NI 43-101 Technical Report
Update of Cap Oeste Project
Santa Cruz Province, Argentina

Prepared for:
Patagonia Gold Plc

Effective Date: 07 October 2011
Report Date: 14 November 2011

087114

Prepared by:
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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 Plc (PGSA) to update and validate the exploration database for the Cap Oeste Project (Cap Oeste or the Project), located within the El Tranquilo I MD claim in the province of Santa Cruz, Argentina. The authors have visited the property on several occasions; most recently, CAM geologist Craig Bow was on site 16-18th August, 2011. The effective date of the mineral resource estimate is 7 October, 2011.

1.2 Property

The Cap Oeste 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 between exists between PGSA and the previous owners, Barrick Exploraciones S.A. and Minera Rodeo S.A., both subsidiaries of Barrick Gold, for 2.5% net smelter return royalties upon production. In accordance with the original agreement, PGSA has fulfilled all its investment commitments.

The El Tranquilo I MD exploration claim is one of several claims in PGSA's El Tranquilo project block. Another drilled prospect area called “COSE”, is located on the same MD exploration claim, centered approximately 1.5 kilometres to the southeast of the Cap Oeste prospect. COSE is the subject of a separate Technical Report.

1.3 Geology

The Cap Oeste property is located in the northwestern part of the Deseado Massif, in Patagonia, southern Argentina. This province 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 flood 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.

Precious metal mineralization at Cap Oeste is spatially related to a curvilinear, west northwest trending structure, termed the Bonanza Fault. The fault dips steeply to the southwest and has been mapped over a strike length exceeding 2.5 kilometres. Mapping peripheral to the main zone of mineralization at Cap Oeste has defined a second, sub-parallel structure 220 meters to the southwest, referred to as the Esperanza Fault. Described as a steeply northeast-dipping zone of faulting and hydrothermal brecciation, the Esperanza Fault has been mapped over a strike distance of approximately 1,500 meters. The opposing dips of the Bonanza and Esperanza faults, together with mapped repetitions and displacements of stratigraphy across these structures suggest the presence of a northwest trending graben or half-graben, approximately 220 meters wide at surface.

1.4 Mineralization

Exploration by PGSA at Cap Oeste is focused principally on discovery and delineation of low sulfidation Au-Ag epithermal deposits of the type well documented throughout the Deseado Massif. Mineralization at Cap Oeste, however, occurs predominantly as a fault-localized breccia / replacement body rather than as quartz veins, a style more typical for deposits elsewhere in the Deseado Massif. Mineralization lies within and adjacent to the Bonanza Fault and is not homogeneously distributed, but concentrated in higher grade lenses or shoots. The locations of mineralized shoots are thought to be controlled by the interaction of lithologic and structural controls; this interplay between structure and rock type has created a broadly repetitive geometrical pattern in which five shoots are currently recognized.

1.5 Exploration and Drilling

Exploration has focused on establishing a core resource in the area of exposed epithermal mineralization. Sawn channel samples from PGSA trenching adjacent to historic Barrick excavations confirmed the presence of an 8 to 25-meter wide zone of stockworked and crackle-brecciated vitric tuff in the hangingwall of the Bonanza Fault, reporting values of the order of 0.3 to 1.0 parts per million (ppm) Au. The fault zone proper contains limonite-hematite rich milled breccia with up to 11 ppm Au over widths up to 8 meters. Further trenches along strike defined a contiguous northwest trending, 900-meter-long by 5 to 15-meter-wide zone of stockwork veining, faulting, and brecciation with anomalous Au, Ag and trace element geochemistry (As, Sb, Hg). Subsequent geophysical, geochemical, and petrographic studies lent important support to these preliminary results, setting the stage for the follow-up drilling programs.

Exploration conducted between 2007 and June 30, 2011 included surface sampling, trenching, ground geophysical surveys, petrologic studies, and 48,858 meters drilling in 295 holes. As a result of this work, PGSA has defined a significant Au-Ag epithermal deposit at Cap Oeste, and has made significant strides in understanding the geologic and structural controls to mineralization. CAM believes significant exploration potential remains at Cap Oeste and on adjacent exploration prospects controlled by PGSA.

1.6 Sampling

Sampling methods employed in the Cap Oeste 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: alternate insertion of a laboratory-certified laboratory standard or blank for every 10th sample;
- RC Drilling: For every alternate 10th 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; and
- Trenching: For every alternate 10th 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.

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

1.8 Density Measurements

A new series of 91 bulk-density (specific gravity) measurements were made, to replace earlier, suspect, measurements. The resultant densities of the various lithologies, oxidation types, and mineralization types ranged between 2.0 and 3.0 tons per cubic meter. The average specific gravity for the oxidized and un-oxidized portion of the three mineralization types is 2.47 and 2.51 respectively. The specified densities by type were used in the block model.

Further bulk-density measurements are needed to increase the precision of the values for each mineralized type present in the deposit.

1.9 Data Verification

CAM verified the database supplied by PGSA, using several electronic and manual methods. The database is suitable for use in mineral resource estimation.

1.10 Mineral Resources

Previous mineral-resource estimates were prepared by CAM in 2008 and by MICON in 2009, but neither report was filed in Canada, because Patagonia Gold was not a public company in Canada at that time. Both estimations were carried out in compliance with NI-43-101 standards.

The resource model was prepared by Patagonia Gold, and was reviewed and edited by CAM. Modeling was carried out initially through the completion of a detailed sectional interpretation on plotted sections and subsequently digitized to create a solid geological grade constraining wireframe model. Vertical sections were created every 25m perpendicular to the strike of the Cap-Oeste mineralization. Grade-constraining polylines were digitized in 3D and snapped to non-composited original samples based on the cutoff and recovery parameters of Au and Ag in oxides and sulfides.

Excellent continuity both along strike and down dip is seen in the Cap-Oeste mineralization model. Models were clipped to surface topography and oxide and sulfide levels were separated by a secondary profile delineating the depth of oxidation. Top cuts were not applied.

A block size of 5m x 5m x2.5m was chosen for the Cap-Oeste model, based on the tight drill spacing (25m x 25m average) for the deposit. Each rock type was assigned a separate block-model code and a partial model was built in GEMS to ensure that each rock type was assigned the correct tonnage from the wireframe model. Grade Interpolation was completed in 2 phases for both elements. Search ellipses were set up for each metal and each mineralization type, 6 in all.

The resources classified according to the criteria discussed above are shown Table 1-1.

Table 1-1
Resource Tabulation by Classification and EQAu Cutoff

Type	Class	CO	SG	Tonnes	Auppm	AuCTOZ	Agppm	AgCTOZ	AuEQppm	AuEQCTOZ
Ox	Ind	0.00	2.40	3153281	1.36	137546	50.62	5132009	2.30	233472
Ox	Ind	0.30	2.40	3150543	1.36	137532	50.66	5131724	2.30	233452
Ox	Ind	0.50	2.40	2960656	1.42	135584	53.49	5091842	2.42	230758
Ox	Ind	1.00	2.40	1902204	1.96	120102	75.19	4598505	3.37	206055
Ox	Ind	1.50	2.40	1308032	2.56	107552	95.55	4018285	4.34	182660
Ox	Ind	3.00	2.40	626558	4.02	81048	146.56	2952320	6.76	136232
Sul	Ind	0.00	2.50	4296531	3.09	426900	100.28	13851710	4.96	685811
Sul	Ind	0.30	2.50	4289881	3.10	426879	100.42	13850823	4.97	685773
Sul	Ind	0.50	2.50	4274411	3.11	426752	100.76	13846479	4.99	685565
Sul	Ind	1.00	2.50	4159067	3.18	424810	103.12	13789420	5.10	682556
Sul	Ind	1.50	2.50	3800106	3.40	415363	110.61	13513622	5.47	667954
Sul	Ind	3.00	2.50	2312629	4.59	341351	157.04	11676430	7.53	559602
Vn	Ind	0.00	2.50	771583	1.42	35346	27.24	675650	1.93	47975

Table 1-1
Resource Tabulation by Classification and EQAu Cutoff

Type	Class	CO	SG	Tonnes	Auppm	AuCTOZ	Agppm	AgCTOZ	AuEQppm	AuEQCTOZ
Vn	Ind	0.30	2.50	742524	1.47	35159	28.21	673457	2.00	47747
Vn	Ind	0.50	2.50	696914	1.55	34711	29.75	666481	2.11	47169
Vn	Ind	1.00	2.50	589469	1.73	32774	33.47	634239	2.35	44629
Vn	Ind	1.50	2.50	443866	1.95	27872	40.47	577567	2.71	38668
Vn	Ind	3.00	2.50	145618	2.86	13376	64.02	299717	4.05	18978
All	Ind	0.00	2.46	8221395	2.27	599792	74.38	19659369	3.66	967257
All	Ind	0.30	2.46	8182948	2.28	599570	74.71	19656004	3.68	966972
All	Ind	0.50	2.46	7931981	2.34	597047	76.88	19604803	3.78	963492
All	Ind	1.00	2.47	6650740	2.70	577686	88.96	19022165	4.36	933241
All	Ind	1.50	2.48	5552004	3.09	550787	101.45	18109473	4.98	889282
All	Ind	3.00	2.48	3084805	4.39	435775	150.52	14928467	7.21	714812
Ox	Inf	0.00	2.40	733965	0.79	18573	24.15	569767	1.24	29223
Ox	Inf	0.30	2.40	733965	0.79	18573	24.15	569767	1.24	29223
Ox	Inf	0.50	2.40	690843	0.81	18088	25.26	561063	1.29	28575
Ox	Inf	1.00	2.40	275836	1.25	11081	44.31	392964	2.08	18426
Ox	Inf	1.50	2.40	111489	1.91	6854	78.73	282205	3.38	12129
Ox	Inf	3.00	2.40	45821	2.82	4151	139.75	205876	5.43	7999
Sul	Inf	0.00	2.50	1379690	2.88	127671	69.28	3073016	4.17	185110
Sul	Inf	0.30	2.50	1379690	2.88	127671	69.28	3073016	4.17	185110
Sul	Inf	0.50	2.50	1379637	2.88	127670	69.28	3073012	4.17	185110
Sul	Inf	1.00	2.50	1366473	2.90	127426	69.82	3067542	4.21	184763
Sul	Inf	1.50	2.50	1237054	3.12	124077	74.55	2964878	4.51	179495
Sul	Inf	3.00	2.50	749931	4.18	100783	96.33	2322715	5.98	144199
Vn	Inf	0.00	2.50	309257	1.03	10239	23.78	236407	1.47	14658
Vn	Inf	0.30	2.50	306596	1.04	10221	23.96	236157	1.48	14635
Vn	Inf	0.50	2.50	292606	1.07	10068	24.86	233891	1.53	14440
Vn	Inf	1.00	2.50	201051	1.29	8363	32.22	208290	1.90	12257
Vn	Inf	1.50	2.50	142561	1.44	6591	39.00	178757	2.17	9932
Vn	Inf	3.00	2.50	11781	2.01	759	78.19	29615	3.47	1313
All	Inf	0.00	2.47	2422912	2.01	156483	49.80	3879190	2.94	228992
All	Inf	0.30	2.47	2420252	2.01	156465	49.85	3878940	2.94	228968
All	Inf	0.50	2.47	2363086	2.05	155826	50.91	3867966	3.00	228125
All	Inf	1.00	2.48	1843359	2.48	146870	61.90	3668795	3.64	215446
All	Inf	1.50	2.49	1491103	2.87	137521	71.46	3425841	4.20	201556
All	Inf	3.00	2.49	807534	4.07	105694	98.53	2558207	5.91	15351

CAM verified the results by running an unconstrained nearest-neighbor resource estimate using a different software system, in this case MicroModel. This check indicated that the PGSA model may be slightly conservative. CAM believes this conservatism is appropriate for a project at this level of development. On the basis of the review of the methodology, visual review of the model, and independent checks, CAM believes the model has been prepared according to accepted engineering practice and is suitable for initial reserve calculations. Some recommendations are shown below in Sections 1.12 and 26.

1.11 Interpretations and Conclusions

Cap Oeste definitely merits further exploration for additional gold-silver mineralization in an epithermal setting.

Substantial exploration has been completed on the Cap Oeste project since completion of the MICON report in 2009). This has included:

- Sufficient in-fill drilling in the resource area to increase drill intercept density to a nominal 25 meter centers;
- Exploration drilling along strike and down dip of known mineralization;
- Additional metallurgical and mineralogical studies;
- Exploration within the district, most notably leading to discovery of the nearby COSE deposit, which is now considered a separate project; and
- Cap Oeste definitely merits further exploration for additional gold-silver mineralization in an epithermal setting.

Review of the expanded database suggests that the previous geologic model for mineralization in substantially correct, but that a new resource can be calculated with a higher component of indicated as opposed to inferred resources.

CAM is of the opinion that this work meets or exceeds best industry practice, and that the resulting exploration database is suitable for use in mineral resource estimation.

1.12 Recommendations

1. The broad zones of stockwork mineralization where the Bonanza and Esperanza faults converge should be drilled, as this presents a potential, bulk-tonnage style target, especially where intersected by plunging ore shoots.
2. Additional drilling should be undertaken to prove the geometry and continuity of the higher-grade pods currently designated as Shoots C and D.
3. PGSA should proceed to generate a new mineral resource for the Cap Oeste project, when the data at hand warrant.
4. The Work Program in Table 1-2 is recommended in order to advance the project. Phase II is dependent on success in Phase I.

Table 1-2
Proposed Work Program, Cap Oeste Project

Item	Basis	Unit Cost, US$	Total Cost, US$	Time Period
Phase I				
Infill & Exploration drilling	50 holes @ 350m = 17,500 m	$ 160/m	$2,800,000	Q 3 & 4, 2011, Q1 2012
Other Drillholes (geotech, RQD, water, etc	5 holes @ 300 m= 1,500 m	$ 200/m	$ 300,000	Q 3 & 4, 2011
Camp, Geology, Assays	3900 m	$ 40/m	$160,000	Q 3 & 4, 2011
Geostatistics/Reporting	resource updates & similar	$ 50k/month	$150,000	Q4, 2011
Project Overhead,	B.A. office, 4months	$ 30,000/month	$120,000	Q4, 2011
SUBTOTAL, PHASE I			$ 3,530,000	
Phase II				
Metallurgical tests	estimate	$50,000	$50,000	Q1 & Q2, 2012
Pre-Feasibility Study	estimate	$600,000	$600,000	Q3 & Q4, 2012, Q1 2013
SUBTOTAL, PHASE I			$ 650,000	
TOTAL, PHASES I & II			$ 4,180,000	

2.0 INTRODUCTION

This report was prepared by Chlumsky, Armbrust and Meyer, L.L.C. (CAM) for Patagonia Gold S.A. (PGSA), to define a gold and silver resource at the Cap Oeste Project, Santa Cruz province, Argentina, which complies with Canada National Instrument 43-101 (NI 43-101). PGSA is a 100% 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 and unpublished data from former owners and explorers (Barrick and Homestake). The report includes data and analysis from contractors, consultants and certified laboratories. The authors' direct knowledge of the property is based on a site visit conducted on the 16th-17th August, 2011. During this time period, the undersigned examined outcrops and the locations of drill holes and surface samples, observed drilling and sampling of DDH and RC pre-collars, observed logging and sampling procedures and reviewed the Project with PGSA staff.

The effective date of the mineral resource estimate is 7 October, 2011, the date of receipt of assays for drillhole CO-300.

2.1 Qualified Persons

Craig Bow, Ph.D. Geology, and Robert Sandefur, P.E., both 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. Sandefur is responsible for report 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.

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

Additional individuals beside the undersigned provided data for this report. These included Alejandra Jindra, Cap Oeste Project Geologist, and Damien Koerber, geologic consultant. 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 Baade.

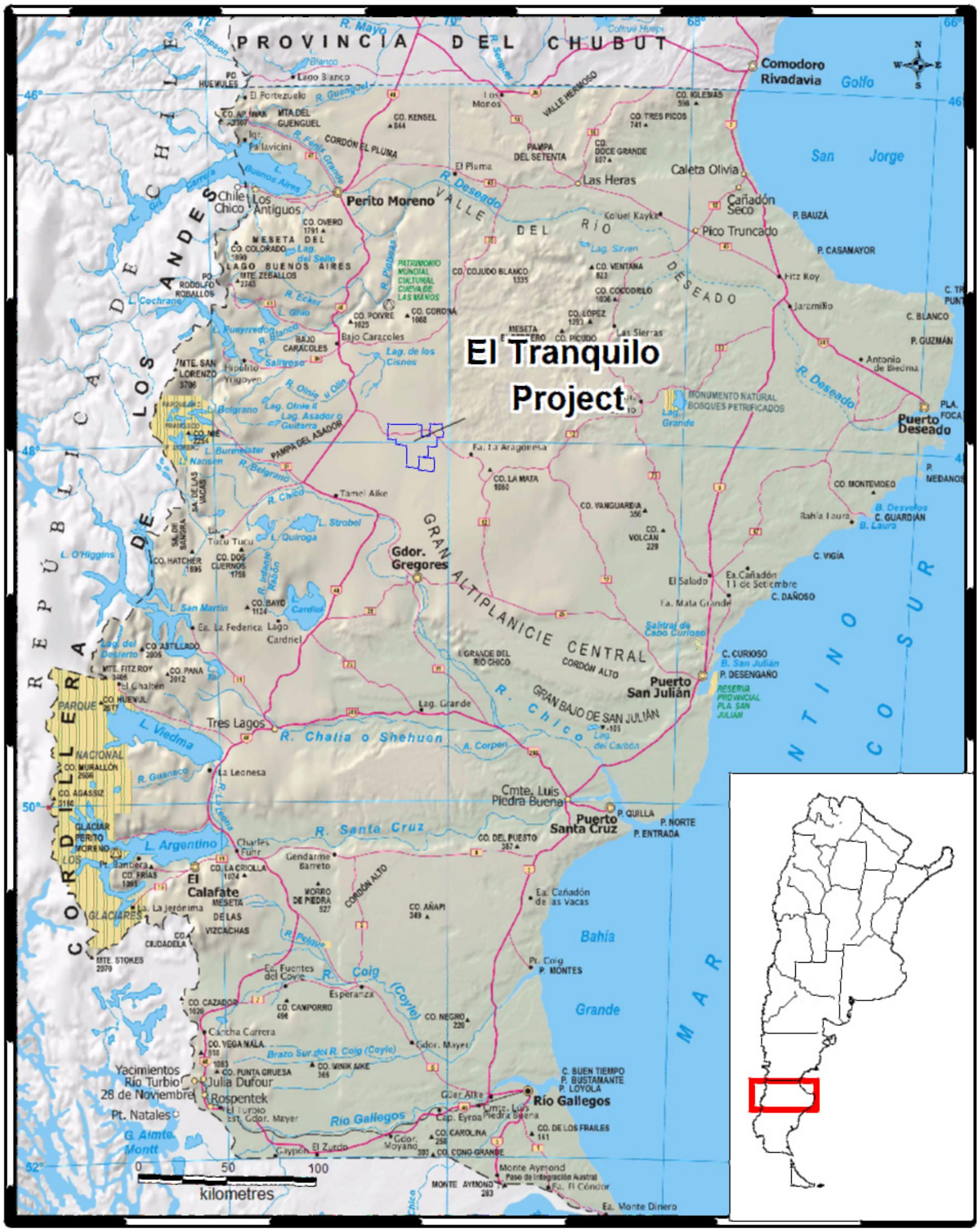


Figure 4-1
Project Location

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.

The Cap Oeste Project area sits within an area which was territorially zoned as a ¨Area of Special Mining Interest” (AEMI) by the Santa Cruz Provincial government which was established in 2009 (see Figure 4-2). Throughout the AEMI the Santa Cruz Province actively supports and encourages exploration and mining.

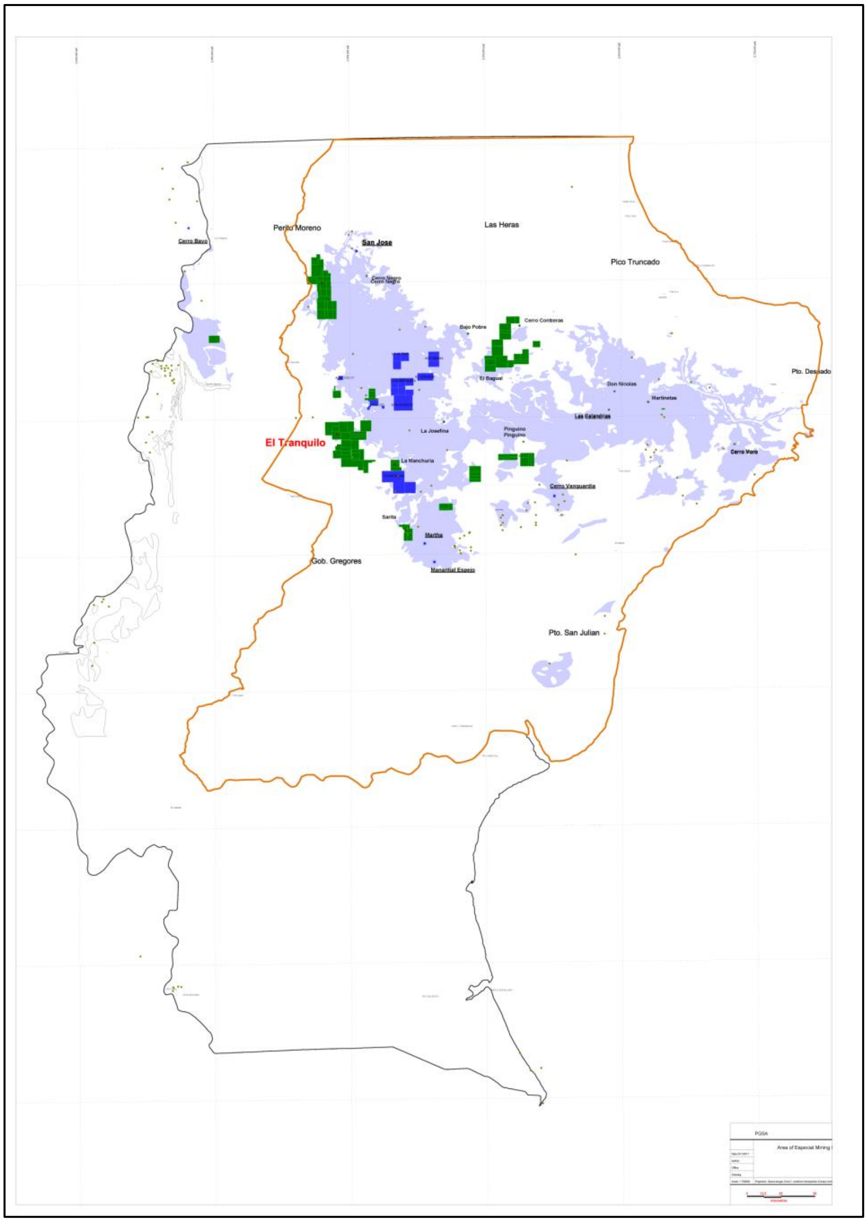


Figure 4-2
Area of Special Mining Interest

4.2 Mineral Tenure and Title

4.2.1 Cap Oeste Project-Patagonia Gold S.A. - Exploration Claims

The Cap Oeste Project is located within the El Tranquilo I Manifestation of Discovery (MD) claim, which is one of seventeen contiguous exploration tenements comprising the El Tranquilo block of properties (79,513hectares) 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). A "cateo" is an initial exploration permit which needs to either be converted to an MD or relinquished after a set period, depending on the mineral potential of the property. The MD El Tranquilo MD (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 location of the Cap Oeste Project area with respect to the El Tranquilo MD claim is displayed in Figure 4-3. The coordinates for the vertices of each property are provided in Table 4-1.

Table 4-1
El Tranquilo Block of Exploration Properties

Name	Property Type	Property File No.	Area (hectares)
La Marcelina	Cateo	412.792/B/04	6,479.00
Venus	MD	402.092/PG/05	2,735.00
La Bajada	MD	404.562/PG/05	5,000.00
La Apaciguada	MD	405.473/PG/05	3,461.00
Monte Puma	MD	406.881/PG/06	1,993.00
Monte Tigre	MD	406.882/PG/06	1,994.00
Marte	MD	409.148/PG/06	1,495.00
Enriqueta	MD	412.519/PG/06	740.70
Maria	MD	412.520/PG/06	2,492.00
La Mansa	MD	413.543/PG/06	1,731.00
Monte Leon	MD	415.664/MR/07	1,981.00
El Tranquilo I	MD	403.094/PG/07	3,724.00
La Cañada I	MD	403.985/PG/07	2,786.00
Las Casuarinas	Cateo	424.914/PG/09	3,626.00
El Mangrullo	Cateo	424.915/PG/09	4,275.00

Table 4-1 El Tranquilo Block of Exploration Properties			
Name	**Property Type**	**Property File No.**	**Area (hectares)**
El Aljibe	Cateo	424.916/PG/09	6,683.00
La Cañada II	MD	427.259/PG/09	1,938.00
Cerro Leon I	MD	423.845/PG/09	3,955.00
La Cañada III	MD	421.630/PG/10	2,752.00
Nueva España II	Cateo	422.216/PG/10	4,447.00
Nueva España I	Cateo	422.217/PG/10	9,955.00
Don Francisco	Cateo	423.465/PG/10	1,798.00
Nuevo	Cateo	423.670/PG/10	3,473.00

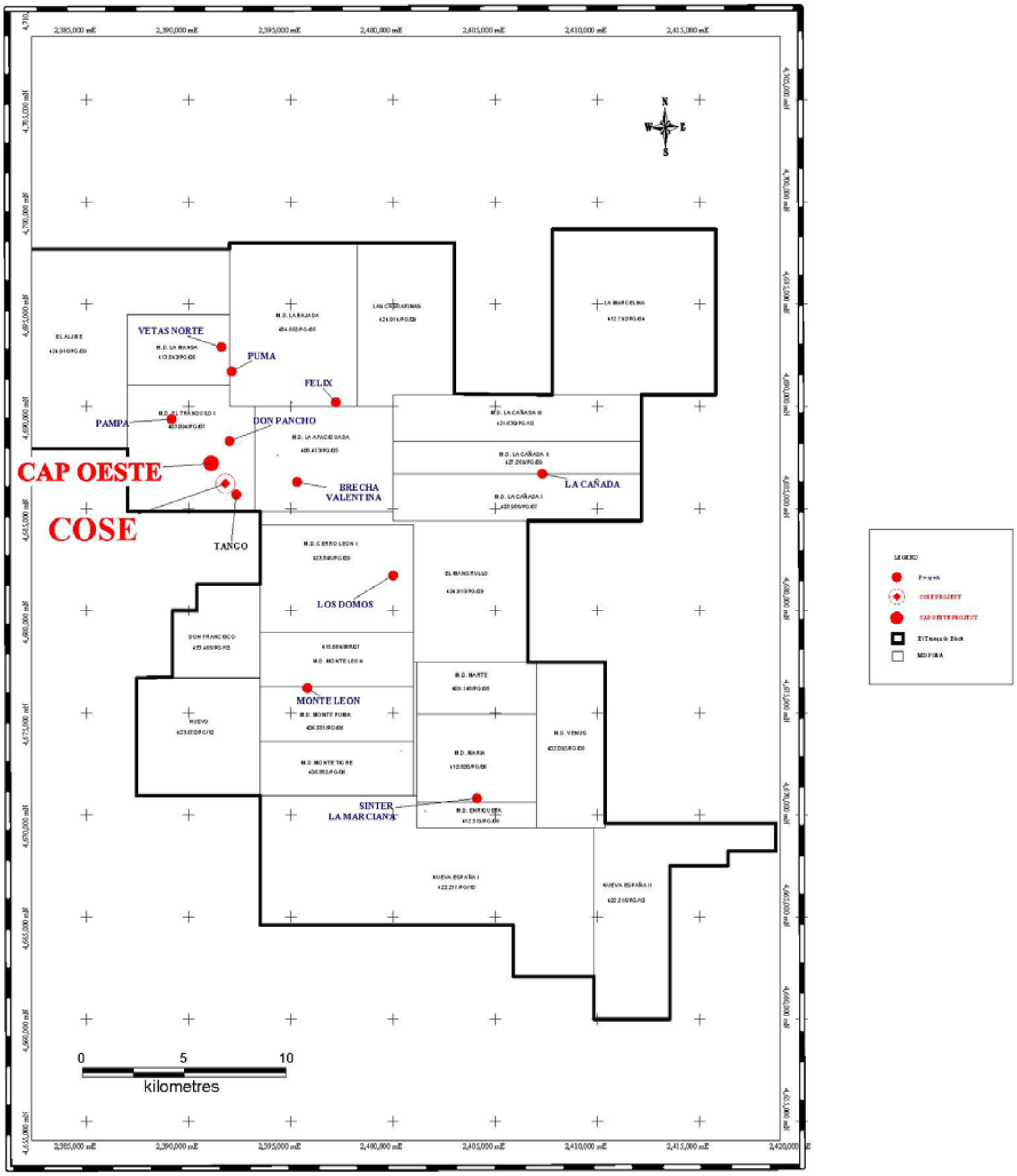


Figure 4.3
Location of the Cap Oeste Project Area
in relation to the COSE project and the El Tranquilo I MD Claim

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'.

The Cap Oeste Project, as defined here, excludes the COSE project, which contains a small, very high-grade mineralized body (CAM, 2011) which is being intensively investigated by PGSA. The excluded area, containing the COSE project, is in a 250 m by 250 m square with lower left corner at northing 4,686,600.00, easting 2,391,400.00, and rotated counter-clockwise 40 degrees about this lower left corner. Mineralized areas outside of this rotated square are in the Cap Oeste project (this report). COSE is the subject of a separate Technical Report.

Surface Rights and Obligations

Surface rights in Argentina are not associated with title either to a mining lease or to an exploration claim and, therefore, must be negotiated with the surface landowner on an individual basis. The Cap Oeste deposit is located mainly on PGSA´s Estancia La Bajada farm bought by the Company in December 2008. The whole Cap Oeste project is transected by the boundary between Estancia La Bajada and the contiguous farm property Estancia El Tranquilo.

In July, 2011, PGSA signed an extension to the pre-existing Access and Exploration Agreement with Ms. Susana Martinic, the landowner of Estancia El Tranquilo, which permits access, road repair and construction, trenching and exploration tasks, including certain drilling on Cap Oeste during the period up until the 30 March, 2012. The Agreement is renewable and will expire in case of the purchase of the property.

Mineral Property Encumbrances

Some 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 Argentinian subsidiaries of Barrick, Minera Rodeo S.A. and Barrick Exploraciones Argentina S.A.

As an integral part of both Barrick's and PGSA's due diligence it was verified that there were no other mineral property encumbrances over the Project or block of properties, except those agreed on the terms and conditions of this Purchase Agreement include:

A US$10,000,000 commitment of approved exploration expenditures within a period of five years, of which US$1,500,000 to be invested during the first 18 months. PGSA notified Barrick´s subsidiaries advising that the investment commitments of US$1,500,000 and US$10,000,000 were exceeded as of

December 31, 2007 and December 31, 2008, respectively. There were no other remaining investment commitments.

PGSA was 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. PGSA delivered to Barrick the previous NI 43-101 Resource Technical Reports prepared by CAM and by MICON to Barrick.

Barrick's Argentine subsidiaries retained a right to 'back-in' up to 70% for any individual property group included in the Purchase Agreement upon written notice, within 90 days upon completion of a NI 43-101 compliant delineation of a two million ounce gold or gold equivalent Indicated Resource, within the respective property group on a forward looking basis which does not include any resources or reserves produced or undergoing development.

On March 2011 PGSA signed with Barrick´s subsidiaries an Amendment to the original Purchase Agreement, eliminating the 'Back in Right' clause in exchange for a 2.5% Net Smelter Return (NSR) Production Royalty. The requirement to provide annual year-end mineral resource statements no longer exists.

"NSR" or "NET SMELTER RETURN" shall mean the net income or profit actually collected from any source, smelting plant, refinery or the sale of mineral products obtained from the Mining Properties (hereinafter, the "Mineral Products") after having deducted the following expenses from the gross income or profit:

- smelting and refining expenses (handling, processing, supplies and sampling expenses, costs of smelter assays and umpire assays, representatives' and arbitrators' fees, fines, wastage and any other expense or loss related to the smelter and/or refining process);
- transportation costs (loading, freight, unloading, handling at port, stowage, demurrage at ports, delays, customs expenses, transaction, handling, haulage and insurance) of ore, metals or concentrates of the products obtained from the place where the Properties are located to any source, smelting plant, refinery or point of sale;
- commercialization costs;
- cost of insurance covering Mineral Products, and customs duties, compensations, state royalties, ad valorem taxes and taxes in general, either levied on production or sale of the minerals or the like, taxes on the use of natural resources existing at the time this Agreement becomes effective or created in the future, export or import taxes or duties on the Mineral Products payable to national, provincial or municipal governmental agencies; and
- royalties payable to any national, provincial or municipal governmental agency or entity.

The following mining properties included in the El Tranquilo block were granted directly to PGSA, either previous to or subsequent to the signing of the Purchase Agreement with Barrick, and, therefore, are not subject to its terms and conditions or the payment of the Net Smelter Return (NSR) Production Royalty to Barrick.

• 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 Francisco	423.465/PG/10
• Nuevo	423.670/PG/10

<u>Environmental Liabilities</u>

No previous mining activity has been conducted on the El Tranquilo block. All the exploration works at El Tranquilo block have been carried out as per the pertaining biannual Environmental Impact Assessments approved by the relevant provincial mining authorities. To the best of our knowledge, the property is not subject to any environmental liabilities related to exploration or mining activities.

<u>Permits</u>

Work at the Cap Oeste project has been conducted in accordance with the legal requirement for an approved biannual Environmental Impact Assessment (EIA) for the El Tranquilo Project block, for which the pre-existing one was renewed and subsequently approved and granted by the State Secretariat of Mining of the province of Santa Cruz on 4 November 2010, with an effective duration of two years.

The approved EIA included a provision for 400,000 metres of drilling on the Cap-Oeste deposit and other prospects within the El Tranquilo block and the development of a decline access at COSE for underground drilling and bulk sampling for metallurgical test works.

PGSA has been collecting meteorological data through its Meteorological Station, and conducting quarterly baseline water sampling throughout the project area since May, 2007, and have been producing independent reports prepared by a private consultant (BEHA). Results of these studies were included in the most recent EIA for the project and submitted to the appropriate authorities.

PGSA has obtained the relevant permits for the use of water during the drilling 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.

A Community Relationship Plan has been implemented since 2008 through the Community Relationship Department of the Company together with the external consultancy of Empoderar RSE.

Since March 2011 the Company has retained Ausenco Vector to commence the Baseline Studies, with the objective of establishing the pre-development environmental and social characteristics of the project and its surrounds, and to prepare an Environmental Impact Assessment for the mining of COSE deposit.

The Baseline Studies and the Environmental Impact Study will detail the Project's area environmental aspects analyzed, including:

- Physical aspect: Geology, Geomorphology, Seismology, Soils, Hydrology, Water Quality (Surface and Groundwater), Air Quality, Acid Rock Drainage, Climate and Hydrogeology among others.
- Biological aspect: Ecosystem Characterization, Flora, Fauna, and Limnology.
- Socio-economic aspect: Archaeology and Paleontology, Landscape, Traffic, Legal and Socio-economic Study.
- Project aspects: Description of the development plan, evaluation of the impacts, description of the environmental management and contingency plans including management of waste and water, and mine closure and post mining monitoring.

The EIA is expected for completion on November 2011, when it will be presented to the State Secretary of Mining for their review, with approval expected for early 2012.

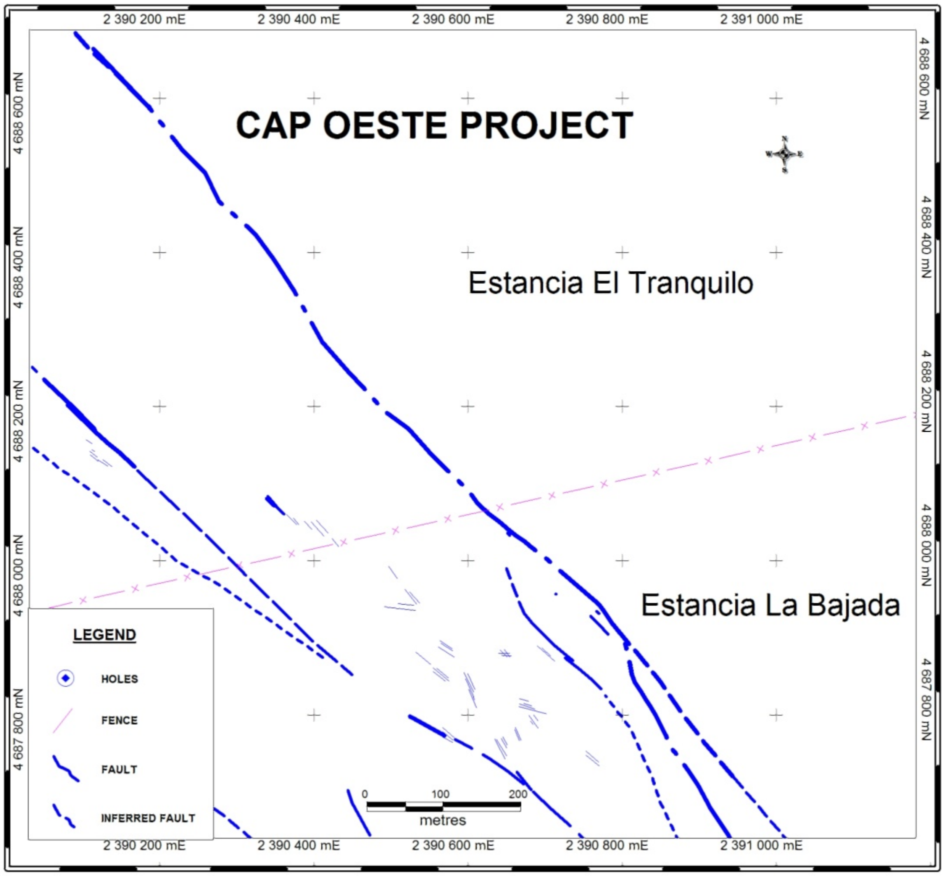


Figure 4-4
Location of Cap Oeste Project area in relation to the fenced boundary between the Estancia El Tranquilo and the Estancia La Bajada

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 to 1,000 meters above sea level (m.a.s.l). 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 500m of the Cap Oeste 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 (mm), 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 than 40 kilometres 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 Cap Oeste Project area is characterized by a predominant northwest-southeast aligned pattern of undulating hills between elevations of 350 and 500 m.a.s.l. In the northwestern portion of 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 called "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 from within the northwestern portion of the Project area to at least approximately 2km further to the northwest. Water supplies for drilling and exploration camp amenities are obtained from these aforementioned 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 along its entirety as part of a major public infrastructure works program. This project is scheduled to be completed by the end of 2012.

