

**Form 43 -101F1
TECHNICAL REPORT**

INDEPENDENT PRELIMINARY ASSESSMENT

ZARUMA GOLD PROJECT

EL ORO PROVINCE - ECUADOR

PREPARED FOR DYNASTY METALS & MINING INC.

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1.0 Executive Summary

1.1 Introduction

This report has been prepared by Consulting Mining Engineer Mr RL Procter BSc (Eng), MBA, MIMMM, CEng, and Consulting Geologists Mr AJ Maynard, BAppSc (Geol) MAIG, MAusIMM and Mr PA Jones, BAppSc (Geol), MAIG. Mr Procter visited the Company's mining and processing operations in February 2014, and has undertaken investigations in respect of the mining, treatment, infrastructure, operations and economics of the property. Mr Maynard has verified previous geological and resource information from previous technical reports in respect of sections 6 to 14 of this report.

The Preliminary Economic Assessment (PEA) is preliminary in nature and includes the use of Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorised as Mineral Reserves, and there is no certainty that the results of the PEA will be realized.

This Report was prepared at the request of Dynasty Metals & Mining Inc. ("Dynasty" or the "Company") and is based on:

- a site visit in February 2014 by R Procter, site visits by A Maynard in March 2013, January 2010, January, 2008 and several trips prior to 2008;
- interviews with Company management and other employees engaged to assist the author to visit the various sites and review the mining, processing and infrastructure operational areas;
- a review of the Company's Technical Reports, in particular the technical report entitled "Independent Preliminary Assessment – Zaruma Project – El Oro Province, Ecuador" prepared by W J Holly and Mr Maynard dated 21 August 2006 (the "2006 Report");
- a review of the Company's mining and processing plans, and historical performance, as prepared by the Company's staff;
- a review of the Company's mining, processing plant and infrastructure design and operations (including all security aspects);
- a review of the Company's head office and head office functions such as health, safety and environmental management; and
- a review of commodity prices and recent commodity trends.

1.2 Property Description and Location

The Zaruma Gold Project is situated in El Oro Province of southern Ecuador, about 175 kilometres south and 60 kilometres southeast of the major port cities of Guayaquil and Machala respectively.

The Zaruma Gold Project comprises 41 concessions covering a total area of 10,121.47 hectares in a readily assessable region of southern Ecuador. The gold mineralization found on the concessions is associated with extensive high grade gold and silver epithermal vein systems.

38 concessions are 100% Dynasty owned. The other three concessions are "condominos": Barbasco Unificado is 25% Dynasty-owned, Barbasco 1 A is 50% Dynasty owned and Soroche Unificado is 75% Dynasty owned. Mineral rights are held 100% by Dynasty.

A royalty of 3.0% of the total gold revenue after deduction of non-payable metal, transport and refining charges is paid to the Ecuadorian Government. A further royalty calculated at 1.5% of net value after deduction of non-payable metal, transport and refining charges is paid to IAMGOLD Corporation ("IAMGOLD").

1.3 History, Exploration and Drilling

Zaruma is a historic mining field based on the mining of narrow high grade quartz veins rich in gold, silver and in some cases base metals. The exploration concept has been to utilise old mining records, mining development and sampling, surface mapping and exploration drilling to demonstrate resources which can be profitably mined.

Dynasty has inherited exploration data from the previous owners of the Zaruma property (TVX Gold Inc. ("TVX") and IAMGOLD). A Technical Report entitled: "Zaruma Project, Ecuador – Technical Report" was completed in respect of this data by M Kalinaj dated October 22, 2004 (the "2004 Report"). Subsequently, an underground sampling program was undertaken by Dynasty and a further Technical Report entitled: "Independent Geological Evaluation – Zaruma Project, El Oro Province, Ecuador" was prepared by A Maynard dated October 21, 2005 (the "2005 Report"). Both reports quoted mineral resources that were confirmed in the 2006 Report.

The Company has established underground mining and processing operations at the Zaruma Gold Project. The underground portal is some 7.5 km from the processing facility. Over the year 2013 the facility treated some 179,000 tonnes (t) for the recovery of approximately 30 thousand ounces (koz) gold and 71 koz of silver at average metallurgical recovery rates for gold and silver of 88.4% and 41.6%, respectively. Mining during the majority of this period was from areas outside of the resource envelope. Whilst the underground mine still has much development to undertake to reach a stage where it is able to produce at a planned steady state, the processing plant has, over the last few years, been processing lesser low-grade material as Dynasty has advanced along the decline and towards the resource and is considered to be performing well (subject to the erratic tonnage and grade of material delivered from the underground workings).

In 2013, underground mining produced 161 kt of processed material grading an average of 6.37 grams per tonne (g/t) Au (including 4,800 t grading an average of 16.6 g/t Au from development headings). It is noteworthy that much of the Company's gold production (except for the latter part of 2013) is from workings that have not been part of the current Mineral Resource inventory.

The Project has numerous underground veins available to it for exploitation, and it is not unreasonable to assess the Project as having at least a 15 year mine life, based on current operating and financial metrics. The underground veins tend to be narrow (mostly <1.5 metres (m)), so generally the rate of production will be limited due to the relatively small size of stopes and the amount of development required to bring stopes into production.

The processing plant (a conventional crush, mill, leach, Carbon-in-Pulp (CIP), elution and electrowinning circuit) is well established. Some 25% of the gold is taken out of circuit via the gravity concentration system. There are other processing flowsheet options available which the Project could embark upon to further increase both throughput and precious metal recoveries (adding increased throughput capacity, for example). A preliminary estimate of capital expenditure requirements for the Project for the 15 year life for which it is assessed in the PEA is \$29 million, which comprises an aggregate \$20 million for both the underground mine and processing facility areas and \$9 million for sustaining capital and Project closure costs combined.

1.4 Geology and Mineralization

Subduction of the Pacific Ocean Nazca Plate beneath the South American continent has produced crustal deformation forming the Andes mountain range over the last 25 million years. The resultant compressional forces have produced several Tertiary and Quaternary tectonic basins along major regional faults that cut both the Pacific and Amazonian piedmonts and the Andean range itself.

The Zaruma Gold Project lies within the compressional Inter-Andean Graben that is bounded by regional scale faults. The graben is composed of thick Oligocene to Miocene volcano-sedimentary sequences that cover the Chaucha, Amotape and Guamote terrains. This structural zone hosts several significant epithermal, porphyry, mesothermal, S type granitoid, VHMS and ultramafic/ophiolite precious metal and base metal mineral deposits.

The Project area consists of a 15 km long vein system that lies immediately north of the Piñas Fault, a major regional structure that separates a metamorphic terrain to the south from a Tertiary volcanic sequence to the north. The metamorphic rocks, subparallel to the Piñas Fault, show that this structure represents a suture dated as Jurassic to early Cretaceous. Tertiary volcanic rocks preserved on the northern side of the Piñas Fault unconformably overlie metamorphic rocks facies of continental origin, implying that the suture marks a site of continent-continent collision in the late Jurassic to early Cretaceous.

Vein textures in the Zaruma project area do not significantly change along 10 km of vein trend. Typical textures for low-sulphidation veins have been observed including rhythmically banded textures with dominantly symmetric geometry, drusy textures, hydrothermal breccia textures, dissolution and replacement textures.

In contrast, veins in the Portovelo system in the southern part of the trend change composition gradually along strike southward from quartz through quartz-calcite, to calcite-dominated compositions over a strike of 2 km. A vertical zonation in composition from quartz dominated deeper portions of veins to calcite dominated assemblages near surface was observed, in particular, within the Portovelo en-echelon block.

The gold and silver distribution differs from that of base metals. Major concentrations of gold and silver were observed closely related to the footwall of southerly dipping thrust faults which lie parallel to the Piñas Fault especially in the northern en-echelon sets of R-nivel, Muluncay and Sesmo-Colorada. While in the Portovelo en-echelon set, almost 90% of precious metal production was from the central part of the veins.

1.5 Mineral Resource Estimates

Estimated Mineral Resources at Zaruma are as follows:

Location	Measured			Indicated			Inferred		
	Tonnes (million)	Grade Au (g/t)	Contained Au (Ozs)	Tonnes (million)	Grade Au (g/t)	Contained Au (Ozs)	Tonnes (million)	Grade Au (g/t)	Contained Au (Ozs)
Cabo de Hornos	1.30	13.99	585,000	0.86	12.46	343,000	3.0	12.3	1,201,000
Barbasco	0.19	11.05	66,000	0.09	10.81	33,000	0.3	12.13	128,000
Ayapamba	0.07	12.64	30,000	0.06	11.63	23,000	0.2	12.4	96,000
Vizcaya	0.03	8.40	8,000	0.02	8.40	6,000	0.1	8.4	23,000
Totals	1.59	13.48	689,000	1.03	12.18	405,000	3.7	12.2	1,448,000

Note: Mineral Resources that are not Mineral Reserves have no demonstrated economic viability.

Table 1: Mineral Resource Summary at Zaruma Gold Project.

A cut-off grade of 2.06 was used to derive the Mineral Resource estimate.

The following is a summary of the various contributions to the Mineral Resource inventory:

- Measured contains 25% of the tonnes and 27% of the contained gold metal
- Indicated contains 16% of the tonnes and 16% of the contained gold metal
- Inferred contains 59% of the tonnes and 57% of the gold contained metal.

This Mineral Resource inventory has been used as a basis for the PEA after applying various parameters and assumptions, which are discussed in this Report.

1.6 Preliminary Economic Assessment

As required by NI 43-101, it should be noted that these evaluations are preliminary in nature and that the Inferred Mineral Resources in particular are considered too speculative

geologically to have economic considerations applied to them that would enable them to be categorized as Mineral Reserves. As such there is no certainty that the PEA will be realised.

The Zaruma Gold Project has sufficient Measured and Indicated Mineral Resources for a mine life of about 10.5 years. The Project also has available to it a significant inventory of Inferred Mineral Resource (59% of all tonnes and 57% in gold content). Adding only 27% of the Inferred Mineral Resource (equating to 25% of gold output) to the Project's production schedule increases mine life to 15 years, which is considered to be a reasonable mine life period to assess the Project on a discounted cash flow basis. This is the base case evaluated for the PEA.

The Inferred Mineral Resource is also assumed to be processed at the end of the mine's life (i.e. after all Measured and Indicated material has been treated), so it suffers the greatest discounted cash flow effect in the analysis.

The results from the PEA (taking into account both taxed and untaxed cases) are provided in Table 2, and the salient information for the Project from the evaluation is the following:

- it produces an earnings before interest, tax, depreciation and amortisation of \$683 million;
- at a 10% discount rate, it produces a taxed net present value (NPV) of \$218 million and untaxed NPV of \$322 million;
- it produces a taxed net cash flow of \$441 million and an untaxed net cash flow of \$653 million;
- it has a total unit cash operating cost of \$855/oz (taxed) and \$658/oz (untaxed);
- the overall operating costs equate to 51% of total revenue;
- total capital expenditure is estimated at \$29 million; and
- it pays \$66 million in royalties and \$213 million in taxes, over its estimated 15 year mine life.

	Unit	Value
Operating Cash Flow (EBITDA)	\$M	682.5
NPV¹ (taxed)	\$M	218.2
NPV (untaxed)	\$M	321.6
Net cash flow (taxed)	\$M	440.9
Net cash flow (untaxed)	\$M	653.4
Unit operating cost (taxed)	\$/oz Au	855
Unit operating cost (untaxed)	\$/oz Au	658
Total capex	\$M	29.1
Royalties	\$M	65.6
Taxes	\$M	212.5

¹ at 10% base case discount rate

Table 2: Key Results for Taxed and Untaxed Cases.

The sensitivity of the Project's NPV and net cash flow to changes in key input parameters, for both taxed and untaxed cases, are shown in Figures 1 to 4, respectively. Sensitivity of the Project's NPV and net cash flow to major changes in discount rate versus gold price (taxed and untaxed cases) are shown in Tables 3 and 4.

These data show that the Project's economics are most leveraged to gold price (grade and metallurgical recovery, on an absolute percentage change basis, have the same effect). Of lesser effect, ranked in decreasing amounts are mining operating costs, processing operating costs, dilution and finally changes to capital expenditure. Given the quantum of the latter, capital costs have no material effect on the Project's economics.

The PEA has not been tested for changes in Project scale. Although the Project will have reasonable sensitivity to scale (considering the discounted cash flow effects), at this stage, and without further feasibility study work, this scenario is considered to be speculative. An equivalent change in scale (production output) will have much less effect on Project economics than gold price, material grade or metallurgical recovery changes.

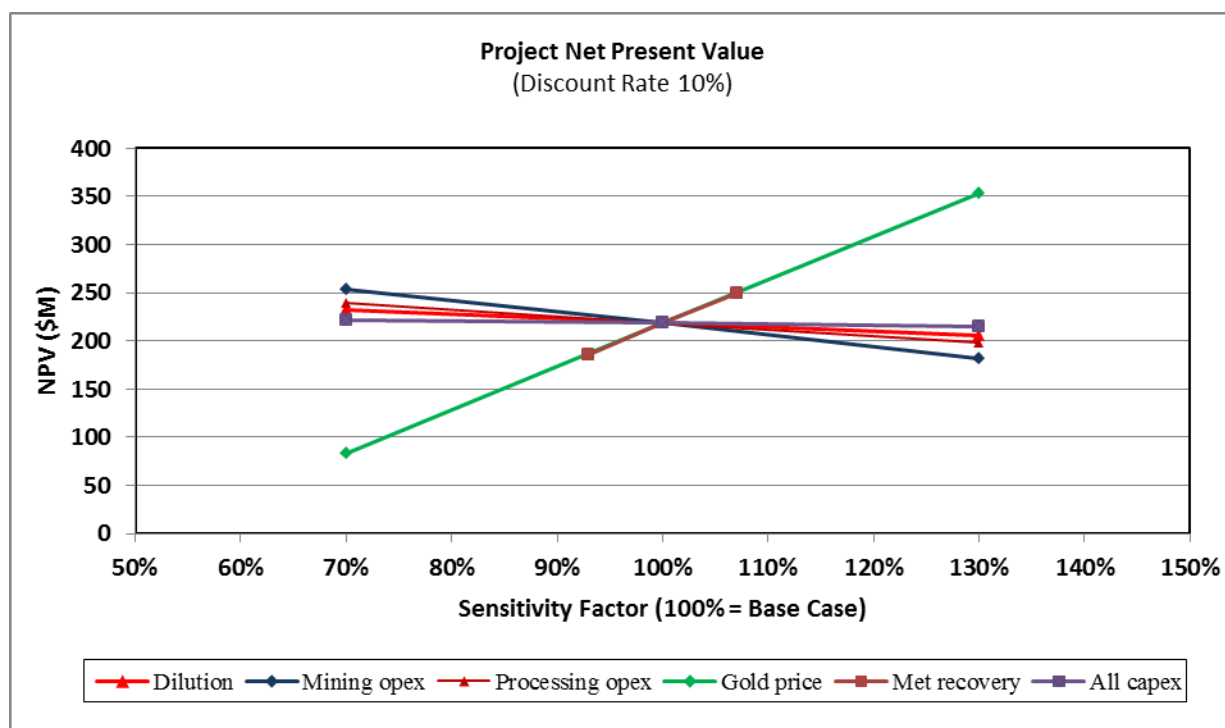


Figure 1: Net Present Value Sensitivities (taxed).

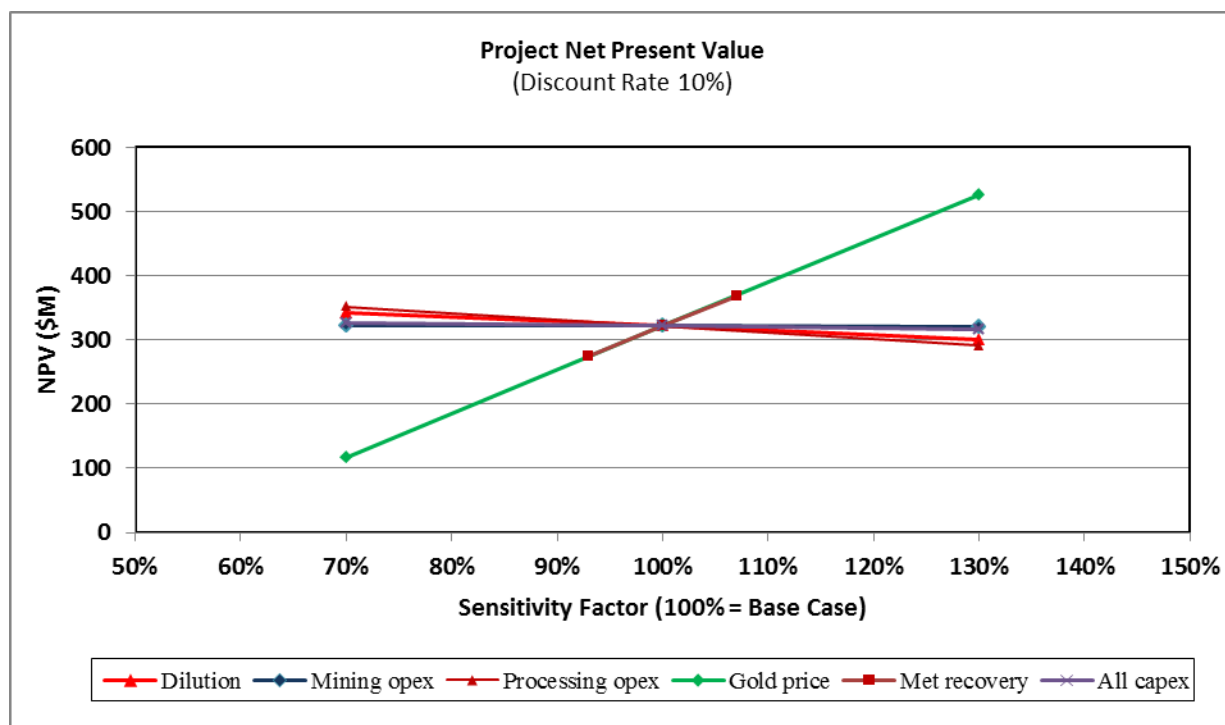


Figure 2: Net Present Value Sensitivities (untaxed).

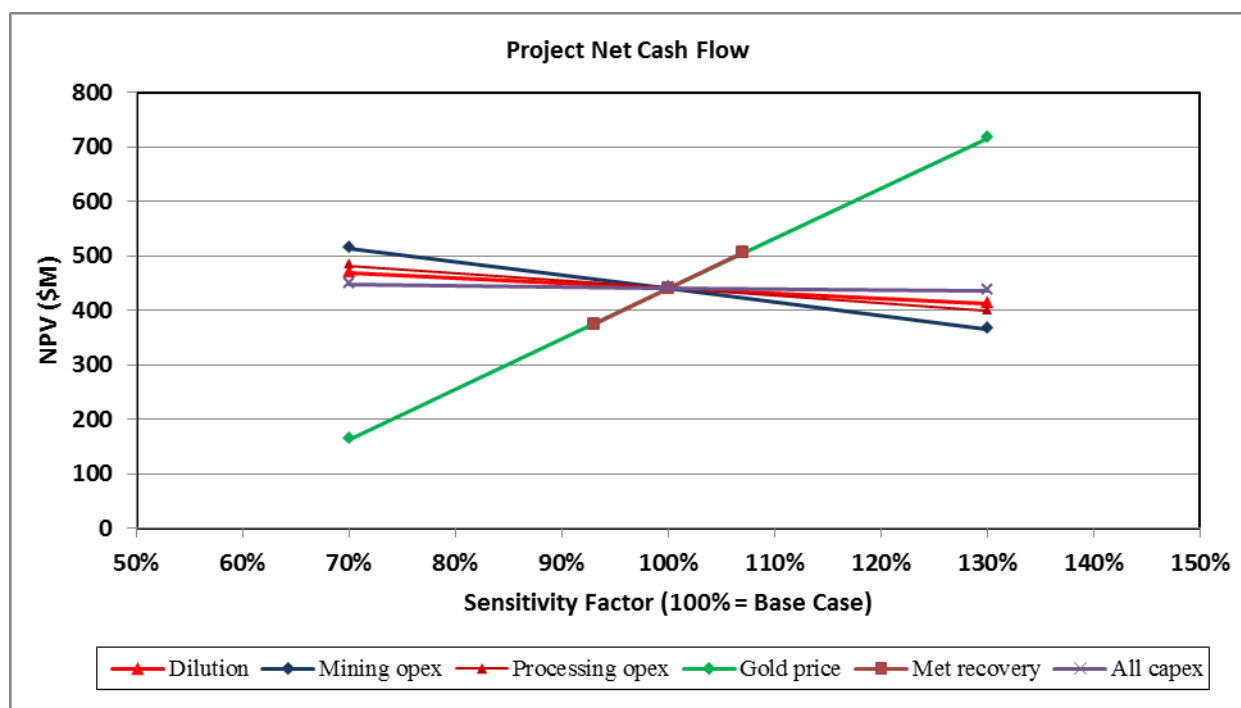


Figure 3: Net Cash Flow Sensitivities (taxed).

