NI 43-101 Preliminary Economic Assessment
COSE Project
Argentina

Prepared for:
Patagonia Gold PLC

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1.0 SUMMARY

1.1 Introduction and Terms of Reference

This report was prepared by Chlumsky, Armbrust & Meyer, LLC ("CAM:) on behalf of Patagonia Gold S.A. ("PGSA") which is a 100 percent owned subsidiary of Patagonia Gold Plc ("PGD") to define and describe a National Instrument 43-101 (NI 43-101) compliant mineral Resource for the Cap Oeste Sudoeste Project (COSE or the Project), located within the El Tranquilo I MD claim in the province of Santa Cruz, Argentina.

Robert Sandefur and Craig Bow, Qualified Persons (QP) of Chlumsky, Armbrust & Meyer LLC visited the property on November 21-22, 2010. Steve Milne QP of CAM and Greg Chlumsky QP of CAM visited the site in May 2011.

1.2 Property

The COSE Project is situated in the central portion of the El Tranquilo I MD exploration claim which is held 100 percent by PGSA, the Argentine subsidiary of Patagonia Gold Plc. An agreement exists with the previous owners, Barrick Exploraciones S.A. and Minera Rodeo S.A., both subsidiaries of Barrick Gold, which have certain back-in rights, and PGSA has certain investment commitments to fulfill.

The El Tranquilo I MD exploration claim is one of several claims in PGSA's El Tranquilo project block. Another drilled prospect area called "Breccia Valentina," is located on the adjacent La Apaciguada MD exploration claim, centered approximately 5 kilometres to the southeast of the Cap Oeste Property.

1.3 Geology and Structure

The COSE project is located in the northwestern part of the Deseado Massif, in Patagonia, southern Argentina. This geological terrane is characterized by a sequence of Middle-to-Upper Jurassic volcanic rocks which are partially covered by Cretaceous volcaniclastic sediments, and by later Tertiary to Quaternary plateau basalts and fluvial-glacial sedimentary cover. Widespread epithermal mineralization is hosted by the Jurassic rocks, specifically the Chon Aike and La Matilde Formation bimodal volcanic suites.

Bedrock in the COSE Project comprises a + 300 meter thick sequence of rhyolitic ignimbrite and tuff units of the Chon Aike Formation. Based on extensive drilling of the target, the Chon Aike rocks have been further split into twelve subunits. This detailed geologic subdivision has permitted determination of relative fault displacements, and, in addition, permitted construction of detailed geologic sections which reveal important lithologic controls on the type and distribution of precious metals mineralization.

The COSE prospect is broadly coincident with a prominent IP geophysical anomaly which extends southeastwards from Cap Oeste. The northeastern margin of this anomaly corresponds to the Bonanza Fault and its proposed extension, the COSE Fault. The juxtaposition of differing rock types across the COSE Fault requires normal displacement of at least 50 meters, west side down. The COSE Breccia Fault – which is intimately related to the high grade shoot – is interpreted as a second order zone of brecciation and gouge, subparallel to and southwest of the COSE Fault.

1.4 Mineralization

Gold-silver mineralization at COSE is interpreted to be of the epithermal, low-sulfidation type The Deseado Massif volcanic province hosts several producing and advanced stage epithermal projects, which typically consist of banded, chalcedonic fissure veins and local vein/breccias, characterized by high Au and Ag contents and ratios of Au:Ag generally greater than 1:10. Exploration drilling by PGSA has been conducted within an approximate seven kilometer radius of the COSE Project area at eight prospects; Cap Oeste, Cap Oeste extension, Puma, Felix, Pampa, Vetas Norte, Don Pancho and Breccia Valentina . Cap Oeste has been the subject of previous 43-101 reports, which include resource estimations.

Precious metals at COSE are contained within a suite of hydrothermal breccias and their adjacent wallrocks. Drilling to date has defined a high grade shoot, approximately 130 meters long and 12-15 meters wide, situated in the immediate hangingwall of the COSE Breccia Fault. The ore shoot pitches steeply over an approximate 120 meter vertical interval, extending from 135 meters to 255 meters vertically below surface. Blind to the surface, mapping, trench sampling and drilling confirm that the high grade shoot is overlain by a broad zone of more diffuse mineralization which yields low level precious metal and trace element anomalism.

Two main styles of mineralization are apparent in drill cores from the COSE prospect. Highest grade Au-Ag concentrations are hosted by a distinctive suite of sinuous to weakly bifurcating breccias, comprising argillic altered fragments of volcanic host rock in a matrix of fine grained grey quartz, illite, and carbonaceous material. Precious metals occur as native metal, alloys and sulfides, in close association with base metal sulfides, pyrite, and arsenopyrite. The immediate hangingwall and footwall rocks to COSE breccias exhibit lower grade mineralized envelopes, in which precious metals occur in veinlets and disseminations.

Exploration continues along the COSE / Cap Oeste corridor, and CAM considers the potential for additional discoveries throughout the area to be high, and to include the following targets:

- Down plunge extensions to the COSE Breccia shoot
- Strike extensions to the upper portion of the COSE breccia system, which could constitute broad intervals of low grade, oxidized mineralization with higher Ag/Au ratios (see Figure 11.5.2)
- Repetitions of the COSE orebody along the COSE Breccia Fault and/or the COSE / Bonanza Fault system.

1.5 Exploration and Drilling

The COSE deposit was discovered in 2008, initially in trench samples and subsequently drill intersections. Since that time, exploration has focused on establishing a core resource in the area of strongest epithermal mineralization, although step out exploration drilling is planned for the 2011 field season. CAM considers all aspects of diamond and RC drilling to have been carried out to acceptable 43-101 standards.

1.6 Sampling

Sampling methods employed in the COSE drilling and trenching work were carried out by PGSA personnel to acceptable NI 43-101 standards.

1.7 Assaying and QA/QC

Quality control measures implemented during the trenching and drilling programs included the submission of a series of certified standard and blanks, which were incorporated and dispatched with the drill samples, according to the following protocol:

- Diamond Drilling:
- RC Drilling:
- Trenching:

Two labs were contracted for analysis of the samples: Alex Stewart and Acme Labs, both accredited laboratories compliant to ISO Certified - 9001:2000. Alex Stewart served as the principal lab, and Acme as the check lab for Au fire assay and ICP. CAM believes that preparation and analysis of samples are acceptable and within NI 43-101 standards.

1.8 Density Measurements

Measurements of specific gravity (SG) were performed by Alex Stewart Assays and on site by PGSA on a total of 94 individual, 1/2 HQ core pieces taken from one meter drill core intervals, for which the average dry sample weight was 0.66 kilograms. The specific gravity for a total of 64 samples was measured by Alex Stewart Assays and 30 samples were determined by PGSA.

The first step in reviewing density was to make sure that the calculations were properly performed by recalculating specific gravity by weight in water and weight in air. Results of these checks by CAM indicate that the original calculations were correctly performed.

The global average of all the specific gravity values based on all rock, oxidation and mineralization types for the 94 samples is 2.40. With respect to the zones of mineralization, the highest average SG values relate to the hydrothermal breccia, with an overall average of 2.44 in the oxide and 2.40 in non oxide respectively. The peak specific gravity values (2.59, 2.57, and 2.51) relate to intervals from weakly silicified breccia gouge with relatively high sulfide mineral concentrations.

1.9 Data Verification

On the basis of these statistical checks, and the checks of data entry discussed previously, CAM believes that the exploration database has been prepared according to NI 43-101 norms and is suitable for the development of geological and grade models.

1.10 Mineral Resources

The COSE resource estimate was based on data provided to CAM by PGSA. Data included the exploration database, surface topography and interpreted cross-sections. The mineral Resources and minable resources in this report were calculated using the Canadian Institute of Mining, Metallurgy and Petroleum (CIM), CIM Standards on Mineral Resources and Reserves, Definitions and Guidelines prepared by the CIM Standing Committee on Reserve Definitions and adopted by CIM Council on Dec 11, 2005.

1.10.1 Resource Estimation Results

CAM has completed the audit and review of the PGSA resource model and reports the following resources for COSE project:

<u>NI 43-101 Compliant Resource Statement</u>

Resources were calculated by Matthew Boyes of Patagonia gold and reviewed and classified as indicated and inferred by Robert L. Sandefur PE of CAM through the use of statistical methods. Resources are based on 38 holes which intersect the COSE ore shoot. CAM classified the resource as indicated based on the criteria that there is a less than 10% chance than less than 85% of the contained ounces will actually be mined. Because this deposit has only been sampled by surface drilling there is greater uncertainty than if the deposit had been estimated on the basis of channels samples a meter apart in drifts separated by 25 m vertically. Approximately 70% of the contained ounces are carried on five of the 38 holes. The silver to gold equivalent ratio of 53.5:1 is based on a gold price of $1204 per troy ounce and a silver price of $23.75 per troy ounce and gold and silver recoveries of 95 and 90% respectively were applied when calculating AuEq metal content. Metals prices were calculated as the three-year past two-year future rolling average as of March 1, 2011. CAM believes additional drilling to confirm the area of influence of the five high-grade holes is prudent.

Resources are summarized in Tables 1-1 and 1-2.

Table 1-1
NI 43-101 Compliant Resource Statement
Total INDICATED Resources Undiluted COSE Project

Tonnes	Grade			Contained Metal (Ounces)		
	Au (g/t)	Ag (g/t)	AuEq (g/t)	Au	Ag	AuEq
20,637	60.06	1,933.07	96.21	39,850	1,282,582	63,835
Gold equivalent (AuEq) values are calculated at a ratio of 53.5:1 Au;Ag.						

Table 1-2
NI 43-101 Compliant Resource Statement
Total INFERRED Resources Undiluted COSE Project

Tonnes	Grade			Contained Metal (Ounces)		
	Au (g/t)	Ag (g/t)	AuEq (g/t)	Au	Ag	AuEq
13,758	60.06	1,933.07	96.21	26,566	855,055	42,557
Gold equivalent (AuEq) values are calculated at a ratio of 53.5:1 Au;Ag.						

1.11 Minable Resources

The COSE deposit is located 150 metres below surface and will therefore be mined by underground methods with a decline access.

CAM have suggested a mechanized cut and fill mining method be adopted for the extraction of the COSE deposit, this style of mining method, although initially requiring greater quantities of sublevel development, is more appropriate for mining of narrow vein structurally controlled deposits such as COSE as dilution and ore-loss can be far better controlled. A total ore movement of 120 tonnes per day or 3600 tonnes per month has been used as the base case production forecast for the mine.

The minable resources, adjusted for dilution and mining recovery, are summarized in Tables 1-3 (Indicated) and Table 1-4 (Inferred).

Table 1-3 Minable Indicated Resources with Dilution and Mining Recovery Factors					
Tonnes	Au gms/tonne	Ag gms/tonne	Au Eq. gms/ tonne	Contained Au oz	Contained Ag oz
23,467	51.76	1665	87.19	39,049	1,256,541
Note: Dilution Width = 0.25m on each side. Mining Recovery = 98%.					

Table 1-4 Minable Inferred Resources with Dilution and Mining Recovery Factors					
Tonnes	Au gms/tonne	Ag gms/tonne	Au Eq. gms/ tonne	Contained Au oz	Contained Ag oz
15,644	51.76	1665	87.19	26,032	837,694
Note: Dilution Width = 0.25m on each side. Mining Recovery = 98%.					

1.12 Preliminary Economic Assessment

Base case metal prices used for Preliminary Economic Assessment are Au $1204/oz, Ag $23.75/oz, with recoveries of 95% and 90% respectively. The "Probable" mining resource estimates are in part based on Inferred resources as a scoping level assessment and are therefore non compliant under the NI 43-101. All cash flow calculations are based upon an undiscounted model due to total project timeline of 23 months and include a 10% royalty payable for exported concentrates. Dilution of 0.25 metres either side of the stope and a 98% recovery factor was applied to calculate the diluted minable resources.

1.12.1 Assumptions

Mining Capex

Mining CAPEX is estimated at $US 24,439,943, which includes the 1,980 meters of main decline ramp access, ore development, cross cuts and stoping of the ore. Total cost per tonne for Production during the 11 month production period are estimated at US$167/tonne and total Development cost is estimated at US$ 14,252,000.

Process Capex

The three main treatment or process routes considered for the treatment of the COSE ore include:

1. Direct Shipping, involves mining and crushing of the material on site and then shipping the ore via road and sea to a suitable smelter for direct smelting to recover the gold and silver and Ag.

2. Construction of a crushing and Cyanide leaching circuit at the La Bajada property and processing through a Merrill Crowe circuit and production of Dore' on site.
3. Gravity separation and smelting of Au and Ag on site to produce Dore'.

Process facility CAPEX estimates and metal recoveries for the three separate treatment routes are shown in Table 1-5.

Table 1-5
CAPEX Estimates and Metal Recoveries for Treatment Options

Treatment Route	CAPEX Requirement (US$ 000's)	Metal Recoveries Au;Ag
Direct Shipping	2,700	93; 90
High NaCN Leach Merrill Crowe	8,100	87; 65
Gravity concentrate-Smelting	5,900	60 ;50

<u>Cash-flow Assumptions</u>

Cash-flow calculations for the three different scenarios and a sensitivity analysis for adjusted Au and Ag prices are shown below. All cash-flow sensitivities were run on the Direct Shipping option treatment route due to the smaller initial CAPEX (US$ 2,768,000) and higher potential revenue.

Base case metal price assumptions were provided by CAM and represent a trailing 36-month and future looking 24-month calculated price giving a base case Au Price of $US 1204/oz and a base case Silver Price of $US 23.75/oz.

Au-Ag Price sensitivity analysis was run for the following metal Prices:

Au Price (US$)	Ag Price (US$)	NPV ($US M)
1,203	23.75	63.7
1,000	20	46.5
1,100	22	55.2
1,400	30	84.7
1,418	35	93.8

1.12.2 Payback

68% of contained Au and Ag will be mined within the first 4 months of production enabling payback of capital after just 14 months from commencement of the decline.

1.12.3 Cash Flow Results

The results of the cash flow include:

- a cash cost of US 167 per equivalent gold ounce produced;
- a net revenue of US$ 63.78 million; and
- an IRR of 870%.

1.12.4 Planning

PGSA has already received approval within the El Tranquilo Environmental Impact report (EIR) from the State Secretary of Mining for the development of a decline access for underground drilling at COSE as well as a bulk sampling for metallurgical test works.

With the receipt of the Resource and the Preliminary Economic Assessment, PGD can now finalize the permit application for the mining of the entire ore-body and the construction of infrastructure and processing facilities on site at the COSE project for approval.

The mineralised structure containing the COSE deposit remains open at depth and along strike. Future deeper drilling in order to expand the deposit will be carried out from underground. Additional drilling is planned between COSE and the Cap-Oeste deposit to the north-west.

1.13 Interpretations and Conclusions

1. Exploration has defined a zone of significant epithermal gold-silver mineralization at COSE, hosted within and adjacent to a moderate to high angle normal fault over widths of 5-12 meters, and a minimum strike length of 130 meters. To date the zone has been drill tested over a vertical interval of 120 meters.
2. Technical work has been conducted in a professional manner and carried out to NI 43-101 standards including the analysis, quality assurance and quality control protocols.
3. Drill intercepts identified as significant to delineation of a precious metals resource have been verified and substantiated sufficiently to pursue a Resource calculation.
4. Work on the property has been successful in identifying mineralization of potential economic interest, and further work is warranted.
5. Based on the positive results of the PEA, PGSA should proceed with a feasibility study for the COSE deposit, noting that:
    a. Additional drilling may be required to convert currently inferred resources into indicated.
    b. Because of the fact that a large proportion of the ounces are contained in a small amount of the tonnes, options which minimize capital should be considered in the feasibility.

2.0 INTRODUCTION

This report was prepared by CAM for PGSA, to define a gold and silver resource at the COSE Project, Santa Cruz province, Argentina, which complies with Canada National Instrument 43-101 (NI 43-101). PGSA is a 100 percent owned subsidiary of Patagonia Gold Plc which is listed on the London AIM stock exchange. Data contained in this report are drawn from original work by PGSA, unpublished data from former owners and explorers (Barrick and Homestake), and third party studies on various aspects of the deposit. The report includes data and analysis from contractors, consultants, certified laboratories, and CAM's Qualified Persons.

The authors' direct knowledge of the property is based on a site visit conducted November 21-22, 2010. During this time period, the undersigned examined outcrops and the locations of drill holes and surface samples, observed drilling and sampling of diamond core, observed logging and sampling procedures and reviewed the Project with PGSA staff.

2.1 Qualified Persons

Craig Bow, Ph.D. Geology, Greg Chlumsky, Steve Milne, P.E. and Robert Sandefur, P.E., all Qualified Persons as defined by NI 43-101, prepared this report, with input by other individuals as listed in Section 3.0. Dr. Bow is responsible for Sections 1 to 13, 15, 16, and 18 to 24 of this report, Mr. Chlumsky is responsible for Sections 16 and 19, Mr. Milne is responsible for Section 19 and Mr. Sandefur is responsible for Sections 14 and 17.

2.2 Conventions

All references to dollars ($) in this report are in US dollars unless otherwise noted. Distances, areas, volumes, and masses are expressed in the metric system unless indicated otherwise

2.3 Units and Abbreviations

For the purpose of this report, all common measurements are given in metric units. All tonnages shown are in metric tonnes of 1,000 kilograms, and precious metal values are given in grams or grams per metric tonne.

To convert to English units, the following factors should be used:

1 short ton = 0.907 metric tonne (MT)

1 troy ounce = 31.103 grams (g)

1 troy ounce/short ton = 34.286 g/MT

1 foot = 30.48 centimetres = 0.3048 metres

1 mile = 1.61 kilometre

1 acre = 0.405 hectare

The following is a list of abbreviations used in this report:

Abbreviation	Unit or Term
AARL	Anglo American Research Laboratory
AA	atomic absorption
Ag	Silver
ARD	acid rock drainage
AR$	Argentinean peso
Au	gold
CAM	Chlumsky, Armbrust and Meyer, L.L.C.
CIC	carbon in column
C-I-L	carbon in leach
°C	degrees Celsius
Cu	copper
EIA	Environmental Impact Assessment
gm or g	gram
g/t or gpt	grams per tonne
g/cc	grams per cubic centimetre
GIS	geographic information system
GPS	global positioning system
ha	hectare
HCl	hydrochloric acid
IP	induced polarization (geophysical survey)
ICP-ES	Inductively Coupled Plasma-Atomic Emission Spectrometer
ISO	International Organization for Standardization
kg	kilogram
km	kilometre
kT	1,000 tonnes
lb	pound
m	metre
M	million
Ma	million years before present
NGO	Non-governmental Organization
NI 43-101 or 43-101	Canadian Securities Administrators' National Instrument 43-101
ounce or oz	troy ounce
PGD	Patagonia Gold Plc
PGSA	Patagonia Gold S.A.

ppb	parts per billion
ppm	parts per million
Project	Cap Oeste Project
QA	quality assurance
QC	quality control
RC	reverse circulation
RFP	Request for Proposal
RQD	rock quality designation
Std. Dev.	standard deviation
t or tonne	metric ton
TSF	tailings storage facility
UG	underground
US$	United States dollars
y or yr	year
/	per

3.0 RELIANCE ON OTHER EXPERTS

Other persons beside the undersigned provided data for this report. These included Damien Koerber, geologic consultant and COSE Project Geologist, Alejandra Jindra. Others include Mathew Boyes, PGSA project manager, and NCL Chile.

4.0 PROPERTY DESCRIPTION AND LOCATION

4.1 Location

The Cap Oeste Project area is located in the central portion of Santa Cruz province, in the Department of Rio Chico, southern Argentina (Figure 4-1). The core resource area is situated within the El Tranquilo I MD ("*Manifestación de Descubrimiento*"), within the El Tranquilo block of exploration properties approximately 65 kilometres southeast of the small township of Bajo Caracoles.

The closest cities to the Project site by road are Perito Moreno (208 kilometres northwest of the Project) and Gobernador Gregores (190 kilometres south of the Project). The Project is accessed via the partially-sealed National Highway 40 heading south for approximately 166 kilometres from Perito Moreno, passing via the township of Bajo Caracoles to a junction titled "Cinco Buzones." This highway infrastructure is currently being upgraded to an all bitumen double lane highway. A secondary improved gravel road is then followed east for approximately 42 kilometres to the Project site, approximately five kilometres to the northwest of the Estancia La Bajada.

Figure 4-1
Project Location

The COSE prospect is centered approximately 1.5 kilometers to the south-southeast of the Cap Oeste Project area. The principal zone of interest occupies a low ridge which extends over an approximate NW-SE trending, 120 meter by 900 meter area. Access to the prospect is gained via a track which extends to the west for approximately 3.5 kilometers from the main access road that connects Estancia la Bajada and the Cap Oeste Project area. The Estancia La Bajada comprises a main farmhouse and several outbuildings which provide space for an exploration base camp, including logging, core cutting, sample preparation, and core storage facilities.

Infrastructure improvements to the property include a graded single track road and several secondary side access tracks to drilling platform areas. There are no mineral reserves, historic mine workings, tailings, tailings ponds, or waste deposits in the Project area.

4.2 Mineral Tenure and Title

4.2.1 COSE Project-Patagonia Gold S.A. - Exploration Claims

The COSE Project is located within the El Tranquilo I Manifestation of Discovery (MD) claim, which is one of seventeed contiguous exploration tenements comprising the El Tranquilo block of properties (60,056 hectares), controlled 100 percent by PGSA.

The El Tranquilo I MD claim was largely constituted from a pre-existing cateo claim block titled El Tranquilo (government file 404.195/MR/02), and a subsidiary portion originally covered by the La Apaciguada MD (government file 405.473/MR/05). The MD El Tranquilo (government file 403.094/PATAGONIA/07) was staked in September 2007 under the "Manifestation of Discovery" covering the last portion released of the original El Tranquilo Cateo.

In accordance with the Argentine mining code, all of the exploration properties are spatially registered in the Gauss Kruger Projection and Campo Inchauspe datum system in the corresponding longitudinal belt defined between 68°-70° West (Faja 2). The details of each property are provided in Table 4-1. The location of the COSE Project area with respect to Cap Oeste and the El Tranquilo MD claim is displayed in Figure 4-2.

Table 4-1 El Tranquilo Block of Exploration Properties			
Name	**Property Type**	**Property File No.**	**Area (hectares)**
La Mansa	MD	413.543/MR/06	1,736.50
El Tranquilo I	MD	403.094/PATAGONIA/07	3,736.20
La Apaciguada	MD	405.473/MR705	3,472.50
La Bajada	MD	404.562/PATAGONIA/05	5,000.00
La Cañada	CATEO	412.791/Barrick/04	7,499.10
La Cañada I	MD	403.985/PATAGONIA/07	2,794.50
Cerro León	CATEO	406.025/MR/02	3,968.10
La Marcelina	CATEO	412.792/Barrick/04	6,500.10
Monte León	MD	415.664/MR/07	1,987.40
Monte Puma	MD	406.881/MR/06	2,000.00
Monte Tigre	MD	406.882/MR/06	2,000.00
Marte	MD	409.148/MR/05	2,500.00
María	MD	412.520/MR/05	743.10
Enriqueta	MD	412.519/MR/05	1,500.00
Las Casuarinas	CATEO	424.914/PG/09	3,637.90
El Mangrullo	CATEO	424.914/PG/09	4,275.60
El Aljibe	CATEO	424.914/PG/09	6,704.70
Nueva España I	CATEO	422.217/PG/10	9,955.00
Nueva España II	CATEO	422.216/PG/10	4,447.00
Don Fransisco	CATEO	423.465/PG/10	1,798.00
Nuevo	CATEO	423.670/PG/10	3,473.00

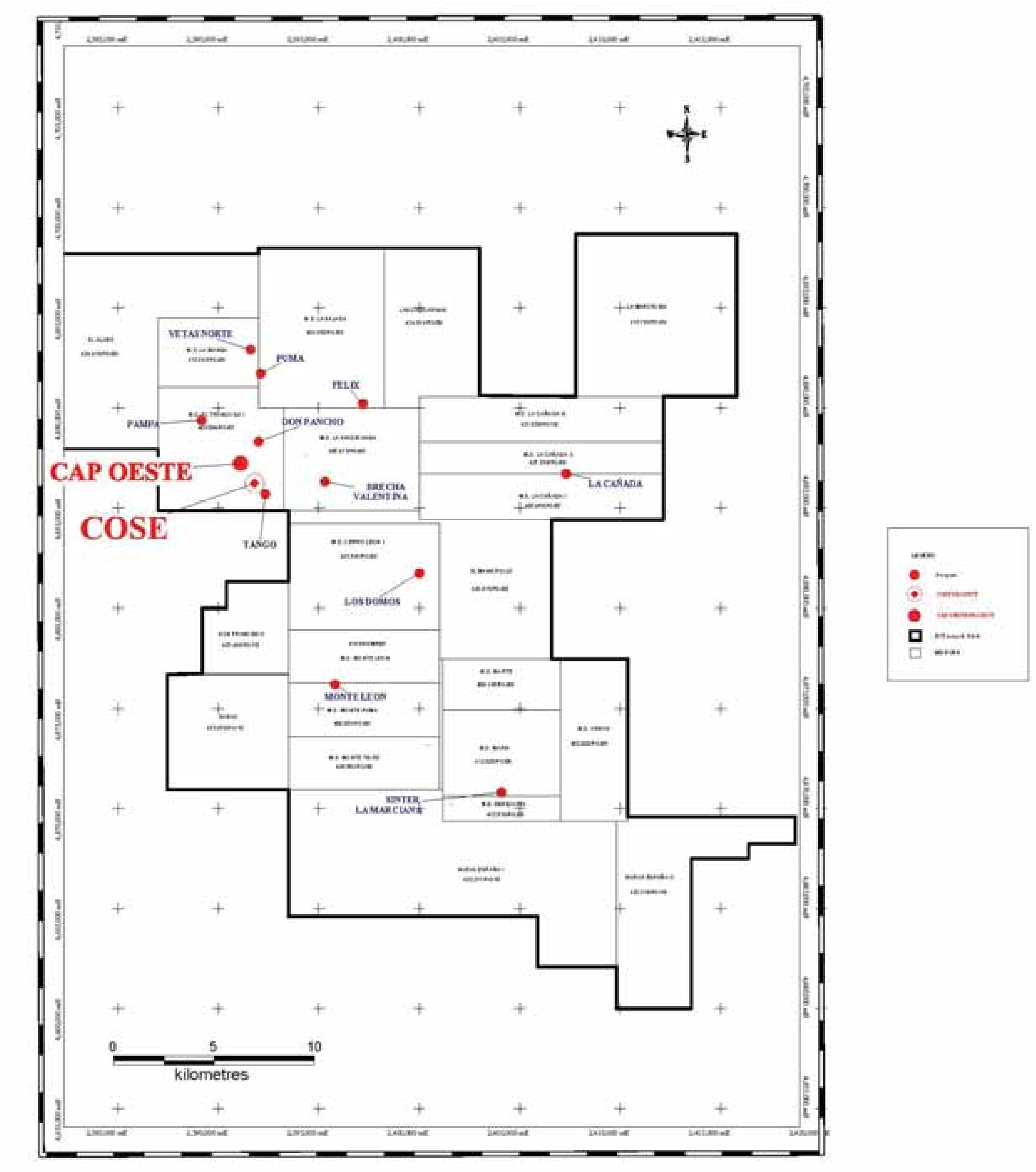

Figure 4-2
Location of the COSE Project in relation to Cap Oeste and the El Tranquilo I MD Claim

The claim titles are current and renewed annually by fee. The renewal is contingent on continued exploration work on the claim within each year. All the MD's are within the legal period prior to which

PGSA has to survey individual concessions (pertenencias) so as to eventually constitute a mining concession or 'Mina'.

4.3 Surface Rights and Obligations

Surface rights in Argentina are not associated with title to either a mining lease or exploration claim and must be negotiated with the landowner. The COSE Project occurs wholly within the Estancia La Bajada, which was purchased in December 2008, including the farmhouse and outbuildings.

4.4 Mineral Property Encumbrances

The majority of the properties of the El Tranquilo block, including the El Tranquilo property, were acquired as part of a Purchase Agreement signed in February 2007 between PGSA and the Barrick Gold S.A. Argentinean exploration subsidiaries, namely Minera Rodeo S.A. and Barrick Exploraciones S.A.

Terms and conditions of this Purchase Agreement include:

1. A US$10,000,000 commitment of approved exploration expenditures within a period of five years, of which US$1,500,000 must be invested during the first 18 months. PGSA has already sent the legal notification to Barrick´s subsidiaries advising that the investment commitments of US$1,500,000 and US$ 10,000,000- have been exceeded as of December 31, 2007 and December 31, 2008 respectively.
2. PGSA is required to provide an annual year-end resource estimation statement completed by an independent qualified person and the provision of the data used for the generation of such statements.
3. Barrick Gold S.A. holds the right to 'back-in' up to 70 percent for any individual property group included in the Purchase Agreement upon written notice, within 90 days upon completion of a 43-101 compliant delineation of a two million ounce gold or gold equivalent Indicated Resource, within the respective property group. This is on a forward looking basis which does not include any resources or reserves produced or undergoing development. Upon exercise of the 'back-in' right PGSA must transfer the property group to a separate joint-venture corporation ("JV Company") which will be free from any and all encumbrances. The back-in right will survive any sale by PGSA of any portion of the property group.

As an integral part of both Barrick Gold's and PGSA´s due diligence it was verified that there are no other mineral property encumbrances over the Project or block of properties.

The following mining properties included in the El Tranquilo block were granted directly to Patagonia Gold S.A., either previous to or subsequent to the signing of the Purchase Agreement with Barrick Gold S.A., and therefore are not subject to its terms and conditions.

- La Bajada 404.562/PATAGONIA/05
- Las Casuarinas 424.914/PG/09
- El Mangrullo 424.914/PG/09
- El Aljibe 424.914/PG/09
- Nueva España I 422.217/PG/10
- Nueva España II 422.216/PG/10
- Don Fransisco 423.465/PG/10
- Nuevo 423.670/PG/10

4.5 Environmental Liabilities

No previous mining or significant exploration activity has been conducted on the El Tranquilo block. To the best of CAM's knowledge, the property is not subject to any environmental liabilities related to exploration or mining activities.

4.6 Permits

Work at the COSE Project was conducted in accordance with the legal requirement for an approved biannual Environmental Impact Assessment (EIA) for the El Tranquilo Project block, for which the preexisting one was renewed and subsequently approved and granted on October 8th 2008, with an effective duration of two years.

The company has been conducting quarterly baseline water sampling throughout the Project area since May 2007, and producing independent reports prepared by a private consultant (BEHA). Results of these studies were included in the newly-presented EIA for the Project and submitted to the pertinent authorities.

PGSA has obtained the relevant permits for the use of water during the drill campaigns, issued by the pertinent government water resources authority of the Santa Cruz Province (Recursos Hídricos), subsequent to the approval by the corresponding surface owners. No other permits are required for the continuation of exploration and/or definition drilling within the property block.

5.0 CLIMATE AND TOPOGRAPHY, ACCESS AND INFRASTRUCTURE, ENVIRONMENTAL AND SOCIAL ISSUES

5.1 Climate, Topography, and Vegetation

The Patagonian region of southern South America is characterized by arid, windy and generally treeless expanses of rolling hills, interspersed with isolated plateaus which rise to elevations of 250 - 1,000 meters above sea level. Field work is generally feasible from September to June while mid-winter (June-August) is typically a recess period. In the absence of excessive snow and rain, exploration occasionally continues into this period due to frozen ground conditions which permit access over otherwise wet areas.

The closest meteorological information available is sourced from the cities of Perito Moreno and Gobernador Gregores, which are located at similar elevations to the Project area at straight line distances of 160 and 90 kilometres respectively. In order to commence baseline environmental studies within the Project area a Davis Model Vantage Pro2 wireless weather station was installed by PGSA on 3rd November 2008 within 1500 meters of the COSE Project area. This equipment records a comprehensive array of meteorological data each minute which is subsequently downloaded by PGSA once per month.

Based on meteorological information sourced from the cities of Perito Moreno and Gobernador Gregores the average annual rainfall at the Project area is estimated to be 300 millimeters the majority of which falls in the period June-September. Snow frequently accumulates on site between June and August, and infrequent snowfall events can deliver up to 100 mm, based on limited historic data. Annual potential evaporation is estimated at between 750 and 1,250 mm. Temperatures at the Project area are characteristic of the central plateau of the Santa Cruz, with short warm summers, and winters with temperatures commonly below 0 degrees Celsius. Based on regional data, the annual average temperature is approximately 8.9 degrees Celsius. Average monthly temperatures above 10 degrees Celsius generally occur between November and March, whereas temperatures below 5 degrees Celsius generally occur from June through August. Strong winds (greater that 40 kilometers per hour) occur year round but typically are strongest during the spring and summer. The dry, windy climate accentuates the aridity of the region by generating an extremely high rate of evaporation and constitutes a strong natural erosive mechanism for the sparse vegetation cover.

The southeastern portion of the COSE Project area is characterized by a predominant northwest-southeast aligned pattern of undulating hills between elevations of 400 and 460 m.a.s.l. Throughout the Project area, topography is generally low and flat. Vegetation constitutes approximately 50 percent of the ground cover and is characterized by grass and bushes; the former typically include the varieties Stipa sp, Poa sp and Festuca sp which are locally named "coiron." Subordinate plant species include Neneo (Mulinum sp), Adesmia (Adesmia sp), Calafate (Berberis sp), Senecio (Senecio sp), Zampa (Atriplex sp), and Mata Negra (Verbena sp).

Despite the general scarcity of surface water throughout the area, several significant fresh water springs (each producing more than 4 liters per second) occur in a northwest trending, geologically controlled corridor extending at least three kilometers from the northwestern corner of the Project area. Water supplies for drilling and exploration camp amenities are obtained from these local springs and water courses with permission of the surface owners and respective provincial authorities.