It should be noted that, on rare occasions (perhaps once per decade) access to the property is affected by falls of volcanic ash emanating from volcanos in the Chilean Andes to the west. Such events may hamper access for periods of days to weeks. The last such major occurrences were from Mt. Puyehue in volcano in mid-2011 for 10 weeks and from Mt Hudson volcano in 1991, lasting for 12 weeks.

Within each individual regional population centre, 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 kilometres 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 Empoderar RSE 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 Cap Oeste 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

PGSA staked the cateo 'La Bajada' in 2005 and the exploration claims 'Casuarina', 'El Aljibe' and 'El Mangrullo' in 2009 and Nueva España I, Nueva España II, Don Fransisco and Nuevo in 2009.
None of these properties are subject to the terms and conditions of the Purchase Agreement signed in 2007 with Barrick.

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 meters).
- Pole-Dipole Induced Polarization and resistivity surveying along 8 lines spaced 150 to 300 meters apart, totaling 27 line kilometres.
- Regional spaced ground magnetic surveying along 16 lines spaced 100 meters 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 identified in order to help guide subsequent exploration.

As follow-up at Cap Oeste, Barrick took a total of 144 lag samples covering a 600 meter long, northwest trending zone of poorly exposed hydrothermal breccia, silica/hematite flooding, and sheeted, limonitic quartz veining. This sampling returned weakly anomalous Au, As and Hg values over an approximate 300-meter wide by 800-meter long area. Barrick tested this anomaly with three trenches (TR 4 –TR 6) totaling 420 meters, which were excavated perpendicular to the exposed mineralization over a strike length of 270 meters. Significant values were returned from two of these trenches, approximately 145 meters apart:

- Trench TR-4: 38 m @ 1.0 ppm Au (using 0.25 ppm Au cutoff) including
  7.5 m @ 1.88 ppm Au, and 33 ppm Ag (using 1.5 ppm Au cutoff)
- Trench TR-5: 14.8m @ 0.55 ppm Au (using 0.25 ppm Au cutoff) including
  2 m @ 1.05 ppm Au (using 1.0 ppm Au cutoff)

The mineralization was spatially coincident with a prominent chargeability and resistivity anomaly highlighted by the one Pole-Dipole Induced polarization (IP) and resistivity survey line that transected the zone.

In summary, the Homestake-Barrick exploration program defined a 10 km by 25 km, northwest-trending, epithermal district hosting extensive zones containing anomalous precious and trace element metal contents, hydrothermal alteration, and coincident chargeability/resistivity targets. Within this area, at least three main corridors have been broadly delineated, namely the Cap Oeste, Breccia Valentina, and Vetas Norte Corridors, as shown in Figure 6-1.

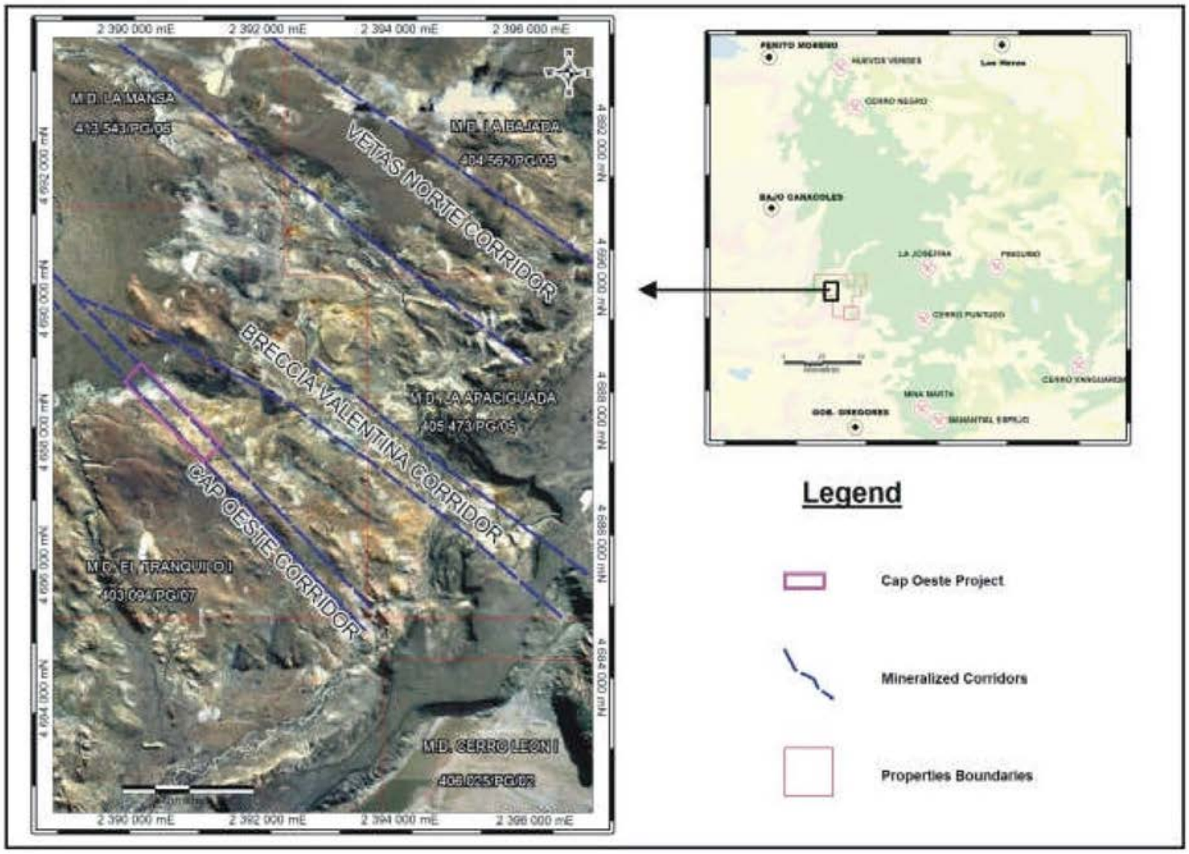


Figure 6-1
Regional Structural Corridors – El Tranquilo Property
(Figure courtesy of Patagonia Gold)

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 9 through 12 of this Technical Report.

As part of its exploration activities on the El Tranquilo land holdings in 2008, PGSA commissioned an initial mineral resource estimate by CAM of the mineralization that had been delineated at the Cap Oeste deposit at that time. The results of the mineral resource estimate have been presented in CAM (2008), and are summarized in Table 6-1. The CAM report was not filed in Canada, as Patagonia Gold was not listed in Canada; however a summary of it (at a uniform 0.3 g/t cutoff) is available on Patagonia Gold's website (see References).

Table 6-1 Cap Oeste 2008 Resource estimate (CAM, 2008)						
Classification	**Gold Cutoff Grade (g/t)**	**Tonnes**	**Gold Grade (g/t Au)**	**Contained Gold (oz)**	**Silver Grade (g/t Ag)**	**Contained Silver (oz)**
Total, Indicated	0.30	3,228,222	1.44	149,842	35.15	3,647,751
Total, Inferred	0.30	3,384,347	1.42	154,257	30.16	3,282,074

1. Mineral resources which are not mineral reserves do not have demonstrated economic viability. The estimate of mineral resources may be materially affected by environmental, permitting, legal, title, taxation, sociopolitical, marketing, or other relevant issues.
2. The quantity and grade of reported Inferred Resources in this estimate are conceptual in nature and there has been insufficient exploration to define these Inferred Resources as an Indicated or Measured Mineral Resource. It is uncertain if further exploration will result in the upgrading of the Inferred Resources into an Indicated or Measured Mineral Resource category.
3. The quantity and grade of reported Inferred Resources in this estimation are conceptual in nature and there has been insufficient exploration to define these Inferred resources as an Indicated or Measured Mineral Resource. It is uncertain if further exploration will result in the upgrading of the Inferred Resource into an Indicated or Measured Mineral Resource category.

Following the 2009 exploration season, PGSA elected to commission a second resource estimate to include the expanded drill database; these results are presented in MICON International (2009) and summarized below as Table 6-2. The MICON report was not filed in Canada because Patagonia Gold was not a listed company in Canada, but it is available in its entirely on Patagonia Gold's website (see References).

Table 6-2 Cap Oeste Cap Oeste 2009 Resource Estimate (MICON, 2009)					
Category	**Tonnes**	**Au capped, OK**	**Oz Au capped**	**Ag capped, OK**	**Oz Ag capped**
Indicated	5,629,645	1.89	342,120	65.04	11,773,380
Inferred	1,053,990	1.35	45,750	41.34	1,401,030

1. OK refers to Ordinary Kriging
2. Contained ounces rounded to nearest 10 oz
3. The average density of 2.39 t/m3 used to determine the tonnage is derived by application of a correction factor of +7.5% to the average density as determined by PGSA.
4. Mineral resources which are not mineral reserves do not have demonstrated economic viability. The estimate of mineral resources may be materially affected by environmental, permitting, legal, title, taxation, sociopolitical, marketing, or other relevant issues.
5. The quantity and grade of reported Inferred Resources in this estimate are conceptual in nature and there has been insufficient exploration to define these Inferred Resources as an Indicated or Measured Mineral Resource. It is uncertain if further exploration will result in the upgrading of the Inferred Resources into an Indicated or Measured Mineral Resource category.
6. The quantity and grade of reported Inferred Resources in this estimation are conceptual in nature and there has been insufficient exploration to define these Inferred resources as an Indicated or Measured Mineral Resource. It is uncertain if further exploration will result in the upgrading of the Inferred Resource into an Indicated or Measured Mineral Resource category.

7.0 GEOLOGICAL SETTING AND MINERALIZATION

7.1 Regional Setting

The Cap Oeste Project is contained within the Deseado Massif geological province, which occupies a 70,000 square kilometres area in the northern third of Santa Cruz Province. The geology of Santa Cruz has been mapped and compiled at 1:750,000 scale, and 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.

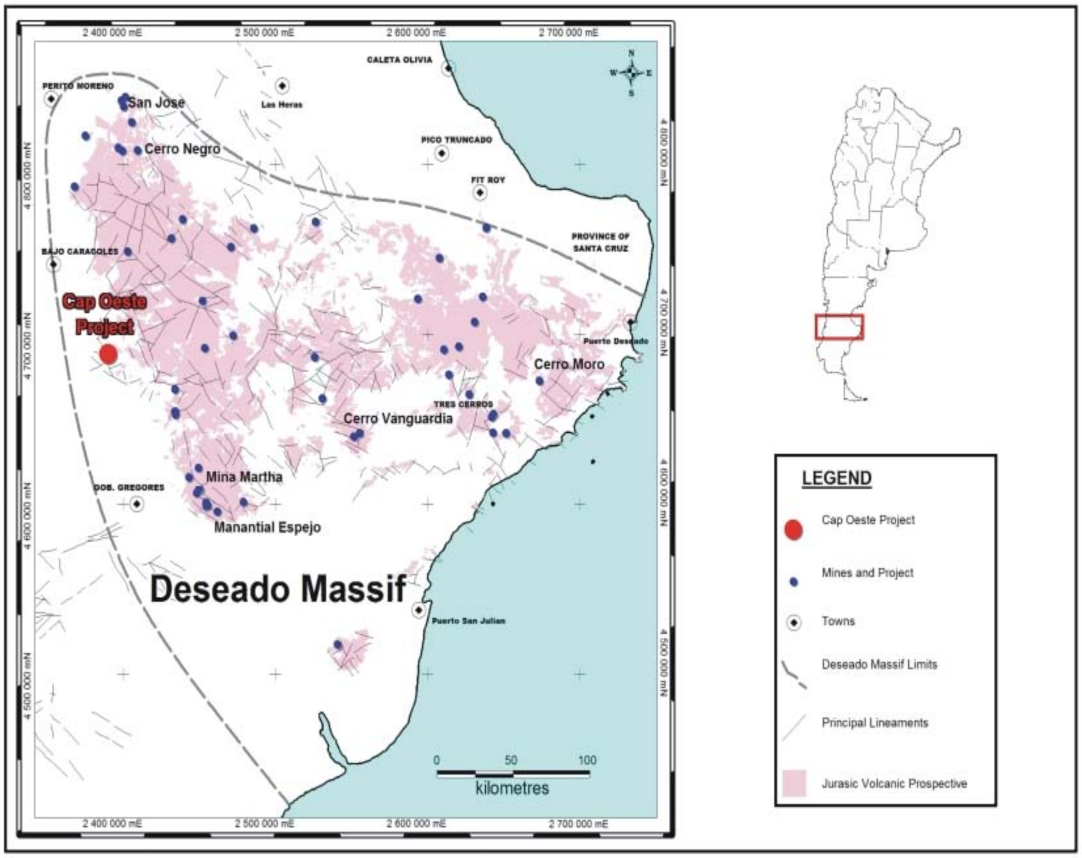


Figure 7-1
Regional Geology of the Deseado Massif, Santa Cruz Province, Argentina

Within the project area, the Jurassic volcanic suite is comprised dominantly of rocks assigned to the Bahia Laura Group (Figure 7-2). 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:

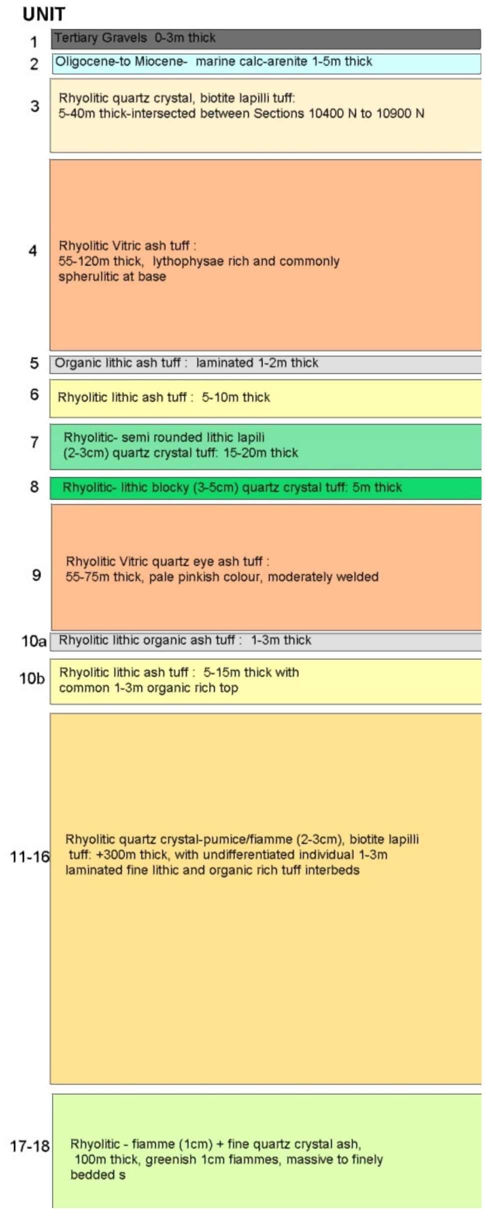


Figure 7-2
Regional Stratigraphy

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 Cap Oeste Project and the surrounding region are consistent with regional east-west to northeast-southwest extension as has been documented for many low sulphidation, epithermal precious metal deposits throughout the province.

7.2 Property Geology

7.2.1 Stratigraphy

The bedrock in the Cap Oeste area comprises a thick (greater than 500 m) sequence of rhyolitic ignimbrite and tuff units of the Chon Aike Formation, overlain by a veneer of Oligocene-to Miocene-aged shallow marine calc-arenite sediments of the Centinela Formation. These are in turn overlain by unconsolidated, Quaternary-aged fluvio-glacial gravels. The stratigraphy is shown in Figure 7-2. The surface and subsurface distribution of these units defined by mapping and drilling, excluding outcrop of the Quaternary gravel cover, is shown in Figure 7-3.

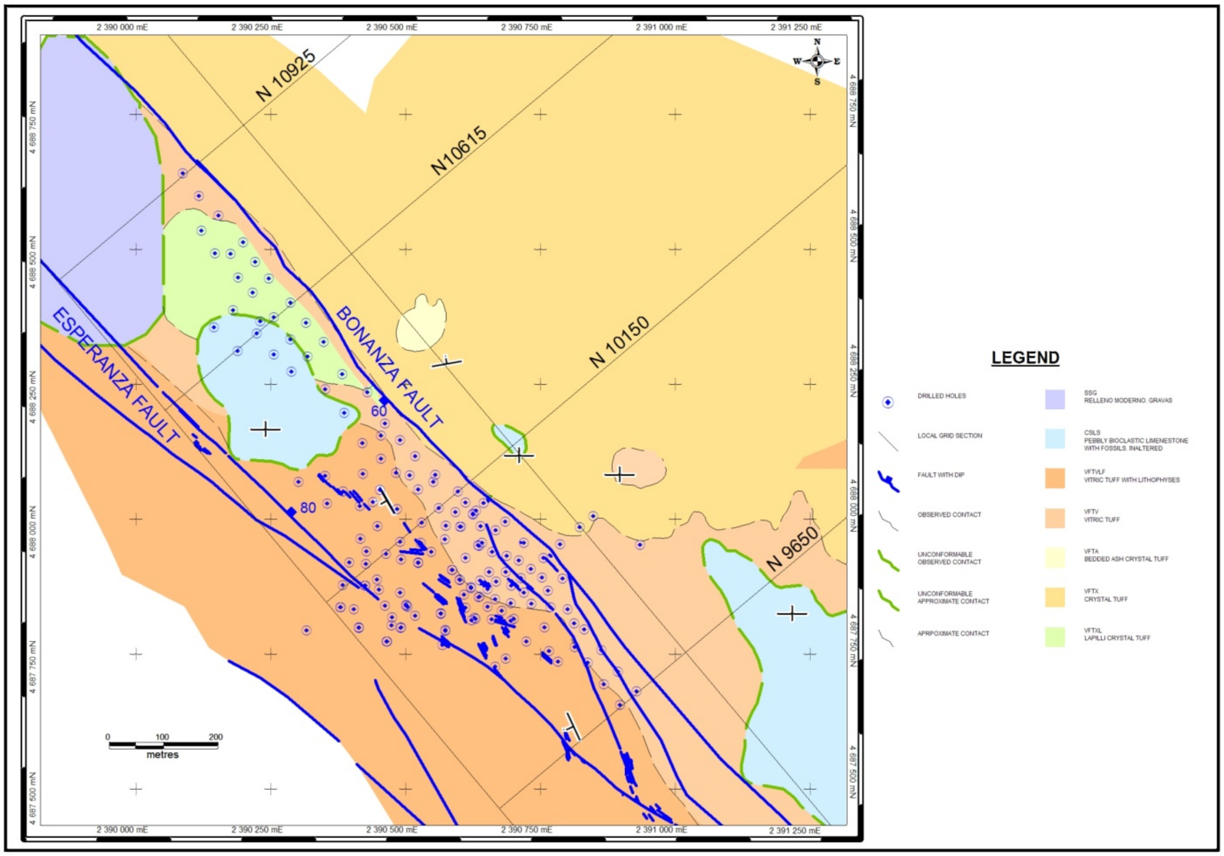


Figure 7-3
Geology and structure of the Cap Oeste Project

From information gained from drilling completed throughout the Cap Oeste project area, the local stratigraphy of the Chon Aike Formation has been defined by PGSA geologists into eight sub-horizontal units throughout an approximate 200 by 1200 m area down to a maximum vertical depth from surface of varying between 100 to 500 m. The surface distribution of the various members of the Chon Aike Formation is strongly controlled by a series of at least two sub-parallel, northwest-trending (320º), moderate to steeply southwest and northeast dipping normal faults, respectively named the Bonanza and Esperanza Faults as shown in Figure 7-4.

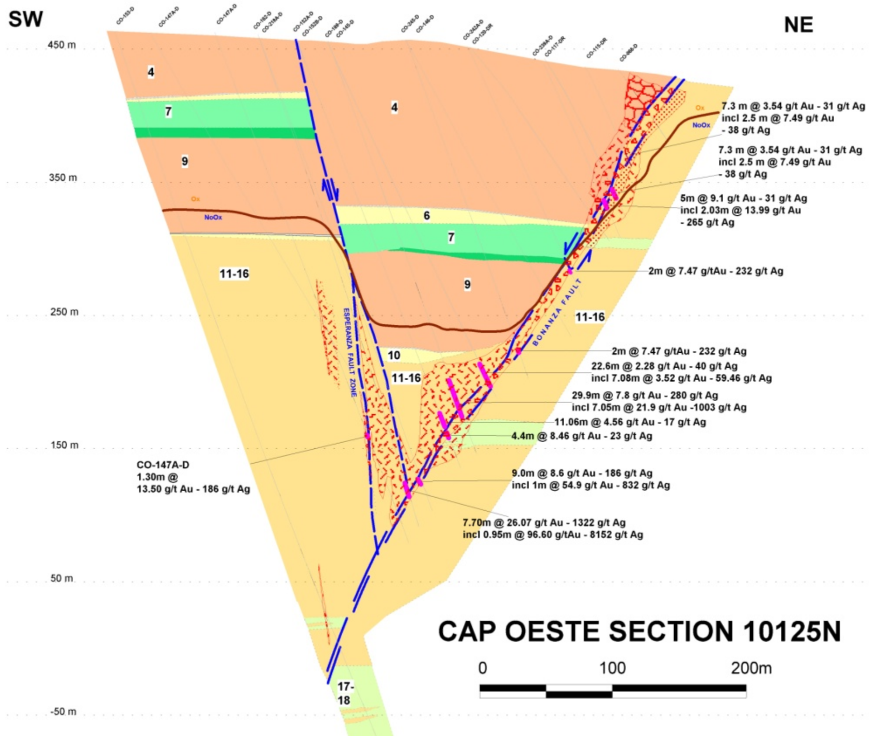


Figure 7-4
Cross Section 10125 N (Looking Northwest)
Showing the Orientations of the Esperanza and
Bonanza Faults and Significant Mineralized Intervals

In summary, the upper 200 m of stratigraphy that hosts the Cap Oeste mineralization defined to date has been found to comprise a crystal-poor, rhyolitic, vitric ignimbrite whose distribution is restricted to the hanging wall to the Bonanza Fault and the upper footwall section of the Esperanza Fault. In contrast, the near-surface stratigraphy of the footwall of the Bonanza Fault is dominantly comprised of a quartz-feldspar-biotite crystal-rich ignimbrite unit.

At depth (i.e. below 0 m RL or approximately 450 m below surface), a series of additional units, typically comprising ignimbrite flows ranging from 25 to 100 m in thickness that are separated by thin interbeds of volcano-sedimentary material, are found in the footwall of the Bonanza Fault.

A more detailed description of the individual lithologies and their respective thicknesses is provided below in order of increasing depth from surface, and relative to their positions with the Esperanza and Bonanza Faults.

Footwall (FE) and Hangingwall (HE) to Esperanza Fault

- (VfTv) Rhyolitic Vitric ash tuff (< 0.25mm diameter) with abundant drusy lined, 2 to 4 cm diameter, flattened lythophysae-rich interval (10 m thick) with weakly welded and partially devitrified volcanic glass ash fragments. This unit comprises the prominent topographic high nominated Cap Oeste, which occurs immediately to the southwest of the mineralized zone. To date, current drilling has intersected a 50-m thick portion of this unit in the footwall to the Esperanza Fault, and throughout the block comprising both the hangingwall to the Esperanza and Bonanza Faults, a 135-m thick portion of this unit has been defined. The relatively increased thickness of this unit in the hangingwall to the Esperanza Fault is interpreted to be a result of the relatively greater preservation of this normally displaced downthrown stratigraphy within the interpreted graben (i.e. between the Esperanza and Bonanza Faults).
- (VfTl) 20-m thick Rhyolitic Lapilli tuff (0.4 to 3 cm in diameter) characterized by a fining upward sequence comprising basal block to lapilli and upper laminated, variably carbonaceous ash tuff 1 to 5 m thick.
- (VfTb) 5 to 10-m thick Rhyolitic Block tuff (>3 cm diameter).
- (VfTv) 70-m thick Rhyolitic Vitric quartz eye ash tuff –moderately welded.
- (VfTvamo) 5-m thick Organic rich laminated rhyolitic ash tuff.
- (VfTx) Rhyolitic quartz, plagiaoclase biotite, crystal coarse ash (0.25 to 4 mm diameter).

Although this lowest unit appears to host proportionally more quartz crystal fragments, it is currently unclear if this unit can or cannot be correlated with the crystal tuff in the footwall to the Bonanza Fault (FB Unit 2). To date, the limit of current drilling has defined a minimum 290 m thickness of this unit in the footwall to the Esperanza Fault and intersected the 80-m thick upper portion of this normally displaced unit in the hangingwall to the Esperanza Fault.

Between sections 10025 N and 10150 N, many of the finer grained units in the lower portion of the stratigraphy in the central graben structure appear to pinch out peripheral to the Bonanza Fault contact, suggesting at least part of the units' deposition was influenced by syn-depositional faulting.

Additionally, to the northwest (between sections 10400 N to 10900 N), a 5-to 40-m thick rhyolite quartz, biotite, plagioclase crystal tuff (rock code VfTxp) containing abundant 0.52-cm sized pumice fiamme clasts has been defined by drilling to be present that conformably overlies the vitric tuff unit (rock code VfTv). This unit underlies a veneer of post Jurassic aged cover rocks, and its lateral extension is confined to between the Bonanza Fault and the Esperanza Fault.

Footwall to the Bonanza Fault (FBz)

- (VfTv) 5 to 20-m thick Rhyolitic vitric ash (<0.25 mm) tuff weakly welded and partially devitrified volcanic glass ash fragments. Given the relative position of this unit elsewhere within the volcanic stratigraphy of the project area (i.e. not found stratigraphically conformable above quartz + plagiaoclase + biotite crystal tuff), it is interpreted that this unit may have possibly been deposited unconformably on the underlying Unit 2 (below) as possibly the latter stage of deposition of the much thicker accumulation of vitric tuff defined in the hanging wall side of the Bonanza Fault (i.e. to the southwest).
- (VfTx) +500-m thick Rhyolitic quartz+plagiaoclase+biotite, crystal coarse ash (0.25 to 4-mm), including two, 10 to 20-m thick interbeds of fiamme and lithic lapili and towards the base of the current limit of drilling, 5-m thick interbeds of organic rich laminated ash volcanic units

7.2.2 Paleo-Volcanic Depositional Setting

These volcano-sedimentary sequences are found in the central portion of the graben structure formed between the Esperanza and Bonanza Faults and are interpreted to represent an extra-caldera, paleo-depositional setting in which a depressed area received numerous ignimbrite flows, potentially erupted from more than one ash-flow caldera centre. The depositional setting must have been of low relief, judging by the commonly fine grain size of the epiclastic rocks and the abundance of carbonaceous intervals reflecting vegetation growth, possibly in a lacustrine environment. The lack of coarse-grained epiclastic rocks near the Bonanza Fault may be indicative that a prominent scarp was not present during the volcanism and sedimentation.

In several localities which fall within a 6km radius of the Cap Oeste Project area relative small exposures (100m x 200m) of dacitic to rhyolitic domes have been mapped as intruding stratigraphy of the Chon Aike Formation, for which no age dating has been conducted. These domes are commonly characterized by auto and hydrothermally brecciated carapaces and host strong flow foliation.

7.3 Mineralization

7.3.1 Regional mineralization

The Deseado Massif volcanic province hosts several producing and advanced stage projects as summarized in Table 7-1.

Table 7-1
Selected Gold-Silver Deposits of the Santa Cruz Deseado Massif, Argentina

Deposit	Past Production /Remaining Resources Million Oz	Resource-Metric Tonnes (million)/ Grade g/t Au-Ag	Operation Type	Plant Type/ Annual Production '000oz	Ownership	Data Source
Cerro Vanguardia	2.9 Au/ Vein 3.03 Au / 24.9 Ag Heap Leach 0.53 Au, 44 Ag Vein Underground 0.58 Au,	Vein 15.49/6.09 Au, 63.9 Ag Heap Leach 25.11/0.66 Au, 63.9 Ag Vein Underground 1.58/8.01 Au	Open Pit/ Underground	CIL /Heap Leach : 209 Au , 2,800 Ag	AngloGold Ashanti 92.5%Formicruz 7.5%	'Mineral Resource and Ore Reserve Report 2010' http://www.anglogold.com/subwebs/InformationForInvestors/Reports10/financials/files/AGA-resource-reserves-2010.pdf
Marta Mine	18 Ag /2.7 Ag, 0.003 Au	0.25/328Ag, 0.44 Au	Underground	Flotation Concentrate 1,600 Ag, 1.84 Au	Coeur d'Alene Mines Corporation	Reserves Table Dec 2010 http://www2.coeur.com/resources-table.html
Manantial Espejo	3 Ag/ 55.14 Ag/,0.745Au	15.6/ 110.1 Ag, 1.49 Au	Open pit /Underground	CIL 3960 Ag, 62.8 Au	Pan American Silver	Reserves Table Dec 2010 http://www.panamericansilver.com/operation/argentina215.php
Cerro Negro Project	5.36 Au, 43.7 Ag	22.18/7.5 Au, 61 Ag	Planned Open pit /Underground	Feasibility	Goldcorp	Goldcorp 2011 http://www.goldcorp.com/operations/cerro_negro/
San Jose	47.7 Ag, 0.82 Au	7.5 / 6.0 Au, 455 Ag	Underground	CIL/Gravity 84.3 Au, 5,324 Ag	Hochschild Mining 51% plc. /Minera Andes 49%	Minera Andes 2010 http://www.minandes.com/projects/san-jose-mine/default.aspx

Throughout the northern portion of the El Tranquilo Block exploration claims, PGSA has defined several areas hosting Au-Ag mineralization and pathfinder geochemical anomalism (e.g. As, Sb, Hg) based on historic Barrick and recent PGSA exploration data. These areas are spatially related to three, 2-3km spaced, and northwest trending regional scale mineralized structural corridors, namely the Cap Oeste, Don Pancho and Vetas Norte corridors (Figure 7-5). These corridors extend throughout an approximate 8-kilometer wide by 10-kilometer long window of variably clay-silica-Fe oxide altered Chon Aike volcanic rocks which is surrounded by post Jurassic cover rocks.

Scout exploration drilling by PGSA has been conducted predominantly within an approximate 6 km radius from the Cap Oeste Project area at seven prospect areas namely Puma, Felix, Pampa, Cose, Vetas Norte, Don Pancho and Breccia Valentina (Figure 7-5).

Precious metal mineralization intersected at these prospects show a range of structurally low controlled sulphidation Au-Ag mineralization styles including silica poor + iron oxide - sulfide replacement, silica-sulfide rich hydrothermal breccias and chalcedonic veining all of which host strong geochemical correlation with As, Sb, Hg +- Mo. The style of mineralization intersected at the Pampa and Cose prospect areas, which are located along the interpreted strike continuation of the Bonanza Fault approximately 2 km to the NW and 1.5km SE of the Cap Oeste Project respectively, both share strong similarities in terms of geochemistry and structural control with that of mineralization at the Cap Oeste Project.

The Cose Prospect Drilling to date has defined a high grade shoot, approximately 130 meters long and 12-15 meters wide, situated in the interpreted southeast extension of the Bonanza Fault. The high grade ore shoot pitches steeply to the west north west 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. The 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.

The Don Pancho prospect is centered on at least two, west-north west trending (300°), subvertical to steep easterly dipping faults where both illite-silica-marcasite-pyrite goldsilver and silver-only, proustite-dominated mineralization are observed. Scout drilling programs completed to date have outlined this mineralization over widths between 5 to 20 m, along a strike length of approximately 150 m and down to a vertical depth of 80 m. The limits of this mineralization have not been defined and the mineralization remains open down dip and along strike to the south east. To the northwest of the Don Pancho prospect, geologic mapping suggests that the Don Pancho structural corridor becomes more west-northwest trending and eventually intersects with the Cap Oeste Structural Corridor.

At the Puma, Felix and Vetas Norte prospect areas anomalous to high grade precious metal mineralization intersected to date is characteristically hosted in low to moderate (40 º) NNE to E dipping, structurally controlled zones currently interpreted as detachment faults, varying in width from between 1-25m. These zones are characterized by banded, opaline to chalcedonic silica veins and hydrothermal breccias that are hosted within wide zones of iron oxide-pyrite-stibnite-arsenopyrite-bearing silica-illite-smectite alteration which have been mapped in outcrop and as float over strike lengths of more than 300 m.

Precious metal mineralization at the Breccia Valentina prospect is best developed within a 150 by 300-meter area throughout which bedrock comprises altered ash and lapilli tuff, which are interpreted to enclose a brecciated subvolcanic rhyolite dome. Alteration is accompanied by extensive silicification and disseminated marcasite and arsenopyrite. The most significant Au and Ag values to date report to a series of steep east/northeast dipping, 5 to 40-m wide zones hosting hydrothermal breccia, sheeted to irregular crystalline / comb quartz / chalcedony veinlets and milled matrix breccia. Proustite (ruby silver) occurs as disseminations in the tuff matrix, in breccia matrix, in drusy veins, and rarely with kaolinite in vein selvages. Trace realgar and orpiment have also been observed. These mineralogical features at Breccia Valentina resemble those at Cap Oeste and are taken as indicative of a similar origin of the mineralization (R. Sillitoe, 2008).

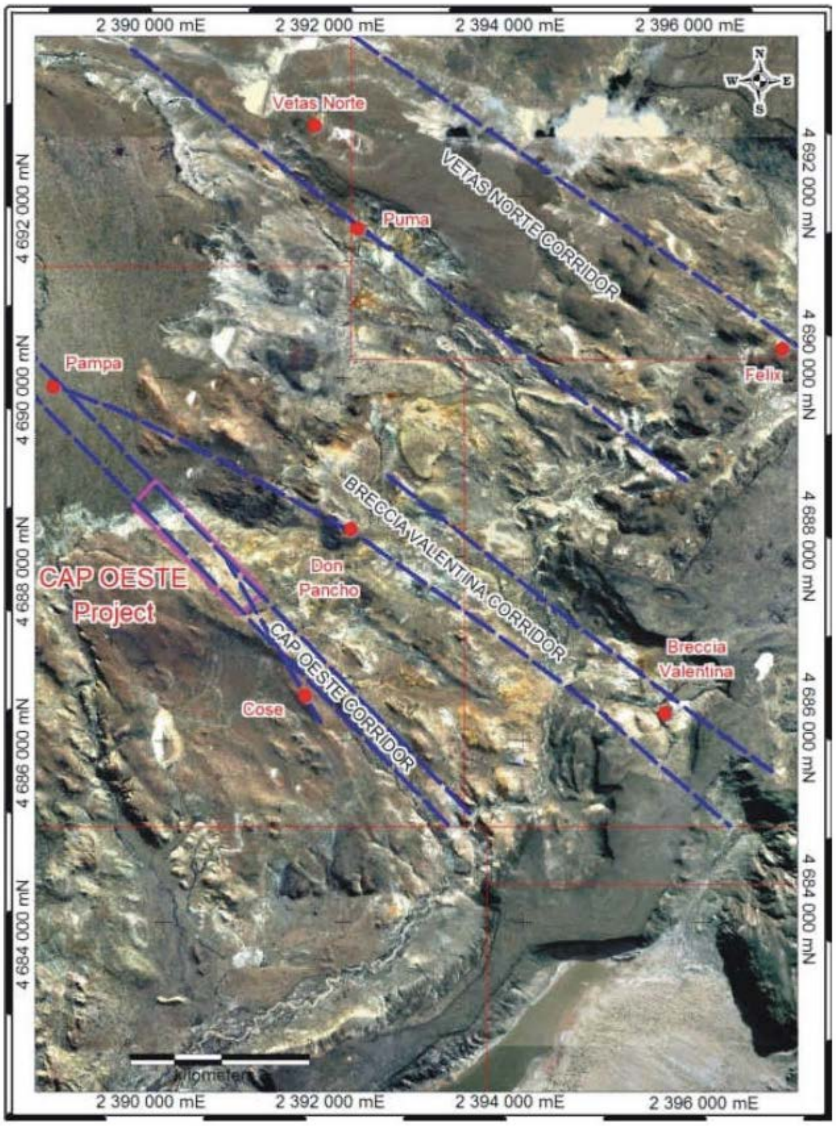


Figure 7-5
El Tranquilo Block Prospect Locations

7.3.2 Property Mineralization

<u>Description and Distribution</u>

As described in Section 7.2, Au-Ag mineralization at Cap Oeste is predominantly hosted by the northwest- trending Bonanza Fault, which dips 40 to 80º to the southwest. Drilling has therefore been orientated towards the northeast (50º true north) along grid lines orthogonal to a baseline trending 140º. The fault juxtaposes crystal-poor ignimbrite to the west with dominantly crystal-rich ignimbrite to the east, reflecting "west side down" normal displacement, and is interpreted to be one of the bounding structures to a Late Jurassic graben or half-graben, as previously discussed.

Transecting the mineralized zone in section, from the hangingwall to the foot wall of the Bonanza Fault, three successive types of mineralization have been generally defined (Figure 7-6).