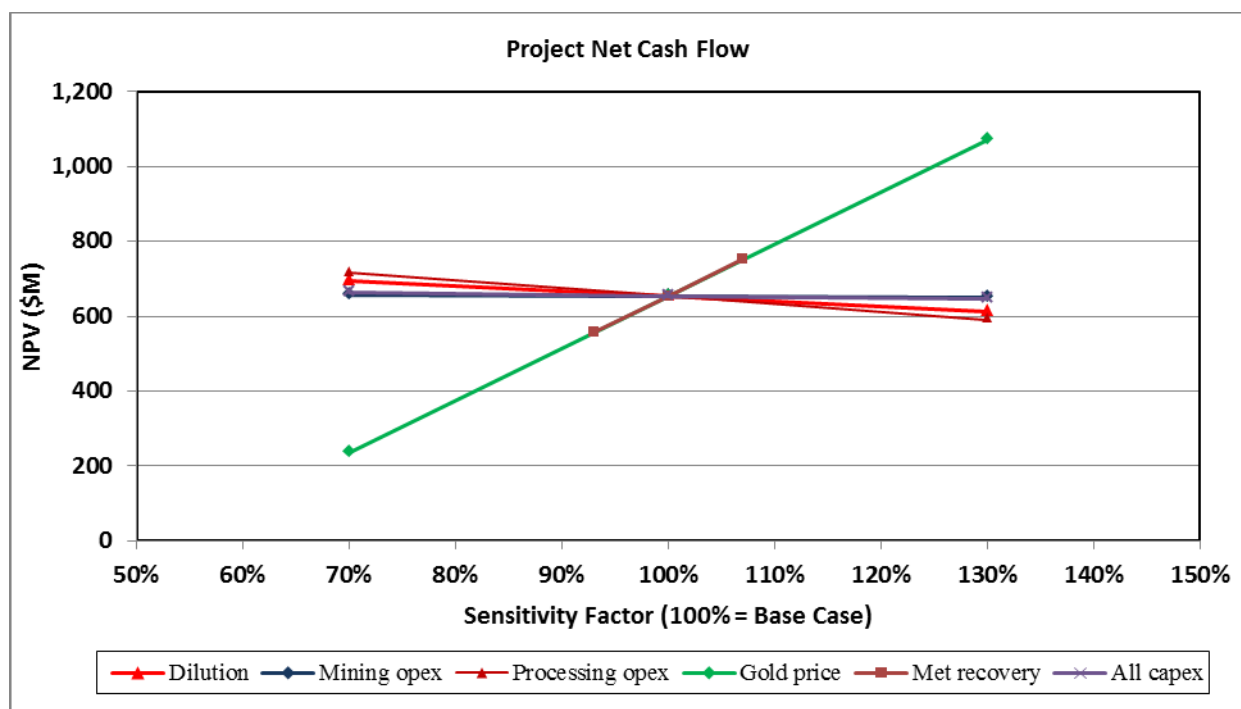


Figure 4: Net Cash Flow Sensitivities (untaxed).

		Discount Rate (%)			
		20	15	10	5
Gold Price	\$ 1500/oz	170,216	216,695	286,746	397,443
	\$ 1400/oz	149,461	190,249	251,709	348,804
	\$ 1300/oz	129,953	165,162	218,158	301,796
	\$ 1200/oz	108,991	138,490	182,874	252,886
	\$ 1100/oz	88,833	112,733	148,635	205,178
	\$ 1000/oz	69,179	87,408	114,738	157,695

Table 3: NPV (\$000) Sensitivity (taxed) - Discount Rate versus Gold Price.

		Discount Rate (%)			
		20	15	10	5
Gold Price	\$ 1500/oz	252,370	321,854	426,598	592,138
	\$ 1400/oz	221,387	282,302	374,103	519,144
	\$ 1300/oz	190,404	242,749	321,608	446,150
	\$ 1200/oz	159,420	203,197	269,114	373,157
	\$ 1100/oz	128,437	163,644	216,619	300,163
	\$ 1000/oz	97,454	124,091	164,124	227,170

Table 4: NPV (\$000) Sensitivity (untaxed) - Discount Rate versus Gold Price.

On a NPV breakeven basis, the following results (Table 5) are achieved for both the taxed base case and the taxed (without Inferred Mineral Resources) scenarios:

	Unit	Base Case	Without Inferred
Gold Price	\$/oz	686	683
Grade	g/t Au	6	5.5
Operating costs	\$/oz	1,278	1263

Table 5: Breakeven Indicators without Inferred Resources.

These data show that due to the cash flow discounting effect, the elimination of the Inferred Mineral Resources does not have a major impact on the key breakeven indicators.

An alternative case was also analysed, which evaluates the Project's economics if all the Inferred Mineral Resource (equating to 27% of the tonnes and 25% of the contained gold metal in the base case production schedule) are eliminated from the base case. The results of this case, comparing the base cases with this alternative case (which does not include Inferred Mineral Resource material), are summarised below and shown in Table 6.

- EBITDA is decreased by 24% to \$521 million;
- the NPV in the taxed and untaxed cases are decreased by 15% and 14%, respectively;
- net cash flow in the taxed and untaxed cases are decreased by 25%;
- it produces a taxed and untaxed NPV at a 10% discount rate of \$187 million and \$277 million, respectively;
- royalties payable are decreased by 25%, whereas taxes payable are decreased by 23%;
- the production schedule is reduced in tonnage by 27% and the contained gold by 25%;
- the overall production grade increases by 7%; and
- mine life decreases to about 10.5 years (assuming the same production profile as the base case).

		With Inferred	Without Inferred	Difference
	Unit	Value	Value	%
Operating Cash Flow (EBITDA)	\$M	682.5	520.9	-24%
NPV ¹ (taxed)	\$M	218.2	186.5	-15%
NPV (untaxed)	\$M	321.6	277.1	-14%
Net cash flow (taxed)	\$M	440.9	329.8	-25%
Net cash flow (untaxed)	\$M	653.4	491.9	-25%
Unit operating cost (taxed)	\$/oz Au	855	849	-1%
Unit operating cost (untaxed)	\$/oz Au	658	647	-2%
Total capex	\$M	29.1	27.1	-7%
Royalties	\$M	65.6	49.2	-25%
Taxes	\$M	212.5	164.1	-23%

¹ at 10% base case discount rate

Table 6: Summary Results – Alternative Case.

1.7 Critical Risk Factors

It is the conclusion of the authors of this report that the PEA summarised herein contains adequate detail and information to support the potentially positive economic result. The PEA proposes the use of industry standard equipment and operating practices. To date, the authors are not aware of any fatal flaws for the project.

The most significant risks associated with the Project are commodity price risk, production risk due to smaller scale mining methods, labour intensive mining methods, and risks of changes to regulatory requirements. These risks are common to most mining projects in South America.

Whilst this PEA is conceptual in nature, a feasibility study will provide greater weight to the risk assessment process, provide a basis for their rating, as well as define a management plan to either mitigate or ameliorate such risks. In time, it is also usual for many of these risks to be reduced as the mine achieves its steady state capacity and operations proceed with greater reliability and become more predictable.

1.8 Conclusions and Recommendations

The Project has made significant progress with underground development, processing plant operations and achieving gold production and hence cash flow. Notwithstanding, the Project is now reaching a stage where it needs to carry out a feasibility study, which will provide the operations with greater certainty about the tonnes and grade of material to be processed, and

hence give greater confidence about potential gold output and therefore the generation of cash flow. It is also considered imperative that in the feasibility study stage, significant attention is given to mining method selection. This is due to the underground mine being a small-scale operation, in which case mining inefficiencies and ineffectiveness (cost per tonne stoped and factors such as dilution and mining recovery) will have adverse impacts on the Project's cash flow capability if not well managed.

The Company has numerous mineralised veins, some exhibiting very high-grade material that are available for exploration. This work, if adequately funded, is considered to have a high probability to provide longer-term feed to the processing operation based on sample results to date. The result will be the eventual development of a sustainable, higher grade mining operation.

Further, the general area is considered to be highly prospective for additional mineralised veins, again based on previous sampling results.

To this end, it is recommended that further exploration continues in order to determine potential new Mineral Resources from veins in the northern sector of the property and from extensions to known veins.

Surface mapping and sampling along with targeted drilling below the known mineralised outcrops should be undertaken to determine the grade characteristics of the veins with depth should continue.

Underground and surface oriented drill holes spaced 50m apart along the vein strike should be applied to aim for the Inferred Resources to be upgraded to Indicated and/or Measured. The same platform may then be used for two or more oriented drill holes with different angles (dips).

Costs to complete the advanced exploration work and feasibility study are currently estimated to be as follows:

- Underground mapping	\$ 50,000
- Surface trenching and mapping	\$ 50,000
- Drilling (approximately 8,100m)	\$ 1,650,000
- Sample assays	\$ 70,000
- Environmental control	\$ 50,000
- Social	\$ 70,000
- Feasibility Study.....	<u>\$ 1,000,000</u>
Total.....	\$ 2,940,000

2.0 Introduction

2.1 Introduction

Dynasty was incorporated under the *Business Corporations Act* (Yukon) in 2000. Its common shares are listed and posted for trading on the Toronto Stock Exchange under the symbol "DMM".

Dynasty has 41 concessions comprising a total Zaruma Project area of 10,121.47 ha. 38 concessions are 100% Dynasty owned. The other three concessions are "condominos": Barbasco Unificado is 25% Dynasty owned, Barbasco 1 A is 50% Dynasty owned and Soroche Unificado is 75% Dynasty owned. Five concessions are each subject to a 3% royalty under the conditions applicable to a 'small-scale mining licence'. These concessions comprise the current focus of mining at Zaruma. The Company has not applied for licences for the remaining concessions, some of which may include the resource.

The Zaruma Gold Project is located in the cantons of Zaruma and Portovelo, Province of El Oro, Ecuador and is centred at latitude 03°41'20" south and longitude 79°36'31" west (see Figures 5 and 6).

Access to the Zaruma Gold Project is primarily by paved roads to the general area and thence via tracks on steep hills in certain parts. Zaruma and Portovelo are towns and each has a population of about 20,000 people, most of whom are dependent on small near surface gold mining operations. The towns have a strong underground mining culture.

No preliminary feasibility study has been undertaken on the Project. This PEA is preliminary in nature and it includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorised as Mineral Reserves, and there is no certainty that the PEA will be realized.

2.2 Terms of Reference

The writers have been requested by the directors of Dynasty to update the 2006 Report which involves the mining and treatment of the mineral resources within the traditionally gold and polymetallic Zaruma-Portovelo mining district in volcanics of the Cordillera Occidental of Andes mountain range.

2.3 Report Purpose

This report is to be submitted to applicable Canadian securities regulatory authorities and is to be used by the Company to potentially raise funds for the preparation of a feasibility study and further exploration.

2.4 Field Involvement

Author, RL Procter, visited the Zaruma Gold Project site from February 22 to 25, 2014 in company with project geologist Luis Bravo and other Dynasty personnel. This visit included

accessing the underground mining, processing and infrastructure operations. Discussions and investigation took place in respect of the mining plans, processing plant operations and their performance (including tailings material disposal), gold security systems as well as reviewing general operational health, safety and environment processes and management procedures.

The Company has in place its own workforce and all mining and processing plant activities are carried out by Dynasty employed personnel.

The field trip gave the author a detailed appreciation of the Company's position vis-à-vis the local towns with which it is associated (Zaruma and Portovelo) and the availability of infrastructure. It also provided the author with a good overview of the the Zaruma / Portovelo geological landscape in relation to the gold deposits being exploited, and the Company's likely requirements in the future for mining and material processing operations.

Co-author AJ Maynard has made a site trip to the Zaruma Gold Project area on four separate occasions. The most recent was in March 2013. On this most recent site trip underground inspections of the workings were conducted by the mine manager explaining progress and planned works. Discussions were held in the site office and geological maps and plans were reviewed along with Dynasty sample locations and methods.

2.5 Qualified Person Responsibilities

The Qualified Persons (QPs) preparing this technical report are specialists in the fields of geology, exploration, mineral resource and mineral reserve estimation and classification, geotechnical, environmental, permitting, metallurgical testing, mineral processing, processing design, capital and operating cost estimation, and mineral economics.

None of the QPs or associates employed in the preparation of this report have any beneficial interest in Dynasty. The QPs are not insiders, associates, or affiliates of Dynasty. The results of this technical report are not dependent upon any prior agreements concerning the conclusions to be reached, nor are there any undisclosed understandings concerning any future business dealings between Dynasty and the QPs. The QPs are being paid a fee for their work in accordance with normal professional consulting practice.

The authors of this report, RL Procter, P.A. Jones and A.J. Maynard, by virtue of their education, experience and professional association, are considered independent QPs as defined by NI 43-101, and are members in good standing of appropriate professional institutions.

2.6 Units, Currency & Rounding

Unless otherwise specified or noted, the units used in this technical report are metric. Every effort has been made to clearly display the appropriate units being used throughout this technical report. Currency is in United States dollars (\$) unless otherwise noted.

This report includes technical information that required subsequent calculations to derive subtotals, totals and weighted averages. Such calculations inherently involve a degree of

rounding and consequently introduce a margin of error. Where these occur, the QPs do not consider them to be material.

2.7 Sources of Information and Data

The sources of information for this report are:

- information provided by Dynasty staff, in particular those involved with the mining, processing and environmental aspects of the Project;
- previous NI 43-101 documentation, primarily the 2006 Report; and
- historic reports detailed in the section entitled References.

3.0 Reliance on other Experts

In preparing this Technical Report, the authors have relied substantially upon detailed discussions with staff, consultants, and directors of Dynasty with respect to legal, political and environmental information compiled by them in regards to title and ownership information at Items 3.0 and 5.0, Item 18.0 in respect of contractual obligations and Item 19.0 in respect of environmental studies and other matters. The information is considered by the authors to be accurate. In particular, the authors have not personally reviewed the registered ownership or option agreements covering the property and claims of Dynasty known as the Zaruma Gold Project.

The authors consider this Report it to be a true and accurate representation of the potential viability of the Zaruma Gold Project.

When referring to the 2006 Report, which incorporates and confirms the results of the 2004 Report and 2005 Report, the authors have no reason to believe that there have been any material changes in the information supplied therein.

4.0 Property Description and Location

4.1 Property Description and Location

Dynasty has 41 concessions with a total of 10,121.47 ha. 38 of the concessions are 100% Dynasty owned while three concessions are "condominos": Barbasco Unificado is 25% Dynasty owned, Barbasco 1 A is 50% Dynasty owned and Soroche Unificado is 75% Dynasty owned.

The Zaruma Gold Project is located in the cantons of Zaruma and Portovelo, Province of El Oro, Ecuador and is centred at latitude 03°41'20" south and longitude 79°36'31" west (Figures 5 and 6). The Project is situated about 175 kilometres south and 60 kilometres southeast of the major port cities of Guayaquil and Machala, respectively (Figure 6).

Access to the Zaruma Gold Project is primarily by paved roads to the general area and thence via tracks on steep hills in certain parts.

Mineral rights are held 100% by Dynasty. For details of individual concessions owned and/or controlled by the Company see Table 7, below.

At the project site, hill slopes are moderately steep to very steep with elevations ranging from 1,040 to 1,600 m above sea level. Being traditionally a mining district, areas which are not disturbed by mining activity are used for farmlands, grazing and local minor secondary forestry. Dynasty's decline mining developments are underground and consequently will involve minimal disruption to current surface activities.

The climate is subtropical and humid with temperatures ranging from 18° to 30° C. Yearly rainfall averages 1,341 mm.

4.2 Environmental Impact

The agreement by which IAMGOLD transferred the Zaruma project areas to Dynasty contains a declaration by IAMGOLD that its work on the concession was done in such a manner as not to result in any damages that would require restitution to the state or to third parties.

Mining activities within the project area are limited to rustic hand mining using explosives, milling of the mineralized material in a small mechanized "Chilean Mill", gravity concentration of the milled material in concrete settling channels and final concentration by hand panning. Gold is extracted from the resultant gravity concentrate by mercury amalgamation. Several plants on the site where gold is extracted from mineralized material use a cyanide process. Both the mercury amalgamation process and the cyanide extraction process have the potential to release harmful chemicals into the environment.

The Company has an Environmental Impact Assessment (EIA) approved by the Ecuador Environmental Authority based on report OFTZ-MAE-UCA-2013-0125, submitted via memorandum MAE-UCAEO DPAEO-2013-0284 dated May 20, 2013, that determined that

the processing plant meets the technical and legal requirements established in current environmental regulations.

For the report, the area of influence for physical impact occupies a strip of land 500 m wide around the perimeter of the project facilities while the impact on water is defined by 3 km downstream in the Calera River.

Most of the study area is represented by a degraded grassland pastures composed of dry and eroded *Pennisetum purpureum* (elephant grass), *Panicum maximum* (Chilean grass) and *Aristida* sp. (Grass pointer). These pastures are of poor quality and have limited value for livestock.

Native vegetation is restricted to the steeper slopes, some margins of streams and rivers that have not been affected by clearing. These sites may be considered as being sensitive areas that should be preserved for natural recovery within a framework of conservation and protection of the flora and fauna resources.

No sensitive wildlife ecosystems are recorded within the study area.

The nearest population center to the project is located approximately 5km to the east at the City of Portovelo. There are no traditional or ancestral, indigenous communities with anthropological roots or cultural values that identifies with their race, values and customs within the study area.

El Oro Province is one of the areas undergoing archaeological study in recent years. The area is considered to be of low archaeological sensitivity.

The determination of compliance or non-compliance in the environmental audit was based on the verification of each of the programs by the current environmental management plan, laws, regulations and other applicable legal environmental parameters. This mainly concentrated on the operation of the tailing dam and related activities (detoxification plant and ancillary infrastructure) taking into consideration existing environmental conditions.

The Environmental Management Plan is aimed at mitigating any impacts that may affect the environment by conserving the natural environment and remaining in harmony with the socio-economic development of villages influenced by the operation of the Plant.

This Management Plan consists of the following programs:

Program for the Prevention, Mitigation and Control of Environmental Pollution:

- Management Discharge Program
- Chemical Management Program
- Management flora and fauna, Revegetation Site Program
- Waste Management Program
- Occupational Health and Safety Program
- Signage Program
- Emergency and Contingency Program

- Training Program
- Noise Monitoring, Air, Water, Soil and Sediment Program
- Community Relations Program
- Program to Monitor Environmental Management Plan
- Closure and Abandonment Program

The implementation of these programs will be carried out through its technical, administrative staff and field workers.

There is no issue that restricts the Company to extract its material. The only one limit relates to the small-scale mining concessions permit, which limits mining production to 300 tonnes per day.

4.3 Royalties

A royalty of 3.0% of the total gold revenue after deduction of non-payable metal, transport and refining charges is paid to the Ecuadorian Government. A further royalty calculated at 1.5% of net value after deduction of non- payable metal, transport and refining charges is paid to IAMGOLD.

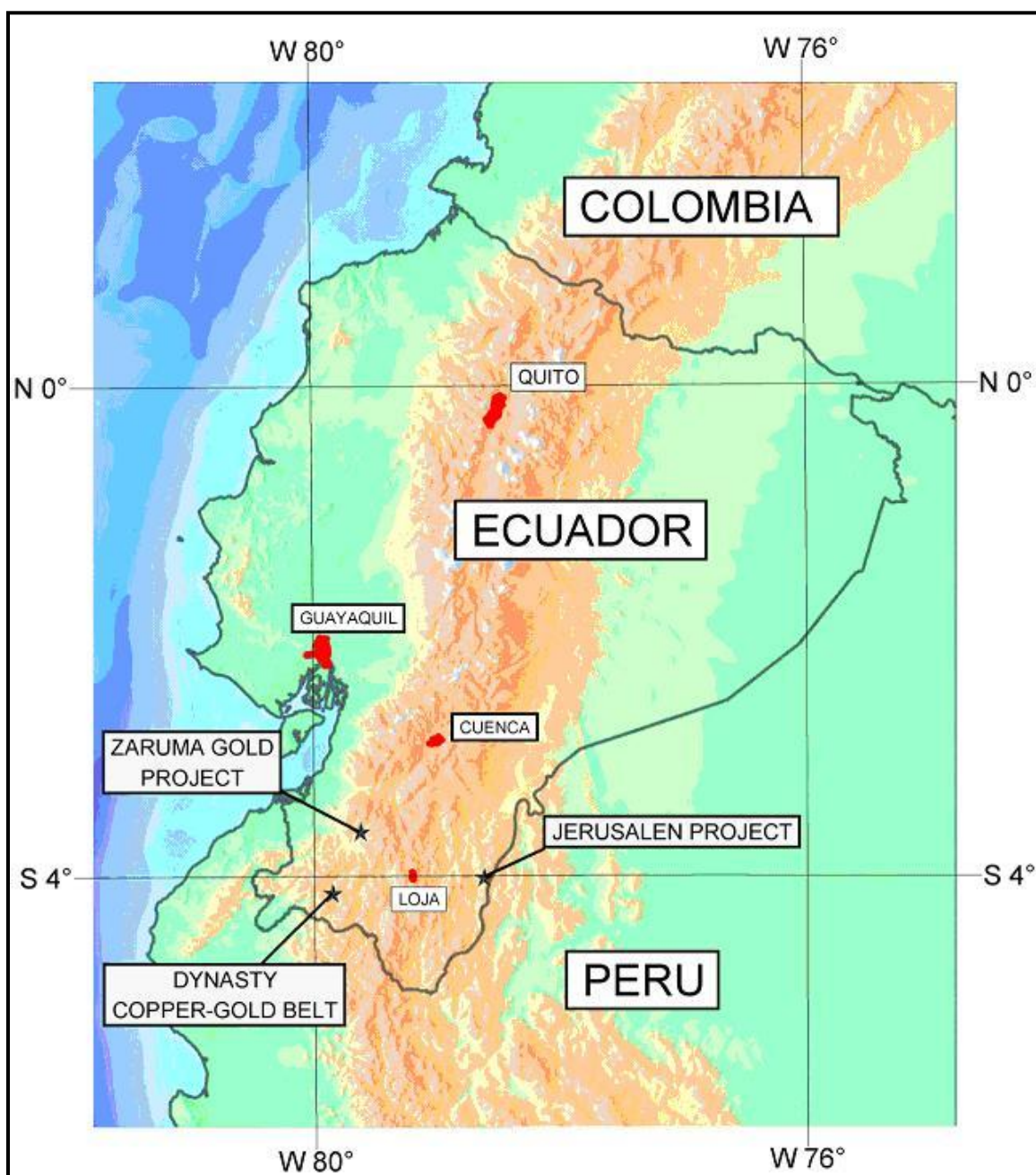


Figure 5: Location Map.