5.2 Access and Infrastructure

As described in section 4.1, the Project area is accessed from the capital city of Buenos Aires by commercial air service and a network of improved highways. The Ruta 40 highway infrastructure throughout the province of Santa Cruz is currently being upgraded to an all bitumen double lane highway scheduled to be completed by the end of 2012.

Within each individual regional population center, including Perito Moreno and Bajo Caracoles, electrical power is supplied via local diesel generators. Within the Project area, electrical power is supplied through company owned or leased generators. The nearby towns generally source local groundwater supplies to meet their needs.

The closest fixed line telephone to the area is situated in Bajo Caracoles (65 kilometers from the Project) and since there is no mobile network coverage throughout the Project area, communication from the exploration camp at Estancia La Bajada is via satellite phones and satellite-based broadband internet.

5.3 Environmental and Social Responsibility

As described in Section 4, exploration has been conducted in accordance with an approved Environmental Impact Assessment (EIA). The Santa Cruz Provincial Mining Directorate's agents together with representatives from the local communities have inspected PGSA's exploration activities, specifically during drilling, and have reportedly expressed satisfaction as to the manner in which the company has carried out operations.

Although once a large wool and mutton producing region, the area encompassing the Project is currently uninhabited, destocked, and unproductive as a result of overgrazing, gradual desertification, and severe loss of productivity following the eruption of the Hudson Volcano in Chile in 1991. To the extent practical, PGSA utilizes local communities to source food, accommodation, fuel, minor vehicle repairs and field labor. More specialized goods and services must be obtained in Caleta Olivia (Santa Cruz), Comodoro Rivadavia (Chubut) and Buenos Aires. The local workforce comprises mainly unskilled workers who receive safety, environmental and exploration methodology training. Senior project management and engineering positions are generally filled by professionals from outside the local communities.

Patagonia Gold S.A. has contracted Vector Argentina S.A. as consultant for community relations throughout the Santa Cruz Province. Under their auspices, public relation meetings have been conducted which involve open-forum discussions focused on industry best practice policies and social responsibility.

6.0 HISTORY

6.1 Early History

No historic mineral production is known to have occurred within or in close proximity to the COSE Project. The earliest modern exploration in the area was reportedly carried out during the mid 1990's by Western Mining Corporation and Homestake Mining, who initially targeted the area using Landsat imagery. Interpretation of the imagery highlighted the presence of regional-scale, northwest-trending lineaments and large zones of coincident clay alteration which served to focus the reconnaissance mapping and sampling. This work led to the staking of exploration claims by the Homestake Mining subsidiary Minera Patagonica S.A., which were held until July 2002. Subsequent to the merger between Barrick Gold and Homestake Mining, the ground was again staked as the El Tranquilo Project by Barrick Gold's subsidiary Minera Rodeo S.A.

6.2 Homestake-Barrick Exploration

Exploration of the El Tranquilo Property Block by Barrick Gold spanned the period May 2002 to May 2006, at which time the decision was made to divest the project areas. The combined Homestake-Barrick exploration programs conducted throughout the El Tranquilo property block during this period included:

- Target generation incorporating information from the Homestake Mining geochemical database, supplemented by ASTER and Landsat Band Ratio image analysis
- Regional scale geological and structural mapping (1:25,000 to 1:100,000) and TM based alteration mapping at 1:50,000
- Geochemical sampling including 334 lag samples, 569 regional rock chip samples and 469 sawn channel samples taken from 11 trenches (1694 metres).
- Pole-Dipole Induced Polarization and resistivity surveying along 8 lines spaced 150 to 300 metres apart, totaling 27 line kilometres.
- Regional spaced ground magnetic surveying along 16 lines spaced 100 metres apart, totaling 35.2 line kilometres.
- Petrographic studies.

As a result of this program of work, several significant Au-Ag targets were defined along a series of sub-parallel, northwest trending structural lineaments which proved to contain the Cap Oeste (originally referred to by Barrick as the Zona Central), Breccia Valentina, and Vetas Norte prospects. With the assistance of external consultants, conceptual genetic models were developed for the various styles of low sulfidation precious metal mineralization encountered, in order to help guide subsequent exploration.

In summary, the Homestake- Barrick exploration program defined a 10km wide x 25km long northwest trending epithermal district hosting extensive zones of precious and trace element anomalism, hydrothermal alteration, and coincident chargeability/resistivity targets. Within this area, three main corridors have subsequently been delineated; the Cap Oeste, Breccia Valentina, and Vetas Norte Corridors, as shown in Figure 6-1. During the above work, no geochemical samples were taken nor geophysics conducted over the COSE prospect area.

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6.3 Patagonia Gold Program

PGSA visited the project and began negotiations for the purchase of the properties in September 2006. Subsequent to the Purchase Agreement reached on February 5, 2007, exploration activities commenced including gridding, surveying, trenching, and drilling programs which are detailed further in Sections 10 through 12 of this Technical Report. From 2008-2010, work conducted throughout the Cap Oeste Prospect by Patagonia Gold defined a 43.101 compliant resource of 655,932 Au equivalent ounces (Table 6-1), which led in turn to the initiation of economic scoping studies.

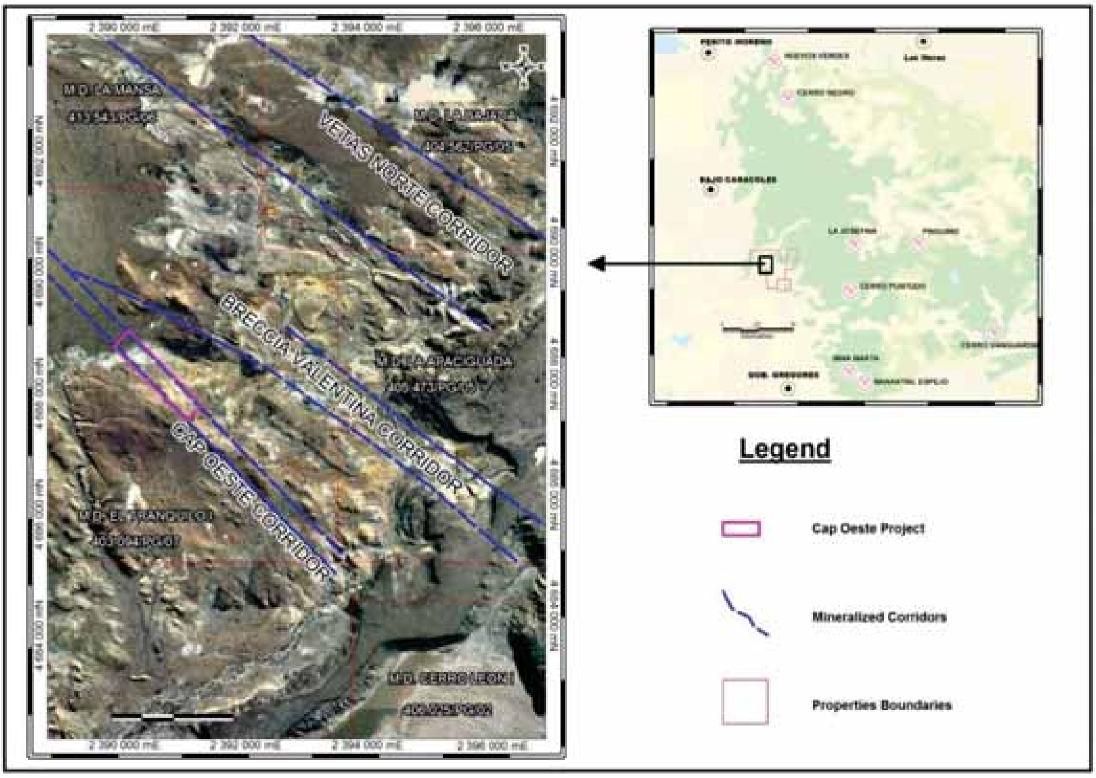


Figure 6-1
Hydrothermal Alteration and Mineralized Corridors: El Tranquilo Property Block

7.0 GEOLOGICAL SETTING

7.1 Regional Geology

The COSE Project is located within the Deseado Massif geological province, which occupies a 70,000 square kilometer area in the northern third of Santa Cruz Province. The geology of Santa Cruz has been mapped and compiled at 1:750,000 scale, published by SEGEMAR in 2003 (Figure 7-1).

Both the Deseado Massif and a second uplifted block, the Somuncura Massif (exposed in Chubut and Rio Negro Provinces to the north), are interpreted to have developed during large-scale continental volcanism accompanying extensional rifting of the Gondwanaland supercontinent and the opening of the Atlantic Ocean (Feraud, et.al, 1999). Bedrock comprises a bimodal suite of andesitic to rhyolitic ignimbrites and tuffs, with lesser flows and intrusions, which was erupted over a 50 million year interval in the middle to late Jurassic (125 to 175 Ma). Its aerial extent places this geological province amongst the most extensive rhyolite platforms worldwide. The Deseado Massif is bordered by two Cretaceous petroliferous basins, the San Jorge Basin to the north, which separates it from the Somuncura Massif, and the Austral-Magallanes Basin to the south. These basins contain thick sequences of non-marine sedimentary rocks which host Argentina's largest producing oil and gas fields.

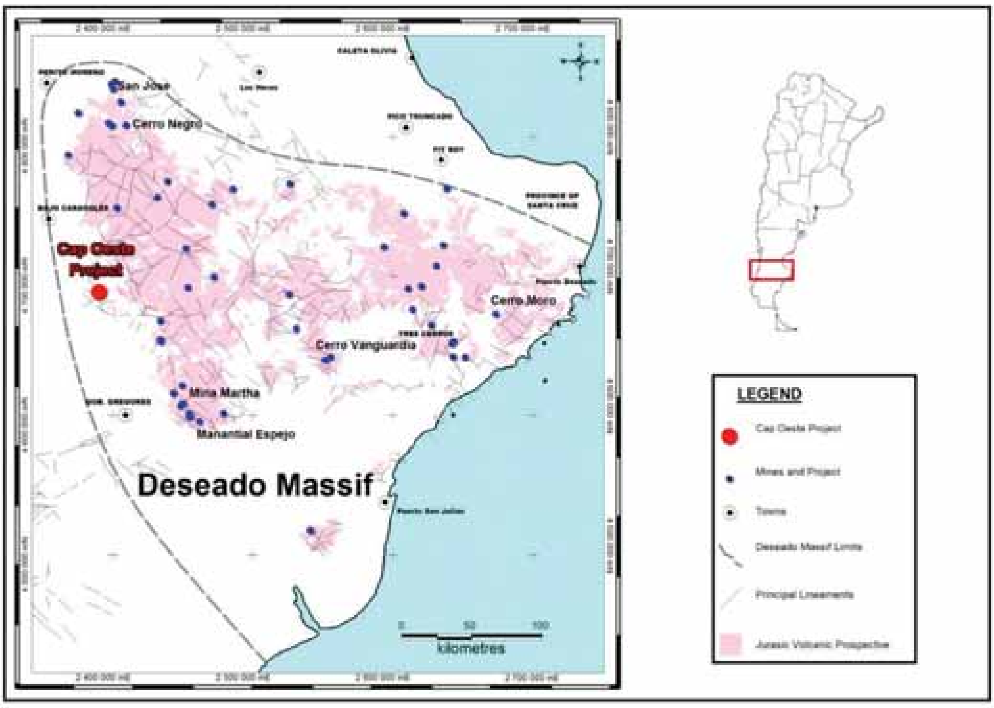


Figure 7-1
Regional Geology of Deseado Massif

Within the project area, the Jurassic volcanic suite is comprised dominantly of rocks assigned to the Bahia Laura Group. The volcanic stratigraphy of the Bahia Laura Group is the best exposed rock sequence in the Deseado Massif, covering more than half of its area, and comprises three formational members:

Bajo Pobre Formation (175-166 Ma): andesitic to basaltic flows, agglomerates, and minor hypabyssal porphyry intrusives which intercalate upwards with mafic tuffs, conglomerates and sediments. Olivine basalts common in the lower part of the formation are thought to be products of fissure eruptions from rifts related to early stages of the Gondwana breakup and continental separation.

Chon Aike Formation (166 – 150 Ma:): high-Si, high-K rhyolitic to rhyodacitic ignimbrites, tuffs and lesser volcanic breccias, flows and domes which attain a cumulative thickness up to 1,200m (Sanders, 2000). Volcanic rocks assigned to the Chon Aike Formation are coincident in space and time with the most significant precious metal deposits in the province.

La Matilde Formation (upper age of approximately 142 Ma): fine grained fossiliferous lacustrine sediments, volcano-sedimentary rocks and airborne tuffs.

The Bahia Laura Formation is underlain by an extensive sequence of basement rocks ranging in age from Precambrian to early Jurassic. Younger cover sequences include small windows (less than 300 meters in diameter) of flat-lying Tertiary marine sediments (which have filled structural controlled and/or erosional basins) and alkalic basalts, which form extensive plateaus throughout the region. Finally, unconsolidated Quaternary glacial - fluvial sediments form characteristic elevated gravel terraces throughout the province.

In a regional structural sense, northwest-southeast extensional faults active during the period of Jurassic volcanism formed grabens, half-grabens and horst blocks with pervasive eastern dips. Since the Jurassic, rocks have been cut by normal faults that probably represent reactivated basement fracture zones. The Jurassic rocks have undergone only minor subsequent deformation and remain relatively flat to gently dipping, except on a local scale proximal to faults and subvolcanic intrusions.

Fault kinematics throughout both the COSE Project and the surrounding region are consistent with regional east-west to northeast-southwest extension as has been documented for many low sulfidation, epithermal precious metal deposits throughout the province.

7.2 Property Geology

7.2.1 Stratigraphy

Bedrock in the COSE Project comprises a + 300 meter thick sequence of rhyolitic ignimbrite and tuff units of the Chon Aike Formation. Based on extensive drilling of the target, the Chon Aike rocks have been further split into twelve, gently SW dipping units (Figures 7-2, 7-3). This detailed geologic subdivision has permitted determination of relative fault displacements, not only in COSE but also extending to the northwest throughout the Cap Oeste Extension, Cap Oeste, and Pampa prospect areas. In addition, construction of geologic sections incorporating detailed stratigraphic breakdowns has demonstrated important lithologic controls on the type and distribution of precious metals mineralization.

Figure 7-2
Detailed Stratigraphy of the Chon Aike Formation: COSE Project

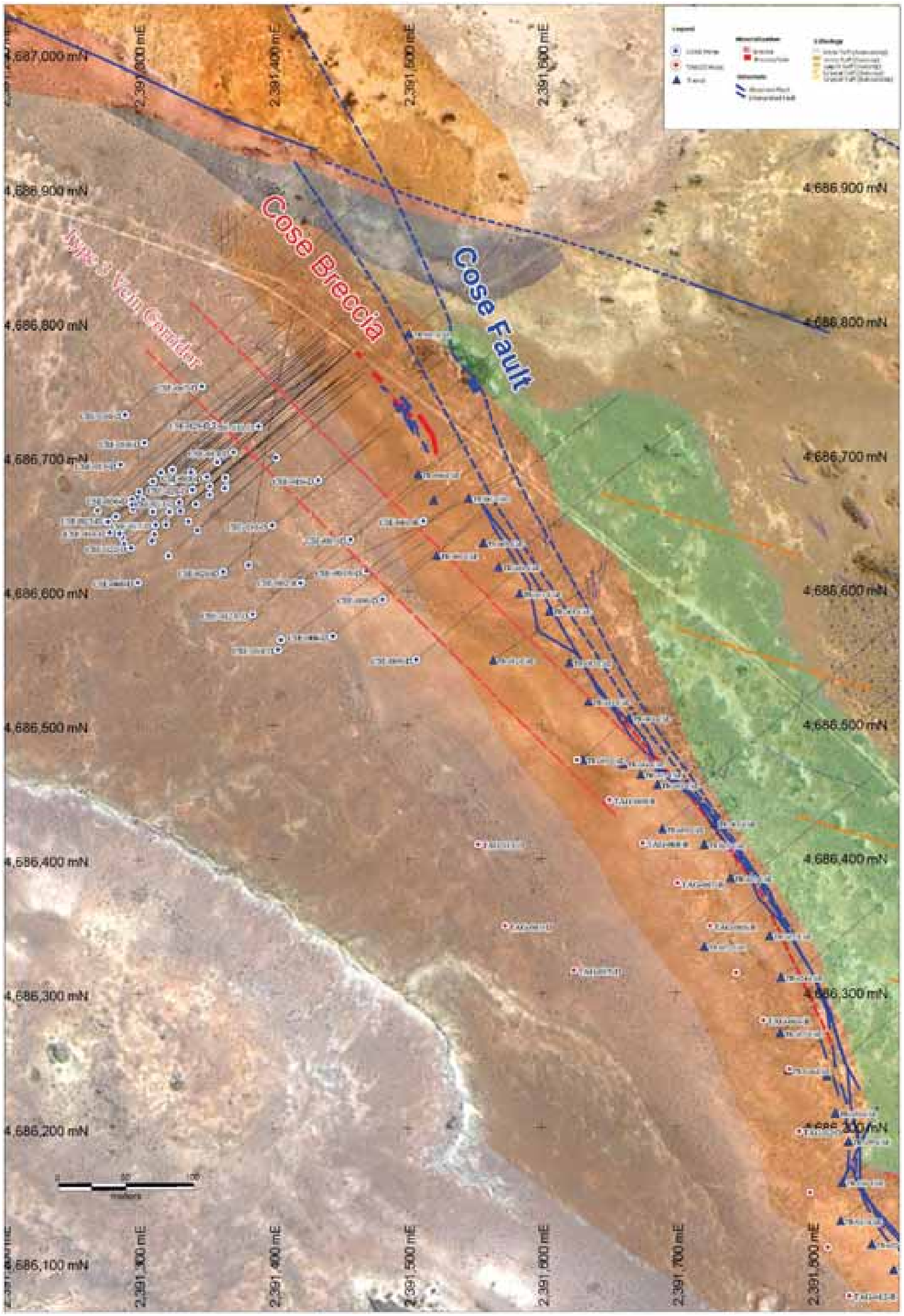


Figure 7-3
COSE Project Area Geology and Structure

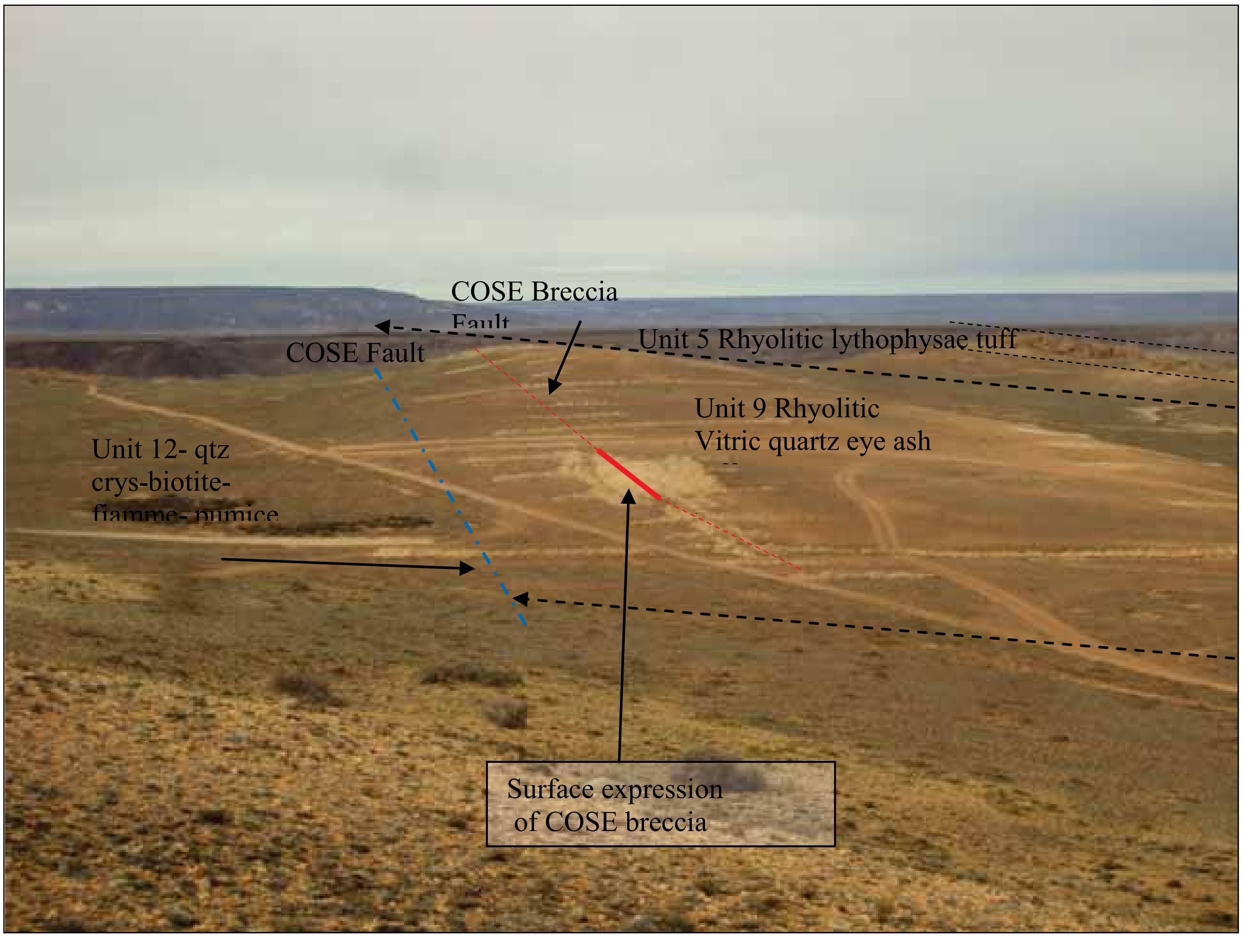
7.2.2 Structure

The COSE prospect area is broadly coincident with a prominent IP geophysical anomaly which extends southeastwards from Cap Oeste. The northeastern margin of the anomaly corresponds to the Bonanza Fault and its proposed extension, the COSE Fault. The COSE Fault can be traced at surface along the entirety of the prospect area as a 25 meter wide corridor with local zones of gouge and brecciation. The juxtaposition of vitric quartz eye tuff against fiamme-rich, lapilli tuff across the COSE Fault requires normal displacement of at least 50 meters, with the west block downthrown. The COSE Breccia Fault – which is intimately related to the high grade ore shoot – is interpreted as a second order zone of brecciation and gouge, subparallel to and southwest of, the COSE Fault (Photo 7-1).

As currently understood, the COSE ore shoot lies in the immediate hangingwall of the COSE Breccia Fault. Where mineralization would project to surface, the orthogonal distance between the COSE Breccia Fault and COSE Fault is approximately 40 meters, with the two structures converging to the southeast (Photo 7-1).

A prominent series of west-northwest trending, sub-vertical fracture sets transects the COSE prospect area, and are best developed in the fiamme-rich, lapilli tuff unit. It has been proposed that the intersection of one of these broadly spaced fracture sets with the COSE Breccia Fault acted as a secondary structural control on the location of the COSE Shoot (Figure 7.2.2.1). The line of intersection of these two structures closely corresponds with the interpreted pitch axis of the high grade mineralization down the plane of the COSE Breccia Fault.

CAM believes the structural observations summarized above are reasonable and consistent with observed features of the geology and associated mineralization. Exploration is at a relatively early stage, however, and additional work – particularly further exploration drilling and underground development – may require revision of this model.



Oblique photo looking south, showing juxtaposition of rock types along COSE Fault and its convergence with the COSE Breccia Fault.

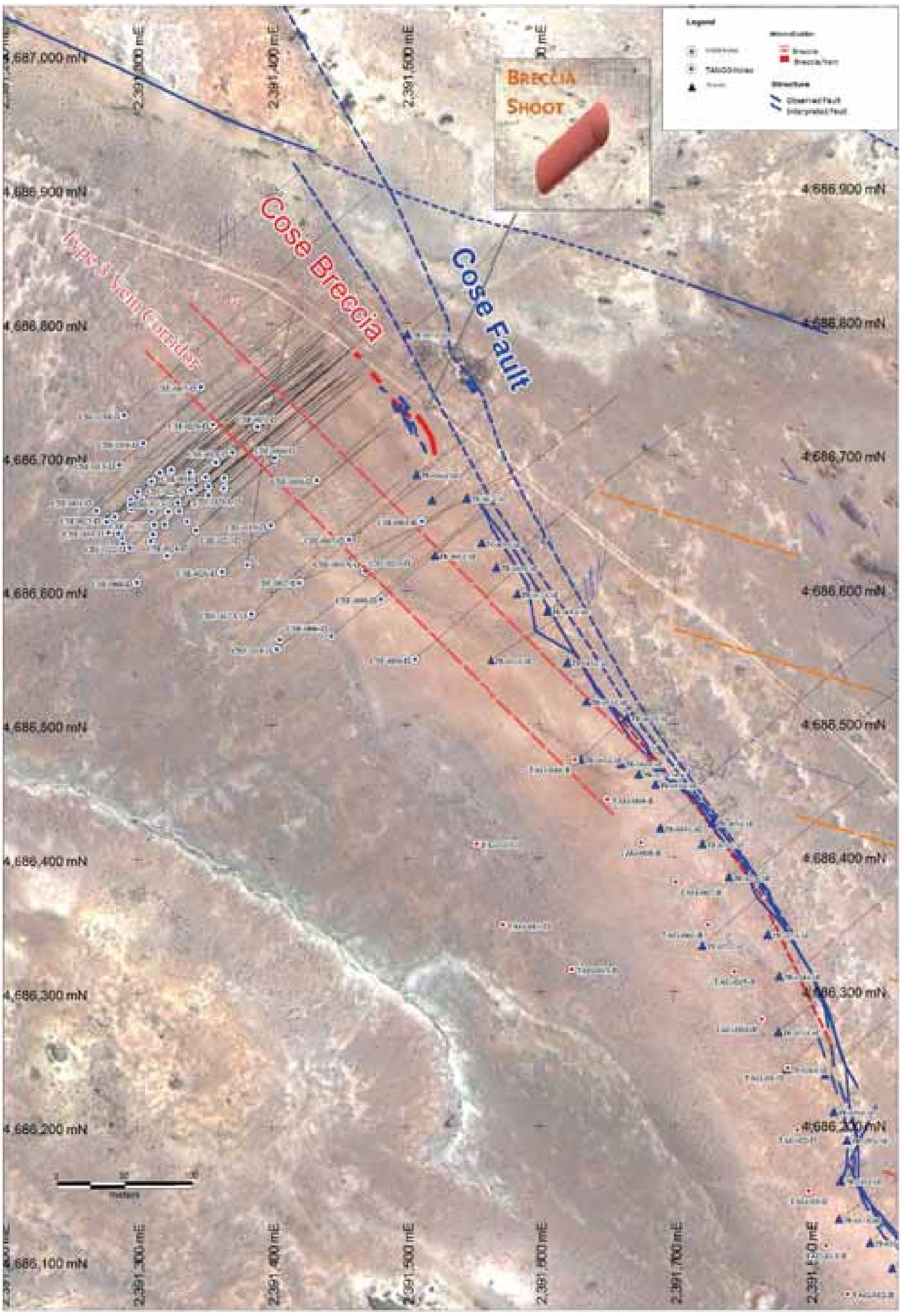


Figure 7-4
COSE Structural interpretation in plan–showing schematic plunge of COSE Breccia (box), COSE Breccia Fault, inferred trends of WNW trending fractures sets (orange) and COSE and Victoria Faults

8.0 DEPOSIT TYPES

Exploration by PGSA throughout the El Tranquilo Block is focused principally on discovery and delineation of low sulfidation, Au-Ag epithermal mineralization of the type well documented throughout the Deseado Massif [e.g. White and Hedenquist (1990 &1994), Corbett, G.J. (2001) and Sillitoe, R.H. (1993)]. Mineralization typically comprises banded fissure veins and local vein/breccias characterized by high Au and Ag contents and ratios of Au:Ag generally greater than 1:10. Mineralized veins and breccias consist of quartz (colloform, banded, and chalcedonic morphologies), adularia, bladed carbonate (often replaced by quartz), and dark sulphidic material termed *ginguro* (fine grained electrum or Ag sulphosalts banded with quartz). Discrete vein deposits develop where mineralizing hydrothermal fluids are focused into dilatant structures, producing ore shoots which host the highest precious metal grades. Low sulfidation mineralization can also develop where mineralizing fluids flood permeable lithologies to generate large tonnage, low grade disseminated deposits (e.g. Round Mountain, Nevada; McDonald Meadows, Montana)

Studies of alteration patterns and fluid inclusion data demonstrate that precious metal precipitation generally occurs between 180^{o} to 240^{o} Celsius, corresponding to depths 150 to 450 meters below the paleosurface (Figure 8-1). Deposits often exhibit a top to bottom vertical zonation:

- Precious metals poor, paleosurface, sinter (Hg-As-Sb).
- Au-Ag-rich, base metal poor “bonanza zone” (Au-Ag-As-Sb-Hg).
- Ag-rich, base metal zone (Ag-Pb-Zn-Cu).

Alteration is controlled by the temperature and pH of the circulating hydrothermal fluids and its distribution therefore can be spatially zoned. Alteration minerals that occur proximal to mineralization include illite, sericite, calcite and adularia whereas smectite and chlorite typically occur in a more distal setting. Additional variants include pervasive silicification of wall rock as envelopes to quartz veins and breccias, and advanced argillic alteration (alunite, jarosite, kaolinite, vuggy silica) in steam heated horizons at higher structural levels (Figure 8-1).

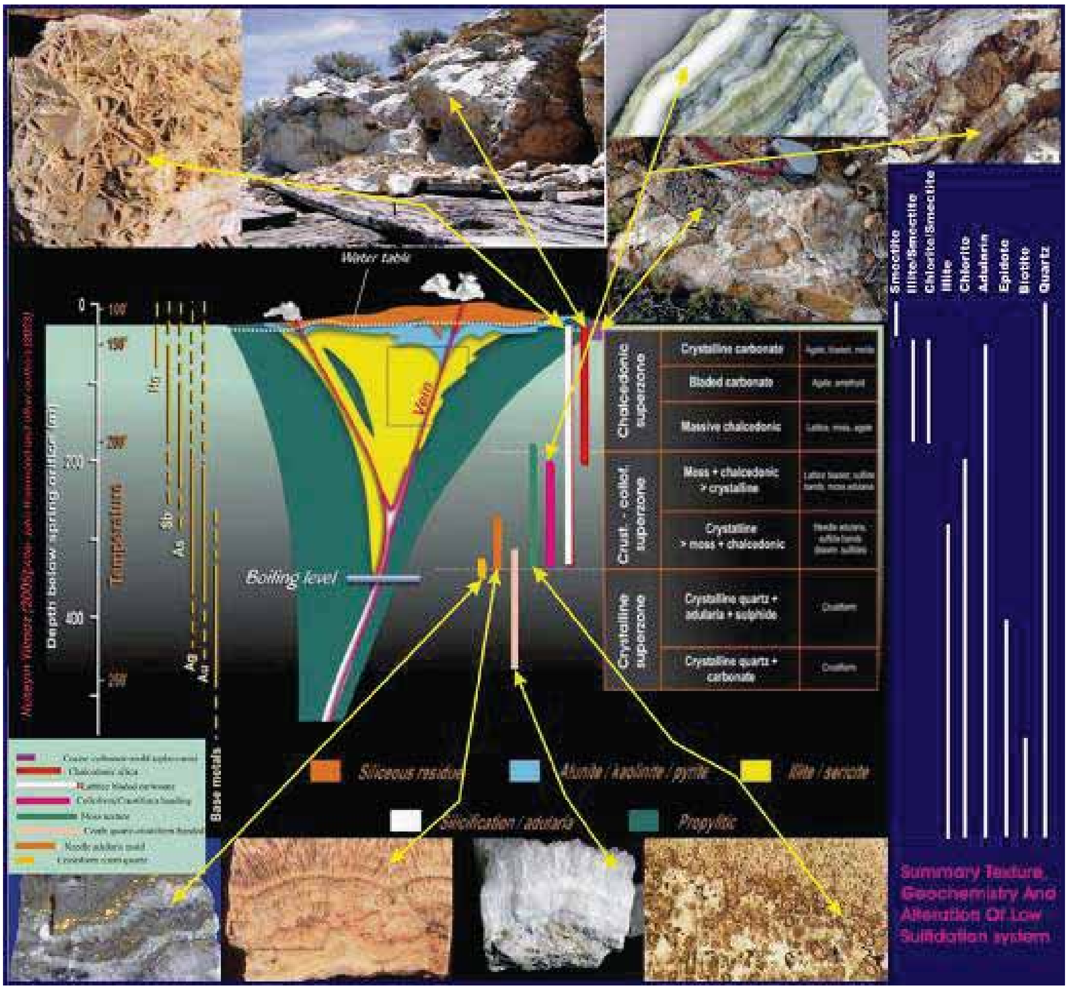


Figure 8-1
Geochemical zonation, quartz type and alteration patterns of low sufidation hydrothermal system (Hammond 2003)

Mineralization at COSE is also assigned to the low sulfidation type, based on the presence of fine-grained replacement quartz and adularia, widespread illite alteration, bladed textures indicative of hydrothermal boiling, and a mineral assemblage dominated by marcasite, arsenopyrite and silver-bearing sulphosalts. The presence of anomalous copper and molybdenum associated with higher grade Au-Ag mineralization suggests a component of magmatic-derived fluid.

The COSE deposit occurs predominantly as hydrothermal breccia, in combination with replacement, veinlet and disseminated styles of mineralization, rather than as one or more discrete quartz veins. This is somewhat atypical for the Deseado Massif deposits, perhaps reflecting a lack of open space during hydrothermal fluid flow.

