to reduce the length of the ducting. The raises will then be connected to each other to serve as the main fresh air conduct.

AMEC has calculated that the requirement of fresh air in the mine will be 59 m^3/s. A leakage factor of 20% has been applied given that the presence of open stopes will increase the chances of air recirculation.

A 2.4 x 1.8 m ventilation raise will be provided from surface to the lowest level of the mine. The permanent ventilation circuit for the mine will be for the full mine airflow to enter the mine via downcasting through the ventilation raise to the bottom of the mine and then exhausting up through the ramp to surface. Bulkheads with regulators will be constructed in each of the raise access drifts along the ramp to prevent short circuiting of air out onto the ramp prematurely. If required, air can be drawn from the flow in the raise through one of these bulkhead regulators to be fed more directly to one of the sublevels. A set of ventilation airlock doors will be required at the entrance to the ventilation transfer drift to control airflow at that location.

The ventilation air will be directed into the active sublevels from the ramp by axivane fans hung in the ramp and forcing air through ducting to the work places in the sublevels. Temporary air brattices will be installed in the stope access drifts to control short circuiting of air through the open stopes.

A 112 kW permanent fan will be mounted on top of the ventilation raise to force the air into the mine – the system will be a "push" ventilation system.

18.1.3 Production Schedule

The mine is scheduled to produce at a rate of 1,000 t/d of ore for 365 d/a for a total of 365,000 t/a. During the preproduction period (Year -1) 34.2 Kt of ore will be mined and stockpiled at the plant until processing starts in Year 1. In addition to the 365,000 t/a of ore production a total of 84.3 Kt of low grade ore will be mined and stockpiled and then processed at the end of the mine life. In the first two years the annual ore production of 365,000 t will be mined from the open pit. In Years 3 to 5 the underground will be in production resulting in a major reduction in the pit production requirement.

The production schedule is based on starting mining in the small Kamila SE pit while the Kamila Main Pit is being prepared. As soon as the Kamila SE pit is mined out then mining of ore will start at the Kamila Main Pit. The low grade is kept separate in the schedule because it will be stockpiled and only processed at the end of the mine life. The overall production schedule for the Project is given in Table 18-5.

Table 18-5
Project Production Schedule

Mining Area	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Total
Open Pit							
Tonnes (kt)	34,196	364,971	365,000	232,380	168,821	102,810	**1,268,179**
Au (g/t)	7.04	4.57	7.01	5.95	4.36	2.50	**5.40**
Ag(g/t)	71	54	107	115	113	101	**93**
AuEq (g/t)	7.92	5.25	8.35	7.39	5.77	3.75	**6.56**
Underground							
Tonnes (kt)	-	-	-	132,620	196,179	110,471	**439,270**
Au (g/t)	-	-	-	3.24	2.98	3.72	**3.24**
Ag(g/t)	-	-	-	209	203	153	**192**
AuEq (g/t)	-	-	-	5.85	5.52	5.63	**5.65**
Subtotal - Direct Mill Production (Open Pit + Underground)							
Tonnes (kt)	34,196	364,971	365,000	365,000	365,000	213,281	**1,707,449**
Au (g/t)	7.04	4.57	7.01	4.96	3.62	3.13	**4.84**
Ag(g/t)	71	54	107	149	161	127	**118**
AuEq (g/t)	7.92	5.25	8.35	6.83	5.64	4.73	**6.32**
Stockpile							
Tonnes (kt)	-	24,014	23,208	14,599	10,344	12,149	**84,315**
Au (g/t)	-	1.34	1.27	1.16	1.13	1.22	**1.25**
Ag(g/t)	-	18	23	29	33	30	**25**
AuEq (g/t)	-	1.56	1.56	1.53	1.54	1.59	**1.56**
Total - Open Pit + Underground + Stockpile							
Tonnes (kt)	34,196	388,986	388,208	379,599	375,344	225,430	**1,791,763**
Au (g/t)	7.04	4.37	6.67	4.82	3.55	3.03	**4.67**
Ag(g/t)	71	52	102	144	158	122	**114**
AuEq (g/t)	7.92	5.02	7.95	6.62	5.53	4.56	**6.10**

18.1.4 Waste

Over the life of the Project the mine will produce a total of 10 Mt of waste rock. Most of the waste will be produced from the open pits with a small portion also coming from the underground mine development. Of this total 8 Mt will be deposited in a waste dump located just to the south east of the Kamila open pit and 2 Mt will be disposed of as backfill in the underground mine and dumped back into Kamila pits.

From a mining point of view, the waste dump is well located being adjacent to the Kamila SE pit and 350 m from the main Kamila pit.

Some waste rock is classified as low-grade potential mill feed and will be stored separately within the waste dump such that it can be recovered and fed to the mill if higher base gold and silver prices than those used in the Feasibility Study are realized.

Once the Kamila main pit has been mined out some waste will be returned to the underground mine as backfill. This waste will be supplied by hauling waste being stripped from Mercado direct to the underground mine.

The schedule of waste rock production is shown in Table 18-6:

Table 18-6
Waste Rock Schedule (t x 1000)

Mining Area	Year						Total
	-1	1	2	3	4	5	
Waste Mined							
Kamila Main Pit	104	3,682	2,458	1,408	962	132	8,747
Kamila SE Pit	362	-	-	-	-	-	362
Mercado Pit	-	-	-	-	-	678	678
Potential	1	38	35	17	16	24	130
Underground			26	59	40		125
Total	**466**	**3,720**	**2,519**	**1,484**	**1,019**	**834**	**10,042**
Backfill Pits and Underground							
Road Construction	100						100
Kamila SE Pit			317				317
Kamila Main Pit				400	400	500	1,400
Return to Underground for Backfill						225	225
Total	**100**	**-**	**308**	**-**	**-**	**725**	**2,042**
Stockpile (at end of year)(1)	366	4,086	6,288	7,372	7,891	8,000(2)	

Note (1) includes potential stockpile; (2) ultimate waste stockpile at end of mine life.

18.1.5 Material Handling

Ore from both the open pit and underground will be trucked to the Run of Mine ore stockpile located at the plant. The ore will be loaded from the stockpile by a front-end loader and hauled to the feeder to the crusher. Any oversize will be moved to one side and the transported to a blasting pad where the large pieces will be blasted. To even out grade fluctuations to the mill blending of ore from the open pit and underground and from higher grade sources can be achieved by controlling dump areas the sequence with which the front-end loader feeds the mill.

The low grade ore will be placed on a separate stockpile and stored there until it will be processed at the end of the mine life. The potential mineralization will be stockpiled to one side of the waste dump and would be fed to the mill at the end of the mine life if metal prices at that are high enough to pay for the loading and processing.

A dozer will be assigned full time to the waste dump to spread the waste after it has been dumped by the trucks.

18.1.6 Mine Services

The services to be provided for maintenance and infrastructure for mining will be minimal due to the simple and compact nature of the mine and the short mine life planned.

The only permanent maintenance facilities to be provided will be a shop on surface which will service both the open pit and underground operations. All equipment repairs on the mobile equipment will be completed in that shop. The only exception to this would be routine repairs to the open pit and underground as these are slow moving units and cumbersome to bring to the shop. However, if major repairs would be required these units would be brought to the surface shop.

The power requirements for the open pit are minimal as almost all equipment will be diesel powered including the production drills. The main power requirement before the start of the underground mine will be to supply power to the mining contractors surface maintenance and office facility as well as for pit dewatering. The underground will use a significant quantity of electrical power for ventilation fans, production and development drill rigs, pumps, compressors and lighting. These loads will start in the middle of Year 2 as the ramp development begins and then will reach full load in the remaining years. The total average estimated load during Year 1 and the first part of Year 2 will be 73 kW increasing to a total of 480 kW for Year 3 and onwards.

Electrical power for the underground mine will be provided to the mine at 6.9 kV and a 6.9 kV electrical distribution cable will be installed in the ramp system to provide power to the mine. A total of three portable substations will be used to provide 400 V power to the equipment including the drill rigs, fans, pumps and some local lighting. One of the substations will be located on surface at the underground portal and the other two will be installed in cut-outs underground close to the workplaces.

Underground mining compressed air will be required for operation of jackleg and stoper drills for rockbolting, explosive loaders, raise development and on occasion shotcreting. The compressed air requirements have been estimated to be an average of 1,200 m^3/h with a occasional peak of 2,700 m^3/h and this will be provide by two 1,700 m^3/h compressors located on surface close to the portal. Only one of these compressors will be required during normal operations, the second is to be provided as a spare and for peak requirements. During development a 150 mm diameter pipeline will installed in the ramp and sublevel access drifts to distribute the compressed air throughout the mine.

18.1.7 Mine Contractors

A specialist mining contractor will be retained to perform all the mining activities. The contract will be all inclusive with the contractor supplying all the required manpower, mining equipment, support equipment and infrastructure facilities to meet the scheduled mining rates.

Some services will be supplied by Intrepid:

- Prepare the site, prior to the mining contractor's arrival, and provide a levelled area suitable for the contractor to install temporary office, dry, maintenance shop and camp facilities (for location see site plan).
- Provide fresh water (drill water, not treated) to the site. The contractor will be responsible to connect to this supply and provide for all his required distribution.
- Provide the mine discharge water settling ponds for water pumped from the development.
- Check surveys only.
- Provide the location of the site facilities (plant, waste dump/tailings impoundment, warehouse, and powerhouse).
- Elaborate and provide a life of mine production and development schedule.
- Geology, grade control and mine planning including blasthole layouts will be the responsibility of Intrepid.

The mining contractor will as part of their services:

- Provide all equipment and consumable items required to meet production targets including trucks, excavators, drills, scooptrams, support equipment pumps, explosives, drilling supplies, and for underground ventilation fans and ducting, rockbolts, mesh, and shotcrete.
- Provide electrical power and all required substations and cables for all their requirements except during initial construction while the generating plant is being installed.
- Provide all required temporary offices, dry, warehouse and maintenance facilities.
- Provide for sewage and grey-water collection and also be responsible for the disposal of all used material from his facilities to the adequate waste dump locations where a specialized contractor will collect them. These costs should be listed separately since the owner may decide to provide waste disposal services.
- Supply employee housing off-site (city of Calingasta) because there will not be an on-site housing camp.

- Construction and maintenance of
- all pit haulage roads both within the pits and to the plant and waste dump.
- Construction of the initial 10 m wide pioneering road to the top of the Kamila pit.
- Be responsible for all aspects of the underground development including equipment, supplies, manpower, supervision, safety and surveying.
- For underground the permanent services in the main ramp, to be included in unit prices, will comprise 150 mm schedule 40 compressed air pipeline, 100 mm schedule 40 water discharge pipeline
- Provide ground support for the surface portal trench including resin grouted rebar, weld mesh and shotcrete as required prior to start of tunnelling.

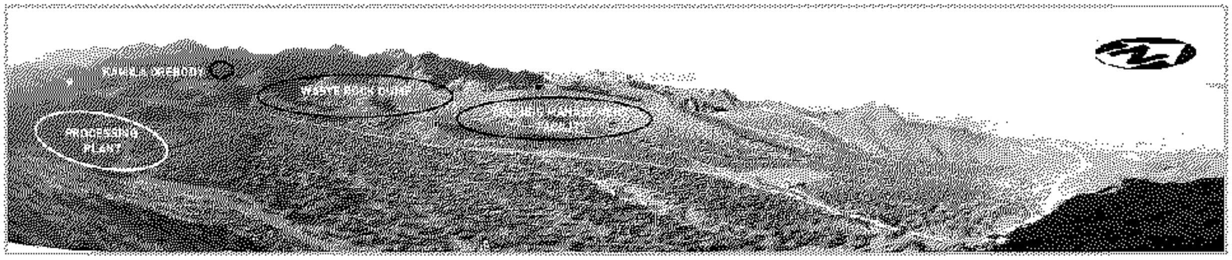
AMEC developed a contract mining document to solicit budget price proposals for mining at Casposo. A budget price proposal was received from a mining contractor with experience in Argentina and personnel located in San Juan. In addition to costs, this proposal included a plan on how the contractor proposed to execute the open pit mining portion of the contract. This plan included operations methodology, manpower levels, equipment requirements, and infrastructure building needs.

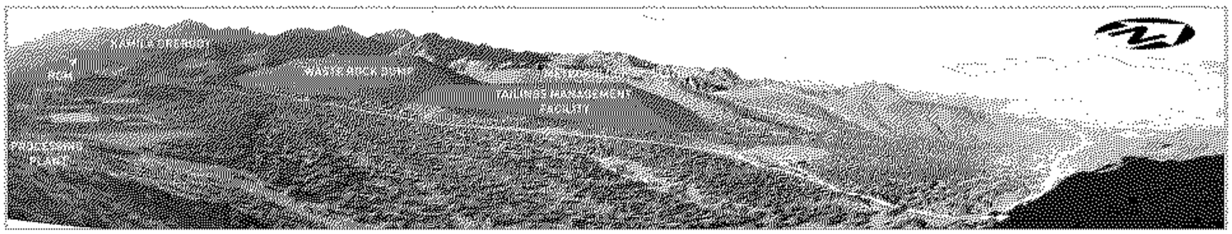
18.2 Mine Site Infrastructure

18.2.1 Site Selection and Investigations

Field investigations and a site selection study were carried out to identify the best locations for the plant site, waste dump and tailings facilities. These locations were examined by AMEC, Intrepid and Knight Piésold (KP) personnel during field site investigations. The finalized position of the facilities is shown in Figure 18-5.