4.4 Tenure

The list of concessions owned and/or controlled by the Company is listed in Table 7 and further described at Table 8, and shown in Figure 6. The property has not been legally surveyed since tenement (Concession) boundaries are based on map grid coordinates, however 465 points with precision better than 0.1m have been surveyed by IAMGOLD employing differential GPS instruments including mine entrances, starting and ending points of major trenches, drilling platforms and collars which can be considered and used as appropriate geographical control points. Additionally, 2126 points with precision equal or better than 0.1m have been surveyed by IAMGOLD in underground as shown on Figure 7.

N°	Concession Name	Code	Area Hectares
1	NUEVA ESPERANZA	152	5
2	PACAYURCO BAJO	189	9
3	BARBASCO UNIFICADO ⁽¹⁾	297	10
4	SAN JOSÉ 2	385	5
5	BARBASCO 1A ⁽²⁾	447	77.24
6	LILLY RAI	473	7.76
7	SOROCHE UNIFICADO ⁽³⁾	506	32.8
8	AGUILA DE ORO	3059	10
9	EL SALVADOR X-3	5012	15
10	ZUDOS 1	5200	323.47
11	SALVADOR X-1-A	5213	57
12	NUEVA ESPERANZA 2	300085	42
13	NUEVA ESPERANZA 4	300086	135.66
14	NUEVA ESPERANZA 3	300089	30
15	LA DURA	300101	129.55
16	NUEVA ESPERANZA 6	300195	77.8
17	SAN ANTONIO DE PADUA	300210	67
18	ROMA	300216	661.92
19	ARCAPAMBA 2	300273	15
20	SAICHUMA 1	300274	466
21	LOS CIPRESES	300302	2394
22	MACHAY	300320	1273
23	LA CALERA	300321	890
24	BETHZABETH	300350	1280
25	EL RETAZO 3	300391	32
26	NICOLE	300444	8.48
27	ANA MICHELLE	300445	20
28	MALVAS 1	300462	480
29	IAM ZARUMA	300477	2.5
30	SUCA	300519	3.8
31	SUCA 4	300523	1
32	MARA 8	300537	5

(continued on following page)

N°	Concession Name	Code	Area Hectares
33	SOLEMARIA	300545	23.81
34	LOS LAURELES 2	300558	100
35	LOMA RICA 1	300559	19.58
36	RESUC 4	300579	3.5
37	RUTH	300580	2
38	LORO	300604	9.6
39	LA ENVIDIA	300630	1346
40	EL TABLON	300861	42
41	EL TABLON 1	300862	8

Notes: (1) Dynasty equity 25%
 (2) Dynasty equity 50%
 (3) Dynasty equity 75%

Table 7: Listing of Zaruma Concessions.
(Yellow highlighted are in Exploitation Phase, remainder are Exploration Phase)

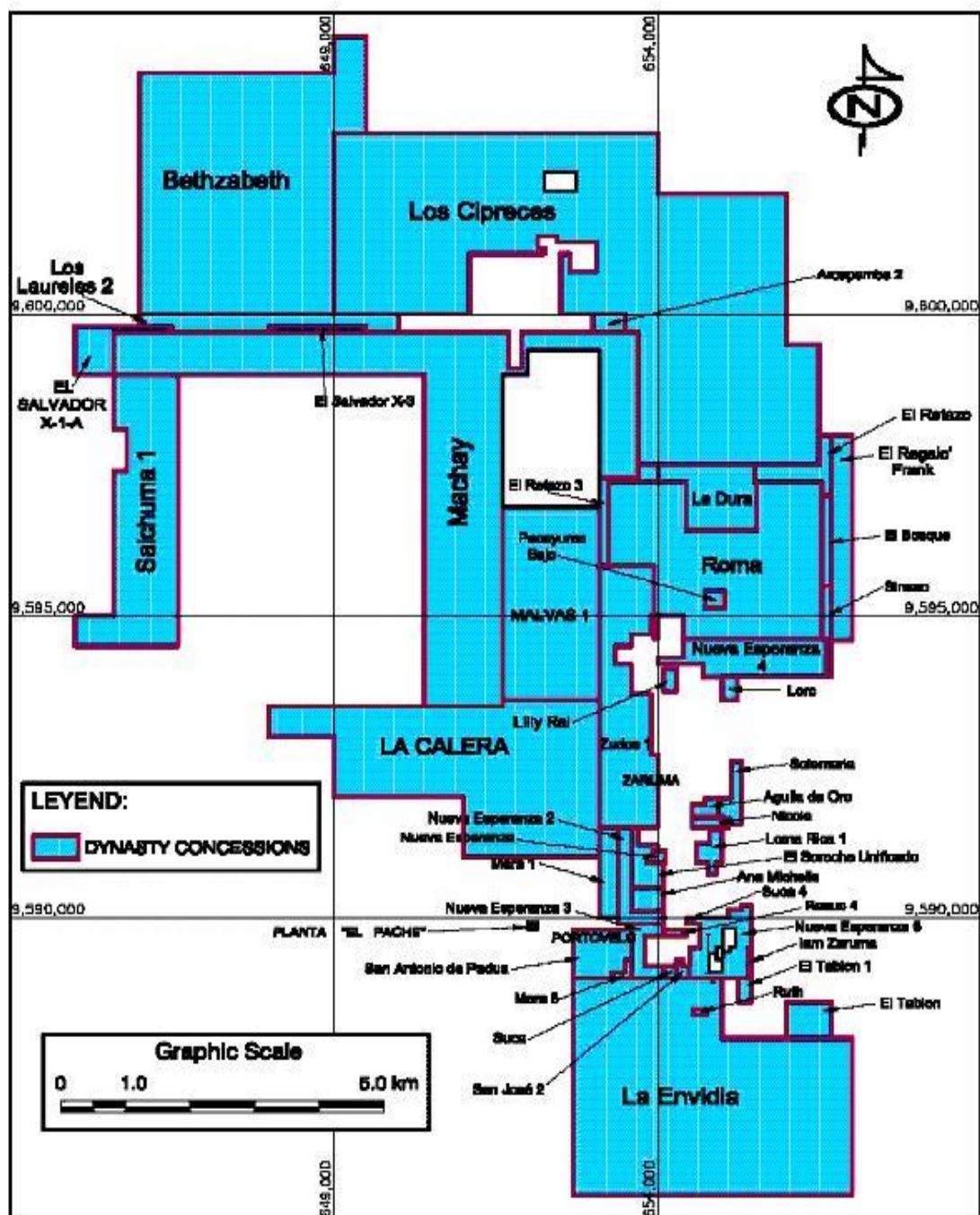


Figure 6: Land Tenements Owned or Controlled by Dynasty Metals & Mining Inc.

Concession Name	Authorisation Date	Registration Date	Expiry Date	Concession Limits	Easting UTM PSAD56	Northing UTM PSAD 56
AGUILA DE ORO	July 9, 2001	July 10, 2001	July 10, 2031	Initial point Point 1 Point 2 Point 3 Point 4 Point 5	654,535 654,735 654,735 654,935 654,935 654,535	9,591,878 9,591,878 9,591,978 9,591,978 9,591,678 9,591,678
ANA MICHELLE	Nov. 6, 2001	Dec. 1, 2001	Dec. 1, 2031	Initial point Point 1 Point 2 Point 3	653,600 653,600 654,100 654,100	9,590,100 9,590,500 9,590,500 9,590,100
ARCAPAMBA 2	Sep. 12, 2001	Oct. 1, 2001	Oct. 1, 2031	Initial point Point 1 Point 2 Point 3	653,000 653,500 653,500 653,000	9,600,000 9,600,000 9,599,700 9,599,700
BETHZABETH	Aug. 6, 2001	Aug. 16, 2001	Aug. 16, 2031	Initial point Point 1 Point 2 Point 3 Point 4 Point 5 Point 6 Point 7	649,000 649,000 646,000 646,000 649,000 649,000 649,500 649,500	9,603,000 9,600,000 9,600,000 9,604,000 9,604,000 9,604,600 9,604,600 9,603,000
EL BOSQUE	Apr. 4, 2002	Jul. 9, 2001	Jul. 9, 2031	Initial point Point 1 Point 2 Point 3	656,700 656,700 656,538 656,538	9,597,000 9,595,500 9,595,500 9,597,000
EL REGALO FRANK	Sep. 18, 2001	Oct. 1, 2001	Oct. 1, 2031	Initial point Point 1 Point 2 Point 3	656,700 657,000 657,000 656,700	9,598,000 9,598,000 9,594,600 9,594,600
EL RETAZO	Apr. 4, 2002	Oct. 1, 2001	Oct. 1, 2031	Initial point Point 1 Point 2 Point 3 Point 4 Point 5 Point 6 Point 7	656,700 656,538 656,538 655,500 655,500 656,500 656,500 656,700	9,597,000 9,597,000 9,597,223 9,597,223 9,597,500 9,597,500 9,598,000 9,598,000
EL RETAZO 3	Sep. 18, 2001	Oct. 1, 2001	Oct. 1, 2031	Initial point Point 1 Point 2 Point 3 Point 4 Point 5 Point 6 Point 7	653,700 654,000 654,000 653,238 653,238 653,100 653,100 653,700	9,597,500 9,597,500 9,597,223 9,597,223 9,595,823 9,595,823 9,597,300 9,597,300
EL SALVADOR X-3	Oct. 17, 2001	Oct. 17, 2001	Oct. 24, 2031	Initial point Point 1 Point 2 Point 3	649,500 649,500 648,000 648,000	9,599,800 9,599,700 9,599,700 9,599,800

Concession Name	Authorisation Date	Registration Date	Expiry Date	Concession Limits	Easting UTM PSAD56	Northing UTM PSAD 56
EL TABLON	Aug. 21, 2002	Sep. 2, 2002	Sep. 2, 2032	Initial point Point 1 Point 2 Point 3	655,250 655,450 655,450 656,700	9,589,000 9,589,000 9,588,600 9,588,000
EL TABLON 1	Oct. 14, 2002	Oct. 22, 2002	Oct. 22, 2032	Initial point Point 1 Point 2 Point 3	655,250 655,450 655,450 655,250	9,589,000 9,589,000 9,588,600 9,588,600
IAM ZARUMA	July 5, 2001	July 9, 2001	July 9, 2031	Initial point Point 1 Point 2 Point 3	655,400 655,450 655,450 655,400	9,589,500 9,589,500 9,589,000 9,589,000
LA DURA	July 10, 1998	July 9, 2001	July 9, 2031	Initial point Point 1 Point 2 Point 3 Point 4 Point 5 Point 6 Point 7	654,000 655,500 655,500 655,538 655,538 654,438 654,438 654,000	9,597,500 9,597,500 9,597,223 9,597,223 9,596,423 9,596,423 9,597,223 9,597,223
LA ENVIDIA	Sep. 18, 2001	Oct. 2, 2001	Oct. 2, 2031	Initial point Point 1 Point 2 Point 3 Point 4 Point 5	652,700 652,700 655,000 655,000 657,000 657,000	9,585,400 9,589,000 9,589,000 9,588,000 9,588,000 9,585,400
LOMA RICA 1	Sep. 18, 2001	Oct. 1, 2001	Oct. 1, 2031	Initial point Point 1 Point 2 Point 3 Point 4 Point 5 Point 6 Point 7 Point 8 Point 9	654,600 654,800 654,800 655,000 655,000 654,900 654,900 654,900 654,782 654,782 654,600	9,591,225 9,591,225 9,591,450 9,591,450 9,590,907 9,590,907 9,590,707 9,590,707 9,590,907 9,590,907
LOS CIPRECES	April 4, 2002	Oct. 5, 2001	Oct. 5, 2031	Initial point Point 1 Point 2 Point 3 Point 4 Point 5 Point 6 Point 7 Point 8 Point 9 Point 10 Point 11 Point 12 Point 13 Point 14 Point 15 Point 16 Point 17 Point 18	649,000 654,000 654,000 656,000 656,000 656,500 656,500 653,700 653,700 653,500 653,500 652,500 652,500 652,600 652,600 653,100 653,100 652,500 652,500	9,603,000 9,603,000 9,602,000 9,602,000 9,599,500 9,599,500 9,597,500 9,597,500 9,599,700 9,599,700 9,600,000 9,600,000 9,601,000 9,601,000 9,600,700 9,600,700 9,601,200 9,601,200 9,601,300

Concession Name	Authorisation Date	Registration Date	Expiry Date	Concession Limits	Easting UTM PSAD56	Northing UTM PSAD 56
				Point 19 Point 20 Point 21 Point 22 Point 23 Point 24 Point 25	652,100 652,100 652,200 652,200 651,100 651,100 649,000	9,601,300 9,601,100 9,601,100 9,601,000 9,601,000 9,600,000 9,600,000
LOS LAURELES	Oct. 23, 2001	Oct. 24, 2001	Oct. 24, 2031	Initial point Point 1 Point 2 Point 3 Point 4 Point 5 Point 6 Point 7 Point 8 Point 9	650,000 650,000 649,500 649,500 648,000 648,000 646,500 646,500 646,000 646,000	9,600,000 9,599,700 9,599,700 9,599,800 9,599,800 9,599,700 9,599,700 9,599,800 9,599,800 9,600,000
LORO	April 4, 2002	Oct. 1, 2001	Oct. 1, 2031	Initial point Point 1 Point 2 Point 3	655,240 655,240 655,000 655,000	9,594,000 9,539,600 9,593,600 9,594,000
MARA 1	Oct. 19, 2001	Oct. 22, 2001	Oct. 22, 2031	Initial point Point 1 Point 2 Point 3	653,100 653,100 653,400 653,400	9,590,000 9,591,500 9,591,500 9,590,000
MARA 8	Oct. 19, 2001	Oct. 22, 2001	Oct. 22, 2031	Initial point Point 1 Point 2 Point 3 Point 4 Point 5	653,600 653,600 653,300 653,300 653,500 653,500	9,589,300 9,589,000 9,589,000 9,589,100 9,589,100 9,589,300
NICOLE	May 7, 2003	May 16, 2003	May 16, 2033	Initial point Point 1 Point 2 Point 3	655,100 655,100 654,535 654,535	9,591,650 9,591,500 9,591,500 9,591,650
RESUC 4	Oct. 19, 2001	Oct. 22, 2001	Oct. 22, 2031	Initial point Point 1 Point 2 Point 3	654,100 654,100 654,450 654,450	9,589,700 9,589,800 9,589,800 9,589,700
SAICHUMA 1	Sep. 12, 2001	Sep. 25, 2001	Sep. 25, 2031	Initial point Point 1 Point 2 Point 3 Point 4 Point 5 Point 6 Point 7 Point 8 Point 9	645,000 645,600 645,600 645,800 645,800 645,600 645,600 646,600 646,600 645,000	9,595,000 9,595,000 9,597,400 9,597,400 9,598,100 9,598,100 9,599,000 9,599,000 9,594,500 9,594,500
SAN ANTONIO	July 9, 2001	July 10, 2001	July 10, 2031	Initial point Point 1 Point 2	652,700 653,600 653,600	9,589,800 9,589,800 9,589,300

Concession Name	Authorisation Date	Registration Date	Expiry Date	Concession Limits	Easting UTM PSAD56	Northing UTM PSAD 56
DE PAUDA				Point 3 Point 4 Point 5 Point 6 Point 7	653,500 653,500 653,300 653,300 652,700	9,589,300 9,589,100 9,589,100 9,589,000 9,589,000
SINSAO	April 4, 2002	Oct. 1, 2001	Oct. 1, 2031	Initial point Point 1 Point 2 Point 3 Point 4 Point 5	656,700 656,600 656,600 656,538 656,538 656,700	9,594,000 9,594,000 9,594,623 9,594,623 9,595,500 9,595,500
RUTH	Sep. 18, 2001	Oct. 1, 2001	Oct. 1, 2031	Initial point Point 1 Point 2 Point 3	654,554 654,754 654,754 654,554	9,588,481 9,588,481 9,588,381 9,588,381
SOLEMARIA	April 4, 2002	April 18, 2002	April 18, 2032	Initial point Point 1 Point 2 Point 3 Point 4 Point 5 Point 6 Point 7 Point 8 Point 9	655,300 655,300 655,100 655,100 655,000 655,000 654,935 654,935 655,135 655,135	9,592,585 9,591,550 9,591,550 9,591,650 9,591,650 9,591,678 9,591,678 9,591,978 9,591,978 9,592,585
SUCA	Oct. 19, 2001	Oct. 22, 2001	Oct. 22, 2031	Initial point Point 1 Point 2 Point 3	654,100 654,290 654,290 654,100	9,589,200 9,589,200 9,589,000 9,589,000
ZUDOS 1	July 5, 2001	July 9, 2001	July 9, 2031	Initial point Point 1 Point 2 Point 3 Point 4 Point 5 Point 6 Point 7 Point 8 Point 9 Point 10 Point 11 Point 12 Point 13 Point 14 Point 15	653,100 653,100 653,938 653,938 653,900 653,900 653,600 653,600 653,350 653,350 653,600 653,600 653,900 653,900 654,000 654,000	9,591,500 9,595,823 9,595,823 9,595,000 9,595,000 9,594,700 9,594,700 9,594,500 9,594,500 9,594,200 9,594,200 9,593,700 9,593,700 9,592,700 9,592,700 9,591,500
ROMA	June 10, 2003	Aug. 6, 2001	Aug. 6, 2031	Initial point Point 1 Point 2 Point 3 Point 4 Point 5 Point 6 Point 7 Point 8	656,538 654,400 654,400 654,000 654,000 653,938 653,938 653,238 653,238	9,594,623 9,594,623 9,595,000 9,595,000 9,594,623 9,594,623 9,595,823 9,595,823 9,597,223

Concession Name	Authorisation Date	Registration Date	Expiry Date	Concession Limits	Easting UTM PSAD56	Northing UTM PSAD 56
				Point 9 Point 10 Point 11 Point 12 Point 13 Point 14 Point 15 Point 16 Point 17	654,438 654,438 655,538 655,538 656,538 655,038 655,038 654,738 654,738	9,597,223 9,596,423 9,596,423 9,597,223 9,597,223 9,595,423 9,595,123 9,595,123 9,595,423
SUCA 4	Oct. 19, 2001	Oct. 30, 2001	Oct. 30, 2031	Initial point Point 1 Point 2 Point 3	654,450 654,550 654,550 654,450	9,590,000 9,590,000 9,589,900 9,589,900
PACAYURCO BAJO	June 11, 2003	Aug. 6, 2001	Aug. 6, 2031	Initial point Point 1 Point 2 Point 3	655,038 655,038 654,738 654,738	9,595,423 9,595,123 9,595,123 9,595,423
NUEVA ESPERANZA 3	July 6, 2001	July 9, 2001	July 9, 2031	Initial point Point 1 Point 2 Point 3 Point 4 Point 5 Point 6 Point 7	653,600 654,100 654,100 653,800 653,800 654,100 654,100 653,600	9,589,900 9,589,900 9,589,700 9,589,700 9,589,200 9,589,200 9,589,000 9,589,000
NUEVA ESPERANZA 6	July 6, 2001	July 9, 2001	July 9, 2031	Initial point Point 1 Point 2 Point 3 Point 4 Point 5 Point 6 Point 7 Point 8 Point 9 Point 10 Point 11 Point 12 Point 13 Point 14 Point 15 Point 16 Point 17 Point 18 Point 19 Point 20 Point 21 Point 22 Point 23 Point 24 Point 25 Point 26 Point 27 Point 28 Point 29 Point 30 Point 31	654,490 654,490 654,500 654,500 654,650 654,650 654,550 654,550 655,100 655,100 655,300 655,300 655,450 655,450 655,400 655,400 655,015 655,215 655,215 655,115 655,115 655,015 654,915 655,015 655,015 654,915 654,800 654,900 654,900 655,000 655,000	9,589,000 9,589,200 9,589,200 9,589,500 9,589,500 9,589,900 9,589,900 9,590,000 9,590,000 9,590,130 9,590,130 9,590,200 9,590,200 9,589,500 9,589,500 9,589,000 9,589,824 9,589,824 9,589,524 9,589,524 9,589,424 9,589,424 9,589,524 9,589,524 9,589,324 9,589,324 9,589,400 9,589,400 9,589,300 9,589,300 9,589,100

Concession Name	Authorisation Date	Registration Date	Expiry Date	Concession Limits	Easting UTM PSAD56	Northing UTM PSAD 56
					654,800	9,589,100
NUEVA ESPERANZA 4	July 5, 2001	July 9, 2001	July 9, 2031	Initial point Point 1 Point 2 Point 3 Point 4 Point 5 Point 6 Point 7 Point 8 Point 9	654,700 654,700 654,000 654,000 654,400 654,400 656,600 656,600 655,000 655,000	9,593,980 9,594,200 9,594,200 9,594,300 9,594,300 9,594,623 9,594,623 9,594,000 9,594,000 9,593,980
NUEVA ESPERANZA	July 6, 2001	July 9, 2001	July 9, 2031	Initial point Point 1 Point 2 Point 3 Point 4 Point 5	654,135 653,835 653,835 653,935 653,935 654,135	9,590,910 9,590,910 9,591,010 9,591,010 9,591,110 9,591,110
NUEVA ESPERANZA 2	July 9, 2001	July 10, 2001	July 10, 2031	Initial point Point 1 Point 2 Point 3 Point 4 Point 5	653,600 654,100 654,100 653,400 653,400 653,600	9,590,100 9,590,100 9,589,900 9,589,900 9,591,500 9,591,500
SAN JOSE 2	July 6, 2001	July 9, 2001	July 9, 2031	Initial point Point 1 Point 2 Point 3 Point 4 Point 5	654,290 654,390 654,390 654,490 654,490 654,290	9,589,300 9,589,300 9,589,200 9,589,200 9,589,000 9,589,000
LILLY RAY	Oct. 17, 2001	Oct. 24, 2001	Oct. 24, 2031	Initial point Point 1 Point 2 Point 3 Point 4 Point 5	654,100 654,300 654,300 654,290 654,290 654,100	9,594,130 9,594,130 9,593,784 9,593,784 9,593,740 9,593,740
EL SOROCHE UNIFICADO	July 6, 2001	July 10, 2001	July 10, 2031	Initial point Point 1 Point 2 Point 3 Point 4 Point 5 Point 6 Point 7 Point 8 Point 9 Point 10 Point 11	654,100 653,600 653,600 653,700 653,700 654,000 654,000 653,935 653,935 653,835 653,835 654,100	9,590,500 9,590,500 9,591,500 9,591,500 9,591,200 9,591,200 9,591,110 9,591,110 9,591,010 9,591,010 9,590,910 9,590,910
MACHAY	Sep. 12, 2001	Oct. 1, 2001	Oct. 1, 2031	Initial point Point 1 Point 2 Point 3 Point 4 Point 5 Point 6	651,600 651,600 650,400 650,400 645,600 645,600 651,700	9,599,000 9,593,500 9,593,500 9,599,000 9,599,000 9,599,700 9,599,700

Concession Name	Authorisation Date	Registration Date	Expiry Date	Concession Limits	Easting UTM PSAD56	Northing UTM PSAD 56
				Point 7	651,700	9,599,100
				Point 8	651,900	9,599,100
				Point 9	651,900	9,599,700
				Point 10	653,700	9,599,700
				Point 11	653,700	9,597,300
				Point 12	653,100	9,597,300
				Point 13	653,100	9,599,400
				Point 14	652,000	9,599,400
				Point 15	652,000	9,599,000

Table 8: Zaruma Project Land Tenure.