9.0 MINERALIZATION

9.1 Regional mineralization

The Deseado Massif volcanic province hosts several producing and advanced stage projects (Table 9-1).

Table 9-1
Selected Gold-Silver Deposits of the Deseado Massif.
Data from Company Annual Reports.

Deposit	Inventory	Inventory Type	Operator	Status
Cerro Vanguardia	3.73 M oz Au 68.9 M oz Ag	2008 Resources (also past prod'n of 2.5 M oz Au)	AngloGold –Ashanti	producing
Martha Mine	0.06 M oz Au 15.0 M oz Au	2008 Resources	Coeur d'Alene Mines	producing
Manantial Espejo	0.75 M oz Au 583 M oz Ag	2008 Reserves	Pan American Silver	construction
Cerro Negro Project	3.1 M oz Au 25.0 M oz Ag	2010 Resource	Andean Resources	advanced exploration
San Jose	0.69 M oz Au 44.76 M oz Ag	2008 Resource	Minera Andes	producing

Exploration drilling by PGSA has been conducted within an approximate seven kilometer radius of the COSE Project area at eight prospects; Cap Oeste, Cap Oeste extension, Puma, Felix, Pampa, Vetas Norte, Don Pancho and Breccia Valentina (Figure 9.1.1). Precious metal mineralization at these prospects exhibits a range of deposit characteristics:

- silica poor, iron oxide - sulfide replacements;
- silica-rich, sulfidic hydrothermal breccias; and
- chalcedonic silica veins.

Precious metals in general are strongly correlated with anomalous concentrations of As, Sb, Hg, Cu, and occasionally Mo. Overall, mineralization intersected at the Pampa, Cap Oeste and Cap Oeste Extension prospect areas, which are located along the strike continuation of the COSE/Bonanza Fault system, is very similar in terms of geochemistry and structural control to that at the COSE Project.

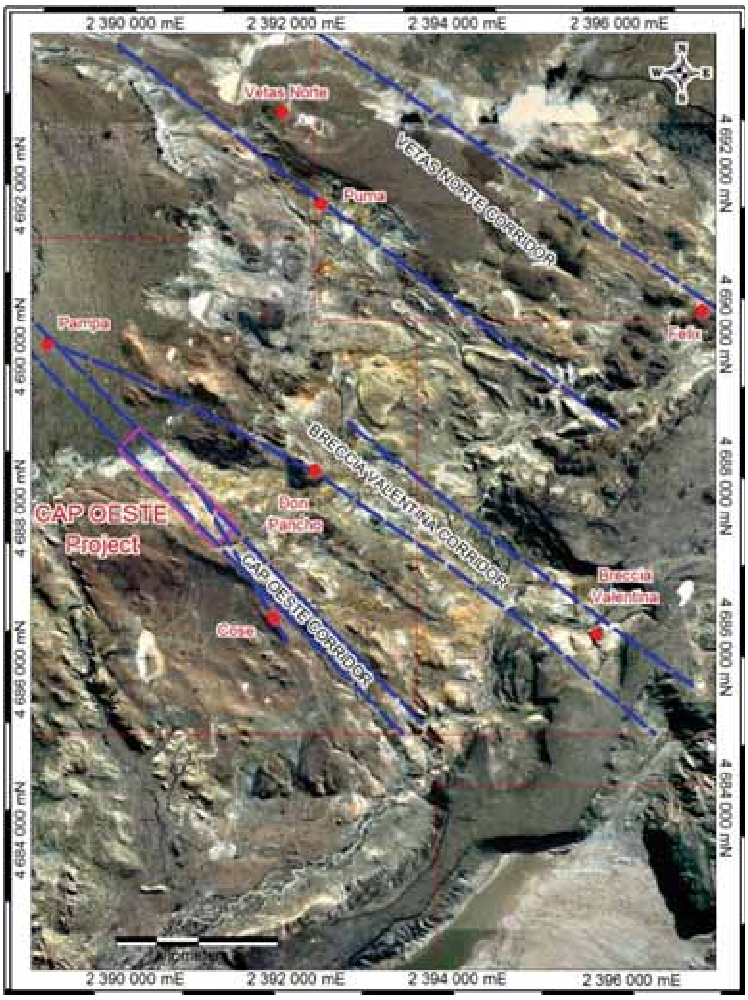


Figure 9-1
El Tranquilo Block Illustrating Exploration Prospects

9.2 Property Mineralization

9.2.1 Description and Distribution

As described in Section 7, Au-Ag mineralization at COSE is hosted by the northwest- trending COSE Breccia Fault, which dips 75 - 90 degrees to the southwest. Drilling has therefore been orientated towards the northeast (050° true north) along grid lines orthogonal to a baseline trending 140°. Based on interpretation of drill cores, the COSE Breccia Fault extends along strike for at least 65 meters to either side of the COSE ore shoot.

Two main styles of mineralization are apparent in drill cores from the COSE prospect. Highest grade Au-Ag concentrations are hosted by a distinctive suite of sinuous to weakly bifurcating breccias, composed primarily of argillic altered fragments of volcanic host rock in a matrix of fine grained grey quartz, illite, sulfide, and carbonaceous material. Precious metals occur as native metal, alloys and sulfides, in close association with base metal sulfide minerals. As described in Section 7, the high grade breccia shoot exhibits a northwest strike with steep, southwest dips, consistent with intersecting, steeply dipping fault surfaces. Certainly the localization of breccia at a structurally juxtaposed lithologic contact seems to require the existence of at least one precursor fault, although there is no compelling evidence for deformation of the mineralised rocks during and after the deposition of sulfides.

The immediate hangingwall and footwall rocks to COSE breccias exhibit lower grade mineralized envelopes (Figure 11.5.2). On the footwall, there is typically a 3-10 meter wide zone of millimetric scale (2-3 veins per metre), sub-vertical veinlets. These veins are silica- poor and dominated by illite and sulfide. On the hangingwall, pervasive silicification of host volcanic rock is accompanied by sheeted chalcedonic veinlets with clay-sulfide selvages. Veining is best developed in more competent lithologies (vitric quartz and quartz crystal rich tuff) and generally shows poor continuity in both grade and thickness down dip and along strike.

The high grade COSE ore shoot apexes approximately 120 meters below surface and is therefore blind. The upward projection of the shoot comprises a series of sheeted veinlets similar to the hangingwall mineralization described above. This broad interval of significant Ag-Au anomalism provides a useful exploration vector to focus deeper drilling for additional high grade targets. The rapid downwards transition from low grade veinlet style mineralization to high grade breccia may be due in part to stratigraphy: the more disperse, vein style of mineralization favored by more competent quartz- rich tuff in comparison to the stratigraphically lower, quartz -poor, ash tuff which typically hosts breccia style mineralization (Figure 9-2).

To date, drilling has focused on defining the dimensions of the high grade shoot, assisted by composite grade x width long sections (Figures 9-3). These long sections were generated from the mineralized intersections. Potentially ore grade mineralization was defined at a minimum cutoff grade of 1.0 ppm Au or 70 ppm Ag. Drill intervals comprising low to anomalous Au-Ag values in zones peripheral to the main shoot but which indicate continuity of mineralization along the COSE Breccia Fault, were used to complete the pierce points on the respective long sections. Pierce point intersections of vein dominant, hangingwall and footwall mineralization were also modeled as individual shapes, thematically colored for the respective grade/widths.

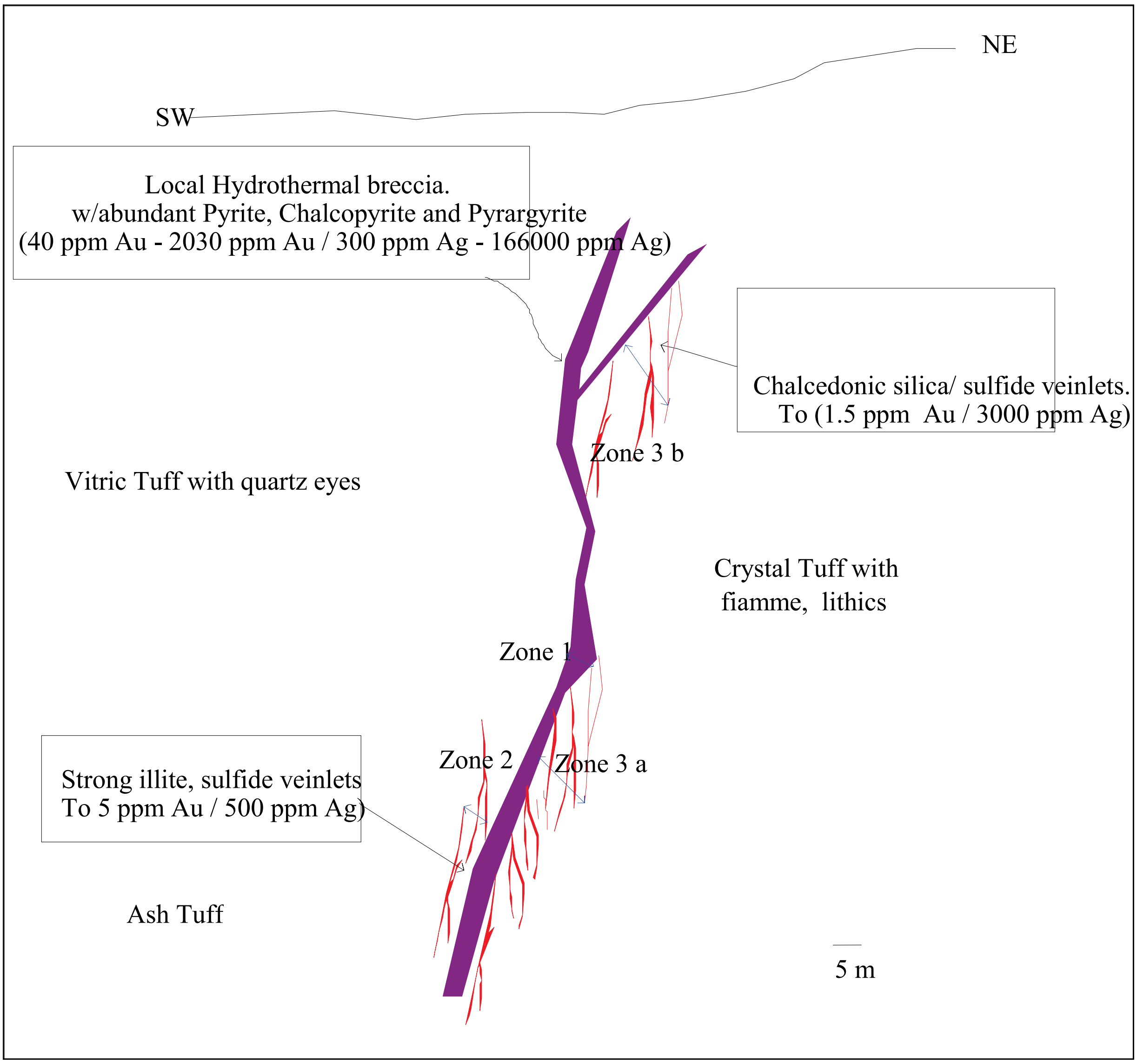


Figure 9-2
COSE Schematic Mineralized Section

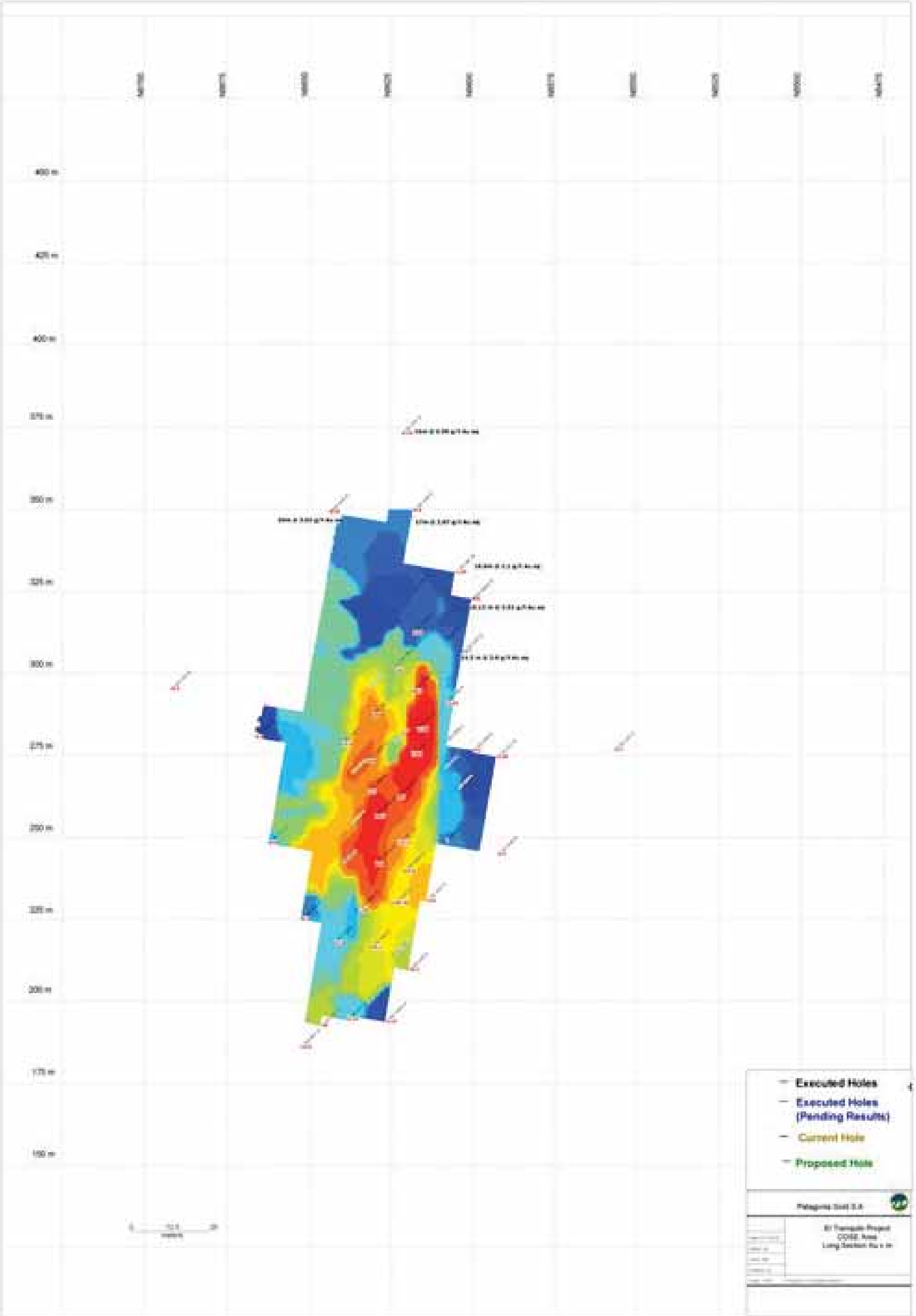


Figure 9-3
COSE Longitudinal Section- Au equivalent ppm x metres

As shown in Figure 9-3, the longitudinal section for Au equivalent gram x meter displays the combined, broadly enhanced continuity of the shoot with a widening of precious metal concentrations in the upper levels.

9.2.2 Mineralogy and Paragenesis

Mineralogy of oxide and sulfide assemblages has been described from hand specimen examination of drill cores, supplemented by a petrographic report on four polished thin sections (Ashley, 2010). Complete oxidation extends down to generally 30-40m vertically below surface peripheral to the main COSE Breccia and COSE Faults but extends to depths in excess of 120 meters within narrow, limonite-hematite rich faults and fractures.

Two stages of alteration are proposed based on these studies. Early potassic alteration affected the groundmass component of the volcanic host rocks, leading to replacement by fine grained K-feldspar (e.g. adularia), quartz, and minor disseminated pyrite, arsenopyrite and trace rutile. This would have had a hardening affect on the host rocks, prior to faulting and ore deposition.

Argillic alteration overprinted the earlier-developed potassic alteration, by replacement of K-feldspar with abundant, fine grained "illite-sericite" and in some cases, kaolinite. Argillic alteration is characterized by small amounts of disseminated pyrite and arsenopyrite, and in a few samples, traces of associated marcasite, chalcopyrite, and sphalerite. In places, argillic alteration is also accompanied by deposition of silica with subordinate adularia, and it is this later event which appears to be most directly related to the formation of hydrothermal breccias and veins. Two types of hydrothermal breccia matrix are observed:

Sulphide-clay-carbon rich tuffaceous material dominates "soft matrix" breccias with little or no accompanying quartz (Photos 9-1, 9-2). The origin of the carbonaceous material remains enigmatic; it could have been mobilised from precursor source rocks, or it could represent an inorganic precipitate via redox reaction between CO_2 and CH_4 in the hydrothermal fluids.

Quartz-rich, "hard matrix" breccias are characterized by silica textures ranging from finely granular to drusy to prismatic. In places small cavities are lined with quartz, pyrite, and arsenopyrite; carbonate (in places bladed), illite, and K feldspar occur locally.

Assay data for the suite of petrographic samples indicate that the highest grade precious metal values occur in association with the clay-rich breccias; however, significant precious metal grades occur as well in the quartz-rich variants.

Photo 9-1
Cose Breccia (DDH-CSE-013D) exhibiting semi massive chalcopyrite and pyrargyrite- proustite, associated with pyrite, marcasite, and arsenopyrite

Photo 9-2
"Soft matrix" Cose Breccia (DDH.CSE-013D) exhibiting sulfide-carbon rich infill; blue staining is ilsemannite

Sulfides are dominated by pyrite and arsenopyrite; fine grained; composite aggregates of the two minerals are common and in many samples pseudomorph a former bladed/prismatic phase, likely carbonate or pyrrhotite. Precious metal and base metal sulfides are trace to minor components of many samples and textural relationships imply that they were deposited after pyrite and arsenopyrite. The precious metal phases and base metal sulfides also occur in composite aggregates, in close association with pyrite-arsenopyrite (Photo 9-3, 9-4). The most abundant precious metal phase is the ruby silver mineral suite, proustite-pyrargyrite. Exceptional concentrations of these minerals to five volume percent correlate with intervals reporting values up to 402 g/t Au and 23,341 g/t Ag (DDH CSE-13:209.7 m). Gold-electrum occurs in composite aggregates with sulfides and as inclusions in gangue minerals (Photo 9-5). Elevated As correlates with modal arsenopyrite, arsenian pyrite and proustite-pyrargyrite. High Mo grade intervals are characterised by a bluish stain upon relatively rapid oxidation in air, likely due to the presence of ilsemannite ($Mo_3O_8 \cdot n(H_2O)$.

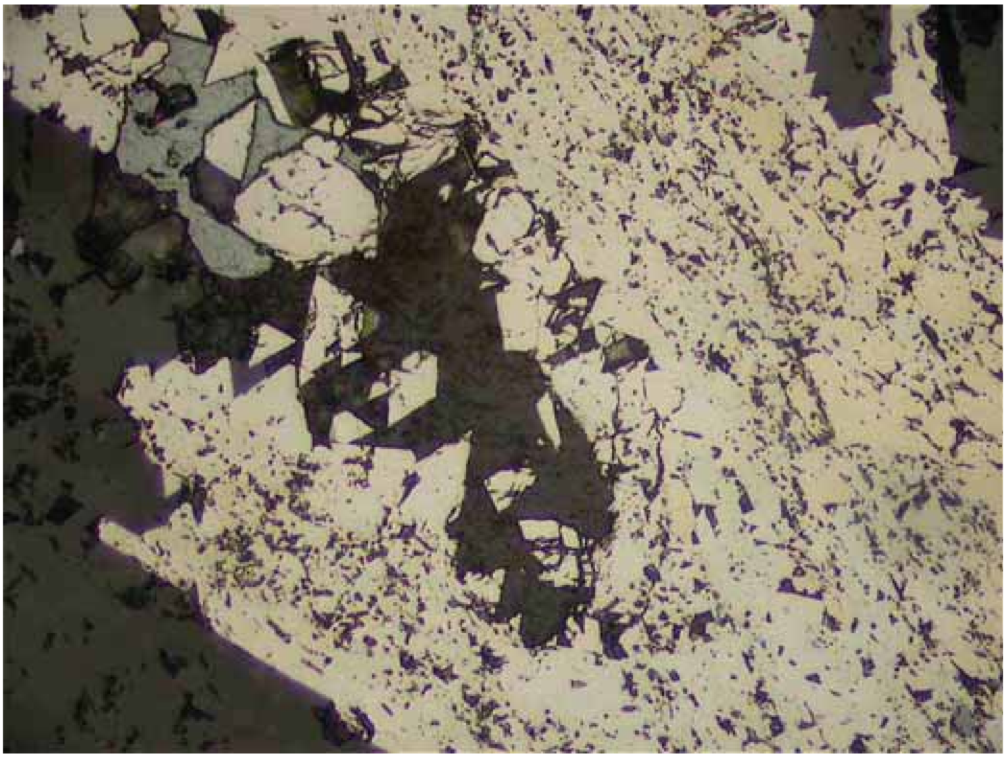


Photo 9-3
DDH CSE-13 (208.3m). Portion of a large sulfide aggregate showing intergrown pyrite (pale yellowish), with arsenopyrite and marcasite (both off-white) and at upper left, a small cluster of proustite-pyrargyrite grains (mid-grey).

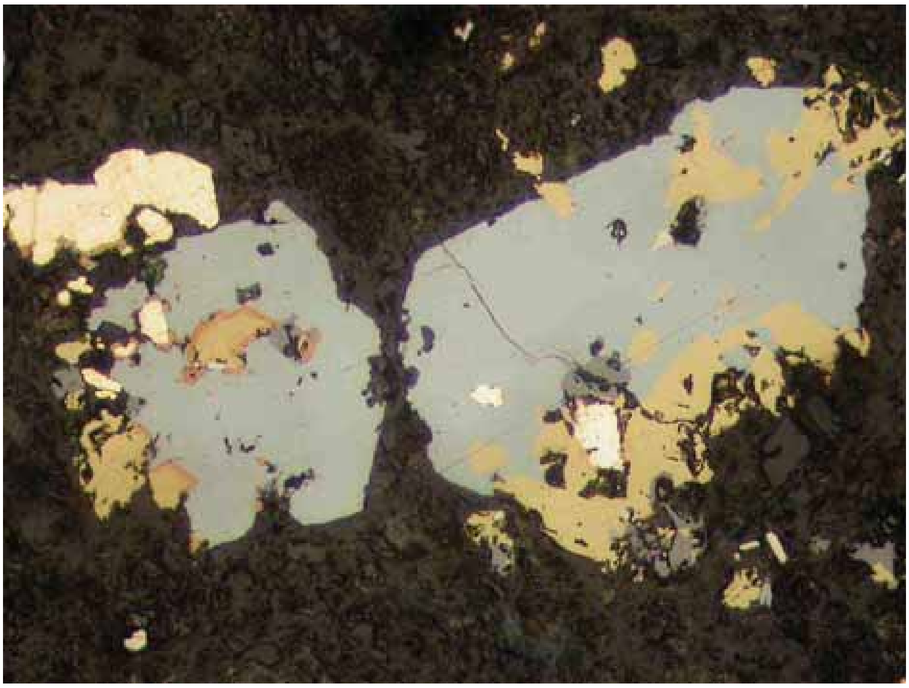


Photo 9-4
DDH CSE-13 (209.7m). Composite aggregates of proustite-pyrargyrite (pale blue-grey) and chalcopyrite (yellow), with bright creamy grains of gold-electrum in illite-sericite -quartz gangue.

9.2.3 Controls on Mineralization and Ore Fluid Paragenesis

Although high-grade mineralization tends to be relatively quartz-poor, adjacent, intensely silicified rocks in the hangingwall are considered as integral parts of the mineralizing event. Sillitoe (2008) has postulated that silicification and associated stockwork development may have occurred early on, with the stockworks the product of fluid overpressuring and release into the overlying hanging wall of the COSE Breccia Fault. Deposition of high-grade mineralization is considered to have overlapped with or immediately followed the main silicification event, potentially in multiple, discrete stages. As is the case at Cap Oeste, the ore-bearing fluids were focused along the footwall side of the silicified zone, resulting in intense illite-sericite alteration. The source of the fluid may have been felsic magma similar to that which formed rhyolitic domes a few kilometers distant at Breccia Valentina (Sillitoe, 2008).

9.2.4 Exploration Potential

CAM considers the exploration potential throughout the immediate COSE Project Area to be high, and to include the following targets:

- Down plunge extensions to the COSE Breccia shoot

- Strike extensions to the upper portion of the COSE breccia system, which could constitute broad intervals of low grade, oxidized mineralization with higher Ag/Au ratios (see Figure 11.5.2)
- Repetitions of the COSE orebody along the COSE Breccia Fault and/or the COSE / Bonanza Fault system.

10.0 PGSA EXPLORATION PROGRAM

Upon signing the purchase agreement with Barrick (February 5, 2007) Patagonia Gold S.A. began exploration activities throughout the El Tranquilo claim block. During November-December 2008 the first trenching and mapping was conducted at COSE, to be followed up by drilling programs in 2009 and 2010.

Work completed to date includes:

- IP/resistivity surveys (twenty 200m spaced, 1.2km long lines totaling 24 line kilometer gradient array-Block 2).
- Geologic mapping at 1:2,000 scale.
- Rock chip and sawn channel samples (57 samples).
- Excavation and sampling of a total of forty trenches (totaling 1,110 meters and 245 channel samples) and excavation of 15m x 65m area for a total of 25 samples.
- A total of 10,280 m in forty three drill holes comprising:
  - Two RC drill holes (totaling 300 meters) averaging 150m in depth and totaling 152 samples.
  - Forty one HQ diamond drill holes (9980 meters averaging 243.4m) and a total of 3438 samples.
- Petrographic analysis of four HQ drill core samples.
- Surveying of all drill hole and trench locations in x, y, and z dimensions with a differential GPS.
- Topography surveying with a differential GPS and generation of topography contour map.

10.1 Gridding, Topography and Surveying

The applied local grid at COSE is tied into the same grid that extends from La Pampa and Cap Oeste. This grid is tied into the Gauss Kruger Projection and Campo Inchauspe Faja 2 datum coordinate system with coordinates recorded using a double frequency (L1 and L2), TOPCON Model GB-1000 differential GPS which generally gives precision of X=1 centimeter, y=1 centimeter and Z (altitude) =1.5 centimeter.

The same equipment was employed to survey trench and drill hole collar locations by a qualified surveyor in addition to providing both topographic control and contours. Topographic control was facilitated with the collection of coordinate and altitude data on a 10m by 10 m meter grid spacing over a 160 hectare area from which data points were subsequently contoured using triangulation parameters.

10.2 Trenching

In 2008, a total of five trenches totaling 148.3 meters were mechanically excavated in order to better define the subtle surface expression of the COSE Fault, originally identified in outcrop in the southeastern portion of the prospect area. A total of thirty seven, 0.5-1.5 meter wide samples were subsequently taken, focused on narrow, steeply dipping zones of fault gouge and crackle breccia. The most significant of these samples returned 0.18 g/t Au and 14.23 g/t Ag, accompanied by anomalous values of As (344 ppm) and Sb (45-73 ppm).

In order to better define the up-plunge geometry of the high grade COSE Breccia Shoot, a 15 meter by 65 meter excavation was made by backhoe and subsequently swept clean, grid staked, mapped, and channel sampled (Photo 10.4.1). This work defined a broad zone containing narrow hydrothermal breccias and gouge thought to be the up plunge projection of the COSE shoot, as well as the subparallel trace of the COSE Breccia Fault.

From a total of 25 samples, these structures returned no anomalous Au values but did report anomalous silver and pathfinder elements to 2.1 ppm Ag, 518 ppm As, 143 ppm Sb, 156 ppm Hg, and 16.4 ppm Mo. Although these elements correlate positively with the high grade mineralization at depth, results emphasize the very subtle character of mineralization at surface above the high grade ore shoot.

Photo 10-1
Excavation along the interpreted up-plunge expression of the COSE Breccia Shoot with annotated COSE Breccia Fault trace (red). Photo looks northwest.

A further thirty five trenches, totaling 960 meters, were excavated along the eastern margin of the IP chargeability anomaly that remains untested by drilling to the south east of the COSE resource area, in order to try and define the surface expression of other, concealed shoots. Mapping identified a series of subparallel, 3-8 meter wide zones of weak fault gouge and hydrothermal breccia, related to a 170 meter long flexure in the COSE Breccia Fault. The highest Au result from this trenching returned a 1.1 meter wide interval of 0.23 g/t Au, with anomalous Ag and As. This mooted extension of the COSE Breccia Fault will be one focus of exploration drilling during 2011.

10.3 Geophysics

Based on the observed correlation of Au-Ag mineralization with disseminated sulfides and varying degrees of silicification, and the effective application of regionally spaced, pole-dipole IP surveying by Barrick Gold, gradient array geophysical surveys were conducted as a potential tool for the detection of additional concealed mineralization at COSE. Data were collected on 25 meter spaced stations along

200-meter spaced lines over the entire COSE prospect area. The area covered by the survey is 1150 meters wide and extends approximately 800 meters to the north northwest and 3000 meters to the south-southeast respectively from holes CSE 001-002.

Given limited apparent effectiveness throughout the neighboring Cap Oeste Project area, no ground magnetic surveying was carried out at the COSE prospect.

10.3.1 Gradient Array Induced Polarization

The survey highlighted a continuous 270 meter wide x 1100 meter long zone of high chargeability, bounded on the northeast by the projection of the COSE Fault (Figure 10-1). The symmetry of the chargeability contours suggests a west-southwest dipping sulfide-bearing body consistent with the COSE shoot as defined by current drilling and trench mapping.

Based on the distribution of disseminated sulfides at depth and the effective depth penetration of the survey, the source of the chargeability response is likely the rhyolitic lapilli tuff unit, known to host disseminated pyrite and arsenopyrite. The survey also highlighted a linear north-northwest trending, 150 meter wide x 450 meter long zone of relative high resistivity, with its axis broadly coincident with the mapped outline of the compact, siliceous quartz crystal tuff unit (Figure 10-2).