Figure 18-5
Proposed Infrastructure Layout - Casposo Site





Plant and Administration Site

Three alternative plant sites locations (PL) were identified during the site visit. The first two sites are located to the south and down slope of the Kamila orebody area, while the third, PL3, is located west and upslope of the deposit. PL1 and PL2 are the preferred sites due to their relative location down slope of the mine and the proximity to the preferred tailings deposit alternative TD1.

The geotechnical investigation showed the ground in the preferred PL1 concentrator/administration area is adequate to support the proposed structures. Laboratory testing of a limited number of bedrock outcrop samples in the Project Area suggests that the foundation bedrock strength will be variable, but generally of high resistance. In the general area of the plant and tailings management facility the groundwater level associated with the lightly weathered and moderately fractured rhyolite bedrock appears to have stabilized at a depth of about 21 m below ground surface.

Waste Rock Dump and Tailings Management Facility

The Casposo Mine Project will generate an estimated 10 Mt of waste rock over its planned five-year operating life. About 2 Mt of the mine waste rock is currently planned for use in road construction, in-pit disposal and underground backfill. The balance will be stored in a waste rock dump (WRD).

Two potential WRD sites were identified, one in the head of a catchment area located to the southwest of the planned Kamila Open Pit and the other just southeast of the Kamila Open Pit area, along the south side of a low-lying bedrock ridge and north of the planned process plant location. AMEC carried out the estimation of the waste storage volume for these locations and there was enough capacity for all of the expected waste production only in the latter option. The site to the southeast of the Kamila Open Pit was identified as the preferred alternative, based on its more favourable relationship of storage capacity versus base area and its overall lower ground surface elevation compared to the alternative site to the southwest of the Kamila Open Pit. This is the site shown in Figure 18-5.

The Project will generate about 1.8 Mt of tailings over its planned five-year operating life. The following criteria were considered when locating the tailings impoundment site.

- Dam Stability – The tailings dam should be stable with minimal risk of failure considering foundation, flooding, seismic activity and construction materials.
- Aesthetics – The tailings dam should be located, constructed, operated, reclaimed and closed in a manner that minimizes the effects on environment as much as practical.
- Economics – Location and construction of the tailings dam should consider construction, operation, and closure costs.
- Water Quality – The location, construction, operation, reclamation, and closure of the dam should consider water quality from dam discharge and seepage.
- Watershed – The dam and impoundment should be located in a watershed that is adequate to contain the volume of tailings generated over the life of the mine and in a watershed that requires minimal routing or flow through of water runoff.

The estimated tailings mass for the proposed life of the mine development will require around 1.9 Mm^3 of storage capacity if disposed as slurry or around 1.4 Mm^3 if filtered and dry stacked. During initial field investigations AMEC identified five possible locations for the tailings disposal facility.

The five alternatives were studied to determine storage capacity, amount of material required for dam construction, ratio of storage capacity against dam volume and upstream catchment area.

Dams were used during the site evaluation only for comparison purposes since AMEC recommends that filter tailings be disposed as dry stack. The disposal options contemplated for the cyanide leached tailings included:

- Conventional, slurried tailings discharge by gravity pipeline to an impoundment (tailings dam) with water reclaim to the process plant.
- Thickened high density tailings discharge by pumping to a confined storage area (earthen embankment) by pipe.
- Filtered tailings transported by truck to a dry stack tailings facility.

TD1 is located down slope and east of the mining area and preferred plant site locations, has sufficient storage capacity and a relatively small catchment area. TD1 is the recommended tailings facility site. TD2, located northeast of the Kamila deposit, was discarded due to the large (1,377 ha) catchment area upstream and the associated difficulty and cost of constructing a system to divert large volumes of water.

TD3, located northwest and upslope of the mining area, has a 703 ha catchment area and the lowest rate of storage capacity to dam volume. Due to TD3's location, upslope from the mining area and preferred plant site, the tailings would need to be transported upslope from the preferred plant site or if the non preferred plant site is selected, the ore would need to be hauled uphill.

Two other alternatives, TD4 and TD5 were reviewed during the study, but discarded due to their inadequate storage capacity.

In general subsurface conditions in the waste rock dump area comprise shallow to no soil overlying competent rhyolite bed rock. The subsurface conditions encountered by the test pit exploration work near the preferred TMF area (TD1) consist of dense, dry, silty or clayey, sandy gravel with medium-plasticity fines. Fine-grained soils exist to 2 m below ground surface, underlain by 3 m of coarse grained granular alluvial sediments. Similar soil characteristics are inferred for the proposed WRD site, where soil cover exists. The granular soil encountered is underlain by moderately-weathered, medium hard to hard, greenish, greyish-brown volcanic rock. The depth to the inferred bedrock surface reportedly varies from 0 to 5 m. Moderately silicified, light-coloured rhyolite bedrock was reported from 5 m to 17 m depth in the borehole, then moderately altered and extremely weathered rhyolite to 48 m below ground surface. From 48 m to 80 m below ground surface, lightly weathered and moderately fractured rhyolite was reported. Based on this the foundation bedrock strength will be variable, but generally of high resistance.

Mine Access Road

Based on discussions with Intrepid personnel at site and in San Juan, AMEC understands that the current alignment of the gravel access road was developed several years ago after sections of the previous route were damaged by a significant storm runoff and erosion event. The current access road is located primarily along high ground in the alluvial deposits, but still crosses several large, naturally-established storm channels. AMEC agrees that the current access road alignment should be maintained for the Project, with modifications to the road width, curves and grades as required for mine transport. As well, modifications should be made to the storm channels at the access road crossings to reduce the extent of damage and temporary road loss in the event of another significant runoff and erosion event during the projected five-year mine operation.

Monitoring Piezometers and Project Water Supply

On the basis of hydrology and hydrogeological studies and surveys conducted by Knight Piésold a test water supply borehole was drilled, monitoring piezometer installed and pump tests conducted upstream in the Quebrada Vallecito river bed.

The most feasible water supply (pipeline) route to the plant appears to be along the north side of Quebrada Vallecito, then along the existing access road between Quebrada Vallecito and the proposed plant site. The water supply well is located far enough upstream in Quebrada Vallecito, to minimize pumping requirements as much of the head will be provided by gravity.

Geotechnical Investigations

General

Two geotechnical investigation programs were completed by AMEC on the Casposo site. One took place during December 2005 and the other during October 2006.

The first program included geotechnical mapping of outcrops and the excavation of trenches to investigate foundation conditions on alternative plant sites. In addition, potential sources for borrow materials, access routes and water diversion requirements were identified.

The purpose of the second program was to review the following:

- transport corridor conditions and potential concerns for large-vehicle movement between Calingasta and the turnoff to the Project Site on Route 412,

- report on and supervise a test pit exploration program within the revised proposed plant site,
- storm runoff structures and alternative routing options for the project's gravel access road,
- proposed monitoring and groundwater supply borehole locations proposed by Knight-Piésold which were subsequently drilled under their supervision later that month.

Geotechnical Structural Mapping

Structural mapping was carried out by AMEC along 507 m of exploration road cuts, covering 21 different areas. Of these areas, 18 were tested by window sampling and 3 with the scanline sampling technique. Over the 507 m length, 470 m covered rhyolite rock units and 37 m were in andesite. The total number of structures mapped was 750; 703 in the rhyolite and 47 in the andesite.

During structural mapping the following parameters were recorded: type of structure, rock alteration, number of fractures, persistence, termination, field resistance, aperture, fill, fill resistance, structure alteration, roughness, waviness, and water condition.

Test Pits

AMEC recorded the subsurface conditions as encountered in 23 test pits that were completed in the Project Area.

Six of the test pits were completed in the tailings dam alternative 1 site (DR1), two in the tailings dam alternative 2 site (DR2), three in the tailings dam alternative 3 site (DR3), one in the general area of plant alternative sites 1 and 2 (PL1 and PL2), three in the plant alternative 3 site (PL3), and eight in the revised proposed plant site (RPL). Test pit exploration work was not conducted in the proposed waste rock dump site, as ground reconnaissance indicated that the subsurface conditions generally comprised shallow to no soil overlying the bedrock.

Laboratory Testing

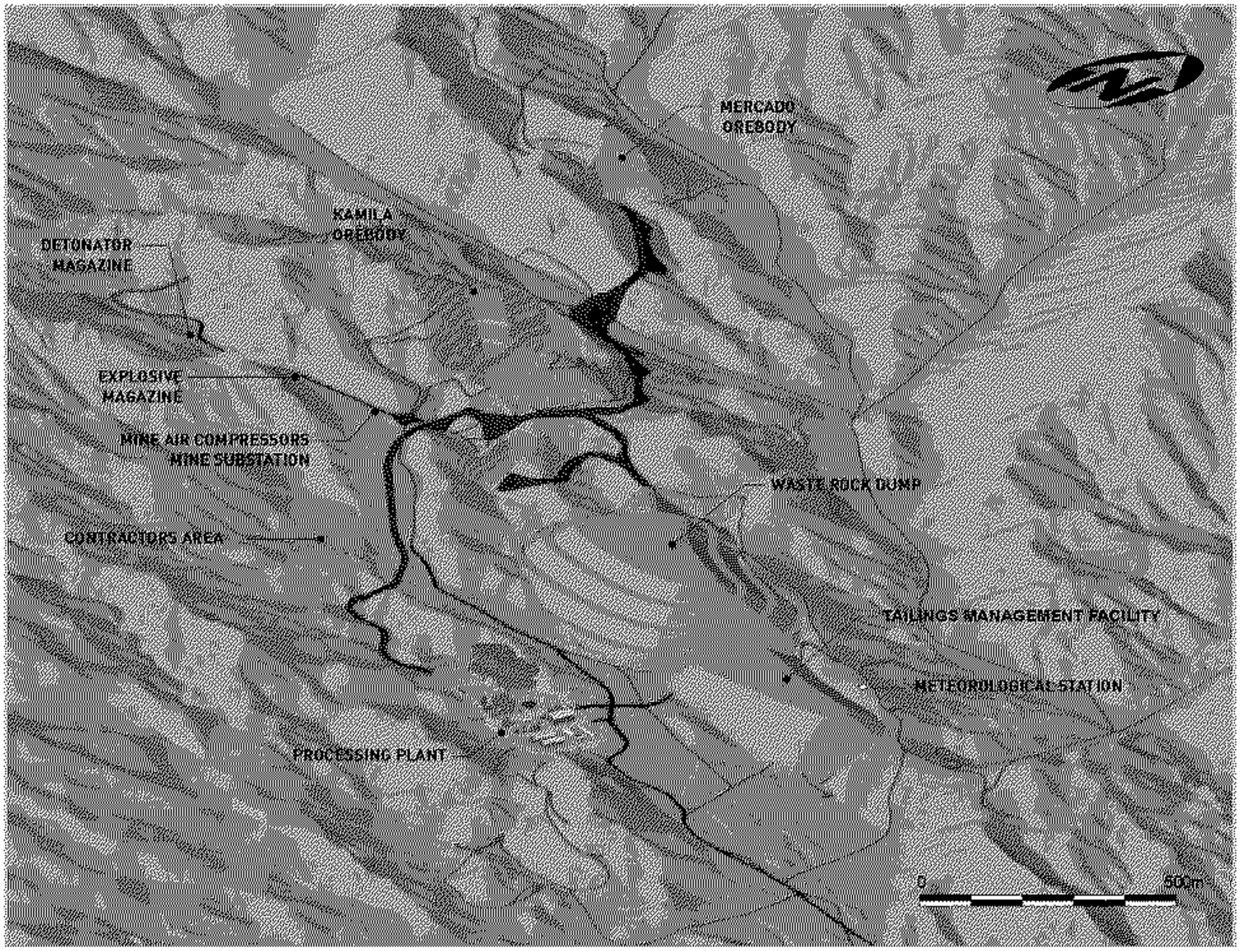
Four Unconfined (Uniaxial) Compressive Strength (UCS) and forty-three Point Load Strength (PLS) tests of individual rock pieces were carried out by the rock mechanics laboratory, at the Universidad Nacional de San Juan in December 2005. The samples for the tests came from exploration boreholes CA-04-99 and CA-04-97. The PLS of andesite rocks ranges from 3 to 9 MPa, whereas those for rhyolite ranged from 2 to 12 MPa. The mean UCS of the andesite was 83.4 MPa, whereas the mean rhyolite UCS was 105.9 MPa.

Fourteen samples collected from 11 of the test pits and three possible quarry alternatives were sent to San Juan for soil classification purposes. The samples were classified through a particle size analysis (sieve analysis). Ten Proctor tests (belonging to 5 TD1 samples, 2 TD2 samples, 3 TD3 samples and 1 PL1 sample) were carried out by the Universidad Nacional de San Juan. The moisture content of seven samples (3 TD1 samples, 1 TD2 samples, 2 TD3 samples, 1 PL1 sample, and 2 quarry alternative samples) were also evaluated by the Universidad Nacional de San Juan, as was the apparent and absolute density of three samples, QU1–QU3.

Final Site Layout Design

The waste dump will be located southeast of the Kamila Open Pit area, along the south side of a low-lying bedrock ridge and abutting the dry stack tailings facility. The administration, services, process, and tailings filtration facilities will be located near the dry stack tailings facility, also within easy access to the mine site access road. The mine maintenance facilities will be located on a bench about 1 km from the western edge of the ultimate pit. The water supply well will be located in the upstream in Quebrada Vallecito. A three-dimensional schematic of the site layout is shown in Figure 18-6.

Figure 18-6
Overall Site Facilities Layout 3-D



18.2.2 Site Facilities

Planned site facilities include:

Ore Storage

Run-of -mine coarse ore stockpile.

Plant Area

- Crusher plant
- Fine ore storage bin and emergency stockpile
- Grinding and gravity
- Counter current decantation thickener plant
- Filter plant and tailings stockpile
- Cyanide destruction and process water distribution
- Process water collection pond
- Plant control room (with internal office and slurry sample preparation room).