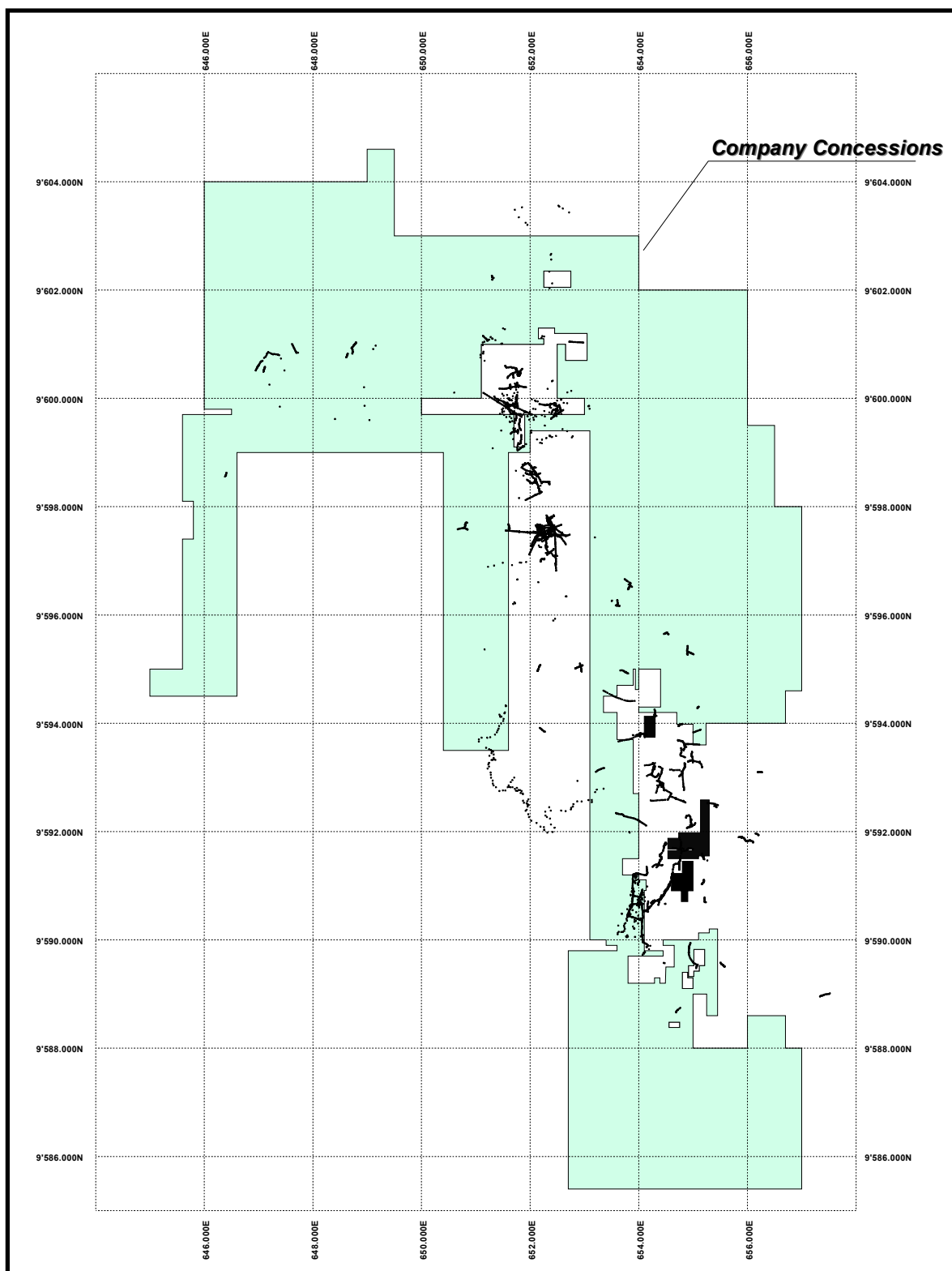


Figure 7: IAMGOLD ground and underground survey control point locations
(A total of 2,591 points are plotted on the map)

5.0 Accessibility, Climate, Local Resources, Infrastructure and Physiography

5.1 Location

The Zaruma Gold Project is situated in El Oro Province of southern Ecuador, about 175 kilometres south and 60 kilometres southeast of the major port cities of Guayaquil and Machala respectively (Figure 6). The centre of the property is located on 654,500 East and 9,592,000 North (UTM PSAD-56, zone 17S) (Provisional South American 1956 projection) or longitude 79° 36' 31" West and latitude 3° 41' 20" South.

5.2 Access

Good asphalted roads provide access to the mining towns of Zaruma and Portovelo about 2 hours drive from the city of Machala. Access on the project area is facilitated by a relatively dense network of secondary roads. The project area is also traversed by a main highway to the city of Loja.

5.3 Climate, Physiography and Vegetation

The property lies on the western flank of the Cordillera Occidental of Andes mountain range. Hill slopes are moderately steep to very steep with elevations ranging from 1,040 to 1,600 m above sea level. Being traditionally a mining district, areas which are not disturbed by mining activity are used for farmlands, grazing and local minor secondary forestry. The climate is subtropical and humid with temperatures ranging from 18° to 30° C. Yearly rainfall averages 1,341 mm with heaviest rainfalls occurring in the months of January to June. In southern Ecuador the irregular topography and climate result in high species diversity and a large range of very different vegetation types in a relatively small area. Semi-evergreen lowland and premontane tall forest types are prevalent in the Zaruma area.

5.4 Local Resources

The location of two medium-sized cities, namely Zaruma with population of approximately 29,000 and Portovelo with population of approximately 14,000, within the Project area provide both the labour force and all other facilities a mining operation needs e.g. hotels, food supply, material supply, communication resources, public security and government institution representatives. A strong mining background for the district guarantees skilled workers for underground mining and the availability of basic mining-related material in the Project area. High-tension power lines providing electricity are connected to both Zaruma and Portovelo and the Rio Amarillo and Rio Calera rivers are able to supply adequate water for large-scale mining operations throughout the year.

6.0 History

6.1 General History

The mountains of Zaruma and Portovelo have been mined for gold and silver for centuries. The Incas were already extracting gold and silver in the area with hydraulic mining of the oxidized parts of veins when Mercadillo, one of Pizarro's force, followed the Rio Amarillo River upstream and encountered the Inca mine and founded the town of Zaruma in 1549 (Billingsley, 1926). Exploitation of the Zaruma and Portovelo districts continued during the time of Spanish colonization until 1870 when an Ecuadorian-Chilean company was established.

In 1880, Grant Zaruma Company, from England, bought controlling shares of the Ecuadorian-Chilean mining company. Operational rights were immediately endorsed to Southern American Development Company (SADCO), an American company, who operated the mine from 1897 to 1950 by gaining control of the district's main gold deposits in 1897. Exploration programs of SADCO commenced in 1896 and brought the mine into production at 108 t/day in 1905. The mine was subsequently deepened to 13 levels located at an elevation of 270 m above sea level and 800 m below the uppermost workings (PRODEMINCA, 2000 in Spencer, R.M. et al., 2002). In the 53 years that followed, SADCO recovered some 3.5 million ounces of gold and 12 million ounces of silver from 7.6 million tonnes of ore at a cut-off grade of 14.4 g/t Au and 48.9 g/t Ag (Van Thournout et al., 1996).

After the lower levels of the Portovelo mine were flooded in 1944 and facing increasing costs, taxes and a complicated political situation, SADCO finally withdrew from the country in 1950 and the Ecuadorian government took over the mine. A state-owned company, CIMA, took over the mining operations in the area until 1980 and it is estimated to have produced a further 375,000 oz of gold by 1965. In 1984, thousands of poverty-stricken miners invaded the old SADCO pits and small-scale and artisan mining has been going on in the area ever since. An additional 35,000 to 50,000 oz of gold has been produced each year since then by informal miners, small-scale operating mining societies and principally from the family-owned BIRA. Statistical information from the 1990s indicates that mining from the Zaruma and Portovelo areas totalled 3 million tonnes (INEMIN – CODIGEM, 1990, in Jemielita and Bolaños, 1993).

In the mid-1990s, several overseas companies attempted to consolidate the area and carried out systematic exploration programmes. In 1995, the Canadian-based TVX reached an agreement with most of the Zaruma area mining societies and informal miners and started with underground and surface mapping particularly within the Zaruma and Muluncay areas from March 1995 until February 1996.

After TVX withdrew from Ecuador in 1998, all information was acquired by IAMGOLD. IAMGOLD continued with more extensive exploration programmes including surface trenching, surface and underground sampling, surveying and diamond drilling. IAMGOLD significantly improved the Zaruma Gold Project area with legal consolidation of titles, adding valuable information to the project database, employing systematic geochemical data quality control and undertaking geological modelling.

In August 2003, all the IAMGOLD properties and the project database were transferred to Dynasty, which continued the exploration since then and is ongoing.



Figure 8: View of the Old SADC Vertical Shaft in Portovelo.

6.2 Regional Exploration History

Systematic exploration activity closely related to mining advancement was carried out by SADC from 1897 to 1950, however only limited information is available from that period. Detailed underground maps and 103,657 assay result records which have been rescued from local miners and from local archives by TVX and IAMGOLD are now in the possession of Dynasty.

Modern exploration activity within the Zaruma–Portovelo Mining District began in 1995 when a first attempt of property area consolidation and district-scale exploration was made by TVX, a Canadian-based company. For over one year, a crew of more than 10 geologists, GIS and support staff studied the property. During this period, over 40 km of underground workings were surveyed and mapped at 1:500 scale.

Valuable historic information, mostly SADC's maps and assay results, were rescued from local miners during this time. The total number of samples collected by TVX is difficult to estimate, however it is estimated that over 4,000 underground channel samples were taken based on daily reports of mapping geologists. Sampling databases submitted to Dynasty by IAMGOLD contain 733 samples, however several hundred sample locations and assay results were digitised from original maps by the Company's GIS staff.

Following TVX's withdrawal from Ecuador in 1998, all their information was acquired by IAMGOLD, which continued exploration, including surface trenching, surface and underground sampling, surveying and diamond drilling. IAMGOLD significantly improved the Zaruma Gold Project area legal consolidation, added valuable information to project databases and was the first company working within the area to employ geochemical data quality control and geological modelling. IAMGOLD databases contained:

680	Surface rocks channel and chip samples
2,126	Underground channel samples
5,415	Soil samples
37	Diamond drill hole results including sample assay results
1,114	DDH core samples
2,591	Topographical control points survey
39	Stream sediments samples
369	Channel samples form surface trenching

In August 2003, all the IAMGOLD properties and project databases were transferred to Dynasty who continued with exploration programmes up to the date of this Report.

The Company had three geologists re-sampling selected IAMGOLD and TVX underground sampling channels to verify the original assay results.

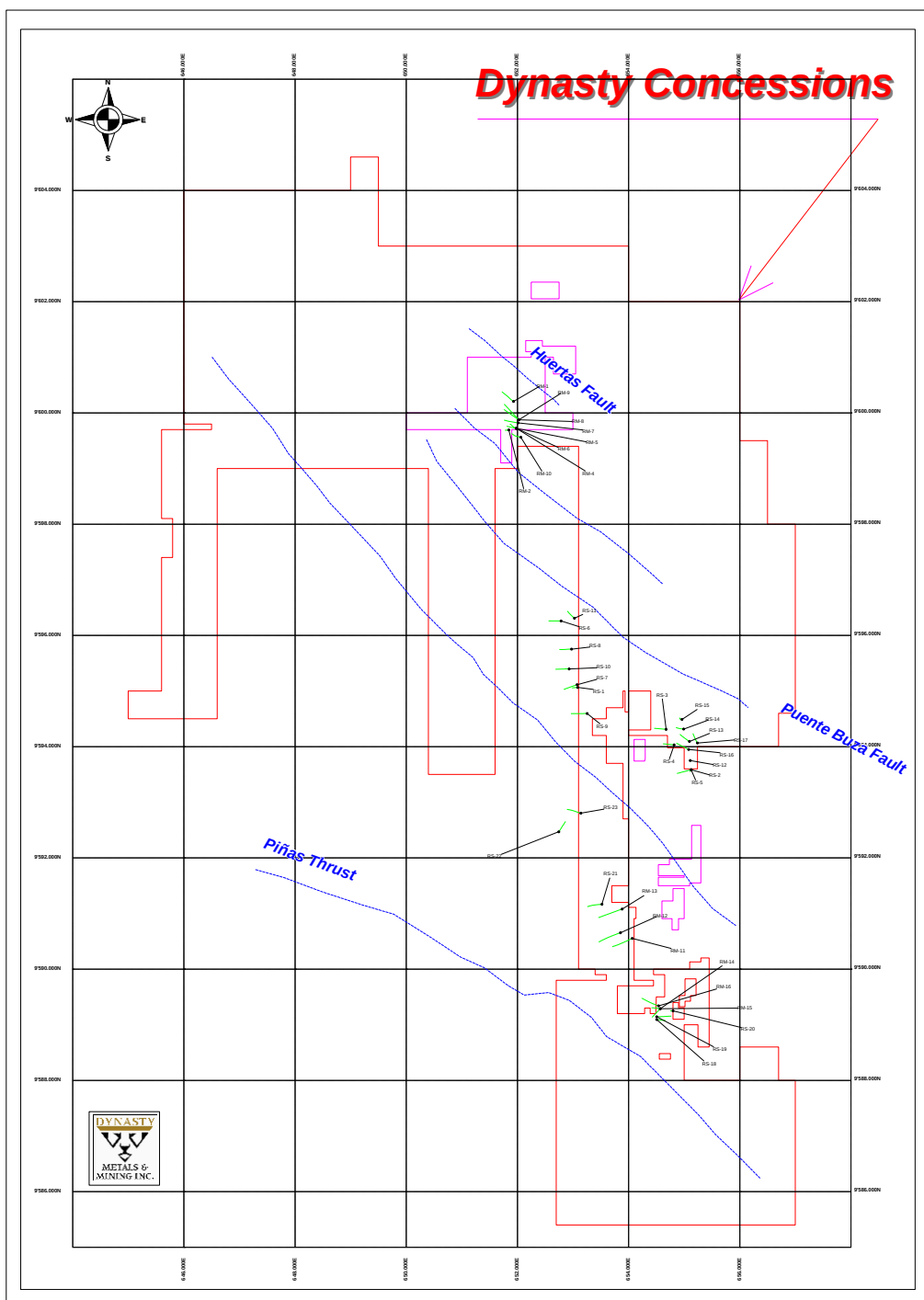


Figure 9: IAMGOLD Drilling Program and Zaruma Gold Project Tenements.

7.0 Geological Setting and Mineralization

7.1 Regional Geology

Subduction of the Pacific Ocean Nazca Plate beneath the South American continent has produced crustal deformation forming the Andes mountain range over the last 25 million years. The resultant compressional forces have produced several Tertiary and Quaternary tectonic basins along major regional faults that cut both the Pacific and Amazonian piedmonts and the Andean range itself.

The Zaruma Gold Project lies within the compressional Inter-Andean Graben that is bounded by regional scale faults. The graben is composed of thick Oligocene to Miocene volcano-sedimentary sequences that cover the Chaucha, Amotape and Guamote terrains. This structural zone hosts several significant epithermal, porphyry, mesothermal, S type granitoid, VHMS and ultramafic/ophiolite precious metal and base metal mineral deposits.

The general geological setting is described in Billingsly (1926), Van Thournout et al. (1996), and Spencer et al. (2002).

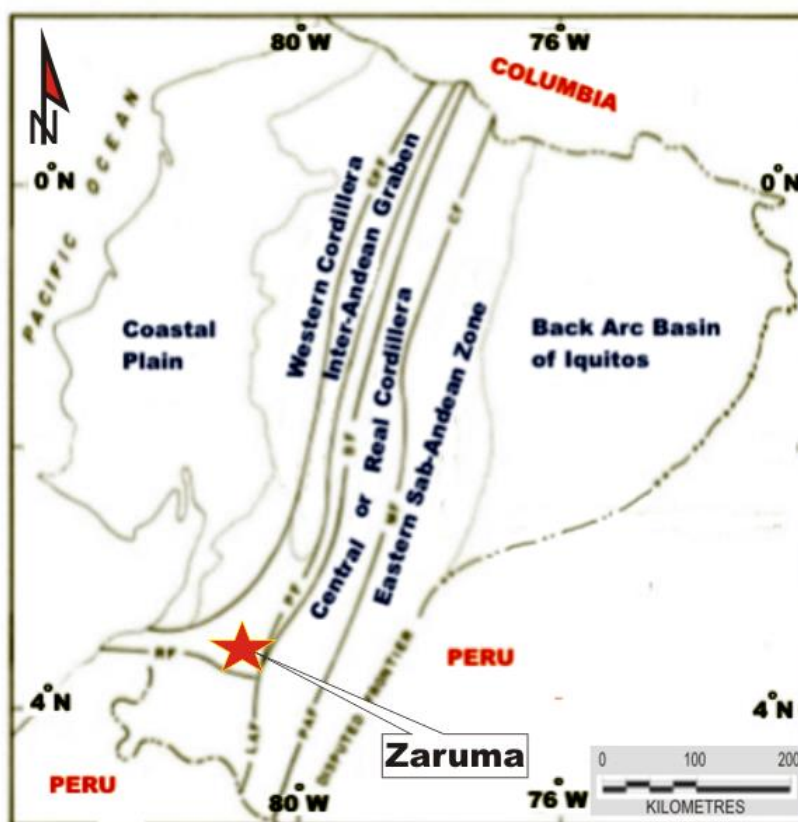


Figure 10: Simplified Structural Zones of Ecuador (after Litherland and Zamora, 1991).

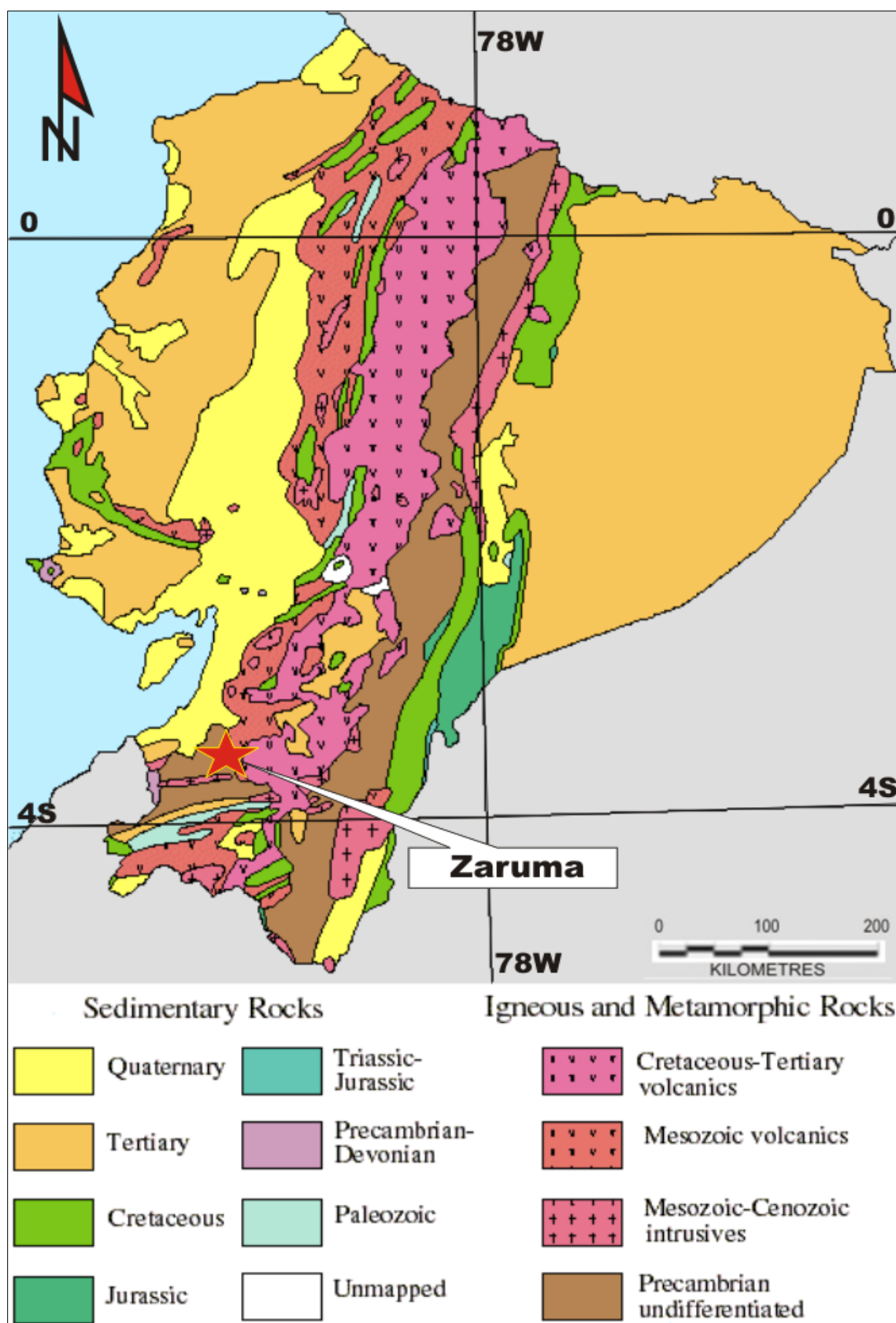


Figure 11: Simplified Geology of Ecuador (derived from US Geological Survey OFR 97-470D).