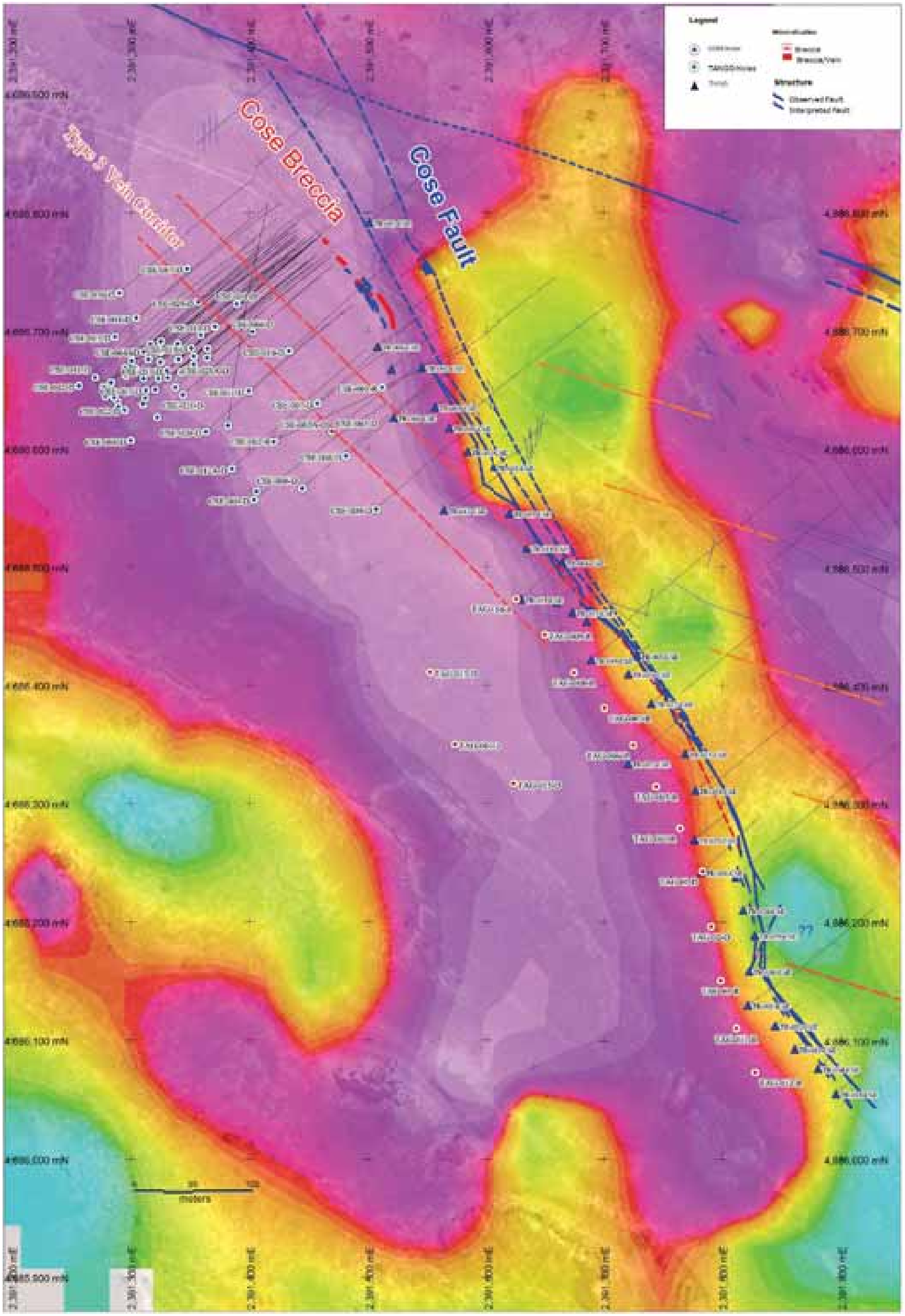


Figure 10-1
COSE Prospect-Plan map of IP Chargeability (Chargeability purple > green)

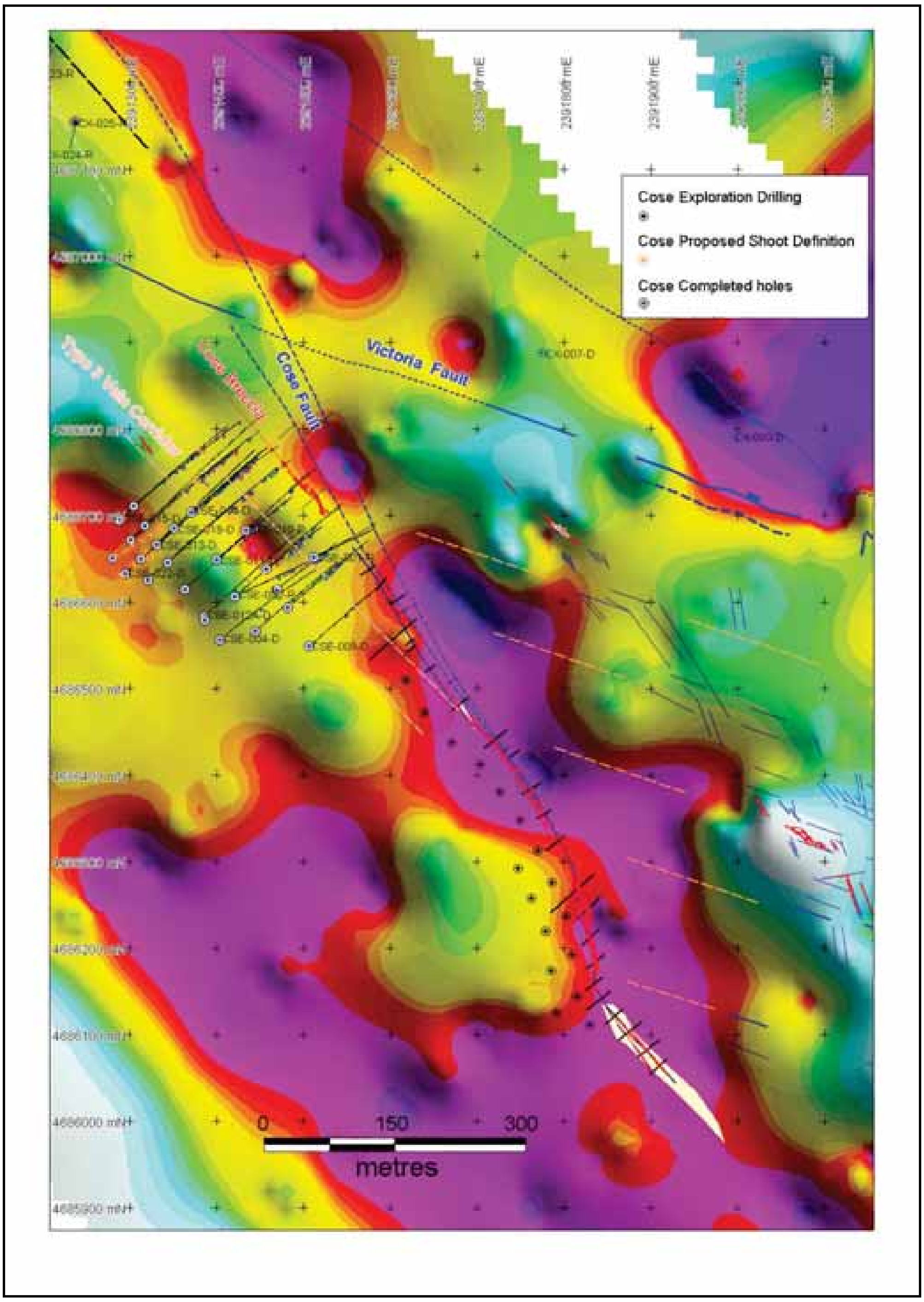


Figure 10-2
COSE Prospect-Plan map of IP Resistivity-(Resistivity: green > purple)

11.0 DRILLING

11.1 Introduction

Two drilling campaigns have been conducted on the COSE target; drill hole survey data are summarized Table 11.1 with collar locations shown in Figure 11.1.1. Drilling of reverse circulation (RC) and diamond holes (DDH) at COSE was carried out under contract by Major Drilling S.A, utilizing truck and track mounted Universal UDR 650 rigs.

Drill holes are identified as follows:

- Project- prefix CSE (COSE)
- Hole Number -(3-digit number)
- Hole Type - suffixes of R (RC) or D (DDH) – where a DDH hole was pre-collared by RC the hole suffix is DR
- For example: CSE-016-DR is COSE diamond drill hole #16 with RC pre-collar
- Abandoned/re-drilled Holes: In the case where a drill hole deviated significantly and was subsequently abandoned and re-drilled from surface the number of the new hole was the same but the suffix of the new hole included a 'A' or in the case of subsequent re-drilling 'B' e.g. COSE Abandoned hole CSE-152-D replaced by CO-152A-D and subsequently CSE-152B-D.

In 2008, two RC holes totaling 300 meters were drilled on Section 8488N inclined 50° as a stepback pair along an azimuth of 065° in order to intersect the COSE Fault and broadly transect the geophysical anomaly perpendicular to strike. During the period September 2009 - June 2010, a total of 41 DDH holes totaling 9980 meters were drilled between grid sections 8413N and 8663N, over a 250 meter strike length.

With the exception of one hole (CSE-026-D) all drilling was inclined at an azimuth of 050° at varying dips between 50°-70° in order to broadly intersect the overall structure orthogonally with increasing depth. Subsequent to the high grade intersection in hole CSE-013-D most drill holes were collared on nominal 12.5 meter spaced sections, extending approximately 40 meters to the northwest and southeast respectively in order to provide greater definition of the shoot.

11.2 Diamond Drilling Methods

Drill collars were located by hand-held GPS, in addition to triangulation from adjacent, previously drilled and surveyed collars. For each drill hole, the azimuth and inclination were set by PGSA geologists using a Suunto compass and Sola inclinometer. Generally the deeper holes were collared and aligned with respect to the target position taking into account the expected deviation over the hole length.

Diamond drilling was carried out under in 12-hour night and day shifts under supervision of PGSA trained technicians who tracked several parameters , including drilling time, reaming time, additives, core recovery, down hole survey information. Radio contact was maintained between the PGSA technician at the drill site and the PGSA geologists at base camp.

Table 11-1
COSE Project - Drill Collar Data

HOLE-ID	LOCATIONX	LOCATIONY	LOCATIONZ	LENGTH	AZIMUTH	DIP
CSE-012A-D	2391386	4686579	440	232.5	47.2	-49.6
CSE-013-D	2391330.776	4686666.376	440.434	234	46.9	-49.2
CSE-014-D	2391371.024	4686703.669	445.834	183	48.5	-50.1
CSE-015-D	2391287.018	4686694.683	439.199	282	48.7	-50.1
CSE-016-D	2391304.874	4686711.062	440.917	221.9	48.1	-50
CSE-017-D	2391313.554	4686649.625	438.902	291	45.1	-49.9
CSE-018-D	2391351.242	4686685.307	442.592	216	47.3	-50.5
CSE-019-D	2391316.82	4686688.632	440.426	252	48.3	-50
CSE-020-D	2391363.159	4686614.684	440.139	243	48.1	-49.7
CSE-021-D	2391343.951	4686645.576	439.902	255	47.8	-50
CSE-022-D	2391294.982	4686632.818	437.637	300	47.5	-50.3
CSE-023-D	2391276.556	4686653.952	437.721	315	45.3	-50.7
CSE-024-D	2391322.55	4686626.6	438.31	288	48.2	-49.7
CSE-025-D	2391277.877	4686653.426	437.728	309	47.2	-50
CSE-026-D	2391381.78	4686619.8	441.65	294	7.6	-49.5
CSE-027-D	2391323.345	4686659.588	439.803	300	47.3	-49.9
CSE-003A-D	2391469.566	4686614.942	454.617	135	52	-49
CSE-004-D	2391403.728	4686556.768	440.845	210	50.7	-49.5
CSE-005-D	2391420.536	4686606.241	445.227	201	49.2	-49.8
CSE-006-D	2391444.605	4686566.405	445.426	193	48.6	-49.5
CSE-007-D	2391457.423	4686638.515	452.914	177	47.9	-49.6
CSE-008-D	2391481.43	4686593.853	455.671	189	49.6	-49.4
CSE-009-D	2391506.568	4686548.913	455.166	165	48.7	-49.8
CSE-010-D	2391433.405	4686682.751	450.869	168	48.2	-49.9
CSE-011-D	2391399.541	4686649.03	444.674	174	48.5	-49.5
CSE-028A-D	2391339.394	4686675.013	441.288	246	47.9	-50
CSE-028-D	2391339.394	4686675.013	441.288	63	48.8	-50
CSE-029-D	2391356.84	4686724.75	445.64	181	47.7	-47.8
CSE-030-D	2391290.56	4686732.11	441.28	224	47.6	-47.8
CSE-031-D	2391389.41	4686723.02	448.83	158	45	-50.6
CSE-032-D	2391310.38	4686637.76	438.07	294	47.3	-49.9
CSE-033-D	2391288.771	4686640.936	437.494	303	46.5	-50
CSE-034-D	2391278.86	4686644.266	437.308	306	46.2	-52.1
CSE-035-D	2391289.749	4686636.012	437.441	317.6	48.1	-52.1
CSE-036A-D	2391287.076	4686644.382	437.617	330	47.3	-54.9
CSE-036-D	2391286.387	4686643.448	437.563	102	49.8	-55.3

Table 11-1 COSE Project - Drill Collar Data						
HOLE-ID	LOCATIONX	LOCATIONY	LOCATIONZ	LENGTH	AZIMUTH	DIP
CSE-037-D	2391277.286	4686653.111	437.713	321	47.7	-54.1
CSE-038-D	2391343.008	4686665.075	441.071	219	46.6	-50
CSE-039-D	2391353.26	4686671.76	441.732	207	49	-49.5
CSE-040A-D	2391365.36	4686678.13	442.97	207	49.7	-50.1
CSE-040-D	2391365.36	4686678.13	442.972	15	51.1	-50.4
CSE-041-D	2391270.028	4686660.641	437.657	315	47.9	-54.9
CSE-042-D	2391256.676	4686653.38	437.009	336	46.7	-55.2
CSE-043-D	2391364.65	4686684.78	443.387	177	49	-50.1
CSE-044A-D	2391310.968	4686659.552	439.034	253.4	48	-50
CSE-002-R	2391421.42	4686607.177	445.386	180	65	-50
CSE-001-R	2391511.642	4686652.122	457.382	120	65	-50
CSE-051-D	2391324.98	4686682.083	440.865	228	48	-50.5
CSE-050A-D	2391315.605	4686676.299	439.971	252	48.3	-50.5
CSE-048-D	2391303.905	4686668.386	440.559	270	48	-50
CSE-046A-D	2391284.277	4686656.48	437.915	291	48	-51.5
CSE-045A-D	2391294.306	4686648.938	439.46	288	48.5	-51.5
CSE-047A-D	2391296.687	4686662.153	438.572	291	48	-51
CSE-049A-D	2391300.361	4686665.129	440.284	282	48	-50.5
CSE-052A-D	2391339.935	4686689.124	442.048	219	49	-50
CSE-053B-D	2391332.14	4686660.592	440.274	235	48.4	-50
CSE-054-D	2391337.477	4686652.191	440.237	255	48.5	-50.8
CSE-055-D	2391320.835	4686649.937	439.425	255	47.7	-50
CSE-056-D	2391295.413	4686668.762	438.823	330	48	-50.5
CSE-057-D	2391325.283	4686673.361	440.076	249	48.5	-50.5
CSE-058-D	2391276.334	4686652.427	437.717	309	52.5	-53.8
CSE-012-D	2391385.177	4686583.174	440.584	15	50	-50
CSE-059-D	2391312.048	4686642.706	438.834	30	48	-49.1
CSE-060-D	2391353	4686678	438.946	303	49.3	-50.5
CSE-061-D	2391358.711	4686695.5	444.241	210.1	50.2	-50.8
CSE-062-D	2391325.819	4686690.099	441.724	234.1	49.5	-50
CSE-064B-D	2391310.28	4686680.47	440.258	264.1	48.2	-50.5
CSE-065-D	2391300.97	4686674.21	439.16	270	48.5	-50.5
CSE-063-D	2391277.35	4686653.06	437.72	320	49	-54
CSE-059A-D	2391312.17	4686642.826	438.946	282	48.6	-50.1

All diamond drilling was of HQ diameter and utilized a 3-meter core barrel where ground conditions permitted; in three cases were holes reduced to NQ. For diamond drilling conducted October 2009 - May 2010, a core barrel sleeve tube (HQ3) was deployed prior to entering the zone of interest in order to maximize recovery and rock quality. No orientated core was drilled during the course of the program.

Upon termination of each drill hole, down hole surveys were taken by the drill contractor every 25 or 50 meters utilizing a digital, single shot, FLEXIT down hole survey tool. The hole inclination, direction (azimuth), magnetic field strength, gravity roll angle, magnetic tool face angle and temperature were provided. Depending on the presence and depth of casing in each hole, collar survey photos were generally taken to within 12 meters of the collar. Each photo or series of drill hole orientation surveys were reviewed by both the drill contractor and the PGSA field technician on site, and subsequently recorded in both the drill contractors log and the respective section on the PGSA Drill Log sheet by the PGSA field technician. Holes that were found to be deviating significantly over the first 50m (i.e. more than 2 degrees inclination and or azimuth) were re-drilled (e.g. a total of 6 holes including CSE-03A-D, CSE-12A-D, CSE-18A-D, CSEO-28A-D, CSE-36A-D and CSE-40A-D).

Based on these data, there is a consistent tendency for the DDH holes to deviate clockwise to the south east averaging between 2-6o over 300 meters; the majority of this deviation occurs in the first 150 meters of the hole. Hole inclinations show a tendency to drop between 1-3o over the hole length.

Following termination of each hole, the collars were marked with capped PVC tubing cemented in a square concrete base, and surveyed by differential GPS.

11.3 Drill Core Logging

Core logging was carried out at Estancia La Bajada, approximately four kilometers from the COSE Project. Based on detailed geological mapping completed prior to the drill campaigns, a set of lithology, alteration, and mineralization codes were established and the logging methodology defined in order to standardize nomenclature amongst the geologists involved in the project. Geological information recorded during logging included:

- Lithology- rock type, grain size and composition;
- Alteration- mineral identification, especially type and intensity of clay and silicification;
- Structure – measurement of structural elements relative to the core axis;
- Mineralization type- breccia types, vein composition and widths, sulfide species and concentrations; and
- Oxidation–degree of oxidation of rock by weathering including oxidized/partially oxidized (transitional) and unoxidized.

High resolution digital photographs of each core box were taken by PGSA technicians and stored as a virtual core library in the PGSA drilling database. The logging process as conducted by the geologist involved the definition, marking and numbering of sample intervals on the core and core boxes; sample intervals were based on the above geological criteria in preference to meter by meter sampling. As a broad guide, minimum and maximum sample intervals of 0.5 and 1.5 meters were utilized. Exceptions to this rule were applied in zones of very low recovery where in rare cases several consecutive down hole meter intervals were composited in order to provide a critical mass of core material for analysis.

All the graphical and coded logs were recorded on paper at a scale of 1:100, or entered directly into a digital log template on a laptop computer; sample intervals and sample numbers were appended as defined by the PGSA geologist. This information was subsequently entered digitally into an access database and validated by both the PGSA technician and the geologist. All geological logging information was recorded on sectional plans on a continual basis in order to allow ongoing interpretation of the lithology and mineralization and compilation of a daily summary for PGSA management.

11.4 Reverse Circulation Drilling Methods

Reverse circulation (RC) drilling was conducted on a 12 hour per day basis during which the entire drilling and sampling process was supervised by a PGSA geologist on site. As stated previously, due to generally high water table levels and emphasis on achieving good sample quality all RC drilling subsequent to hole CO-010-DR was limited to the top of the water table, and thereafter diamond drilling was used.

During RC drilling, a 5 1/4-inch face return hammer was utilized and a PVC tube and sealed dust T box was installed at the collar with which to channel dust away from the drill area and prevent caving around the mouth of the hole. Individual one metre intervals were clearly marked on the drill mast which acted as a guide for the drilling contractors in sample collection. Subsequent to each six metre rod change, the hole was routinely conditioned and cleaned prior to the placement of the bulk sample bag beneath the cyclone for the sampling of the subsequent drill interval.

RC logging of sieved washed drill chips from each interval was accomplished on-site and contemporaneous with the drilling of each hole. Representative drill chips from individual on metre samples were saved in the respective marked chip trays.

11.5 Results of Drilling

A total of 12 geological sections were generated by PGSA geologists using Mapinfo/Discover GIS software, which incorporate interpreted lithological boundaries, zones of oxidation, mineralization, structural features and Au values. The locations of holes and sections in plan are provided in Figure 11-1 and a representative section is shown in Figure 11-2. Significant results received from all the drilling to date are listed in Table 11-2.

Table 11-2 Significant Drill Intercepts: High Grade COSE Breccia Shoot					
Hole No.	**SECTION**	**From m**	**Interval m**	**Au g/t**	**Ag g/t**
CSE-002-R	N8488	128	18	2.11	5.3
CSE-013-D	N8600	208.3	4.1	540.0	28088.9
Including		208.3	0.9	166.0	5464.0
Including		209.2	0.8	402.0	23341.0
Including		210	0.85	2030.4	106507.0
CSE-014-D	N8600	146.4	1.25	1.7	3156.0
CSE-015-D	N8650	247	20	1.4	2.7
Including		252.7	0.8	3.3	1.0
CSE-016-D	N8650	151.3	7.7	4.6	162.3
Including		157	1	15.8	37.2
and		163.8	1	18.7	346.0
CSE-017-D	N8600	241.5	20	9.5	685.2
Including		244.78	2.32	43.0	4930.7
CSE-018-D	N8600	170	1.5	2.4	701.3
and		188	2.55	1.6	171.1
CSE-019-D	N8625	179	1	11.1	1612.0
and		209.9	1	5.5	666.0
CSE-022-D	N8600	269.9	3.6	31.4	863.6
Including		269.9	1.25	80.1	2351.0
CSE-023-D	N8625	280	2.4	1.6	7.6
CSE-024-D	N8589	235.6	7.7	12.5	313.3
Including		240	3.3	23.9	373.0
CSE-025-D	N8625	283.96	1.74	6.2	56.0
CSE-027-D	N8595	214.17	13.93	159.2	626.6
Including		219.2	1.6	1284.2	3977.3
CSE-028A-D	N8600	188.6	22.7	2.6	355.8
Including			0.9	9.1	2872.0
Including			0.65	8.0	3115.0
CSE-029-D	N8620	138	1.7	2.1	515.0
CSE-030-D	N8670	157.3	4.5	4.0	345.2
and		162	1	5.8	216.0
CSE-033-D	N8600	269.5	5	17.6	334.8
including		269.5	2	36.3	689.4
CSE-034-D	N8600	283.6	2.8	18.4	266.9
CSE-035-D	N8600	285.08	3.12	16.9	237.0
including		286.6	1.6	27.0	409.9
CSE-038-D	N8580	199.9	2.1	13.0	601.1
and		207	4.5	4.3	41.5
CSE-040A-D	N8580	175.45	0.81	4.0	3210.0
CSE-041-D	N8623	297	2.1	18.1	619.1
including		298.6	0.5	72.5	1439.0
CSE-042-D	N8625	306.7	3.1	3.2	307.8

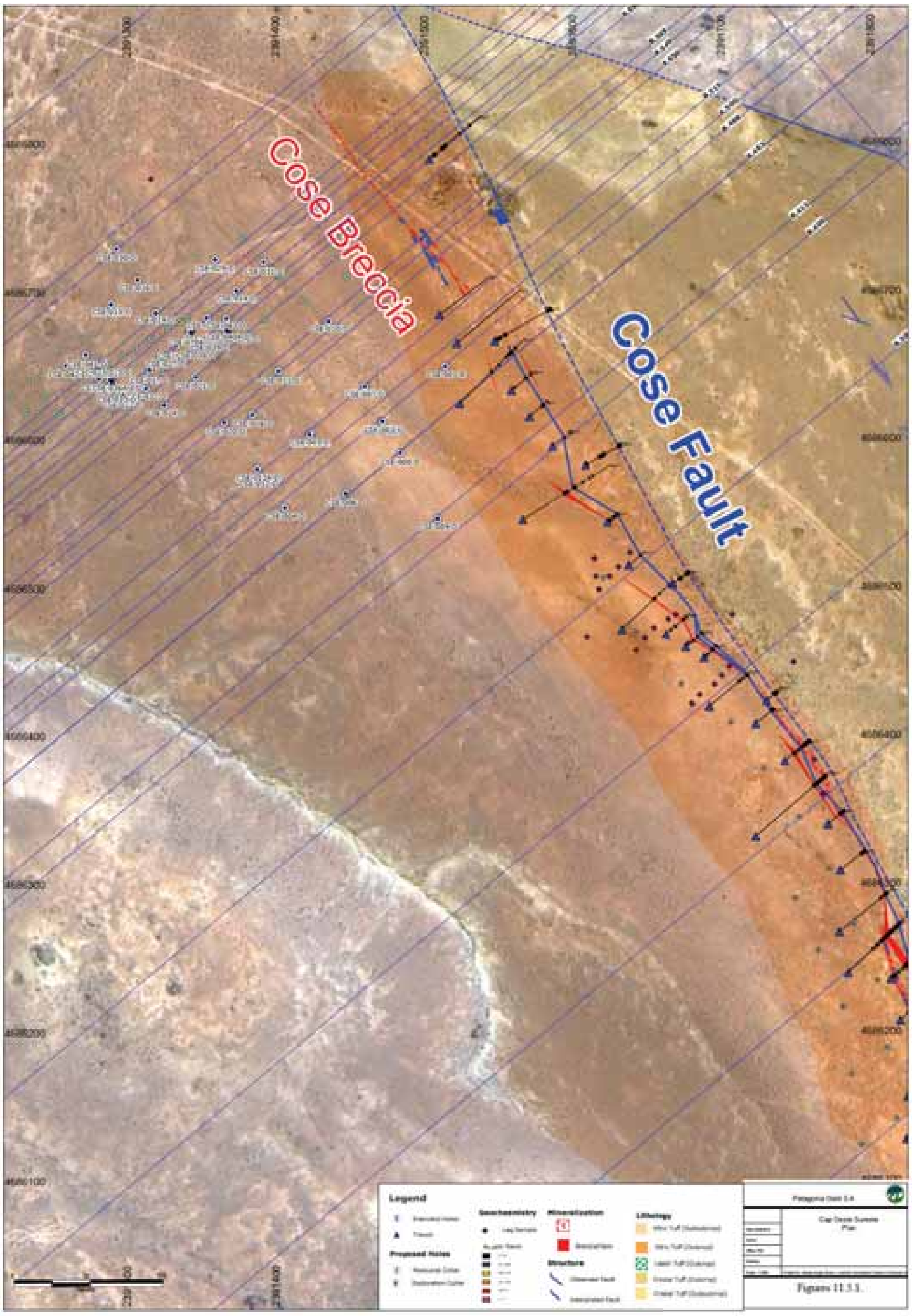


Figure 11-1
COSE Drill holes and sections

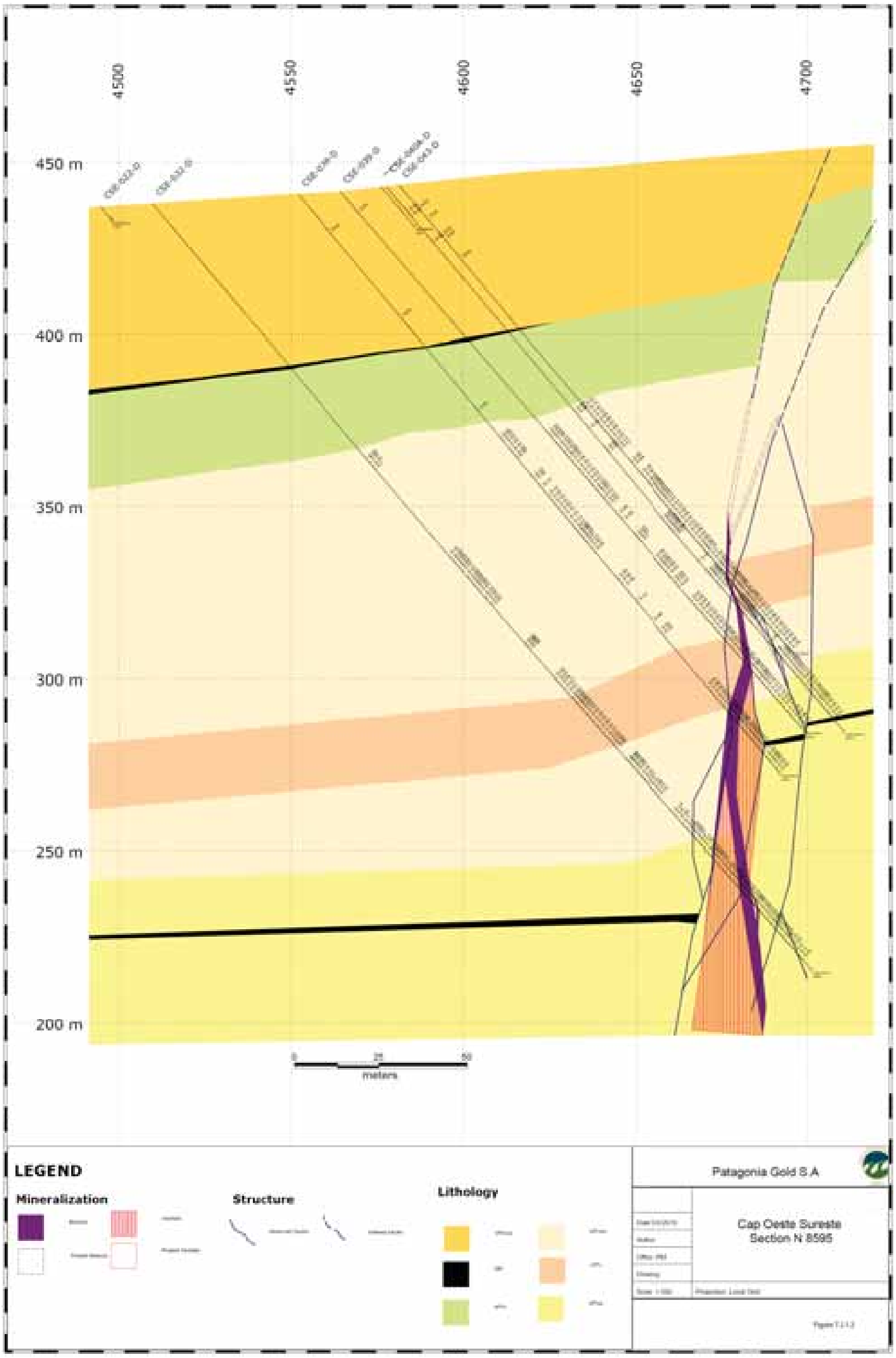


Figure 11-2
Section North 8595, illustrating the geometry of mineralization and its relationship to the COSE Breccia Fault.

12.0 SAMPLING METHODS AND APPROACH

12.1 Trench Samples

Trenches were laid out with Brunton compass and hand-held GPS. Topsoil removed by the backhoe excavator was stockpiled separately for later backfilling, and trenches were subsequently excavated down to bedrock to a maximum depth of three meters. The trenches were then cleaned and two parallel, five-centimeter by five-centimeter slots were mechanically dry sawn, cleaned, and sampled. Bags were tagged, sealed, and subsequently transported back to the base camp where each sample was weighed and recorded for final laboratory dispatch. Final surveying of the trenches position was completed using a differential GPS.

12.2 Reverse Circulation Sampling Methods

PGSA field technicians processed each one meter sample as follows:

- Weighing on-site of the sample and recording sample weight and type (e.g. dry, moist, wet).
- Riffle splitting to achieve a representative 4 kilogram sub sample which was bagged immediately in a plastic polyurethane bag (dry samples), or in polypropylene cloth bags (wet samples). Samples were weighed at various times during drilling for quality control.
- The rifle splitter was cleaned between each sample interval with compressed air sourced from the drilling rig. The cyclone was thoroughly cleaned between drill holes and every effort made to ensure quality control on-site.

In the case of wet RC drilling conditions, a rotary splitter was utilized in lieu of the conventional cyclone which allowed for a 1/8 and 7/8 split of the bulk one meter interval. Individual interval samples were taken from the 1/8 split portion of the splitter, placed in consecutively numbered lines peripheral to the drill platform and subsequently weighed when the excess water had drained through the pores of the polypropylene cloth bags. The wet splitter was thoroughly cleaned between each hole to minimize contamination.

12.3 Diamond Drilling Sampling Methods

During drilling, the diamond core samples were managed according to the following protocol:

- Extracted core was immediately placed in a core cradle, ensuring that core was maintained intact and in the correct order.
- Core was washed with segments "puzzled" together in order to reconstruct in situ position as much as possible. The vertices of any mineralized structures were preferentially aligned with the upper axis of the core.

- In combination with the drilling meterage blocks, as defined and provided by the driller, the PGSA technician calculated and marked the individual metre limits on the core.
- Recovery length and percentage of both the total drilled interval and each complete unit depth meter interval was calculated and recorded on the Drill Log sheet.
- Rock quality designation (RQD) for each core run was measured on the sum total interval of individual core pieces that measured over 10cm in the core run.
- Core was placed into numbered wooden core boxes in which meterage was marked, and wooden blocks inserted.

12.4 Drill Sample Recovery

12.4.1 Diamond Core Recovery

A summary analysis of the recoveries achieved in the different geological zones is shown in Table 12-1. Based on results from the 9,970 meters drilled throughout the program, overall diamond core recoveries averaged 98.7percent.

Table 12-1
Diamond Drill Recoveries

Geological Zone	Recovery DDH (%)
Oxide	98.4
Partial Oxide	98.7
Non Oxide	99.1
Mineralization Type	
Hydrothermal Bx	97.6

In order to maximize recoveries throughout mineralized zones of interest, the use of HQ triple tube drilling was implemented, from a point prior to intersection of the main mineralized breccias to the end of hole. Generally good recoveries were achieved for non oxide and partially oxidized mineralized zones, averaging 99.1 and 98.7 percent respectively. Slight core loss (average recovery 98.4 percent) occurred throughout the oxide zone, likely a product of the friable and clay-rich nature of mineralization.

The drill core recovery within the main COSE Breccia reported good but lower overall average recoveries (97.6 percent) the enhanced loss being a consequence of the commonly clay rich breccia matrix, fault gouge and fractured rock.

12.4.2 Reverse Circulation Sample Recovery

Average recoveries for the RC drilling sample intervals were calculated, as shown in Table 12.4.2. Recovery was calculated by dividing the dry weight per meter by the theoretical weight of the volume of rock per meter in which rock densities used were derived from the respective rock specific gravity values defined below in Section 12.7.

Theoretical sample weight/meter values utilized in recovery calculations for non oxide and oxide zones were calculated as follows:

- Oxide: 3.1417 (pi) x 0.066 sq (radius meters squared) x 2.1 (density) = 28.7 kg

The reverse-circulation drilling recoveries calculated for geological intervals are shown in Table 12-2.

Table 12-2 Reverse Circulation Drilling Recoveries	
Geological Zone	**Recovery (%)**
Oxide Zone	98.0
Partial Oxide	98.6

Drilling throughout the oxide zone yielded good average recoveries, with relatively small losses preferentially throughout the first 1 to 3 meters where supergene clay alteration is strongest and the presence of open space fractures is greatest.

12.5 True Width and Orientation of the Drill Target and Drill Intercepts

The overall form of the mineralized envelope at COSE in section is planar and broadly sigmoidal with an average dip of 55° southwest, with local variations between 40o and 80o. The holes drilled to test the zone (drilled 50o to 70o degrees towards the northeast), generally intersected mineralization at relatively high acute angles with respect to the core axis. Although no orientated core was obtained, these overall angles correlate with those recorded in the structural logging including fault planes, hydrothermal breccia fabrics and sheeted veinlets, relative to the core axis.

Given the consistent orientation of drill holes, the true widths of the intersected mineralization generally equate to approximately 80 to 95 percent of intersected widths. In a rare number of circumstances mineralization was intersected at a lower acute angle of 55o which equates to approximately 80-95 percent of the intersected widths.

12.6 Specific Gravity (Bulk Density) Determinations

Measurements of specific gravity (SG) were performed by Alex Stewart Assays and on site by PGSA on a total of 94 individual, 1/2 HQ core pieces from individual one meter drill core intervals, for which the average dry sample weight was 0.66 kilograms. The specific gravity for a total of 64 samples was measured by Alex Stewart Assays and 30 samples were determined by PGSA.

Based on the 9,970 meters of available core intervals, this sample set represents approximately 2 percent of the total sample population, drawn predominantly from zones of mineralization. The samples were systematically selected to represent all major lithological, alteration, and mineralization types with differing degrees of oxidization.

12.6.1 Specific Gravity Methodology

For specific gravity determinations of core conducted on site and at the Alex Stewart facility, intervals were first selected by the project geologist for specific gravity determinations from whole HQ core samples measuring at least 10 cm long and sufficiently robust so as not to break up or crumble during the measurement process. For each interval the project geologist recorded the relevant lithology, mineralization type and oxidized state information.