Administration

- Gatehouse and weigh scale
- Administration and engineering offices
- Assay laboratory
- Warehouse, and storage yard
- Maintenance workshops
- Canteen and infirmary
- Power generator plant
- Fuel storage
- Reagent mixing
- Fresh water and fire water storage and distribution
- Potable water plant
- Air services
- Merrill Crowe and refinery
- Site water collection pond

Mine Area

- Contractors area (offices, warehouse, heavy vehicle repair shop, fuel tank) and facilities
- Explosives magazine
- Detonator magazine
- Underground air compressor services
- Underground mine water settling pond

Water Well Supply Area

- Wells
- Fresh water supply tank and distribution pipe

18.2.3 Plant Site Drainage

In the administration zone there are several channels with a slope of 1.5% whose purpose is to collect and divert the rain. The water from the channels is collected by a marginal ditch. These waters are then directed to the site water collection pond.

In the plant zone run-off as well as potential spillage from the piperack and emergency overflows will be collected in a central trench and directed towards the process water pond.

18.2.4 Site Roads

The design of the haul roads considers a road width of 10 m, a maximum slope of 10% and a minimum radius of 40 m.

For the design of the new access roads the configuration of the existing roads will be taken into consideration; these have an average width of 4.5 m and an average slope of 10%.

The design parameters of the new access roads include a road width of 6 m, a maximum slope of 10% and a minimum radius of 40 m. The main purpose of the access roads is the communication among the different facilities of the mine

Most of the access roads are located over existing roads; in these cases the grade line needs to be improved and road width needs to be broadened. Only one section of about 570 m of length is new, it is located between the processing plant and the tailings management facility.

18.2.5 Waste Rock Dump and Tailings Management Facility

Waste Rock Dump

The WRD will be located just southeast of the Kamila orebody zone, with a ramp constructed of waste rock being initiated and raised as required to provide access to the working platform. The dump will be developed in a conventional manner, with haul trucks transporting and end-dumping the waste rock and dozers spreading and configuring the material on the working platform.

About 95% by mass of the waste rock from the Kamila and Mercado Open Pit operations will be comprised predominantly of rhyolite (87%) and andesite (8%). Quartz porphyry, quartz vein rock, pyroclastic breccia and other rock types will constitute the remaining 5% of the waste rock material. The host rock in the planned open pit zones is expected to be very competent and require close blasting patterns, and should produce well-graded, strong, durable and angular particles. Sulphur contents greater than 0.5% by mass, which can be an indicator for acid mine drainage (AMD) potential, have been identified in approximately 12% of the rhyolite and about 8% of the andesite in the Kamila zone. The Mercado zone, which represents about 10% of the total mining tonnage, has a lithological composition similar to the Kamila zone. Geological modelling indicates that the high sulphur content zones within the Kamila zone are well-defined, with little distribution or spreading of the higher sulphur values in the surrounding rock.

The results of geochemical analyses and metallurgical mapping indicate that the quartz vein ore rock will not have significant sulphur contents with respect to AMD potential; data from humidity cell testing to date support this inference.

Annual waste rock generation will range from about 0.8 Mt to 3.7 Mt over the planned five-year mine operating life, with an additional 0.5 Mt estimated from pre-production stripping work. A quantity of waste rock will be used for road construction in the pre production and latter stages of the mine operation, and as in-pit disposal material and underground backfill in the final year of operation.

Tailings Management Facility

The washed, rinsed and filtered tailings will be transported by haul truck to the TMF where they will be end-dumped and dozer-tracked. A heavy compactor will be used to provide additional densification to the tailings and reduce the potential for wind erosion during operation, as required. The TMF will abut the southeast end of the WRD and extend further to the southeast, and will be provided with a geomembrane liner over the prepared foundation surface. The liner will extend within the east side of the WRD, in order that a barrier to any downward seepage infiltration to the native foundation soils is maintained beneath the entire tailings deposit.

The rinsed and filtered tailings will be a silt-sized material having geotechnical properties typical of milled, hard rock gold tailings. Residual cyanide concentrations in the rinsed tailings will be less than 50 parts per million (ppm), and the tailings will be dewatered to a moisture content of about 18% (mass water to mass dry solids) by the belt-filtering process.

The TMF will be progressively closed as it is operated, using non-reactive, suitable waste rock to construct a cover resistant to erosion by wind, direct precipitation and water runoff.

Monitoring

Geotechnical and environmental monitoring of the WRD and TMF operation will include regular topographic surveys, water balance evaluations and monitoring well sampling and laboratory testing. Additional geotechnical and/or environmental monitoring instrumentation and procedures will be introduced as required. The monitoring programs will be continued into the mine closure and post-closure periods as required to verify that the geotechnical and geochemical design criteria for mine closure have been achieved.

18.2.6 Power Supply and Distribution

The Casposo site power station will be a diesel fuelled generation facility based on multiple medium speed reciprocating engines, placed on an open slab, complete with cooling and all auxiliary equipment. The generator controls and the general distribution board that gathers and distributes the power from the generators will be located in a building adjacent to the power generators.

The power output is rated to meet the process plant, ancillary support loads and the mine requirements. An additional installed spare unit will be provided to allow any one unit to be on standby or out of service for maintenance or repair.

It has been estimated that the power requirements of the first open pit phase will require the use of 3 power generators capable of supplying 1,000 kW each of constant power at constant load and an extra standby generator of the same characteristics. For the underground phase an additional generator will be required to supply the power requirements of the Kamila underground mine; totalling 5 power generators in an N+1 configuration.

The power station will be located close to the largest loads, which are located in the grinding area of the process plant, to minimize both losses and capital cost. Power to the site loads will be distributed from the powerhouse in 3 phases at 6.6 kV and 50 Hz using cable and overhead lines as appropriate.

The main electrical loads are as follows:

- Main process plant (and crushing)
- Service complex
- Ancillary loads including water distribution/handling and treatment
- Underground Mine

18.2.7 Communications

The Casposo communications system will be satellite based and provide the following:

- Voice-over internet protocol for offsite telephone communications
- A local area network on site, which can be expanded to provide a wide area network incorporating Intrepid's head office in Toronto
- 20 telephone trunk lines, which will simultaneously support 15 to 20 phone calls, a fax and a number of computers on the internet, with high quality service. Should this prove to be insufficient, the bandwidth can be increased in less than one hour, remotely by the service provider.
- Toronto dial tone (no long distance charge). Other city dial tones can be provided should it be necessary.

18.3 Capital Costs

The capital cost estimated to design and build the Casposo Project is US$45.5 M. All costs are expressed in fourth quarter 2006 US dollars, with no allowance for escalation, interest during construction, or taxes. All estimates herein are quoted in fourth quarter 2006 US dollars, and 3.097 ARS/$ and 1.1371 CAD/$.

AMEC developed a baseline cost estimate of US$3.7 M for the EPCM costs to execute the Casposo Project.

No taxes have been included because it is assumed IVA tax paid during the construction period will be recovered. However an allowance of 4.0% of the direct capital equipment cost has been included in the estimate for duties (ad-valorem, custom agent and others). The contingency used is 12.0% based on an @Risk Simulation analysis.

The capital cost summary is shown in Table 18-7.

Table 18-7
Capital Cost Estimate Summary ($US x 1,000)

AREA	DESCRIPTION	Imported Supply	Local Supply	CONSTRUCTION and ERECTION MHs. x1000	Direct Cost	Indirect Cost	Sub Contract	S_TOTAL	TOTAL COST
	TOTAL CAPITAL COST ESTIMATE	**10,449**	**7,295**	**381**	**4,827**	**5,595**	**17,301**	**27,724**	**45,468**
100	**Open Pit**			**3**			**2,466**	**2,466**	**2,466**
110	Pit Haul Roads			3			186	186	186
120	Pit Preproduction Stripping						1,568	1,568	1,568
160	Pit Dewatering						9	9	9
190	Mining Contractor Facilities						702	702	702
200	**Process Plant**	**8,320**	**4,607**	**232**	**1,972**	**2,961**	**475**	**5,408**	**18,335**
210	Crushing & Fine Ore Storage	720	1,103	32	270	400		670	2,494
220	Grinding	2,246	1,029	35	245	379	250	873	4,148
230	Gravity	314	23	4	24	37		61	397
240	Leach Circuit	615	1,106	58	475	634		1,110	2,831
250	CCD Wash & Filtration	2,390	499	73	569	840	75	1,485	4,374
260	Merrill Crowe	904	489	13	194	346	75	616	2,008
270	Refinery	484	63	1	76	137		213	760
280	Cyanide Destruct & Process Water	48	62	4	47	61		107	217
290	Reagents Handling	208	137	5	25	47	75	148	493
295	Plant Services Equipment	390	97	7	46	80		126	613
300	**Process Plant Buildings**		**111**	**16**	**186**	**216**	**403**	**805**	**916**
305	Plant Service		5	2	9	11	50	71	76
310	Electrical Room			1	4	5	203	212	212
315	Merrill Crowe and Refinery		102	12	164	190		354	456
320	Reagents		4	1	7	8	36	51	55
325	Control Room			0	1	2	113	117	117
400	**Site and Services**	**634**	**1,511**	**105**	**1,978**	**1,608**	**2,268**	**5,854**	**7,999**
410	Site Prep and Onsite Civil Works and Roads		64	41	1,361	807		2,168	2,232
415	Mine Haulage Road								
417	Truck Scale	27	1	2	14	19	4	37	64
420	Access Road			7	207	117		324	324
435	Plant Mobile Equipment		587						587
440	Fresh Water Supply & Distribution	61	292	10	54	94	14	161	513
450	Power Supply, Generators & HVAC		73	13	132	155	2,250	2,536	2,609
460	Plant Site Substation and Primary Distribution	350	151	11	83	201		284	785
470	Fire Protection and Prevention	73	64	4	25	44		69	206
490	Fuel Storage	63	249	10	58	102		160	472
495	Sewage	61	30	7	45	70		115	206
500	**Tailings and Waste Rock Management**						**1,250**	**1,250**	**1,250**
520	Surface Tailings Impoundments						1,000	1,000	1,000
530	Surface Waste Rock Management Facility						250	250	250
600	**Ancillary Facilities**		**99**	**25**	**174**	**210**	**1,400**	**1,784**	**1,883**
610	Administration Building		8	3	24	28	106	157	166
620	Plant Warehouse		16	5	24	30	200	254	270
625	Infirmary & Canteen		4	1	9	10	49	68	71
630	Maintenance Shop		55	9	41	53	280	375	430
635	Small Tools, Shop Equipment, Warehouse Shelving & Bins						90	90	90
638	Administration Furniture, Canteen & Office Equipment						100	100	100
655	Security Mine Access (Gate & Cabin)		0	0	0	0	3	3	3
670	Assay Laboratory		16	5	25	30	158	212	228
675	Laboratory Equipment & Initial Supplies						250	250	250
680	Communications						145	145	145
685	Surveying Equipment						20	20	20
690	Site Water Management			1	51	59		110	110
900	**Indirects**	**376**	**186**				**6,077**	**6,077**	**6,638**
930	Freight & Duties						2,371	2,371	2,371
940	Capital Spare Parts & First Fills	376	186						561
950	Engineering, Procurement, Construction Management (EPCM)						3,707	3,707	3,707
3000	**Owner Costs**						**1,243**	**1,243**	**1,243**
3010	Pre-production Employment & Training						500	500	500
3100	Canteen Operations						50	50	50
3200	Accomodation Allowance						35	35	35
3250	Mobile Equipment and Vehicles						35	35	35
3300	Communication Expenses						84	84	84
3400	Insurance						25	25	25
3500	Corporate Services						86	86	86
3600	Environmental						30	30	30
3700	Security/Medical						12	12	12
3800	Travel and Transportation						75	75	75
3900	Community Development						50	50	50
3920	Vendor Representatives & Commissioning						96	96	96
3950	Land Acquisition & Holdings Costs						100	100	100
3960	Consultants						50	50	50
3960	Legal, Permits, and Fees						15	15	15
9000	**Other**	**1,120**	**782**		**517**	**600**	**1,721**	**2,837**	**4,738**
9800	Contingency (12 %)	1,120	782		517	600	1,721	2,837	4,738

18.4 Operating Costs

Three distinct operating phases are considered in the development of the operating costs:

- Years 1 to 2 Initial mining in the Kamila open pit and process plant operations.
- Years 3 to 5 Combined open pit and underground mining and process plant operations.
- Year 5 End of mine production and three months of process plant operations reclaiming and processing 84,000 t of stockpiled low grade ore.

The operating costs include all costs required to produce 1,000 t/d including the normal underground mine development and extension of principal underground mine ramps and raises. Not included are sustaining capital which, due to the short mine life, comprise only four main items: initial Kamila underground mine development and associated support infrastructure until the start of underground production, addition of a power generator and electrical substation to support the underground operation, extension of the haul road to Mercado, and closure costs. Sustaining capital is incorporated directly into the project cash flow model.

Life of-mine operating costs are shown in Table 18-8, and annual operating costs in Table 18-9.