The Project area consists of a 15 km long vein system that lies immediately north of the Piñas Fault, a major regional structure that separates a metamorphic terrain to the south from a Tertiary volcanic sequence to the north (Figure 11). The metamorphic rocks, subparallel to the Piñas Fault, show that this structure represents a suture dated as Jurassic to early Cretaceous (Aspden et al., 1995). Tertiary volcanic rocks preserved on the northern side of the Piñas Fault (Figure 12) unconformably overlie metamorphic rocks facies of continental origin, implying that the suture marks a site of continent-continent collision in the late Jurassic to early Cretaceous.

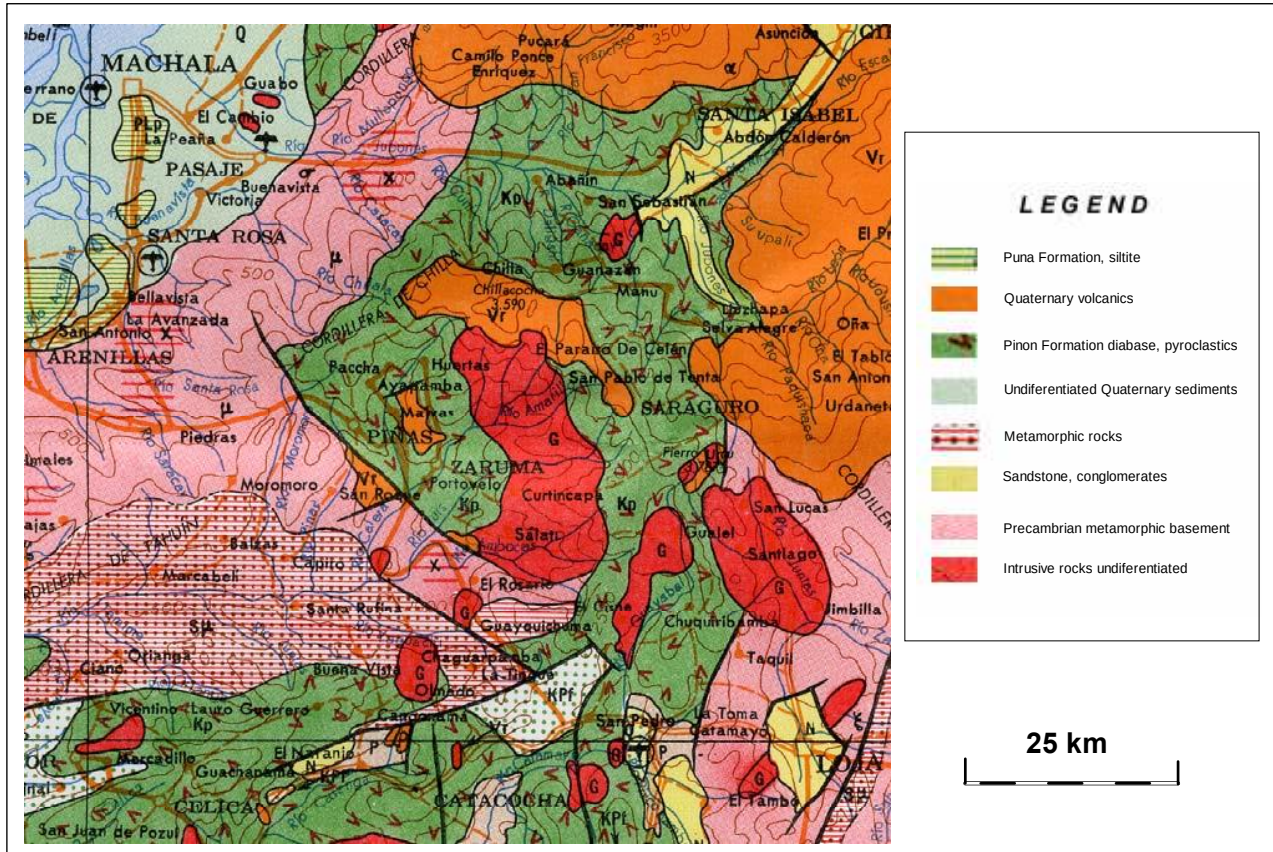


Figure 12: Regional Geological Map of the Zaruma – Portovelo District.
(Segment of Geological Map of Ecuador, Governmental Geological Service, 1969)

Preserved volcanic units show southward-thickening that define growth sequences formed in a half graben during normal movement on the Piñas Fault (Coward, 2001 in Spencer, R.M. et al., 2002) and as a consequence of deceleration in the rate of subduction of the Nasca plate beneath the South American plate at 37 Ma. North-south-directed compression increased the rate of the plate convergence at 20 Ma and caused the hanging wall of the Piñas Fault to bulge into an open, regional-scale antiform.

Host rocks of the Zaruma–Portovelo vein system are late Oligocene to early Miocene (28.4 – 21.5 Ma) in age (Dunkley and Geybor, 1997 in Spencer, R.M. et al., 2002) and range in composition from calc-alkaline to alkaline, compatible with their emplacement in an extensional cordillera setting. Four porphyry intrusions of dioritic to granodioritic composition have intruded the hinge of the Cangrejos antiform; one has been dated at 16 Ma (Pratt et al.,

1997). On one of these, the Fierro Urcu porphyry, drilling revealed a mass of coalescing sills compatible with intrusion into a compressive stress field (PRODEMINCA, 2002 in Spencer, R.M. et al., 2002).

7.2 Local & Property Geology

Vikentyev et. al. (2005) describes the local geology as follows, which also applies to the property geology:

The large Portovelo-Zaruma gold field is associated with Early Miocene continental arc magmatism. Quartz-sulphide veins crosscut andesitic lavas and tuffs. Pyrite, chalcopyrite, sphalerite and galena are the dominant ore minerals (5-10 vol.%). Gold occurs in visible and submicroscopic modes, mainly in chalcopyrite (up to 421 g/t, INAA). Petzite, hessite and tetradymite have been identified in druse quartz in assemblages with native gold.

In central and southwestern parts of Ecuador, the Eocene to Late Miocene igneous complexes related to epithermal and porphyry-Cu ores originated from mantle-derived calc-alkaline magmas fractionated at shallow levels inside rifted continental crust (Chiaradia et al., 2004).

Epithermal style gold veins of the Portovelo-Zaruma field (Figure 13) are associated with Early Miocene continental arc magmatism. The mining district consists of a tectonic block, 7 km in width, with NW-trending fault boundaries. The block is composed of andesites and tuffs of three series (K): Muluncay, Portovelo and Faique. There are numerous subvolcanic rhyolite dikes and stocks which crosscut the volcanic units.

The Portovelo-Zaruma field comprises a large system of longitudinal gold-polymetallic veins within a surface area of more than 4 x 12 km, aligned along a strike-slip structure.

Historical production was focused on the southern part of the system (Portovelo), which is characterized by the highest gold grade. More than 30 gold-bearing veins are identified. They dip east at angles from 45 to 70°, have lengths of hundreds of metres up to 2.5 km, and thickness of 0.8-1.5 m (reaching 3-5 m in some places). The vein system is complicated by the presence of ore shoots with a relatively gentle northern declination. Ore veins are crossed by rhyolite dikes related to the Pisayambo Formation (N1) that corresponds to the upper limit of gold-sulphide mineralization.

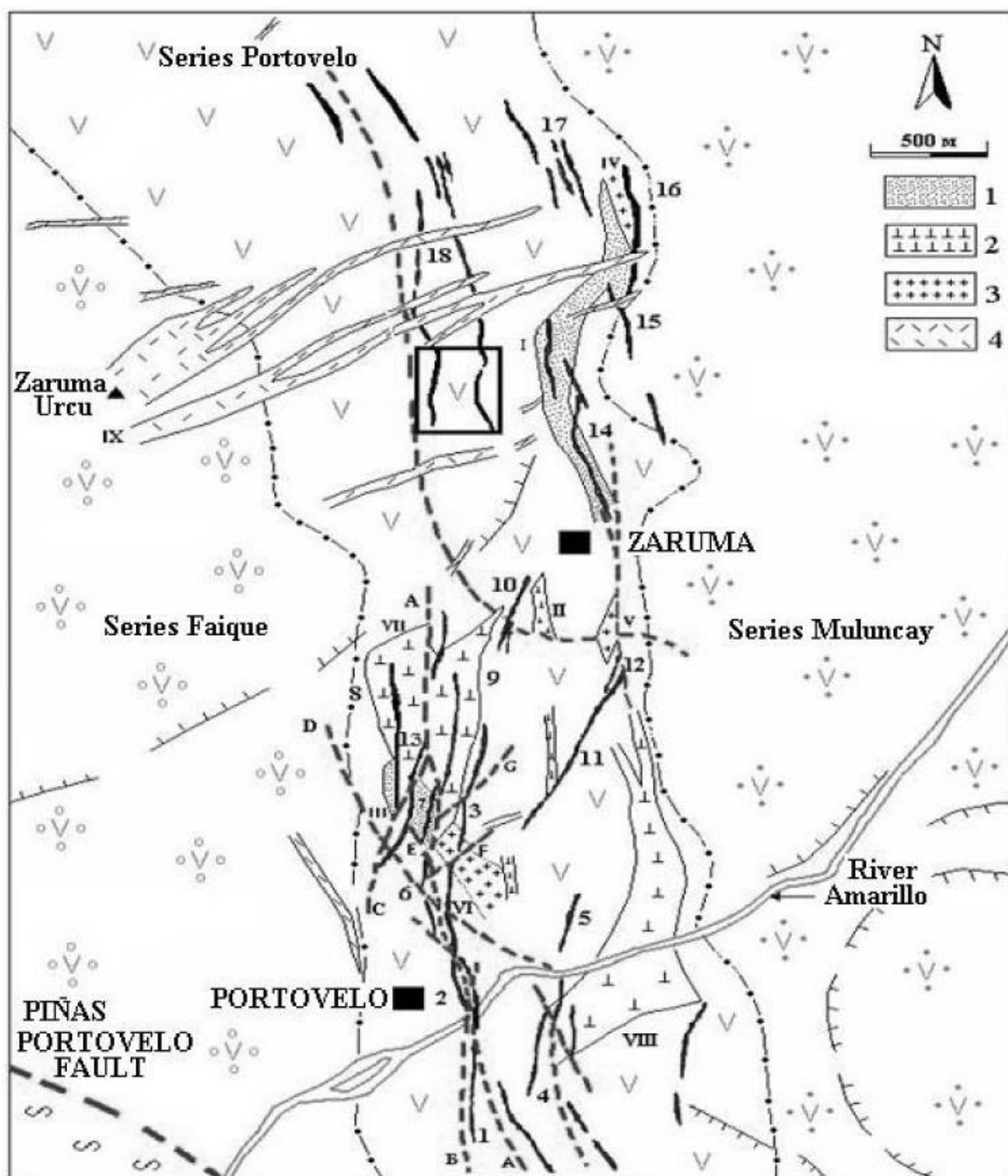


Figure 13: Map of Portovelo-Zaruma gold field (Vikentyev et al., 2005).

1-4 – intrusives: 1 - quartz-monzonite, 2 – granodiorite, 3 –diorite, 4 - rhyolite (I- Sesmo, II- Jorupe, III- Soroche, IV- 10 Vetas, V- Tres Reyes, VI- Castillo, VII- Agua dulce, VIII- Curipamba, IX- Zaruma Urcu). Veins: 1- Abundancia; 2- Portovelo; 3- Cantabria; 4- Curipamba; 5- Barbazco; 6- 26 y Nudo; 7- Soroche; 8- Tamayo; 9- Agua Dulce; 10- Bomba de Pachapamba; 11- Jorupe; 12- Tres Reyes; 13- Quebrada; 14- Sesmo; 15- Salvador; 16- Veta Ancha; 17- Gobernadora; 18- Vizcaya. Faults: A- Abundancia; B- Portovelo; C- Matalanga; D- “25”; E- Cerro Russia; F- “24”; G- Nivel “F”; H- Tres Reyes

The Zaruma Project area is underlain by intrusions and mafic to intermediate volcanic rocks that constitute the Portovelo Unit of the Saraguro Group. Aerial photography has revealed a distinct 10km in diameter ring-type structure which has been tentatively interpreted as a caldera structure (Quiroga, J.G., 1995). Rock types are discussed in greater detail at Item 8.

The monotonous volcanic sequence in the Zaruma–Portovelo area has been subdivided (Billingsley, 1926) into three units:

- The lowest part of the sequence has been nominated the Muluncay Series and is dominantly volcanic breccias, tuffs, lava flows and minor ignimbrites. Breccias include autobreccias, flow breccias, pyroclastic breccias and matrix-supported conglomerates which have been interpreted as mud-flow deposits
- The middle part of the local stratigraphy consists of medium-grained andesitic sills of the Portovelo Unit which constitutes the concordant limb of a lapolithic intrusion of melanodioritic composition which is similar in composition to the enclosing andesitic volcanic rocks
- The uppermost unit, the Faique Series, consists of volcanic rocks which are similar to the Muluncay Series except that the breccias tend to be coarser grained.

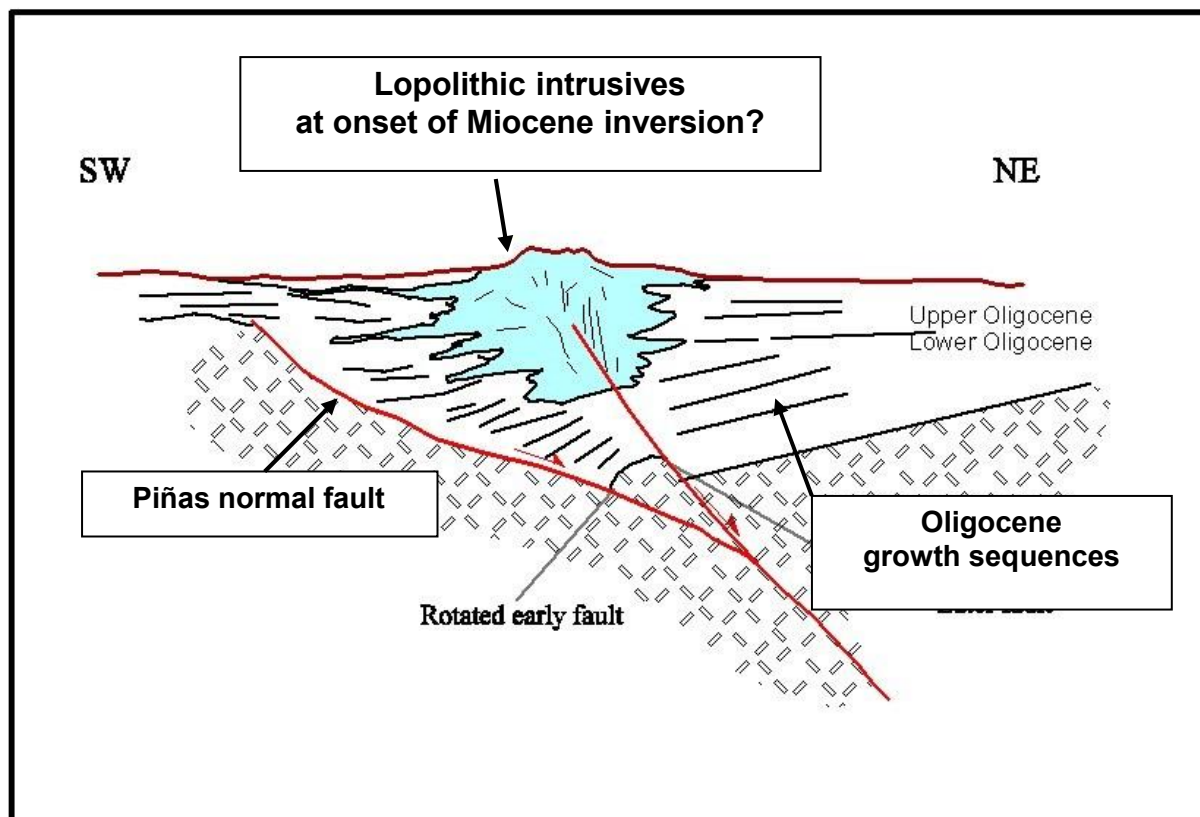


Figure 14: Schematic cross-section showing a model for the development of the Portovelo Unit volcanics associated with normal faulting (after R. Spencer, 2002).

Considering similarities between the Muluncay Series and the Faique Series it was suggested that both units may form part of an upward-coarsening sequence that was intruded along flat-lying fractures by a melanodioritic lopolith (Figure 14) which has an eastern limb that constitutes the Portovelo Unit (Spencer, R.M. et al., 2002).

Numerous, mostly fine-grained intrusions varying in composition from melanocratic diorite to quartz monzonite are known in the project area and are interpreted as high-level, subvolcanic intrusive bodies. Sigmoidal shapes of the intrusive bodies in plan view are suggesting an asymmetry which is compatible with intrusion into a right-lateral stress field. In some cases, it was observed that melanocratic diorite dykes run parallel to the main veins (Figure 15) suggesting a close spatial relationship between dykes and mineralising fluids that are suspected to be related to a late stage of melanocratic diorite intrusive activity (Quiroga, J.G., 1995).

The main host rocks to mineralisation are the volcanic units described above, however, immediately to the NW of Zaruma city a strongly altered porphyry was also observed as a host rock (Quiroga, J.G., 1995).

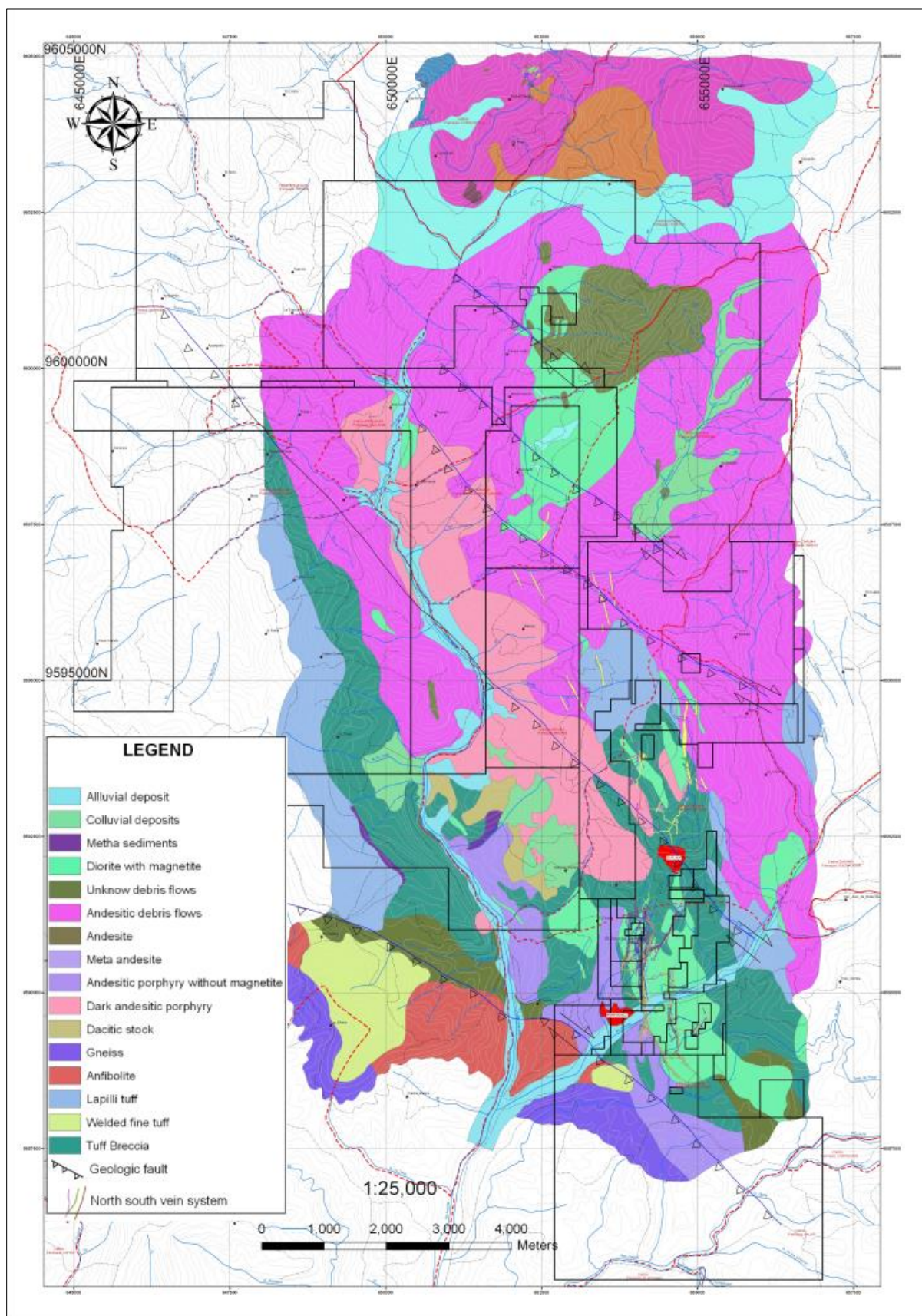


Figure 15: Zaruma Gold Project Geological Map (after IAMGOLD and Kalinaj, 2004).