For specific gravity determinations conducted by Alex Stewart Assays, the chosen samples were securely packaged for dispatch to Mendoza and transported by a freight contractor directly to the laboratory. At the Alex Stewart Assays, laboratory in Mendoza core samples the procedure used included:

- Core samples were firstly kiln dried
- Each core sample is weighed = P1
- Each core sample is immersed in liquid paraffin and weighed = P2
- Each core sample is immersed in liquid paraffin and then immersed in a container with water and weighed = P3

The calculation used to calculate the SG of each sample is:

SG =P1/ (P2-P3 – ((P2-P1)/0.86*))
(*Specific weight of paraffin)

For specific gravity determinations conducted on site, the process was carried out by PGSA field technicians under the geologists´ supervision. Prior to weighing of the chosen core samples an aluminum alloy cylinder of known stable mass was weighed in both air and when submersed in water in order to provide a check that the scales were functioning correctly. No laboratory specified density has been assigned to the cylinders for direct comparisons of the respective known and calculated density using the immersion technique on site.

For each selected core piece, the dry weight was measured and subsequently the core was sprayed with hair spray and its weight when fully submerged in clean fresh water was recorded.

The specific gravity of each core sample was defined using the following equation:

SG = weight dry / (weight dry – weight submerged)

CAM reviewed the density database to ensure that the calculations were properly performed by recalculating specific gravity by weight in water and weight in air. Results of these checks by CAM indicate that the original calculations were correctly performed as disccuesed below in resourse estimation.

12.6.2 Specific Gravity Results

The range of SG values calculated for samples representing the complete range of lithologies, mineralization types and oxidization states are shown in Table 12-3.

Table 12-3
Summary of Specific Gravity Results

Zone	Oxide State	S.G. Mean	S.G Maximum	S.G Minimum	SG Std Dev.	No. Samples
All samples	NA	2.40	2.59	2.03	0.12	76
Mineralization Type						
Stockwork – Veinlets	Oxide	2.35	2.53	2.07	0.12	13
	Non oxide	2.44	2.57	2.22	0.12	9
Fault-Hydrothermal Breccia	Oxide	2.44	2.51	2.31	0.08	5
	Non oxide	2.40	2.59	2.03	0.13	15

The global average of all the specific gravity values based on all rock, oxidation and mineralization types for the 94 samples is 2.40.

With respect to the zones of mineralization, the highest average S.G values relate to the hydrothermal breccia, with an overall average of 2.44 in the oxide and 2.40 in non oxide respectively. The peak specific gravity values (2.59, 2.57, and 2.51) relate to intervals from weakly silicified breccia gouge with relatively high sulfide mineral concentrations.

12.7 Summary of Sampling

CAM are of the opinion that PGSA's drilling and sampling approach and procedures yielded samples of sufficient reliability to be appropriate for use in Resource estimation.

13.0 SAMPLE PREPARATION, ANALYSIS AND SECURITY

13.1 General Description

Sampling was performed on site, in the case of trenching and RC drilling, and at the Estancia La Bajada base camp, in the case of diamond core. Field technicians were given appropriate training and were supervised by a PGSA geologist. Care was exercised to eliminate sources of potential contamination:

- Wearing of jewelry was prohibited;
- Sample bags and core boxes were closed immediately upon the insertion/placement of the respective sample and kept above the ground surface on pallets;
- Care was taken during the transporting and processing of core samples, and the subsequent storing of samples and core boxes;
- Sample bags were kept in a dust-free environment and individual sample bags were stapled closed and maintained in burlap bags subsequent to sampling, which were immediately zip tied closed; and
- No sample reduction of any of type was conducted at the base camp other than the ½ splitting of the diamond core. The only sample reduction that took place in the field was the splitting of the RC samples (as described previously in Section 12).

CAM concludes the sampling methods employed in COSE drilling and trenching were carried out by PGSA to acceptable industry and NI 43-101 standards.

13.2 Trench Samples

As previously described in Section 10, trench samples were prepared and bagged in the field at the Cap Oeste Project area. Upon arrival at the base camp they were collectively bagged in burlap bags and subsequently labeled, zip-tied, weighed and recorded in the sample dispatch log, and stored ready for shipment.

13.3 Reverse Circulation Drill Samples

As previously described in section 10.2 and 12.2, the RC drill samples were collected and prepared at the drilling site. Other than packaging in sealed burlap bags, labeling, documenting and weighing prior to shipment, no other preparation was performed on the samples.

13.4 Diamond Drill Core Samples

As described in Section 12.3 and Section 12.4, drill core was placed in the core cradle and subsequently washed, orientated, marked and the recovery and RQD were measured and recorded.

In order to standardize sampling methodology and allow for reconstruction of the drillhole in 1/2 core, the convention of utilizing the left hand side of each cut core portion for subsequent geochemical analysis and the right hand piece to be retained as the reference core was applied. At the end of each sample interval, a perpendicular saw cut was made to clearly mark the end and beginning of the consecutive sample. During the cutting, the core sample intervals and corresponding numbers were repeatedly crosschecked.

Half core samples for individual intervals were placed in clean, tagged plastic sample bags which were immediately closed after sampling, and the corresponding interval in the core was marked with a stapled aluminum tag. After the individual samples were bagged they were placed in numbered burlap bags and subsequently weighed and recorded ready for transport. The marking, sampling, and bagging process was conducted by the PGSA field technicians under supervision of the Project geologist.

13.5 Storage and Transport

Samples pending shipment were stored onsite at Estancia La Bajada in a secure storage area and shipped weekly via a contracted private courier in a closed and locked truck compartment. The samples were transported directly to the designated laboratory in Mendoza, Argentina and were always accompanied by a required provincial transport permit in addition to a shipping dispatch and a letter addressing the particular analyses required, sample numbers, quantity and weights for the laboratory. The PGSA data manager was notified immediately upon reception of the samples in the laboratory by the laboratory staff.

13.6 Laboratories, Methods and Procedures

Alex Stewart Assayers Argentina S.A., which is an international recognized and accredited laboratory compliant to ISO Certified - 9001:2000 standards, was contracted for the geochemical analysis of the samples generated during the two drilling campaigns at Cap Oeste, and for exploration holes drilled outside the Cap Oeste Project area. ACME Labs of Vancouver BC Canada performed check assays on selected samples.

13.7 Quality Control

Quality control procedures conducted include the routine incorporation of certified geochemical standards, blanks and sample duplicates (RC percussion) which are submitted with geochemical samples to the laboratories and check assaying.

13.7.1 Geochemical Standards & Field Duplicates

Quality control measures implemented during the trenching and drilling programs included the submission of a series of certified standard and blanks, which were incorporated and dispatched with the drill samples, according to the following protocol:

- **Diamond Drilling:** alternate insertion of a laboratory certified laboratory standard or blank for every 10th sample.
- **RC Drilling:** For every 10^{th} sample, a duplicate sample of the preceding interval was taken as a field duplicate, or a certified laboratory check standard or blank sample was submitted respectively.
- **Trenching:** For every 10^{th} sample, a duplicate sample of the preceding interval was taken as a field duplicate, or a certified laboratory check standard or blank sample was submitted respectively.

A total of 263 individual standards, with a range of certified grades between 0.03 and 47.24 ppm Au and 0.5 to 462.7 g/t Ag, were submitted with drill samples for quality control along with 129 blanks and 8 duplicates. The analytical results for each individual standard were plotted against three upper and lower limits defined by + 2 and + 3 standard deviations from the respective certified value, in addition to +10 percent relative variance from the value.

All Au standards returned within the + 3 standard deviation limits of the laboratory certified value with 18 exceptionst. For each of these failures, five of the adjacent drilling samples within the batch were reanalyzed. As part of these rechecks, a total of 88 pulps were re-analyzed for Au, together with a total of 11 standards.

The results for the original and recheck pulps show good correlation within + 10 percent (Figure 13-1) and all the standards that were included likewise returned values within + 10 percent of the certified values. Based on these results, PGSA geologists believe that the original standards which indicated large variation from the expected values were either erroneously submitted and/or recorded, or somehow contaminated during preparation and handling.

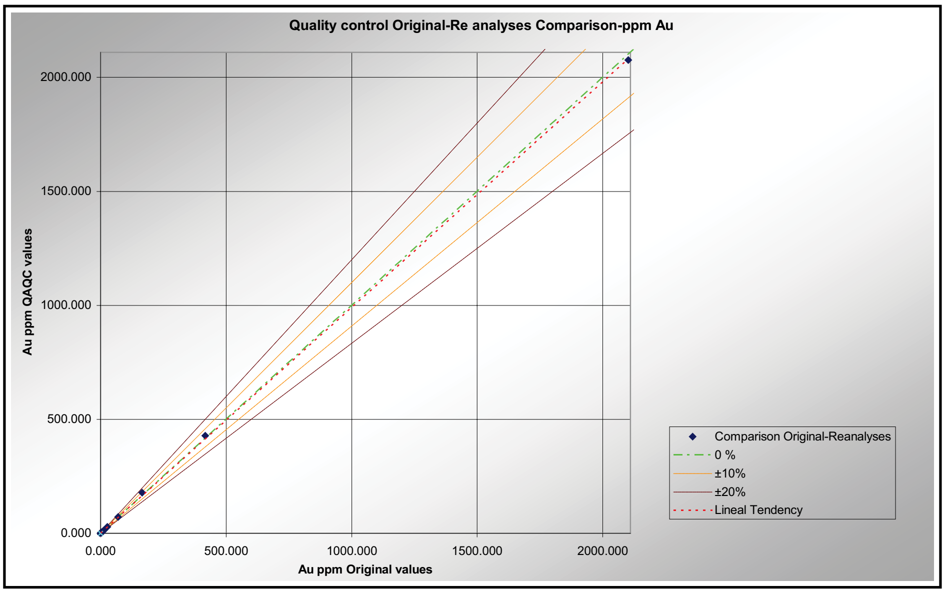


Figure 13-1
Scatterplot results of the reanalysis of 88 drill intervals comprising individual intervals consisting of five of the adjacent drilling samples within the batch, relative to each of the 18 submitted standards

Field Duplicates

The eight field duplicates taken from RC chips showed acceptable (+10 percent) correlations (Figure 13-2).

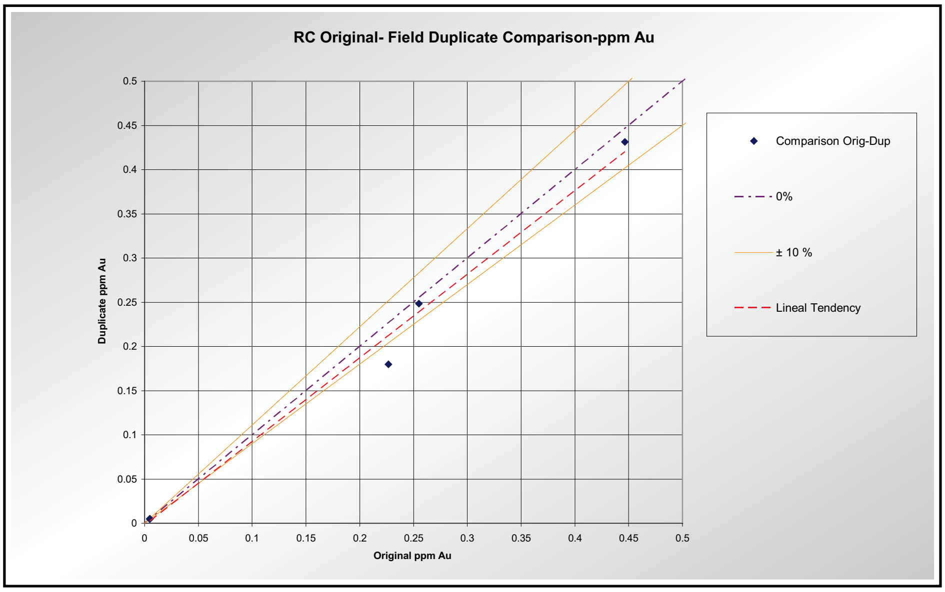


Figure 13-2
Field duplicate comparison CSE-001-R & CSE-002-R

Check Assays

A total of six batches of check assays were conducted for Holes CSE-001-R to CSE-43-D, which overall comprises approximately 5 percent of the total drill sample interval population. Resubmitted samples consisted of pulps (69 samples) and coarse rejects (125 samples), drawn primarily from mineralized intervals; all check assay submittals were accompanied by laboratory certified standards.

These samples were submitted to:

a) Alex Stewart Assayers S.A (32 pulps, 55 coarse rejects, 35 standards)
b) Acme Laboratories (37 pulps, 70 coarse rejects, 29 standards)

Results reported by Alex Stewart are shown as scatter plots in Figures 13-3 – 13-6 (pulps) and in Figures 13-7 to 13-0 (coarse rejects). Final results for several of the standards from these batches are still awaited.

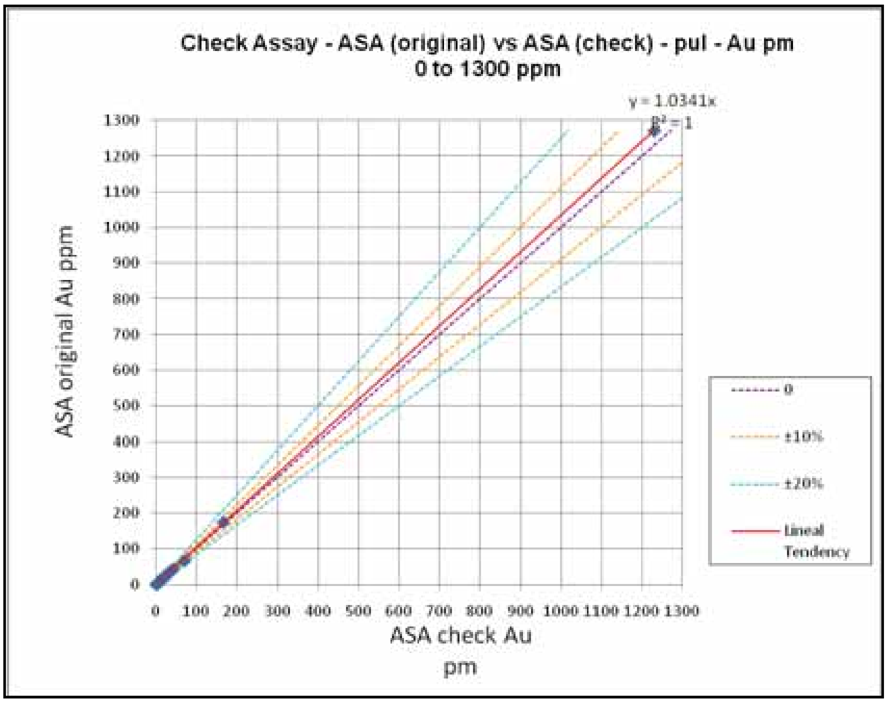


Figure 13-3
Check Assays- Alex Stewart- Pulps Au values between 0-1300 ppm Au

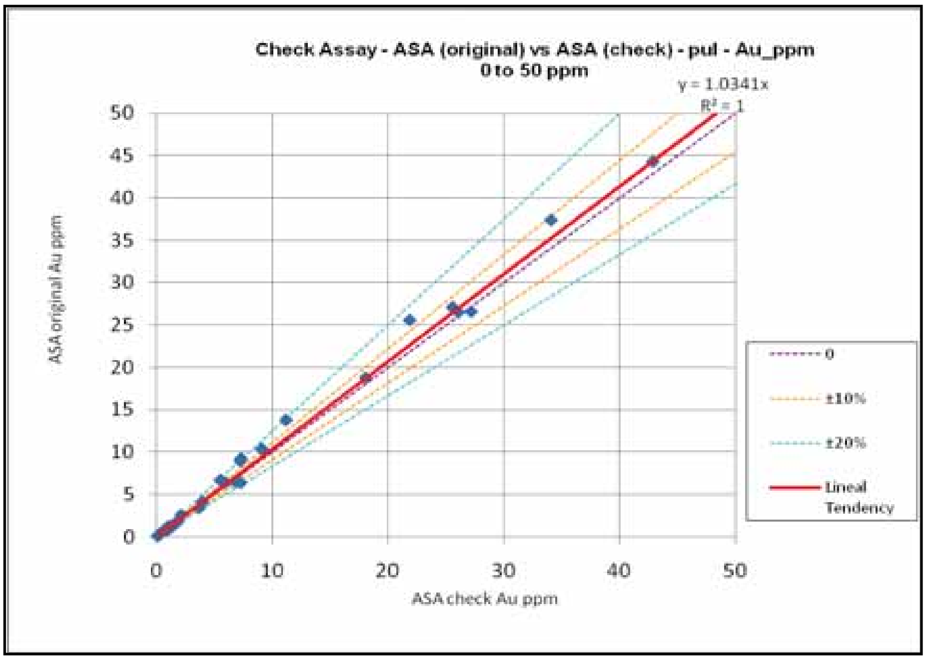


Figure 13-4
Check Assays- Alex Stewart- Pulps Au values between 0-50 ppm Au

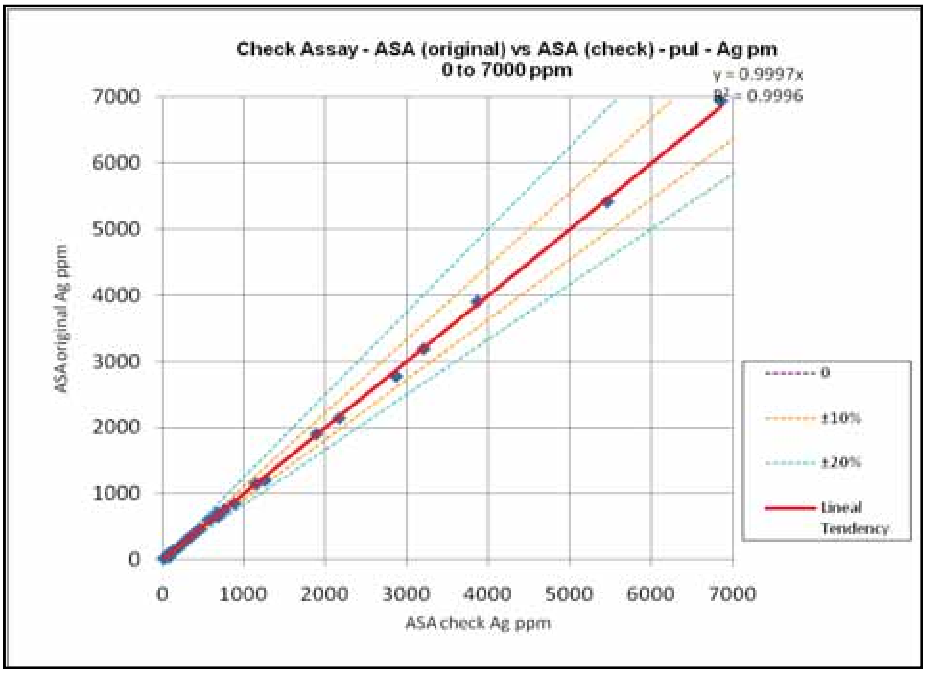


Figure 13-5
Check Assays- Alex Stewart- Pulp values between 0-7000 ppm Ag

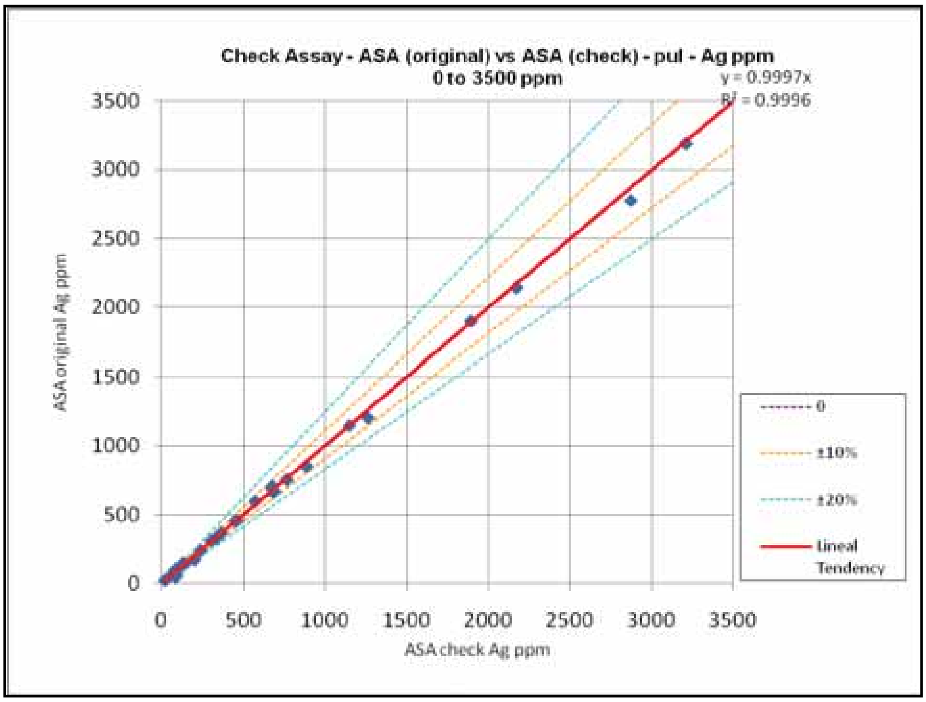


Figure 13-6
Check Assays- Alex Stewart- Pulps Ag values between 0-3500 ppm Ag

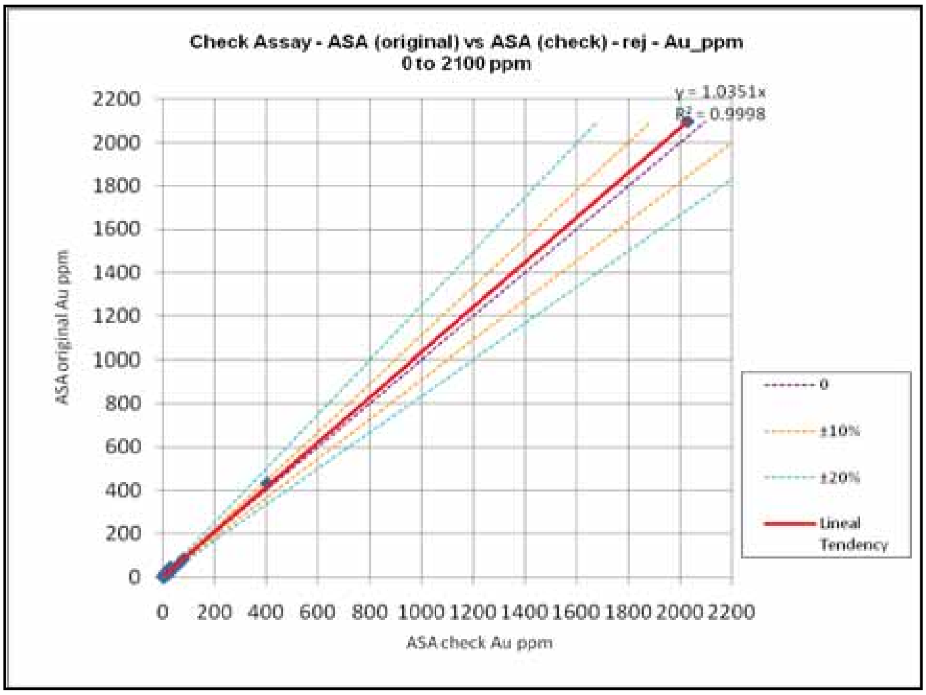


Figure 13-7
Check Assays- Alex Stewart- Rejects Au values between 0-2200 ppm

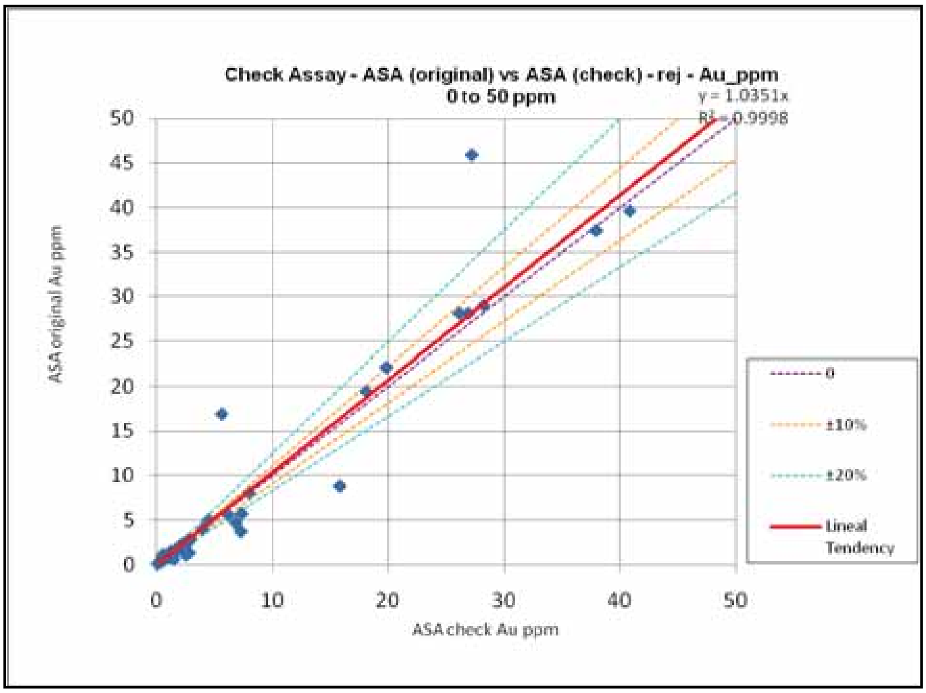


Figure 13-8
Check Assays- Alex Stewart- Rejects Au values between 0-50ppm

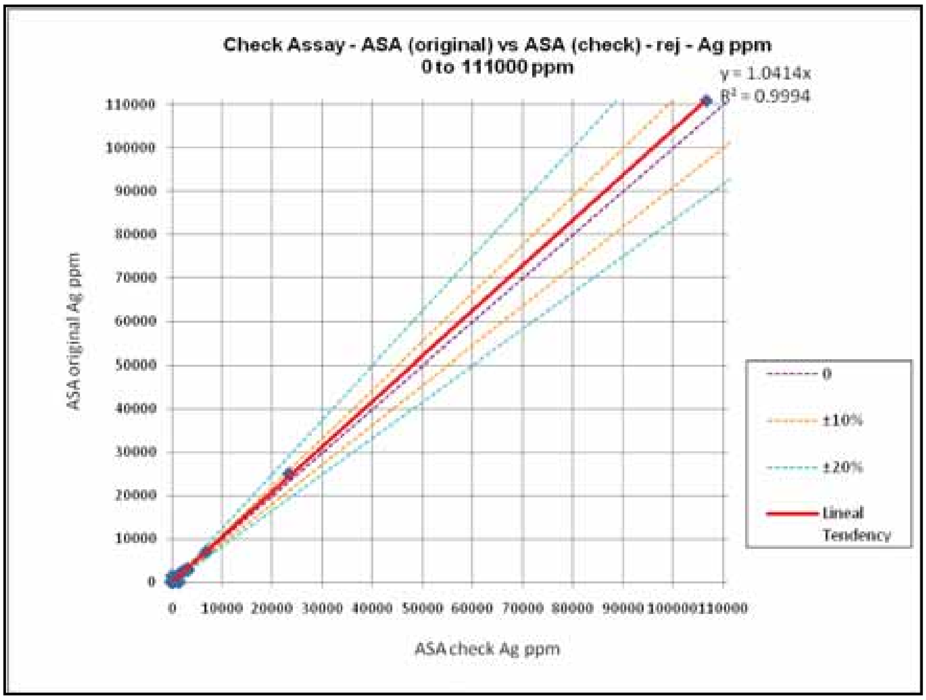


Figure 13-9
Check Assays- Alex Stewart- Rejects Ag values between 0-111000ppm

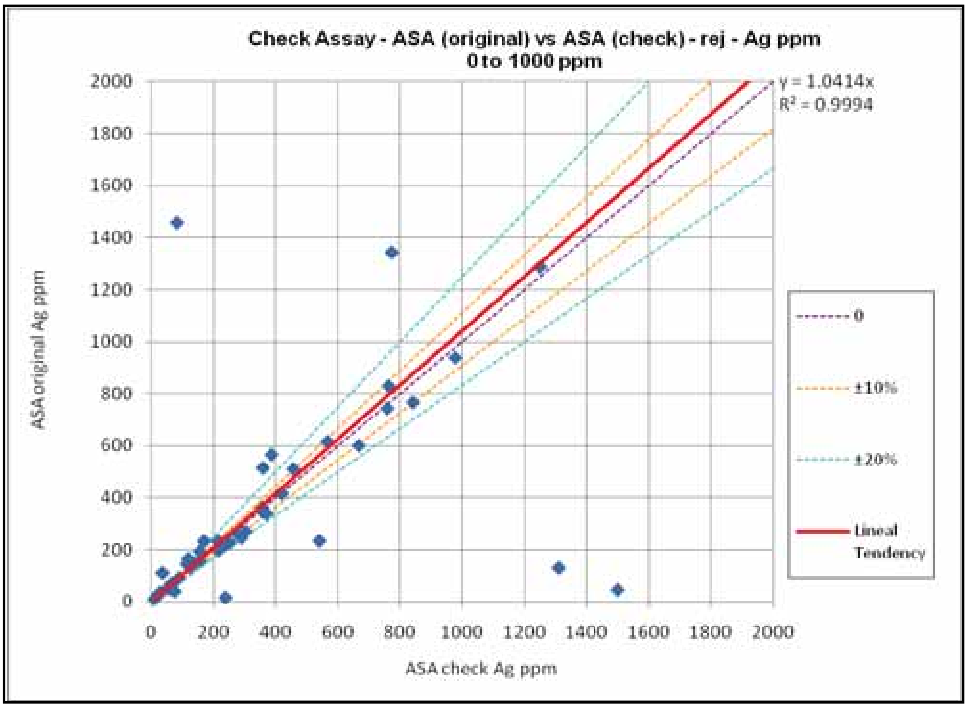


Figure 13-10
Check Assays- Alex Stewart- Rejects Ag values between 0-1000ppm

<u>Discussion of Check Assay Results</u>

Preliminary interpretations of the scatter plots for the provisional Alex Stewart check assay results took into consideration the correlation of original and check assay values that were duplicated within plus or minus 10 and 20 percent limits, the linear regression trends generated by the respective values and the relative precision of the laboratory values reported for the standards that were submitted within the respective check assay batches.

For the check assays conducted on the pulps generally good overall correlation was achieved for both Au and Ag with, in the case of Au, an indicated slight overall positive bias (+5 percent) of the original values with respect to the check assay values. The respective original and check assay values for Ag showed overall better correlation than those for Au.

Analysis of the provisional results for the coarse rejects reported by Alex Stewart, indicate good correlation for high grade Au and Ag results (i.e. Au 400-2100 g/t Au, Ag 20,000-210,000 g/t Ag) whereas results for the ranges of 0-50 g /t Au and 0-1500 g/t Ag show considerable widespread variations. These results are currently being analyzed will be further compared with those for ACME once those results become available.

13.8 Check Assay Results

Statistical results for the check assay data were generated in Excel spreadsheets. Correlation coefficients indicate an excellent correlation for all of the gold values and the rechecks of pulps with the independent laboratory (i.e. ACME Laboratories), as well as an internal check of Alex Stewart Assayers S.A.

The interpretations of the scatter plots took into consideration the correlation of original and check assay values that were duplicated within plus or minus 10 and 20 percent limits, the linear regression trends generated by the respective values and the relative precision of the laboratory values reported for the standards that were submitted within the respective check assay batches.

CAM believes that the check assay results indicate that the database is acceptable.

13.9 Adequacies of Sample Preparation, Security, and Analytical Procedures

CAM believes that preparation and analysis of samples are acceptable and within industry standards. However, CAM did not independently supervise any drilling and sampling or sample preparation.

14.0 DATA VERIFICATION

Data was validated utilizing visual review of digital and paper files, as well as computer- aided checking systems. This validation also included the physical re-checking (in some case re-surveying) of field locations including survey stations, trenches and drill collars. Validation also included review of historic core samples and volumes of digital and paper data, including maps and assays. Data verification included database searches, certificate validation, and QA/QC tests on assay results. Other forms of validation included the twining of drill holes and trenches and review of the geophysical data. Robert Sanderfur, P.E., a Qualified Person, performed the data verification.

Minor limitations on validation include the few supporting documents from the historic data set from Barrick; the Barrick data was only from trenches and was not used in Resource estimation. In most cases the data was re-generated, surveyed or duplicated for confirmation.

14.1 Surface Topography

Surface topography was provided as elevations on a 5 by 5-metre grid in MapInfo format. These data were exported into the MicroModel software system and checked against surveyed drillhole collar elevations. The fit was found to be very good, and the surface topography was thus accepted. N

14.2 Standard Checks

CAM uses automated data processing procedures as much as possible in constructing and auditing geologic databases to assure consistency and minimize errors. These procedures depend heavily on consistent alphanumeric attribute codes and consistent and non-duplicated field labels and drillhole IDs. While many of the issues flagged by these automated procedures are obvious to a human, CAM requires a clean and consistent database before proceeding with geological modeling. Common inconsistencies in submitted databases include:

1. Misspellings;
2. Confusion of 0 (zero) and O or o;
3. Inconsistent use of upper and lower case;
4. Inconsistent usage or space _ and -;
5. Trailing, leading or internal blanks. (blanks are routinely changed "_" to positively identify this problem);
6. Inconsistent use of leading zeros in hole IDs;
7. Inconsistent analytical units (e.g. PPM, PPB, opt, percent); and
8. Inconsistent coordinate systems and units and state plane and mine grid: feet and metres.

These issues are not uncommon for a project at this level of development and all of the corrections to the Cap Oeste database were obvious. CAM does not regard these issues as critical, but they need to be addressed as the project proceeds.