Table 18-8
Life-of-Mine Operating Cost, US$ x 1000

	Labour	Expenses	Total Cost	US$/t Milled
Mine operations	1,857	36,699	38,556	21.52
Processing operations	4,823	29,979	34,803	19.43
Administration	3,927	7,512	11,439	6.39
Total	**10,608**	**74,190**	**84,798**	**47.34**

Table 18-9
Annual Operating Cost, (US$ x 1,000)

Year	Mining	Processing	Admin	Total	Open Pit US$/t Mined(1)	Underground US$/t Mined(2)	US$/t Milled	US$/oz Au Eq(3)	US$/oz Au incl. Ag Credit(4)
1	8,541	6,969	2,437	17,946	2.10	-	50.43	302	275
2	7,028	7,139	2,437	16,603	2.47	-	45.49	185	123
3	10,474	7,258	2,437	20,169	2.89	42.99	55.26	249	153
4	8,917	7,282	2,437	18,636	3.24	26.66	51.06	273	147
5	3,596	6,155	1,693	11,445	2.38	12.66	33.62	262	153
Total	**38,556**	**34,772**	**11,439**	**84,798**	**2.36**	**28.07**	**47.34**	**248**	**168**

Note: (1) Open tonnage mined - including ore + stockpile + waste
(2) Underground tonnage mined – ore only
(3) Gold US$500/oz, Silver US$8.50/oz
(4) Silver US$8.50/oz net of cost of silver sales

18.5 Environmental

18.5.1 Baseline Studies

Baseline data collection for the feasibility study commenced in January 2006 and for reporting purposes the baseline data reported was cut-off in September 2006.

Knight Piésold (KP) developed and supervised the baseline monitoring program. During 2006 monthly baseline monitoring (meteorology, water quality, hydrology, hydrogeology, air quality), seasonal (flora and fauna, limnology and ichthyology) and special studies (geomorphology, seismicity, soils, landscape), were conducted by KP, and extends those initiated by Intrepid prior to the feasibility report. Intrepid also commissioned archaeological studies which KP has incorporated into their report.

The KP baseline study concluded that:

- The region is classified as having a dry desert (BW) climate, with an arid humidity regime, characterized by high net evaporation rates. Consequently, precipitation that can originate runoffs from the waste dump and tailing deposit will be very low and high evaporation rates are also favourable for avoiding acid generating conditions.
- The BW climate categorization also supports a negative project water balance and minimizes potential hydrology impacts. No major impacts are expected on water quality and quantity since the process water will be reclaimed to operations, and mitigation measures to prevent infiltration of components related to the metallurgical process have been considered.
- ARD geochemical test work indicates tailings and practically 90% of the waste rock are not producers of acid rock drainage.
- Soils at the Project Site are generally considered to be of such low edaphological development that it is highly improbable (without the use of technically advanced irrigation procedures) they can be used for commercial agricultural or animal production. From a socioeconomic point of view, mining activity at the site is unlikely to compete with other potential activities for land and soil use, and may be the only productive alternative. Consequently the potential impact on soils, current land use and agriculture potential are regarded as low.
- Regionally, and despite the severe seismic classification of the Province, records demonstrate the zone in which the Casposo Project is located has a relatively low seismic density, with earthquakes of Ms ≤6 magnitude and foci (hypocentre) depths between 50 km and 100 km. No active, regional faults have been identified close to the Project Site Area. The potential presence of active, local faults close to the Project Site will be investigated in more detail as part of a site-specific geological hazards study.

- Only one flora or fauna endemic species was discovered in the Study area; Asthenes Steinbachi pale basketweaver, of the Furnaridae family, but this is not expected to be irreversibly, negatively affected by the mine operations.
- No unique species or microhabitats exist at the Project. Site; all the species and habitats are found on a regional scale.
- The Project is not located close to a water body. The Quebrada Vallecito and del Potrerito are the two main streams relatively near to deposit area. Rivers and streams, during the whole study period, have alkaline characteristics with average pH values between 7,48 and 8,18. Average concentrations of Dissolved Oxygen are favourable for the development of aquatic ecosystems, with values around 8 mg/l. Chemical analysis does not show concentrations of Cyanides, Hydrocarbons or Pesticides in the waterways, as well as no significant concentrations of total metals.
- Hydrogeological analysis indicates a production well system, installed in the aquifer of the middle section of the Quebrada Vallecito, can permanently deliver the small 12 L/s water supply requirement for the Project. Measurements indicate that gravels in the Project Area and below the Quebrada Vallecito have moderate hydraulic conductivities, but enough to allow surface water infiltration. Gravels adjacent to the Project and in the lower part of Quebrada Vallecito are not saturated. In this sector, the aquifer would be restricted to fractured rock media with depths of at least 30 to 40 m. Consequently the water resources in the area should not be significantly altered by the Project development.
- The open pits, WRD and TMF represent irreversible, though localized alterations to the site topography in the Mine area. Given the visual access restrictions of the site by virtue of its location in the geographic basin, and the Project's location some 17 km removed from National Route 419, views of the permanent Mine site development will be limited to the visual basins in which it is situated. The prominence of the WRD and TMF on the local landscape will also be attenuated by developing them adjacent to and abutting a natural slope to which they will also be roughly profiled during site closure work.
- The location and configuration of the Project facilities in this Study will not directly affect any recognized archaeological sites; mitigation measures will be provided for a few identified in the proximity.

18.5.2 Project Development Environmental Management Plan

A preliminary environmental management plan (EMP) has been developed to summarize the significant, adverse impacts that the development, operation and closure of the Project may have on the environment and to suggest measures that will mitigate these recognized, potential adverse impacts to acceptable levels. General recommendations for the management and care of the physical environment, biological environment and archaeological and cultural heritage have been made in the preliminary EMP. The effectiveness of the EMP will be judged by performance monitoring throughout the mine construction, operation and closure stages, and as far into the post-closure period as required to demonstrate satisfactorily that the EMP objectives have been satisfied and will remain so in the long term. The EMP covers:

- Geomorphology
- Water
- Air Quality and Noise
- Soils
- Biological Environment including Protected Species
- Archaeological and Cultural Heritage
- Process Plant
- Waste Rock Dump and Tailings Management Facility
- Performance Monitoring
- Safety and Health Plan
- Contingency Plan
- Dangerous Goods Transport Management Plan
- Community Relationships Plan
- Waste Management Program including Solid Domestic and Industrial Non-Dangerous Waste and Management of Hazardous Solid Waste.

18.5.3 Preliminary Closure Plan

The closure and reclamation plan developed for the Feasibility Study summarizes the following:

- The anticipated conditions of the principal site facilities and infrastructure at the end of the mine operation.

- The closure criteria or objectives as determined from the estimated post-closure environmental and land-use objectives.
- The strategies and time frame required to achieve these criteria.

The purposes of this plan are to lay out the closure measures and estimated closure costs as input to the Project Feasibility Study, and also to form part of the Environmental Impact Assessment to indicate the post-mining reclamation activities that will mitigate potential environmental impacts.

The closure and reclamation plan is considered preliminary since some elements of the plan, such as post-closure monitoring requirements and associated cost estimates, may require revision upon Project approval. Further, the specific details of the Mine Closure and Reclamation Plan will evolve as mining progresses, and so the plan will be updated periodically during the mine life. The final plan will be generated two years before mine closure.

In general terms, the closure plan promotes, where technically and economically feasible, the rehabilitation of disturbed areas as much as possible to their pre-operation state. Areas permanently altered by the mine operation will be closed in a manner acceptable for physical, environmental, land-use and safety considerations.

Areas to be rehabilitated to their pre-operation state include the process plant site and the internal haul roads. All machinery and equipment will be dismantled and removed for resale or scrap value. Concrete foundation elements will be demolished and the rubble disposed of in the WRD or in another acceptable, permitted facility. All mineral, hydrocarbon or otherwise contaminated soil at surface will be excavated and the material treated and/or disposed of according to applicable environmental regulations. Stormwater diversion ditches and ponds not required for the post-closure design will be backfilled. The general area will be backfilled and/or regraded to match surrounding topography. Residual features may include large cut or fill slopes for which their predicted, acceptable long-term geotechnical, geochemical and safety performance can be demonstrated.

Permanent landform changes brought about by the Project include the open pits, WRD and TMF. The conceptual closure plan for these structures may include (for the open pits) infilling and will include (for the WRD and TMF) regarding and cover construction. Closure plan objectives for these structures include ensuring long-term geotechnical and geochemical stability, resistance to surface erosion processes (precipitation runoff and wind), aesthetically acceptable appearance, compatibility with predicted long-term land use and reduction of safety hazards to an acceptable level. Direct precipitation on the WRD and TMF will be managed by a store and release cover design, which is believed to be the most suitable for the environmental and meteorological conditions at site.

Progressive closure of these waste material facilities should be carried out during the mine operation, to allow evaluation and refinement of the closure strategy and to reduce post-operation closure costs.

Openings to surface from the underground mine operation will be permanently sealed to discourage unauthorized entry. Unless high phreatic levels are detected during the detailed design investigation and operation stages, the underground seals will comprise rock particle and soil backfill.

The preliminary closure plan will be developed into the definitive site closure plan at least two years before the anticipated end of mining operations. Decommissioning and final closure construction is anticipated to require less than one year. The success of the closure plan will be determined by way of post-closure monitoring of geotechnical, geochemical, biological, environmental and socio-economic conditions to compare the closure design criteria with the short-term post-closure conditions. The results of the short-term post-closure monitoring program will determine the need for any additional site remediation work, the extent and duration of any further monitoring, and the predicted point at which no further activity or monitoring is required, i.e. when the site can be considered permanently closed.

The earthworks closure construction and subsequent, short-term post-closure monitoring are estimated to cost around US$0.9 M. This figure does not include the cost of dismantling and removing or disposing of any structures, installations and equipment other than the concrete foundation elements. Some major equipment will have a salvage value net of the dismantlement cost, while other equipment, such as tanks, vessels, pump boxes and piping are assumed to have a scrap value, net of their dismantlement and removal cost.

Environmental Controls

Environmental controls will be incorporated into the design, construction, operation and closure aspects of the process facilities. The controls relate principally to health and safety, environmental protection and water management. The main areas of environmental concerns are cyanide elimination, surface land disturbance and the storage and handling of reagents. The envisioned controls for design, construction and operation are summarized below:

- The mill facilities will be designed to meet Argentine regulatory requirements and World Bank standards for health and safety.
- The berm areas will be designed to contain the volume of 110% of the contents of the single largest tank, or a major spill from the circuit. Internal sumps and bermed areas will be constructed to control all water and process streams. Thus, all spills can be contained in the mill area.

- The crushing and stockpile facilities are designed with dust control and dust recovery systems to minimize particulate loss to the air.
- A retort system will be included to ensure the removal of any trace mercury from zinc precipitates. Condensed mercury will be stored in flasks and shipped off site for by product refining by others. It is assumed the mercury credit will cover the costs of its treatment off-site and this is reasonable. Trace amounts of mercury are stripped from the condenser off-gas stream by pulling it through canisters loaded with activated carbon. The retorts and condensers are accommodated in a ventilated refinery building.
- Off-gas handling will be designed into the dorè furnace to control gas emissions from the furnace.
- The chemical and reagent storage and make-up facilities will be contained in a separate enclosed and ventilated building with dry storage and spillage control systems.
- Process chemistry will be monitored, including cyanide destruction, to ensure ongoing optimal plant efficiency.
- Industrial hygiene monitoring will be conducted in the mill and in other areas to protect worker health.
- Chemical and reagent storage areas will be inspected regularly to ensure the integrity of the storage structures and secondary containment.
- All emergency equipment will be inspected regularly to ensure optimal function.

18.6 Financial Evaluation

The capital and operating costs generated for the development, operation, and closure of the Casposo Project have been evaluated using discounted cash flow analysis as a means of determining the financial aspects and economic indicators for the Project. The cash flow analyses have been based upon data developed by AMEC, KP and Intrepid. The tax information basis has been reviewed by R y J López Aragón, Intrepid's Argentine tax consultant.

The results of the economic analysis represents forward-looking information that are subject to a number of known and unknown risks, uncertainties and other factors that may cause actual results to differ materially from those presented here.

The results of the economic analysis represents forward-looking information as defined under Canadian securities law. The results depend on inputs that are subject to a number of known and unknown risks, uncertainties and other factors that may cause actual results to differ materially from those presented here. Factors that could cause such differences include, but are not limited to: changes in commodity prices, costs and supply of materials relevant to the mining industry, the actual extend of the mineral resources compared to those that were estimated, actual mining and metallurgical recoveries that may be achieved, technological change in the mining, processing and waste disposal, changes in government and changes in regulations affecting the ability to permit and operate a mining operation, forward-looking information in this analysis includes statements regarding future mining and mineral processing plans, rates and amounts of metal production, tax and royalty terms, smelter and refinery terms, the ability to finance the project, and metal price forecast.

The date of this projected financial analysis commences on 1 January 2008, when pre-production costs will initiate. The following year (1 January 2009) full scale mining, milling operations and gold and silver production are proposed to begin.

18.6.1 Financial Analysis Methodology

A discounted cash-flow analysis was used to review the Casposo Project. This method requires projecting annual cash inflows (or revenues), and then subtracting annual cash outflows such as operating costs, sustaining capital costs and taxes. The resulting net annual cash flows are discounted back to the date of financial analysis at a chosen discount rate, and totalled to determine the project's net present value. For discounting purposes, cash flow is accounted for at the end of each year.

An internal rate of return is calculated, equivalent to the rate of discount at which net present value equals zero. The payback period is stated in terms of the number of years from the production start date required to pay back the initial capital investment, excluding sunk costs, based on the undiscounted cash flow.

For the purposes of the cash flow analyses, all pre-production costs occur in Year 0 (which is taken to be the year prior to 1 January 2008). Full scale mining, milling operations and gold and silver production will begin in Year 1 (January 2008 to December 2009).