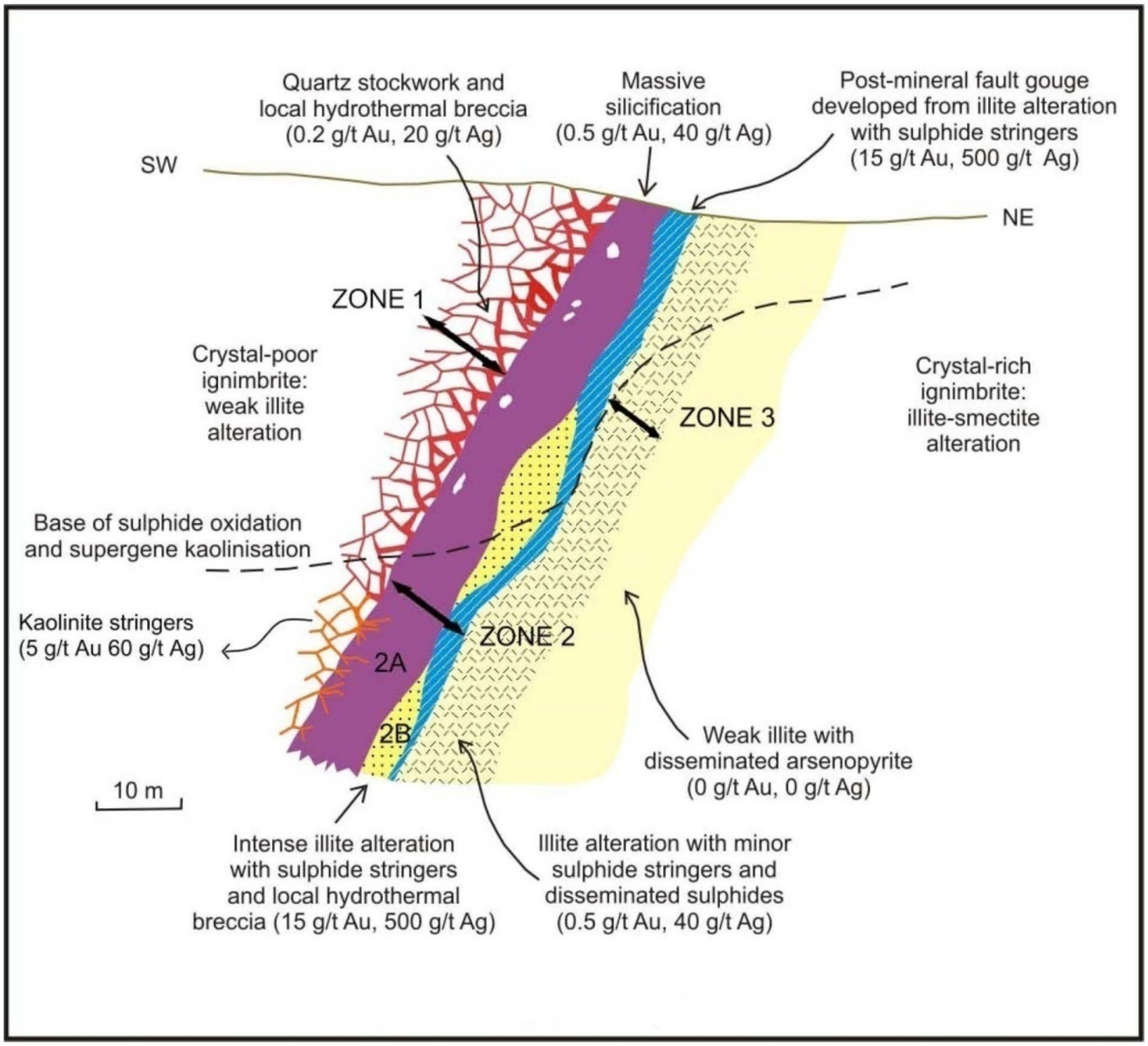


Figure 7-6
Mineralized Section Schematic (after Sillitoe 2008)

Distinguishing features of these zones include the following:

Zone 1 (Hangingwall Crackle Breccias and Veinlets)

In the upper levels peripheral to the Bonanza Fault (within 80 m from surface) this zone is developed over 10-to 40-m true width, preferentially within hangingwall vitric tuff. Irregular, multi-directional, quartz (chalcedony) stockwork veinlets and matrix-supported breccia occur with associated pyrite, goethite, and hematite which report persistently anomalous precious metals averaging of the order of 0.5 g/t Au and 20 g/t Ag.

Where unoxidized, this zone contains generally low concentrations of disseminated sulfides (<1.0%) and low-order precious-metal values, typically in the order of 0.5 g/t Au and 40 g/t Ag.

This zone tends to narrow and become less well developed with depth, possibly as a result of increasing lithostatic pressure and its effect on ascending hydrothermal fluids and commonly passes a transition to predominantly disseminated mineralization where it intersects the less competent rhyolitic lapilli and block tuff proximal to 300 m RL.

Below a depth of approximately 250 m from surface (approximately 275 m RL) in the hanging wall adjacent to the Bonanza Fault, this zone is manifested as a 10-to 20-m wide zone comprising high grade, drusy quartz lined, kaolinite filled stringers developed predominantly in the lower, generally unoxidized, hangingwall vitric quartz eye tuff and underlying crystal tuff unit. This mineralization typically hosts low concentrations of disseminated sulfides (<1.0%) and moderate to high precious-metal values, typically in the order of 5 g/t Au and 60 g/t Ag.

Zone 2a (Fault Zone Silicified Breccia)

Relict hydrothermal breccia textures are preserved within the Bonanza Fault and generally extend 1 to 5m into the foot wall crystal tuff. Zone 2a breccias are commonly overprinted by pervasive chalcedony+hematite (where oxidized) and disseminated marcasite-pyrite (where unoxidized), thought to be introduced during cyclic re-brecciation and healing events. This zone hosts gold and silver values typically in the range of 0.5 to 1 g/t Au and 40 g/t Ag, respectively.

Zone 2b (Fault Zone Sulfide Stringer/Sulfide Matrix Breccia)

Distinctive breccias consisting of silicified clasts cemented by a marcasite-pyrite-illite chalcedonic silica-rich matrix are interpreted to have been converted by post-mineral faulting to fine-grained gouge containing clay-altered (illite) and silicified clasts. This gouge is commonly black in color due to the presence of crushed sulfide minerals as shown in Figure 7-7. This zone varies in width between 5 and 15 m, and occurs most commonly along the contact between the contrasting lithologies that are juxtaposed across the Bonanza Fault. The zone has also been identified along sub-parallel fault splays which locally cut the hangingwall stratigraphy and within underlying footwall rocks (Figure 7-6). This zone hosts gold and silver values typically in the range of 15 g/t Au and 500 g/t Ag, respectively.

Figure 7-7
DDH Core from hole CO-054-D: Example of mineralized breccia in Zone 2b below silicified hematite rich oxidized fault contact

Zone 3 (Footwall Stringer/Disseminated Zone)

Sheeted to stockwork textured, marcasite-pyrite veinlets and disseminations containing up to 5% fine sulfide occur peripheral and sub-parallel to Zones 2a and 2b, across a true thickness of 10 to 40 m. Precious metal values diminish progressively into the footwall, together with an increase in the smectite:illite clay alteration ratio and a decrease in abundance of arsenopyrite needles. Zone 3 lacks the presence of hydrothermal quartz veinlets or stringers and occasionally hosts rare calcite-filled stringers. This zone hosts gold and silver values typically in the range of 0.5-1.0 g/t Au and 40 g/t Ag, respectively.

Drilling completed to date in the immediate Cap Oeste project area and along the strike extension of the Bonanza Fault has defined gold-silver mineralization and/or anomalous indicator geochemical signatures over a strike length exceeding 3.5 km that are broadly coincident with the Bonanza Fault.

To date, the majority of the step out drilling at depth along this zone has been focused on delineating mineralization which appears to comprise a series of broadly defined shoots over a strike length of approximately 1,025 m between sections 9725 N and 10800 N, as shown in the longitudinal projections for gold, silver and gold-equivalent grade-thickness products (Figures 7-8, 7-9 and 7-10, respectively).

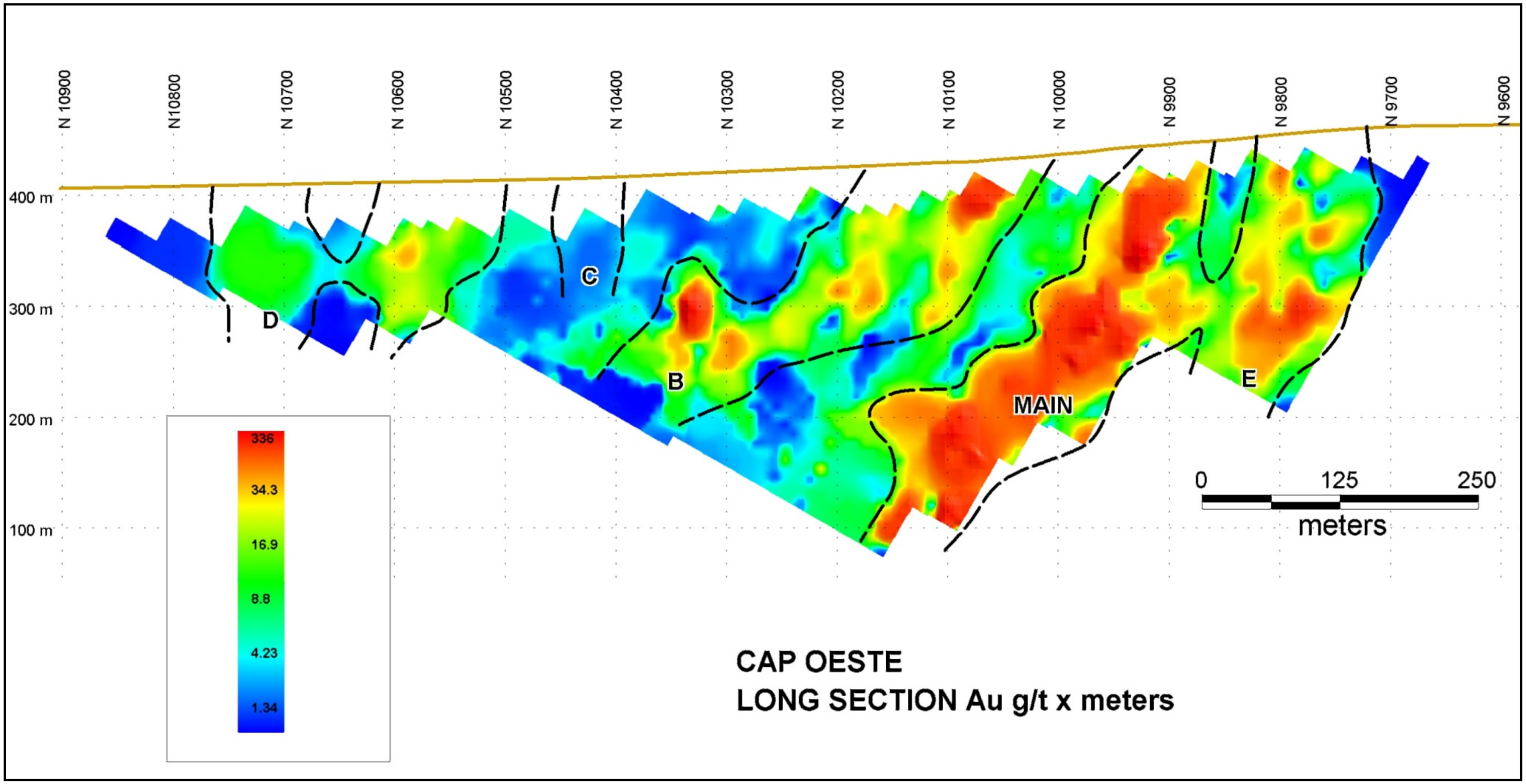


Figure 7-8
Cap Oeste Deposit -Longitudinal Projection
of the Au Grade-Thickness (Au g/t x meters)

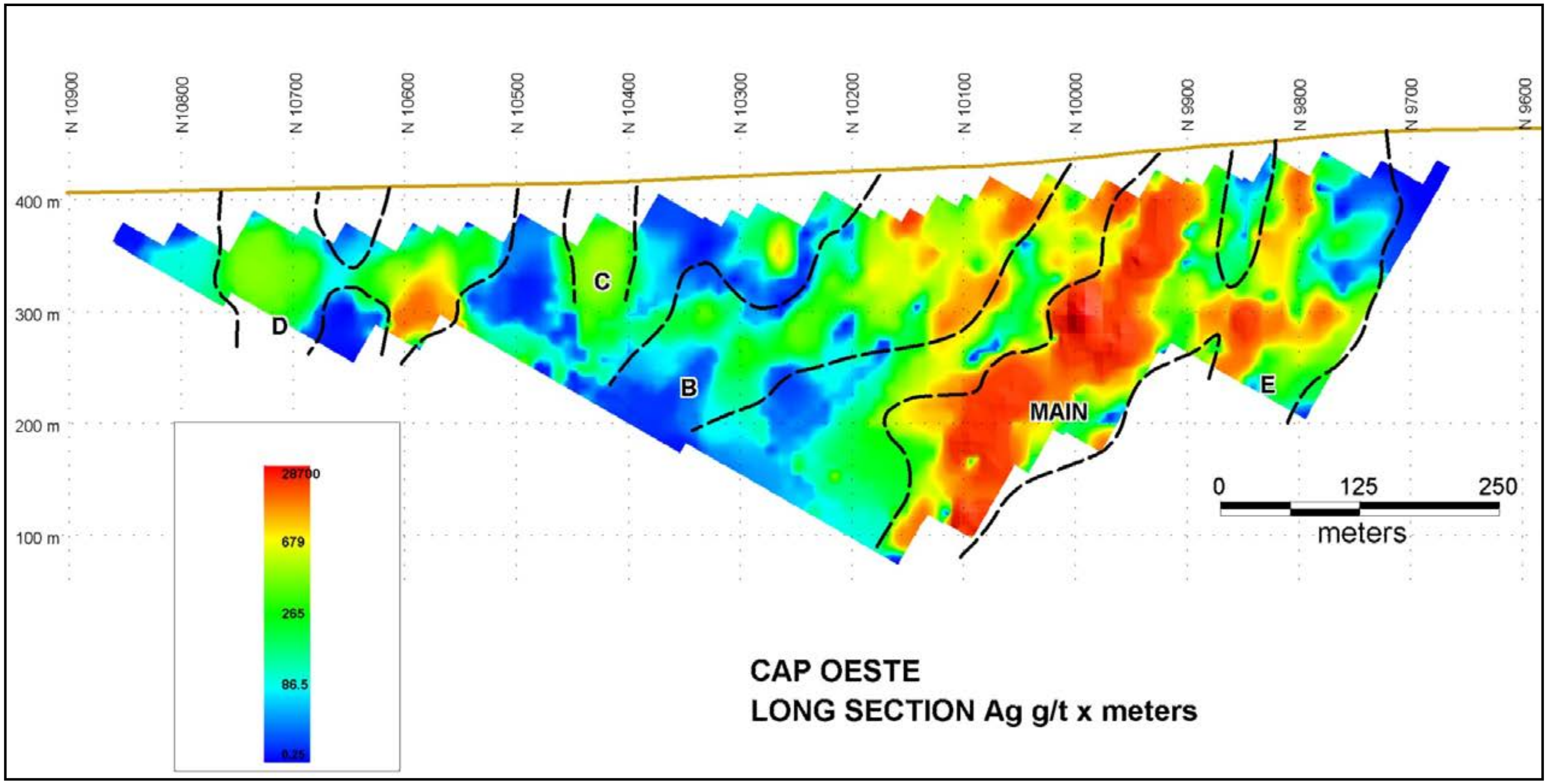


Figure 7-9
Cap Oeste Deposit -Longitudinal Projection
of the Ag Grade-Thickness (Ag g/t x meters)

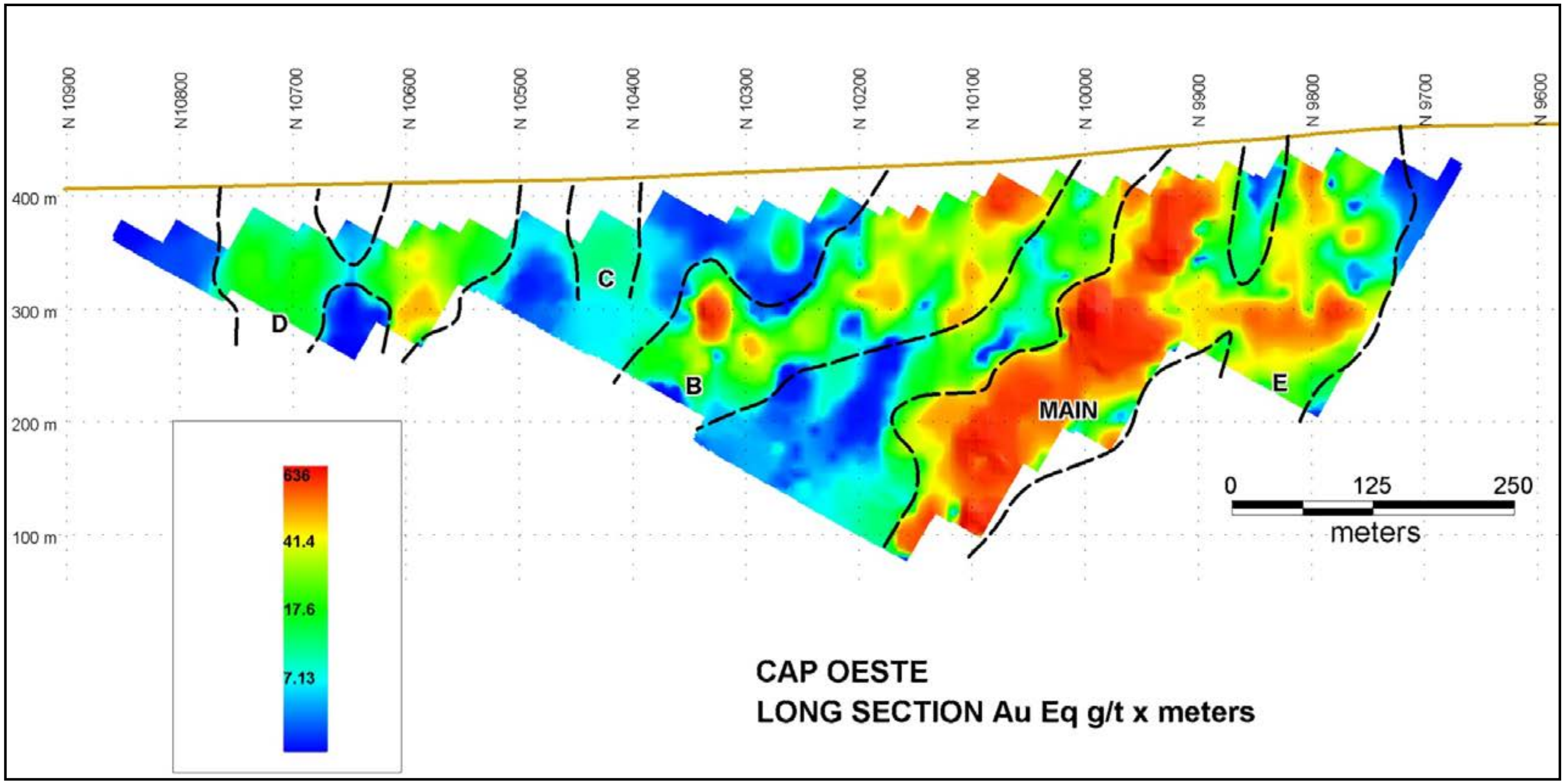


Figure 7-10
Cap Oeste Deposit -Longitudinal Projection of the Au equivalent Grade-Thickness (Au eq g/t x meters)

These longitudinal projections were generated from the mineralized intersections from drill holes that are predominantly localized in breccia style mineralization along the Bonanza Fault as tabulated in Section 10 (refer to Table 10-1). In creating these mineralized intersections, potentially economic grade mineralization was identified by application of a minimum cutoff grade of 0.5 g/t Au or 35 g/t Ag. The core lengths were used in the calculation of the grade-thickness products.

For areas peripheral to the main shoots that contain low to anomalous level gold-silver values, drilled intervals which indicate continuity of mineralization along the Bonanza Fault were selected to maintain continuity on the longitudinal projections.

The longitudinal projection for silver displays a relatively more cohesive medium to high grade silver zone (defined by silver grade x thickness greater than 350 gram-metres) defined by the partial union of the Main shoot and Shoot B, compared to the gold long section. Local differences between respective high gold and high silver zones are considered to be due to the presence of differing mineral assemblages, particularly evident by the gold-rich, silver poor lower portion of Shoot B and high silver-low gold signature of Shoot C and Shoot D. The longitudinal projection for the gold-equivalent gram-metre data essentially confirms the continuity of the individual shoot geometry.

It is currently interpreted that mineralized shoot localization is controlled by the intersection of the Bonanza Fault with the crosscutting fracture corridor described above and subtle strike changes along the former. Additionally, where this structural combination transects more competent lithologies it is believed that there is a tendency for enhanced gold and silver values throughout, or immediately peripheral to, the lower respective contact. Throughout the Cap Oeste project area this appears to have created a broadly repetitive geometrical pattern of a series of at least five shoots developed along the plane of the Bonanza Fault described as follows:

Main Shoot

The 'Main Shoot' is the most significant high grade mineralized lens defined to date and can clearly be distinguished on the longitudinal projections of the gold grade-thickness contours.

This shoot is principally defined by a gold grade-thickness composite value of greater than 20 gram-metres, and is interpreted to extend from surface down plunge at a pitch of approximately 50° along the plane of the Bonanza Fault for a distance of approximately 450 m between sections 9850 N and 10150 N. The average height (as measured in a direction perpendicular to the plunge of the shoot) is approximately 70 m and width of the shoot is approximately 10 m.

As defined by the current level of drill information as of June, 2011, mineralization at the currently defined lower limit of the Main Shoot approximately 350 m below surface (<90 m RL) remains open, albeit with reduced dimensions compared to the up plunge extension.

Shoot B

Shoot B is centered at surface approximately 160 m to the northwest of Main Shoot and from trenching and drilling results to date has been broadly indicated to extend over an approximate down plunge length of 350 m at a plunge of approximately 35o to the northwest along the plane of the Bonanza Fault. The average height and width of the shoot is approximately 35 m and 10 m, respectively within which this shoot shows more variability than the Main shoot in terms of grade and width.

Based on the longitudinal projections and the gold grade contours shown in plan view projection (Figure 7-11), the individual Main and B shoots have the following geometries along the plane of the Bonanza Fault:

- Main Shoot: (between section 9850 N and10150 N). Plunge 35-55° to 280°.
- Shoot B: (between 10000 N and 10375 N). Plunge 30° to 300°.

Shoot C

This shoot is defined by a single drill pierce point on the longitudinal projections (C0-073-D) comprising a high grade silver interval of 2.30 m at 0.66 g/t Au, 187.26 g/t Ag. Potential for the enhanced down plunge continuity of this mineralization remains untested by drilling.

Shoot D

Shoot D is centered approximately 600 m to the northwest of the central outcropping portion of the Main Shoot, between Sections 10550 N and 10775 N. This shoot, as has been defined to date by relatively sparse drilling, is seen to comprise a series of poorly defined, disjointed zones of mineralization. The approximate height of the shoot is 35 m and the approximate width of the shoot is 10 m. The indicated northwest trending plunge length of this shoot is approximately 100 m. Based on the limited level of drilling, the geometry of this shoot is not well-defined. However, it is currently interpreted as holding potential to project to depth maintaining a hypothetical rake angle similar to the other Main Shoot and Shoot B of approximately 35°.

Shoot E

Shoot E is centered approximately 125m to the southeast of the Main Shoot, at the site of a subtle change in strike of the Bonanza Fault, and has been defined by drilling to date to pitch overall more steeply down the plane of the Bonanza Fault as compared to the Main and B Shoots. Drilling to date has defined this shoot to a depth below surface of approximately 250m and with the combination of high Au and Ag values at approximately 300m RL (150m below surface) mineralization appears to be more strongly developed and the shoot appears to exhibit continuity with the Main Shoot.

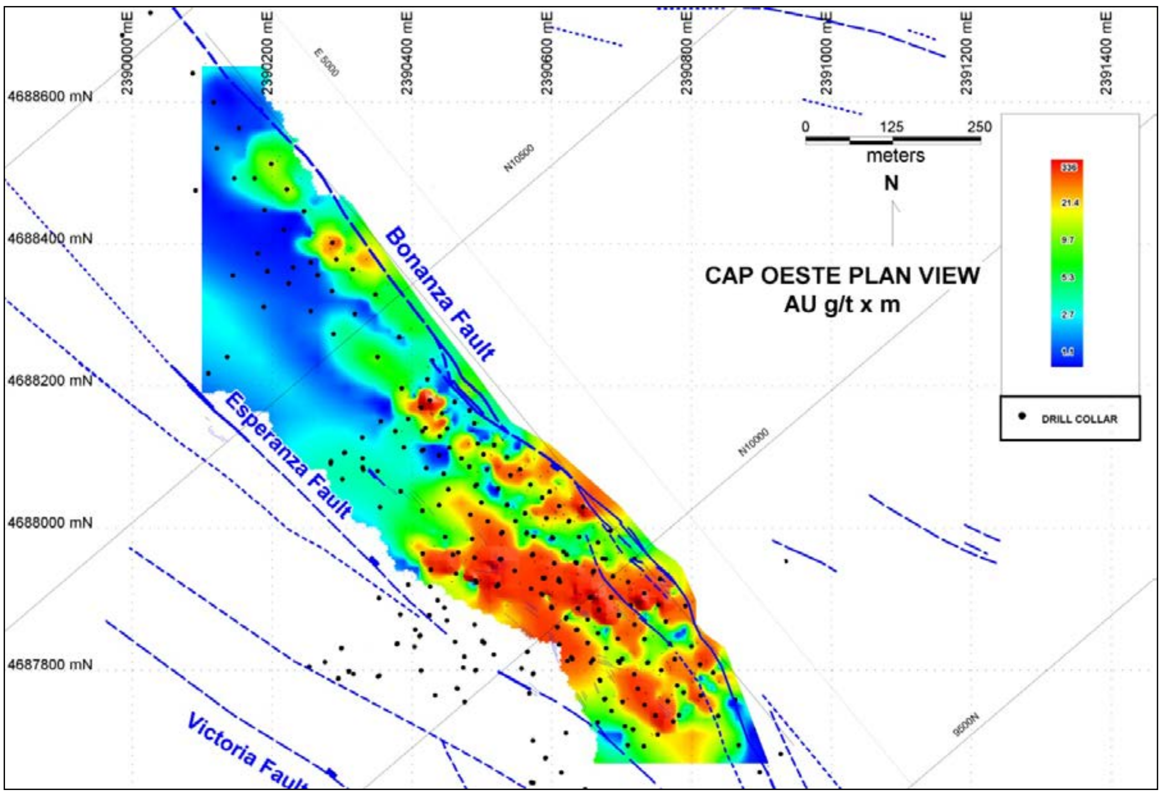


Figure 7-11
Contoured Plan for Gold Grade-Thickness Product
Showing Au Distribution and Geometry for the Cap Oeste Main

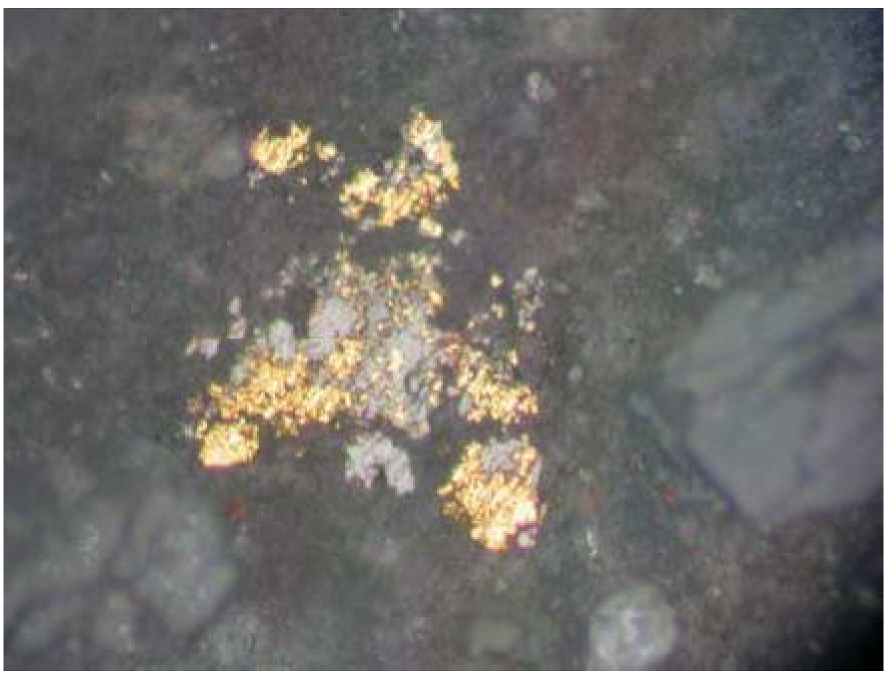
Mineralogy and Paragenesis

Based on observations from core in hand specimen, thin and polished section petrographic samples (total of 28 samples) and studies by computed axial tomography (CAT scan; 2 samples), the respective mineralogical characteristics of oxide and sulfide assemblages have been determined and are discussed below.

Oxide Mineralogy

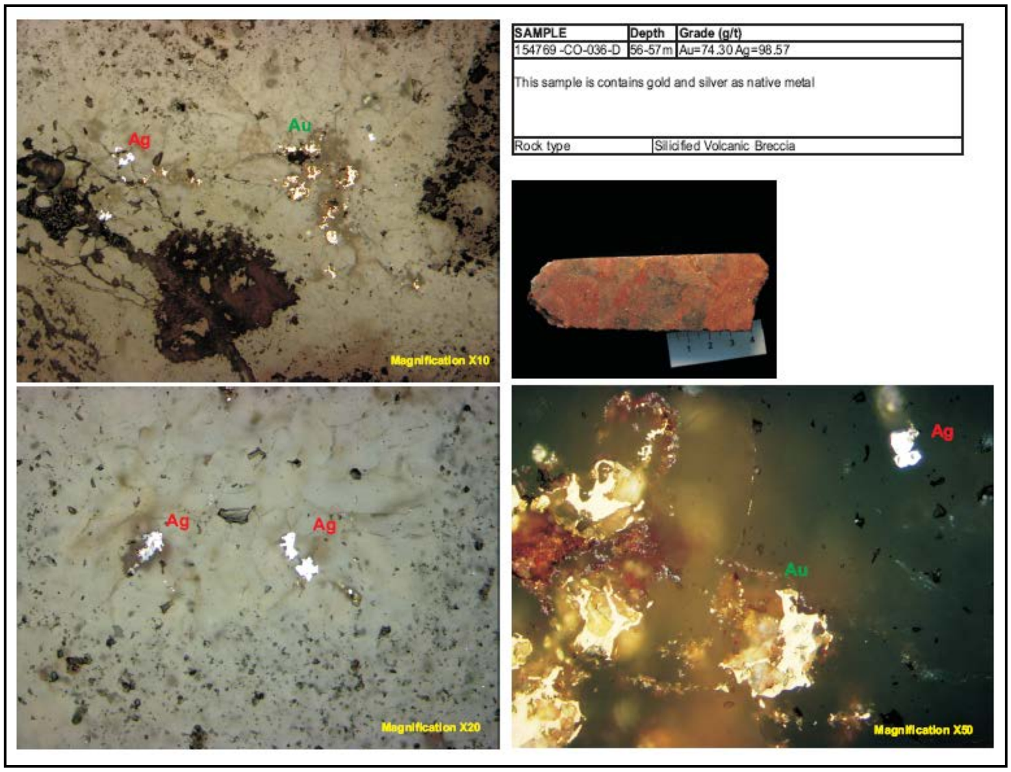
Partial to complete supergene oxidation of high-grade Au-Ag mineralization (Zones 2a and 2b) has occurred to an average depth interval of 70 to 120 meters, with the consequent destruction of all sulfide minerals and the development of abundant hematite, jarosite, limonite, and kaolinite. The oxide/ sulfide boundary is transitional, and generally mirrors the southwest dipping trace of the fault, with oxidation consistently reaching greater depths on the hangingwall side of the Bonanza Fault. This has been interpreted as due to the lower rock permeability (i.e. mores resistant to the circulation of oxidizing fluids) caused by the preferential development of illite and smectite clay in footwall rocks (Sillitoe, 2008).

Within the zone of oxidation, gold occurs in the native state; discrete grains of gold (up to approximately 30 µm across) were observed and interpreted to be of both relict hypogene and supergene occurrence (Photographer N 7.1). The fineness of the gold may have been increased due to preferential silver removal during oxidation of hypogene electrum. Native silver has also been defined by both petrology and CAT scan, some of it potentially inherited from hypogene assemblages.



Photograph 7-1
CO_054-DR (132-133.1m; 7.86 ppm Au, 87.2 ppm Ag). Composite aggregate of gold-electrum with argentite-acanthite (pale grey) (Ag_2S) enclosed in quartz and illite-sericite (dark grey), with slight development of supergene Fe oxides (red-brown hue). Plane polarized reflected light, field of view 0.2 mm across (after Ashley 2008).

Similarly, in the oxidized sample examined by CAT, gold and silver were observed to occur in the native state and also as electrum, as shown in Photograph 7-2.



Photograph 7-2
(CO-036-D; 56-57m) – Computed axial tomography (CAT) scan image
LHS image represents a rendered 2D image of the distribution of native Au/Ag at the flat surface face of the core sample, the RHS image represents the 'see through' 3D projection showing the pattern produced by the Au/Ag distributed throughout the whole volume of the sample

Apart from minor amounts of scorodite ($FeAsO_4 \cdot 2H_2O$.), no other supergene minerals have been identified to date and it is assumed that strongly anomalous values of Au, Ag, As and Sb may also be hosted in supergene Fe oxides. In addition, Ag is suspected to be present as one or more halides including chlorargyrite (AgCl), embolite (Ag(Br,Cl)), bromargyrite (AgBr) and iodargyrite (AgI) given the semi-arid climatic conditions and consequent elevated chloride, bromide and iodide contents of local ground water (Sillitoe, 2008).

As a product of post-drilling superficial oxidation of molybdenite, numerous high grade, sulfide Au-Ag drill intervals hosting original elevated concentrations of molybdenite (MoS_2) reflect high visually prominent (i.e. blue staining) concentrations of ilsemannite ($Mo_3O_8 \cdot n(H_2O)$).

Sulfide Mineralogy

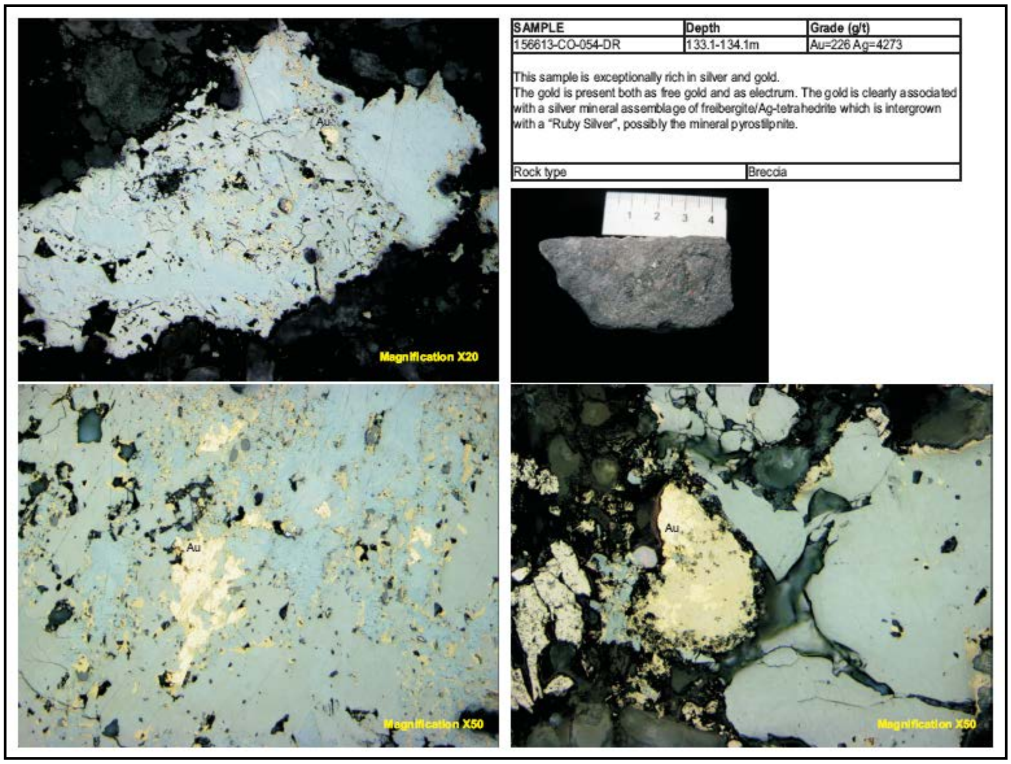
Based on hand lens observations, the main sulfide minerals in the Cap Oeste Project zone are pyrite and marcasite; pyrite typically occurs as small (less than 0.5 millimeters) isolated crystals and marcasite as fine (less than 0.2 millimeters) disseminations. Sulfides occur in different sites including altered phenocrysts (e.g. former ferromagnesian and feldspar pseudomorphs), altered groundmass of volcanic fragments, in strongly silicified fragments in breccias, as components of hydrothermal breccia matrices and veins, and within fault gouge.

In hand specimen the pyrite appears to be early in the paragenesis of precious metal mineralization and typically occurs as sub-radiating and bladed aggregates, whereas the marcasite appears to be more closely related to the gold and silver mineralization. Arsenopyrite (FeAsS) is widespread as an accessory to the iron sulfides particularly in the footwall stringer Zone 3 where it is paragenetically later than pyrite-marcasite.

Precious metals within hypogene mineralization occur dominantly in finely disseminated proustite (Ag_3AsS_3), argentite/ acanthite (Ag_2S), sternbergite ($AgFe_2S_3$), lautite (CuAsS), and gold/electrum, either singly or in aggregates. Gold values in excess of 10 grams per tonne occur with concentrations of acicular arsenopyrite without appreciable Ag and Sb, most commonly on the immediate footwall side of the high-grade zone. In several of the petrology samples hosting high grade hypogene mineralization no discrete Au-bearing phases were recognised, suggesting that a proportion of the gold might be held in arsenopyrite ± pyrite.