For manually generated databases, CAM generally regards an error rate of less than one in 500 good, an error rate of less than one in 100 acceptable and an error rate greater than two in 100 as unacceptable. The acceptability or unacceptability of the database also depends heavily on the impact of the errors. Hence the values for acceptability in unacceptability may easily change by an order of magnitude depending on the nature of the errors. For example a dropped decimal point in a value of 37 for an actual value is 0.37 is much more serious than the entry of a 0.36 for a 0.37. For computer-generated databases any errors may be indicative of problems in data processing procedures and these require resolution of the source of the problem.

CAM also reviews the procedures used to prepare the database and is particularly critical of the common practice of cutting and pasting to obtain the database. Different companies and even geologists within the same company have different methods for drilling, sampling, sample prep and analysis and record-keeping. In some cases it may be necessary to de-weight the results of certain drilling campaigns or types of drilling.

Over the years CAM personnel have developed procedures for mathematically and statistically validating exploration databases. This check procedure includes:

- Check for duplicate collars.
- Check for twin holes.
- Check of surface collared holes against surface topography
- Check for statistically anomalous downhole surveys.
- Check for overlapping assays
- Check for zero length assays
- Review of assay statistics by grade class.
- Review of assay statistics by length class.
- Checks for holes bottomed in ore.
- Check for assay values successively the same.
- Check for assay spikes.
- Check for downhole contamination by decay analysis.
- Check of total grade-times-thickness globally and by mineral zone.

In evaluating an existing database CAM uses values flagged by these automated procedures as a starting point for database review and has found that if the error rates in the statistically anomalous values is acceptable then the entire database is generally acceptable.

A few anomalies were noted, and forwarded to PGSA, but the number and type of anomalies were within industry norms for databases of this size, and even if the anomalies turn out to be errors, they would have no effect on the overall resource estimate. On the basis of these statistical checks, and the checks of data entry discussed previously, CAM believes that the exploration database has been prepared according to industry norms and is suitable for the development of geological and grade models.

The standard check includes an approximate accuracy calculation on the XYZ location of the drillhole based on the two limiting assumptions that the hole turns to the next downhole survey direction just after a survey and that it continues at the same direction to the next downhole survey point. Results of his calculation are given in Table 14-1.

Table 14-1
De-survey Differences of More than 2.5 metres

Hole ID	Depth	DTOT	DX	DY	DZ
CO-036-D	108.1	2.5	-1.4	2.0	-0.1
CO-088-DR	50.0	2.5	-1.9	-0.9	-1.3
CO-090-DR	40.0	2.5	-2.2	0.9	-0.9
CO-034-D	150.0	2.7	2.4	0.0	1.2
CO-059-D	58.0	2.7	-1.1	2.5	0.5
CO-070-D	153.0	2.7	2.3	0.2	1.3
CO-093-D	213.0	2.7	1.8	1.4	1.5
CO-024-D	78.1	2.8	2.3	0.5	1.4
CO-072-D	144.0	2.8	2.5	-0.1	1.3
CO-083-DR	125.0	2.8	1.3	2.3	0.8
CO-012-DR	55.0	2.9	1.5	1.9	1.6
CO-054-DR	70.0	3.8	-3.1	1.8	-1.0
CO-043-DR	110.0	4.0	1.7	3.0	2.1
CO-057-DR	120.0	4.0	-2.1	-2.5	-2.4
CO-050-D	111.0	4.1	-3.5	1.3	-1.5
CO-089-DR	50.0	4.1	-3.6	0.8	-1.8
CO-065-DR	186.0	4.2	0.5	3.8	1.8
CO-087-DR	198.0	4.3	-0.8	4.1	1.0
CO-082-DR	232.0	5.6	1.9	4.8	2.3
CO-053-DR	86.0	5.8	3.1	3.8	3.1
CO-081-DR	205.0	5.9	2.3	4.9	2.4
CO-055-DR	186.0	6.0	3.1	4.4	2.6
CO-085-DR	220.0	6.1	0.0	5.6	2.2
CO-086-DR	226.0	6.1	2.0	4.9	3.0
CO-056-DR	180.0	6.2	1.8	5.2	2.9

Table 14-1 De-survey Differences of More than 2.5 metres					
Hole ID	**Depth**	**DTOT**	**DX**	**DY**	**DZ**
CO-084-DR	214.0	6.4	1.2	5.6	2.8
CO-080-DR	231.0	7.0	3.3	5.8	2.2
CO-063-DR	120.0	7.9	3.0	6.1	4.0
CO-062-DR	153.0	9.5	8.2	1.5	4.6
CO-078-DR	232.0	9.8	-1.5	9.2	2.8

This table indicates that for some holes the downhole XYZ location could be off by up to 9.8 metres. While a 9.8-metre difference is not significant in terms of the resource calculation it does have implications on the minable continuity of the deposit. For this reason CAM recommends that all future holes be downhole gyroscopically surveyed and that some downhole surveys be duplicated to assure that the uncertainty in location is less than 0.5 meters.

14.3 Drillhole Database

The Cap Oeste exploration database was provided to CAM as a Microsoft Access (MDB) database. CAM exported the data to ASCII and reformatted the data for import into the MicroModel geological modeling and mine planning system. Basic statistics on the database as provided the CAM are given in Table 14-2

Table 14-2 Drilling Statistics from Assay Database		
Item	**Number**	**Length (m)**
Surveyed hole collars in Assay Database	95	11263.2
Below-collar survey shots	183	9546.5
Downhole survey shots down-hole (incl collars)	278	9546.5
Holes with below--collar downhole surveys	70	9565.6
Surveys up-hole	0	0.0
Assay intervals (Au ppm)	8600	9632.5
Assayed intervals (Au ppm)	8600	9632.5

15.0 ADJACENT PROPERTIES

As previously discussed, significant precious metal mineralization has been defined at the adjacent Cap Oeste project and at several prospects within an approximate, seven kilometer radius around the COSE target.

Further exploration drilling is scheduled throughout the Cap Oeste, Cap Oeste Extension, Pampa, Don Pancho and Vetas Norte prospect areas in the coming 2010/2011 field season (refer to Figure 2-1).

While CAM acknowledges that this ongoing work may lead to eventual expansion of the Cap Oeste Project, none of the exploration results from adjacent properties were used by CAM in preparing this report, and the mineral Resources estimated herein rely solely on the Cap Oeste database.

16.0 MINERAL PROCESSING AND METALLURGICAL TESTING

Mr. Gustavo Almeyda of Patagonia Gold S.A. requested SGS Minerals Services in Santiago, Chile to perform several tests on a set of samples from the COSE project including:

- assays of gold in the metallic fractions;
- cyanide leaching in bottle tests; and
- gravity separation tests.

The samples were received, and entered into the SGS system, and prepared for testing. A total of 70 samples were received, which were composited into 25 samples at Patagonia Gold's request. The 25 composite samples and metallurgical tests are summarized in Table 16-1.

16.1 Screen Fire Assays

A total of 16 coarse residues (95 percent less than 10 # ASTM) from composite, high and mid grade intervals were analyzed by SGS Laboratories via the screen fire assay technique in order to determine the size/distribution character of gold mineralization.

The technique is designed to concentrate the potentially larger gold particles in the coarse fraction sample, given the tendency for gold grains to flatten during grinding, and enable semi-quantitative analysis on the potential presence and effects of coarse gold on sample analysis reproducibility of relatively small (50 gram) sample sizes used routinely for analysis.

Sample preparation involves firstly the milling of the coarse reject to 95 percent less than 200# ASTM after which the undersize is sieved, weighed and split into three subsamples which are each subsequently fire assayed. The oversize is weighed and subsequently fire assayed for which values over 10 g/t Au are determined with a gravimetric finish.

16.2 Gravitational Concentration Testing

In order to perform gravitational concentration tests a total of 3 composites weighing between 3.5 to 5 kg were submitted to SGS Minerals.

The samples will be firstly milled in a roll crusher to 100 percent less than 10# ASTM (2mm) and subsequently for each sample, two representative 500 gram sub samples will be obtained using a Knelson concentrator. One of these samples will be retained as a reference sample and the other one further pulverized to 100 percent below 150# ASTM and analyzed by fire assay to determine the head gold grade. Each sample that returns results over 5 g/t Au will be re-analyzed by gravimetric finish.

16.3 Bottle Roll Tests

A total of 25 composites were submitted for bottle roll tests to SGS Minerals.

The samples were composited from 70 individual course rejects selected from 15 diamond holes taken throughout oxidized, partially oxidized and non oxidized portions of fault-hydrothermal breccia hosted and veinlet Au mineralization.

The coarse reject composite samples will be ground to 95 percent less than 100 microns (# ASTM 150), homogenized and split into 800 gram pulps.

Each of the composites were tested as follows:

a) 45 element ICP scan after multi acid digestion.
b) 50-gram Fire assay.
c) Active Cyanide Leach on each 500-gram sample with 1 percent NaCN solution with sampling of the pregnant CN liquor after 24, 48 and 72 hours.
d) Analysis of gold and silver in solid residue after cyanidation by 50-gram fire assay method.

The gold ranged between 1.2 and 1,276 grams per tonne (g/t) and the silver ranged between 13 to 45,118 g/t.

16.4 Preliminary Results

The gravity tests were run utilizing a lab-model Knelson concentrator. In general the gold recovery averaged 60 percent and the silver recovery averaged 35 percent.

In the rolling-bottle cyanide-leach tests the results were slightly better, with gold recovery averaging above 76 percent and the average silver recovery was 55 percent.

It was determined that a simple solution was not at hand to increase recoveries of both the gold and silver, and that more extensive tests were needed to investigate higher recovery methods.

After reviewing the preliminary metallurgical results, and considering the tonnage of ore present and the grade in the COSE ore shoot, CAM concluded that it would be much simpler to mine the ore in the shoot and ship it to a smelter. The direct-shipping ore would have a value on the order of US$2,980 per diluted tonne using metal prices of US1204 /oz for gold and US23.75 per ounce for silver.

CAM estimates that a gold recovery of 95 % would be conservative and a silver recovery of 90% could be reasonably expected.

The refining losses were estimated from past contracts at 1 percent for gold for a final recovery of 99 percent for gold and a refinery loss for silver at 3 percent for an overall refinery recovery of 97 percent for silver.

Table 16-1
COSE Composite Intervals and Metallurgical Tests

Composite	Hole	From	To	Interval	Bottle Roll	Screen Fire Assay	Gravity Separation
1	CSE-033-D	269.50	271.50	2.00	√	√	X
2	CSE-033-D	271.50	274.50	3.00	√	X	X
3	CSE-030-D	156.40	160.00	3.60	√	√	X
4	CSE-027-D	214.17	217.70	3.53	√	X	X
5	CSE-027-D	217.70	219.20	1.50	√	√	X
6	CSE-027-D	219.20	220.80	1.60	√	√	X
7	CSE-027-D	220.80	223.50	2.70	√	√	√
8	CSE-027-D	223.50	228.10	4.60	√	X	X
9	CSE-024-D	240.00	243.30	3.30	√	√	√
10	CSE-022-D	269.90	272.00	2.10	√	√	X
11	CSE-022-D	272.00	273.50	1.50	√	X	X
12	CSE-017-D	241.50	244.78	3.28	√	X	X
13	CSE-017-D	244.78	247.10	2.32	√	√	√
14	CSE-017-D	247.10	250.50	3.40	√	X	X
15	CSE-017-D	250.50	252.98	2.48	√	√	X
16	CSE-017-D	252.98	261.50	8.52	√	X	X
17	CSE-013-D	208.30	210.85	2.55	√	√	X
18	CSE-013-D	210.85	213.40	2.55	√	X	X
19	CSE-002-R	142.00	146.00	4.00	√	√	X
20	CSE-034-D	283.60	286.40	2.80	√	√	X
21	CSE-035-D	285.08	286.60	1.52	√	√	X
22	CSE-035-D	286.60	288.20	1.60	√	√	X
23	CSE-014-D	123.00	126.80	3.80	√	√	X
24	CSE-041-D	298.60	299.80	1.20	√	√	X
25	CSE-039-D	180.00	183.70	3.70	√	X	X
TOTAL					25	16	3

16.5 Process

After the ore is mined and brought to the surface, it will be crushed in a conventional jaw crusher and further crushed in a roll crusher. The final product will be screened at 1 millimeter and loaded into 2-

tonne bags for shipment. The bags will be loaded into 20-tonne ocean cargo containers and these will be loaded onto a truck for shipment to the port of Comodore Rivadavia.

The containers will then be shipped by ocean going vessels to smelters in Japan, South Africa or Europe.

17.0 MINERAL RESOURCE ESTIMATE

17.1 Summary

Resources were calculated by Matthew Boyes of Patagonia Gold and reviewed and classified as indicated and inferred by Robert L. Sandefur, PE of CAM using statistical methods. Resources are based on 38 holes which intersect the COSE ore shoot. CAM classified the resource as indicated based on the criteria that there is a less than 10 percent chance than less than 85 percent of the contained ounces will actually be mined. Because this deposit has only been sampled by surface drilling there is greater uncertainty than if the deposit had been estimated on the basis of channels samples a meter apart in drifts separated by 25 meters vertically. Approximately 70 percent of the contained ounces are carried on five of the 38 holes. The silver to gold equivalent ratio of 53.5 is based on a gold price of US$1204 per troy ounce and a silver price of US$23.75 per troy ounce and gold and silver recoveries of 95 and 90 percent respectively were applied when calculating AuEq metal content. Metals prices were calculated as the three-year past two-year future rolling average as of March 1, 2011. CAM believes additional drilling to confirm the area of influence of the five high-grade holes is prudent. Resources are summarized in Tables 17-1 and 17-2.

Table 17-1
NI 43-101 Compliant Resource Statement
Total INDICATED Resources Undiluted COSE Project

Tonnes	Grade			Contained Metal (Ounces)		
	Au (g/t)	Ag (g/t)	AuEq (g/t)	Au	Ag	AuEq
20,637	60.06	1,933.07	96.21	39,850	1,282,582	63,835

Gold equivalent (AuEq) values are calculated at a ratio of 53.5:1 Au;Ag.

Table 17-2
NI 43-101 Compliant Resource Statement
Total INFERRED Resources Undiluted COSE Project

Tonnes	Grade			Contained Metal (Ounces)		
	Au (g/t)	Ag (g/t)	AuEq (g/t)	Au	Ag	AuEq
13,758	60.06	1,933.07	96.21	26,566	855,055	42,557

Gold equivalent (AuEq) values are calculated at a ratio of 53.5:1 Au;Ag.

17.2 PGSA Resource Model

The PGSA resource model was developed by Matthew Boyes of PGSA using the GEMCOM software system. The ore shoot was defined on the basis of lithologies and grades and the hanging wall and footwall intersections were used as a basis to build a wireframe of the ore shoot. A block model using the following geometric parameters was constructed: (Table 17-3).

Table 17-3 PGSA Model Model Geometric Parameters					
Origin (Meters)		Number of		Block Size (Meters)	
Northing Easting Elevation	4686600.00 2391400.00 100.00	Rows Columns Benches	100 100 120	Row Column Bench	2.50 2.50 2.50
Rotation Angle (320.00)					

Block partials were constructed by intersecting the wireframe with the block model. Nominal 1-meter length composites were constructed inside the wireframe. Grades were calculated using inverse distance cubed with a sector search oriented at an azimuth of 40, a dip of -75 with radii of 15x10x15 meters. Gold and silver composites over 600g/t and 20,000 g/t were restricted to radii of 7.5x5x7.5 meters and 7.5x5.07.5 meters, respectively. A density of 2.4 was used in the resource estimate, calculated from 92 measurements taken from a representative selection of core samples.

17.3 Model Validation and Review by CAM

DataBase

The database was provided to CAM as a series of spreadsheets. Basic statistics on the provided database are given in Table 17-4.

Table 17-4 COSE 2011 Drilling Statistics from Assay Database		
Item	Number	Length (Meters)
Holes	86	17,239.5
Holes with non-collar downhole surveys	79	16,873.9
Non-collar survey records	581	16,688.9
Downhole surveys down	667	16,688.9
Assay intervals (Au)	5,221	6,150.2
Assayed intervals (Au)	5,221	6,150.2

The standard CAM database check was unremarkable but for a shoot of this size downhole surveys may be important.

Downhole Survey Review

Top Bottom Desurvey

There are a variety of ways of desurveying drillholes (converting downhole azimuth and dip to XYZ locations). The CAM check method for finding desurvey differences calculates the XYZ location by:

1. Assuming that the hole is straight at a given azimuth and dip to the next shot.
2. Assuming the hole immediately deviates to the next azimuth and dip 0.001 millimeter beyond the shot.

The average deviation was 2 meters and the maximum was 5 meters. As shown it the Figure 17-1 the red dot is almost always to the left of the back dot. This indicates that hole deviation is probably consistent.

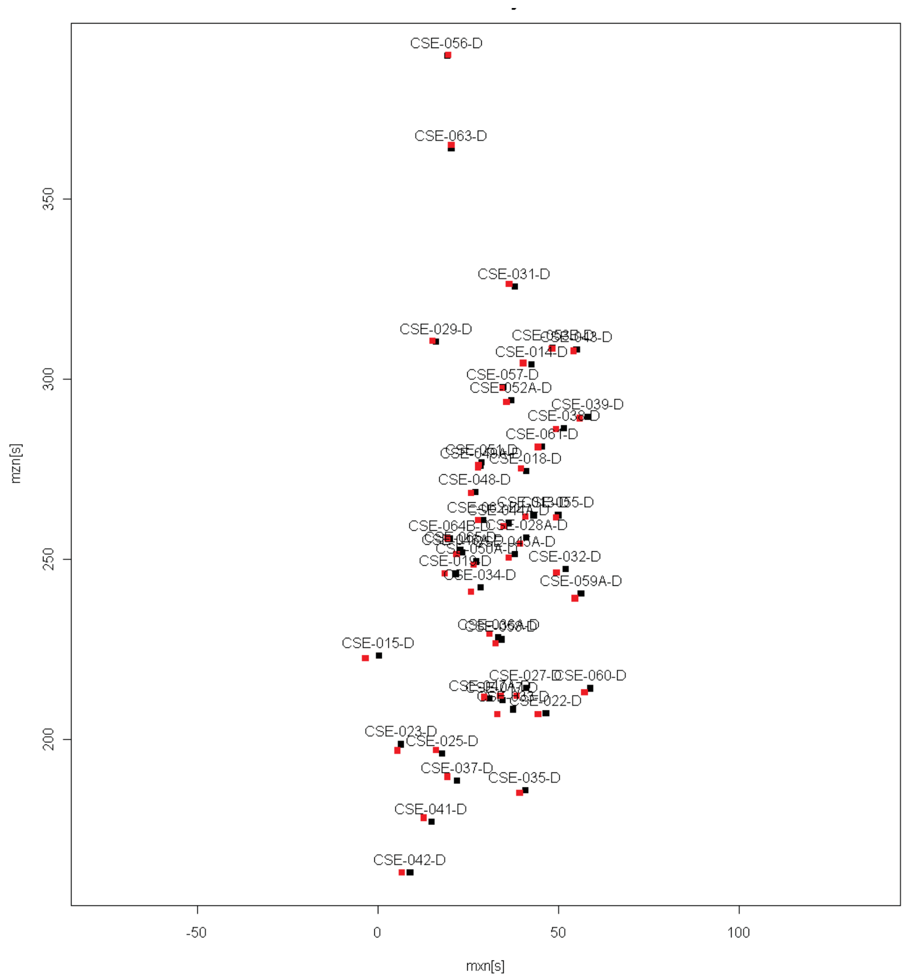


Figure 17-1
COSE Ore Shoot Rotated Model Maximum Desurvey Deviations

17.4 CAM Resource Recalculation

To check the quantities of the PGSA resource estimate, CAM used a nearest neighbor estimate with a spherical search radius of 7.5 meters. This estimate, while geometrically unbiased, completely ignores the geology of the deposit and is suitable only for verifying quantities associated with a properly geologically constrained estimate. This type of estimate basically projects the value of each composite perpendicular to the drillhole half way to the nearest drillhole and is thus based on the geometry of the drilling and completely ignores the geometry of the deposit. Results of this calculation are given in Table 17-5.

Table 17-5
CAM Nearest Neighbor Check of the PGSA Resource Model

Item	Model	Tonnes	AuPPM	AgPPM	EQAuPPM	AuCToz	AgCToz	Eq53.5	AvgDst
Value	50x50x50_Unconstrained_NN 7.5m		56.31	2178.02	97.02				4.12
	PGSA_model_tabulated_by_CAM		60.06	1933.07	96.19				NA
Percent	50x50x50_Unconstrained_NN 7.5 m	97.78	93.76	112.67	100.86	91.67	110.17	98.62	4.12
	PGSA_model_tabulated_by_CAM	100	100	100	100	100	100	100	NA

17.5 Resource model review by CAM

The COSE deposit is characterized by some spectacular grades, the highest of which is 0.4 percent gold equivalent. The reproducibility of assays is excellent. This is the only high grade ore shoot that CAM has worked on that was not found by drifting underground along a structure. The deposit was found by surface drilling and has only been sampled by surface drilling. A relatively small number of intersections through the shoot are available. In an underground operation typically a shoot such as this would have been sampled by at least two raises with samples on about 1 meter separation.

In many mines spectacular high grade values found in samples are not reflected in actual production so it is usual engineering practice to limit the influence of very high grade samples by top cutting or capping values above a certain value to that value. In the case of COSE there is no production data available which may be used to calibrate an appropriate top cut. For this deposit CAM has used statistical methods to characterize the risk associated with the resource estimate ASSUMING THAT THE SAMPLES OBTAINED THUS FAR ARE REPRESENTATIVE OF THE DEPOSIT AND OF ALL ACTUAL GRADE THAT WILL BE MINED. This assumption can only be validated by actual mining but is geologically reasonable given the compact geometry of the ore shoot and the fact that black rocks are almost always ore and white rocks tend to be lower grade.

The ore shoot is essentially vertical and strikes approximately north 40° east. For this analysis CAM rotated the model to an azimuth of 50° about an arbitrary point X6 the choice of the origin is arbitrary. T his rotation gives a model in which the y-axis corresponds very closely to the true width of the shoot. A block size of 1-meter vertically, 1-meter along strike and 0.1-meter across strike was selected. Blocks whose centroid fell within the wireframe of the ore shoot were defined as ore shoot blocks. Downhole composites 0.1-meter in length were constructed and backmarked by block. The approximate true width for each hole was calculated as the difference in rotated Y values for each hole along with grade thickness for gold and silver.

Relevant variables for the deposit for a global check of the resource are apparent thickness, total gold grade*apparent thickness, total silver grade*apparent thickness, and equivalent gold grade*apparent thickness. The equivalent gold grade*apparent thickness is calculated as the sum of gold grade plus silver grade divided by 53.5. (Note that initially a value of 47 was used for equivalence and some figures may be based on this value. This difference is not substatantive.)

A total of 38 drillholes intersect the ore shoot. All four relevant variables are more or less lognormal. The distribution is shown in the next four figures.

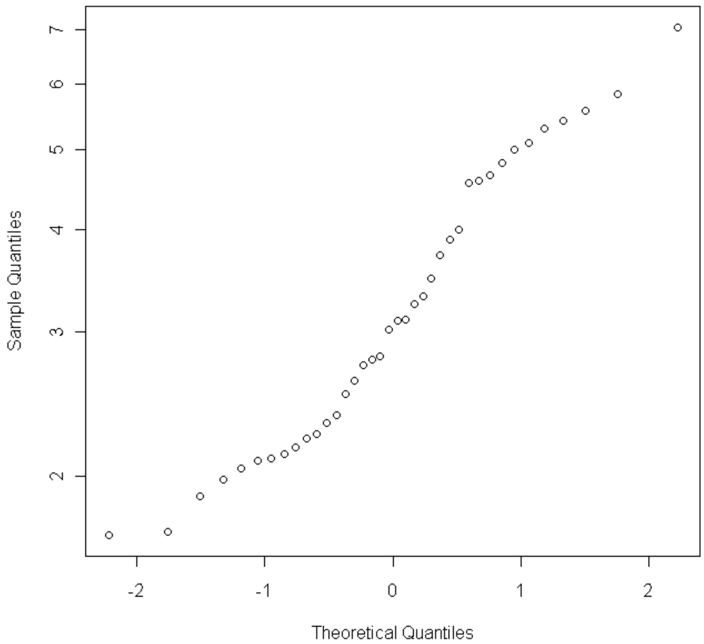


Figure 17-2
In Ore Shoot Rotated 0.1 Meter Backmarked Composites
EDA Log Cumfreq Apparent Total Thickness (m)

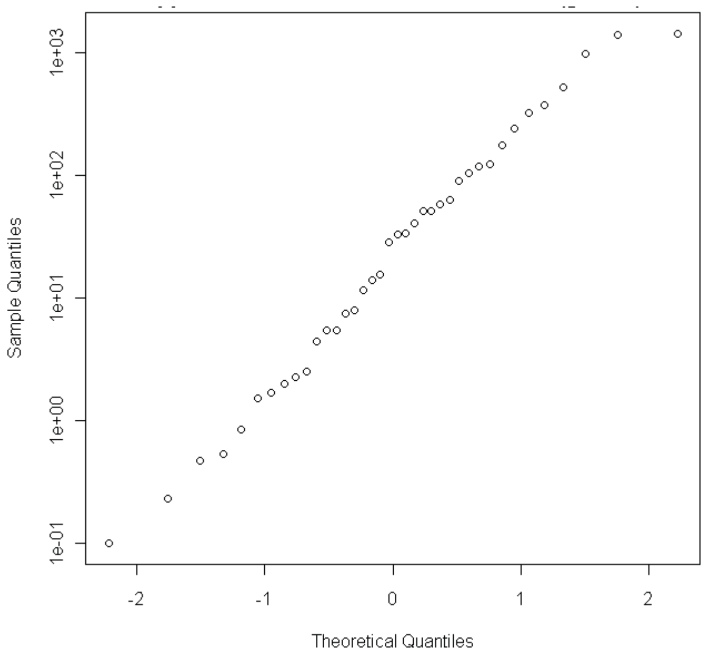


Figure 17-3
In Ore Shoot Rotated 0.1 Meter Backmarked Composites
EDA Log Cumfreq Apparent Total Au+0.1 Grade Thickness (gm/t*m)

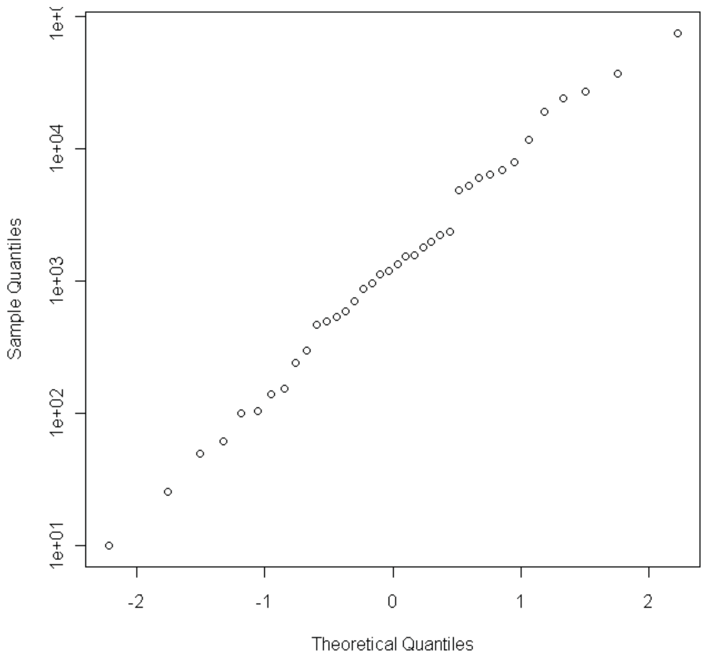


Figure 17-4
In Ore Shoot Rotated 0.1 Meter Backmarked Composites
EDA Log Cumfreq Apparent Total Ag+10 Grade Thickness (gm/t*m)

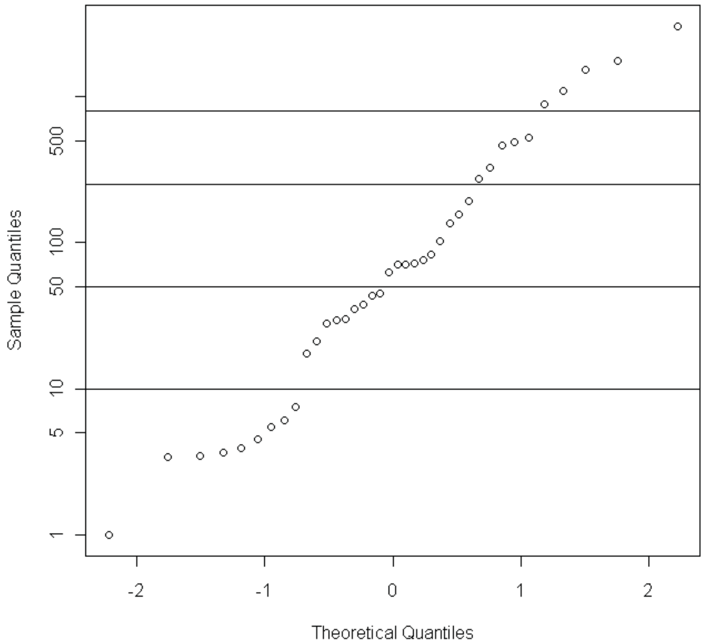


Figure 17-5
In Ore Shoot Rotated 0.1 Meter Backmarked Composites
EDA Log Cumfreq Apparent Total AuEq+1 Grade Thickness (gm/t*m)

The horizontal lines on the above plots may correspond to different regions within the ore shoot. A view of the model color coded by equivalent gold grade thickness is given in the next figure.

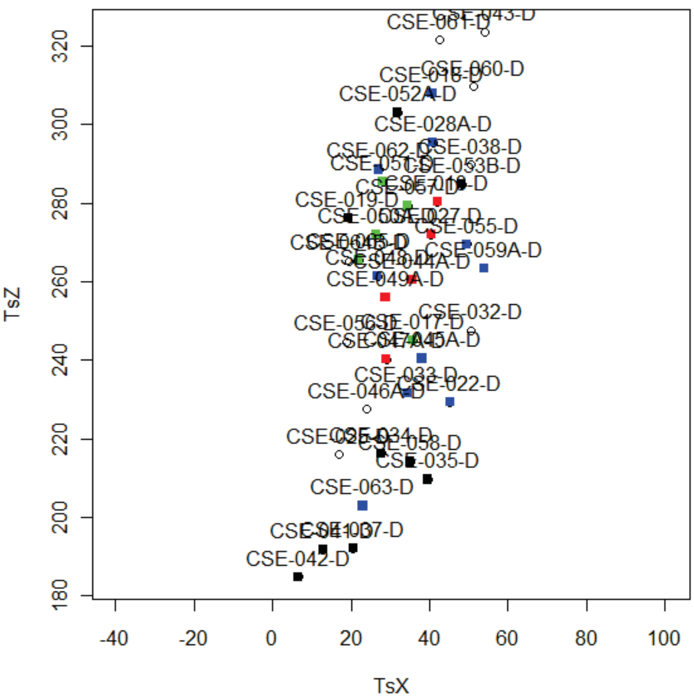


Figure 17-6
In Ore Shoot Rotated 0.1 Meter Backmarked Composites
EDA Rotated XZ Hole Centroid Plot EqAt Total Grade Thickness
Red > 800, Green > 250, Blue > 50, Black > 10, Open Circle <= 10

This figure is encouraging because it shows continuity of the highest grade thickness values (greater than 800 gm*m/t) and reasonable continuity of material greater than 250*gm*t. However, even though there appears to be good continuity of the highest grade grade thickness values slightly off vertical, there is insufficient close spaced drilling to define the width of this higher grade 5 point "structure". A grade thickness of 800 gives 62 equivalent gold ounces per square meter of apparent vein area (US$86,400 per square meter of apparent vein area), a grade thickness of 250 gives 19 equivalent gold ounces per square meter of apparent vein area, and a grade thickness of 50 gives 4 equivalent gold ounces per square meter of apparent vein area (US$5,600 per square meter of apparent vein area) or put another way a difference house 12 m² between the red dot area of influence and the blue dot area of influence is $1 million in contained metal values.

CAM has seen similar or greater differences in vein type deposits; however, in those the Quartz containing free gold and the wall rock were very distinct. The same high grade zones show in grade and ounces per hole as shown in the next two figures.