18.6.2 Financial Model Parameters

Precious Metal Prices

The base case for economic analysis of the Project has assumed long-term prices of US$500/oz for gold and US$8.50/oz of silver. These prices are significantly below the prevailing market price for both gold (US$639/oz) and silver (US$13.01/oz), at 1 January 2007. The equivalent base-dated two year and three year trailing-average gold prices are US$525/oz and US$486/oz respectively. Silver accounts for about 25% of total revenue in the Project Cash Flow. The base case utilizes higher metal prices than those used in either the resource estimation or reserve estimation and pit optimization (US$450/oz for gold and US$7/oz for silver).

Projecting gold and silver prices involves complex issues other than straight supply and demand, and as such, it was felt that selecting a gold price of US$500/oz for the base case analysis was a reasonable reference for the economic analysis in this study. A similar argument was considered for the silver price. The base case project IRR determined in this study is most sensitive to price.

Production Schedule

The production schedule assumes 90% of full production (at full indirect cost) in the first quarter, to allow for a ramp-up in production following commissioning, and 100% of production thereafter.

Capital Costs and Development Expenses

Project capital costs are detailed in the cash flow model by type, in order that the proper methods of tax and depreciation can be applied.

The Mining Investment Law in Argentina, Article 13, allows mining companies to choose an accelerated depreciation regime for their investments in new projects. This accelerated regime considers two categories of assets:

- Buildings, civil works, roads and infrastructure in general, with a depreciation schedule of 60% during the first year of production and 20% the second and third years.
- Equipment and vehicles, with a depreciation schedule of one third during the first three years of production.

One important consideration of the depreciation scheme allowed by the Mining Investment Law is that depreciation can only be deducted for tax purposes as long as there is taxable income. If there is no taxable income in any particular year, depreciation can be carried forward until the last year of the assets' life (in that year, remaining depreciation can be deducted completely even though it causes the company to have a fiscal loss).

Project non-depreciable expenses include the owner's costs (permitting, insurance and legal costs), EPCM contract costs, and working capital. Working capital will be recovered at the end of Project life.

Working capital assumes approximately two months of operating supplies costs.

Pre-production costs and expenses are detailed in the cash flow model by year. AMEC developed the Owners costs and EPCM component of the pre-production costs. Exploration and development costs and expenses were provided by Intrepid and AMEC has not verified these expenditures, or their classification. They are in reasonable agreement with the amounts disclosed in the Intrepid Annual Financial Reports that AMEC reviewed for the relevant periods for Casposo, and AMEC has accepted that the data can be used in the base case analysis. The capital costs have been documented in Section 18.3, specifically in Table 18-7.

Operating Costs

Operating costs are detailed in the financial model by year. Operating costs are documented in Section 18.4, specifically in Table 18-8 for the life-of-mine, and Table 18-9 for the annual operating costs.

Sustaining Capital

Due to the relatively short Project life no sustaining cost are envisaged for the mill or support facilities, and sustaining capital comprises only four mining items: initial Kamila underground mine development and associated support service infrastructure until the start of underground production, addition of a power generator and electrical substation to support the underground operation, and extension of the haul road to the Mercado pit. There is no significant development required prior to ore production at Mercado and so this cost was incorporated into the mining operating cost. The assumed sustaining capital costs are included in Table 18-10.

Table 18-10
Sustaining Costs, (US$ x 000)

Item	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Total
Power (for underground)		862.5				862.5
Kamila UG Mine Mobilization and Services		135.0	58.5			193.5
Kamila UG Mine Development *		1,096.4				1,096.4
Mercado Haul Road					35.0	35.0
Total		**2,093.9**	**58.5**		**35.0**	**2,187.4**

US$ 246,000 of this cost is power supplied to the underground mine by Intrepid during development.

Cost of Sales

The cost of sales used in the model are based upon terms as received in a budget quotation communication from Johnson Matthey (AMEC, 2007), and include melting, refining and treatment charges. The preliminary terms in the quotation may change before the Intrepid reaches an agreement with a buyer.

The base transportation and insurance cost will be US$12,500 (transport with two trucks with capacity of approx. 4.4 t) plus US$5.00/kg of dorè sold.

In addition, Intrepid will have the following deductions/charges:

- Melting charge: Intrepid will be paid 99.5% of the contained gold and silver.
- Refining charge: a deduction of US$0.65/oz of gold would be applied.
- Treatment charge: a deduction of US$0.22/oz of gold and silver will be applied.

Royalties, Duties and Taxes

The royalties and taxes that the Project is and will be subject to are discussed in Sections 4.2.7 and 4.2.8 respectively.

Mine Closure Costs

These mine closure cost estimate basis is detailed in Section 18.5.3. Mine closure costs have been estimated at US$890,000 and, for the cash flow analysis, have been assumed to occur in the year following the last year of the project.

Salvage Value

The capital cost estimate for the process facilities and infrastructure in this Study assumes the use of all new equipment. Typically equipment of this type will have a useful economic life cycle of about 20 years and given the 5 year mine life of the Project it is reasonable to assume most will have a salvage value net of the cost of dismantlement and removal on closure. AMEC estimated a salvage value of about US$4.5 M for the main capital equipment utilized in the Project, net of the cost of dismantlement, removal and taxes, a residual salvage value of about 30% of the main equipment.

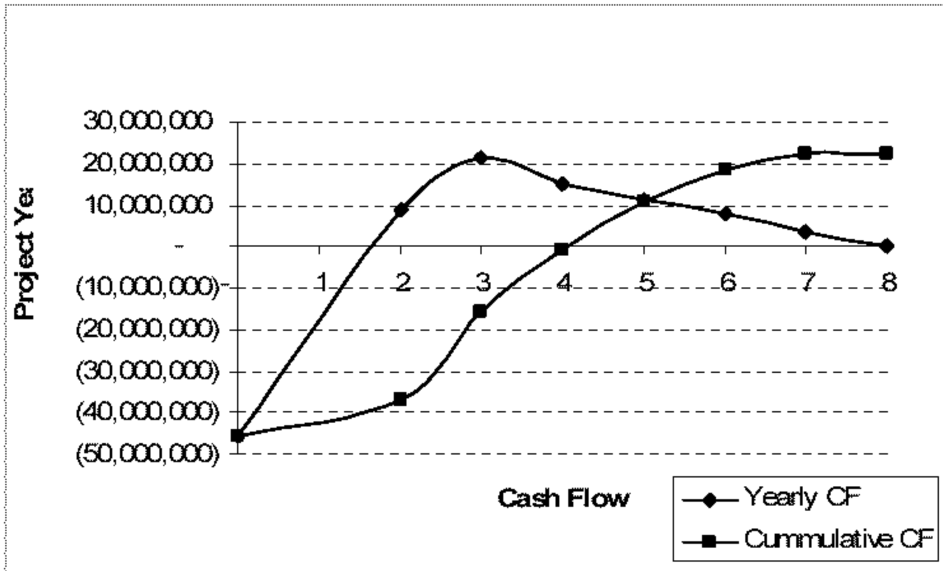
18.6.3 Base Case Economic Analyses Results

The base case for economic analysis of the Project has an assumed price of US$500/oz for gold and US$8.50/oz of silver.

As will be shown in the next section the Project IRR is most sensitive to revenue, or metal prices. There are a number of factors reasonably supporting the argument the Project may be able to take advantage of higher prices that have prevailed for the past three years. These include its relatively short five year project life cycle, and planned implementation to bring it into production relatively quickly, by April 2008, using a combination of open pit production initially and established processing technology. Prior to making a production decision, Intrepid may also elect to use financial options in its financing strategy to realize higher project precious metal prices than those used by AMEC in the Study base case economic analysis.

The cash flows generated for the Project base case are shown graphically in Figure 18-7. The time at which the cumulative cash flow line becomes positive represents the payback period for the Project. In this case, it is approximately three years from the beginning of production operations at the start of Year 1. Note that all cash flows are calculated at year end (at the end of Year 0 the cash flow is a negative US$37.2 M).

Figure 18-7
Annual and Cumulative Cash Flow US$ Projections (After Tax)
at Au US$500/oz and Ag US$8.50/oz



The base case economic indicators for the Project are shown in Table 18-11.

Table 18-11
Economic Indicators

Economic Indicator	After Tax
Internal Rate of Return	14.8%
Total Cash Flow, undiscounted	US$22.2 M
Project NPV, 6% discount[(1)]	US$10.836 M
Project NPV, 8% discount	US$7.865 M
Project NPV, 10% discount	US$5.219 M
Project Payback Period	3.1 years

(1) Net present value is at the start of Year 0 and assumes that all cash flow amounts occur at the end of each year of the Project. Gold price is US$ 500 per ounce, and silver price is $8.50 per ounce.

18.6.4 Sensitivity Analyses

Sensitivity analyses were performed to the base case net cash flow to estimate the sensitivity of the Project economics. To do this 30% positive and 20% negative variations were independently applied to each of the following parameters:

- Revenue (changes in gold and silver metal prices, recovery and grades).
- Capital cost of the Project (including working capital).

- Operating costs (not including working capital).

An increase in gold (or silver grades) and/or recovery will result in more gold (and silver) produced, which will slightly raise operating costs. This was accounted for in the sensitivity analysis for revenue, but the resulting effect is considered to be minor. The results of the sensitivity analyses are presented in Table 18-12 and Figure 18-8. The economic indicator chosen for sensitivity evaluation is the internal rate of return (IRR). This analysis demonstrates that the Project is, as expected, most sensitive to revenue, and the use of higher prices than the assumed base case price of US$500/oz for gold and US$8.50/oz of silver will significantly increase the IRR achieved. It also indicates that the Project is likely to stand up better to variation in development and operating costs.

Table 18-12
IRR Sensitivity (After Tax)

% Base Case	Revenue	Operating Cost	Capital Investment
80%	-8.9%	25.0%	24.8%
85%	-2.5%	22.5%	21.9%
90%	3.6%	20.0%	19.3%
95%	9.3%	17.4%	16.9%
100%	14.8%	14.8%	14.8%
105%	20.0%	12.1%	12.8%
110%	25.1%	9.4%	10.9%
115%	30.0%	6.5%	9.3%
120%	34.8%	3.7%	7.7%
125%	39.5%	0.7%	6.2%
130%	44.0%	-2.3%	4.9%

Figure 18-8
IRR Sensitivity (After Tax) at US$500/oz Au and US$8.50/oz Ag

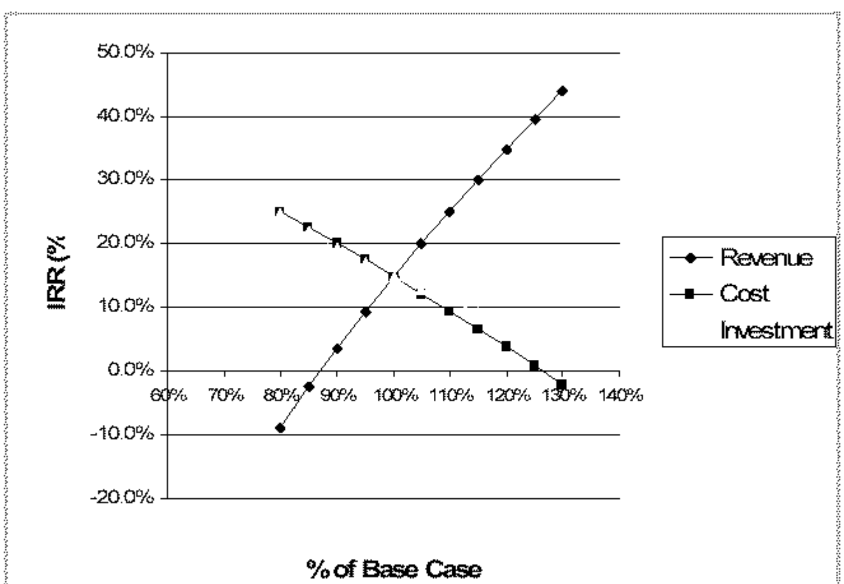
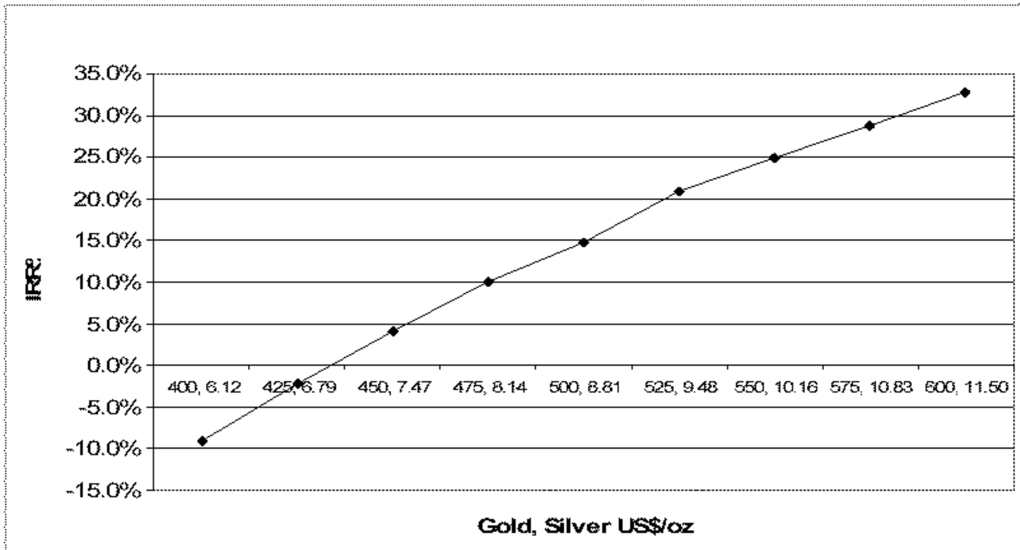


Figure 18-9
Price Sensitivity – IRR



Project Optimization

Two potential areas of potential project optimization opportunities during the mine life were noted:

- Defer open pit stripping. It is possible about one Mt of Year 1 stripping could be deferred from Year 1 to Year 2, deffering about US$2 M in stripping costs. As a result, some higher grade ore in Year 2 could also be mined in Year 1 increasing the average grades mined during Year 1. For the purpose of this analysis AMEC assumed the Year 1 and 2 grades will average out
- Defer underground mine development. It is possible that underground development could be deferred six months into Year 3. On this basis it was assumed the underground mine could be mined at a higher rate and half the Year 3 underground operating costs could be deferred to Year 4. Half the cost of the total underground mine development cost (in sustaining capital) was also deferred from Year 2 to Year 3. It was also assumed the grades in Years 3 and 4 could be maintained.