From the CAT study of hypogene samples, Au and Ag were found to occur both in the native state and as electrum - both of which show a strong association with freibergite $(Ag, Cu, Fe)_{12}(Sb, As)_4S_{13}$, argentiferous tetrahedrite $(Cu,Ag)_{10}(Fe,Zn)_2Sb_4S_{13}$ and possibly pyrostilpnite (Ag_3SbS_3), as shown in Photograph 9-3).

Hand specimen observations suggest that tennantite ($(Cu,Fe)_{12}As_4S_{13}$) argentiferous tetrahedrite ($(Cu,Ag)_{10}(Fe,Zn)_2Sb_4S_{13}$) mineralization identified on the basis of its characteristic chestnut-colored streak is broadly confined to Zone 2b, and gives rise to close correlations between gold, silver, copper, antimony, arsenic and mercury values. There are also minor occurrences of high-grade silver-gold mineralization that lack any correlation with elevated copper, arsenic and antimony values, perhaps due to the presence of acanthite (Ag_2S), electrum and/or native silver, all of which have been identified locally in drill core.



Photograph 9-3
Hole CO-054-DR: 133.1- 134.1m; 226 ppm Au, 4273 ppm Ag Computed axial tomography CAT scan image – showing Au associated with Freibergite (Ag, Cu, Fe)$_{12}$(Sb, As)$_4$S$_{13}$ and Ag tetrahedrite (Cu,Ag)$_{10}$(Fe,Zn)$_2$Sb$_4$S$_{13}$ and possibly pyrostilpnite (Ag_3SbS_3). LHS image represents a rendered 2D image of the distribution of native Au/Ag at the flat surface face of the core sample, the RHS image represents the 'see through' 3D projection showing the pattern produced by the Au/Ag distributed throughout the whole volume of the sample.

In Zone 2b, one or more ruby silver minerals, probably proustite (Ag_3AsS_3), and/or pyrargyrite (Ag3SbS3), occur as monomineralic veinlets or, where gouge development has taken place, as clastic grains. There is a strong suggestion that these silver sulphosalts were deposited late with respect to the rest of the gold-silver mineralization at the Cap Oeste Project which were in turn followed by deposition of trace amounts of realgar (As4 S4) and orpiment (As2S3). The geochemical association of silver with other metals as described suggests that supergene silver enrichment is not an important contributor to bonanza-grade values, and that appreciable silver introduction as supergene acanthite is unlikely (Sillitoe, 2008). Particle size for individual Ag-rich minerals ranges up to 0.5 to 1.0 millimeters, with local aggregates up to a few millimetres.

Controls on Mineralization

Ore Fluid Genesis

The ore fluid responsible for mineralization at Cap Oeste is postulated to have been focused within dilatant sites along the Bonanza Fault, with its expulsion potentially linked directly to fault-displacement events. The source of the fluid may have been felsic magma similar to that which formed rhyolitic domes a few kilometres distant at Breccia Valentina (Sillitoe, 2008).

Petrological examination by Ashley (2010) reported evidence of at least two alteration events the first characterized by potassic alteration of comprising replacement of the groundmass, sanidine-plagioclase feldspar and biotite by fine grained K-feldspar and quartz, and a subsequent retrograde argillic illite- -sericite+-kaolinite overprint that was locally accompanied by silicification.

The argillic and local silicic alteration of the volcanic host rocks appears in many samples to be related temporally to the formation of hydrothermal breccias and veining which is interpreted to have occurred in at least two episodes.

The first is dominated by fine to medium grained quartz characterized by local late stages of deposition of sulfides, mostly pyrite and arsenopyrite, clay, and rare carbonate. Carbonate in bladed/prismatic form could have been formerly more common in breccia infillings, but has been pseudomorphed by quartz, or possibly by bladed aggregates of fine grained pyrite ± marcasite, overgrown by arsenopyrite. Although high-grade mineralization is relatively quartz-poor, adjacent, intensely silicified rocks of Zone 1 are considered as integral parts of the mineralizing event (Sillitoe, 2008). It is 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 Bonanza Fault. The decrease in stockwork development with depth hence reflects increasing lithostatic pressure. However, fluid that accessed the immediate foot wall of the fault appears to have not undergone the same degree of cooling; hence, the complete absence of both silicification and quartz veining. A lack of open space during the faulting is considered the most likely explanation for the absence of the banded quartz typical of low-sulphidation deposits.

The second phase of breccia infill tends to contain abundant sulfides and illite-sericite, with little or no quartz which hosts sulfides including paragenetically early arsenopyrite and pyrite, with later-deposited Ag minerals, base metal sulfides, along with gold-electrum. Sillitoe (2008) postulates that the ore-bearing fluids were focused along the footwall side of silicified zone originating from the 1st episode, resulting in intense illite-sericite alteration.

Following alteration and mineralization, fault displacement is suggested by Sillitoe (2009) to have continued and been localized by the rheologically weakest part of the fault zone: the intense illite-sericite alteration along the immediate footwall of the massive silicification, which given that this zone was also the site of high-grade mineralization, much of the potential ore occurs in fault gouge.

However, petrological examination by Ashley (2010) concludes that little or no textural evidence exists that suggests there was any significant deformation (e.g. shearing) occurring during, or after the formation of hydrothermal breccias and association veining. Contents of most breccias, particularly those with quartz-rich infill, remain unstrained and lack fracturing. It could be implied that in some of the breccias in which illite-sericite is the dominant matrix infill, that there is a weak anastomosing foliation, but enclosed delicate sulfide aggregates (including “hard” arsenopyrite and pyrite, and “soft” precious and base metal phases) do not show any cataclastic or foliation effects.

Stratigraphic Control

Based on detailed stratigraphic logging of the volcanic litholgies and their spatial relationship with mineralization it is interpreted that host-rock lithology does not act as a fundamental control on the localization of the main mineralized shoots. However the more competent volcanic stratigraphic units e.g. the moderately welded Rhyolitic Vitric quartz eye ash tuff (VfTvxq) –appear to have influenced the formation of some of the highest grade and widest portions of the main shoots where they were transected by the structural zones.

Structural Control

The interaction of the respective orientations of the Bonanza Fault with that of the cross cutting fracture corridor is interpreted to have potentially created the enhanced dilationary setting within which the enhanced development of the mineralized shoot was created.

7.4 Structure

7.4.1 Bonanza Fault

The main Au-Ag mineralization defined to date at the Cap Oeste project area gold-silver mineralization occupies an approximate 800-m strike length within and immediately adjacent to the curvilinear, northwest (310-320º) trending Bonanza Fault.

The fault dips moderately to steeply (50-70 º) to the southwest and has been defined by mapping and drilling over a strike extent exceeding 5 km.

The relative displacements of individual stratigraphic units across the Bonanza Fault suggest a normal displacement of the southwestern (i.e. hanging wall) block of at least 180 m down the plane of the fault (i.e. throw of 150 m and heave of 70 m). On a macroscopic scale, indicators of normal displacement include the interpreted fault drag deformation of the hangingwall units adjacent to the Bonanza Fault.

7.4.2 Cross Cutting Fracture Corridor

Outcrop mapping between Sections 9925-9975N in the immediate hangingwall to the Bonanza Fault has defined two prominent main fracture trends characterized by 10-50cm spaced, narrow (0.5cm-2cm wide), planar, limonitic silica veinlets and hydrothermally brecciation (Figure 7-12). One of the fracture sets occurs parallel to the Bonanza Fault and is interpreted to have formed in response to movement along the latter. The other is interpreted to correspond to a cross cutting fracture set dipping 80-90º towards 185-195° (Figure 7-13). The intersection of these fracture trends is interpreted to have spatially controlled the development of the dominantly WNW plunging mineralized shoots along the plane of the Bonanza Fault explained above.

The gold mineralization occurs in a pipe-like ore shoot that plunges to the NW and is associated with a strong chargeability anomaly. From plans and sections drawn up from the extensive drilling of the deposit there is no apparent change in dip or strike of the Bonanza structure to explain the control of the ore body. This either means that the mineralization occurs at the intersection of the Bonanza Fault with a second structure and/or that the mineralization has been truncated by post-mineralization movement along the reactivated Bonanza structure.

The presence of fine WNW trending (100-120°N) fractures and minor veining in the hangingwall of the Cap Oeste ore body indicate that the Patagonia Gold model of ore body control at the intersection of the Bonanza fault zone and a WNW-trending structural corridor is the best explanation and a predominant NNE dip of the WNW structures explains why the ore body plunges to the NW (Starling 2011).

Figure 7-12
Outcrop Photo of the Cross Cutting Fracture Corridor (Looking Northwest)

Figure 7-13
WNW trending tensional sulfide bearing veining
splaying off a sinistral NW Fault (after Starling 2011)

7.4.3 Esperanza Fault

Mapping and drilling peripheral to the main mineralized zone has defined a second, subparallel structure located some 220 m to the southwest at surface, referred to as the Esperanza Fault. This fault dips steeply (75-85°) to the northeast and is characterized by a semi-planar, 2 to 20 m wide zone of faulting and fracturing that includes narrow 1 to 2 m wide zones of hydrothermal crackle-and matrix-supported brecciation. This structure has been mapped and intersected by drilling over a strike distance of approximately 1,500 m and has returned high grade gold results at depths below 250 to 300 m below surface (100 m to 150 m RL).

The displacement of the lithologies on either side of the Esperanza fault appear to have undergone approximately 80m of down dip displacement in which the eastern hangingwall was downthrown to the east.

7.4.4 Interpreted Structural Setting

The inclination of the Esperanza fault with respect to that of the Bonanza Fault, the repetition of the stratigraphy in the foot wall and hanging wall blocks of the former and the differential amount of respective indicated displacement along the Esperanza and Bonanza Faults (i.e. 80m versus 180m) suggests that this structural pair bound a northwest trending half graben, approximately 220 meters wide at surface (Figure 7-3 and 7-4).

As part of this hypothetical fault array, the Esperanza Fault could comprise the subsidiary, antithetic structure to the main Bonanza fault for which movement along the latter potentially preceded movement along the Esperanza Fault. This would explain how the volcanic stratigraphy in the hangingwall to both the Esperanza Fault and the Bonanza Fault remained subhorizontal (in SE-NW section) during differential movement along the respective faults.

Both from the intersection of the respective Bonanza and Esperanza fault planes in drilling, and the extrapolation of the respective fault planes below the level for which drill information exists, it is interpreted that the graben floor (or the line defined by the intersection of the two fault planes) would comprise a lineation that plunges approximately 10 º towards azimuth 320º (i.e. a generally flat lineation oriented along the strike of the structures). This intersection has been interpreted from drilling to date between Section 9950 N and Section 10150 N to occur at approximately depths of 80 m RL and approximately 125 m RL (approximately 300 m below surface), respectively.

No kinematic indicators (e.g. slickensides) have been observed to indicate the direction of movement along either the Esperanza or Bonanza Faults, although observations on a more regional scale suggest a component of oblique movement is possible.

In addition, correlation of individual units within the hangingwall portion of the volcanic stratigraphy (i.e. that to both the Esperanza and Bonanza Faults) between sections 9950- 10150N indicate a consistent shallow (13º) dip of the hangingwall package to the NW along the axis of the proposed half graben and on individual sections, shallow tilting to the NE (approximately 5º).

Based on these orientations it is interpreted that the gentle inclination of the Bonanza Fault hangingwall block along the graben axis is possibly due to a 'scissor' or 'hinge' style normal faulting which pivoted down to the northwest from a point to the SE along the axis of the half graben, and a minor component of NE side down rotation in the hangingwall block relative to the Bonanza Fault.

Given the different geometrical intersection array between the cross cutting fracture corridor and the Esperanza Fault compared to that of the former and the Bonanza Fault it is considered probable that the resultant plunge of any significant, mineralized shoot along the Esperanza Fault will be steeply plunging to the east southeast (i.e. 80º to azimuth 105º).

An illustration of the interpreted geometries of the various structures discussed above is presented in Figure 7-14.

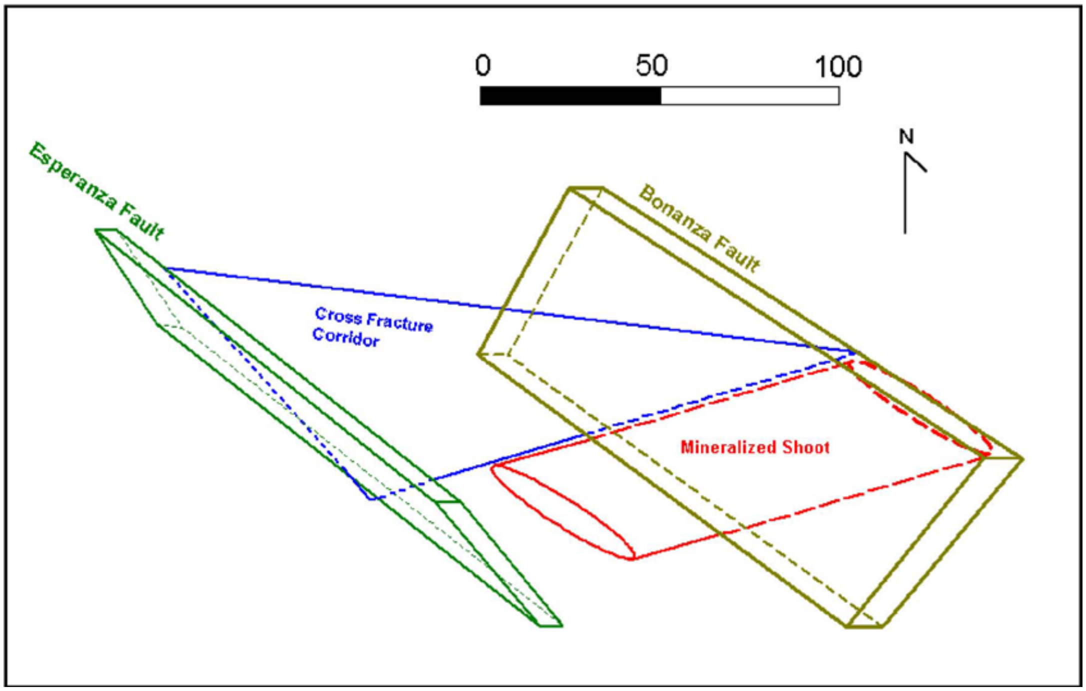


Figure 7-14
Schematic of Interpreted Fault Geometry and
Control of Mineralization at the Cap Oeste Deposit

8.0 DEPOSIT TYPES

Exploration by PGSA at Cap Oeste 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 to Ag generally greater than 1 to 10. Mineralized veins and breccias consist of quartz (colloform, banded, and chalcedonic morphologies), adularia, bladed carbonate (often replaced by quartz), and dark sulfidic material termed ginguro (fine grained electrum or Ag sulfosalts 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 style 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 show that precious metal precipitation generally occurs between 180 to 240 degrees 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).
- Barren pyritic root.

Alteration accompanying low sulfidation epithermal mineralization is controlled by the temperature and pH of the circulating hydrothermal fluids and its distribution therefore can also 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).

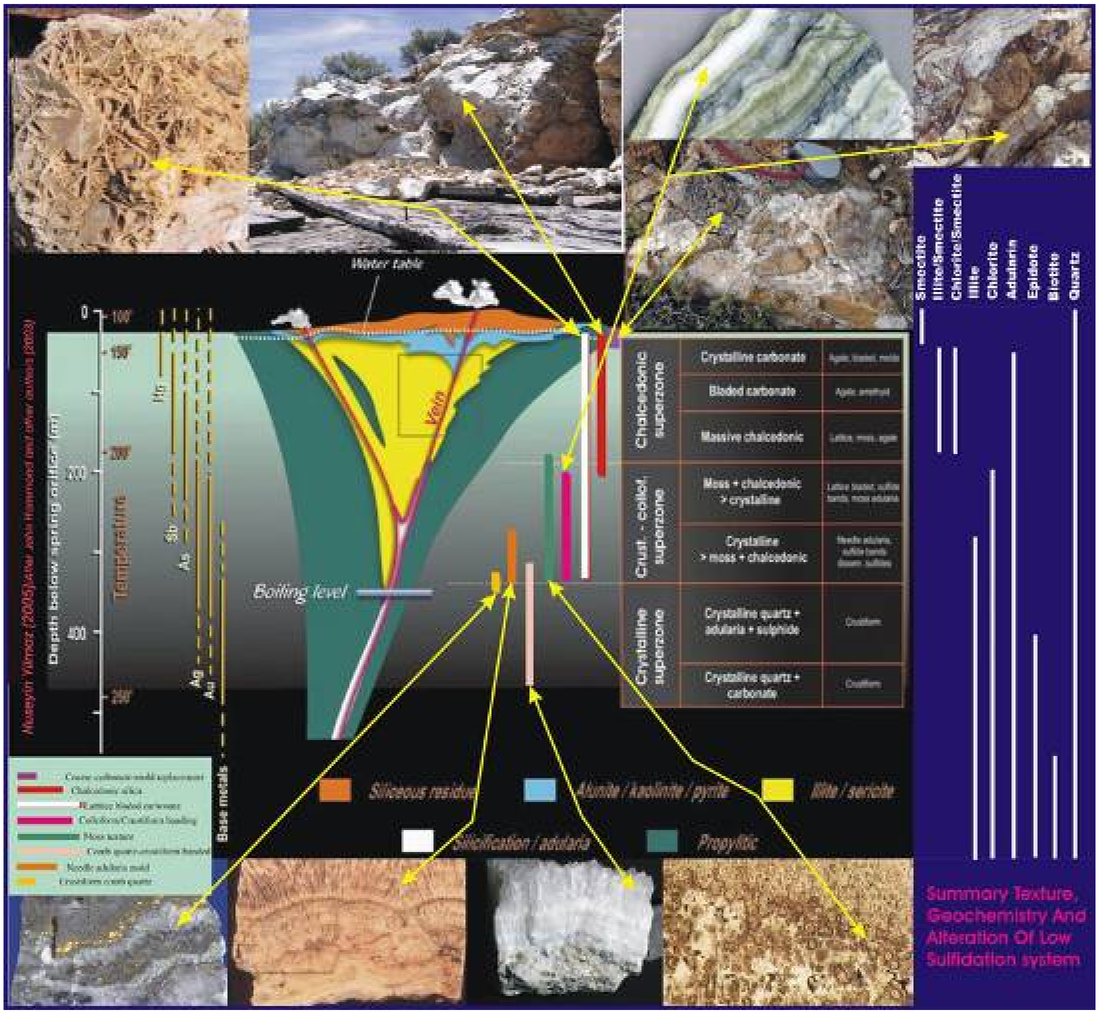


Figure 8-1
Geochemical zonation, quartz type and alteration patterns of low sulfidation hydrothermal system

Based on observations by R. Sillitoe (2008 & 2009), mineralization at Cap Oeste is assigned to a shallow epithermal, low sulfidation type of mineralization, specifically:

> *"The presence of fine-grained replacement quartz, widespread illite alteration, abundant marcasite, silver-bearing sulfosalts and late-stage realgar and orpiment combine to confirm that Cap Oeste formed in the epithermal environment, potentially in relative proximity to the paleosurface. The abundance of arsenopyrite, a sulfide that precipitates under reduced conditions, suggests that the prospect is assignable to the low-sulfidation epithermal category."*

The PGSA staff believes that mineralization occurs as a result of a fault-localized combination of hydrothermal breccia, replacement, veinlet and disseminated style body rather than as one or more discrete quartz veins, as is the typical style for similar deposits elsewhere in the Deseado Massif. As described previously, replacement style, low sulfidation deposits do occur elsewhere and include the disseminated ore bodies in non-welded ignimbrite at Round Mountain, Nevada, the lithic tuff hosted deposit of McDonald Meadows, Montana, and the breccia-hosted orebodies at Ladolam in Lihir Island, Papua New Guinea (Sillitoe, 2008). In contrast to low-sulfidation systems, high-sulfidation epithermal deposits are normally replacement bodies, commonly localized along faults. A high-sulfidation assignation for the Cap Oeste Project has been ruled out by the neutral-pH illite dominated alteration style and the complete absence of vuggy quartz and associated advanced argillic alteration assemblages.

9.0 EXPLORATION

9.1 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. The initial emphasis was to validate Barrick data for the Breccia Valentina and Cap Oeste prospect areas, in preparation for the first stage of drill testing in September 2007.

Work completed to mid-2011 includes:

- Establishment of local grid baseline points at origin- 5000E, 10000N- to allow projection of trench and drill section data on sections perpendicular to the northwest strike of mineralization.
- Geologic mapping at 1:1,000 scale.
- Excavation and sampling of five trenches, (224 meters and 82 channel samples).
- Completion of 48,857.6 meters in 295 drill holes including :
    - 29 RC drill holes (totaling 2,044 meters averaging 66m in depth) and 1,669 samples.
    - 60 holes with RC pre-collar (3,680.60 meters of RC) and HQ DDH tail (7,509.83 meters, with 5,008 samples)
    - 206 HQ diamond drill holes (35,623.20 meters averaging 172.92 m in length) and a total of 9,577samples. Three of the diamond holes are twin holes of earlier RC holes: CO-001-R & CO-036-D, CO-009-R & CO-034-D, CO-010-R, and CO-035-D.
- Petrographic study of 28 samples in thin and polished sections.
- Visits from international-recognized geological consultants Greg Corbett (2007), Richard Sillitoe (2008 & 2009) and Tony Starling (2011).
- Survey topography with a differential GPS and develop a contour map.
- Survey of all drill hole and trench locations in x, y, and z dimensions with a differential GPS.
- IP/resistivity surveys (7 lines totaling 6.3 line kilometer gradient array; 1 line totaling 1.6 kilometres pole-dipole), Ground magnetic survey (10 lines totaling 13 line kilometres).

9.2 Gridding, Topography and Surveying

Local baseline grid points were surveyed with the origin defined at 5000E, 10000N. This grid is tied into the Gauss Kruger Projection and Campo Inchauspe Faja 2 datum coordinate system. Surveys utilized a double frequency (L1 and L2), TOPCON Model GB-1000 differential GPS which generally gives precision of X=1 cm, y=1 cm and Z (altitude) =1.5 cm.

The same equipment was employed to survey trench and drill hole collar locations in addition to providing topographic control. Topographic control was facilitated with the collection of coordinate and altitude data on a 5 by 5 meter grid spacing over a 450-ha area from which the data points were subsequently contoured using triangulation parameters.

9.3 Trenching

By May 2007, five trenches totaling 224 meters were mechanically excavated (PGTR_12 to PGTR_16; Figure 9-1). Trenches PGTR_12 to PGTR_14 were excavated adjacent to Barrick's original trench TR-4 along 50-meter spaced lines and PGTR_15-16, were excavated 550 meters and 750 meters respectively to the northwest.

The most significant precious metals values reported from trench PGTR_14, which returned 37 meters @ 0.52 ppm Au (0.2 ppm Au cutoff), including 8 meters @ 5.77 ppm Au and 17.3 ppm Ag (2.5 ppm Au cutoff).

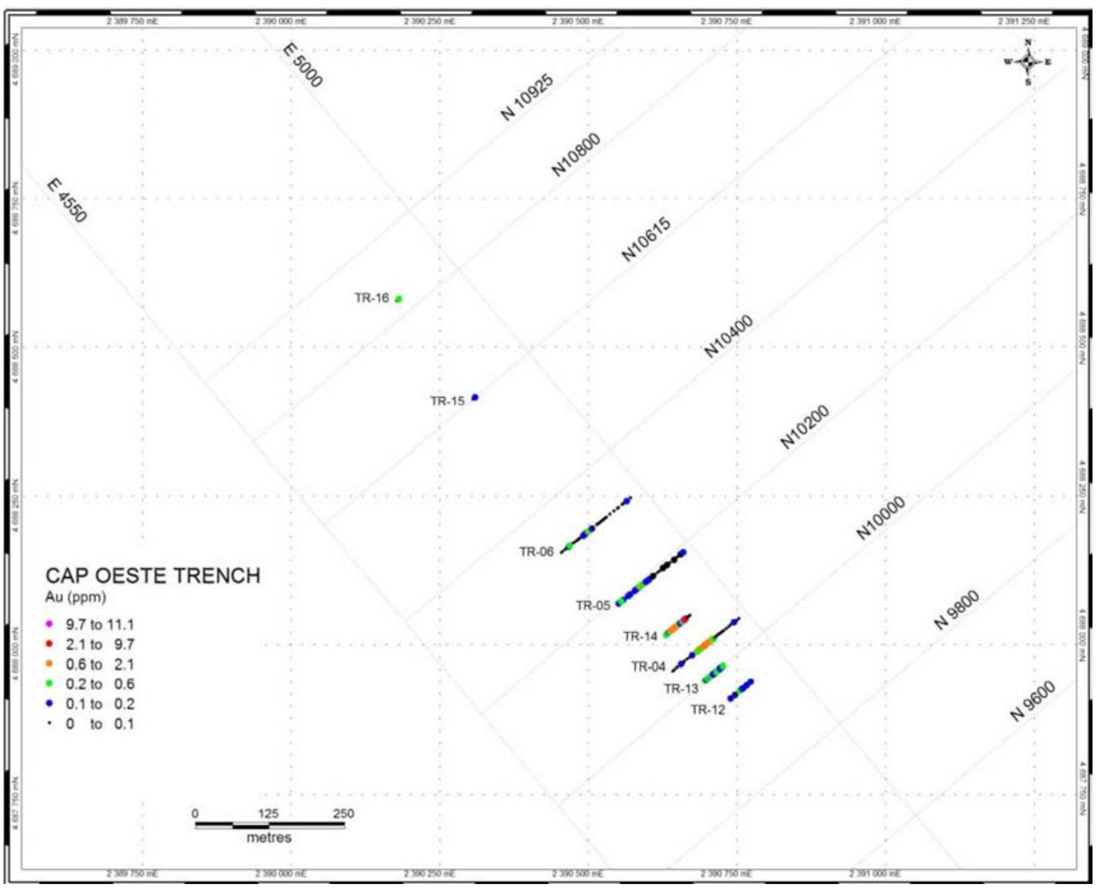


Figure 9-1
Cap Oeste Trench Locations

Trench locations 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-cm by five-cm slots were mechanically dry sawn, cleaned, and sampled. Trench sampling and logging were carried out under the supervision of PGSA geologists; sample intervals were generally marked using a measuring tape following geological criteria (e.g. zones of similar mineralogical/geological features). Sampling of the trenches comprised chipping between the two sawn slots with hammer and chisel to the limits of marked sample intervals and placing the broken material in plastic sample bags. Each sample bag is tagged and staple 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 by a qualified surveyor.

9.4 Geophysics

Based on the observed correlation of precious metals mineralization with disseminated sulfides and varying degrees of silicification, and the effective application of regionally spaced, pole-dipole IP surveying by Barrick Gold, both pole-dipole array and gradient array geophysical surveys were applied as tools for the detection of additional concealed mineralization. Baseline ground magnetic data were also collected in hopes of mapping fault-related displacements within the volcanic stratigraphy.

9.4.1 Pole-Dipole Induced Polarization

Pole-dipole IP surveys were completed by Barrick Gold along a 1600 meter portion of local grid section 9950N between 4100 E and 5700E, across a well-defined, mineralized section. The survey was performed with dipole spacing of 50 meters expanded through 6 separations (n=1 to 6). The chargeability anomaly which occurs broadly in the centre of the test survey line correlates with the occurrence of up to 10 percent sulfide below the level of oxidation, within the Bonanza Fault and its immediate footwall rocks.

Figure 9-2 depicts pole-dipole chargeability survey results along Section 9950N. The inverted section (lower) shows the survey results with geological and drillhole information overlaid. Note apparent correlation of the principal zone of chargeability (magenta-red) with the footwall to the Bonanza Fault structure.

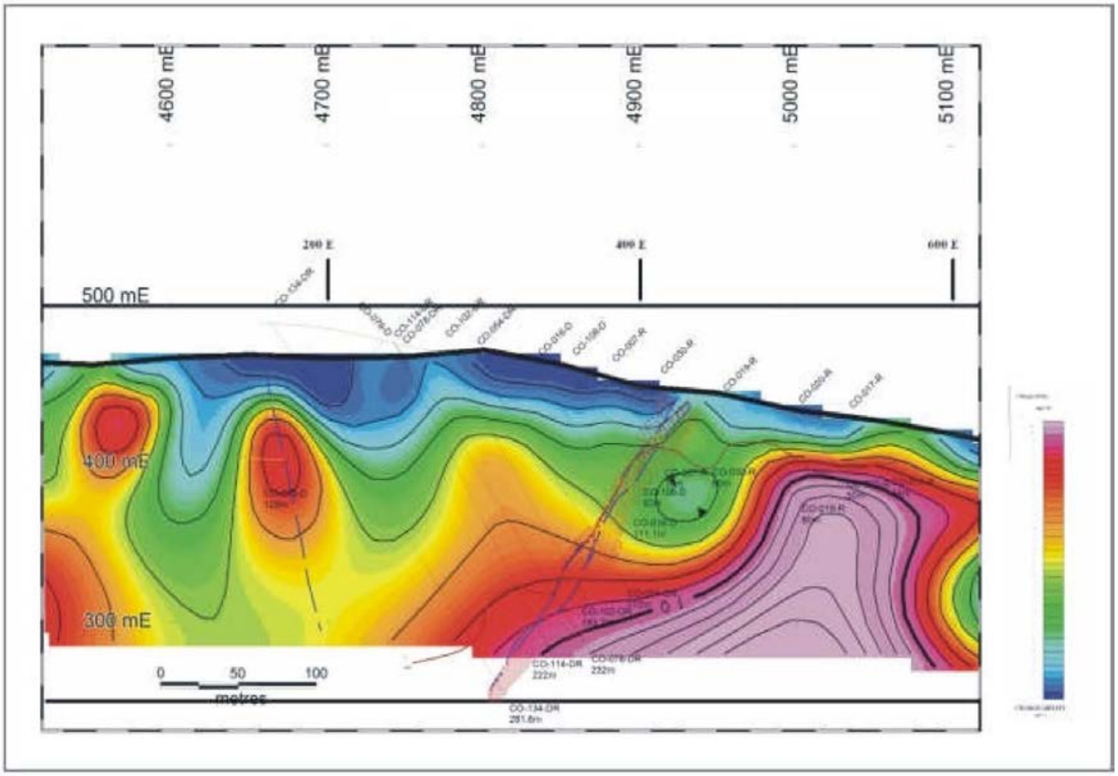


Figure 9-2
Pole-dipole Chargeability Inversion (Section 9950 N) with superimposed drilling, faults and mineralization. Chargeability high appears to reflect sulfide-rich Bonanza Fault and associated footwall alteration

A zone of high apparent resistivity is offset slightly to the west of the conductivity anomaly (Figure 9-3); this is interpreted as due to the presence of silicified breccias within the mineralized envelop, augmented by greater degrees of silicification within the vitric tuff unit.

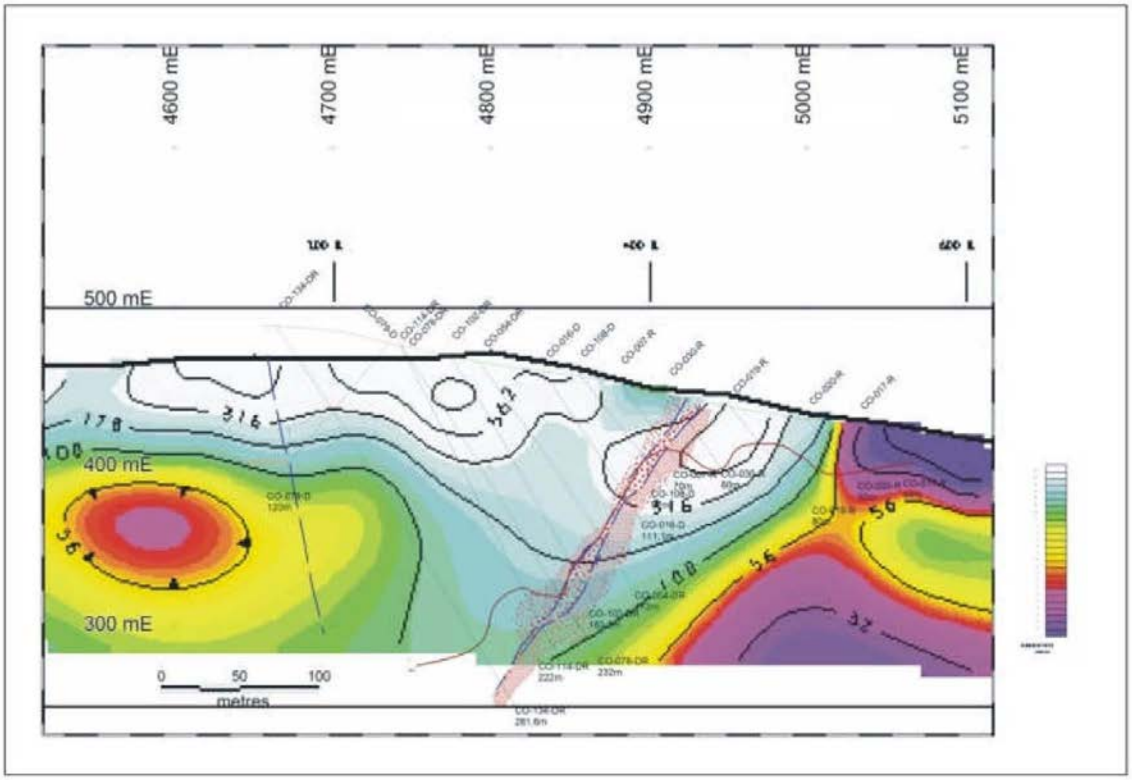


Figure 9-3
Pole-dipole Resistivity Inversion (Section 9950 N) with superimposed drilling, faults and mineralization. High resistivity reflects silicification within the Bonanza Fault

9.4.2 Gradient Array Induced Polarization

A baseline gradient array IP survey was conducted along 100-meter spaced lines over the entirety of the project area. The gradient array data is presented as plan maps of total chargeability and apparent resistivity (Figures 9-4 and 9-5); both are draped with surface geological data and drillhole locations.

Coincident, northwest trending chargeability and resistivity anomalies are evident in these plots which mirror the strongest mineralized zone between Sections 9800N and 10350N. Peak chargeability is broadly coincident with the southwest dipping Bonanza Fault. Towards the northwest the anomaly resolves into sub-parallel anomalies which are coincident with the mapped traces of the Bonanza and Esperanza Faults.

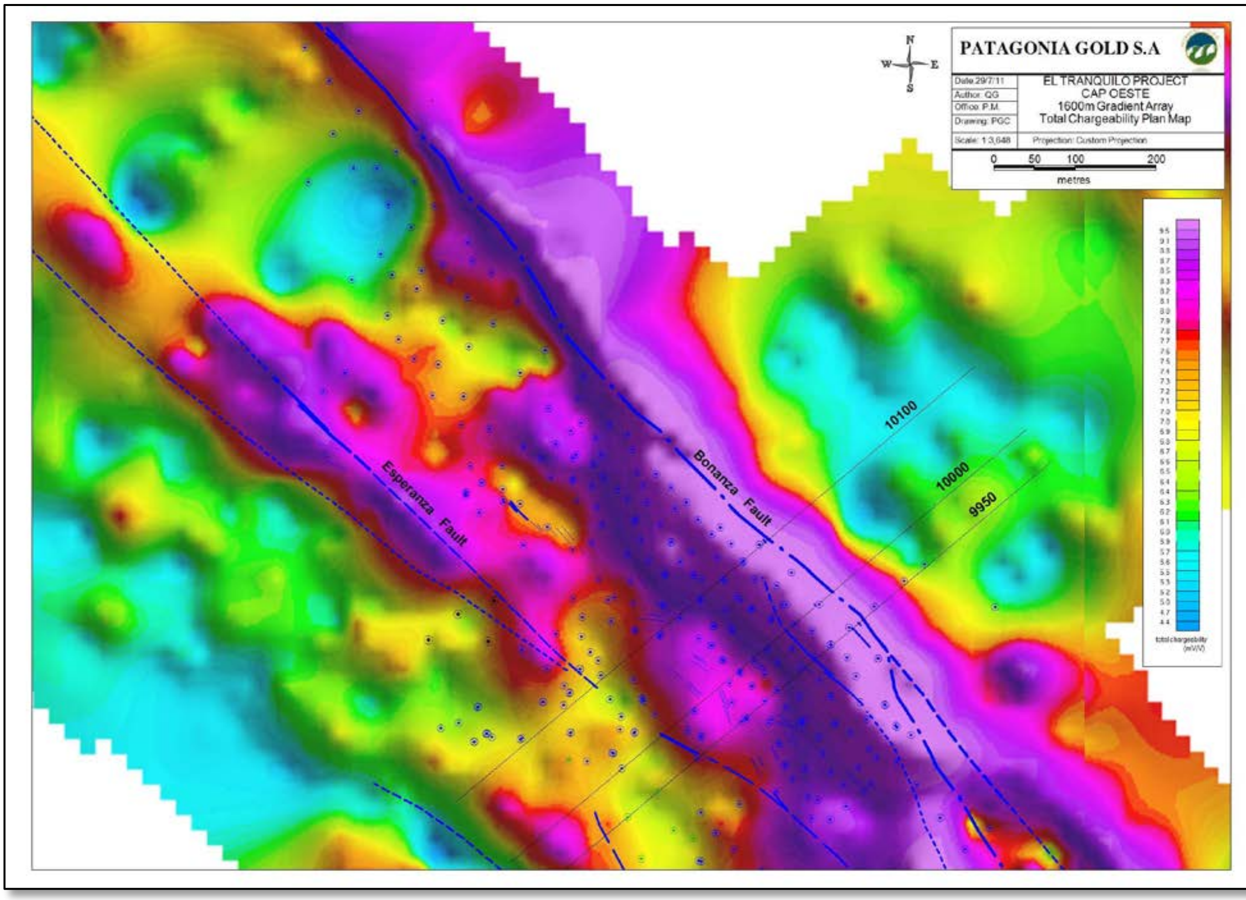


Figure 9-4
Gradient Array Chargeability Plan Map

Peak resistivities are interpreted to reflect the combined effects of silicification along the bounding faults and relatively higher resistivities within the vitric tuff unit mapped within the northwest trending graben. As is the case for conductivity, resistivity anomalies resolve into discrete, linear zones to the northwest of the strongest mineralization, presumably reflecting silicification along the main graben-bounding faults.