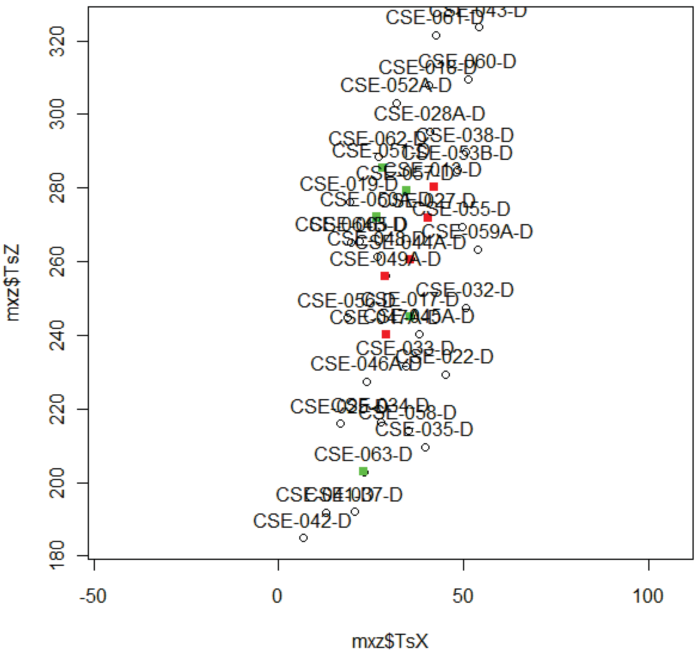


Figure 17-7
COSE Rotated XZ In Shoot Hole Centroid Plot
Percent of Contained Gold Equivalent Ounces
Red > 10, Green > 2, Open Circle <=2

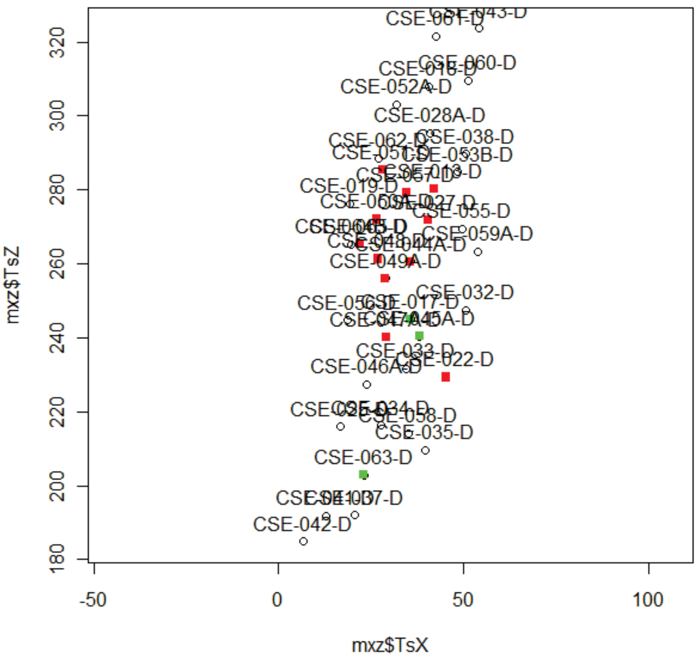


Figure 17-8
COSE Rotated XZ In Shoot Hole Centroid Plot
Au Equivalent Grade TOz/tonne
Red > 2, Green > 1, Open Circle <=1

CAM attempted to construct log variograms for the relevant variables but these were not very satisfactory, so CAM elected to assess the statistical uncertainty of the resource estimate using cross validation. In cross validation a data point is dropped out of the dataset, the value at that point calculated from the other points and the calculated value compared to actual. CAM has developed an automated cross validation procedure to optimize the parameters used in cross validation so the estimation variance is locally minimized. (Local minimization means that changing the estimation parameters very slightly gives a higher value than the optimized point. If there are multiple minima as a function of the parameters local minimization may miss the global minimum.)

To calculate tonnage and contained metal CAM used a nearest neighbor estimate within the footprint of the rotated ore shoot and apparent thicknesses and grade thicknesses.

17.6 Resource Classification

There are no quantitative definitions for the resource classifications on measured indicated and inferred. For measured and indicated CAM normally uses the criteria that there is less than a 10 percent chance than less than 85 percent of the tons and contained metal in indicated plus measured class will be recovered during actual mining operations. . For an operation such as COSE with no actual production experience, CAM does not classify any of the resource as measured.

CAM conducted a cross validation on the 38 holes in the ore shoot and selected the minimum variance for each of the 4 variables. To define indicated tonnage and grade CAM calculated the low 10 percent limit on the mean of each of the four variables assuming a normal distribution. This assumption is incorrect for the log normally distributed data; however, the low 10 percent limit assuming a normal distribution is conservative given the skewed nature (i.e. approximately lognormal) of the distribution. Indicated was calculated by dividing the low 10 percent by 0.85. Table 17-6 summarizes the calculations for the four variables.

Table 17-6
CAM 2D polygonal resource estimate SG=2.500
(This Table Contains Totals of Indicated and Inferred and Is Not Publicly Disclosable)

Item	Percent of Tonnes	AuPPM	AgPPM	AuEQPPM	Percent of TOZAu	Percent of TOZAg	Percent of TOZAuEQ
ALL	100.00	58.83	2315.92	108.11	100	100.00	100.00
LOW10%	87 .88	NA	NA	NA	52.54	45.86	52.43
Indicated%	100.00	NA	NA	NA	61.81	53.96	61.68

The tonnage is 100 percent indicated and contain ounce are about 60 percent indicated, However, having different proportions of indicated for tonnes and ounces may be confusing and CAM and PGSA agreed to classify tonnes and contained ounces as in the resource as 60 percent indicated and 40 percent inferred as shown in Tables 17-6 and 17-7.

17.7 Resources by Hole

Since CAM did a 2-D polygonal resource estimate, it is possible to easily calculate the tonnage, grade and contained ounces associated with each hole. Results of this calculation are given in Table 17-7. Because CAM split the resource proportionally into indicated and inferred, all totals (tonnes, area and contained ounces) are expressed as a percentage and the same percentage are applicable to both indicated and inferred.

Table 17-7
CAM 2D Polygonal resource estimate By Hole
Tonnes, Area and Ounces are expressed as % because of restrictions on Totalling Indicated and Inferred
AgperAu=53.5

Hole	% Tonnes	% Area	Athk m	Au PPM	Ag PPM	AuEQ PPM	% TOZ Au	%TOZ Ag	% TOZ AuEQ	% TOZ AuEQ	Sequence
CSE-013-D	2.40	1.87	3.00	481.49	25045.91	949.64	19.60	25.90	22.27	22.27	1
CSE-049A-D	3.49	2.28	3.59	273.16	10316.16	465.99	16.18	15.53	15.90	38.17	2
CSE-027-D	3.71	2.18	3.99	350.55	1321.80	375.26	22.08	2.12	13.62	51.79	3
CSE-044A-D	5.61	3.22	4.10	91.61	5854.16	201.03	8.74	14.18	11.05	62.84	4
CSE-047A-D	4.41	2.34	4.42	119.03	6102.97	233.10	8.92	11.61	10.06	72.89	5
CSE-057-D	2.17	1.87	2.72	118.63	2341.60	162.40	4.38	2.20	3.45	76.35	6
CSE-051-D	2.50	1.65	3.56	33.43	5311.57	132.71	1.42	5.73	3.25	79.60	7
CSE-063-D	7.24	4.46	3.81	32.40	403.36	39.94	3.98	1.26	2.83	82.43	8
CSE-050A-D	3.01	1.93	3.65	48.76	1868.56	83.69	2.49	2.43	2.47	84.89	9
CSE-017-D	5.88	2.28	6.05	17.40	1318.81	42.05	1.74	3.35	2.42	87.31	10
CSE-065-D	0.33	0.79	0.98	248.53	11819.33	469.45	1.40	1.69	1.53	88.84	11
CSE-018-D	5.94	3.05	4.57	1.61	1305.34	26.01	0.16	3.35	1.51	90.35	12
CSE-048-D	1.82	1.49	2.88	31.08	1689.11	62.65	0.96	1.33	1.12	91.47	13
CSE-022-D	1.28	2.71	1.11	56.88	1626.85	87.29	1.23	0.90	1.09	92.56	14
CSE-028A-D	6.13	3.34	4.31	2.69	550.49	12.98	0.28	1.46	0.78	93.34	15
CSE-055-D	2.51	2.63	2.25	22.86	382.30	30.01	0.98	0.41	0.74	94.08	16
CSE-033-D	2.51	2.54	2.32	21.89	410.52	29.56	0.94	0.45	0.73	94.81	17
CSE-059A-D	4.45	2.16	4.84	8.38	408.05	16.01	0.63	0.78	0.70	95.50	18
CSE-062-D	2.41	2.28	2.48	11.37	889.28	27.99	0.47	0.93	0.66	96.17	19
CSE-045A-D	1.28	1.97	1.52	37.97	384.79	45.16	0.83	0.21	0.57	96.73	20
CSE-034-D	2.17	2.52	2.02	16.65	241.41	21.16	0.61	0.23	0.45	97.18	21
CSE-052A-D	2.45	3.20	1.80	4.39	745.95	18.33	0.18	0.79	0.44	97.62	22
CSE-035-D	2.27	2.52	2.11	15.61	219.62	19.72	0.60	0.21	0.44	98.06	23
CSE-041-D	2.47	3.58	1.62	8.60	432.96	16.69	0.36	0.46	0.40	98.46	24
CSE-053B-D	1.95	2.18	2.10	0.77	735.54	14.52	0.02	0.62	0.28	98.74	25
CSE-037-D	1.23	2.65	1.09	1.76	1088.93	22.11	0.04	0.58	0.27	99.00	26
CSE-042-D	2.49	3.38	1.73	3.13	304.25	8.82	0.13	0.33	0.21	99.22	27

Table 17-7 CAM 2D Polygonal resource estimate By Hole Tonnes, Area and Ounces are expressed as % because of restrictions on Totalling Indicated and Inferred AgperAu=53.5											
Hole	% Tonnes	% Area	Athk m	Au PPM	Ag PPM	AuEQ PPM	% TOZ Au	%TOZ Ag	% TOZ AuEQ	% TOZ AuEQ	Sequence
CSE-058-D	1.26	2.52	1.17	13.10	196.73	16.78	0.28	0.11	0.21	99.43	28
CSE-019-D	1.40	1.85	1.78	3.05	631.00	14.84	0.07	0.38	0.20	99.63	29
CSE-056-D	0.99	3.32	0.70	2.05	207.07	5.92	0.03	0.09	0.06	99.69	30
CSE-061-D	2.88	5.11	1.32	0.10	99.01	1.95	0.01	0.12	0.06	99.74	31
CSE-025-D	1.11	2.48	1.05	4.12	37.67	4.82	0.08	0.02	0.05	99.80	32
CSE-046A-D	1.40	3.68	0.89	2.74	57.93	3.82	0.06	0.03	0.05	99.85	33
CSE-043-D	0.65	2.14	0.72	0.52	402.37	8.04	0.01	0.11	0.05	99.90	34
CSE-032-D	2.07	4.32	1.13	1.91	13.68	2.17	0.07	0.01	0.04	99.94	35
CSE-060-D	1.88	3.20	1.38	0.54	65.97	1.77	0.02	0.05	0.03	99.97	36
CSE-038-D	1.31	2.50	1.23	0.35	77.13	1.79	0.01	0.04	0.02	100.00	37
CSE-064B-D	0.96	1.79	1.26	0.00	0.00	0.00	0.00	0.00	0.00	100.00	38
Total/Avg	100.00	100.00	2.35	58.83	2315.92	102.12	100.00	100.00	100.00		

It is of interest to note that 73 percent of the contained equivalent ounces are contained in the five holes with over 10 percent ounces per hole and that one hole accounts for approximately 22 percent of the ounces and 87 percent of the contained ounces are contained in 10 of the holes. This again confirms the high degree of statistical uncertainty associated with any resource estimate for the COSE deposit. Hole CSE-064B-D is inconsistent between the CAM and PGSA resource estimate. This has negligible effect on the overall resource but should be resolved in the next update.

17.8 SG review by CAM

A total of 60 specific gravity samples of the 94 values provided to CAM were labeled as “Mineralized zone” in the Comments field. A cumulative frequency plot of these samples showed a more or less normal distribution with three high outliers which are quite common in deposits with large amounts of sulphides.

The mean value of the 60 samples is 2.39 and the standard deviation is 0.10 which gives an approximate 95 percent confidence limits on the density of plus or minus 1 percent. This uncertainty is much lower than the statistical uncertainty associated with the tonnage, grade or contained ounces and is adequate for a feasibility study (as opposed to a PEA). On the basis of this review CAM believes the density of 2.4 used by Patagonia Gold is acceptable for use in a feasibility study.

18.0 OTHER RELEVANT DATA OR INFORMATION

CAM is unaware of any other information not included herein, the omission of which would tend to make this report misleading.

19.0 ADDITIONAL REQUIREMENTS FOR TECHNICAL REPORTS ON DEVELOPMENT PROPERTIES AND PRODUCTION PROPERTIES

19.1 Mining

19.1.1 Introduction

The COSE prospect is located in Patagonia in the department of Rio Chico, Santa Cruz province, Argentina, approximately 65 kilometers southeast of the town of Bajo Caracoles.

The deposit at COSE is a subvertical, gold and silver bearing breccia that averages 0.5 m to 4 m in width and dips approximately 75° to the southwest. The interpreted ore shoot has a strike length of about 35 m, with a down dip extension of approximately 145 m. The preliminary estimate of undiluted resources is 34,000 tonnes containing approximately 112,000 gold equivalent ounces.

Exploration has included mapping, rockchip and trench geochemical sampling, induced polarization studies, and both rotary (RC) and diamond drilling (DDH). This work was carried out between 2008 and the present. Gold occurs as sulfides, electrum, and native gold. The wall rocks are primarily volcanic tuffs of different composition and porosity. Neither the geomechanical, nor the hydrogeological information, has yet been consolidated.

A preliminary Scoping Study report on the deposit was prepared by NCL, in December 2010.

The NCL study includes a brief description of the; deposit, mine access, mining method, preproduction development requirements, development and production schedule, production rate, major mining equipment, backfill, mine services anticipated, preliminary ventilation estimates, and a very preliminary estimate of development and mining costs.

All preproduction and mining is proposed to be performed by a mining contractor.

This current desk-top review is based solely on information contained in NCL's Scoping Study, as no site investigation has been performed. To-date, no topography, or deposit sections or plan maps were available to assist this review.

19.1.2 General

The overall mine plan prepared by NCL Ingeniería y Construcción Ltd (NCL), at a Scoping Study level, is generally acceptable, and would allow for accessing and mining the deposit. However, CAM believes that the current plan could be improved upon in several areas, and that the currently-estimated cost for developing and mining the deposit should be revised, based on the CAM's proposed changes to the mine access and the mining method.

Based on NCL's mine plan, their estimated cost to develop and mine the deposit, over an approximate two-year period, is $15,313,496. CAM's estimate for the same work is $24,439,943. All work is to be accomplished by a mining contractor.

Based on a new access decline inclination, and the mining method proposed by CAM, all preproduction take-off quantities presented in the NCL's Study were revised, and new cost projections estimated. The NCL-proposed production rate of 120 tonnes per day (3,600 tonnes per month) was accepted.

The NCL-proposed decline inclination, at 16.7 percent through the ore zone, is considered too steep. This grade may save a few meters of development, but results in very high equipment operating cost, and is unsafe for loaded trucks (with backfill) returning into the mine. A grade of 12.5 percent (8:1) is recommended.

A standard, mechanized, overhand, cut and fill mining method, utilizing resuing where applicable, should be considered instead of the sublevel stoping method proposed in the NCL Study.

No allowance for external dilution in the mining phase was made in the NCL Study. The use of a sublevel stoping method in a narrow deposit could easily result in 30+ percent dilution, plus a 10% to 15% loss of ore recovery. In addition, the dilution tonnes should been added to the estimated approximate 34,000 insitu resource tonnes, and considered in the mining cost summary.

Utilization of a cut and fill/resuing method would provide much more flexibility in the mining process, would reduce dilution significantly, and would improve ore recovery significantly. The external dilution should be less than 20 percent. These are all important factors, when dealing with high-grade precious metal ores. The slight decrease in productivity, when compared with a sublevel stoping method, should easily be justified, when the advantages of a more selective mining method are considered.

The use of this method will also allow for the use of "resuing" (separation of ore and waste within the stoping process), when mining in very high grade areas.

Additional time needs to be allowed in NCL's project schedule for site preparation, contractor set-up, and the portal construction. This change will probably add three to four months to the current schedule.

In CAM's opinion, the ventilation circuit needs to be reversed to bring in fresh air through the decline and exhaust through the proposed ventilation raise. This is for safety, in case of a fire emergency, and to prolong the life of diesel engines operating within the decline.

No allowance has been made for on-site Patagonia Gold (Company) overall project management and quality control during the developing and mining of the deposit by the contractor. As the development and mining phases are expected to last about two years, a company cost is anticipated over this period to provide for project management and QAQC.

An average contingency of approximately five percent has been included by NCL in their Case 1 cost summary. This figure needs to be in the 25 percent to 35 percent range at this stage of estimating.

19.1.3 Mine Access

The Scoping Study prepared by NCL proposes to access the deposit by a minus 16.7 percent spiral decline from the surface to the bottom of the deposit, a distance of 1,743 m. The proposed cross section of the access decline is 4.3 m wide by 4.5 m high, designed to accommodate 20-tonne trucks and utility lines. Typical ground support, using rockbolts and shotcrete is allowed, where necessary. This work is to be performed by a reputable underground mine contractor.

The present schedule for developing 1,743 meters of decline is approximately one year, at an average advance rate of 150 m per month. This rate should be achievable by a good contractor.

The biggest problem with the decline design is the steepness of the ramp at 16.7 percent through the ore zone. Almost all mine access ramps are designed at minus 12 percent to minus 15 percent. If the ramp is a continual spiral, as is proposed for the COSE deposit, the lower figure would normally be adopted. Continuous turning in a decline is difficult to construct and hard on equipment. The steepness included in the NCL study may have been partially rationalized by the very short mine life.

However, the proposed grade in the decline will be very hard on the loaded 20-tonne trucks traveling up the decline. Maintenance costs could easily be doubled. This is compounded by the fact that the access decline designed by NCL is to be an exhaust airway. Experience with a similar decline ramp, operating in an exhaust mode, resulted in reducing truck engine lives by two-thirds. There is also a severe safety issue with loaded trucks (hauling backfill) down a ramp with an inclination of minus 16.7 percent.

In CAM's opinion the ramp inclination should be lowered to 12.5 percent (8:1) and the change reflected in new capital and operating cost estimates. This change would increase the NCL-estimated decline length by 237 meters, to 1980 meters.

19.1.4 Preproduction Development

The current NCL preproduction plan calls for a 4.3 m x 4.5 m decline access from the surface to the ore deposit, with 4.0 m x 4.0 m access drifts to the ore zone every 16 m vertically. Ventilation raises 2.0 m x 2.0 m in section would be constructed between each level, with a 2.4 m diameter bored raise extending to the surface from the uppermost level. Short, 4.0 m x 4.0 m, ventilation drifts would connect the access drifts to the ventilation raises. The advance rate in the primary openings is estimated at 150 meters per month.

The basic development plan would allow for the exploitation of the deposit, but it underestimates the development quantities and costs, and the time it will take to perform all of the work.

In addition, an advance rate of 150 m per month for constructing a minus 16.7 percent decline, in a continuing spiral, may be difficult to achieve. This rate of advance is typically estimated for minus 12 to 14 percent ramps, with long straight drives between curves. CAM has accepted the advance rate used by NCL, but based their ramp design on longer straight sections between curves, with accesses to the deposit every nine meters, vertically.

CAM has estimated new preproduction development quantities based on a lower decline incline inclination, and a different mining method, which would require access to the deposit every nine meters vertically. The basic ventilation scheme was revised to incorporate a longer, primary raise-bored ventilation raise to surface, with less “short” conventional raises within the mine.

Typically, when preproduction development quantities are estimated, even in scoping studies, an allowance of 10 percent is normally included for turnouts, muck bays, material storage, etc. This allowance does not appear to have been included in NCL’s estimates.

In addition, NCL’s cost estimate, which will be discussed later in this review, is based solely on cubic meters extracted for development and mining, rather than lineal meters of development performed. The use of cubic meters as a pay unit normally presents a problem for the owner in that it pays the contractor for a volume extracted. Unless the contractor is limited to the design section, plus a small allowance for overbreak, payments would be made on the total measured volume extracted, including any excess overbreak.

Table 19-1 provides a summary comparison of the preproduction development included in the NCL report, along with CAM’s estimate. All of CAM’s estimates include an allowance of 10 percent for turnouts, muck bays, material storage, etc.

Table 19-1 Preproduction development Summary					
Cost Center	Size (W x H) (m)	Length (m)		Volume (m3)	
		NCL	CAM	NCL	CAM
Site Preparation	Lump Sum	Lump Sum	Lump Sum	Lump Sum	Lump Sum
Portal Construction		0	100	0	1,450
Decline	4.3 x 4.5	1,743	1,980	34,843	42,144
Access Drifts	4.0 x 4.0	333	480	4,772	8,448
Vent Xcuts	4.0 x 4.0	235	235	3,747	4,136
Short Raises	2.0 x 2.0	181	0	819	0
Raise Bore Raise	2.4 dia.	92	255	417	1,130
Misc. Openings	Various	0	N/A	0	500

19.1.5 Mining Method

The NCL-proposed mining method for the COSE deposit, although referred to as a cut and fill method in their study, is actually a modified blasthole stoping method, where 4.0 m x 4.0 m access drifts to the ore are developed every 16 m, vertically, off of the access decline. Drifts for blasthole drilling and ore extraction will be extended from the access drifts. Once the ore is extracted, the plan is to backfill the stope, with a combination of cemented and un-cemented mine development waste rock. The development waste rock will be hauled out of the mine and stored close to the access portal. 20-tonne ore extraction trucks are proposed to carry the backfill back into the mine, when needed.

As mentioned previously, the deposit is very small (about 34,000 tonnes undiluted). NCL has proposed a production rate of 120 T per day (3,600 tonnes per month, 42,000 tonnes annually). This would extrapolate to approximately nine months of mining ore, if no external dilution were included.

However, when external dilution (0.25 meters) and mining recovery are considered, the mined tonnage becomes 39111, which translates to almost 11 months of production, and increases the total operating cost estimates accordingly.

The above described mining method proposed by NCL would allow for the extraction of the deposit, at a slightly lower unit operating cost, but with much higher dilution and lower ore recovery.

In CAM's opinion, NCL's planned mining method would not be the most appropriate method for mining the COSE deposit.

Small, epithermal vein/breccia pipe deposits are typically very irregular in both the vertical and horizontal directions. As the vein widens (> 4 m), there is generally less irregularity, and the effect of wall rock dilution is diminished substantially. However, at COSE, the deposit widths generally vary from very narrow (0.5 m) up to 4 m, with occasional wider areas.

The use of a blasthole stoping method, where the ore is drilled from a drilling drift above to an extraction drift below, will result in excessive dilution from the deposit walls and reduce the mineral recovery where it "bulges" outside the drilling limits. This would seem to be unacceptable, especially when high-value minerals (gold/silver) are being mined.

The use of a blasthole mining method would be more productive, which would result in a slightly less unit operating cost per tonne mined. However, the resulting greater dilution and lower ore recovery should prohibit the use of this method at this deposit. In addition, a much larger stope must remain open until backfill is placed. Insufficient geotechnical information is currently available to say whether there would be problems with larger stopes standing open for a period of time. In the absence of such data, common sense would indicate that a more conservative approach be considered.

The two mining methods that might be acceptable are shrinkage stoping, or mechanized, ascending cut and fill. Both of these methods are adaptable to narrow, high grade deposits and are both much more selective than a blasthole method. However, since nothing is known at this time about the geotechnical parameters of the deposit, or the wall rocks, it would be safer to base a scoping study mine plan on a cut and fill option.

The use of a mechanized cut and fill mining method would be appropriate for this deposit, especially when definition drilling is limited, and little is known about the geotechnical aspects of the deposit. Flexibility, with regard to deposit boundary variability, would be much greater with the use this method, equipment sizes would be smaller, dilution much less, and ore recovery much better. In addition, backfilling the stope cuts would be simplified, and cement in the backfill could be avoided.

These factors should more than offset any productivity and cost benefits there might be in using a sublevel blasthole mining method. Preproduction development for ramping from the surface to the deposit bottom should be about the same, except for a somewhat longer length due to a lower decline inclination, while access drifting between the main ramp and the deposit would increase due to more access drifts being required.

The installation of a horizontal pillar, about half-way down the deposit (around elevation 248) would allow mining to begin in the upper half of the deposit, while development work continued to the bottom of the deposit. This would permit an earlier cash flow to be generated. The pillar could be recovered at the end of the bottom half mining by leaving a cemented floor in the bottom of the first stope cut in the top half of the deposit.

19.1.6 Ventilation

The current design is to bring fresh ventilating air down through a bored 2.4 m raise, pass the air through the stopes and exhaust up the primary access decline. The required quantity of ventilation was estimated by NCL at approximately 130,000 cubic feet per minute (cfm), or 62 m^3/second. This volume was based on the horsepower of anticipated equipment in use (1,100 HP), personnel in the mine, pressure losses, and acceptable velocities within the airways. A quick check of this quantity indicates that this figure should be acceptable for either mine plan.

However, in CAM's opinion, the fresh air should enter through the access decline, pass through the workings, and then exhaust through a primary ventilation raise. The primary reasons for this are safety and cost.

If there were to be a fire in the mine, the principal mine exit requires a fresh air escapeway, which would be through the access decline. In addition, experience has shown that the maintenance costs on diesel engines, operating on steep declines, particularly in exhaust air streams, increase significantly.

In CAM's opinion, a bored ventilation raise should be constructed between the deposit (~280 elevation) and the surface (~430 elevation). This would allow development to continue downward, and also provide ventilation to early mining in the upper portion of the deposit. When development has reached the bottom of the deposit, the bored ventilation raise can be continued (offset) to the bottom for ventilation of all of the deposit.

19.1.7 Equipment/Services

The equipment list included in the NCL study contains only the major mine equipment. It is understood that the development/mining contractor will supply and maintain all of the necessary underground mobile equipment, including the surface plant, to perform the scope of work. In reality, the contractor's actual equipment requirements will be larger than NCL's list includes, and there will be a price schedule for equipment used in "out-of-scope work", which almost always occurs.

The Company typically provides the fixed surface plant, such as infrastructure, surface utilities, primary fans, substations, compressors, generators, etc., and the maintenance of access roads and company infrastructure. On a turn-key project, such as this appears to be, the contractor may be requested to provide generators for producing electric power, compressors for compressed air equipment, and utility lines and distribution, etc. However, the cost for providing all of such items is going to be added to the contract by the contractor; as a rental-plus maintenance agreement, as a lump sum payment increase to the unit price for cubic meters extracted, or for lineal meters advanced. Unexpected weak ground and excessive water inflows are common "out-of-scope-work extra cost sources.

At the COSE project, the successful contractor will utilize the existing infrastructure for their personnel camp, maintenance requirements, and administration offices. However, the Company will also need an on-site project manager, secretary, and project shift engineers for quality control. This will mandate use of some the existing Company infrastructure. In addition, the Company may have to maintain the site access roads, and provide other services. These items need to be reflected in the cost estimate.

The contract between the Company and the contractor should be carefully constructed to avoid excessive charges by the contractor. In addition, the Company needs to understand that not all liability lies with the contractor. Unforeseen items for additional ground control and excessive water are common, and generally require a good working relationship and contract between the parties to arrive at an equitable agreement for the "extra, or out-of-scope, work".

Services, such as electric power, compressed air, waste water drainage, and fresh water supply, have been adequately addressed for this level of study and available information.

19.1.8 Project Scheduling

The schedules included in the NCL study for preproduction development and mining would appear to be underestimated, thus indicating higher costs for both of these cost centers.

The base case (Case 1) schedule shown in NCL's Figure 4-1 calls for about twelve months of preproduction development, followed by an additional nine months of on-going development and mining.

The primary cost centers in their preproduction period are the driving of the primary access decline from the surface to the bottom of the deposit (1,743 m) and the deposit access drifts from the decline, plus ventilation requirements. The lineal work for the decline and access/ventilation drifts was scheduled at 150 m per month, which resulted in about 15 months of development time. CAM's estimates for the same work would require an additional three months, or 18 months.

In addition, time must be allowed for the site preparation, the contractor mobilization, and the portal boxcut excavation and decline collar brow support. A quick review of these requirements, plus the additional time driving the decline and access drifts, would indicate an additional four to six months should be added to the preproduction schedule. Although not included in NCL's schedule, cost estimates for the contractor mobilization and demobilization, and the portal construction were included in their budget.

The mining phase is currently scheduled to begin in the fourth quarter of the first year, and continue for a year. As previously mentioned, the start of production could be advanced, if desired.

At the scheduled production rate of 3,600 tonnes per month, the 39,111 tonne (diluted/recoverable) deposit would be mined out in approximately 11 months. This would add an additional two months of production to NCL's ore production schedule.

19.1.9 Development and Mining Costs

Based on figures shown in NCL's Table 5-1, during the first three quarters, the average cost per meter for driving the decline and access drifts is approximately $3,000 per meter of advance, which would be an acceptable figure for contractor-performed work in this region. When this figure is applied to the total lineal development, over the life of the mine (2,599 lineal meters – NCL's estimate), the indicated total cost for lineal development drifting is $7,797,000.

The NCL Study (Table 5-1) also includes $505,094 for contractor mobilization and demobilization, site preparation, and construction of the decline portal, plus $10,411,621 (not including contingency and profit) for developing and mining the deposit, for a total estimated cost of $10,916,715.

The difference between the total cost of $10,916,715 and ($7,797,000 + $505,094), should be the approximate amount that was estimated for mining, or $2,614,621. Using the NCL Study resource figure of 33,000 tonnes (undiluted), an estimated cost of $79 per tonne for mining would be indicated. Based on a scheduled production rate of 120 tonnes per day, this figure appears to be low, when compared with mines with similar production rates.

NCL's estimating details provided to CAM show a direct operating cost of $136.28/tonne, without contingency. This figure would translate to a total operating cost of $4,497,240 for mining the 33,000 undiluted tonnes mined included in their study. CAM has used the latter, detailed operating cost in cost comparisons.

Since the Company will need representation for project management, engineering and quality control during all of the preproduction development and mining phases, Owner's labor and infrastructure costs will be realized during this period. CAM is estimating a minimum of one project manager, two project engineers, and one clerical person will be required. This labor cost, plus five percent for infrastructure maintenance and office supplies, etc., over an approximate 24 month project life, has been estimated. This would add approximately $40,000 per month to the project cost.

A contingency of $500,000 (approximately 4.6%) and a contractor profit of $1,637,222 (15%) have been added by NCL to their estimated costs to arrive at the total cost for developing and mining the deposit of $13,054,222, shown in following Table 3-4. This figure is derived from Table 5-1 (Case 1) in their December 2010 report. If the detailed unit operating cost supplied to CAM is used ($163.54/tonne w/contingency), the total becomes $15,410,456.

The estimate for contractor profit of 15 percent is acceptable, but a contingency figure under five percent at this stage of estimating is unacceptable. The figure should be in the range of 25 percent to 35 percent.

Table 19-2 summarizes CAM's estimated development costs. All takeoff quantities include factors for overbreak and allowances for turnouts, muckbays, material storage, etc.

Table 19-2
CAM Estimated Development Cost ($US)

Development Type	Size (W x H)	Unit	Quantity	Unit Price ($)	Estimated Cost
Contractor:					
Site Preparation	Site	each	Lump Sum	500,000	$500,000
Decline Portal	Box Cut + 20 m	m3	2,000	150	$300,000
Decline	4.3 x 4.5	m	1,980	3,500	$6,930,000
Access Drift	4.0 x 4.0	m	480	3,000	$1,440,000
Vent Xcuts	4.0 x 4.0	m	235	3,000	$705,000
Raise Bored Vent	2.4 m dia.	m	255	900	$230,000
Misc'l Openings	Various	m3	500	150	$75,000
Subtotal Contractor					**$10,180,000**
Contingency @ 25%					$2,545,000
Cont. Profit @15%					$1,527000
Total Develop. Cost					**$14,252,000**

CAM has estimated the operating cost (contractor direct and indirect labor and supplies) for mining the COSE deposit at approximately $170 per tonne. This cost was based on zero-based estimates for a contractor's direct and indirect labor and supplies needed for a production rate of 3,600 tonnes per month, over an11 month production period.

Table 19-3 summarizes the mining (operating cost) for a contractor-run mining operation to mine 39,112 tonnes of ore. It assumes that the contractor will supply all labor, equipment, equipment maintenance, supplies and utilities, plus operating and maintaining their portion of Patagonia Gold's camp, which already exists at the site.

The Company would maintain a project manager, project engineers/geologists, and a clerk on site for quality control of the contractor's mining work for the 11 month ore production period. The Company would maintain its portion of the existing infrastructure that they utilize.

Table 19-3
CAM Estimated Mining (Operating) Costs

Cost Center	Labor	Supplies	Total	Cost/Tonne
Contractor (Direct + Indirect)	$4,606,649	$2,041,881	$6,648,530	$170.00
Company	$418,000	$22,000	$440,000	$11.22
Total	$5,024,649	$2,063,881	$7,088,530	$181.22

Table 19-4 is included to compare the NCL Study total estimated cost figures to CAM's total cost estimate.

Table 19-4
Total Cost Estimate Comparison

Cost Center	Quantity		Units		Unit Cost		Estimated Cost		Difference
	NCL	CAM	NCL	CAM	NCL	CAM	NCL	CAM	
Contractor									
Setup/Portal	each	each	L.S.	L.S.	$505,094	$800,000	$505,094	$800,000	$294,906
Development	m3	m3	47,969	62,533	$162/m3	$150/m3	$7,797,000	$9,380,000	$1,583,000
Mining	m3	m3	10,583	19,019	$247/m3	$410/m3	$4.663,520	$6,648,530	$1,985,010
Subtotal Cont.							$12,965,614	$16,828,530	$3,862,916
Contingency							$500,000	$4,207,133	$3,707,133
Contractor Profit							$1,944,842	$2,524,280	$579,438
Company Cost	N/A	24	N/A	month	$0	$40,000	$0	$960,000	$960,000
Total							$15,410,456	$24,519,943	$9,109,487

The primary differences between the two cost estimates are; added development quantities during the preproduction period ($1.78 million), mining external dilution added to the insitu resource tonnes ($1.85 million), contingency/contractor profit markups ($4.87 million), and the inclusion of Company costs for QA/QC for the life of the project ($1 million).

19.1.10 Minable Resources

In order to determine the portion of the measured and indicated resources that might qualify as minable resources, it is first necessary to determine the portion of the resources that can be economically extracted. The economical portion of the resource is then formatted into minable shapes along the vein(s), based on the minimum mining width, and deposit geometry. This part of the mine planning process incorporates internal dilution and eliminates ore blocks that cannot be economically accessed. When the economic viability has been established, factors for mining dilution and mining recovery are applied, based on the; deposit parameters, mining method(s), and ground strength, to arrive at the minable resource portion of the resource.