Overall the effect of these changes increased the IRR by about 0.8%.

Used Ball Mill and Power Generators

Budget quotations were obtained for a new ball mill (and motor) and power generators of about US$2.1 M and US$2.2 M respectively, and these equipment prices are included in the capital cost estimate and investment in this financial analysis. However, the lead times on delivering these items is too long to match the Project Schedule which assumes suitable used units will be procured. AMEC have done a preliminary survey to assess the availability of suitable used equipment and have identified some potential units that will be followed up during engineering. For the basis of this analysis and preliminary pricing obtained for available used ball mills and generators, it was assumed both these units could be procured, reconditioned and delivered for about US$0.8 M each, resulting in a total capital saving of about US$2.7 M. Overall, the effect of this reduction increased the IRR by about 2.5%. It is possible that other used equipment will also be procured, thereby further enhancing this benefit.

18.7 Project Development Timeline

The overall Project capital program will take about 15 months from the scheduled project approval date (coinciding with the issue of an environmental license certificate and finance approval) to complete the detailed engineering and construct the facilities to mechanical completion when pre-operations can start. A four month basic engineering phase precedes the start of detailed engineering and parallels the EIA approval process.

The Casposo Project will use competitive tendering and negotiation to obtain the best quality material, equipment, and contractors for a fair market value. Emphasis will be placed on using local Argentine (San Juan) labour, materials, and equipment wherever reasonable, provided it does not affect cost, schedule, or quality.

The key features and timeline for executing the Casposo Project are to:

- Submit the EIA study and start a four month basic engineering phase within a month of acceptance of the Feasibility Study.
- Optimize the Project open pit mine design and prepare, issue and award a mining contractor bid package.
- Secure a used ball mill and power generation equipment.
- Conduct detailed site geotechnical investigations.
- Complete basic engineering of process and facilities.
- Select an EPCM contractor

- Make a positive development decision (Project approval) within five months of EIA submission and following EIA approval issue an environmental certificate and finance approval.
- Award Project EPCM and start detailed engineering (and permitting process) immediately after a positive development decision.
  - Establish a comprehensive Project Implementation Plan and Procedures Manual.
  - Establish a contract plan and award principal contracts.
  - Establish a practical and effective project cost control system.
  - Establish an Environmental Protection Plan.
  - Issue project documents and size construction materials in metric units. Use the English language during the engineering and procurement stage with Spanish translations as needed.
  - Complete procurement of equipment and material orders (initial focus on site establishment requirements).
  - Complete project permitting process within three months of start of detailed engineering (prior to site establishment).
  - Complete detailed engineering within nine months.
- Construction site establishment and completion of permitting process within three months of issue of environmental certificate (and start of detailed engineering).
  - Comply with the laws and regulations of the Government of Argentina.
  - Maintain a high standard of workplace safety to minimise incidents and accidents.
  - Observe the requirements of Intrepid's Environmental Protection Plan.
  - Minimize disruption to local communities.
  - Complete mine development and construction and commission within 12 months of site establishment and 15 months from start of detailed engineering.
  - Recruit and train operations staff within three months of start-up.
  - Incorporate a three-month ramp up during start up to reach maximum production levels. Production is planned to be 90% of maximum in the first quarter.

19.0 OTHER RELEVANT DATA AND INFORMATION

There are no other data or information available that are relevant to this section of the report.

20.0 INTERPRETATION AND CONCLUSIONS

The Casposo Project is located in northwestern Argentina in the San Juan Province, department of Calingasta, and is approximately 150 kilometres west of the city of San Juan. The Project Site is located in a wide valley at the base of rugged terrain, characterized by low mountains with steep slopes, and ravines associated with dry drainage systems. The area is arid and contains sparse vegetative cover of bushes and small trees. The soil is sandy to gravelly with no continuous soil cover.

Other than a gravel access road and a weather station no other infrastructure currently exists on site.

The land tenure holdings for the Casposo Project cover a combined area of 35 km^2, and comprise a Mining Lease, five exploration Cateos and one Manifestación.

The Casposo gold–silver mineralization is dated at 280± 0.8Ma (K/Ar) and occurs in both the rhyolite breccias and underlying andesite, where it is associated with banded quartz–chalcedony veins. Mineralization is typical of low-to intermediate-sulphidation epithermal systems. It is characterised by low Fe sphalerite+chalcopyrite+sulfosalts+pyrite, and is typically silver and base-metal rich. The opaque mineral assemblage includes translucent (low Fe) sphalerite, chalcopyrite, pyrite, lead sulphide and selenide, several sulphosalt minerals, native metal alloys that range from silver-rich electrum to native silver, silver selenide and sulphide and rare native arsenic.

Several additional targets and showings of similar style mineralization have been identified by Intrepid on the project. Subsequent to additional exploration with positive results, these targets may add to the existing mineral resource and/or reserve base for the project and increase the mine life. A further fifteen holes were drilled in the Kamila deposit after the resource estimate deadline, targeting strike extensions to the defined mineralized resources at Kamila. These new mineralized intercepts will be followed-up during a drill program scheduled to commence in mid-2007.

In AMEC's opinion the geology of the Casposo Project and the controls on mineralization are well understood.

Exploration has been ongoing on the Project since 1993. Battle Mountain Gold (1993-1999) completed 8,626 m of drilling in 46 holes on the Kamila and Mercado zones. Intrepid Mines Limited has held the property since 2003, and have, to date, have undertaken 165 holes (153 diamond holes and 12 RC holes) for 27,274 m.

The database used to estimate the Mineral Resources at Casposo has been verified by AMEC and consists of samples and geological information from 130 diamond drill holes, 70 trenches and 5 pits. The database close-off date was 10 October 2006. A total of 15 diamond holes on the Kamila and Mercado zones were not included in the estimate as assay results had not been received by that date. The remaining 10 drill holes were sited into prospects away from the main Kamila–Mercado zone.

AMEC considers the database to be sufficiently robust for the Indicated and Inferred Mineral Resource estimation classifications that were assigned during the Feasibility Study. However, AMEC has identified a number of issues with the database that could be improved prior to advancing to a higher confidence level (Measured Resources) for the estimate. These include: documentation of the trench and pitting programs; validity of using long channel sample lengths in resource estimation; drill hole recoveries, including grade versus recovery studies; documentation of collar survey methodologies; review of the applicability of using drill holes that have acid-etch downhole surveys in resource estimation; the drilling QA/QC review that identified analytical biases close to the acceptable limits, and use of silver standards that are outside of the average grades encountered to date within the deposits and prospects of the Project; and QA/QC review of the surface (trench and pit) sampling.

Bulk density samples consisted of half core pieces, 10 cm to 15 cm long, which were taken from quartz veins and silicified breccias (47), andesites (28) and rhyolites (19). Bulk density of quartz veins ranges within a relatively narrow dispersion interval, from 2.28 t/m^3 to 2.72 t/m^3, and similar trends can be observed for andesites and rhyolites.

The geological interpretation was completed by Intrepid based on lithological, mineralogical and alteration features logged in drill core, trenches and pits.

Intrepid defined five domain zones to represent the different veins systems at Kamila and Mercado. Domains were defined based on lithology, structure and grade boundaries. At the Kamila zone, domains are referred to as Aztec vein, B vein, Inca vein and High Grade. The High Grade (HG) domain was determined using drill hole intercepts that are above 10.00 g/t Au equivalent within the other domains to separate the high-grade population from the lower grades. At Mercado only one domain is defined.

Intrepid used only one set of east–west oriented, 25 m spaced, vertical sections to interpret and model Aztec, B and Inca veins. High-grade capping was applied to gold and silver grades for all domains. Gold and silver grades are estimated using an inverse distance cubed (ID3) methodology. The block model used blocks of 4 x 4 x 6 m dimensions.

The Intrepid Kamila Zone Mineral Resource model was developed using industry-accepted geostatistical methods. AMEC validated the model estimates and found them to reasonably estimate grade and tonnage. The resource estimate is summarized at a 1.4 g/t AuEq cut-off value, calculated by reference to the gold and silver ratios (77:1), gold price of US$450/oz, process and G&A cost allocations (US$15.70/t and US$3.35/t respectively, and assumed process recoveries (94% for Au, and 80% for Ag).

The resource estimate AuEq equation does not take mining costs or dilution factors into account. Although this cut-off is not the mining breakeven value, since costs vary depending on the mining method used, AMEC considers this value acceptable for reporting the mineral resources.

The mineralization at Casposo was classified into Indicated, and Inferred Mineral Resources, using logic consistent with the CIM Definition Standards referred to in NI 43 101, and is summarized in Table 20-1.

Table 20-1
Casposo Mineral Resources
(Effective Date 10 October 2006)

		Tonnes (Mt)	Au g/t	Ag g/t	Au oz	Ag oz	Au Eq oz
	Kamila	1.960	4.76	120.0	300,130	7,563,468	398,357
Indicated	Mercado	0.225	1.82	81.9	13,148	591,898	20,835
	Total Indicated	**2.185**	**4.46**	**116.1**	**313,278**	**8,155,366**	**419,192**
	Kamila	0.131	2.14	86.1	9,000	362,393	13,706
Inferred	Mercado	0.007	1.66	28.0	354	5,957	431
	Total Inferred	**0.138**	**2.12**	**83.3**	**9.354**	**368,350**	**14,137**

Mineral Reserves were defined for the project at an effective date of 30 March 2007, based on separate underground and open pit operations. The Probable Reserves for the open pit were calculated separately for the Kamila Main, Kamila SE and Mercado open pits. Reserves incorporate considerations for gold and silver prices (US$450 and US$7.00 respectively), recovery, mining, processing and G&A costs, and dilution for open pit reserves and stockpiles. Open pit reserves are shown in Table 20-2. The overall open pit Probable Reserves are estimated to be 1.35 Mt at a grade of 5.14 g/t Au and 89 g/t Ag, using a AuEq cut-off grade of 1.98 g/t AuEq.

The Kamila open pit consists of a large pit (Kamila Main Pit) and a small satellite pit located 100 m to the SE (Kamila SE pit). The underground mine will be accessed by a decline and exploited using a longhole open stoping mining method with sublevel retreat. The underground mine has been designed with a temporary crown pillar below the open pit floor and with the ramp portal and ventilation raise collar located beyond the perimeter of the open pit such that both the open pit and the underground mines can be operated concurrently.

Table 20-2
Summary Open Pit Reserves at a 1.98 g/t AuEq Cut-off; Stockpiles at a 1.70 g/t AuEq Cut-off (Effective Date 30 March, 2007)

Zone	Probable Reserve						
	Tonnage (t)	AU-ID3 Grade (g/t)	AG-ID3 Grade (g/t)	AUEQ-ID3 Grade (g/t)	Contained Metal Au (oz)	Ag (oz)	AuEq (oz)
Ore							
Kamila Main Pit	1,103,513	5.66	94.7	6.85	200,958	3,359,937	242,957
Kamila SE Pit	84,396	4.96	65.7	5.79	13,469	178,386	15,699
Mercado Pit	80,270	2.19	93.5	3.36	5,647	241,173	8,661
Total	**1,268,179**	**5.4**	**92.7**	**6.56**	**220,074**	**3,779,496**	**267,317**
Stockpile							
Kamila Main Pit	74,375	1.25	24.4	1.55	2,983	58,403	3,713
Kamila SE Pit	1,082	1.27	24.5	1.58	44	854	55
Mercado Pit	8,858	1.23	29.2	1.6	350	8,320	454
Total	**84,315**	**1.25**	**24.9**	**1.56**	**3,377**	**67,577**	**4,222**
Total Ore + Stockpile							
Kamila Main Pit	1,177,888	5.39	90.3	6.51	203,941	3,418,339	246,670
Kamila SE Pit	85,478	4.92	65.2	5.73	13,513	179,240	15,754
Mercado Pit	89,128	2.09	87.1	3.18	5,997	249,493	9,115
Total	**1,352,494**	**5.14**	**88.5**	**6.24**	**223,451**	**3,847,073**	**271,539**

The production rate for the mine will be 365,000 t/a for a total five-year mine life, with all production for the first two years coming from the Kamila open pits and underground ore production starting in Year 3. The Mercado open pit will be mined in the final year of the mine life due to its lower grade and since it is located further from the processing plant than the Kamila deposit. Mining will be undertaken using a contractor.

Underground Probable Reserves have only been estimated at the Kamila deposit, and are shown in Table 20-3. Reserves also incorporate considerations for gold and silver prices, recovery, mining, processing and G&A costs, and dilution for underground and remnant areas. The underground Probable Reserves, including remnant material, are estimated to be 0.439 Mt at a grade of 3.31 g/t Au, 192 g/t Ag at a cut-off grade of 3.5 g/t AuEq.