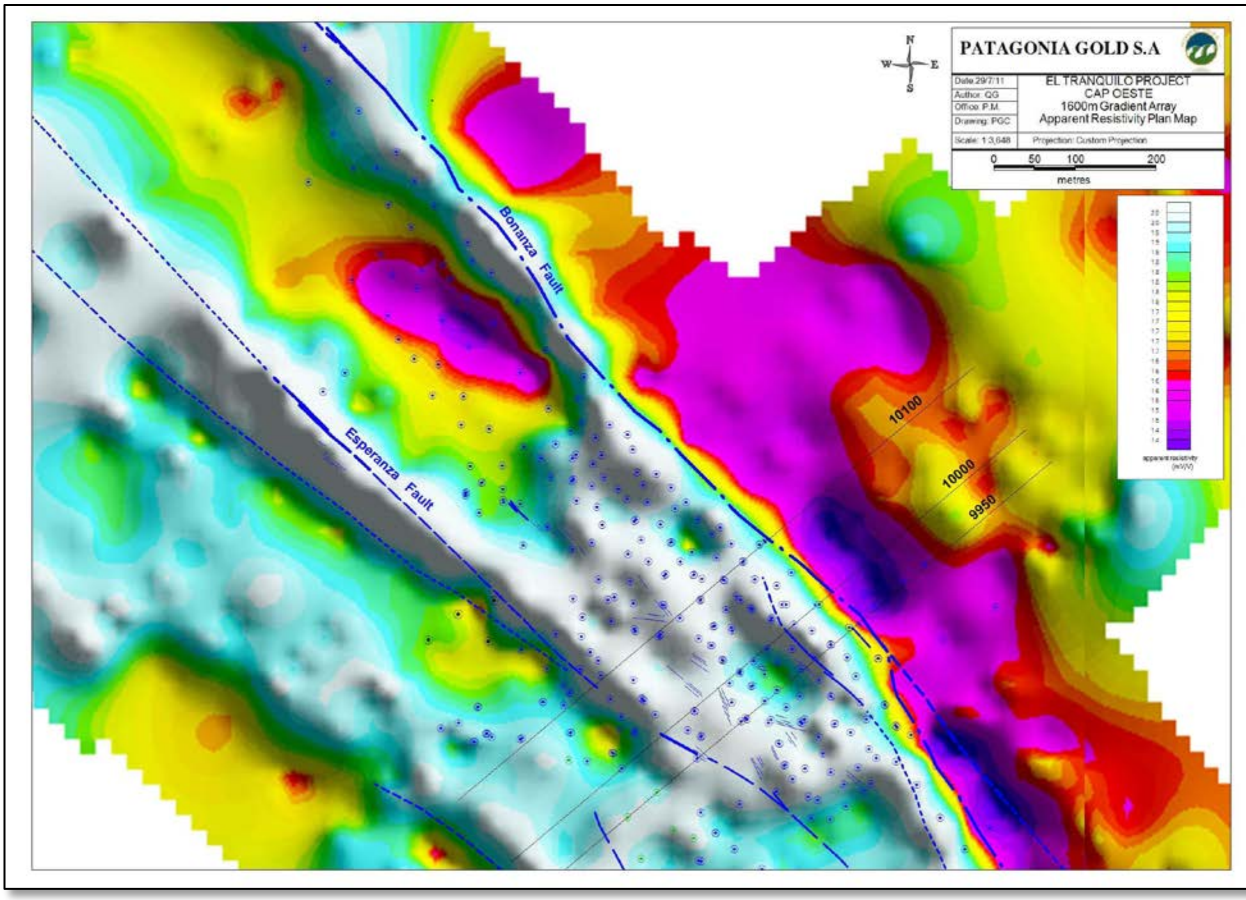


Figure 9-5
Gradient Array Resistivity Plan Map

9.4.3 Ground Magnetic Surveying

Baseline ground magnetic surveying was carried out, utilizing 10-meter spaced points along 100-meter spaced, 1-kilometer long lines throughout the southeastern portion of the project area. Overall this method was only moderately effective at defining structures using these survey parameters.

9.5 Petrography and Computed Axial Tomography

A suite of 28 samples was selected from HQ drill core and shipped to a petrology consultant in Australia for preparation and petrographic analysis. A summary of sample descriptions and interpretations from this study are discussed above in Section 9.3.

In addition, two core samples, representative of both oxidized and un-oxidized mineralization types, were studied using computed axial tomography (CAT scan) at the Department of Mineralogy of the Natural

History Museum, London, UK. A summary of sample descriptions and interpretations from this study have also been discussed previously.

9.6 Exploration Potential

9.6.1 Cap Oeste Project Area

Based on interpretation of exploration results to date, CAM believes moderate to good exploration potential remains within the immediate resource area:

- Drilling between and along strike of ore shoots. In particular, the very high grades and small footprint of the COSE deposit encourage the notion that similar mineralization will occur along extensions of the Bonanza Fault, particularly in those areas with marked variations in strike and which exhibit crosscutting subsidiary structures.
- Drilling for extensions to the broad zone of stockwork mineralization which has been discovered near the intersection of the Bonanza and Esperanza Faults, at a depth of approximately 300 meters below surface (125 m RL).
- Drilling to delineate discrete zones or shoots of economic precious metals within the relatively unexplored Esperanza Fault. Narrow intercepts of chalcedonic veinlets with high grade precious metals are known to occur along the fault (DDH CO-166-D; 1.1meters @ 435 g/t Au, 1006 g/t Ag).

9.6.2 Regional Targets

Exploration outside the Cap Oeste deposit consists of trenching, rock chip sampling, geophysical surveys and drilling (15,908 meters total of which 9,970 meters have been drilled on the COSE deposit) spread between seven prospects. As discussed above in section 7.3.1 , this work has delineated significant additional mineralized occurrences within the Cap Oeste structural corridor as well as within the adjacent Breccia Valentina and Vetas Norte trends. CAM is of the opinion that further discoveries will be made in these areas, utilizing the proven combination of surface prospecting, ground geophysics, and drilling.

10.0 DRILLING

10.1 Introduction

Drilling of RC and diamond holes (DD) at Cap Oeste has been carried out in four separate campaigns under contract by Patagonia Drill S.A and Major Drilling S.A. (October through to June, 2008), Major Drilling S.A. (October, 2008 to May, 2009) and Major Drilling S.A (January, 2011 to present date) utilizing truck-and track-mounted Universal UDR 650 rigs, respectively. Diamond drilling by Major Drilling.

Both Patagonia Drill and Major Drilling conducted the drilling in the first campaign (October, 2007 to June, 2008) and then only Major Drilling conducted the drilling in the second, third and fourth campaigns (October, 2008 to June, 2009 and January, 2011 to current date).
Drillhole naming adopted the following nomenclature:

- *Project*- prefix CO (Cap Oeste)
- *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: CO-016-DR is Cap Oeste diamond drillhole #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. CO-Abandoned hole CO-152-D replaced by CO-152A-D and subsequently CO-152B-D.

Figure 10-1 shows drillhole locations on the project, to June 30, 2011.

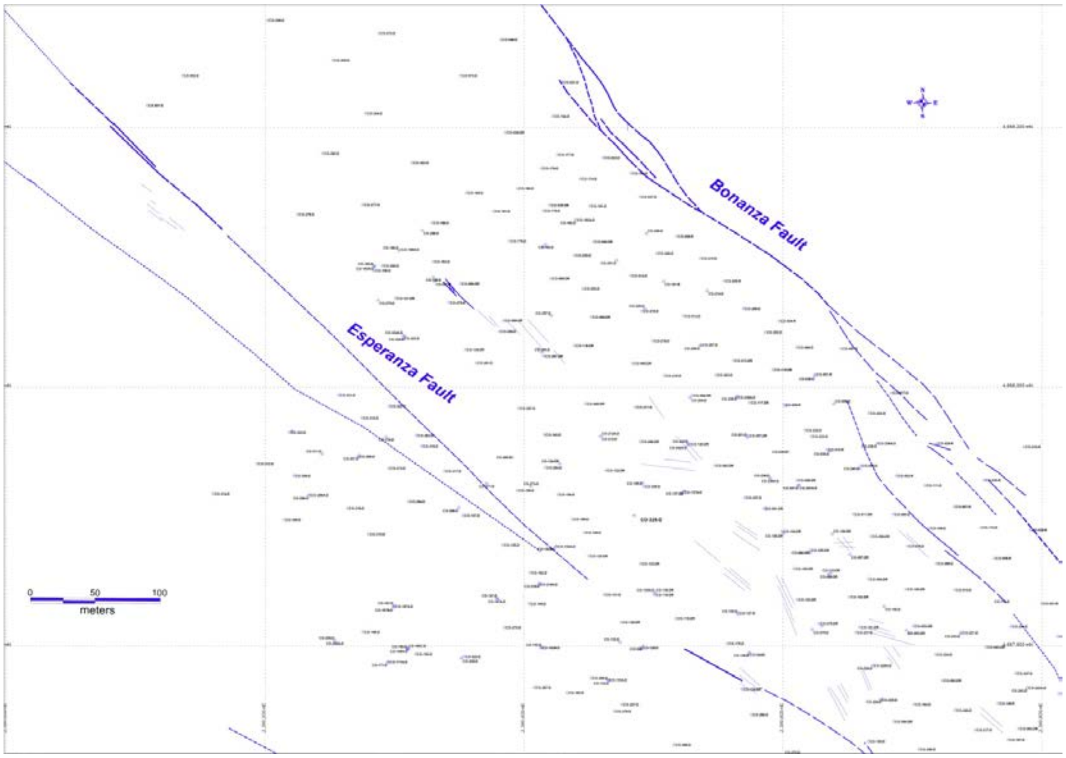


Figure 10-1
Cap Oeste Project Drill Hole Collars to hole CO-300

10.1.1 October 2007- June 2008 Drill Campaign

A first tranche of RC drilling, designed to test the strongest zones of mineralization as defined by trenching, commenced in October 2007 along 50-meter spaced centers (CO-001-R toCO-010-R).

Encouraging results led to the continuation and expansion of the program during 2008, specifically:

- The use of diamond drilling in preference to RC drilling through the deeper (greater than 40 meters down dip) projected zones of mineralization. This was based on the relatively high water table, the silica-poor, clay-sulfide-rich character of mineralization, and resulting concerns over RC recoveries and ability to obtain representative samples.
- Twinning of first stage RC holes with DDH to check influences of wet sample intervals and low recoveries on grade bias (Three of the diamond holes are twin holes of earlier RC holes namely CO-001-R & CO-036-D, CO-009-R & CO-034-D, CO-5010-R, and CO-035-D).
- Drill testing along the entirety of the strike length of the breccia/fault zone identified by trenching and zones extrapolated under areas of post mineral cover between previous drill sections.

Most drill holes were collared on 50-meter spaced sections 050 degree azimuths and with inclinations between -50 and -70 degrees (Figure 11-1). This configuration was designed to intersect the southwest dipping mineralized zone as perpendicularly as possible with increasing depth. One hole CO-079-D was drilled towards 230 degrees GK grid (i.e. Local Grid west), in order to intersect the Esperanza Fault.

10.1.2 October 2009- May 2009 Drill Campaign

From October, 2009 to May, 2009, emphasis was placed on further infill delineation of the Main Shoot with 20 to 25-m spaced drill holes planned to be completed on 25-m spaced sections and further definition of the down plunge extension of the shoot to a depth of approximately 325m below surface (i.e. 100 m RL), through a series of 3 to 5 step-out holes on selected sections.

Initially, the step-out holes were generally drilled with RC pre-collars to the approximate depth of the water table or to a point before the start of the interpolated depth of possible mineralization, after which the universal drill rig was converted to complete the drill hole using conventional coring equipment. During the infill stage drilling between January, 2009 and May, 2009 it was considered that drill hole deviation was best controlled by drilling with conventional core drilling equipment for the entire length of the drill hole.

10.1.3 January 2011 – Current Campaign

From January 2011 and continuing the present date, an infill HQ diamond drill program comprising 25m spaced holes was conducted in order to primarily further delineate the oxide mineral resource between sections 9775 – 10400N, down to approximately 300m RL.

The drill collar information for the Cap Oeste deposit contained within the drillhole database as of 30th June, 2011, is presented in Figure 10-1.

10.2 Diamond Drilling Methods

Drill hole collars were initially located using a hand-held GPS, in addition to triangulation from adjacent previously drilled and surveyed collars. For each drill hole, the direction of drilling (azimuth and inclination) at each collar was defined by PGSA geologists using a Brunton or Suunto compass.

Diamond drilling was carried out on a 24-hour basis using 12-hour, day and night shifts during which PGSA trained technicians were on site at all times in order to record drilling activities in a Drill Log sheet (e.g. drilling, reaming time, additives, core recovery, rock quality density, down hole survey information) and supervise the extraction of the core from the diamond core barrel and placement into the core cradle.

Permanent radio contact was maintained between the PGSA technician at the drill site and the PGSA geologists at base camp.

All diamond drilling was of HQ diameter and utilized a 3-m core barrel where ground conditions permitted. In only two cases the hole diameter had to be reduced to NQ size (CO-147A-D and CO-189-D). For diamond drilling conducted from January, 2009 to present, the use of a core barrel sleeve tube (HQ3) was implemented prior to entering the zone of interest in order or to maximize the recovery and rock quality of the drilled interval.

Water utilized for drilling was sourced from a series of spring-fed pits excavated in the northeastern portion of the project area within a 2 km distance from the project area. No orientated core surveys were carried out during diamond drilling due to the generally fractured state of the rock.

Daily site visits, which collectively comprised several hours onsite time, were made by the PGSA geologist / project geologist for review of drilling progress, drill planning and quality control.

During the drill campaigns conducted up until December 2008, down hole surveys were generally taken by the drill contractor every 50 meters upon termination of each drill hole utilizing either a Eastman single-shot camera (in the case of Patagonia Drill) or a digital, multishot, FLEXIT down hole survey tool (in the case of Major Drilling). In the case of the single-shot camera a downhole photo was produced at the respective depths and in the case of the FLEXIT the hole inclination, direction (azimuth), magnetic field strength, gravity roll angle, magnetic tool face angle and temperature were recorded.

Depending on the presence and depth of casing in each hole, collar survey photos were generally taken to within 5 to 10 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.

During the drill campaigns conducted from January 20 11 to June 2011, which predominantly comprised the drilling of the deeper infill holes, each hole was surveyed at 25m intervals from the collar to a depth of 100m after which the survey intervals were increased to 50m intervals to the end of each hole. 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 28 holes including CO-128A-D, CO-133A-D, CO-135A-D, CO-137A-D, CO-143A-D, CO-147A-D, CO-152A-D and subsequently CO-152B-D, CO-163A-D, CO-165A-D, CO-167A-D, CO-169A,D, CO-188A-D, CO-190A-D, CO-192A-D, CO-198A-D, CO-208A-D, CO-212A-D,CO-236A-D, CO-238A-D, CO-239A-D, CO-241A-D, CO-242A-DC and CO-243A-D, CO-272A-D, CO-272B-D, CO-274A-D,CO-287A-D, CO-294A-D, CO-296A-D, CO-296B-D and CO-297A-D).

For several of the holes drilled during April-May 2009 PGSA rented a FLEXIT MultiSmart™ multishot downhole surveying tool in order to check the accuracy and precision of the downhole survey camera used by Major Drilling and to resurvey several holes drilled/surveyed in previous campaigns. This tool was used to survey the holes on completion at 50m intervals the results of which were then compared to the original surveys conducted by Major during the drilling of the hole. Overall, very close correlation was achieved between the readings taken by the two individual cameras.

Examination of the survey data shows that, overall, there exists a consistent tendency for the diamond drill holes to deviate clockwise to the south east averaging between 2° and 6° over the course of 300 m. Although generally the inclination of the drill hole remained true throughout drilling of each hole, additionally there exists a tendency for the holes to drop in inclination of between 1° and 3° over similar hole lengths.

Following termination of each hole, the collars were marked clearly and permanently with capped PVC tubing cemented in a square concrete base. Following the completion of drilling, the collars were surveyed by a qualified surveyor utilizing a differential GPS.

10.3 Drill Core Logging

Core logging was carried out at Estancia La Bajada, which is situated approximately 5 kilometres from the Cap Oeste Project area. 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 are 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 according to the sampling protocol displayed in Figure 10-2. 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.

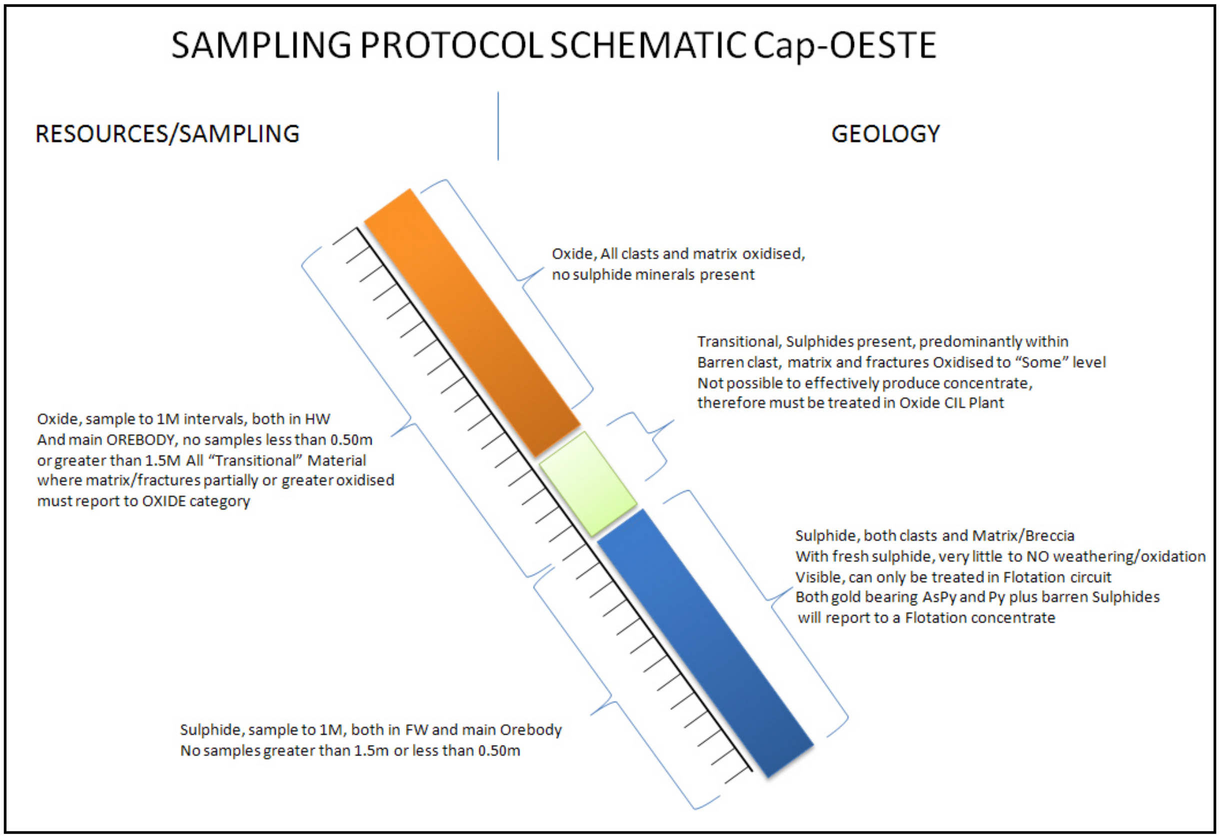


Figure 10-2
Cap Oeste sampling protocol for oxide-transitional un-oxidized mineralization

All the graphical and coded logs were recorded on paper log sheets at a scale between 1:100 and 1:200, depending upon the intervals of interest, in addition to the sample intervals and sample numbers defined by the PGSA geologist. This information was subsequently entered digitally by PGSA technicians into a 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.

10.4 Reverse Circulation Drilling Methods

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 5¼-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 1-m intervals were clearly marked on the drill mast which acted as a guide for the drilling contractors in sample collection. Subsequent to each six meter 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.

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

10.5 Results of Drilling

A series of 25-metre-spaced geological sections were generated by PGSA geologists using Mapinfo/Discover GIS software over the length of the Cap Oeste Prospect area from which interpreted lithological boundaries, zones of oxidation, mineralization and structural features were defined.

A summary of significant drill intersections based on a minimum cutoff grade of 0.5 g/t Au for the Cap Oeste project area is presented in Table 10-1. The majority of these drill intersections relate to the three higher grade shoots (Main, Shoot B and Shoot E) and to one less well defined northwestern plunging shoots (Shoot D).

As drilling to date has largely been directed to the northeast, intersection of the steeply northeast-dipping Esperanza Fault is limited to only several intervals cut sub-parallel or at a low angle to the core axis. As such, it is believed that the potential of the Esperanza Fault to host significant gold-silver mineralization has not been fully evaluated.

Table 10-1
List of Significant Drill Hole Intersections, Cap Oeste Deposit

Hole No.	Depth (m)	Section	From (m)	Interval (m)	Gold Grade (g/t Au)	Silver Grade (g/t Ag)
CO-001-R	80	10100	44	13	11.78	47
	including		50	5	29.28	85
	and		65	5	1.4	28
CO-002-R	74	10050	23	12	0.74	6
	and		48	6	2.36	135
CO-003-R	80	10000	11	12	0.65	3
	and		57	14	0.44	107

Table 10-1 List of Significant Drill Hole Intersections, Cap Oeste Deposit						
Hole No.	**Depth (m)**	**Section**	**From (m)**	**Interval (m)**	**Gold Grade (g/t Au)**	**Silver Grade (g/t Ag)**
CO-004-R	56	10150	24	17	0.69	23
CO-005-R	55	10200	33	22	0.7	11
CO-006-R	60	10250	12	20	0.81	10
CO-007-R	70	9950	47	11	3.14	60
	including		49	5	5.79	57
CO-008-R	70	9900	40	10	3.65	30
	including		41	5	5.8	33
CO-009-R	120	10100	73	17	0.69	21
CO-010-R	111	10050	74	37	0.77	28
CO-011-DR	123.25	10000	103	9.5	2.89	65
	including		106.95	4.7	5.23	133
CO-012-DR	114	10150	83.6	29.4	0.51	19
CO-013-DR	123	10200	77.3	21.2	0.8	25
CO-014-D	111.25	10250	78.3	11.5	0.61	14
CO-015-D	117	9900	75.85	14.35	2.38	19
	including		75.85	3.15	8.58	28
CO-016-D	125	9950	91.95	13.45	11.5	389
	including		91.95	3.65	40.28	1,373
CO-022-D	62.95	10330	31.15	14.85	0.72	8
CO-023-D	68.85	10525	31	7	0.76	1
CO-024-D	78.05	10620	39.1	7.7	0.77	4
CO-025-D	87	10675	22	9	0.68	2
CO-028-R	56	9900	11	20	1.4	18
	including		28	2	5.28	16
CO-029-R	56	10000	27	5	0.85	173
CO-030-R	60	9950	19	10	1.32	18
CO-031-R	68	9900	11	11	2.27	2
	including		18	3	7.08	1
CO-032-R	62	9850	15	5	0.77	1
CO-034-D	150.95	10100	77	14.1	1.58	65
	including		89	2	6.17	40
CO-035-D	146.9	10050	112.7	7.3	1	13
CO-036-D	108.1	10100	47.1	12.3	14.27	56
	including		52.6	5.3	31.61	100
CO-043-DR	110	9900	77	7	0.7	49
	and		91	12.5	0.74	19
CO-044-DR	89	9850	55	5	5.48	33
	including		56	3	7.77	12
CO-045-D	74	10550	35	11.4	2.27	1
CO-050-D	111	10525	65	5	1.21	11
CO-051-D	111	10625	79	12	3.2	27
	including		81	4.1	5.96	27
CO-053-DR	164	9900	120.8	5.2	1.33	128

Table 10-1 List of Significant Drill Hole Intersections, Cap Oeste Deposit						
Hole No.	**Depth (m)**	**Section**	**From (m)**	**Interval (m)**	**Gold Grade (g/t Au)**	**Silver Grade (g/t Ag)**
CO-054-DR	172	9950	132	7	48.11	769
	including		133.1	2.8	118.43	1,875
CO-055-DR	187	10000	160	11.8	2.79	144
	including		168.3	2.7	9.28	453
CO-056-DR	180	10050	116	18	0.82	49
CO-057-DR	170	10100	110.6	30.35	0.58	22
CO-058-D	105	10075	57.42	7.28	2.21	355
	including		59.03	0.97	6.13	1,968
CO-059-D	119	10025	65	22.6	0.56	45
CO-060-D	141	9975	88	7	4.01	51
	including		89	4	6.15	54
CO-062-DR	153	9850	119.9	15.6	1.56	72
CO-065-DR	173	10200	140.9	30.1	1.59	18
CO-066-DR	150	10200	131	11.75	1.63	32
CO-067-DR	46.5	10100	3	24	1.85	76
	including		23	3	5.89	401
CO-068-D	56	10125	23	25	0.67	10
CO-069-D	97.5	10550	76	9.3	1.18	31
CO-070-D	149.5	10625	127	10.2	2.38	178
CO-071-D	150	10550	123.1	4.2	1.22	6
CO-074-D	117	10700	84	11.7	1.03	32
CO-075-D	108	10760	78	5.5	2.33	78
CO-077-D	51	10050	6	19	0.83	18
CO-078-D	232	9950	181.65	37.35	1.04	11
CO-080-DR	231	10000	161	27	7.95	132
	including		170	15.2	13.06	206
CO-081-DR	205	10050	156.6	17	2.24	127
	including		168.2	3.2	8.41	216
CO-082-DR	232	10100	172	6	0.91	32
CO-083-DR	192	9900	145.5	5.4	1.39	23
CO-084-DR	214	9850	175	31.1	1.24	24
	including		179.63	1.82	5.48	143
CO-085-DR	225	10200	187	6	1.16	68
CO-086-DR	226	10150	193.9	7.1	4.84	208
	including		196.1	3.9	7.72	314
CO-087-DR	261.83	10250	196.15	9.85	0.87	9
CO-089-DR	211	10300	182.35	21.65	1.24	17
CO-090-DR	221	10350	192.2	15.3	3.35	20
	including		197	2	19.7	15
CO-096-DR	137.1	9975	107.25	10.75	3.9	142
	including		111.5	1.75	18.6	739
CO-097-DR	146.8	9975	132.92	6.8	10.92	1,711
	including		132.92	2.62	24.27	3,963

Table 10-1
List of Significant Drill Hole Intersections, Cap Oeste Deposit

Hole No.	Depth (m)	Section	From (m)	Interval (m)	Gold Grade (g/t Au)	Silver Grade (g/t Ag)
CO-098-DR	182.7	9975	153.8	16.9	3.2	157
	including		163.7	4	11.19	557
CO-099-D	102	9925	80.1	2.3	51.92	618
	including		80.1	1.18	100.8	953
CO-100-DR	138	9925	112	4	2.24	16
CO-101-DR	192	9925	161.2	21.1	1.68	31
	including		163.4	1.35	7.89	111
CO-102-DR	183.2	9950	154.1	15.4	1.36	27
CO-103-DR	221.5	9975	186.25	10.8	2.14	92
CO-104-DR	198	10025	171	16.75	5.93	1,716
	including		178	9.75	8.87	2,466
CO-105-DR	219	10000	186.95	17.8	15.18	157
	including		193.1	1.9	97.06	111
CO-106-DR	186	10025	156	2	3.59	2,493
	and		161	8	1.22	154
CO-107-DR	210	10050	181	19.6	5.89	246
	including		189.7	8.55	11.07	486
CO-108-DR	92	9959	70.1	9.65	14.59	592
	including		70.1	3.9	25.95	595
	and		75.5	1.25	26.5	2,478
CO-109-DR	150	10000	127.6	11.7	2.3	49
	including		136	2.3	6.49	176
CO-110-DR	177	9975	152.3	4.2	9.77	1,470
	including		152.3	2.08	17.93	2,818
CO-111-DR	75	9975	53.2	5.3	3.63	800
	including		57	1.5	5.21	2,737
CO-112-D	60	9925	40.9	4.3	2.3	47
CO-114-DR	222	9950	176	1.5	7.66	9
	and		194.1	9.1	2.32	39
CO-115-DR	84	10125	70.37	9.13	3	61
	including		78	1.5	13.29	279
CO-116-DR	282	10050	258.5	13.15	4.51	98
	including		264	5.1	7.9	204
CO-117-DR	134.5	10112	110.5	7.3	3.54	31
	including		110.5	2.5	7.49	38
CO-119-DR	285	10025	258	13.2	14.02	186
	including		261	3	29.16	271
	and		265	5.3	15.26	270
CO-120-DR	180	10125	158	10.1	1.99	109
	including		165	1.57	9.06	606
CO-121-DR	255	10350	216	2	2.28	19
CO-122-DR	239	10150	217.5	4	2.53	187
CO-123-DR	282	10075	257.9	16.1	5.92	260

Table 10-1 List of Significant Drill Hole Intersections, Cap Oeste Deposit						
Hole No.	**Depth (m)**	**Section**	**From (m)**	**Interval (m)**	**Gold Grade (g/t Au)**	**Silver Grade (g/t Ag)**
	including		257.9	3.1	6.38	487
	and		269.57	3.53	12.41	619
CO-124-DR	246.7	10175	236.5	4.05	2.05	118
CO-125-DR	306	10100	278	10	5.27	860
	including		280.5	2.08	18.91	3,990
CO-126-DR	324	10050	283	32	2.53	20
CO-127-D	255	10000	234.5	13	5.57	853
	including		244.6	2.9	17.1	2,696
CO-129-D	306.2	10025	278	15.8	5.29	126
	including		281	7	8.4	239
CO-130-D	291.5	10000	258	14.1	3.58	60
	including		268	3	5.38	23
CO-131-D	338.8	10075	279	24	2.62	23
	including		290.5	3	4.79	39
CO-132-D	362.5	10050	280.4	32.7	2.86	45
	including		281.4	2.75	4.95	215
	and		298.65	1.65	5.22	85
	and		305.85	3.8	5.66	22
CO-133A-D	369	10025	300	22.5	1.92	13
CO-138-D	228	9850	203.25	10.05	1.64	13
CO-139-D	282	9962	244.7	11.8	15.21	203
	including		249	5.1	33.34	372
CO-141-R	133	10625	106	8	2.51	75
	including		111	2	5.6	134
CO-143A-D	373.3	10075	324	15.6	1.43	23
CO-144-D	353.8	10100	316	33	5.87	169
	including		323	1	18	452
	including		326.5	5	10.17	150
	including		346.05	1.65	45.13	2,137
CO-145-D	330	10125	271	29.9	7.81	280
	including		289.95	7.05	21.89	1,003
CO-146-D	269.6	10125	256	3.1	2.23	979
CO-147A-D	409.7	10125	323	1.3	13.5	186
CO-148-R	57	9825	31	17	3.61	137
	including		34	7	6.63	310
CO-149-R	127	9825	96	19	2.14	21
CO-151-R	97	10230	71	2	1.49	18
CO-150-D	351	10025	285.15	4.3	4.18	25
CO-152B-D	345	10125	300.94	11.06	4.56	17
	and		315.8	4.4	8.5	23
	Including		317.3	2.9	10.82	24
CO-154-D	306	10150	260.85	3.65	3.65	821
	and		289.35	6.3	5.35	45

Table 10-1 List of Significant Drill Hole Intersections, Cap Oeste Deposit						
Hole No.	**Depth (m)**	**Section**	**From (m)**	**Interval (m)**	**Gold Grade (g/t Au)**	**Silver Grade (g/t Ag)**
	Including		292.4	3.25	7.35	24
CO-155-D	366	10150	257.9	23.1	5.94	49
	Including		259	3.5	6.79	87
	Including		269.45	9.55	9.71	54
	and		300.95	20.5	4.05	189
	Including		313	8.45	6.42	366
	and		330	12.3	2.6	48
	Including		340.7	1.6	6.66	230
CO-156-D	339	10175	265	2.75	4	93.2
CO-157-D	387	10187	216	6	2.10	10.83
CO-161-D	218	9825	172.7	5.55	3.96	22.09
CO-162-D	386	10120	353.3	7.7	26.07	1322.19
	Including		353.3	0.95	96.60	8152.00
	Including		358	1	63.80	1406.00
CO-164-D	188	9850	155	5	4.17	107.91
CO-165C-D	435	10150	392	17.1	1.17	9.54
CO-166-D	414	9800	270	3.4	145.56	336.13
	Including		270	1.1	435.40	1006.00
CO-167B-D	393	10150	376.5	1.4	3.39	57.66
CO-168-D	441	10150	401	14.5	10.13	143.22
	Including		408	4.7	21.19	377.39
CO-170-D	309	9850	261.4	5.87	12.72	265.30
	Including		267.56	4.7	15.31	322.20
CO-171-D	317	10125	406.43	5.17	2.98	31.72
	Including		407.8	1.9	5.62	21.18
	and		417.86	0.44	8.80	63.31
CO-172-D	321	9850	281	1.95	3.33	88.63
	Including		282	0.48	6.94	17.19
CO-173-D	384	9825	334.23	2.99	2.99	72.81
	and		312.57	4.38	1.40	13.61
CO-175-D	135	10325	109	2	0.27	148.51
CO-176-D	186	10325	149.14	2.68	1.40	15.85
CO-177-D	75	10350	61	1	1.20	19.37
CO-178-D	321	10000	274.77	12.65	1.43	7.89
CO-180-D	129	10350	109.3	1.4	10.40	23.04
CO-181-D	165	10350	137.89	6.73	35.53	53.42
	Including		139.34	3.59	61.85	68.48
CO-182-D	402	10025	329.6	12.6	1.80	55.84
CO-183-D	219	10350	194.6	2.15	7.61	18.31
	Including		195.25	1.5	9.88	18.24
	and		202.3	5.47	6.37	5.00
	Including		206	1.77	13.36	11.72
	and		202.3	0.6	4.26	4.82

Table 10-1
List of Significant Drill Hole Intersections, Cap Oeste Deposit

Hole No.	Depth (m)	Section	From (m)	Interval (m)	Gold Grade (g/t Au)	Silver Grade (g/t Ag)
CO-184-D	60	10375	14.1	3	0.95	3.51
CO-186-D	189	10375	165.2	2.95	0.22	75.01
CO-187-D	284	10075	229.1	12.57	3.70	307.41
	Including		232.5	2.5	9.35	567.50
CO-188A-D	225	10375	210.6	7.5	1.83	11.34
CO-189-D	295	10125	267	22.58	2.28	39.88
	Including		269.7	7.08	3.52	59.46
	Including		283.73	1	7.27	247.00
CO-190-D	75	10318	14	2.2	0.90	4.86
	and		33	1.6	0.84	34.50
CO-191-D	96	10318	71.5	2.15	0.70	82.09
CO-192-D	126	10318	93.74	2.76	0.89	14.64
CO-193-D	150	10318	124.58	1.78	0.79	30.63
CO-194-D	180	10318	147.05	4.23	0.70	16.05
	and		163.52	1.39	1.95	8.83
CO-196-D	80	10600	31.17	2.09	7.52	14.40
	and		45	1.06	1.84	14.77
	and		48.2	1.03	1.00	8.50
CO-197-D	156	9786	125.1	2.25	24.50	140.79
CO-198-D	111	10600	85	6	2.55	117.82
CO-199-D	198	9786	157.47	2.87	0.86	5.53
CO-200-D	204	9786	179.75	2.25	1.24	17.87
CO-201-D	177	10100	141	3	1.95	26.93
	and		134	0.9	2.69	54.75
CO-202-D	84	10150	37	3	0.98	49.38
	and		51.3	3.23	1.58	68.40
CO-203-D	134	10150	105.7	8.3	2.16	146.16
	Including		105.7	3.3	4.03	312.66
CO-204-D	174	10150	148.17	3.21	2.30	153.55
CO-205-D	342	9825	296	14	2.64	17.07
	Including		298	8	3.40	20.85
	and		290	2	1.24	13.38
	and		312	1	2.15	0.25
CO-206-D	78	10175	46	17	2.69	344.95
	Including		48	3.48	8.76	1220.72
CO-207-D	117	10175	90.2	11.8	2.08	52.99
CO-209-D	138	10175	102	8.19	1.76	72.93
CO-210-D	162	10175	139.9	3.1	4.06	199.89
CO-211-D	192	10175	167.88	4.12	0.87	116.08
CO-212-D	223	10175	210.6	1.4	1.30	166.36
CO-213-D	252	9800	190	5	8.58	508.04
	Including		194	1	36.10	430.00
	and		200.7	2.6	53.74	700.39