At COSE, the precious metal grades are very high (approximately 87 grams Au_{eq}/tonne), and the deposit is enclosed by sharp, visible, structure boundaries (abrupt grade boundary). Therefore, all tonnes between the deposit boundaries would be considered as ore.

Since no “Measured” resources have yet been delineated, all minable resources tabulated in this evaluation have been based on “Indicated” resource tonnes, and as such, are classified as “Probable” resource tonnes. “Inferred” resource tonnes are also included at this “Scoping” level of Study for mine planning purposes.

19.1.11 Cutoff Grade

The economic portion of the resource is typically determined by the application of a breakeven cutoff grade, or value, that considers the total estimated operating costs for the mine, process plant and administration.

The total operating costs are then formulated into an algorithm that also considers; metal price(s), process recovery(s), applicable royalties, and forward costs for concentrate freight, insurance, smelting and/or refining. These parameters are then equated to determine the minimum grade, grade equivalent, or value of metal(s) that will need to be mined in order to cover these total operating costs. The cutoff grade may also be expressed as a revenue value (US$ per tonne), that equals the total operating costs.

At the Patagonia Gold’s COSE project there will be two payable metals (gold and silver), with gold being the predominant payable metal. Therefore, it would be easier to express the breakeven cutoff as a gold equivalent grade, or as a minimum NSR (net smelter return) value of the smelted concentrate, expressed in $/tonne ore, which covers the total operating cost.

Since the breakeven cutoff grade represents the minimum grade, or value, that will be mined, the average grade delivered to the mill will always be higher. This increment, between the breakeven cutoff grade and the head grade, provides for return of the capital investment and profit.

Other cutoff grades (incremental) may be employed later in the mine planning process by the mine planners/management, to handle situations where mineralized material, with a value below the economic cutoff grade, must be mined in order to reach ore, or to optimize the cash flow. However, these incremental cutoff grades are not normally acceptable for use in calculating the initial cutoff grade for establishing minable resources.

The following basic algorithm illustrates the typical relationship, between the various parameters, to calculate at the breakeven cutoff gold equivalent grade:

$$Au_{eq} = \frac{Total\ Operating\ Cost}{(Au\ Price)\ x\ (Au\ Recovery)} = Grams\ of\ Au_{eq}\ per\ tonne$$

If the breakeven cutoff is expressed as a value (typically an NSR in $/tonne ore), this value can then be compared to the sum of the mine, mill, G&A, and forward unit operating costs. The NSR value must equal, or exceed, the on-site total operating costs for the block, or stope, tonnes to be economically viable and included in the minable resource.

At present, various processing options are still being considered. These options include; all ore shipped directly to a smelter (95% Au and 90% Ag recovery), on-site gravity concentration with the concentrates being sent to the smelter (60% Au and 50% Ag recovery), and gravity concentration followed by a Merrill Crow cyanide leach circuit (87% Au and 65% Ag recovery).

For this review, it has been assumed that a processing plant will be available on, or near, the deposit site, utilizing gravity concentration followed by a Merrill Crow circuit. Doré would be shipped off-site.

At the COSE Project, the following economic parameters, which were based on a mine production rate of 120 tonnes per day (tpd), were utilized in the economic evaluation.

<u>Production Rates (tonnes per day)</u>....................................... 120

<u>Estimated Total Operating Costs (US$ per tonne)</u>

Mining (Contractor direct & indirect) 170
Processing 38
G&A (Company).. <u>12</u>

Total Operating Cost... **210**

Process Recoveries

Gold (%)... 87

Silver (%) 65

Metals Prices

Gold ($US/oz) 1166

Gold ($US/gm) 37.49

Silver ($US/oz).. 22

Silver ($US/gm) 0.690

Credits

Ag as a credit to Au grade (ave. gms Ag per/gm Au) 32.18

Gold equivalent value.. 1.45

Forward Costs

Freight, Marketing, etc ($US/oz Au)………………………..10

Smelting/Refining ($US/oz Au)……..……………………….6

FS&R ($US/gm Au)……………………………………..0.51

Applying the above parameters provides the following breakeven gold equivalent grade for a 120 tpd production rate:

$$Au_{eq} = \frac{(\$170 + \$38 + \$12)}{(\$37.49/gm - 0.51/gm)\ x\ (0.87)} = 6.53\ grams\ of\ Au_{eq}\ per\ tonne$$

This indicates that the gold grade (gms/t) in a block, plus the silver grams per tonne converted to gold grams per tonne in that block, must equal at least 6.53 grams Au/tonne.

Typically, this breakeven cutoff grade would be used to identify the economic portion of the measured and indicated resource. Minable shapes (stopes) are then applied to the economical portion of the resource model. This shape may include some sub-economic material that must be taken in the mine planning and stoping process, which is included as internal dilution. In addition, some of the economic blocks may have to be dropped due to their location outside of the mining shapes, or because they would require excessive development to access and prepare the block for mining. This may decrease the minable portion of the resource recovery somewhat.

19.1.12 Mining Dilution

Internal waste dilution (inside the deposit boundaries) has already been accounted for in the resource block model preparation.

External waste in the mining process generally originates from three principal sources; 1) irregularities in the deposit walls, 2) over-drilling into waste at the deposit boundaries, and 3) mucking up backfill from the floor.

Since the COSE deposit is narrow and "pipe-like" in shape, and an overhand, mechanized cut and fill mining method has been proposed, most of the external mining dilution would be expected from the deposit boundaries, with a small amount resulting from mucking waste from the floors. Floor waste dilution can be minimized with good supervision, so an average 0.25meter (m) halo of dilution, at zero grade, has been assumed around the deposit. This figure should be conservative, which should also allow for some waste dilution from the floors. Table 19-5 summarizes the diluted, economic minable resource at COSE by extraction level.

Table 19-5
Diluted and Recoverable Minable Resources

Elev.	Tonnes	Au gms/tonne	Ag gms/tonne	Au Eq. gms/ tonne	Contained Au oz	Contained Ag oz	Contained AuEq oz
316.5	138	0.743	454.843	10.420	3	2,013	46
313.5	253	1.044	613.983	14.107	8	4,996	115
310.5	368	1.085	666.079	15.257	13	7,875	180
307.5	496	1.197	838.189	19.031	19	13,372	304
304.5	575	1.529	896.433	20.602	28	16,574	381
301.5	610	1.914	759.537	18.074	38	14,898	355
298.5	638	2.284	660.417	16.335	47	13,544	335
295.5	695	2.656	617.766	15.800	59	13,805	353
292.5	781	5.744	762.007	21.957	144	19,126	551
289.5	878	14.004	1,072.258	36.818	395	30,260	1,039
286.5	935	65.227	3,390.458	137.364	1,961	101,943	4,130
283.5	931	126.901	6,321.974	261.411	3,800	189,306	7,828
280.5	944	156.215	7,051.475	306.246	4,740	213,963	9,292
277.5	978	144.751	4,950.179	250.074	4,553	155,715	7,866
274.5	1,012	152.010	2,667.195	208.759	4,943	86,739	6,789
271.5	1,055	128.704	1,436.781	159.274	4,365	48,733	5,402
268.5	1,094	94.213	1,459.865	125.274	3,315	51,359	4,407
265.5	1,105	66.666	1,677.937	102.367	2,368	59,588	3,635

Table 19-5 Diluted and Recoverable Minable Resources							
Elev.	Tonnes	Au gms/tonne	Ag gms/tonne	Au Eq. gms/ tonne	Contained Au oz	Contained Ag oz	Contained AuEq oz
262.5	1,099	50.069	1,739.850	87.087	1,769	61,460	3,076
259.5	1,084	77.015	2,580.663	131.923	2,685	89,960	4,599
256.5	1,117	105.483	3,368.648	177.156	3,789	121,003	6,364
253.5	1,188	91.889	3,437.163	165.020	3,509	131,245	6,301
250.5	1,248	60.971	2,095.824	105.563	2,447	84,111	4,237
247.5	1,312	37.435	1,410.392	67.444	1,579	59,483	2,844
244.5	1,397	40.030	1,642.297	74.973	1,798	73,766	3,367
241.5	1,469	55.094	2,309.040	104.222	2,602	109,039	4,922
238.5	1,541	60.463	1,937.742	101.692	2,996	96,032	5,040
235.5	1,553	57.337	1,189.115	82.637	2,863	59,370	4,126
232.5	1,489	47.231	578.032	59.529	2,262	27,680	2,851
229.5	1,348	29.739	457.460	39.473	1,289	19,824	1,711
226.5	1,183	17.775	403.298	26.356	676	15,337	1,002
223.5	983	15.748	374.989	23.727	498	11,856	750
220.5	836	14.684	316.066	21.409	395	8,498	576
217.5	747	13.641	258.783	19.147	328	6,217	460
214.5	699	13.866	243.891	19.055	312	5,483	428
211.5	633	17.536	262.837	23.128	357	5,345	470
208.5	558	21.358	324.463	28.262	383	5,823	507
205.5	508	22.086	346.552	29.460	361	5,661	481
202.5	470	17.786	339.721	25.014	269	5,135	378
199.5	507	17.088	399.038	25.578	279	6,507	417
196.5	557	14.018	470.797	24.035	251	8,424	430
193.5	506	11.929	567.545	24.004	194	9,240	391
190.5	466	10.224	590.942	22.798	153	8,848	341
187.5	459	8.151	468.919	18.128	120	6,919	268
184.5	410	6.062	410.058	14.786	80	5,405	195
181.5	258	4.831	332.353	11.902	40	2,754	99
Total	39,111	51.756	1,665.456	87.192	65,081	2,094,235	109,639
Note: Dilution Width = 0.25m on each side. Mining Recovery = 98%.							

Included in the minable resources reported above and in the mine plan are inferred quantities of 15,644 tonnes, 26.0 thousand contained gold ounces and 837.7 thousand contained silver ounces.

19.1.13 Mining Recovery

The proposed mining method for mining COSE ore is mechanized cut and fill, utilizing crushed waste rock, or an engineered backfill from process tailings. This mining method is very selective and generally would allow for a very high percentage recovery of the ore. The primary causes for ore loss are; 1) irregularities in the deposit boundaries, 2) pillars left for ground support, and 3) ore and waste mixed to a point where the ore becomes uneconomical. At COSE, the deposit boundaries should be the only contributor to any losses. A horizontal pillar may be left to allow mining in the upper portion of the deposit, while development continues into the lower portion of the deposit, but this pillar (if utilized) can be recovered.

The top two, and the bottom one, three-meter cuts (bench elevations 178.5, 319.5 and 322.5) are below cutoff grade, which would eliminate 110 tonnes. This would reduce the diluted resource total from 40,020 tonnes to 39,910 tonnes.

Based on a review of the deposit parameters and irregularities at the deposit boundaries, the remaining diluted resources are estimated to be 98% recoverable.

19.2 Economic Analysis

Base case metal prices used for Preliminary Economic Assessment are Au $1204/oz, Ag $23.75/oz, with recoveries of 95% and 90% respectively. The "Probable" mining resource estimates are in part based on Inferred resources as a scoping level assessment and are therefore non compliant under the NI 43-101. All cash flow calculations are based upon an undiscounted model due to total project timeline of 23 months and include a 10% royalty payable for exported concentrates. Dilution of 0.25 metres either side of the stope and a 98% recovery factor was applied to calculate the diluted minable resources.

19.2.1 Assumptions

Mining Capex

Mining CAPEX is estimated at $US 24,439,943, which includes the 1,980 meters of main decline ramp access, ore development, cross cuts and stoping of the ore. Total cost per tonne for Production during the 11 month production period are estimated at US$167/tonne and total Development cost is estimated at US$ 14,252,000.

Process Capex

The three main treatment or process routes considered for the treatment of the COSE ore include:

1. Direct Shipping, involves mining and crushing of the material on site and then shipping the ore via road and sea to a suitable smelter for direct smelting to recover the gold and silver and Ag.
2. Construction of a crushing and Cyanide leaching circuit at the La Bajada property and processing through a Merrill Crowe circuit and production of Dore' on site.

3. Gravity separation and smelting of Au and Ag on site to produce Dore'.

Process facility CAPEX estimates and metal recoveries for the three separate treatment routes are shown in Table 19-6.

Table 19-6 CAPEX Estimates and Metal Recoveries for Treatment Options		
Treatment Route	**CAPEX Requirement (US$ 000's)**	**Metal Recoveries Au;Ag**
Direct Shipping	2,700	93; 90
High NaCN Leach Merrill Crowe	8,100	87; 65
Gravity concentrate-Smelting	5,900	60 ;50

Cash-flow Assumptions

The entire project will be constructed and mined out in a 23 month period with a 12 month period of preproduction. The production rate is estimated at 3,600 tonnes-per-month. The overall mining cost is estimated at US$14,252,000 and the process capital cost is estimated at US$2,768,000. Both estimates have a confidence level of ± 30 %.

The project operating cost as shown in the cash flow is US413 per diluted tonne of ore or US167 per equivalent ounce gold produced. The operating costs include the following items, as presented in Table 19-7.

Table 19-7 Project Operating Costs	
Item	**Cost**
Total Mining	$170
Owner Mine Supervision	$11.21
Process	
Crushing/Loading Labor	
Supplies/Material	$7.00
Sundry Item	$5.00
Power	$4.00
Trucking per 20 tonne load	$760.00
Port Charges (per Tonne)	$15.00
Ocean Frieght per tonne of ore	$120.00
G&A Cost per tonne	$10.00
Gold Refining Cost (per ounce)	$3.00
Silver Refining Cost (per ounce)	$0.50
Total Direct Cost	$413

Cash-flow calculations for the three different scenarios and a sensitivity analysis for adjusted Au and Ag prices are shown below. All cash-flow sensitivities were run on the Direct Shipping option treatment route due to the smaller initial CAPEX (US$ 2,768,000) and higher potential revenue.

Base case metal price assumptions were provided by CAM and represent a trailing 36-month and future looking 24-month calculated price giving a base case Au Price of $US 1204/oz and a base case Silver Price of $US 23.75/oz.

Au-Ag Price sensitivity analysis was run for the following metal Prices:

Au Price (US$)	Ag Price (US$)	NPV ($US M)
1,203	23.75	63.7
1,000	20	46.5
1,100	22	55.2
1,400	30	84.7
1,418	35	93.8

Payback

68% of contained Au and Ag will be mined within the first 4 months of production enabling payback of capital after just 14 months from commencement of the decline.

19.2.2 Cash Flow

The cash flow is presented in Table 19-8.

The results of the cash flow include:

- a cash cost of US 167 per equivalent gold ounce produced;
- a net revenue of US$ 63.78 million; and
- an IRR of 870%.

Table 19-8
Cash Flow

Month		1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	Total
Production Summary																									
Material Mined																									
Ore		-	-	-	-	-	-	-	-	-	-	-	-	3,600	3,600	3,600	3,600	3,600	3,600	3,600	3,600	3,600	3,600	3,111	39,111
Waste																									
Total		-	-	-	-	-	-	-	-	-	-	-	-	3,600	3,600	3,600	3,600	3,600	3,600	3,600	3,600	3,600	3,600	3,111	39,111
Ore Grade																									
Au ppm		0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	85.19	67.33	131.77	100.33	3.79	6.53	17.81	15.59	40.60	58.01	40.88	
Ag ppm		0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	1643.00	1643.00	1643.00	1643.00	1643.00	1643.00	1643.00	1643.00	1643.00	1643.00	1643.00	
Contained Ounces																									
Au		0	0	0	0	0	0	0	0	0	0	0	0	9,861	7,793	15,252	11,613	438	756	2,061	1,804	4,700	6,714	4,089	65,082
Ag		-	-	-	-	-	-	-	-	-	-	-	-	190,168	190,168	190,168	190,168	190,168	190,168	190,168	190,168	190,168	190,168	164,337	2,066,018
Ag/Au Ratio		0	0	0	0	0	0	0	0	0	0	0	0	19.29	24.40	12.47	16.38	434.08	251.53	92.25	105.39	40.46	28.32	40.19	31.74
Cash Flow Summary																									
Gold Production - Oz Au														9,861	7,793	15,252	11,613	438	756	2,061	1,804	4,700	6,714	4,089	65,082
Silver Production - Oz Ag														190,168	190,168	190,168	190,168	190,168	190,168	190,168	190,168	190,168	190,168	164,337	2,066,018
Gold Production - Oz Au	95.00%													9,368	7,404	14,489	11,032	416	718	1,958	1,714	4,465	6,379	3,885	61,828
Silver Production- Oz Ag	90.00%													171,151	171,151	171,151	171,151	171,151	171,151	171,151	171,151	171,151	171,151	147,903	1,859,417
Gold After Refining	99.00%													9,274	7,330	14,344	10,922	412	711	1,939	1,697	4,420	6,315	3,846	61,210
Silver After Refining	97.00%													166,017	166,017	166,017	166,017	166,017	166,017	166,017	166,017	166,017	166,017	143,466	1,803,634
Hedged Gold Sales (oz)	0.0%													-	-	-	-	-	-	-	-	-	-	-	-
Unhedged Gold Sales (oz)	100.0%													9,273.91	7,329.60	14,344.17	10,922.02	412.03	711.06	1,938.76	1,696.98	4,420.06	6,314.94	3,846.01	61,210
Hedged Silver Sales	0.0%													-	-	-	-	-	-	-	-	-	-	-	-
Unhedged Silver Sales	100.0%													166,016.80	166,016.80	166,016.80	166,016.80	166,016.80	166,016.80	166,016.80	166,016.80	166,016.80	166,016.80	143,466.18	1,803,634

Table 19-8
Cash Flow

Month		1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	Total
Gross Revenue														$15,108,688	$12,767,743	$21,213,285	$17,093,006	$4,438,979	$4,799,013	$6,277,160	$5,986,065	$9,264,647	$11,546,089	$8,037,913	$116,532,588
Treatment and Refining Charges	$ 200.00													$720,000	$720,000	$720,000	$720,000	$720,000	$720,000	$720,000	$720,000	$720,000	$720,000	$622,200	$7,822,200
Sampling per 20 tonne Container	$ 50.00													$9,000	$9,000	$9,000	$9,000	$9,000	$9,000	$9,000	$9,000	$9,000	$9,000	$7,778	$97,778
Argentina RoyaltyRoyalty	10%													$1,510,869	$1,276,774	$2,121,329	$1,709,301	$443,898	$479,901	$627,716	$598,607	$926,465	$1,154,609	$803,791	$11,653,259
Net Revenue														$12,868,819	$10,761,968	$18,362,957	$14,654,706	$3,266,081	$3,590,112	$4,920,444	$4,658,459	$7,609,182	$9,662,480	$6,604,144	$96,959,352
Total Mining														$612,000	$612,000	$612,000	$612,000	$612,000	$612,000	$612,000	$612,000	$612,000	$612,000	$528,870	$6,648,870
Owner Mine Supervision														$40,392	$40,392	$40,392	$40,392	$40,392	$40,392	$40,392	$40,392	$40,392	$40,392	$34,905	$438,825
Process																									
Crushing/Loading Labor														$18,261	$18,261	$18,261	$18,261	$18,261	$18,261	$18,261	$18,261	$18,261	$18,261	$18,261	$200,872
Supplies/Material	$ 7.00													$25,200	$25,200	$25,200	$25,200	$25,200	$25,200	$25,200	$25,200	$25,200	$25,200	$21,777	$273,777
Sundry Item	$ 5.00													$18,000.00	$18,000.00	$18,000.00	$18,000.00	$18,000.00	$18,000.00	$18,000.00	$18,000.00	$18,000.00	$18,000.00	$15,555.00	$195,555
Power	$ 4.00													$14,400.00	$14,400.00	$14,400.00	$14,400.00	$14,400.00	$14,400.00	$14,400.00	$14,400.00	$14,400.00	$14,400.00	$12,444.00	$156,444
Trucking per 20 tonne load	$ 760.00													$136,800	$136,800	$136,800	$136,800	$136,800	$136,800	$136,800	$136,800	$136,800	$136,800	$118,218	$1,486,218
Port Charges (per Tonne)	$ 15.00													$54,000	$54,000	$54,000	$54,000	$54,000	$54,000	$54,000	$54,000	$54,000	$54,000	$46,665	$586,665
Ocean Frieght per tonne of ore	$ 120.00													$432,000	$432,000	$432,000	$432,000	$432,000	$432,000	$432,000	$432,000	$432,000	$432,000	$373,320	$4,693,320
G&A Cost per tonne	$ 10.00													$36,000	$36,000	$36,000	$36,000	$36,000	$36,000	$36,000	$36,000	$36,000	$36,000	$31,110	$391,110
Gold Refining Cost (per ounce)	$ 3.00													$27,822	$21,989	$43,033	$32,766	$1,236	$2,133	$5,816	$5,091	$13,260	$18,945	$11,538	$183,629
Silver Refining Cost (per ounce)	$ 0.50													$83,008	$83,008	$83,008	$83,008	$83,008	$83,008	$83,008	$83,008	$83,008	$83,008	$71,733	$901,817
Total Direct Cost														$1,497,883	$1,492,050	$1,513,094	$1,502,828	$1,471,298	$1,472,195	$1,475,878	$1,475,152	$1,483,322	$1,489,006	$1,284,397	$16,157,102
Operating Profit														$11,370,936	$9,269,918	$16,849,863	$13,151,878	$1,794,783	$2,117,917	$3,444,566	$3,183,306	$6,125,861	$8,173,474	$5,319,747	$80,802,250
Direct Unit Cash Cost																									
Capital Purchase																									
Pre-Production Capital																									
Mining		$266,667	$266,667	$266,667	$533,077	$533,077	$995,077	$995,077	$995,077	$1,715,077	$1,715,077	$995,077	$995,077	$995,077	$995,077	$995,077	$995,077								$14,252,000
Process		$230,667	$230,667	$230,667	$230,667	$230,667	$230,667	$230,667	$230,667	$230,667	$230,667	$230,667	$230,667												$2,768,000
Total Capital																									
Reclamation																									

Table 19-8
Cash Flow

Month	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	Total
Salvage																								
Working Capital																								
Net Cash Flow	$(497,333)	$(497,333)	$(497,333)	$(763,744)	$(763,744)	$(1,225,744)	$(1,225,744)	$(1,225,744)	$(1,945,744)	$(1,945,744)	$(1,225,744)	$(1,225,744)	$10,375,859	$8,274,841	$15,854,786	$12,156,801	$1,794,783	$2,117,917	$3,444,566	$3,183,306	$6,125,861	$8,173,474	$5,319,747	$63,782,250
Cumulative Cash Flow	$(497,333)	$(994,667)	$(1,492,000)	$(2,255,744)	$(3,019,487)	$(4,245,231)	$(5,470,974)	$(6,696,718)	$(8,642,462)	$(10,588,205)	$(11,813,949)	$(13,039,692)	$(2,663,833)	$5,611,008	$21,465,794	$33,622,595	$35,417,378	$37,535,295	$40,979,861	$44,163,168	$50,289,028	$58,462,502	$63,782,250	
IRR	20.8%		EqAu(fromAg)	AuOz	TotalAuEqOz																			
Site Cash Cost Eq Au oz	$167		35,578	61,210	96,788																			
NPV 0%	$63,782,250																							
Gold Price Hedged	0																							
Gold Price Unhedged	1204																							
Silver Price Hedged	0																							
Silver Price UnHedged	23.75																							

20.0 INTERPRETATIONS AND CONCLUSIONS

Following are CAM interpretations and conclusions with regard to the COSE Project:

1. Exploration has defined a zone of significant epithermal gold-silver mineralization at COSE, hosted within and adjacent to a moderate to high angle normal fault over widths of 5 meters and a minimum strike length of 35 meters.
2. Technical work has been conducted in a professional manner and carried out to NI 43-101 standards including the analysis, quality assurance and quality control protocols.
3. Drill intercepts identified as significant to delineation of a precious metals resource have been verified and substantiated sufficiently to pursue a Resource calculation.
4. Work on the property has been successful in identifying mineralization of potential economic interest, and further work is warranted.
5. Based on 38 drillholes which intersect the mineralized shoot, PGSA has constructed a wireframe limiting the shoot and calculated grades within the shoot.
6. Assuming that the 38 drillholes are representative of the material that will actually be mined from the shoot CAM has calculated an indicated resource based on the statistical criteria that for indicated resource there is only a 10% chance than less than 85% of the contained ounces will be mined.
7. Approximately 73% of the ounces are carried on only five holes. This implies greater risk on the resource estimate than the usual resource Estimate for underground operations were grades and tonnages are estimated based on several hundred channel samples through an ore shoot.
8. A PEA indicates that the project has favorable economics.

21.0 RECOMMENDATIONS

The geometry of the mineralization at COSE the very high grade shoot which carries 73 percent of the ounces appears to have vertical continuity over 40 m however, the nature and width of the mineralization along strike has not been defined in detail and the most important recommendations (1 and 2) by CAM relate to this.

Following are CAM recommendations with regard to the COSE project:

1. Ten holes offset on either side by about 5 meters should be drilled next to the five highest grade holes. It is CAM's understanding that it is difficult to target the intersection of these new holes with the vein from the surface so wedges right and left off the existing holes are suggested.
2. Two holes offset above and below the highest and lowest holes of the five high-grade holes should be drilled. Because of the the targeting difficulties alluded to in item 1 wedging off existing holes should be considered.
3. Specific gravity data should be reviewed and additional specific gravity determinations should be done on the very high grade intercepts to determine if there is a correlation between density and grade.
4. The block size of 2.5 meters used by PGSA is perfectly satisfactory for a PEA, however for convenience the vertical block size should be revised to be an integer multiple of the 3 m block size used by CAM in resource calculations.
5. A cross validation study on a hole by hole basis should be done to optimize the PGSA resource estimate.
6. It should be possible to obtain the weights used in the inverse distance estimation by PGSA in GEMCOM. These should be obtained and used to calculate the tonnes and ounces associated with each hole.
7. The CAM nearest neighbor checks (both 2-D and 3-D) of the PGSA models both show lower gold grades and higher silver grades than the PGSA models. While these differences are well within the accuracy of the resource estimates and are compensating in terms of equivalent gold ounces the source of this difference should be reviewed.
8. The CAM nearest neighbor grade is higher for gold and lower for silver than the PGSA estimate. This compensates in the calculation of equivalent grade and is within the statistical uncertainty of the mean grade calculation but should be reviewed.
9. Hole CSE-063-D is outside the grouping of other holes of the similar grade and grade thickness. There may be nothing wrong with this but the collar and down hole surveys should be reviewed.
10. While CAM was unable to obtain satisfactory variograms. The grouping and apparent trend shown in the equivalent gold ounces per hole, equivalent gold grade thickness and equivalent

gold grade may allow some type of indicator kriging approach to be used. This should be investigated after the results of recommendations one and two are obtained.

11. Hole CSE-064B-D has inconsistent results between the PGSA and the CAM estimates. While this has no substantive effect on the overall resource estimate it should be reviewed prior to feasibility.
12. Based on the positive results of the PEA, PGSA should proceed with a feasibility study for the COSE deposit, noting that:
    a. Additional drilling may be required to convert currently inferred resources into indicated.
    b. Because of the fact that a large proportion of the ounces are contained in a small amount of the tonnes options which minimize capital should be considered in the feasibility.

Depending on the results of recommendations one and two are carried out it may be possible to significantly increase the proportion of indicated resource in the estimate.

22.0 REFERENCES

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23.0 DATE AND SIGNATURE

23.1 Craig Bow

Craig S. Bow
9011 Cascade, Beulah, CO 81023

Phone (719) 485-4202, cellular (719) 252-0018

craigb@csbplats.com

I, Craig S. Bow, of Beulah, Colorado, do hereby certify that:

- I am an Independent Consulting Geologist, at the above address.
- I graduated from the Washington and Lee University in 1971 with a B.S. degree in Geology, and from the University of Oregon in 1979 with a Ph.D. in Geology. I am a Certified Professional Geologist # 08250 of the American Institute of Professional Geologists. I am a Fellow of the Society of Economic Geologists.
- I have practiced my profession continuously since 1979.
- I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
- I am the author of sections 1-13, 15, 16, and 18-23 of the report entitled "NI 43-101 Preliminary Economic Assessment COSE Project, Argentina" dated May 5, 2011 (the "Technical Report"). The Technical Report is based on my knowledge of the Project Area and resource database covered by the Technical Report, and on review of published and unpublished information on the property and surrounding areas. I conducted a site visit on November 21-22, 2011.
- I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which makes the Technical Report misleading.
- I am independent of Patagonia Gold or any of their subsidiary companies applying all of the tests in section 1.5 of National Instrument 43-101.
- I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that Instrument and Form.
- I consent to the filing of the Technical Report with any Canadian stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their website accessible by the public, of the Technical Report.

Dated this 5th day of May, 2011

Craig S. Bow, CPG

23.2 Gregory Chlumsky

Gregory F. Chlumsky, MMSA 01117QP
13092 W. LaSalle Circle
Lakewood, Colorado 80228 USA

I, Gregory F. Chlumsky, of Lakewood, Colorado, do hereby certify that:

- I am a Consulting Mineral Process Engineer, affiliated with Chlumsky, Armbrust and Meyer LLC at 12600 W. Colfax Avenue, Suite A-250, Lakewood, Colorado 80215, USA.
- I am a member in good standing of the Mining and Metallurgical Society of America and have been for 15 years. I am a recognized QP in Metallurgy and Process from that Organization with QP Member Number 01117QP. I have over 35 years of experience in Metallurgy and Process Costing. I graduated from the Colorado School of Mines with a BS degree in Chemistry and a minor in Chemical Engineering, and have practiced my profession continuously since 1970.
- I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
- I am the author of parts of sections 16, and 19 and the relevant parts of Section 1of the report entitled "NI 43-101 Preliminary Economic Assessment COSE Project, Argentina" dated May 5, 2011 (the "Technical Report"). The Technical Report is based on my knowledge of the Project Area and resource database covered by the Technical Report, and on review of published and unpublished information on the property and surrounding areas.
- I visited the COSE Project in May 2011.
- I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which makes the Technical Report misleading.
- I am independent of Patagonia Gold or any of their subsidiary companies applying all of the tests in section 1.5 of National Instrument 43-101.
- I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that Instrument and Form.
- I consent to the filing of the Technical Report with any Canadian stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their website accessible by the public, of the Technical Report.

Dated this 5th day of May, 2011

Gregory F. Chlumsky, MMSA 01117QP

23.3 Steve L. Milne

Steve L.Milne, P.E.
1651 Calle El Cid
Tucson, Arizona 85718

I, Steve L. Milne, of Tucson, Arizona, do hereby certify that:

- I am a Consulting Mining Engineer, affiliated with Chlumsky, Armbrust and Meyer LLC at 12600 W. Colfax Avenue, Suite A-250, Lakewood, Colorado 80215, USA.
- I am Professional Engineer #25589 in the state of Colorado, in good standing.
- I was awarded an E.M. degree in Mining Engineering from the Colorado School of Mines at Golden, Colorado in 1959.
- Since 1959 I have practiced continuously as a mining engineer, supervisor, mine manager, corporate officer, and consultant for mining firms and other mining consulting firms. This work has concentrated primarily on underground mines; encompassing a wide variety of underground conditions, metals, reserve evaluations, production rates, mining planning , equipment selection, and cost analyses throughout the world. I am the author of several publications on subjects relating to the underground mining industry.
- I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
- I am responsible for the preparation of parts of section 19, and relevant portions of section 1 of the report entitled "NI 43-101 Preliminary Economic Assessment COSE Project, Argentina" dated May 5, 2011 (the "Technical Report").
- I visited the COSE Project in May 2011.
- I am independent of Patagonia Gold or any of their subsidiary companies applying all of the tests in section 1.5 of National Instrument 43-101.
- I am not aware of any material fact or change with respect to the subjects of this report which is not reflected in this report, the exclusion of which would make this report misleading.
- I have read National Instrument 43-101 and Form 43-101F1, and the report has been prepared in compliance with that Instrument and Form.
- I consent to the filing of the Technical Report with any Canadian stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their website accessible by the public, of the Technical Report.

Dated this 5th day of May, 2011

Signed ________________________
Steve L. Milne, P.E.

23.4 Robert Sandefur

Robert L. Sandefur
1139 South Monaco
Denver, CO 80224
Phone (303) 472-3240
rlsandefur@aol.com

I, Robert L. Sandefur, of Denver, Colorado, do hereby certify that:

- I am an Independent Consulting Geostatistician, at the above address.
- I am a Certified Professional Engineer (Number 11370) in the state of Colorado, USA, and a member of the American Institute of Mining, Metallurgical and Petroleum Engineers (SME).
- I graduated from the Colorado School of Mines with a Professional (BS) degree in engineering physics (geophysics minor) in 1966 and subsequently obtained a Masters of Science degree in physics from the Colorado School of Mines in 1973.
- I have practiced my profession continuously since 1969.
- I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
- I am the author of sections 14 and 17, of the report entitled "NI 43-101 Preliminary Economic Assessment COSE Project, Argentina" dated May 5, 2011 (the "Technical Report"). The Technical Report is based on my knowledge of the Project Area and resource database covered by the Technical Report, and on review of published and unpublished information on the property and surrounding areas. I conducted a site visit on November 21-22, 2011.
- I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which makes the Technical Report misleading.
- I am independent of Patagonia Gold or any of their subsidiary companies applying all of the tests in section 1.5 of National Instrument 43-101.
- I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that Instrument and Form.
- I consent to the filing of the Technical Report with any Canadian stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their website accessible by the public, of the Technical Report.

Dated this 5th day of May, 2011

R L Sandefur

Robert L. Sandefur, P.E.