The process flowsheet selected to process Casposo ore will use conventional primary jaw and secondary cone crushing, ball milling, gravity concentration for coarse gold and silver, cyanide leach, Counter Current Decantation (CCD) and washing and dewatering of tails by belt filtration. Gold and silver will be recovered by standard Merrill-Crowe (MC) zinc precipitation and smelted to produce dorè bars. The leached tailings, following filtration to recover precious metals, will be washed and rinsed on the same belt filter to remove cyanide. The cyanide wash solution will be collected for the destruction of cyanide using the conventional SO_2/air process. The detoxified solution is recycled to the belt filter as wash solution.

Table 20-3
Underground Reserves Summary at a 3.50 g/t AuEq Cut-off
(Effective Date, 30 March 2007)

Zone	Reserve						
	Tonnage	AU-ID3	AG-ID3	AUEQ-ID3	Contained Metal		
	(t)	Grade (g/t)	Grade (g/t)	Grade (g/t)	Au (oz)	Ag (oz)	AuEq (oz)
Below Pit							
2380 Sublevel	103,649	3.51	167.1	5.59	11,680	556,685	18,639
2370 Sublevel	25,210	5.56	386.1	10.38	4,503	312,957	8,415
2360 Sublevel	83,421	3.55	161.8	5.57	9,515	434,010	14,941
2350 Sublevel	33,019	3.12	294.5	6.80	3,308	312,636	7,216
2340 Sublevel	38,769	4.41	179.1	6.65	5,493	223,256	8,284
2320 Sublevel	27,758	2.23	157.4	4.19	1,986	140,468	3,742
2315 Sublevel	30,359	2.98	228.9	5.84	2,905	223,438	5,698
2300 Sublevel	25,102	1.54	178.2	3.77	1,245	143,844	3,043
2280 Sublevel	20,215	1.15	213.0	3.81	748	138,449	2,479
2258 Sublevel	17,650	0.91	171.8	3.06	519	97,466	1,737
Subtotal	**405,151**	**3.22**	**198.3**	**5.70**	**41,903**	**2,583,211**	**74,193**
Remnants							
2444 Remnant A	8,487	7.52	22.0	7.79	2,051	5,998	2,126
2426 Remnant D	3,196	4.13	56.9	4.84	425	5,851	498
2402 CP Remnant N	10,917	2.63	175.0	4.81	921	61,409	1,689
2402 CP Remnant S	11,040	3.77	135.7	5.47	1,338	48,162	1,940
Subtotal	**33,641**	**4.38**	**112.3**	**5.78**	**4,736**	**121,420**	**6,253**
Total	**438,792**	**3.31**	**191.7**	**5.70**	**46,638**	**2,704,631**	**80,446**

This will minimize the fresh water requirements for the process. Filtered tailings will be trucked to a lined tailings management facility and stacked in compacted lifts.

The crushing plant and mill are designed to operate 365 days a year on two and three shifts (of eight-hours) per day respectively, and process 365,000 t/a of ore. The nominal mill flowsheet throughput of 46.3 t/h of ore is based on processing 1,000 t/d and an assumed plant availability of 90%. For the average life of mine head grades the overall gold and silver recoveries are projected to be 93.7% and 80.6% respectively. Average production is estimated to be about 50,478 oz of gold and 1.1 Moz of silver or 68,483 oz of gold equivalent annually.

The Casposo Mine Project will generate an estimated 10 Mt of waste rock and approximately 1.8 Mt of tailings over its planned five-year operating life. About 2 Mt of the mine waste rock is currently planned for use in road construction, in-pit disposal and underground backfill. On this basis the waste rock dump is designed for 8 Mt. The waste rock dump will be located just southeast of the Kamila Open Pit, and will be developed in a conventional manner, with haul trucks transporting and end-dumping the waste rock and dozers spreading and configuring the material on the working platform.

The Project will require the development of infrastructure to operate such as haul and access roads around the site facilities, the open pit and underground mines, a process plant, a waste rock dump, a tailings dump facility, administration and services (including water and power supply and communications). The current gravel access road to the site will be maintained and upgraded as required for mine transport.

Environmentally no material impact issues have been identified in the baseline studies for the proposed Project development.

AMEC have estimated the capital cost to build the facilities described in this Study to first gold pour to be US$45.5 M, including working capital and 12% contingency. Not included are estimated sustaining costs of US$2.2 M, which are included directly in the Project cash flow. The life of mine Total Operating Costs, including mining, processing, and general and administration, are projected to be US$47.34/t of ore milled, or US$248/oz of gold equivalent and US$168/oz of gold net of silver credits.

The results of the economic analysis represents forward-looking information that are subject to a number of known and unknown risks, uncertainties and other factors that may cause actual results to differ materially from those presented here.

Base gold and silver prices used in the financial analysis were different from those used to estimate reserves; the base case gold and silver prices being US$500/oz and US$8.50/oz respectively. Silver accounts for about 25% of total revenue in the Project cash flow. Results of the base case financial analysis indicate that the Project has a potential after-tax internal rate of return of 15% and net present value of US$10.8 M, at a discount rate of 6%. The base case scenario has a projected payback period of approximately three years, and is shown in Table 20-4. The financial analysis indicates the Project is more sensitive to changes in metal price and grade than either capital or operating costs and is slightly more sensitive to operating costs than capital costs. The IRR rises to 33% if prices of US$600/oz for gold and US$11.50/oz for silver are used.

Table 20-4
Economic Indicators

Economic Indicator	After Tax
Internal Rate of Return	14.8%
Total Cash Flow, undiscounted	US$22.2 M
Project NPV, 6% discount(1)	US$10.836 M
Project NPV, 8% discount	US$7.865 M
Project NPV, 10% discount	US$5.219 M
Project Payback Period	3.1 years

(1) Net present value is at the start of Year 0 and assumes that all cash flow amounts occur at the end of each year of the project.

21.0 RECOMMENDATIONS

Opportunities and recommendations are summarized below.

21.1 Database

The database issues identified in Section 20 should be addressed so that the database is sufficiently free from error so as to support higher-confidence categories of resource estimate (Measured Resources).

21.2 Exploration and Mineral Resources

It is recommended search ellipses for grade estimation of Mercado are reviewed in future resource model updates.

More drilling within the resource model limits could possibly reveal more continuous mineralized zones and therefore additional resources.

Some continuous high-grade intervals (AuEq >10.00 g/t) were found outside of the interpreted boundaries of the High Grade domain. This could cause some dilution since high grades are not confined and adjacent low-grade intervals are used for estimating grades at surrounding blocks. AMEC points to this problem as having locally the potential for upside in grade estimates.

Geological interpretations show that a zone of gold-bearing stockwork veins occurs above 2400 m elevation, which have not been included in the resource because of the difficulty to interpret and correlate these veins. During open pit mining it may be possible to recover a significant quantity of this material based on improved confidence from pit mapping and sampling.

Continue the Project exploration program to identify additional mineralization that can be drilled out to provide supplemental mill feed to the planned Casposo plant.

Future exploration programs should incorporate CIM best practice guidelines and a thorough QA/QC program.

21.3 Mining

Potential stockpile rock may become economically extractable at higher metal prices; the mine plan includes a provision for stockpiling of low-grade rock below the current low grade stockpile cut-off (1.55 g/t AuEq). A cut-off grade of 1.30 g/t AuEq was established for this potential stockpile based on metal prices of Au US$550/oz and Ag US$9.00/oz; slightly higher than the base case pit design metal prices. The cost of mining and stockpiling of about 130 kt of mineralization at a grade of 1.32 g/t AuEq has been included in the mine plan. AMEC recommends that Intrepid separate the rock containing between 1 g/t AuEq and 1.3 g/t AuEq into a third stockpile in the event of even higher gold prices at the end of the mine life.

There is potential for increased quantities of ore and low grade stockpile material over those reported in the Feasibility Study. If metal prices at time of mining are higher than those assumed in the Feasibility Study (Au US$450/oz and Ag US$7.00/oz) then the cut-off grade for these two categories of rock will drop. This is significant for this Project since there is a substantial amount of lower-grade mineralization below the current cut off grades. The result would be an increased Reserve.

The selection of the open pit underground boundary for the Kamila Main deposit was derived using parameters developed in early stage estimates and trade-off studies. It is recommended this interface be reviewed using the final pit parameters prior to initiating the detailed pit design.

The cut-off grades used for the reserve estimation in the Feasibility Study were based on assumed gold and silver base metal prices of US$450/oz and US$7/oz respectively. During the finalization of the Feasibility Study prevailing metal prices increased significantly so the base case gold and silver metal prices used for the economic evaluation were raised to US$500/oz and US$8.50/oz. Based on AMEC´s analysis, at these higher prices there is potential for an increase in the reserves and it is recommended that during detailed design the reserve estimate be reviewed and updated as required

Optimization of the open pit mining schedule could potentially result in the deferral of about one Mt of waste stripping from Year 1 to Year 2 resulting in a deferral of about US$2 M in stripping costs. Additionally, there is the potential to advance the mining of some higher grade ore from Year 2 to Year 1. These opportunities should be evaluated during detail mine design.

Optimization of the underground schedule could potentially result in the deferral of the underground development by six months into Year 3. On this basis the underground mine could be mined at a higher rate and half the Year 3 underground operating costs could be deferred to Year 4.

Open Pit - the estimate of 1.0 m dilution on the grade contact was based on mining limits defined by the block model grade interpretation supplemented with data from sampling of blastholes. Since the mining boundary is defined by a grade cut-off and not geology a comprehensive dilution control program should be instituted during operations comprising good blasthole sampling practices, quick turnaround on assays, detailed geological mapping, well planned ore blasting designs, detailed mark-up of ore blasts by the grade control geologists, and good digging practice by the excavator operator. Failure to implement these practices could result in higher dilution than forecast.

Underground - some sloughing underground is inevitable and regular grade control by the geologist and at the face sorting of waste from ore will be important to minimize dilution from this source. Failure to do this could result in higher underground dilution than forecast.

Pit wall failure - a final pit wall slope of 54° was used in the pit design based on the recommendations of the geotechnical studies. However, there is the risk of a pit wall failure if currently unidentified local weak rock conditions should occur. This risk is greatest on the one high wall of the pit (west wall) which will have ultimate vertical height of 150 m. A pit wall failure could result in production delays, increased stripping costs, damage or loss of equipment and in the worst case injuries or fatalities. This risk can be largely mitigated by regular ongoing geotechnical mapping, data collection and analysis by trained personnel.

Underground stope wall failure - an open stoping mining method has been selected with stope spans of 120 m long by 100 m high. Stope wall failure risks can be mitigated by implementing regular ongoing geotechnical mapping, data collection and analysis by trained personnel.

Underground crown pillar - a detailed geotechnical evaluation will be required prior to the final detailed design of the mining plan for this pillar.

Mining contractor - during operation, close supervision and monitoring of the contractor's work will be required including regular and open discussions on progress and goal setting.

Care should be exercised during the initial pit development with respect to potential rockfall hazards to personnel and equipment on the Kamila pit outcrop slopes.

21.4 Process

Test work indicates no significant difference in gold and silver recovery with or without the use of gravity recovery. However this circuit has been included in the design as AMEC's experience has shown that mineralization in intermediate sulphidation deposits that was not demonstrated either statistically or economically in bench scale test work, does provide a significant source of additional gold and silver credits when a gravity circuit is retrofitted. This circuit could potentially be deferred for future installation.

21.5 Infrastructure

Further pumping tests are recommended to confirm the fresh water source and the detailed design of the well system.

Regional "Zonda" winds are year-round strong gusts of dry hot air. Potential may exist to supplement diesel power generation with wind power for some ancillary facilities. This could be investigated during engineering.

No power factor correction equipment has been considered. This should be reviewed for potential operating cost savings during engineering when the final power factor is established at the main 6.6 kV distribution.

21.6 Capital

The use of all new equipment is assumed in the capital estimate. The ball mill and power generator plant account for about US$2 M and US$4 M of the US$45.7 M project capital respectively. To expedite the Project Schedule it is currently assumed a used ball mill and power plant will be purchased. AMEC has made preliminary enquiries on the availability of used ball mills and generators suitable for the Project. Specific prospects have been identified and will be followed up early in 2007. A total potential capital reduction of about US$3 M is indicated. It is possible other used equipment will also be procured, thereby further enhancing this benefit.

21.7 Operating Cost

Freight rates are regarded as high and opportunities should be investigated to optimize this.

There is some risk of an escalation in wages and salaries with competition for labour and materials developing with other mine development decisions recently announced in San Juan. It is recommended that these costs be monitored and if necessary, updated into the financial model

One mining operation in Argentina is currently exempt from paying ITC tax on fuel and this may be a cost optimization negotiation opportunity.