Table 10-1
List of Significant Drill Hole Intersections, Cap Oeste Deposit

Hole No.	Depth (m)	Section	From (m)	Interval (m)	Gold Grade (g/t Au)	Silver Grade (g/t Ag)
CO-214-D	96	10200	60.83	8.91	3.14	49.93
	Including		64	2	5.97	58.11
CO-216-D	156	10200	110	2	2.10	34.50
	and		120	9	3.61	75.95
	Including		126	3	5.53	125.18
CO-217-D	177	9800	140	4	1.77	12.57
	and		146	6	1.61	3.89
	Including		150	2	3.03	1.22
CO-218-D	171	10225	126	8	6.14	61.97
	Including		130	1	30.32	102.92
CO-219A-D	381	10125	312.6	7.8	1.37	26.54
	and		331.2	13.3	1.92	313.21
	Including		331.2	5.6	2.11	650.80
CO-220-D	156	10225	106	2.34	1.59	64.79
	and		111.55	1.81	2.23	45.41
CO-222-D	399	10100	304	9	3.00	66.92
	Including		310	3	5.37	139.20
	and		352	31	5.29	178.36
	Including		353	3	4.20	86.39
	Including		360	13	8.79	224.66
	Including		381	2	4.49	85.51
CO-223-D	132	9875	100.67	3.33	2.04	14.05
CO-224-D	171	9875	130.3	2.7	3.56	61.18
CO-225-D	210	9875	182	14	1.31	31.77
	Including		182	1.85	4.40	162.93
CO-226-D	231	9875	196.4	7.8	1.84	668.17
CO-227-D	377	10000	302	18	1.92	7.94
	and		322	10	1.48	5.66
	and		335	2	1.26	4.01
	and		344	4.35	14.69	383.35
CO-228-D	225	9900	192	3.1	11.03	455.36
	Including		193.25	0.95	32.47	1233.36
	and		216	4	1.23	6.30
CO-229-D	411	10100	345	2	1.98	113.17
	and		382	4	4.11	107.39
	and		389	2	29.24	562.50
	Including		390	1	54.90	832.00
CO-230-D	174	9900	133.14	7.86	1.27	20.62
CO-231-D	211	9925	189.36	4.44	9.56	63.90
	Including		189.36	2.07	13.15	125.47
CO-232-D	126	10075	100	4	1.55	10.43
CO-233-D	165	10075	97	2	1.06	76.02
	and		114	2	1.15	23.05

Table 10-1
List of Significant Drill Hole Intersections, Cap Oeste Deposit

Hole No.	Depth (m)	Section	From (m)	Interval (m)	Gold Grade (g/t Au)	Silver Grade (g/t Ag)
CO-234-D	291	9850	250	1	1.53	275.00
	and		258	18	1.80	14.14
	Including		266	6	2.57	8.21
	and		279	2	1.06	75.60
CO-235-D	180	10075	118.1	2	1.04	14.23
	and		137.5	3	1.31	25.00
CO-236A-D	234	10075	171.77	2.23	1.10	111.77
	and		193	1	1.40	3.15
CO-237-D	246	10075	193	4	2.13	97.16
CO-238A-D	84	9985	36	2	1.38	8.74
	and		43	1	1.60	10.82
	and		60.23	0.9	1.66	141.13
CO-239A-D	159	10075	120	5	9.12	175.52
	Including		120.97	2.03	13.99	265.16
CO-240-D	138	9950	87	4.14	1.31	86.38
	and		96	1.7	1.47	353.12
	and		100.3	3.12	1.74	126.56
CO-241A-D	159	10050	131	1.6	1.54	58.74
CO-242A-D	207	10075	186	2	7.47	231.72
CO-243A-D	90	9800	86	3	1.41	3.02
CO-246-D	162	9825	145	10	1.68	9.68
CO-247-D	105	9825	80	10	4.20	220.33
	Including		82.5	3.5	7.14	491.68
CO-249-D	99	10275	57	6	0.95	4.41
CO-250-D	210	9825	195	5.85	3.29	36.39
	Including		197	0.9	15.40	224.79
CO-252-D	219	10350	195	10	2.96	5.98
	Including			1	12.20	7.64
CO-254-D	300	9800	255	8	1.87	46.63
CO-257-D	195	10275	159	14	1.76	37.89
CO-258-D	216	10300	197.4	3.6	3.62	69.38
CO-259-D	330	9890	286	6.1	1.04	6.24
CO-260-D	210	10250	180.5	4	1.59	111.00
CO-262-D	231	10250	185.63	7.37	1.42	33.84
CO-263-D	285	9830	240	9	3.01	357.46
CO-264-D	285	10175	256	1	2.83	112.27
CO-265-D	318	10200	272	10	1.67	64.77
CO-266-D	390	10025	301	14.14	1.25	11.04
	and		319	22	3.42	37.83
	including		326.5	1.2	17.11	65.97
	and		343	11	1.61	7.86
	including		353	1	8.12	86.51
CO-267-D	261	10225	244.72	9.28	7.15	206.06

Table 10-1 List of Significant Drill Hole Intersections, Cap Oeste Deposit						
Hole No.	**Depth (m)**	**Section**	**From (m)**	**Interval (m)**	**Gold Grade (g/t Au)**	**Silver Grade (g/t Ag)**
	Including		248.5	4.5	13.18	245.56
CO-268-D	360	10200	297	17	1.76	11.93
	and		341	3	1.49	0.91
CO-269-D	318	9925	273	13	3.12	89.74
CO-270-D	408	10095	375	3	1.81	93.57
CO-271-D	345	10200	273.5	5.5	1.97	30.29
	and		283	17.72	1.05	5.23
	and		310	5	2.92	5.51
CO-272B-D	291	9875	256.85	4.15	1.56	42.77
CO-273-D	321	10175	264	14	2.55	43.09
CO-274A-D	354	9855	302.5	15.58	1.89	16.93
	and		322	10.2	3.15	123.17
	including		326.85	1.05	14.30	699.41
CO-276-D	249	10325	212.75	1.9	36.28	390.31
	including		212.75	0.95	71.20	768.31
CO-278-D	396	10000	327	8	1.73	9.05
	and		357	7	3.42	190.13
	including		359	1	18.27	1330.92
CO-279-D	294	10450	280	1	1.54	9.92
CO-281-D	366	9935	318	20	3.92	48.13
	including		326.42	3.58	9.39	153.67
CO-283-D	363	10250	302.05	1.95	5.93	317.39
CO-284-D	374.5	10225	234	16	5.91	48.70
CO-285-D	including	10275	235	2	16.67	174.48
	including		244	4	11.98	45.74
	and		295	62	3.88	87.72
	including		343	14	8.48	255.20
	393		349.2	19.2	34.29	255.11
CO-286A-D	including	10250	357.34	0.96	434.38	2362.62
	including		357.34	8.91	70.79	517.79
	384.4		384.4	0.8	1.19	25.17
CO-287-D	414	10050	383.55	4.75	1.89	45.90
CO-288-D	393	9950	245.5	4.7	1.88	51.04
CO-289-D	and	9878	324	11	2.50	74.44
	369		207.25	5.75	4.35	106.73
CO-290-D	and	9950	266.6	8.4	4.25	49.79
	including		266.6	0.7	37.19	249.94
	441		250.5	4.5	1.86	77.30
CO-291-D	414	9900	323.4	8.85	3.92	137.17
CO-292-D	including	9900	323.4	0.6	15.70	676.88
	465		378	1	8.24	22.25
CO-296B-D	264	9750	212.5	2.5	48.56	4983.23
CO-297A-D	162	--	127.14	0.90	0.21	4.22

Hole No.	Depth (m)	Section	From (m)	Interval (m)	Gold Grade (g/t Au)	Silver Grade (g/t Ag)
CO-298-D	372	--	314.25	2.25	0.58	4.06
CO-299-D	300	--	259.15	0.50	0.16	6.53
CO-300-D	438	--	351.00	4.00	0.10	1.07

While the limits of oxidation are observed to be highly variable, the base of the oxidized material is broadly coincident with the footwall contact of the Bonanza and Esperanza Faults down to vertical depths of between 70 and 120 m. Generally, the boundary between the zones of complete, partial and no oxidation are sharp (Figure 10-3), with the interval representing the zone of partial oxidation being typically 5 to 10 m wide. The zones of complete and partial oxidation are collectively represented as occurring above the line of oxidation on the sections provided in this report.

Figure 10-3
Example of Transitional Oxidized Zone with Transition to the Hypogene Zone, DDH CO-139-D

10.6 Drill Sample Recovery

10.6.1 Diamond Core Recovery

A summary analysis of the recoveries achieved in the different geological zones and mineralization types, is shown in Table 10-2. Based on results from the 18,373 diamond core intervals drilled throughout the Cap Oeste Project area, overall diamond core recoveries averaged 98.15%.

Table 10-2
Diamond Drilling Core Recovery Statistics

Geological Zone	Recovery DD Holes (%)
Oxide	97.56
Partial Oxide	98.22
Non Oxide	98.98
Mineralization Type	
Crackle Bx (Zone 1)	97.99
Hydrothermal Bx (Zone 2)	97.17
Disseminated and Veinlets (Zone 3)	98.90

It can be seen that good recoveries were achieved throughout Zone 1 and Zone 3 with average recoveries of 97.99 and 98.90 %, respectively. A slight loss of core (average recovery 97.17%) occurred throughout Zones 2a and 2b, which is likely a consequence of the commonly clay rich fault gouge and fractured rock. Based on this tendency, during the January to present drilling campaigns, the use of HQ triple tube diamond drilling through the main zone of interest was implemented.

Generally good recoveries were achieved for non-oxide and partially oxidized mineralized zones, averaging 98.98 and 98.22% respectively. Slight core loss (average recovery 97.56%) occurred throughout the oxide zone, likely a product of the friable and clay-rich nature of mineralization.

10.6.2 Reverse Circulation Sample Recovery

The average recoveries for the RC drilling sample intervals were calculated for differing drilling conditions (wet/dry) and geological parameters including degree of oxidation and mineralized zones, as shown in Table 12-2. 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 11.4. In the case of wet RC samples, the wet bulk sample residues (i.e. after splitting) were left to dry prior to weighing to which the recorded weight of the split laboratory sample was subsequently added to calculate recoveries.

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

- Oxide: 3.1417 (pi) x 0.066 sq (radius meters squared) x 2.07 (density) = 28.3 kg
- Sulfide: 3.1417 (pi) x 0.066 sq (radius meters squared) x 2.11 (density) = 28.9 kg

The RC drilling recoveries calculated for various geological intervals are shown in Table 10-3.

Table 10-3
RC Recovery by Oxidation Zone and Mineralization Type

Geological Interval	Recovery RC Holes (%)
Oxide Zone	89
Non oxide	98.5
Mineralization Type	
Crackle Bx (Zone 1)	87.6
Hydrothermal Bx (Zone 2)	89
Disseminated (Zone 3)	95

Drilling throughout the oxide zone, yielded good average recoveries throughout which relatively small losses typically occurred preferentially throughout the first 15 to 20 m where supergene clay alteration is strongest and the presence of open space fractures is greatest.

Samples of wet RC drill cuttings, which were limited to the deeper holes from the initial campaign (CO-001-R to CO-010-R), generally reported significantly lower recoveries averaging 49%. These results led directly to the policy of limiting future RC drilling to the interval above the water table subsequent to the first drilling campaign. In addition, twin diamond drill holes were completed adjacent to the initial RC holes where mineralization was intersected below the water table to examine for any significant bias that drilling beneath the water table may have generated.

With respect to the sample recoveries as a function of mineralization type, overall good recovery of 87.6% was achieved within the Zone 1 type mineralization, albeit lower than that achieved for Zone 2 and 3 which reported 89% and 95%, respectively.

Given that Zone 1 type mineralization was predominantly tested in the oxide zone, the lower recoveries are likely a product of the higher propensity for minor loss of clay fines in open space fracture and in permeable portions of the host lithologies. Similarly, recoveries throughout Zone 2 type mineralization were likely affected by the clay rich fault gouge and highly fractured rock conditions typical of this zone.

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

The overall form of the mineralized envelope of the Main Shoot at Cap Oeste 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 towards the northeast), generally intersected mineralization at relatively high acute angles of 60º to 85º 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% of intersected widths. In a rare number of circumstances mineralization was intersected at a lower acute angle of 55 º which equates to approximately 70 to 85% of the intersected widths.

The overall planar geometry of the mineralized section of the Esperanza Fault zone which dips steeply (75-85º) to the northeast was partially tested by stepback holes designed principally to test the Bonanza Fault at depths greater than 150-200m RL. On average these holes intersected this structure at 15 to 25º to the core axis for which the true width equates to approximately 20 to 25 % of the intersected widths.

11.0 SAMPLE PREPARATION, ANALYSES AND SECURITY

11.1 Sampling

11.1.1 General Considerations

Drilling and sampling were not actively ongoing at the time of the most recent CAM visit to the project in August, 2011. As a consequence, the drilling, sampling, and sample preparation procedures are drawn from observations on previous site visits, augmented by review of sampling protocols as provided by PGSA.

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 splitting of the diamond core, and the splitting of the RC samples (as described previously in Section 10).

11.1.2 Trench Sampling

Trench locations 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-cm by five-cm slots were mechanically dry sawn, cleaned, and sampled. Trench sampling and logging were carried out under the supervision of PGSA geologists; sample intervals were generally marked using a measuring tape following geological criteria (e.g. zones of similar mineralogical/geological features). Sampling of the trenches comprised chipping between the two sawn slots with hammer and chisel to the limits of marked sample intervals and placing the broken material in plastic sample bags. Each sample bag is tagged and staple sealed and subsequently transported back to the base camp where each sample weighed and recorded for final laboratory dispatch. Final surveying of the trenches position was completed by a qualified surveyor using a differential GPS.

11.1.3 Reverse Circulation Sampling

PGSA field technicians processed each one meter sample as follows:

- 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.
- Samples were weighed on-site, and the sample weight and type (e.g. dry, moist, wet) were recorded. Samples were weighed at various times during drilling for quality control.
- Riffle splitting was used 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).

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.

11.1.4 Diamond Drill Sampling.

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

- The core barrel was retrieved following each 'run' via wire line, after which the diamond core was immediately slid out from the core barrel and placed in a core cradle. For diamond drilling conducted during 2009 and later, the use of a core barrel sleeve tube (HQ3) was implemented, the core was 'pumped' out hydraulically.
- During this process care was taken by the contractor and PGSA field technician to ensure that core was maintained intact and in the correct order within the cradle.
- Core was washed and subsequently orientated in order to reconstruct the core in its predrilled 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 meter 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 by the PGSA field technician on the sum total interval of individual core pieces that measure over 10cm in the core run.
- Core was carefully placed into the numbered wooden core boxes in which meter intervals were marked on core, and core boxes, with wooden meterage blocks inserted in the corresponding position.

In order to standardize sampling methodology and to more easily allow for reconstitution of the drillhole in half-core, a convention was established of utilizing the left hand side of each cut core portion for subsequent geochemical analysis, with the right hand piece retained as reference core. 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.

11.1.5 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 the 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.

11.2 Analysis

11.2.1 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 all the 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. ACME is also ISO 9011-certified.

11.2.2 Methods and Procedures

The core drill samples underwent sample preparation according to Alex Stewart's procedure P-5:

- Reception of samples in Mendonza, check of number identification
- Weigh sample
- Dry sample at 80 – 90ºC;
- Crush all the sample in Jaw crusher (primary and secondary crusher) to 80% - 10 mesh;
- Split the sample in Riffle splitter to obtain 1.2kg;
- Grind 1.2 kg sample to obtain 85% at 200 mesh

11.2.3 Sample Analysis

Gold was analyzed by fire assay and silver by aqua regia digestion with AA finish, according to Alex Stewart's procedures Au4-50 and Ag4A-50. Inductively-coupled plasma (ICP) analyses for a suite of multi-elements (39 elements, including base metals and silver) were performed by procedure ICP-MA-39.. Silver over limits 200 g/t were analysed by fire assay gravimetric finish. Gold above 10 g/t was re-assayed with gravimetric methods.

11.2.2 Screen Fire Assays

A total of 24 coarse rejects (95% less than -10 mesh ASTM) were selected from original, individual high, mid-range and low grade sample intervals as determined by assaying performed by the Alex Stewart laboratory. These were analyzed by Acme Analytical Laboratories via the screen fire assay technique in order to determine the size/distribution character of gold mineralization (Figure 11-1).

This technique is designed to examine whether larger gold particles are present in the coarse fraction of the sample and to enable semi-quantitative analysis on the potential presence and effects of coarse gold on grade reproducibility of relatively small (50 g) sample sizes used during routine analysis.

Sample preparation involves firstly the milling of the coarse reject to 95% less than 200 mesh ASTM (74 µm) after which the undersize is sieved, weighed and split into three subsamples which are each subsequently fire assayed. The oversize is weighed and the entire coarse fraction is subsequently fire assayed, after which those values over 10 g/t Au are determined with a gravimetric finish.

In order to test for the level of repeatability of the high grade gold intervals and potential nugget effects relating to the possible presence of coarse gold, a total of 9 coarse residues from some of the highest grade intervals were re-assayed (Table 11-1). Scatter graphs for the gold and silver results are presented in Figures 11-2 and 11-3, respectively. Results show a high degree of repeatability whereby the respective repeat values returned values within the plus or minus 10 to 15% limits of the original value for both gold and silver.

The combination of the overall good repeatability of both the routine check assay data and high grade re-assay results, and defined presence of gold in the coarse fraction (>74 µm or 200 mesh ASTM) determined by the screen fire analysis, suggests an overall homogenous distribution of fine and coarse gold in the high grade samples that does not negatively influence the level of repeatability achieved by the conventional fire assay technique.

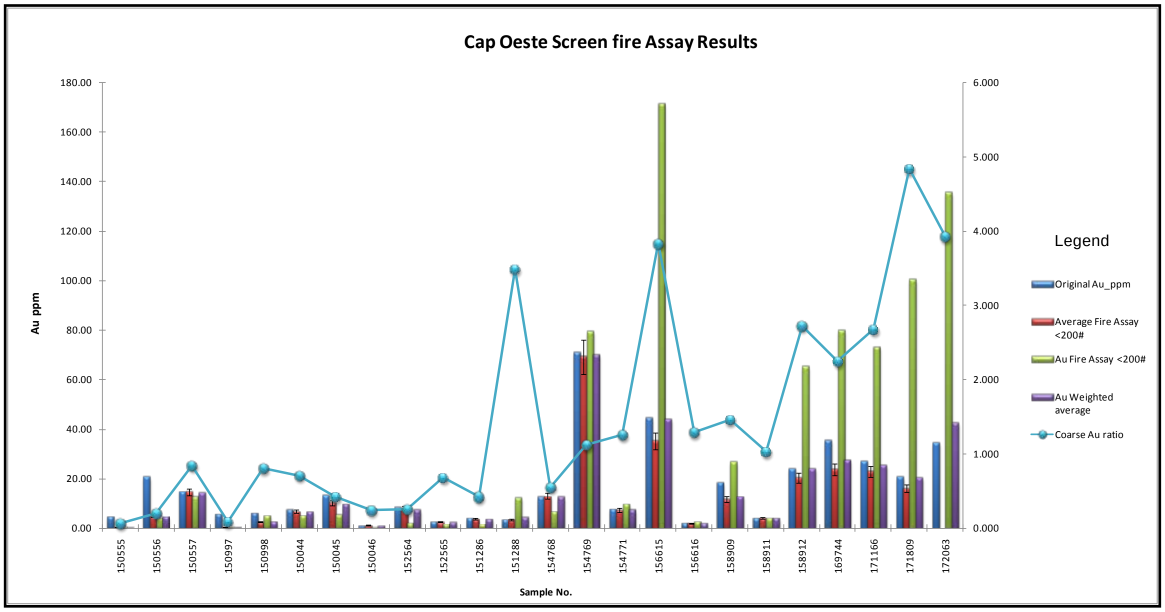


Figure 11-1
Graphical Comparison of Screen Fire Assay Gold Results

11.3 QA/QC - Quality Assurance and Quality Control

CAM reviewed a 57-page report entitled "Cap Oeste Quality Control Report - Geochemical Assays of Patagonia Gold by Gustavo Almeira of PGSA, dated July 2011. QA/QC procedures conducted include the routine incorporation of certified geochemical standards, blanks and RC drilling sample duplicates . In addition, PGSA mandated screen fire-assay tests on 24 samples.

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:

The standards and blanks 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 alternate 10th 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 alternate 10th 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.

11.3.1 Field Duplicates

Field Duplicates –Trenching

Two field duplicates were taken during sawn trench sampling throughout the Cap Oeste deposit area, which reported good repeatability and correlation within plus or minus 10 to 30% of the relative error limits.

Field Duplicates – RC Drilling

From the total of 174 field duplicates analyzed, a good correlation of within +10% variability was received for values between 1.5 to 6 g/t Au and an acceptable correlation within +20% variability was received for values between 0.1 to 1.5 g/t Au, as shown in (Appendix II). Correlation for silver for the field duplicates reported generally within the +10-20% limits and, apart from a single outlier, indicated an overall slight positive bias of the original assay results.

11.3.2 Certified Standards

Certified standards were purchased from Geostats pty ltd, based at Western Australia. The standards had a range of round-robin certified grades between 0.03 and 47.24 ppm Au and 0.5 to 462.7 g/t Ag, thus bracketing the range of expected values.

A total of 1107 individual standards, with a range of certified gold grades between 0.03 and 47.24 g/t Au, 790 blanks and 174 duplicates were submitted with half-core drill samples for quality control throughout the routine drill sample assay process.

The analytical results for each individual standard were plotted on control charts in which the upper and lower limits were defined by plus or minus 2 and plus or minus 3 standard deviations from the respective certified value, in addition to the plus or minus 10% relative variance from the assigned standard value.

The control charts showed that all gold standards performed within the accepted 3 standard deviation limits of the recommended gold value, with the exception of a total of 35 samples which returned values outside these limits. For each of these failed standard samples, five of the adjacent drilling samples within the batch, relative to the standard, were reanalyzed. As part of these rechecks, a total of 289 drill sample interval pulps were re-analyzed for gold, together with a total of 19 standards.

The results for the original and recheck drill sample interval pulps show a good correlation of within +10% (Appendix II) and all the standards that were included with the reanalysis returned values within the +10% variation limits of the certified standard values. As a result, it is considered that the original standards which returned a large variation from the expected values were either erroneously submitted and/or recorded, or that preparation and handling of the standards introduced a degree of error greater than plus or minus 3 standard deviations.

For the quality control of silver results a total of 142 silver standards with certified values between 52 and 1,419.6 g/t Ag were submitted with the drill interval samples during the drill campaigns. Assay values received from the laboratory show good correlation within plus or minus 10% of the certified values.

The Almeira report referred to above reports the analytical results for each of 27 Au standards, some of which were assayed on more than 60 occasions. Examination by CAM of the results indicates that the precision and accuracy of the standards were both acceptable. There a few notable blunders (sample-switching in the lab ?) which is to be expected in an operational setting. Possible laboratory bias (non-accuracy) in excess of 5% relative to the standard was noted in 4 of the Au standards with +30 assays. There was no appreciable temporal drift of results for these standards, and the biases were both positive and negative, suggesting the possibility that these standards were off-key.

A typical control chart is shown in Figure 11-2.

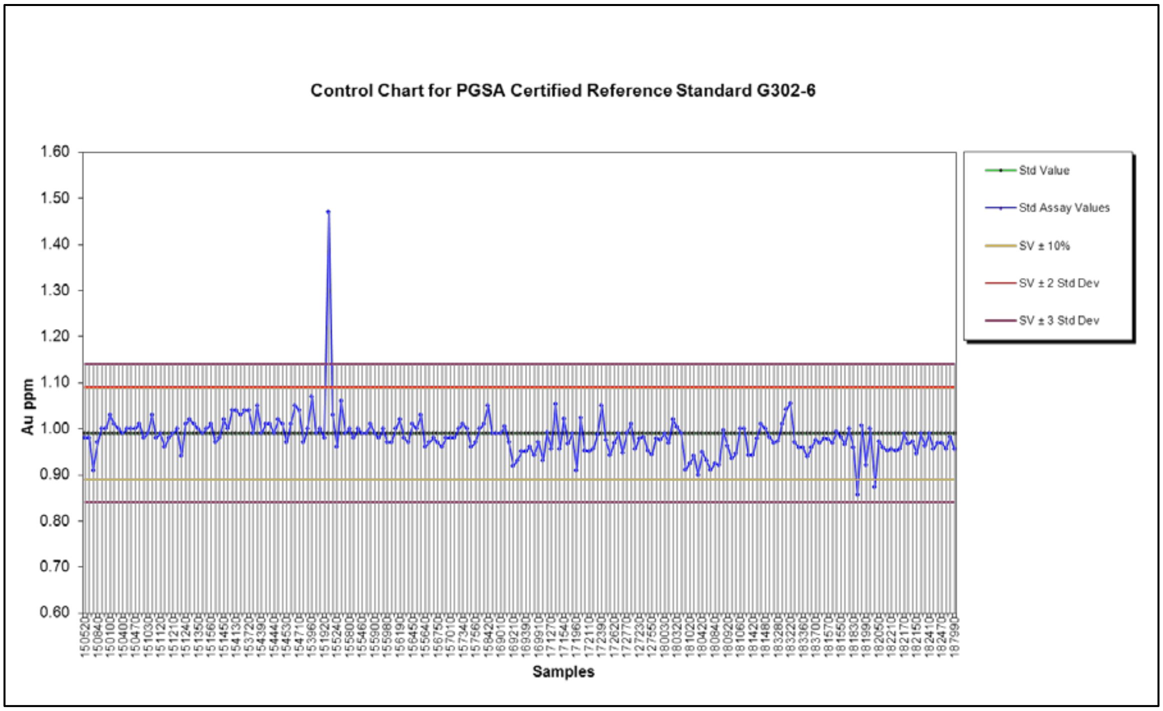


Figure 11-2
Control chart for gold standard G302-6

11.3.3 Blank Samples

Blanks standards were obtained from Geostats Perth. PGSA submitted 579 blank samples into the assay stream. Of these, only two (fewer than 1/2 of one percent of the total) resulted in high values, probably resulting from sample labeling blunders in the laboratory. Eight others (fewer than 1.5% of the total) yielded results of two to four times the limit of detection for Au. CAM considers that these results are acceptable, and that intervention is not necessary.

11.3.4 Check Assays

A total of 17 batches of check assay sampling were conducted during 2008, 2009 and 2011 for holes CO-001-R to CO-136-DR and CO-166-D, which overall comprises approximately 3.5% of the total drill sample interval population. These check samples consisted of sample pulps (278 samples, 85% less than 80 µm or -200 mesh ASTM) and coarse rejects (305 samples, greater than 85% less than 1.7 mm or -10 mesh ASTM). These check samples were taken predominantly from where significant gold-silver drill intervals were reported, which were collectively submitted with a total of 64 laboratory-certified standards.

These samples were resubmitted to both:

- The original laboratory (i.e. Alex Stewart Assayers S.A.) - comprising of 83 pulps and 237 coarse rejects plus 35 standards.
- A certified check laboratory (i.e. Acme Laboratories) - comprising 198 pulps and 68 coarse rejects plus 29 standards.

Additionally, check assaying for silver by the two laboratories was completed, however certified standards containing significant concentrations of silver were not included with these samples. Statistical results for the check assay data were generated in Excel spreadsheets. 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% 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 of the coarse rejects between the two laboratories, values reported by Alex Stewart Assayers S.A. indicated a minor negative bias compared to that of Acme Analytical Laboratories of approximately 10%, highlighted particularly in the range of values between 3 and 8 g/t Au. For silver, the same tendency was reported albeit to a lesser degree (2-5%), apart from a high isolated outlier sample which reported a 12% negative bias relative to the Acme Analytical Laboratories result.

From the internal comparison of check assays for Alex Stewart Assayers S.A., the check results from the coarse rejects suggest a slight negative bias (<5%) compared to those of the original results. The correlation coefficients indicate an excellent overall correlation for all of the gold and silver values for both the coarse rejects and pulps with the independent laboratory (Acme Analytical Laboratories), as well as the internal checks of Alex Stewart Assayers S.A. Results of the check assays are displayed in Table 11-1.

Table 11-1
Comparison of Gold and Silver Re-Assay Results

Hole	Original Results			Check Results using Coarse Rejects		
	Sample	Gold (g/t)	Silver (g/t)	Check Sample	Gold (g/t)	Silver (g/t)
CO-016-D	151404	94.28	3410	170777	89.91	3,188.65
CO-054-DR	156613	218.83	4273	170778	250.17	4,253.18
CO-054-DR	156614	61.04	552	170779	61.07	543.88
CO-080-DR	158902	41.29	24.73	170781	34.39	21.42
CO-105-DR	169213	98.44	105	170782	103.23	105
CO-108-D	169316	31.29	665	170783	33.37	606.44
CO-119-DR	169736	34.42	499.02	170791	37.01	487.21
CO-119-DR	169737	46.22	300.48	170792	46.57	279.81
CO-125-DR	171027	32.73	6,649.54	170793	35.81	6,502.77

Figures 11-3 and 11-4 display scatter plots of the gold and re-assay results.

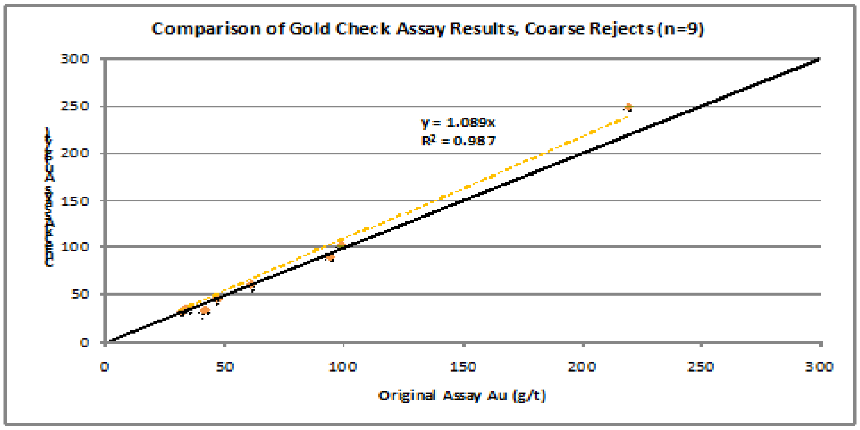


Figure 11-3
Scatter Plot of Gold Re-Assay Results

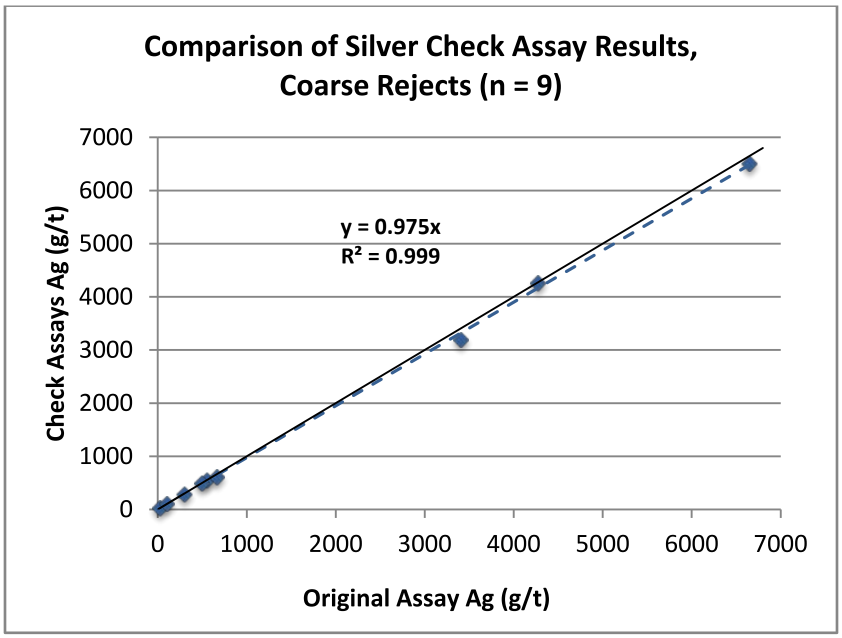


Figure 11-4
Scatter Plot of Silver Re-Assay Results

11.3.5 Suitability of QA/QC

CAM are of the opinion that PGSA's sample preparation, security, and analytical procedures yielded samples of sufficient reliability to be appropriate for use in Resource estimation.

11.4 Specific Gravity

The weight-to-volume ratio in rocks is conveniently referred to as "bulk density", as it refers to that ratio as measured when taking into account the porosity, oxidation, moisture content, and other properties of a rock containing a variety of minerals and voids. This ratio is commonly referred to as "specific gravity", whereas that term may also be reserved for the weight-to-volume ratio of specific mineral species or other homogeneous phases.

11.4.1 Methodology

Measurements of specific gravity were performed on site by PGSA prior to 2011 on 1,691 samples of whole or split core. The global average of all the specific gravity values based on all rock, oxidation and mineralization types for the 1,691 samples was 2.14 t/m^3 .which was rather low considering the rock types involved. I t was later discovered that systematic procedural errors had rendered these readings suspect, so PGSA instituted a verification program, and set aside the early results.

Measurements of specific gravity (SG) were thenceforth performed by Alex Stewart Labs on 91 individual, HQ half-core pieces. Based on the 60,117.38 m of available core this sample set thus represents approximately 0.03 percent of the total sample population, which were taken predominantly from zones of mineralization.

The methodology used by Alex Stewart was by water displacement, as follow:

a) The sample is oven-dried and then allowed to cool
b) The sample is weighed = P1
c) The sample is immersed in liquid paraffin and weighed = P2
d) The sample is hung in an adequate support and then immersed in a container filled with water.
e) The sample is placed in a support and then weighed = P3

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

$$\mathbf{SG} = \frac{\mathbf{P1}}{\mathbf{P2\text{-}P3 - [(P2\text{-}P1)/0.86^{*}]}}$$

11.4.2 *Specific Gravity Results*

The range of SG values calculated from samples representing the spectrum of lithologies, mineralization types and oxidization states in the 91 samples is shown in Table 11-2.

Table 11-2
Summary of Alex Stewart Specific Gravity Results

Zone	Oxide State	S.G. Mean	S.G Maximum	S.G Minimum	SG Std Dev.	No. Samples
All samples	NA	2.5	3	2	0.22	91
Mineralization Type						
Stockwork-Crackle Breccia (Zone 1)	Oxide	2.43	2.5	2.4	0.05	3
	Non oxide	2.5	2.8	2.0	0.2	18
Fault-Hydrothermal Breccia (Zones 2a, 2b)	Oxide	2.47	2.9	2.2	0.13	9
	Non oxide	2.52	3	2.3	0.15	50
Footwall stringer (Zone 3)	Oxide	2.6				1
	Non oxide	2.47	2.7	2.2	0.12	10

The global average of all the specific gravity values based on all rock, oxidation and mineralization types for the 91 samples is 2.50. The average specific gravity for the oxidized and un-oxidized portion of the three mineralization types is 2.47 and 2.51 respectively.

With respect to the zones of mineralization, the highest average S.G values relate to Zone 2 mineralization, with an overall average of 2.47 in the oxide and 2.52 in non-oxide respectively. The peak specific gravity value (3.00) in Zone 2 relates to weakly silicified breccia gouge with abundant sulfide mineral species. The higher specific gravity compared to that of Zone 1 and Zone 3 is considered to be largely due to the comparatively enhanced levels of silica- sulfide / Fe Oxide in that of Zone 2.
The approximate 5% increase in specific gravity between the non-oxide and oxide portions of each respective mineralization types is interpreted to be due to the enhanced levels (5 to 10 percent) of sulfide species (in the non-oxide zone) compared to the more leached and supergene, clay rich, oxidized zone.

PGSA are continuing the improved specific-gravity measurement program, in order to increase the precision and the accuracy of the averaged data by rock type. These measurements should be calibrated with standard density samples (e.g. sealed aluminum tubes); covering the range of densities encountered in the deposit (SG's about 1.9 to 2.7).

11.4.3 Assessment of Specific Gravity Results

CAM believes that the results in Table 11-2 above are credible, and are suitable for use in resource estimation. However additional readings are needed to confirm the values for each of the several mineralized material types present at Cap Oeste.

12.0 DATA VERIFICATION

PGSA did the resource estimate using the GEMCOM software system. CAM always checks resource estimates with a different software system, and used MicroModel for this check. The first step in this check is converting data from the various sources to MicroModel format. Data were provided to CAM as Excel spreadsheets and the Microsoft Access database. Although CAM found some minor errors in merging the data from the various sources, only one significant error (an incorrect sign in one of the downhole survey dips). This error was reported to PGSA but apparently it does not exist in the GEMCOM database.

A summary of the exploration database received from PGSA is given in Table 12-1

Table 12-1
Cap Oeste Drilling Statistics from Assay Database

Item	Number	Length (m)
Holes	337	61933.3
Holes with non-collar downhole surveys	314	60490.5
Non-collar survey records	2876	59276.6
Downhole surveys up	0	0.0
Downhole surveys down	3213	59276.6
Assay intervals defined (Au)	4605	4972.9
Actually assayed intervals (Au)	4604	4971.9
Assay intervals defined (Ag)	4605	4972.9
Actually assayed intervals (Ag)	4604	4971.9

The only thing significant in the above table of drilling assay statistics is the small proportion of assay and assayed intervals relative to the total amount of drilling. This ratio (slightly less than 10%) is typical of drilling in a structurally-controlled deposit, where values are mainly restricted to narrow intervals.

CAM uses automated data processing procedures as much as possible in constructing and auditing geologic databases to assure consistency and minimize errors and costs. 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 include:

- Misspellings.
- Confusion of 0 (zero) and O or o.
- Inconsistent use of upper and lower case.

- Inconsistent usage or space _ and -.
- Trailing, leading or internal blanks. (CAM routinely changes all blanks to positively identify this problem).
- Inconsistent use of leading zeros in hole IDs.
- Inconsistent analytical units (e.g. PPM, PPB, opt, %, etc).
- Inconsistent coordinate systems and units (e.g. NAD27 and state plane or mine grid): ft and m.

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 of 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.

The CAM check procedure generates a number of false positives (possible issue which are actually correct). In general if the number of items flagged is less than 2% of the total records the database is acceptable.

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 different personnel 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 a procedure for mathematical 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
- Calculate approximate difference in XYZ location due to differences in hole desurvey algorithms
- Check for overlapping assays
- Check for 0 length assays

- Check for long assay intervals
- 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 thickness in toto and by mineral zone
- Bias testing between drilling campaigns and drilling type as appropriate

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.

Two errors (sign of dip noted above and a collar bust on a short hole) and a few anomalies were noted, and reported to Cap Oeste Project personnel, 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 substantive 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. CAM also believes that the data have been properly vetted and are suitable for use in preparing a resource model for feasibility and financial decisions.

13.0 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Gravity Concentration Testing

For gravity-concentration tests, of 18 samples weighing between 10 and 13 kg were submitted to Acme Analytical Laboratories which subsequently sub-contracted SGS Minerals to conduct the work.