Veins in the Zaruma–Portovelo District are arranged in four principal en-echelon sets controlled by NW-SE striking oblique faults parallel to the Piñas Fault (Figure 16). The majority of the veins strike north and some show a sigmoidal geometry in which southeast-striking tails rotate into the north-striking central portions of the veins (Spencer, R.M. et al., 2002).

Faults that control the en-echelon sets formation strikes NW and dip to the southwest at 40° to 60° . The oblique right-lateral thrust movement on the north-west-striking thrust ramps have been defined. It is suggested that Zaruma, Sesmo-Colorada, Muluncay and R-level Thrust root on the Piñas Fault.

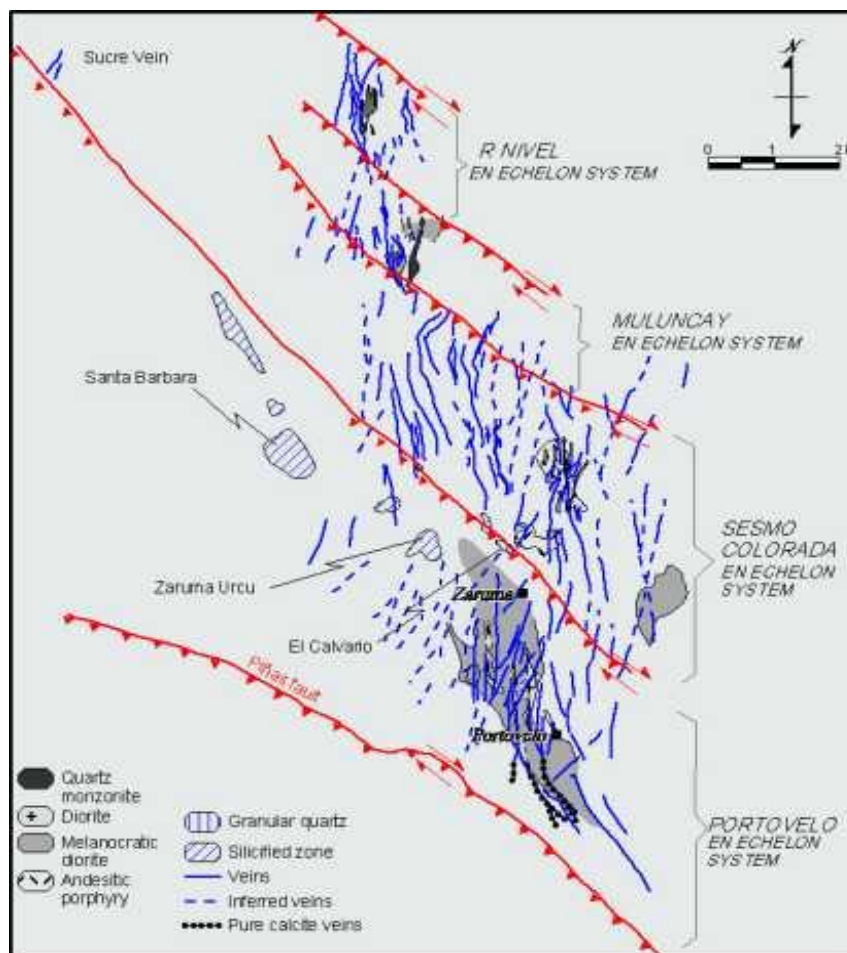


Figure 16: Northwest striking thrusts control the en-echelon set formations and the sigmoidal geometry of the veins.

Alteration styles within the Zaruma–Portovelo area show a transition from high temperature assemblages in the NE portion of the area centred on El Poglio porphyry to low temperature assemblages towards the southwest and south (Figure 17).

While veins are exposed up to 900 m below a secondary biotite- tourmaline-magnetite altered El Poglio porphyry, still further north and at higher elevations, K silicate altered dioritic porphyries passes upward into a high-temperature advance argillic altered zone with a pyrophyllite-diaspore-topaz-tourmaline assemblage defined as a lithocap (Sillitoe, R., 2000). Advance argillic alteration comprising vuggy silica, dickite, pyrophyllite and minor alunite caps high-relief hills in the southwestern part of the project area near Zaruma Urcu and Santa Barbara and are interpreted as relicts of a SW dipping advance argillic-altered volcanic sequence that formed part of the lithocap related to the El Poglio porphyry cluster (Sillitoe, R., 2000). Veins closest to the El Poglio porphyry lie adjacent to the area of secondary biotite and the vein infill consists of dog-tooth quartz intergrown with banded Fe-rich chlorite, epidote, tourmaline, magnetite and in some cases, coarse-grained muscovite.

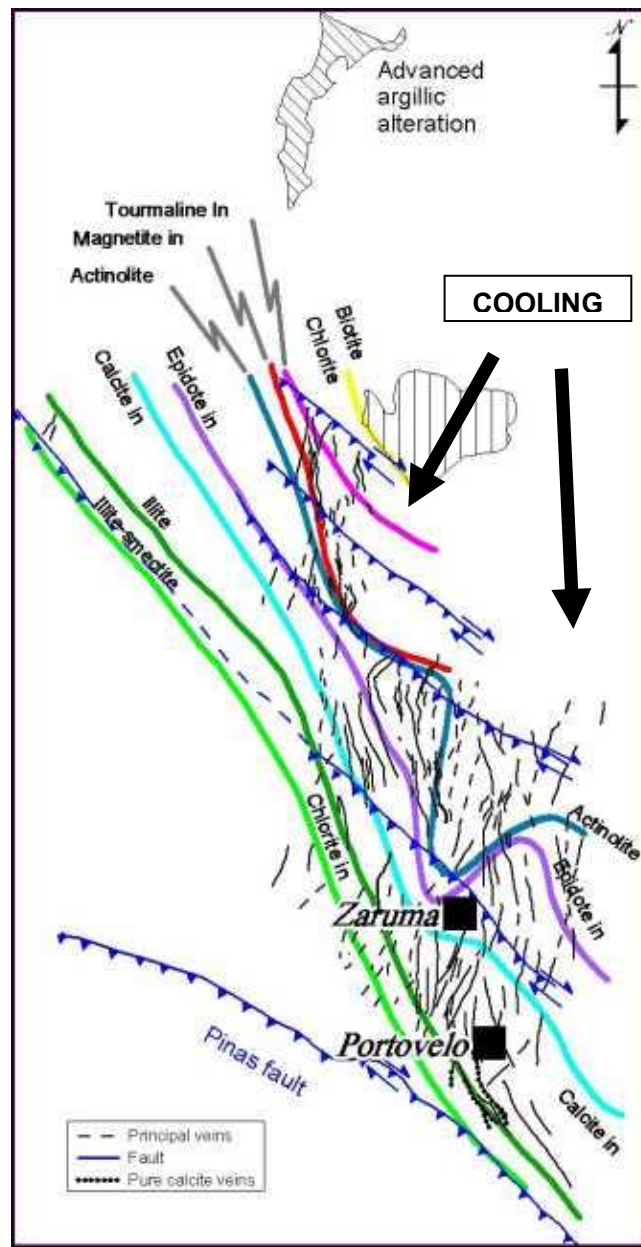


Figure 17: Alteration mineral zonation within the Project area (after R. Spencer, 2002).

From north to south through vein outcrops a gradual decrease in temperature was observed in wall-rock alteration, culminating in illite-smectite mix-layered clay in proximity to the Piñas Fault (Reyes, 1990 in Spencer, R., 2002).

8.0 Deposit Types

8.1 General

The Zaruma Mineral Deposits are a 15 km long polymetallic epithermal precious metal vein system that lies immediately north of the regional Piñas Fault. The vein system is hosted by a monotonous Tertiary volcanic sequence of intermediate to mafic volcanic breccias, tuffs and andesites. The majority of veins in the Zaruma–Portovelo District strike north, dip steeply to the west or east and are arranged in four principal en-echelon sets from south to north: the Portovelo en-echelon system → the Sesmo-Colorada en-echelon system → the Muluncay en-echelon system → the R-nivel en-echelon system, controlled by NW striking faults parallel to the Pinas Fault. The vein textures are for the most part unchanging along strike and range from rhythmically banded, massive to drusy quartz with some hydrothermal breccias. The best development of veins was observed in the immediate footwall of the northwest striking thrust faults where they are wider and continuous along strike when hosted by competent rocks. Most veins throughout the strike length of the Zaruma–Portovelo vein system dip to the east at an average of 60°. Further geological details are provided at Item 7.0.

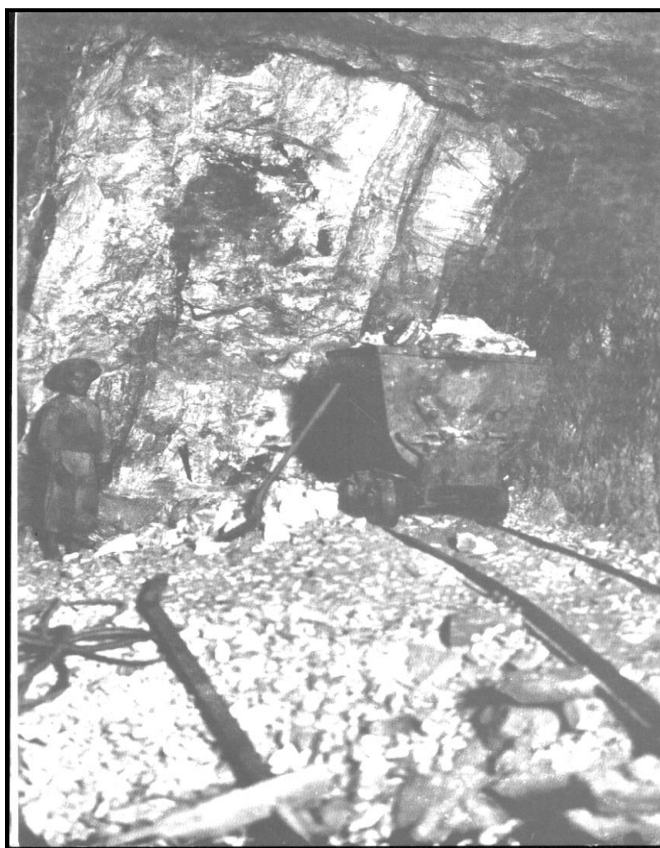


Figure 18: Typical High-Grade Vein, Zaruma-Portovelo Mining District.



Figure 19: Banded Textures, Cristina Vein, Zaruma (after Kalinaj, 2004).

8.2 Structural Setting

Veins in the Zaruma–Portovelo District are arranged in four principal en-echelon sets controlled by NW-SE striking oblique faults parallel to the Piñas Fault (Figure 20). The majority of the veins strike north and some show a sigmoidal geometry in which southeast-striking tails rotate into the north-striking central portions of the veins (Spencer, R.M. et al., 2002).

The faults that control the en-echelon sets formation strike NW and dip to the southwest at 40° to 60°. It is suggested that Zaruma, Sesmo-Colorada, Muluncay and R-level Thrust root on the Piñas Fault.

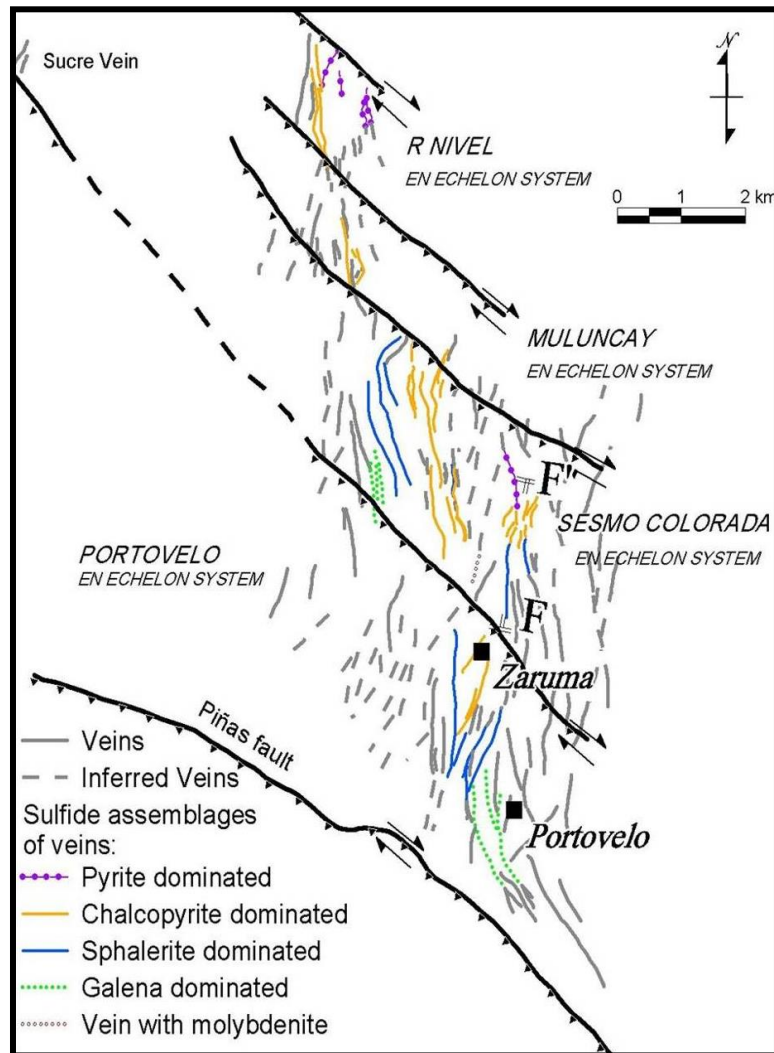


Figure 20: Mineralisation Zonation in the Zaruma–Portovelo Mining District (after Spencer, 2002).

8.3 Mineralisation

Vein textures in the Zaruma project area do not significantly change along 10km of vein trend. Typical textures for low-sulphidation veins have been observed including rhythmically banded textures with dominantly symmetric geometry, drusy textures, hydrothermal breccia textures, dissolution and replacement textures.

In contrast, veins in the Portovelo system in the southern part of the trend change composition gradually along strike southward from quartz through quartz-calcite, to calcite-dominated compositions over a strike of 2 km. A vertical zonation in composition from quartz dominated deeper portions of veins to calcite-dominated assemblages near surface was observed, in particular, within the Portovelo en-echelon block.

Mineral assemblages within veins showing a zonation is a function of cooling direction (Figure 20). While northern portions of veins are dominated by relatively higher temperature

assemblages with disseminated chalcopyrite-pyrite-magnetite in K-silicate altered El Poglio porphyry and banded quartz-chalcopyrite-pyrite-magnetite-dark-green chlorite within veins in R-nivel enechelon set.

Going south, the mineral group progresses into the quartz-chalcopyrite-pyrite dominated Muluncay en-echelon set then into quartz-chalcopyrite-sphalerite-galena in the Sesmo-Colorada en-echelon up to the sphalerite-dominant, quartz-carbonates-sphalerite-pyritegalena $\pm$ chalcopyrite assemblage in the Portovelo en-echelon system.

Similar zonation can be observed in an EW section with higher temperature assemblages in eastern and central part of the district dominated by chalcopyrite changing into sphalerite-dominated to galena-dominated assemblages in the easternmost part of the district as shown on Figure 20.

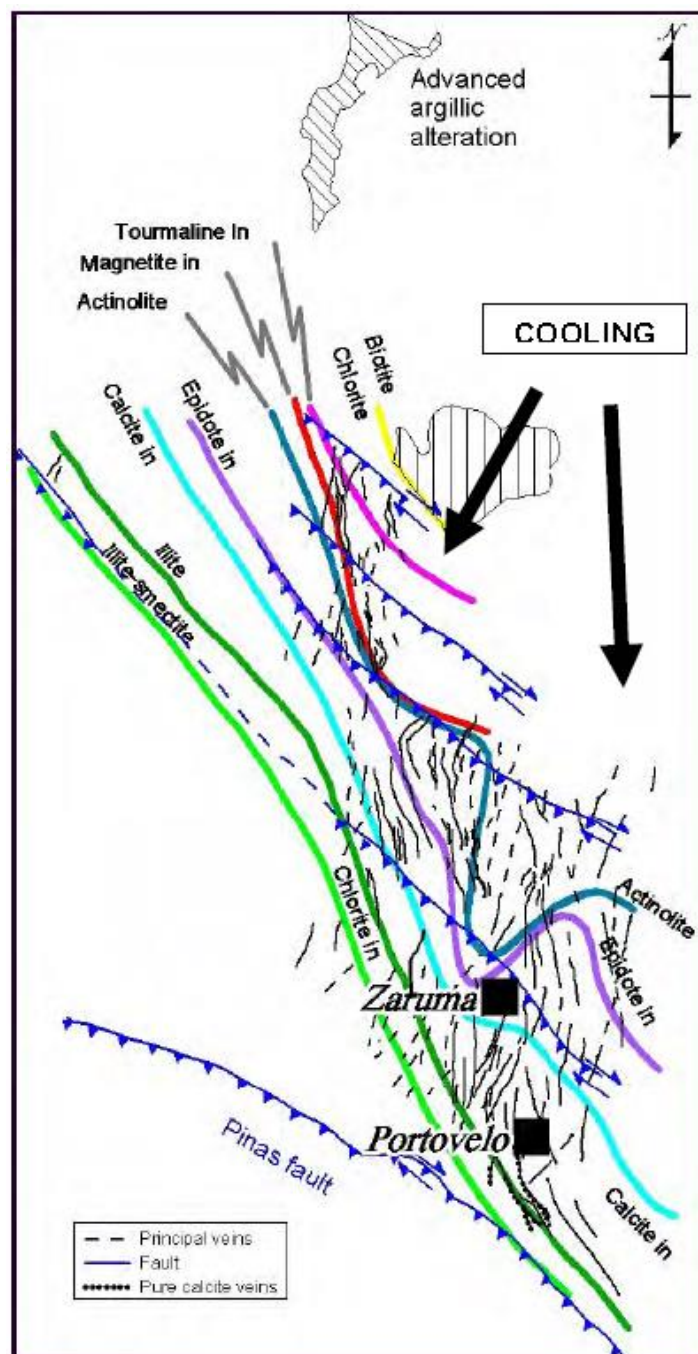


Figure 21: Alteration mineral zonation within the Project area (after Spencer, 2002).

8.4 Mineral Assemblage Studies

Kalinaj carried out petrographic and mineral assemblage microscopic studies in 1995. The results from these studies are summarized below. A total of 9 petrographic sections were used for transcurrent light studies and 5 polished sections were used in reflected light studies.

Rock Types:

1. *Quartz Andesite* – a crystalline rock with phenocrysts and a vitreous groundmass. The rock belongs to group of rocks with a low presence of quartz and without K-feldspars. The dominant mineral is idiomorphic phenocrysts of Na-Ca feldspars (andesine) with poikilitic inclusions of quartz and mafic minerals. Strong albitisation and carbonitisation was observed. Mafic minerals, mainly biotite and amphibole, are strongly chloritised. Irregularly-shaped quartz crystals are present in less than 5% of the rock. The groundmass is vitreous and consists of strongly altered clay (kaolinite) with undefined opaque minerals. Pseudomorphs of chlorite and carbonate after mafic minerals are common (Botoneros labour, 10m north from BT-009 survey point, Zaruma).

2. *Pyroxene Andesite* – this rock is characterized by a holocrystalline texture with a dominance of groundmass over phenocrysts. The presence of K-feldspars was detected. Dominant minerals are Na-Ca feldspars (andesine to labradorite) and a monoclinic pyroxene (35° - 48°), which is relatively fresh, frequently twinned in a 100° axes and determined as probably augite. Chlorite and carbonate are present as pseudomorphs after pyroxene and/or other mafic minerals. The groundmass texture is finely crystallised and is composed by quartz and feldspar (Naranjitos labour, reference point N-032).

3. *Dacite Porphyry* – two types of porphyries have been detected:

- * A porphyritic type with mega-crystals of feldspars and quartz in a microcrystalline groundmass of similar composition
- * A less porphyritic variety with only medium-size laminar and randomly-oriented feldspar phenocrysts which were apparently xenoliths from an older intrusive phase

Only Na-Ca feldspars of andesine composition have been observed in both types of porphyry with frequently pronounced albitisation and carbonitisation alteration. Megacrystals of quartz are common and with the crystals size, liquid > gas fluid inclusions have been detected. Mafic minerals have been observed only as pseudomorphs of chlorite, epidote / zoisite and carbonate without relict textures which make the primary mineral identification difficult, although amphibole is possible (Pacayurco area, 655.875E / 9°59'05.050 N).