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22.2 Abbreviations and Units of Measure

80 Percent Passing	p80
Abrasion Index	AI
Acid Base Accounting	ABA
Acid Mine Drainage	AMD
Agricultural Establishment Unit	EAP
American Association of State Highway and Transportation	AASHTO
American Society for Testing and Materials	ASTM
Ampere	A
Analysis of Health Situation	AHS
Annum (year)	a
Argentine Peso	$
Average	AV
Bench Face Angle	BFA
Best Value	BV
Bond Work Index	BWI
Break Horse Power	BHP
Canadian Securities Administrators	CSA
Carbon Dioxide	CO_2
Centro Regional de Sismología para América del Sur	CERESIS

Centimetre	cm
Check Samples	CS
Chile – Argentina Geophysical Experiment	CHARGE
Coefficient of Determination	R^2
Comisión Evaluadora Multidisciplinaria Ambiental Minera	CEMAM
Confidence Interval	CI
Counter Current Decantation	CCD
Cubic centimetre	cm^3
Cubic feet per minute	cfm
Cubic metre	m^3
Curve Number (Hydrogeological)	NC
Day	d
Days per week	d/wk
Days per year (annum)	d/a
Dead weight tonnes	DWT
Degree	°
Deterministic Seismic Hazards Analysis	DSHA
Diameter	ø
Dirección de Planeamiento y Desarrollo Urbano	DPDU
Dry metric tonne	dmt
Dump Stability Class	DSC
Dump Stability Rating	DSR
Elevation (metres)	el
Environmental Impact Assessment	EIA
Environmental Impact Declaration	EID
Engineering Procurement Construction Management	EPCM
Escribania de Minas	EDM
Factor of Safety	FS
Foot	ft
Gallon	gal
Gemcom Software (Geological Modelling Software)	GEMS
Gram	g
Global Positioning Satellite/System	GPS
Gold	Au
Gold Equivalent	AuEq
Grams per tonne	g/t
Greater than	>
Hectare (10,000 m^2)	ha
High Density Polyethylene	HDPE
Horsepower	hp
Hour	h
Hours per day	h/d
Hours per week	h/wk

Hours per year	h/a
Inductively Coupled Plasma (Chemical Analysis Instrument)	ICP
Infant Mortality Rate	IMR
Inputs and Outputs	I/O
Instituto Nacional de Estadística	INDEC
Instituto Nacional de Tecnología Agropecuaria	INTA
Instituto Nacional de Prevención Sísmica	INPRES
Inter Ramp Angle	IRA
International Industrial Uniform Codes	IIUC
Intensive Cyanidation Unit	ICU
Inverse Distance Cubed (Grade Estimation Method)	ID3
Kilo (thousand)	k
Kilogram	kg
Kilograms per cubic metre	kg/m^3
Kilograms per hour	kg/h
Kilograms per square metre	kg/m^2
Kilogram per year	kg/a
Kilometre	km
Kilometres per hour	km/h
Kilopascal	kPa
Kilovolt	kV
Kilovolt-ampere	kVA
Kilovolts	kV
Kilowatt	kW
Kilowatt hour	kWh
Kilowatt hours per tonne (metric ton)	kWh/t
Kilowatt hours per year	kWh/a
Less than	<
Life of Mine	LOM
Linear Low Density Polyethylene	LLDPE
Litre	L
Litres per minute	L/m
Mass Spectrometer (Analysis Instrument)	MS
Mass Submerged in Water	Mw
Maximum Credible Earthquake	MCE
Maximum Precipitation in 24 Hours	P_{24}
Maximum Design Earthquake	MDE
Measure of Acidity or Alkalinity of a Solution	pH
Megapascal	MPa
Megavolt-ampere	MVA
Megawatt	MW
Mercury	Hg
Merrill Crowe	MC

Metre	m
Metres above sea level	masl
Metres per minute	m/min
Metres per second	m/s
Micrometre (micron) 10^{-6}m	µm
Milliamperes	mA
Milligram	mg
Milligrams per litre	mg/L
Millilitre	mL
Millimetre	mm
Million	M
Million Dollars (US)	US$M
Millions of Ounces	Moz
Million Pascals	MPa
Million tonnes	Mt
Mine Environmental Authority	AAM
Minute (plane angle)	'
Minute (time)	min
Month	mo
Motor Control Centre	MCC
National Agricultural Census	CAN
National Earthquake Information Center	NEIC
Nearest Neighbour (Resource Estimation Technique)	NN
Neutralization Potential	NP
Net Neutralization Potential	NNP
Net Present Value	NPV
Norwegian Geotechnical Institute	NGI
Neutralization Potential Ratio	NPR
Ounce	oz
Ounce per year	oz/a
Operating Basis Earthquake	OBE
Optical Emission Spectroscopy (Analysis Instrument)	OES
Overall Bias	OABias
Particle Material (10µg) Concentration expressed as µg/m^3	PM10
Parts per billion	ppb
Parts per million	ppm
Pascal (newtons per square metre)	Pa
Pascals per second	Pa/s
Percent	%
Peak Horizontal Ground Acceleration	PHGA
Point Load Strength	PLS
Precipitation (Maximum in 24 hour period)	P_{24}
Probable Maximum Precipitation	PMP

Potassium/Argon (Geological Dating Industry Standard)	K/Ar
Pound(s)	lb
Probabilistic Seismic Hazards Analysis	PSHA
Process Control System	PCS
Programmable Logic Controller	PLC
Pounds per square inch	psi
Polyvinyl Chloride	PVC
Quality Assurance and Control	QA/QC
Reduction-to-Major-Axis	RMA
Registro Nacional de Precursores Químicos	SEDRUNAR
Regression Line Slope	RLS
Resistance Temperature Detectors	RTD
Revenue Factor	RF
Reverse Circulation	RC
Rock Mass Rating	RMR
Rock Quality Designator	RQD
Run of Mine	ROM
Scanning Electron Microscopy	SEM
Second (plane angle)	"
Secondary Education Schools for Adults	CENS
Second (time)	s
Seismic Magnitude	M_s
Silver	Ag
Slope Stability Analyses Software (Geo-Slope, 2006)	SLOPE/W
Small and Medium Sized Companies	PyMES
Sodium Cyanide	NaCN
Software for analysis of orientation based geological data	DIPS
Standard Deviation	SD
Standard Samples	SS
State Mining Secretary	SMS
Specific gravity	SG
Synthetic Precipitation Leaching Procedure	SPLP
Square centimetre	cm^2
Square foot	ft^2
Square inch	in^2
Square kilometre	km^2
Square metre	m^2
Sulphur Dioxide	SO_2
Tailings Management Facility	TMF
Thousand tonnes	kt
Tonne (1,000 kg)	t
Tonnes per day	t/d
Tonnes per hour	t/h

Tonnes per Metre	t/m
Tunnelling Index	Q
Tonnes per year	t/a
Unconfined Compressive Strength	UCS
Underground	UG
Unified Soil Classification System	USCS
United States Environmental Protection Agency	USEPA
US Dollar	US$
Uniaxial Compressive Strength	UCS
Unified Soil Classification System	USCS
Universal Transverse Mercator (Co-ordinate System)	UTM
United Stages Geological Survey	USGS
Volt	V
Waste Rock Dump	WRD
Week	wk
Weight	WT
Wireless Digital Communications Protocol	HART
X-Ray Diffraction	XRD
Wet metric tonne	wmt
Year (annum)	a

22.3 Glossary

The glossary definitions are sourced from the glossary maintained by the Canadian "Northern Miner" publication group (www.northernminer.com/Tools/glossary.asp).

Aeromagnetic survey: A geophysical survey using a magnetometer on board or towed behind an aircraft.

Alluvium: Relatively recent deposits of sedimentary material laid down in river beds, flood plains, lakes, or at the base of mountain slopes. (adj. alluvial)

AMD: Acid mine drainage, characterized by low pH, high sulphate, and high iron and other metal species.

Amortization: The gradual and systematic writing-off of a balance in an account over an appropriate period.

Anfo: A free running explosive used in mine blasting made of 94% prilled aluminium nitrate and 6% No. 3 fuel oil.

Anomaly: Any departure from the norm which may indicate the presence of mineralization in underlying bedrock.

ARD: Acid Rock Drainage. See AMD

Assay: To determine the amount of metal contained in an ore.

Channel sample: A sample composed of pieces of vein or mineral deposit that have been cut out of a small trench or channel, usually about 10 cm wide and 2 cm deep

Comminution: Crushing and/or grinding of ore by impact and abrasion. Usually, the word "crushing" is used for dry methods and "grinding" for wet methods. Also, "crushing" usually denotes reducing the size of coarse rock while "grinding" usually refers to the reduction of the fine sizes.

Concentrate: The concentrate is the valuable product from mineral processing, as opposed to the tailing, which contains the waste minerals. The concentrate represents a smaller volume than the original ore.

Cordillera: The continuous chain of mountain ranges on the western margins of North and South America.

Cut and Fill Stoping: If it is undesirable to leave broken ore in the stope during mining operations (as in shrinkage stoping), the lower portion of the stope can be filled with waste rock and/or mill tailings. In this case, ore is removed as soon as it has been broken from overhead, and the stope filled with waste to within a few feet of the mining surface. This method eliminates or reduces the waste disposal problem associated with mining as well as preventing collapse of the ground at the surface.

Cyanidation: The process of extracting gold and silver by leaching with cyanide (CN-). Cyanide, usually added in the form of a salt (e.g., NaCN, KCN), dissolves gold by the following reaction:

4Au + 8CN- + O2 + 2H2O = 4Au(CN)2- + 4OH-

Cyclone (hydrocyclone): A classifying (or concentrating) separation machine into which pulp is fed so as to take a circular path. Coarser and heavier fractions of solids report at the apex of a long cone while the finer particles overflow from the vortex.

Decline: A sloping underground opening for machine access from level to level or from the surface. Also called a ramp.

Depreciation: The periodic, systematic charging to plant expenses, assets reflecting the decline in economic potential of the assets.

Development: Underground work carried out for the purpose of opening up a mineral deposit.

Diamond drill: A rotary type of rock drill that cuts a core of rock that is recovered in long cylindrical sections, two cm or more in diameter.

Dilution (mining): Rock that is, by necessity, removed along with the ore in the mining process, subsequently lowering the grade of the ore.

Dore bar: The final saleable product of a gold mine, usually consisting of gold and silver.

EM survey: A geophysical survey method which measures the electromagnetic properties of rocks.

Epithermal deposit: A mineral deposit consisting of veins and replacement bodies, usually in volcanic or sedimentary rocks, containing precious metals, or rarely, base metals.

Extractive metallurgy: The processes of chemically separating the valuable metal from its mineral matrix (ore or concentrate) to produce the pure metal. Includes the disciplines of hydrometallurgy and pyrometallurgy.

Flowsheet: An illustration showing the step-by-step sequence of operations by which ore is treated in a milling, smelting, or concentration process.

Footwall: The rock on the underside of a vein or ore structure.

Gangue: The fraction of ore rejected as tailing in a separating process. It is usually the valueless portion, but may have some secondary commercial use.

Grab sample: A sample from a rock outcrop that is assayed to determine if valuable elements are contained in the rock. A grab sample is not intended to be representative of the deposit, and usually the best-looking material is selected.

Grade: Percentage of a metal or mineral composition in an ore or processing product from mineral processing.

Gravity separation: Exploitation of differences in the densities of particles to achieve separation. Machines utilizing gravity separation include jigs and shaking tables.

Hangingwall: The rock on the upper side of a vein or ore deposit.

Heap leaching: A process whereby valuable metals, usually gold and silver, are leached from a heap or pad of crushed ore by leaching solutions percolating down through the heap and collected from a sloping, impermeable liner below the pad.

Induced polarization: A method of ground geophysical surveying employing an electrical current to determine indications of mineralization.

Metal complexes: An ion consisting of several atoms including at least one metal cation

Metallurgy: The science and art of extracting metals from their ores, refining them, and preparing them for use. Metallurgy consists of three major disciplines: mineral processing metallurgy, extractive metallurgy, and physical metallurgy.

Mill: Includes any ore mill, sampling works, concentration, and any crushing, grinding, or screening plant used at, and in connection with, an excavation or mine.

Mine: An opening or excavation in the earth for the purpose of extracting minerals.

Mineral: A naturally occurring, solid, inorganic element or compound, with a definite composition or range of compositions, usually possessing a regular internal crystalline structure.

Mineral processing: Preparation of ores by physical methods. A subcategory of metallurgy. Methods of mineral processing include comminution, classification, flotation, gravity separation, etc.

Native metal: A natural deposit of a metallic element in pure metallic form, not combined as a mineral with other elements.

Open pit: A mine that is entirely on the surface. Also referred to as an open-cut or open-cast mine.

Open Stope: In competent rock, it is possible to remove all of a moderate sized ore body, resulting in an opening of considerable size. Such large, irregularly-shaped openings are called stopes. The mining of large inclined ore bodies often requires leaving horizontal pillars across the stope at intervals in order to prevent collapse of the walls.

Ore: A natural deposit in which a valuable metallic element occurs in high enough concentration to make mining economically feasible.

Overburden: Material of any nature, consolidated or unconsolidated, that overlies a deposit of ore that is to be mined.

pH: The negative logarithm of the hydrogen ion concentration, in which pH = -log [H+]. Neutral solutions have pH values of 7, acidic solutions have pH values less than 7, and alkaline solutions have pH values greater than 7.

Prill: Small pieces of pure precious metal extracted from assay samples. Usually resembling a cone shape in appearance.

Reclamation: The restoration of a site after mining or exploration activity is completed.

Refining: A high temperature process in which impure metal is reacted with flux to reduce the impurities. The metal is collected in a molten layer and the impurities in a slag layer. Refining results in the production of a marketable material.

Resistivity survey: A geophysical technique used to measure the resistance of a rock formation to an electric current.

Riparian: Pertaining to the bank of a natural watercourse.

Spoil: Debris or waste material from a mine.

Stockpile: Broken ore heaped on the surface, pending treatment or shipment.

Stope: An excavation in a mine, other than development workings, made for the purpose of extracting ore.

Tailings: Residue from milling processes (e.g., flotation tailings, gravity tailings, leach tailings, etc.).

Trunnion: The part of a ball valve which holds the ball on a fixed vertical axis and about which the ball turns.

Vein: A mineralized zone having a more or less regular development in length, width, and depth to give it a tabular form.

Vezin: Name for a sampler or cutter which is a circular path cross-stream sampler for primary or secondary sampling from a falling stream of slurry or free flowing solids.

Zone: An area of distinct mineralization.

23.0 DATE AND SIGNATURE PAGE

The effective date of this Technical Report titled "Casposo Project, San Juan Argentina, Technical Report on Feasibility Study, is March 30, 2007.

Signed,

"signed and sealed"

AMEC (Perú) S.A. 25 May, 2007