The samples were first milled in a roll crusher to 100% less than 10 mesh ASTM (approximately 2 mm). For each sample, two representative 500-g crushed sub-samples were obtained using a rotary splitter. One was retained as a reference sample, and the other was further pulverized to 100% passing 150 mesh ASTM (105 µm), and analyzed by fire assay to determine the gold head grade. Each sample that returned results over 5 g/t Au was re-analyzed and the gold content was determined by means of a gravimetric finish. The head grades of the samples were found to range from 1.1 g/t to 21.2 g/t Au with an average of 4.6 g/t Au.

The calculated head gold grade of the gravity separated samples was determined using the weights of each product and the respective gold analyses, which were then compared with the analyzed head grade from the original sample split.

Based on the results, it appears that under the above test conditions, the concentration of gold by gravity separation is relatively ineffective. Results suggest that this method is able to recover between 10 to 20% of the contained gold of a given sample.

13.2 Bottle Roll Cyanidation Tests

Bottle roll cyanide leach tests have been conducted on 4 batches during 2008-2010 by three laboratories:

- Bottle roll testing of the Batches 1 and 2 employed the following parameters:
  - 45 element ICP scan after multi acid digestion.
  - 50-gram Fire assay.
  - Active Cyanide Leach on each 500-gram sample with 1 percent NaCN solution with sampling of the pregnant CN liquor after 6, 12 and 24 hours.
  - Analysis of gold in solid residue after cyanidation by 50-gram fire assay method.
- **Batch 1, July, 2008:** OMAC Laboratories, an affiliate of Alex Stewart Assayers (with an ISO 17025 accreditation) based in Loughrea, County Galway, Ireland. This batch consisted of 15 gold-mineralized samples, which were composited from 97 individual course rejects selected from 7 RC and 12 diamond holes. The samples selected were from the oxidized and partially oxidized portions of fault-hydrothermal breccia hosted gold mineralization from the Main Shoot.

At the time of selection of these samples, the relative importance of silver was not considered. Consequently, the focus of attention was directed towards the gold contents and recoveries.

- **Batch 2, August, 2008:** Alex Stewart Assay and Environmental Laboratories Ltd, Kara-Balta, Kyrgyzstan. This batch comprised 12 samples of both gold and silver mineralization, which were composited from 97 individual coarse rejects selected from 9 mineralized intervals from 7 diamond holes. These composites comprised material from predominantly non-oxidized portions of fault-hydrothermal breccia hosted and disseminated gold mineralization at Cap Oeste from both the Main and B Shoots.
- **Batch 3, July, 2009: SGS Laboratories, Santiago, Chile.** The third batch comprised 4 samples submitted for both gold and silver mineralization, consisting of predominantly non-oxidized fault-hydrothermal breccia and disseminated gold mineralization from the above-mentioned shoots, for which the methodology of the test was similar to that for Batches 1 and 2 except that the leach time was extended to 72 hours
- **Batch 4, October, 2010: SGS Laboratories, Santiago, Chile.** The fourth batch comprised 3 main categories of sample types derived from comprised mainly fault-hydrothermal breccia hosted and disseminated gold mineralization from the Main Shoot which were sourced from 109 individual drilling samples from 15 diamond holes, 75 individual drilling samples from 6 diamond holes and 156 individual drilling samples from 10 metallurgical diamond holes (COM1-COM 10) and 2 RC holes. These samples were selected to yield composite grades similar to what was envisaged as likely material (based on grade and oxidation state) for Heap Leach, Dump Leach and Hypogene extraction and leaching scenarios.

The categories were denominated as:

- Heap - 12 oxide ore composites
- Dump (ROM) - oxide ore composites
- Hypogene- 12 sulfide ore composites

In the case of the ‘Heap’ and ‘Dump’ categories, the principal aim of this test work was to simulate pad leaching and determine the effect of varying grind sizes (75 µm, ¼’’ (6.3mm) and ½’’ (12.5mm)) and leach times, on the bottle roll recoveries of Au and Ag.

The leach tests were initially conducted on both Au and Ag recoveries for the 75 µm fraction for each composite in each category over 1, 6, 10 and 24 hours. Cyanide leach tests on 2 kg samples were then conducted on the 1/4″ and 1/2″size fractions, for the ‘Heap’ and ‘Leach’ composites respectively, over 1, 4, 7, 11 and 14 days (corresponding to 24, 96, 168, 264, 336 hrs).

In the case of the Hypogene category, the principal aim of the test work was to determine the Au and Ag recoveries for the 75 µm fraction over varying bottle roll leach times between 1, 6, 10 and 24 hr.

The individual Heap, Dump and Hypogene category composites were analyzed for Au, Ag, total sulphur, sulfide and trace elements by ICP. As part of the sample preparation for the bottle roll tests for the hypogene samples, representative sub-samples were retained for future Bond Rod Mill Work Index tests.

13.2.1 Batch 1 Test Results

Sample characteristics, trace element geochemistry and bottle roll gold leach results from the first batch are shown in Figure 13-1. In summary, the results showed average recoveries of 96.3, 97 and 97.3 percent, after 6, 12 and 24 hours, respectively. The three highest grade composite samples, between 17.5 to 26.75 g/t Au (average 22.67 g/t Au), returned an average recovery of 98.7, 98.5 and 99% after 6, 12 and 24 hours respectively.

One of the composites (No. 4) that returned relatively lower recoveries (93.3% recovery after 6 hours) consists of oxidized and partially oxidized mineralized material. The other composite sample (No. 6) with a similar mix of materials returned a higher average recovery of 96.3%. During these tests no lime or cyanide consumption concentrations were analyzed.

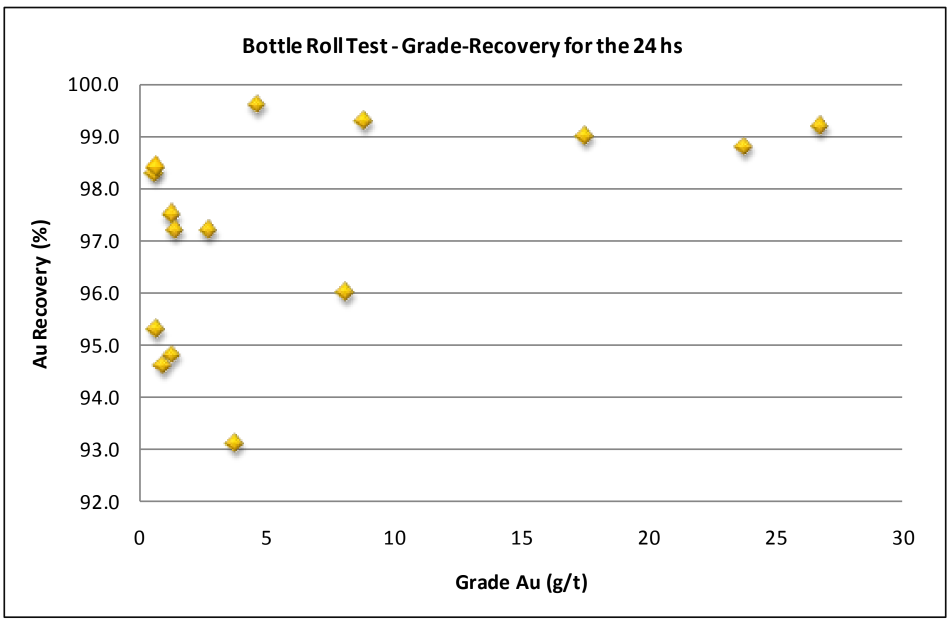


Figure 13-1
Gold Grade vs. Recovery Results for the 24-hr Bottle Roll Tests, Batch 1

13.2.2 Batch 2 Test Results

All but two of the composites tested in this batch were from un-oxidized core. Graphs showing the percentage recoveries of gold by selected grade ranges (between 0 to 18 ppm Au and 0 to 154 ppm Au) are provided in Figures 13-2 and Figures 13-3. Graphs showing the percentage recoveries of silver by selected grade ranges (between 0-100 ppm Ag and 0 to 2,500 ppm Ag) are provided in Figures 13-4 and 13-5.

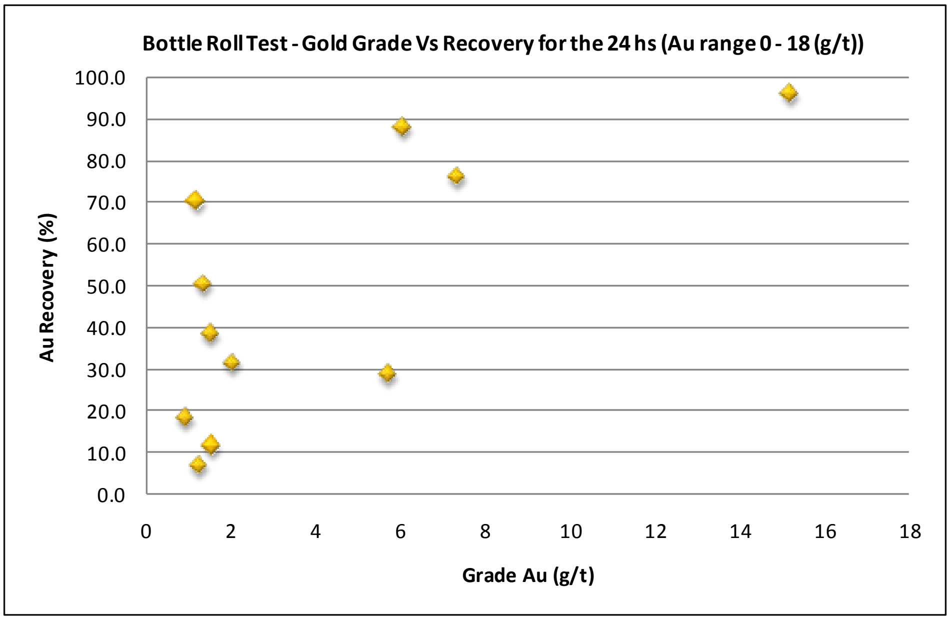


Figure 13-2
Batch 2-Gold Grade vs. Recovery (0 – 18g/t Au)

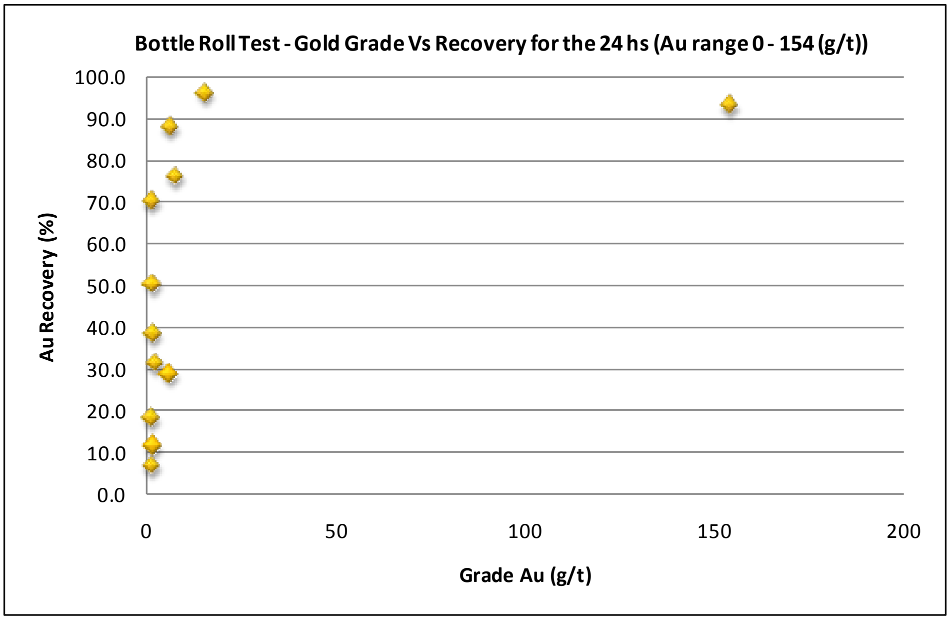


Figure 13-3
Batch 2- Gold Grade vs. Recovery (0 – 154 g/t Au)

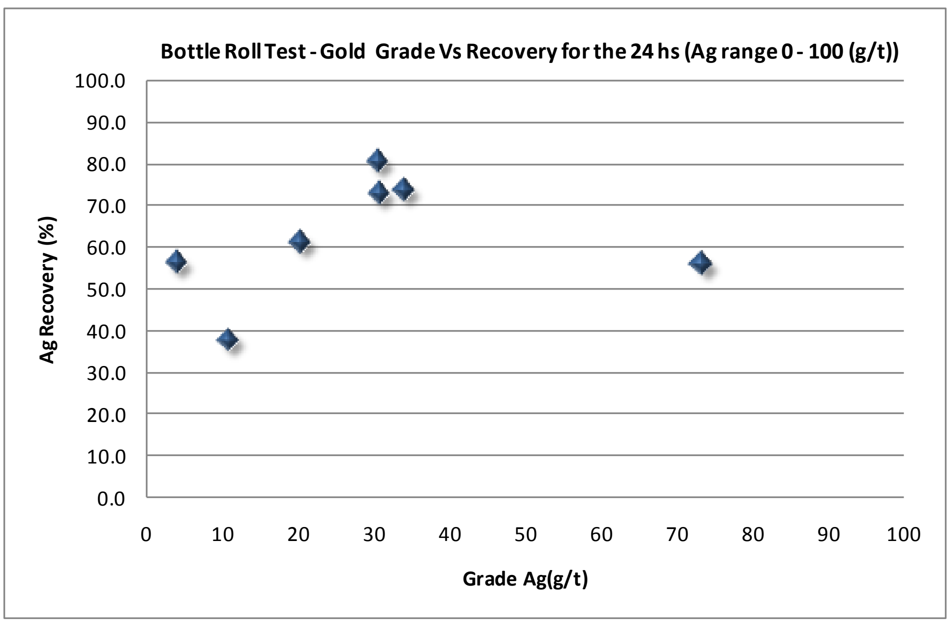


Figure 13-4
Batch 2- Silver Grade vs. Recovery (0 – 100 g/t Ag)

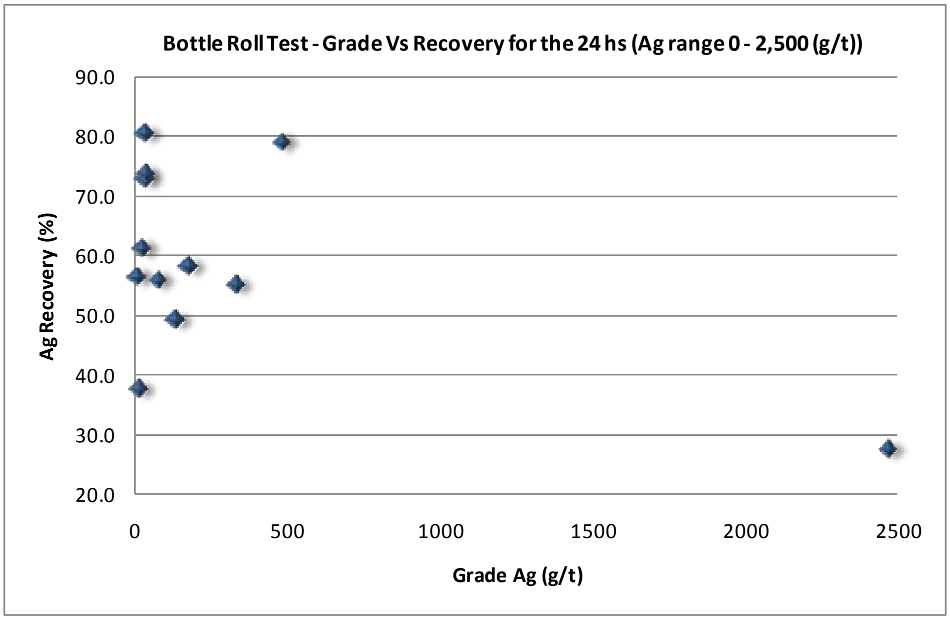


Figure 13-5
Batch 2- Silver Grade vs. Recovery (0 – 2,500 g/t Ag)

The relationship between arsenic concentration and gold and silver recoveries is shown in Figures 13-6 and 13-7, respectively.

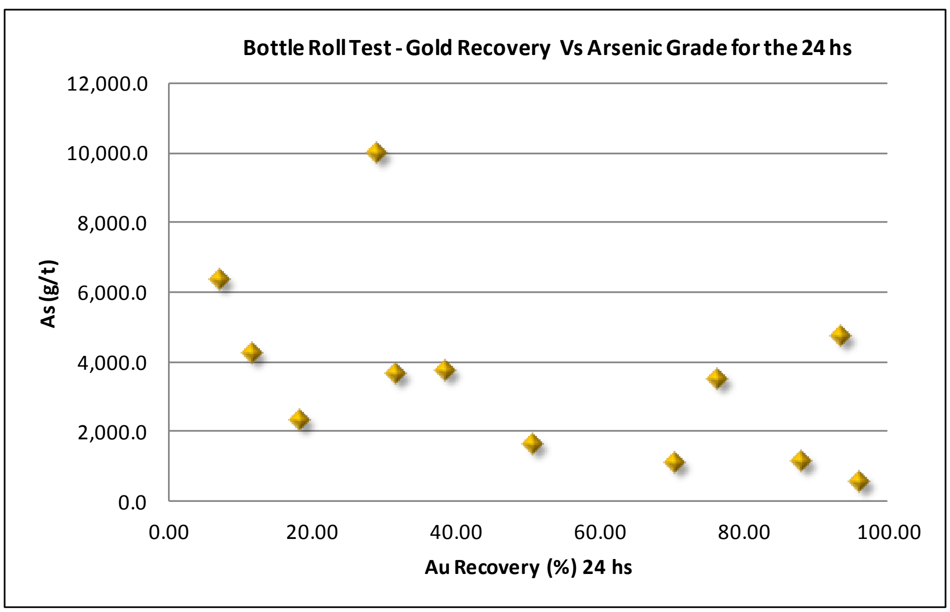


Figure 13-6
Batch 2-Comparison of Arsenic Values vs. Gold Recoveries

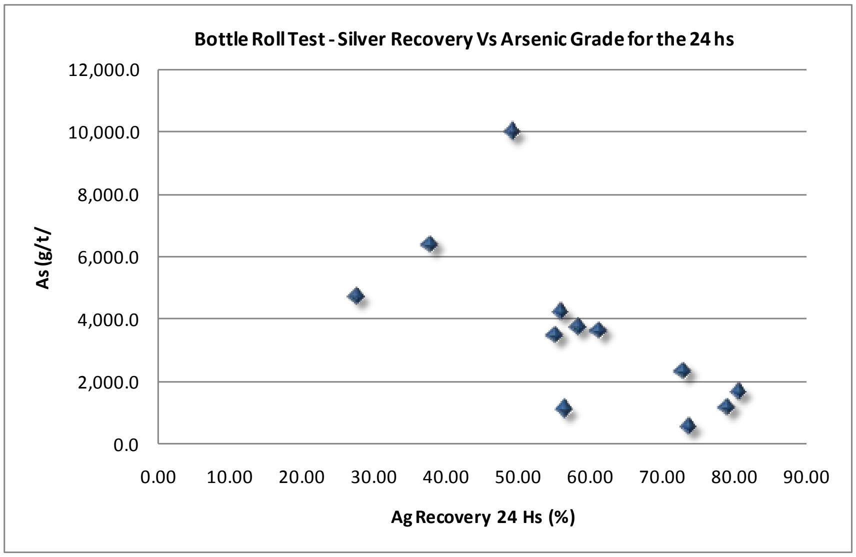


Figure 13-7
Batch 2-Comparison of Arsenic Values vs. Silver Recoveries

13.2.3 Batch 3 Test Results

Results of the gold and silver recoveries versus grade are shown graphically in Figures 13-8 and 13-9 respectively. In summary, the results suggest that the longer leach time of 72 hours consistently achieved an average of approximately 12% and 7% more recovery for Au and Ag respectively, compared to those of 24 or 48 hrs.

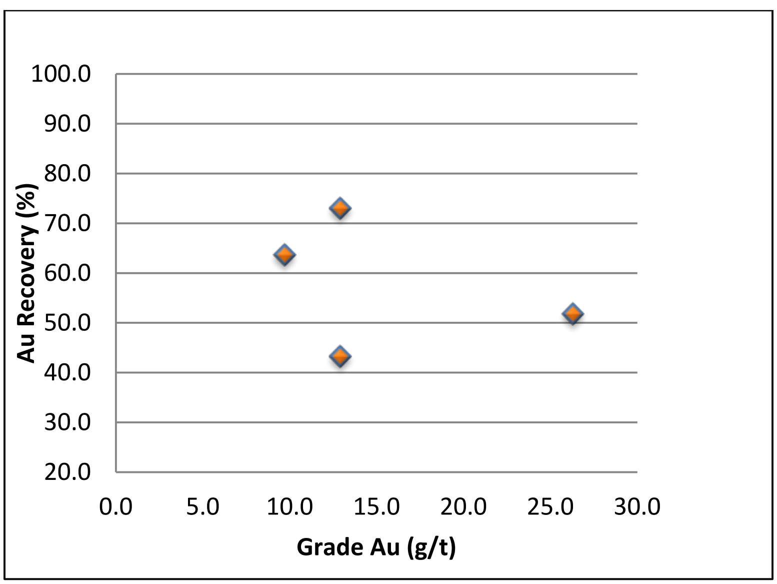


Figure 13-8
Batch 3- Gold Grade vs. Recovery- 72 hr

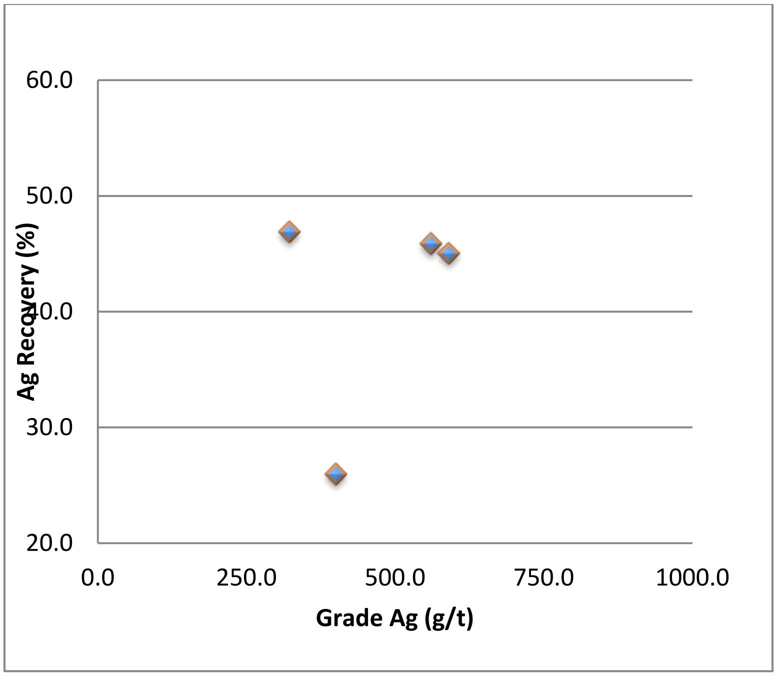


Figure 13-9
Batch 3- Silver Grade vs. Recovery 72 hr

13.2.4 Batch 4 Test Results

<u>Heap Composites</u>

Graphs showing the gold recovery versus leach times for the 75µm (1, 6, 10 & 24 hr) and ¼'' fractions (24, 96, 168, 264 and 336 hr) are provided in Figures 13-10 and 13-11.

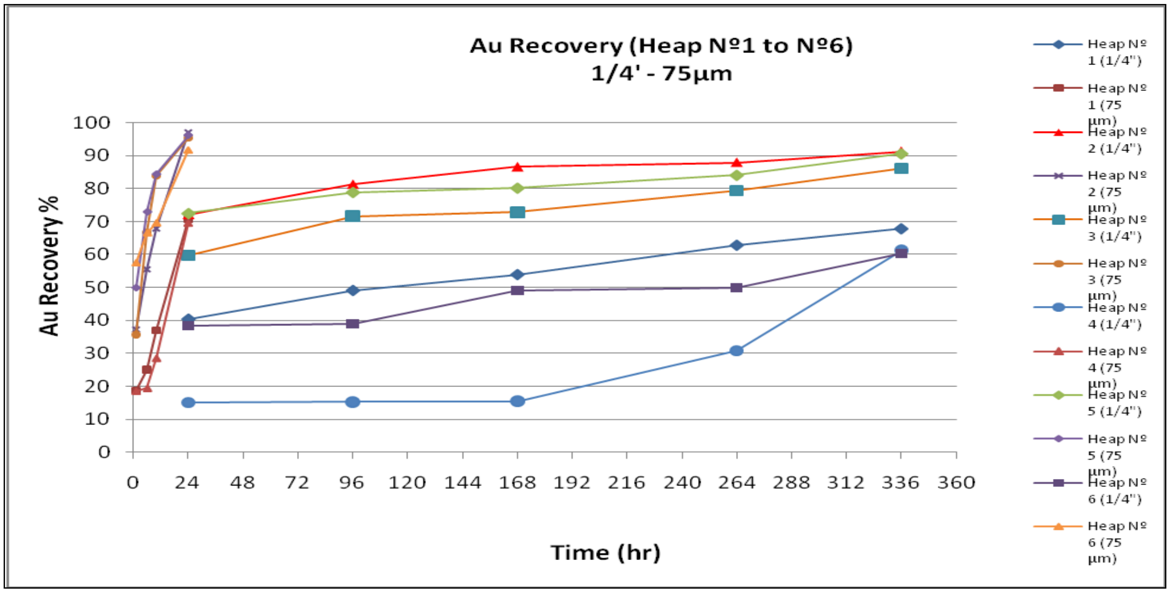


Figure 13-10
Batch 4- Heap Category Composite Bottle Roll Test Results (No. 1- 6) for 1, 6, 10 and 24 hours (75µm and ¼'' fractions) and 24, 96, 168, 264, 336 hr (1/4'' fraction)

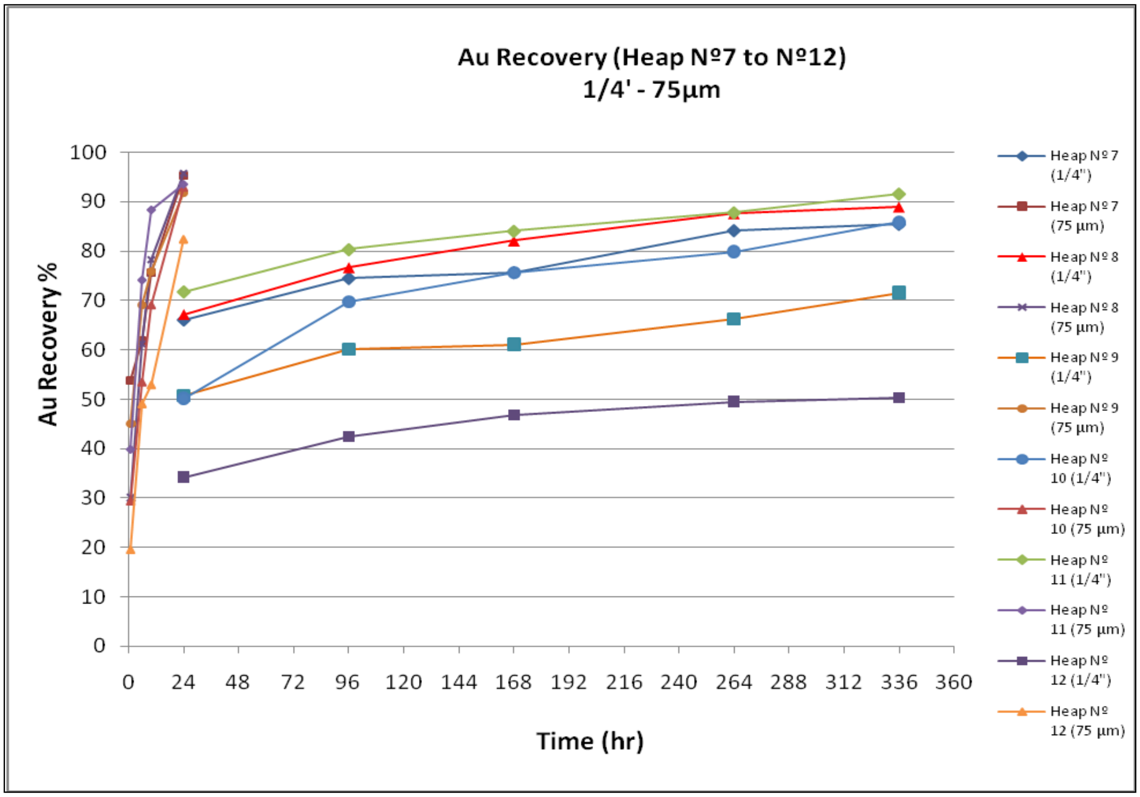


Figure 13-11
Batch 4- Heap Category Composite Bottle Roll Test Results (No. 7- 12)
for 1, 6, 10 and 24 hours (75µm) and 24, 96, 168, 264, 336 hr (1/4'' fraction)

<u>Dump Composites</u>

Graphs showing the gold recovery versus leach times for the 75µm (1, 6, 10 & 24 hr) and 1/2'' fractions (24, 96, 168, 264 and 336 hr) are provided in Figure 13-12.

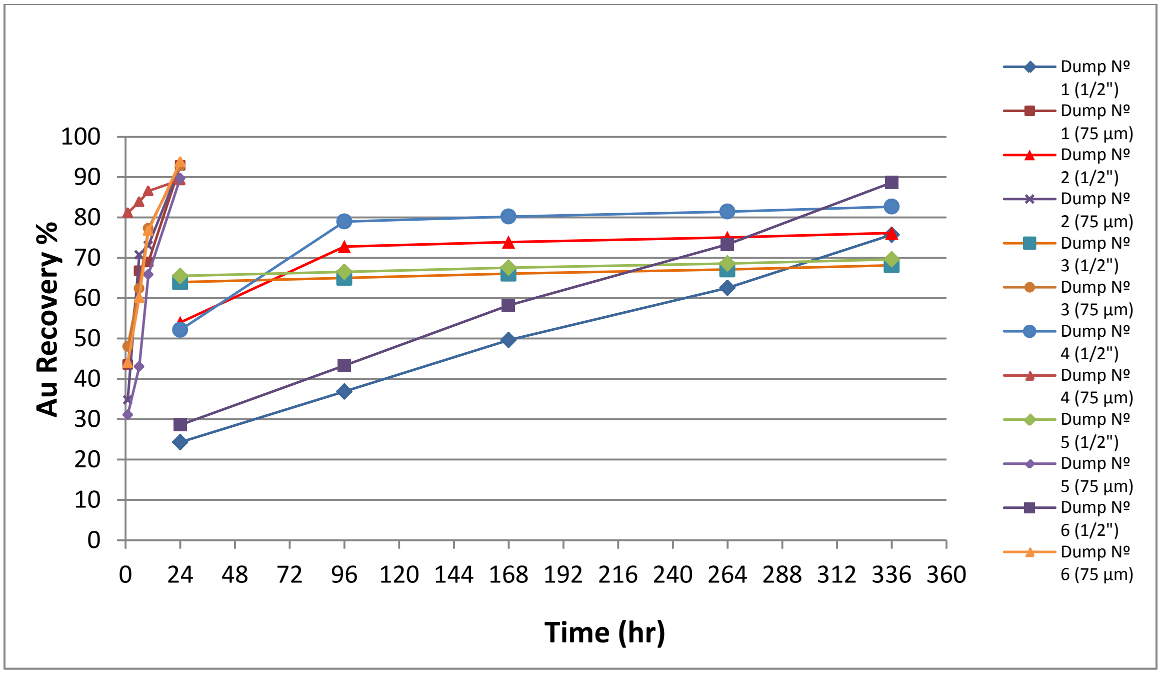


Figure 13-12
Batch 4- Dump Category Composite Bottle Roll Test Results (No. 1- 6)
for 1, 6, 10 and 24 hours (75µm) and 24, 96, 168, 264, 336 hr (1/2'' fraction)

<u>Hypogene Composites</u>

Graphs showing the recoveries for gold and silver versus the respective grades are provided in Figures 13-13 and Figure 13-14 respectively.

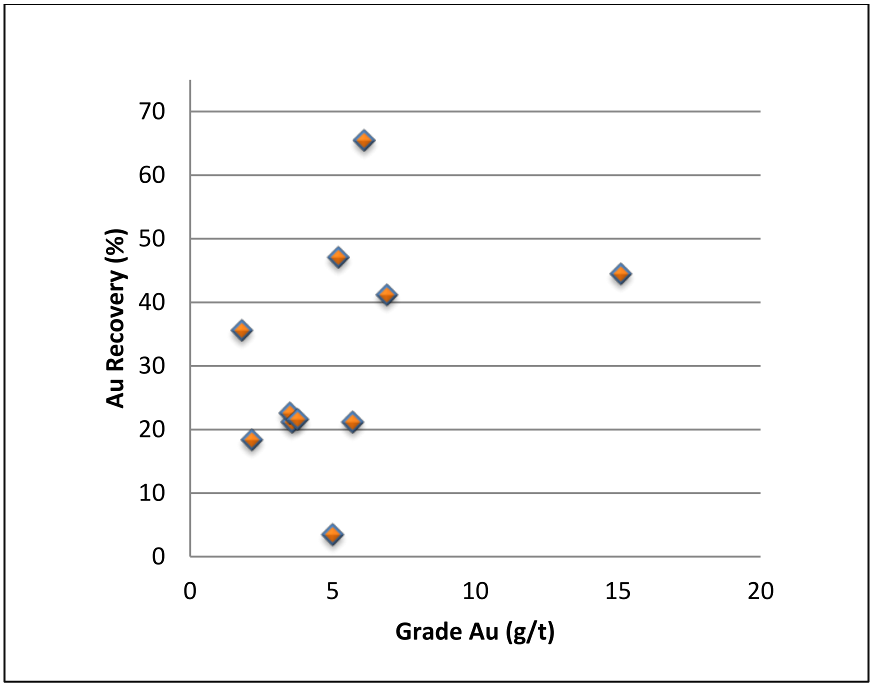


Figure 13-13
Batch 4- Hypogene Composites- Gold Grade vs. Recovery- 24hr-75µm

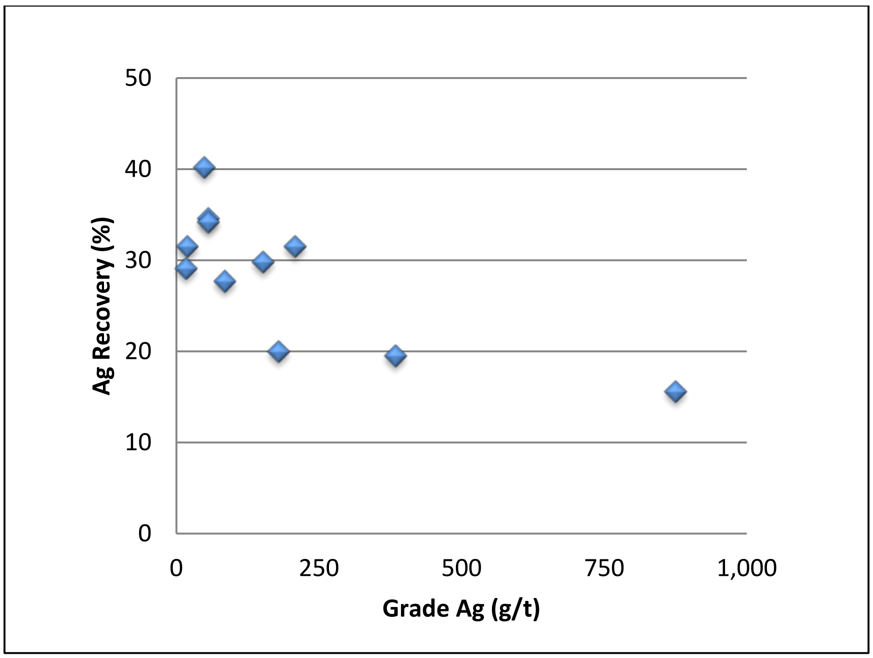


Figure 13-14
Batch 4- Hypogene Composites- Silver Grade vs. Recovery- 24hr-75µm

13.2.4 Discussion of Bottle Roll Results

Batch 1

Tests on oxide gold mineralization from 15 samples in Batch 1 indicate that the gold is highly amenable to cyanide extraction of gold, as an average of 96.3% of gold was leached in the first 6 hours. Partially oxidized mineralization tested in two samples in Batch 1 returned good average recoveries of 95.1%. No analysis for silver recoveries has been conducted to date on any samples from Batch 1.

Batch 2

Fresh, sulfide-hosted gold-silver mineralization tested in 10 samples in Batch 2 returned variable cyanide leach recoveries between 6.9 to 96.1% for gold (averaging 47.7%) and 27.5 to 79.5% for silver (averaging 54.6%), after 24 hours.

Partially oxidized-hosted gold-silver mineralization tested in two samples from Batch 2 returned variable cyanide leach recoveries between 63.25 to 68.5% for gold (averaging 63.3%) and 58.1 to 78.9% for silver (averaging 68.5%) after a 24 hour leach time.

Bottle roll extraction periods were limited to a maximum of 24 hours for all the composites, at which point all the tests indicate that leaching was continuing.

Fresh, sulfide-hosted (NoOx) and partially oxidized (POx) mineralization reported differing average gold and silver extraction rates over 6, 12 and 24 hours respectively including:

Gold:
1hr: NoOx 35.7% vs. POx 62.6%
12 hr: NoOx 37.2% vs. POx 59.1%
24 hr: NoOx 48.4% vs. POx 63.3%

Silver:
1hr: NoOx 22.1% vs. POx 43.0%
12 hr: NoOx 30.1% vs. POx 53.1%
24 hr: NoOx 49.7% vs. POx 67.7%

As shown in Figure 13-2, a positive correlation between gold value and gold recovery is evident particularly between the grade ranges between 0 and 18 ppm Au.

As shown in Figure 13-6 (albeit based on a relatively low number of data points), an overall negative correlation is indicated between arsenic concentrations and gold recoveries for arsenic values greater than approximately 6,000 ppm. It is interpreted from comparisons between concentrations of arsenic and gold in the non-oxide (NoOx) mineralization that several populations exist, of differing types of gold-silver mineralization, which are characterized by differing arsenic:gold ratios. A strong negative linear correlation between arsenic concentrations and gold recoveries suggest that a considerable portion of the gold and silver occurs in a refractory state with pyrite and arsenopyrite.