4. *Granodiorite Porphyry* – this rock is characterised by intensively albitised Na-Ca feldspars with minor K-feldspars. Interstitial spaces between the feldspars phenocrysts are infilled with quartz. Mafic minerals are totally replaced by carbonates and chlorite, and only pseudomorphs have been observed. (San Antonio labour, Muluncay are, reference point SA-017)

5. *Dacite* – this rock has a significant amount of quartz, very low to absent K-feldspars and hemicrystalline to porphyritic textures. Vesicular textures observed are different in size and mineral infill. Small vesicles are oval-shaped and usually filled with finely

laminated and radial-crystallised chlorite; while the larger vesicles are infilled with quartz, albite, chlorite, epidote / zoisite, actinolite and carbonate. Albitisation is the whole rock alteration style and is replacing mainly the feldspars. (La Chonta labour, Muluncay area, Zaruma. Reference: 5m east from CH-008 underground survey point)

6. *Amphibole-biotite Granodiorite* – this rock is characterised by feldspar twinning of Carlsbad-type in euhedral crystals with poikilitic inclusions of plagioclase (andesine), quartz and mafic minerals. Potassic feldspars, frequently in paragenetic association with quartz and biotite, show a lower crystalline grade as Na-Ca feldspars are present. In some mega-crystals of Na-Ca feldspars, potassic feldspars have been observed as poikilitic inclusions. Monoclinic amphiboles form one dominant mineral (20°). Anhedral quartz fills the interstitial space between plagioclase crystals and finely laminated biotite with Fe-Ti oxides and ilmenite. (Muluncay zone north from the Zaruma project boundary)

Mineralisation assemblages:

1. *Quartz* – forms the common and dominant mineral in all ore-samples studied. At least two generations of quartz have been identified:

- * An older generation of coarsely-granulated quartz with interstitial space infilled with sulphides (mainly chalcopyrite and pyrite), native gold and hematite. Gold impregnations, up to 50 µm in size, are abundant in this quartz generation (La Chonta labour, reference point CH-045, Jane vein)

- * A younger generation of crystallised, euhedral quartz and micro-drusy quartz frequently with sulphides inclusions. Quartz crystal surface corrosion by sulphides is common

2. *Pyrite* – occurs in the main as single, disseminated hexahedral crystals in the older generation quartz. Pyrite aggregates infill with chalcopyrite and native gold have been observed (La Chonta labour, CH-045, Jane vein and Las Cañas labour, 5m south of SC-43, Cristina vein). Pyrite crystal surface corrosion textures by gold have been observed.

3. *Chalcopyrite* – forms commonly as infill of drusy-quartz, in pyrite aggregates and in cataclased pyrite grains. Isometric inclusions of native gold are frequent. Chalcopyrite is defined as an older mineral in comparison with galena and sphalerite due to the corrosion textures of chalcopyrite on contact with younger galena and sphalerite.

4. *Sphalerite* – is a common mineral in studied assemblages and corrodes chalcopyrite grain surfaces together with or slightly before galena. Evidence of the sphalerite/chalcopyrite solid solution segregations are usually observed as > 5 µm grains of chalcopyrite in sphalerite. Textures of sphalerite surface corrosion by native gold and polybasite have been observed. Concentration of chalcopyrite inclusions in sphalerite decrease from early stage-formed sphalerite to younger stage-formed sphalerite were observed in zoned sphalerite crystals. Based on colour changes seen in zoned sphalerite crystals, two varieties of the mineral are present – iron-rich

marmatite as a older sphalerite generation and an iron-poor sphalerite, honey coloured cleophane, as a younger generation.

5. *Galena* – occurs less abundantly than sphalerite and is apparently slightly younger. Together with sphalerite, galena has been observed corroding chalcopryite surfaces and in turn the galena surface has shown corrosion by native gold and polybasite (Pacay Blanco labour, Cristina Vein)

6. Polybasite – forms a frequent silver mineral in association with gold and exhibits corrosion textures on sphalerite and galena grain surfaces.

7. *Acanthite* – a common silver mineral in association with gold and polybasite; infilling micro-fractures in galena crystals.

8. *Gold* – a ubiquitous mineral in all samples. With silver minerals (polybasite and acanthite) native gold forms inclusions in chalcopryite, sphalerite and galena. Gold also shows corrosion textures on sphalerite and galena grains, infilling exfoliation fractures in galena crystals and infilling micro-fractures within pyrite crystals. Gold is frequently observed in association with dark-green chlorite in the northern part of the Zaruma District. Spectacular native gold specimens are well-known from the Zaruma area in the form of hairy-gold in drusy-quartz and coarse gold, including cubic gold crystals, occurs as a carbonate vein infill in the Portovelo area. Free-gold specimens vary in colour from bright yellow to dark yellow-greenish which suggest a gold purity in the range of 600/1000 to 850/1000.

9. *Carbonates* – form coarsely-crystallised minerals which are in the main calcite and occur as the youngest drusy-infill with quartz.

8.5 Gold and Silver Distribution

The gold and silver distribution differs from that of base metals. Major concentrations of gold and silver were observed closely related to the footwall of southerly dipping thrust faults which lie parallel to the Piñas Fault especially in the northern en-echelon sets of R-nivel, Muluncay and Sesmo-Colorada. While in the Portovelo en-echelon set, almost 90% of precious metal production was from the central part of the veins (Figures 22 and 23).

The decrease of gold and silver in proximity to the Piñas Fault is probably a result of alteration zonation where this part of the District is well within the illite-smectite wall-rock alteration zone (Spencer, R.M. et al., 2002). This alteration mineral assemblage represents relatively cool hydrothermal conditions with temperatures ranging from 100° to 200°C and mineral deposition in subsurface levels (Hedenquist, J.W. et al., 2000).

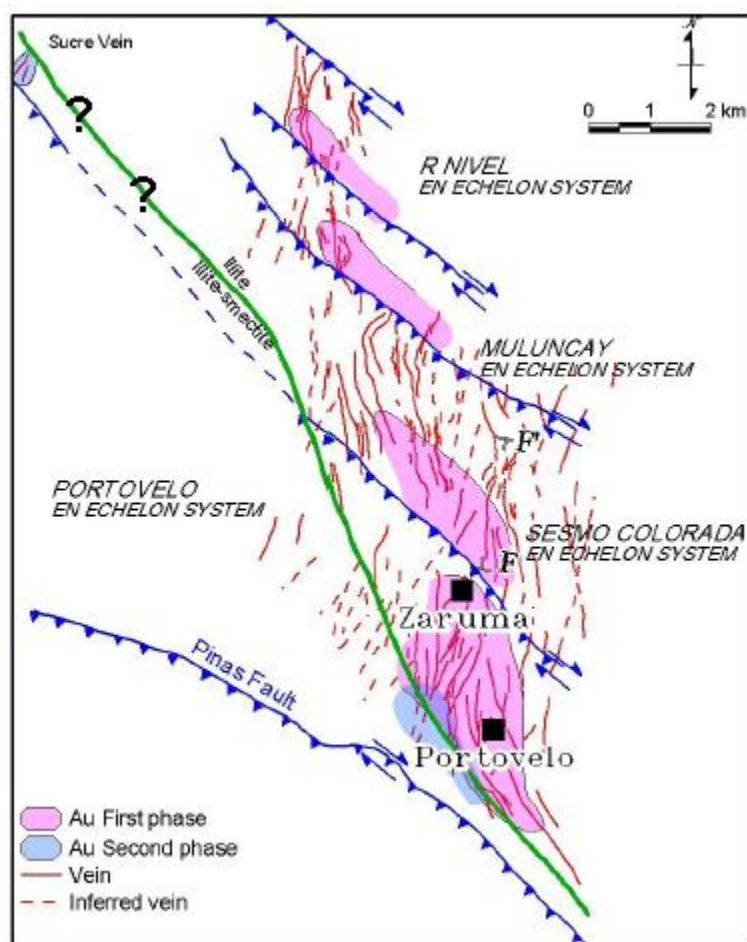


Figure 22: Gold Distribution in Zaruma – Portovelo Mining District (after R. Spencer 2002).

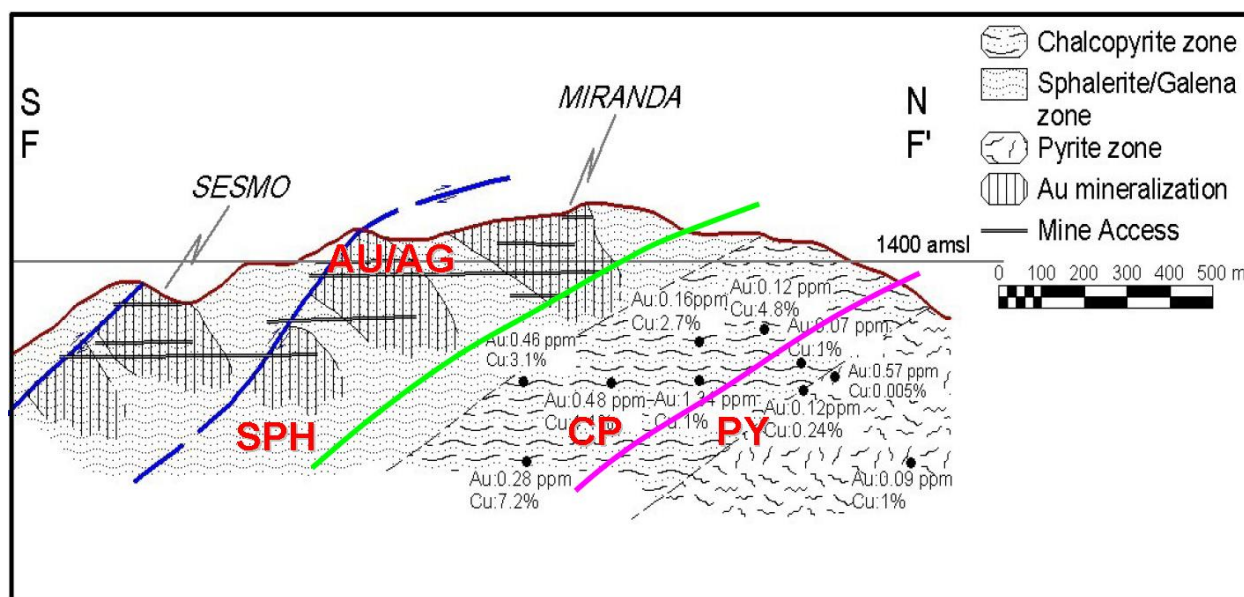


Figure 23: Longitudinal Section along Sesmo Vein showing gold and polymetallic mineralisation (after R. Spencer, 2002).

Two stages of gold and silver mineralisation have been detected in the Zaruma – Portovelo Mining District.

- Stage 1 – Quartz-adularia-base metals-gold-silver veins are considered as the earliest stage of mineralisation (Figure 24). Vein assemblages contain pyrite, chalcopyrite, sphalerite, Ag-bearing galena and minor bornite and silver-poor tetrahedrite. In near surface levels also chalcocite and covellite can be present as well as copper-oxides (Quiroga, J., 1995). Mineral assemblages vary according to NS and EW zonations described earlier.

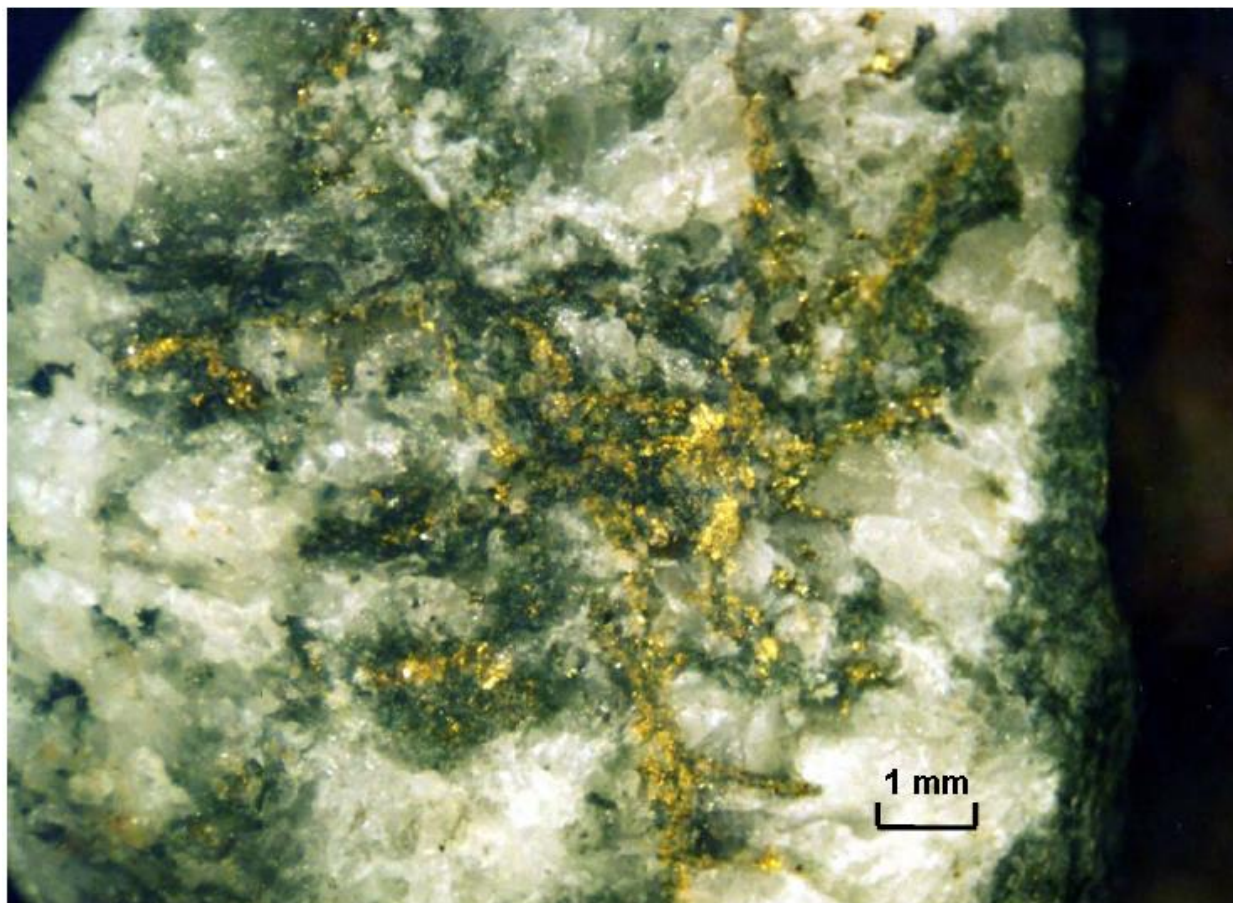


Figure 24: Early stage gold intergrowth with dark-grey chlorite. Jane Vein, Zaruma. (Kallinaj, 2004).

- Stage 2 – Quartz-calcite-chlorite-electrum-Ag sulphosalts are considered to be a younger mineralisation event and characterised by low to zero sulphides and free coarsegold (Figure 25). This sulphide-poor stage contains a wide spectrum of sulphosalts including tetrahedrite, tennantite, Ag-tetrahedrite (freibergite), safflorite; Ag-sulphosalts as pyrargyrite, proustite, stephanite and including gold-telluride like nagyagite. Free, colloidal gold is reported from the dark, banded botryoidal quartz with disseminated finegrained pyrite in the Sucre Vein (Marikovsky, 1958 in Spencer, R.M. et al., 2000).

The total sulphide content in veins varies in a north-south direction from 10 to 15 % in the R-nivel and Muluncay en-echelon systems dominated by chalcopyrite and pyrite through to 5-10% in the central part of the Sesmo-Colorado en-echelon set dominated by sphalerite-pyrite galena to >5% to nil sulphides in Portovelo en-echelon set. A similar distribution can be observed in an east-west direction with low, >5% to nil sulphides in eastern part of the district increasing gradually to west with 20-50% sulphides and mainly sphalerite-galena dominated assemblages (Sillitoe, R.H., 2000).



Figure 25: Late stage coarse and partially crystallised gold (shown by arrow) in association with calcite and quartz. Abundancia Vein, Portovelo area (Kalinaj, 2004).

9.0 Exploration

9.1 Exploration History

Exploration history is described at Item 6.1.

The exploration work carried out by Dynasty at Zaruma is summarised in Tables 9 and 10.

Prospect	Drill Holes	Total Metres Drilled	Geotech Holes	Geotech Metres Drilled	Exploration Holes	Exploration Metres Drilled	Drilling Samples Assayed
Cabo de Hornos	38	6,615.35	9	890.40	29	5,724.95	471
Barbasco	29	5,392.65	1	200.50	28	5,192.15	272
Ayapamba	15	3,896.55	3	486.90	12	3,409.65	95
Pacayurco	5	1,608.60	0	0.00	5	1,608.60	24
Playitas	5	874.00	0	0.00	5	874.00	16
Colorada	2	405.50	0	0.00	2	405.50	7
Collapsed (not finished holes)	2	37.10	0	0.00	2	37.10	
Others Geotech (G.Valley Plant)	7	499.25	7	499.25			
Totals	103	19,329.00	20	2,077.05	83	17,251.95	885

Table 9: Summary of Dynasty Drilling at Zaruma.

Prospect	Clearing Tunnels for Sampling (metres)	U/ground Channels	U/ground Channel Metres	U/ground Channel Samples	U/ground Samples Assayed	Mapping (scale)	Surface Geochem Samples
Cabo de Hornos	437	1,542	1,222	1,564	348	1 : 500	0
Barbasco	275	167	137	103	42	1 : 500	4
Ayapamba	269	276	312	303	89	1 : 500	12
Pacayurco	117	23	15	19	14	1 : 2500	4
Playitas	134	7	7	6	6	1 : 2500	8
Colorada	245	17	24	17	34	1 : 1000	6
Others Geotech (G.Valley Plant)						1 : 250	0
Totals	1,477	2,032	1,717	2,012	533		34

Table 10: Summary of Dynasty Exploration Works at Zaruma.

During 2005-2008, the Company mapped outcrops along creeks in the northern sector of the concessions. Only limited samples were collected due to poor outcrop exposure. Many of the veins in this region had been mined from the surface and therefore only voids could be shown on maps.

Clearing, mapping and sampling of old mining development at Ayapamba, Cabo de Hornos, Barbasco, Playitas, Pacayurco and La Colorada mines commenced in 2006. Diesel compressors ensure good ventilation in the tunnels and power the sampling equipment used for the channel sampling, perpendicular to the veins at 3 - 4 meter intervals along the drives.

The majority of this work was carried out at Cabo de Hornos, the largest decline developed by the Company, where 1,564 samples were collected from the Matalanga, Nudo and San Ernest (Tamayo) veins. A further 19 channel samples were collected at Pacayurco in the French and Pacayurco veins.

A total of 303 underground channel samples were taken along various underground levels at Ayapamba from the Sucre, Bolívar and Tres Diablos veins. Mapping and 17 samples collected from the old workings in the Malvas vein at Colorada confirmed the existence of spectacular gold and copper values in the northern extension of this vein until its convergence with the Tres Reyes vein which was also recorded historically as a higher gold grade vein.

9.2 Adequacy of Exploration

Details of sampling methods and quality, as well as relevant details of the samples collected, are provided at Item 10.

In the opinion of the authors the exploration programs completed to date were appropriate to the known deposit mineralization styles and this exploration has delineated significant epithermal gold-silver deposits and will continue to develop the exploration potential of the Project.

There are reasonable prospects for future exploration and drilling programs being able to add to the known mineralization, and potentially to add to the mineral resources.

10.0 Drilling

10.1 Dynasty Drilling

Dynasty has conducted exploration drilling since 2008 using two company owned Hydracore 2000 diamond drill rigs. By the end of 2013, approximately 18,600 m of drilling had been completed. Drilling by a prior operator, IAMGOLD, is summarized at Item 6.2. Drilling data by prior operators was not applied in determining the Mineral Resource estimate contained herein due to the wide spacing of their drilling programme. As such, the results of such prior drilling programmes are not relevant to this report and have not been included.

The main areas drilled by Dynasty are (Figures 26 to 32):

- a) **Cabo de Hornos** (Ana Michelle, Soroche Unificado and Nueva Esperanza 2 concessions);
- b) **Barbasco** (Barbasco 1A, Barbasco Unificado and southern part of Loma Rica 1 concessions);
- c) **Ayapamba** (Bethsabeth concession);
- d) **Pacayurco** (Pacayurco and Rome concessions);
- e) **Playitas** (Machay concession) and
- f) **Colorada** (Malvas 1 concession).