Lime consumption varied between 1.14 and 3.34 kg/t (average 1.59 kg/t) and cyanide consumption varied between 0.26 and 1.32 kg/t (average 0.47 kg/t)

Batch 3

This batch consisted of four samples of hypogene ore material from four holes located in the Main shoot for which the leach time was extended to 72 hours and the recoveries were determined at 48 hours and 72 hours.

As shown in Figure 13-8, gold recoveries after 72 hrs for the 4 samples averaged 58% and varied from between 43.3% and 73%. As shown in Figure 13.9, silver recoveries after 72 hrs for the 4 samples averaged 41% which varied from between 26% and 46.9 %. As shown in Figures 13-10 and 13-11, an increase in recovery was achieved with the extended leach time whereby the comparative increase in recovery with respect to the 48hr leach time achieved an average of 12% and 7% for gold and silver respectively.

Batch 4, Heap Leach Composites

Geochemical analysis reported a head grade between 1.0 and 9.6 g/t Au (average 4.4 g/t Au) and 1213 Ag (average 167.8 g/t Ag) for the Heap composite samples.

Leach tests over 24hr for the 75 µm fraction for this category achieved average gold recoveries of 89%, (ranging from 69.6% and 97%), and for Ag achieved an average recovery of 54.8% (ranging from 14% and 81.3%).

Leach tests over 24hr for the ¼’’ fraction for this category achieved average gold recoveries of 53.2%, (ranging from 15% and 72.5%) and for Ag achieved average recoveries of 23.5% (ranging from 5.9% and 53.3%).

Leach tests over 336 hr for the ¼'' fraction for this category achieved average gold recoveries of 77.6%, (ranging from 50.3% and 91.6%) and for Ag achieved average recoveries of 39.6% (ranging from 10.1% and 72%).

The ¼'' fraction of the individual composites 1, 4, 6, 9, 10 and 12 returned gold recoveries averaging 38% (ranging from 15-58%) and silver recoveries averaging 28% (ranging from 13.4-53.3%). These latter composites host elevated values of sulphur (0.7-1.5%) and/or higher concentrations of Arsenic which suggest the existence of minor remnant partial sulfide material. For these latter composites, finer grinding to 75 µm increased the recoveries for both gold and silver to averages of 83% (ranging from 69.6-92.9%) and 59% (ranging from 48.7-81.3%) respectively.

Overall for all the composites from this batch, the gold and silver recovery (24 hr) on 75 µm is approximately 30 - 50% higher than for the ¼" fraction, and the gold and silver recovery on the ¼" fraction in 14 days was 10% less than the recovery on the 75 µm fraction over 24 hr.

As shown in Figures 13.10 and 13.11, leach time curves for the ¼'' fraction for each composite to 336 hours display shallow to moderate gradients which indicates continued leaching potential for longer leach cycles.

Batch 4, Dump Leach Composites

Geochemical analysis reported a head grade between 0.37 and 0.6 g/t Au (average 0.45 g/t Au) and 8 to 29 g/t Ag (average 17.8 g/t Ag) for the Dump composite samples.

Leach tests over 24hr for the 75 µm fraction for this category achieved average gold recoveries of 91.9%, (ranging from 89.3% and 93.8%), and for Ag achieved an average recovery of 31.5% (ranging from 8.2 % and 55.3%).

Leach tests over 24hr for the 1/2'' fraction for this category achieved average gold recoveries of 48.1%, (ranging from 24.3% and 65.5%) and for Ag achieved average recoveries of 9.6 % (ranging from 1.4% and 23.1%).

Leach tests over 336 hr for the 1/2'' fraction for this category achieved average gold recoveries of 76.9 %, (ranging from 68.2 % and 88.7 %) and for Ag achieved average recoveries of 14.7 % (ranging from 3.7 % and 35.5%).

Overall for all the composites from this batch, the gold and silver recovery (24 hr) on 75 µm is approximately 90% and 300% higher respectively than for the 1/2" fraction, and the gold and silver recovery on the ½'' fraction in 14 days was 20 and 30% less respectively than the recovery on the 75 µm fraction over 24 hr.

As shown in Figure 13-12, the leach time curves for the 1/2'' fraction for each composite to 336 hours display shallow to steep gradients, especially in the case of Composites 1 and 6, which serve to indicate good continued leaching potential for longer leach cycles.

Batch 4- Hypogene Composites

Geochemical analysis for the Hypogene composite samples reported a head grade averaging 5.3 g/t Au (ranging between 1.8 and 15.1 g/t Au) and averaging 189.1 g/t Ag (ranging between 17 and 875 g/t Ag).

Results for this batch returned relatively low and highly variable cyanide leach recoveries for the 75µm fraction after 24 hours which averaged 31.1% for gold (ranging from 3.5 to 65.50 %) and 25.5% for silver (ranging from 15.6 to 40.2 %) which are consistent with the relatively low results reported from hypogene samples from Batches 2 and 3.

13.3 Summary of Mineral Processing Tests

It is apparent that the concentration of gold by gravity separation is relatively ineffective. Concentration of values by flotation has not yet been tested.

Bottle-roll tests using cyanide give recoveries above 95% for gold in some optimal cases and above 80% for silver in some optimal cases, depending upon grind size, oxidation state, metal grade, and test time.

There is a negative correlation between arsenic concentrations and gold recoveries.

CAM believes that the bottle-roll cyanidation tests are encouraging, and that further testing will allow optimized processing flowsheets to be established for the various types of Cap Oeste mineralization.

14.0 MINERAL RESOURCE ESTIMATES

The resource model was prepared and much of this section was written by Patagonia Gold. It has been reviewed and edited by CAM. Any significant work by CAM is indicated by "CAM" in a sentence or paragraph.

14.1 Modelling Mineral Domains/Types

Cap-Oeste modelling was carried out initially through the completion of a detailed sectional interpretation on plotted sections and subsequently digitised in 3D. A solid geological grade constraining wireframe model was created. Vertical sections were created every 25m perpendicular to the strike of the Cap-Oeste mineralization, the orientated grid being aligned at azimuths 320 and 130 degrees. All sections were interpreted looking SE (130). Grade constraining polylines were digitised in 3D and snapped to non-composited original samples based on the following criteria:

- AuEq cut-off grade for Oxide 0.30 g/t
- Au Recovery for Oxide 90%
- Ag Recovery for Oxide 65%
- AuEq cut-off grade for Sulfide 1.00 g/t
- Au Recovery for Sulfide 92%
- Ag Recovery for Sulfide 93%
- Minimum ore width 2m
- Maximum internal dilution of 3m for sub Cut-off grade material
- No Hanging wall and footwall dilution added

A section showing the Topography, Oxide Profile, Oxide mineralization and Sulfide mineralization is shown in Figure 14-1.

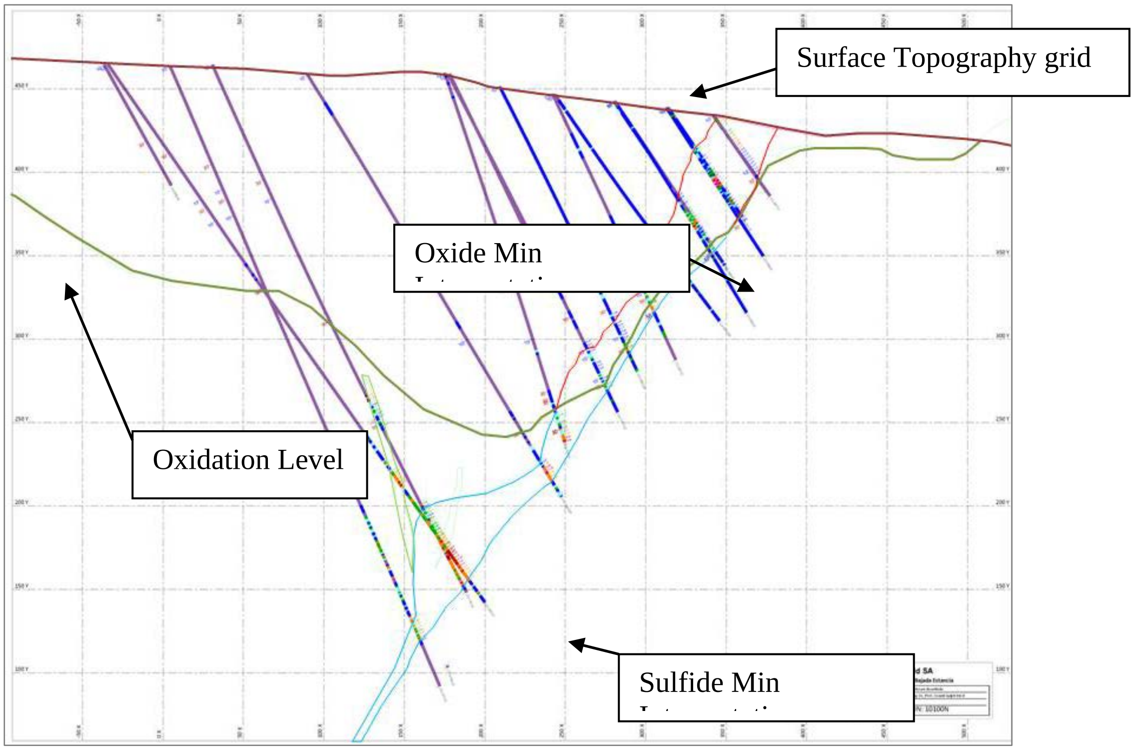


Figure 14-1
SE looking section (10,100N) for Cap-Oeste showing sectional interpretation and Drillhole data

Interpretations of Lithology, oxidation and mineral domains (Oxide, Sulfide and sub-vertical Veins) were completed for each 25m section. Mineralization is structurally controlled by the Footwall Bonanza fault and is persistent and a series of cross-cutting faults which culminate in the formation of “Shoots” plunging 45 Deg to 320. No intermediate or “Transitional” oxidation state was assigned to the resource and any semi or moderately oxidised material was assigned a Sulphidic rock code, the quantity of “Transitional” material is considered to be very low and not material to the resource

Gold and Silver domains were interpolated within the same geological grade domain, a weighted gold-equivalent (Oxide Au 90% Rec, Ag 65% and Sulfide Au 92% and Ag 93% Rec) value was used as a final check to be certain all potentially economic material fell within the constraining model

Excellent continuity both along strike and down dip is seen in the Cap-Oeste mineralization model, the high density of the drilling completed by PGSA has enabled a good quality 3D model to be produced with a high level of confidence in the dimensions, extent and morphology of the interpreted mineralization.

Once 3D models were stitched together and checked for accuracy all Drillhole data falling within the enclosed models was selected and subset into different rock codes (Oxide, Sulfide and Veinlets) for the interpolation process. A 2m composite length was applied and used as the minimum composite length for the interpolation stage of the estimation.

All models were clipped to surface topography and oxide and sulfide levels were separated by a secondary profile delineating the depth of the oxidised level at the deposit.

Basic global statistics were carried out to recognise any outliers or trends indicating a need for top cutting of Au and Ag or restraining of search parameters.

Top cuts were not applied although a restricted search ellipse was used for both elements in both Oxidised and Sulphidic material (MB need distances and grades).

Table 14-1 shows summary statistics for each metal within the respective mineralization types.

Table 14-1
Summary Statistics for each Mineralization Type

Statistic	Au	Ag
OXIDE		
Mean	1.45	52.05
Median	0.42	15.68
Geometric Mean	0.50	16.79
Natural LOG Mean	-0.69	2.82
Standard Deviation	5.68	178.35
Variance	32.29	31810.00
Log Variance	1.41	1.86
Coefficient of Variation	3.93	3.43
Moment Coefficient of Skewness	10.77	11.04
Moment Coefficient of Kurtosis	135.48	161.49
SULFIDE		
Mean	2.96	105.94
Median	1.19	18.98
Geometric Mean	1.21	21.50
Natural LOG Mean	0.19	3.07
Standard Deviation	8.66	371.00
Variance	74.97	137639.08
Log Variance	1.64	2.80
Coefficient of Variation	2.93	3.50
Moment Coefficient of Skewness	16.09	8.84
Moment Coefficient of Kurtosis	375.49	109.80
VEINLETS		
Mean	1.43	26.62
Median	0.90	8.84
Geometric	0.70	9.18

Table 14-1 Summary Statistics for each Mineralization Type		
Statistic	**Au**	**Ag**
Natural	-0.36	2.22
Standard	2.46	65.19
Variance	6.05	4249.83
Log	1.89	1.93
Coefficient	1.72	2.45
Moment	7.80	6.36
Moment	89.79	53.28

14.2 Block Model

A block size of 5m x 5m x2.5m was chosen for the Cap-Oeste model, based on the tight drill spacing (25m x 25m average) for the deposit

The block-model parameters are shown in Table 14-2.

Table 14-2 Block-Model Parameters.	
Columns	**212**
Rows	248
Levels	260
Origin	
X	2390706.9
Y	4687099.9
Z	550
Rotation	40
Block Sizes	
X	5
Y	5
Z	2.5

Each rock type was assigned a separate block-model code and a partial model was built in GEMS to ensure that each rock type was assigned the correct tonnage from the wireframe model. Percent models were created and the solid model wireframes intersected with each block model with a resulting percent and tonnage assigned to each block falling within the wireframes. Final checks were carried out to ensure wireframe volumes were equal to the block volume assigned.

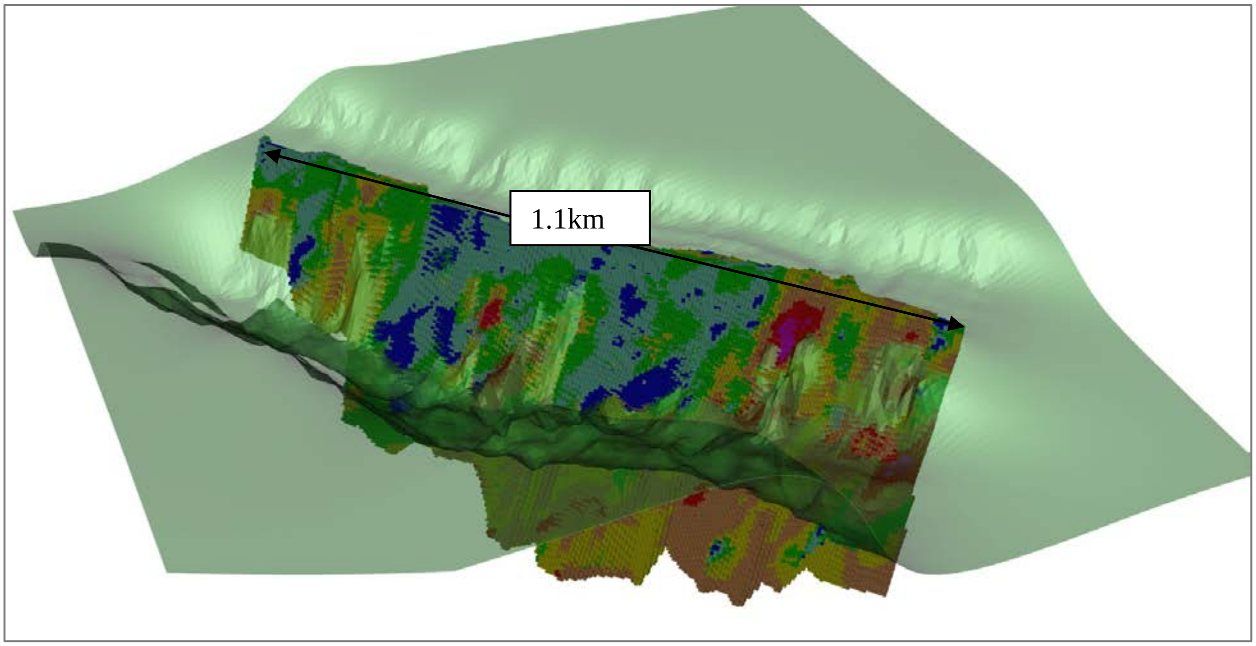


Figure 14-2
Au Block Model in 3D with Ox_Prof Surface Wireframe

Grade Interpolation was completed in 2 phases for both elements. Search ellipses were set up for each metal and each mineralization type, 6 in all, Ox Au, Ox Ag, Sul Au, Sul Ag, Vn_Au and Vn Ag, as shown in Figure 14-3.

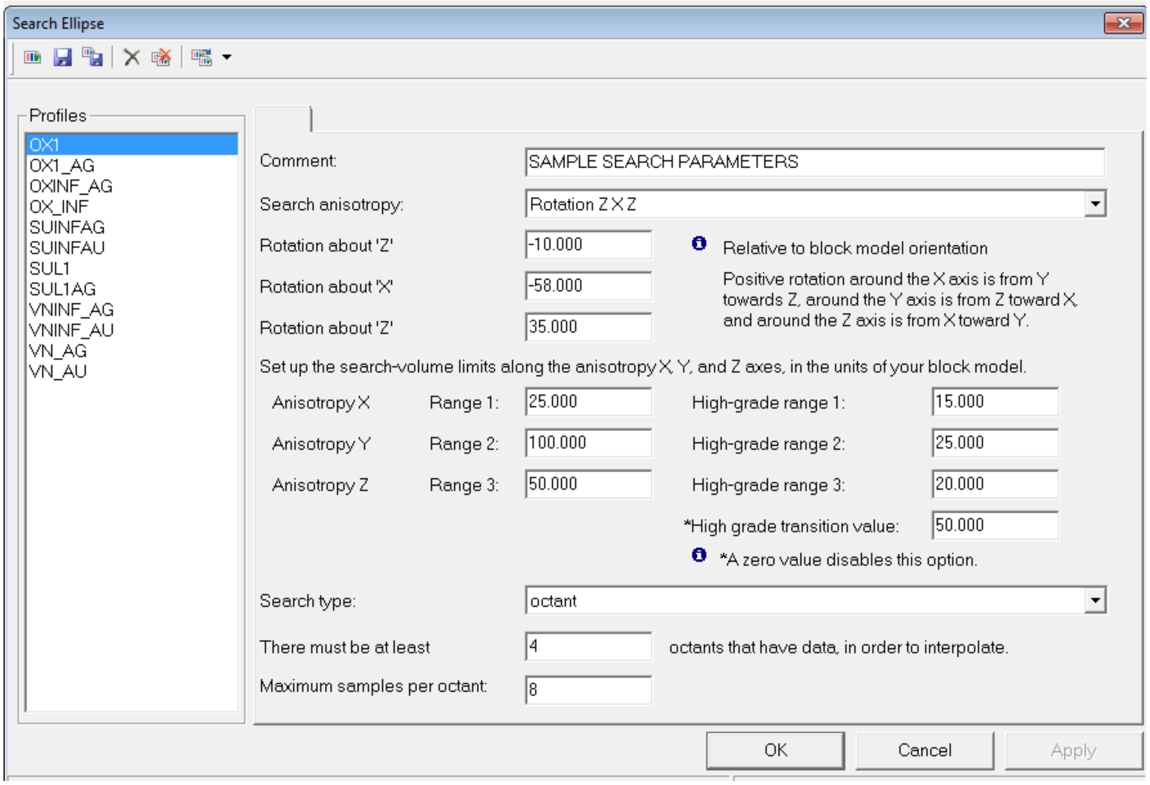


Figure 14-3
Search Ellipse Parameters

The ellipses created for Gold Oxide, Sulfide and Silver Oxide Sulfide varied little in anisotropy and dimension

14.3 Resource Classification

There are no quantitative definitions of measured, indicated, and inferred mineral resources. However, if a distance criteria for resource classification is used, a deposit may be usually considered as at least indicated if it is drilled on a rectangular grid corresponding to the range of the first structure of the variogram. This assures that every block within such a grid is estimated by at least two points within the range of the variogram of that block. For a perfectly square grid this means that the block at the center of the square is 0.7071 times the side of the square away from the nearest sample point. Hence CAM regards blocks as at least inferred if it is within 0.7071 of the range of the first structure the variogram.

CAM initially attempted to construct variograms using the nominal 2 m composites used in estimation; however these variograms were greatly influenced by the downhole data. Hence CAM went to a 2-D approach which is appropriate given the vein like nature of most of the Cap Oeste structure. Variograms using the apparent grade thickness of the composites used in the estimate were constructed using data points at the centroid of the composites for each hole. These data were approximately lognormally distributed so CAM constructed relative variograms from logs for AuEQ*(apparent thickness). The apparent thickness used was the thickness of the composites for each hole perpendicular to the vertical plane at in azimuth of 320° (320° is the approximate strike of the structure. This variogram is shown in Figure 14-4.

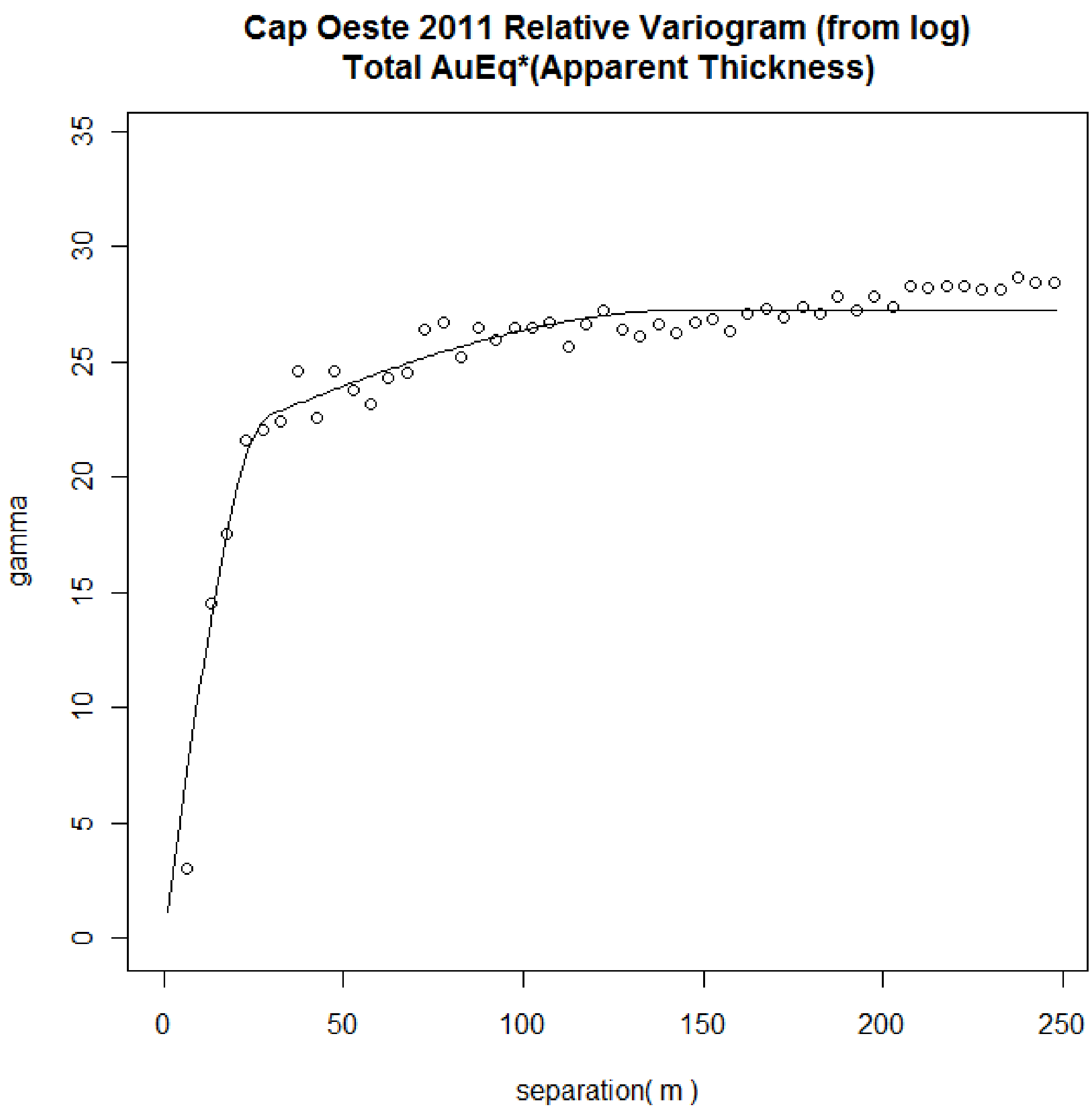


Figure 14-4
Variogram for AuEq times Apparent Thickness.

The autofit range of the first structure is 29.3m, which according to the CAM criteria means that any block within 20.72 m of a composite may be classed as indicated resource. The first structure of 29.3 m is not inconsistent with the approximate 30 m first structure observed in the downhole variograms. There are a limited number of pairs on the first three points of the variogram, and there is only limited closely-spaced data to define the along-strike and down-dip mineable continuity of the deposit. Since there is no mining experience on the deposit, CAM does not classify any of this resource as measured. CAM restricted the inferred resource to a maximum distance of 50 m from the nearest composite. These criteria were provided to Patagonia for use in resource classification. A detailed geostatistical study is in progress by the consultants Geovariances, and it is likely that the resource classification criteria may change somewhat in the next update.

14.4 Resource Tabulation

The resource classified according to the criteria discussed above is given in Table 14-3.

Table 14-3
Resource Tabulation by Classification and EQAu Cutoff

Type	Class	CO	SG	Tonnes	Auppm	AuCTOZ	Agppm	AgCTOZ	AuEQppm	AuEQCTOZ
Ox	Ind	0.00	2.40	3153281	1.36	137546	50.62	5132009	2.30	233472
Ox	Ind	0.30	2.40	3150543	1.36	137532	50.66	5131724	2.30	233452
Ox	Ind	0.50	2.40	2960656	1.42	135584	53.49	5091842	2.42	230758
Ox	Ind	1.00	2.40	1902204	1.96	120102	75.19	4598505	3.37	206055
Ox	Ind	1.50	2.40	1308032	2.56	107552	95.55	4018285	4.34	182660
Ox	Ind	3.00	2.40	626558	4.02	81048	146.56	2952320	6.76	136232
Sul	Ind	0.00	2.50	4296531	3.09	426900	100.28	13851710	4.96	685811
Sul	Ind	0.30	2.50	4289881	3.10	426879	100.42	13850823	4.97	685773
Sul	Ind	0.50	2.50	4274411	3.11	426752	100.76	13846479	4.99	685565
Sul	Ind	1.00	2.50	4159067	3.18	424810	103.12	13789420	5.10	682556
Sul	Ind	1.50	2.50	3800106	3.40	415363	110.61	13513622	5.47	667954
Sul	Ind	3.00	2.50	2312629	4.59	341351	157.04	11676430	7.53	559602
Vn	Ind	0.00	2.50	771583	1.42	35346	27.24	675650	1.93	47975
Vn	Ind	0.30	2.50	742524	1.47	35159	28.21	673457	2.00	47747
Vn	Ind	0.50	2.50	696914	1.55	34711	29.75	666481	2.11	47169
Vn	Ind	1.00	2.50	589469	1.73	32774	33.47	634239	2.35	44629
Vn	Ind	1.50	2.50	443866	1.95	27872	40.47	577567	2.71	38668
Vn	Ind	3.00	2.50	145618	2.86	13376	64.02	299717	4.05	18978
All	Ind	0.00	2.46	8221395	2.27	599792	74.38	19659369	3.66	967257
All	Ind	0.30	2.46	8182948	2.28	599570	74.71	19656004	3.68	966972
All	Ind	0.50	2.46	7931981	2.34	597047	76.88	19604803	3.78	963492
All	Ind	1.00	2.47	6650740	2.70	577686	88.96	19022165	4.36	933241
All	Ind	1.50	2.48	5552004	3.09	550787	101.45	18109473	4.98	889282
All	Ind	3.00	2.48	3084805	4.39	435775	150.52	14928467	7.21	714812
Ox	Inf	0.00	2.40	733965	0.79	18573	24.15	569767	1.24	29223
Ox	Inf	0.30	2.40	733965	0.79	18573	24.15	569767	1.24	29223
Ox	Inf	0.50	2.40	690843	0.81	18088	25.26	561063	1.29	28575
Ox	Inf	1.00	2.40	275836	1.25	11081	44.31	392964	2.08	18426

Table 14-3 Resource Tabulation by Classification and EQAu Cutoff										
Type	Class	CO	SG	Tonnes	Auppm	AuCTOZ	Agppm	AgCTOZ	AuEQppm	AuEQCTOZ
Ox	Inf	1.50	2.40	111489	1.91	6854	78.73	282205	3.38	12129
Ox	Inf	3.00	2.40	45821	2.82	4151	139.75	205876	5.43	7999
Sul	Inf	0.00	2.50	1379690	2.88	127671	69.28	3073016	4.17	185110
Sul	Inf	0.30	2.50	1379690	2.88	127671	69.28	3073016	4.17	185110
Sul	Inf	0.50	2.50	1379637	2.88	127670	69.28	3073012	4.17	185110
Sul	Inf	1.00	2.50	1366473	2.90	127426	69.82	3067542	4.21	184763
Sul	Inf	1.50	2.50	1237054	3.12	124077	74.55	2964878	4.51	179495
Sul	Inf	3.00	2.50	749931	4.18	100783	96.33	2322715	5.98	144199
Vn	Inf	0.00	2.50	309257	1.03	10239	23.78	236407	1.47	14658
Vn	Inf	0.30	2.50	306596	1.04	10221	23.96	236157	1.48	14635
Vn	Inf	0.50	2.50	292606	1.07	10068	24.86	233891	1.53	14440
Vn	Inf	1.00	2.50	201051	1.29	8363	32.22	208290	1.90	12257
Vn	Inf	1.50	2.50	142561	1.44	6591	39.00	178757	2.17	9932
Vn	Inf	3.00	2.50	11781	2.01	759	78.19	29615	3.47	1313
All	Inf	0.00	2.47	2422912	2.01	156483	49.80	3879190	2.94	228992
All	Inf	0.30	2.47	2420252	2.01	156465	49.85	3878940	2.94	228968
All	Inf	0.50	2.47	2363086	2.05	155826	50.91	3867966	3.00	228125
All	Inf	1.00	2.48	1843359	2.48	146870	61.90	3668795	3.64	215446
All	Inf	1.50	2.49	1491103	2.87	137521	71.46	3425841	4.20	201556
All	Inf	3.00	2.49	807534	4.07	105694	98.53	2558207	5.91	15351

14.5 Resource Verification by CAM

Even though the resource estimation procedure used by PGSA is reasonable and done in a well- known software system (GEMCOM), there is a small chance that a procedural or other error has occurred in the estimation process. To check for this unlikely possibility, CAM always reruns an unconstrained nearest-neighbor resource estimate using a different software system (in this case MicroModel).

The nearest neighbor estimate, while well-known to be geometrically unbiased, over-predicts grade and under-predicts tons but is generally close on contained metal. Because this estimate is unconstrained, it is inferior to a geologically constrained model for locating ore, but if the unconstrained estimate shows less contained metal it is indicative of some type of problem in the estimation process. For the unconstrained estimate CAM used one-meter uncapped composites, with a search ellipse oriented parallel to the vein structure. The same distances for resource classification were used as per the PGSA estimate.

Because the unconstrained estimate did not make the oxide-sulfide vein split, CAM used a density of 2.4 for all mineralization types. As would be expected for a properly constructed model, the CAM nearest neighbor grades were higher and the tonnage was lower. However, in all cases the contained ounces in the CAM model were more, indicating that the PGSA model may be slightly conservative. CAM believes this conservatism is appropriate for a project at this level of development.

14.6 Conclusions and Recommendations for Resource Estimation

On the basis of the review of the methodology, visual review of the model, and independent checks, CAM believes the model has been prepared according to accepted engineering practice and is suitable for initial reserve calculations. CAM notes that the model may have to be revised slightly to account for the different grade-control procedures to be used in open pit and underground and that the model will likely have to be revised as actual production experience is gained.

Recommendations are as follow:

1. Extend the surface topography model to include the entire area of the Cap Oeste block model, with such extensions as necessary to cover any planned dumps for the open pit.
2. Once the Geovariances geostatistical study is complete, review the need for additional closely-spaced drilling (probably using wedges) to define the short range mineable continuity in the plane of the vein for both the oxide and sulfide portions of the orebody.
3. Once the close spaced drilling recommended into is complete, review the estimation procedures to make sure that the model is consistent with the results expected with the grade control procedures for both open pit and underground.
4. Proceed with the resource calculation for both open pit and underground scenarios. This should include a balancing limit between open pit and underground as well as a breakeven pit. Once these designs are obtained, review the need for additional drilling to precisely define the open pit bottom.

15.0 MINERAL RESERVE ESTIMATES

No mineral reserves are disclosed in this Technical Report.

16.0 to 22.0 ADDITIONAL REQUIREMENTS FOR ADVANCED PROPERTIES

These sections are omitted.

23.0 ADJACENT PROPERTIES

As previously discussed, significant precious metal mineralization has been defined at the adjacent COSE project and at several other prospects within a radius of approximately seven kilometres around Cap Oeste. The nearby COSE project is discussed in a separate NI 43-101-format report by CAM, submitted to Patagonia Gold in September, 2011. The COSE report has not yet been filed in Canada, as Patagonia Gold is not listed in Canada. Further exploration drilling is scheduled throughout the COSE, Pampa, Tango, Don Pancho and Vetas Norte prospect areas in the future (refer to Figure 7-5).

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. The mineral resources discussed herein lie entirely on the Cap Oeste project area as defined.

24.0 OTHER RELEVANT DATA AND INFORMATION

The authors are not aware of any additional information, the exclusion of which from this report make it misleading.

25.0 INTERPRETATION AND CONCLUSIONS

Cap Oeste definitely merits further exploration for additional gold-silver mineralization in an epithermal setting.

Substantial additional exploration has been completed on the Cap Oeste project since completion of the MICON report in 2009:

sufficient in-fill drilling in the resource area to increase drill intercept density to a nominal 25 meter centers

exploration drilling along strike and down dip of known mineralization

additional metallurgical and mineralogical studies

exploration within the district, most notably leading to discovery of the nearby COSE deposit, which is now considered a separate project.

CAM is of the opinion that this work meets or exceeds best industry practice, and that the resulting exploration database is suitable for use in mineral resource estimation.

Metallurgical tests have shown that the concentration of gold by gravity separation is relatively ineffective. Concentration of values by flotation has not yet been tested. Bottle-roll tests using cyanide yield variable but often high recoveries for gold, and somewhat lower recoveries for silver, depending upon grind size, oxidation state, metal grade, and test time. There is a negative correlation between arsenic concentrations and gold recoveries. CAM believes that the bottle-roll cyanidation tests are encouraging, and that further testing will allow optimized processing flowsheets to be established for the various types of Cap Oeste mineralization.

26.0 RECOMMENDATIONS

The broad zones of stockwork mineralization where the Bonanza and Esperanza faults converge, should be drilled as this presents a potential, bulk-tonnage style target, especially where intersected by plunging ore shoots.

Additional drilling should be undertaken to prove the geometry and continuity of the higher-grade pods currently designated as Shoots C and D.

PGSA should proceed to generate a new mineral resource for the Cap Oeste project, when the data at hand warrant.

Additional bulk-density measurements should be made on core, preferably at least 50 measurements for each type of lithology and mineralization. These measurements should be calibrated with standard density samples (e.g. sealed metal tubes) covering the range of densities encountered in the deposit (SG's about 1.9 to 2.7).

Once the geovariances geostatistical study is complete, review the need for additional close spaced drilling (probably using wedges) to define the short range mineable continuity in the plane of the vein for both the oxide and sulfide portions of the orebody.

Proceed with the resource calculation for both open pit and underground scenarios. This should include a balancing limit between open pit and underground as well as a breakeven pit. Once these designs are obtained, review the need for additional drilling to precisely define the open pit bottom.

The Work Program in Table 26-1 is recommended. Phase II is dependent on success in Phase I.

Table 26-1
Proposed Work Program, Cap Oeste Project

Item	Basis	Unit Cost, US$	Total Cost, US$	Time Period
Phase I				
Infill & Exploration drilling	50 holes @ 350m = 17,500 m	$ 160/m	$2,800,000	Q 3 & 4, 2011, Q1 2012
Other Drillholes (geotech, RQD, water, etc	5 holes @ 300 m= 1,500 m	$ 200/m	$ 300,000	Q 3 & 4, 2011
Camp, Geology, Assays	3900 m	$ 40/m	$160,000	Q 3 & 4, 2011
Geostatistics/Reporting	resource updates & similar	$ 50k/month	$150,000	Q4, 2011
Project Overhead,	B.A. office, 4months	$ 30,000/month	$120,000	Q4, 2011
SUBTOTAL, PHASE I			$ 3,530,000	
Phase II				
Metallurgical tests	estimate	$50,000	$50,000	Q1 & Q2, 2012
Pre-Feasibility Study	estimate	$600,000	$600,000	Q3 & Q4, 2012, Q1 2013
SUBTOTAL, PHASE I			$ 650,000	
TOTAL, PHASES I & II			$ 4,180,000	

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28.0 DATE AND SIGNATURE PAGE

28.1 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 2 to 9, 13, 23 to 27, and the relevant parts of sections 1, 10, and 11, of the report entitled "NI 43-101 Technical Report, Update of Cap Oeste Project, Santa Cruz Province, Argentina" dated 14 November, 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 site visits on 22-24 April, 2008, 21-22 November, 2010 and 16-17 August, 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, for which the omission to disclose would 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 14th day of November, 2011

Craig S. Bow, CPG

28.2 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 Society for Mining, Metallurgy, and Exploration (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 Master 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 12, 14, and 15, and parts of sections 1, 10 and 11, of the report entitled "NI 43-101 Technical Report, Update of Cap Oeste Project, Santa Cruz Province, Argentina", dated 14 November, 2011 (the "Technical Report"). The Technical Report is based on my knowledge of the Project Area and drilling database included in the Technical Report, and on review of published and unpublished information on the property and surrounding areas. I conducted site visits on 22-24 April, 2008, and 21-22 November, 2010.
- 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, for which the omission to disclose would 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 14th day of November, 2011

R L Sandefur

Robert L. Sandefur, P.E.