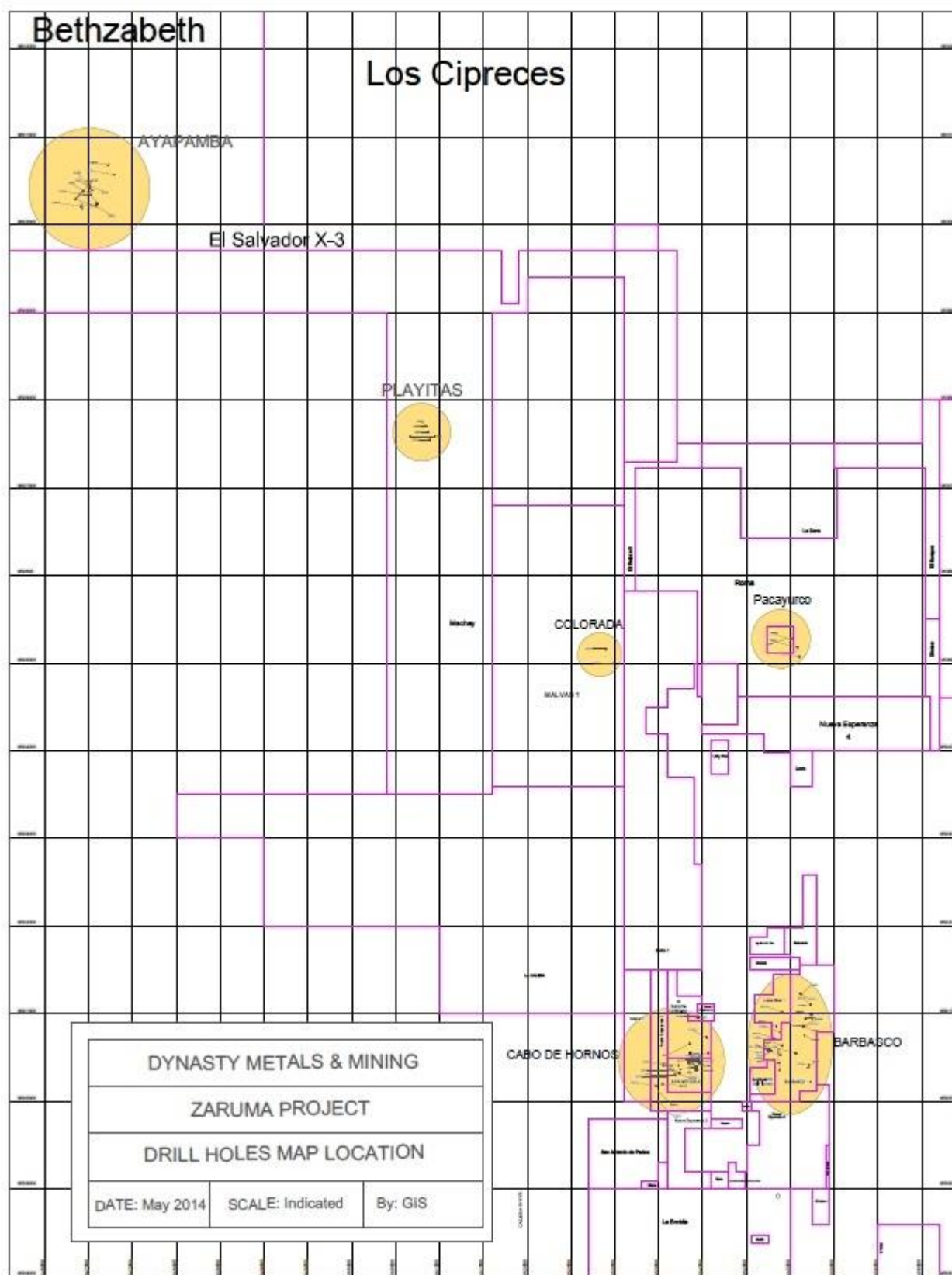


Figure 26: Zaruma Project Drill Holes Location Map.

The area covered by drilling totals approximately four square kilometers.

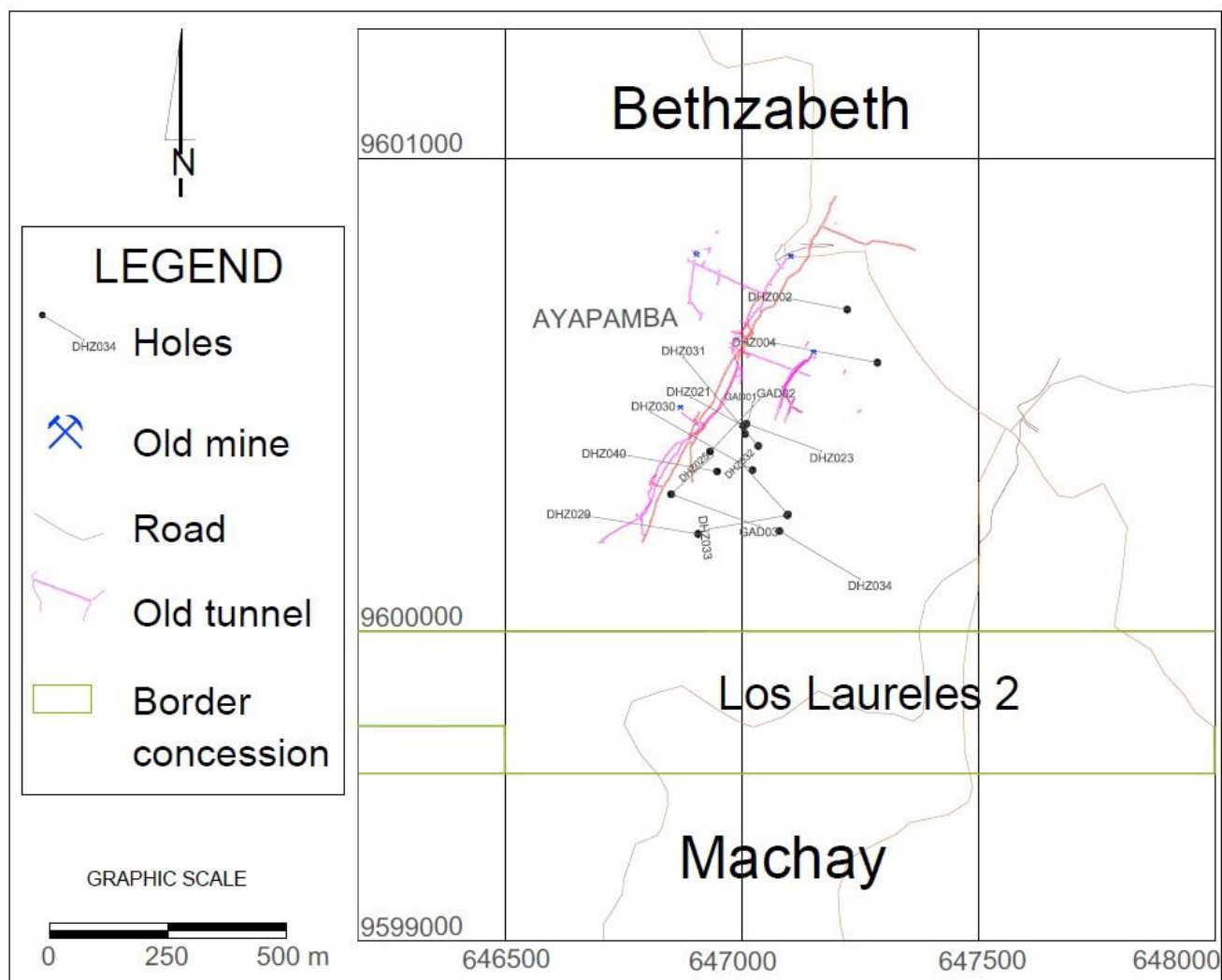


Figure 27: Drill hole locations at Ayapamba.

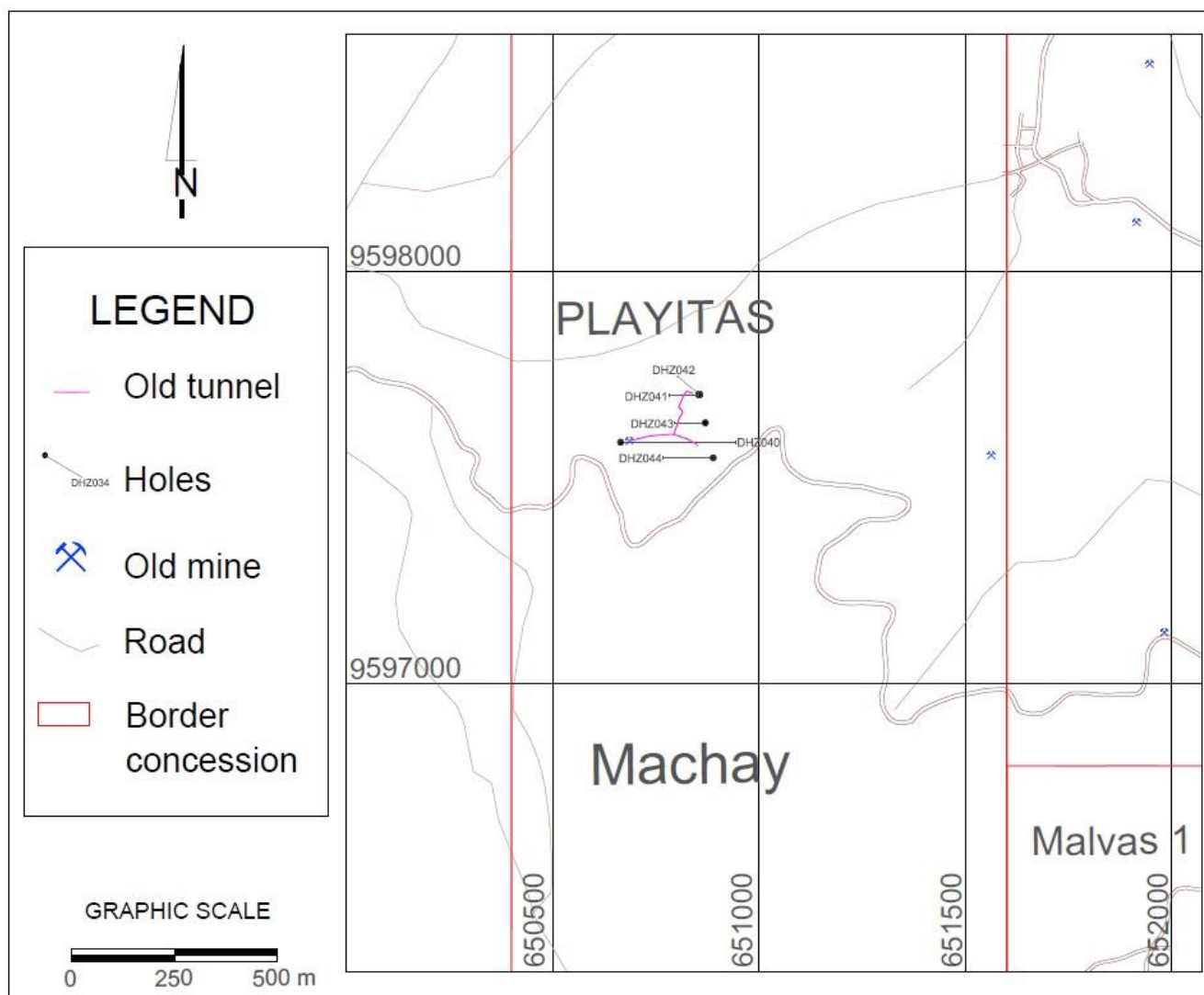


Figure 28: Drill hole locations at Playitas.

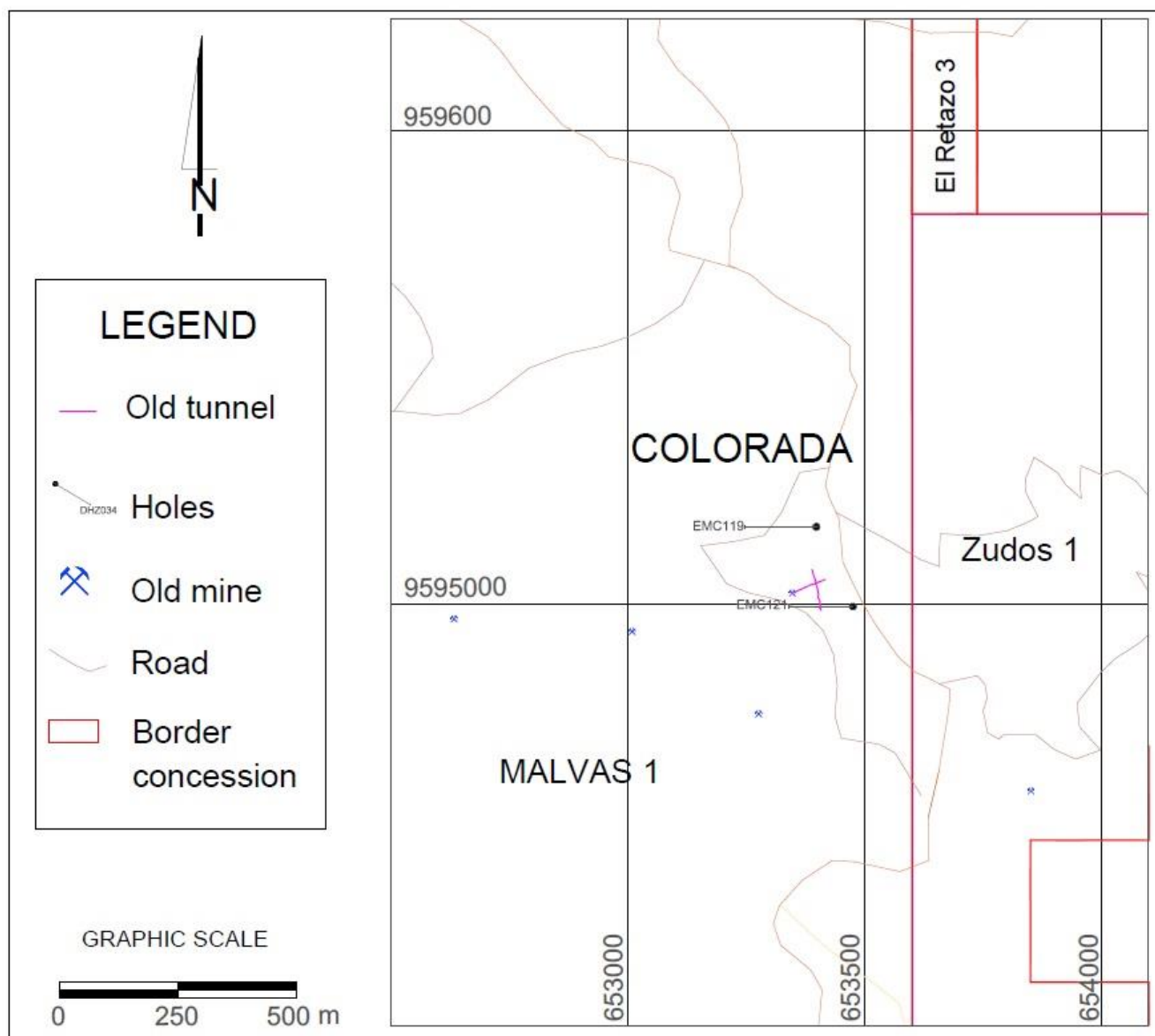


Figure 29 Drill hole locations at Colorada.

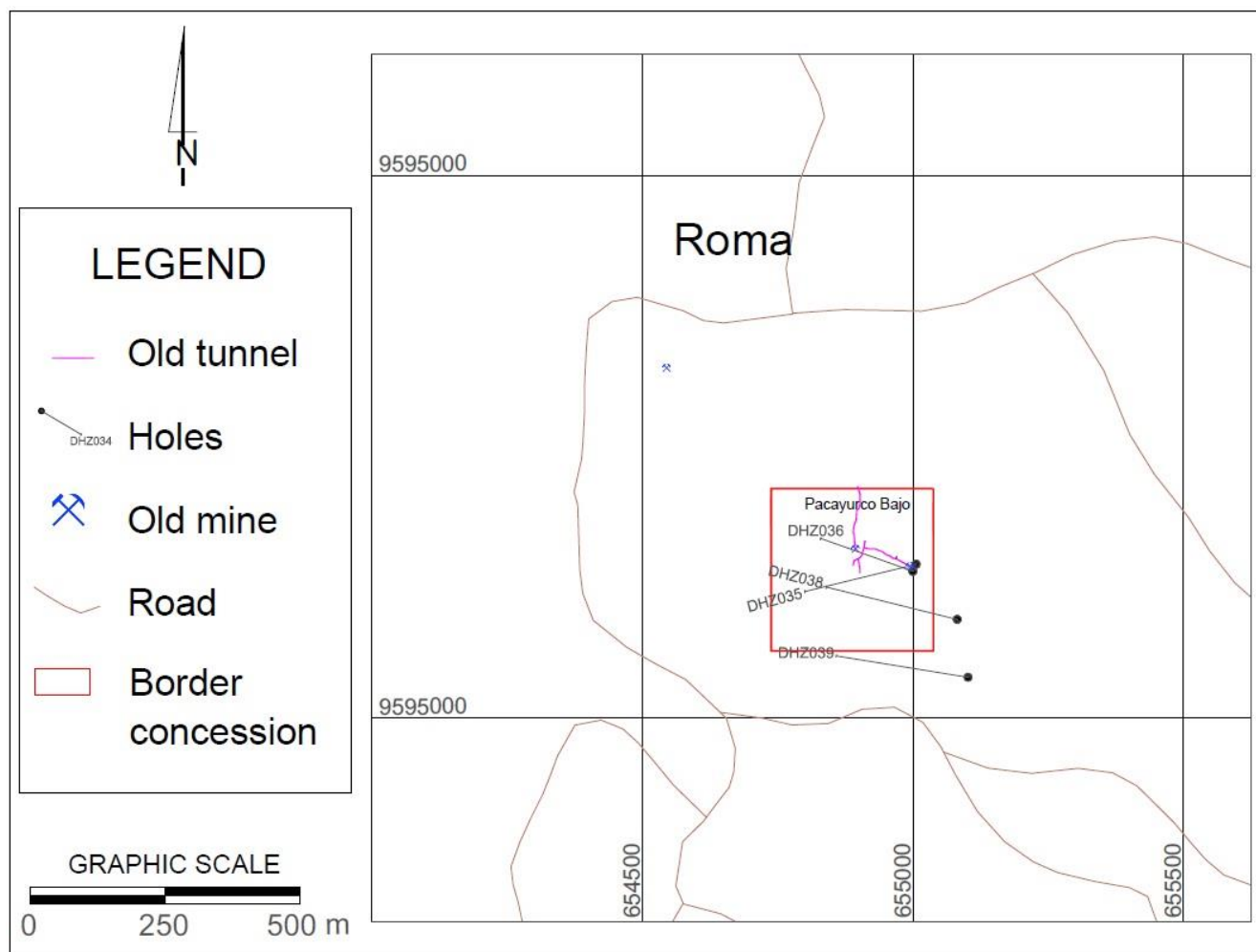
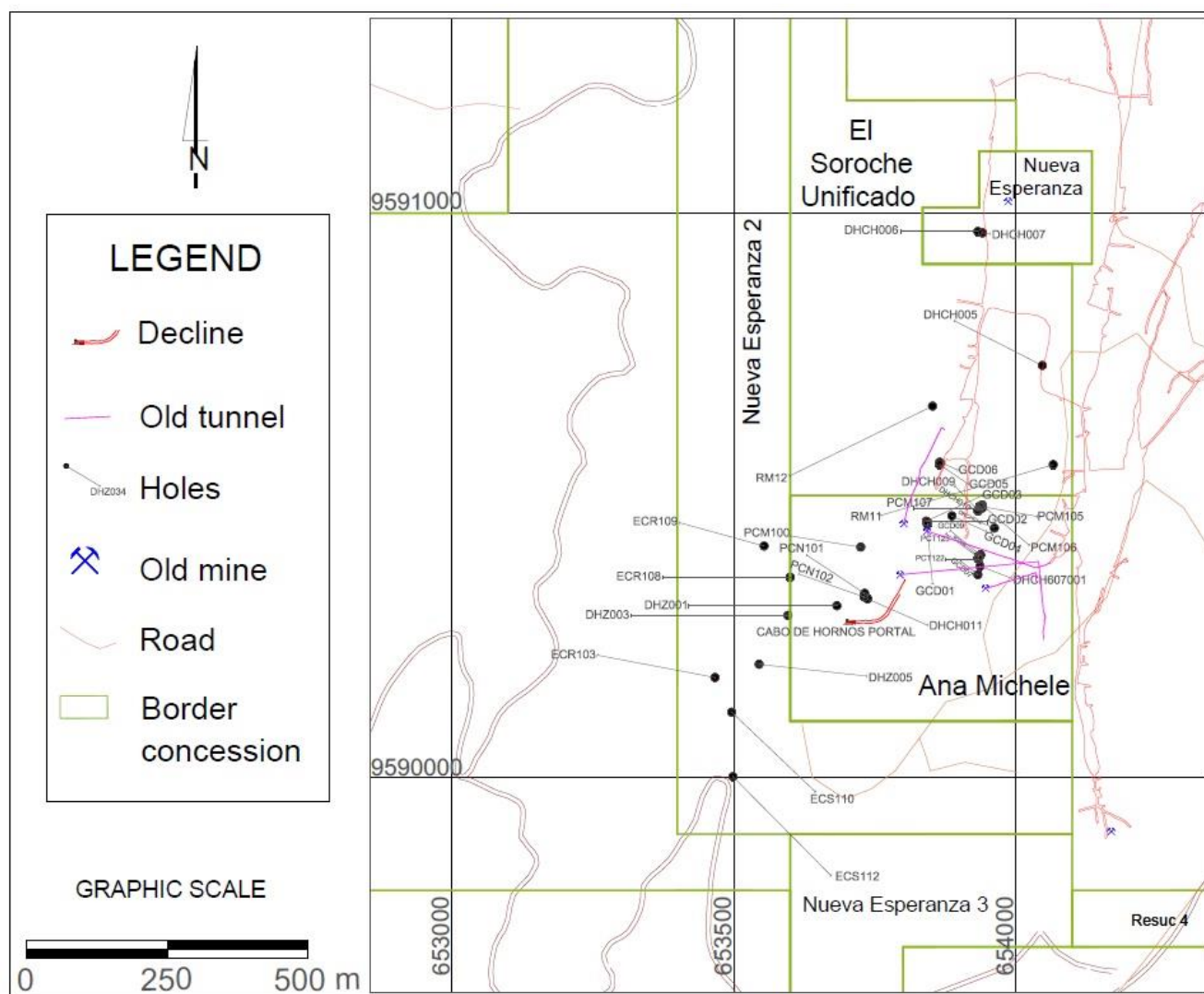


Figure 30: Drill hole locations at Pacayurco.



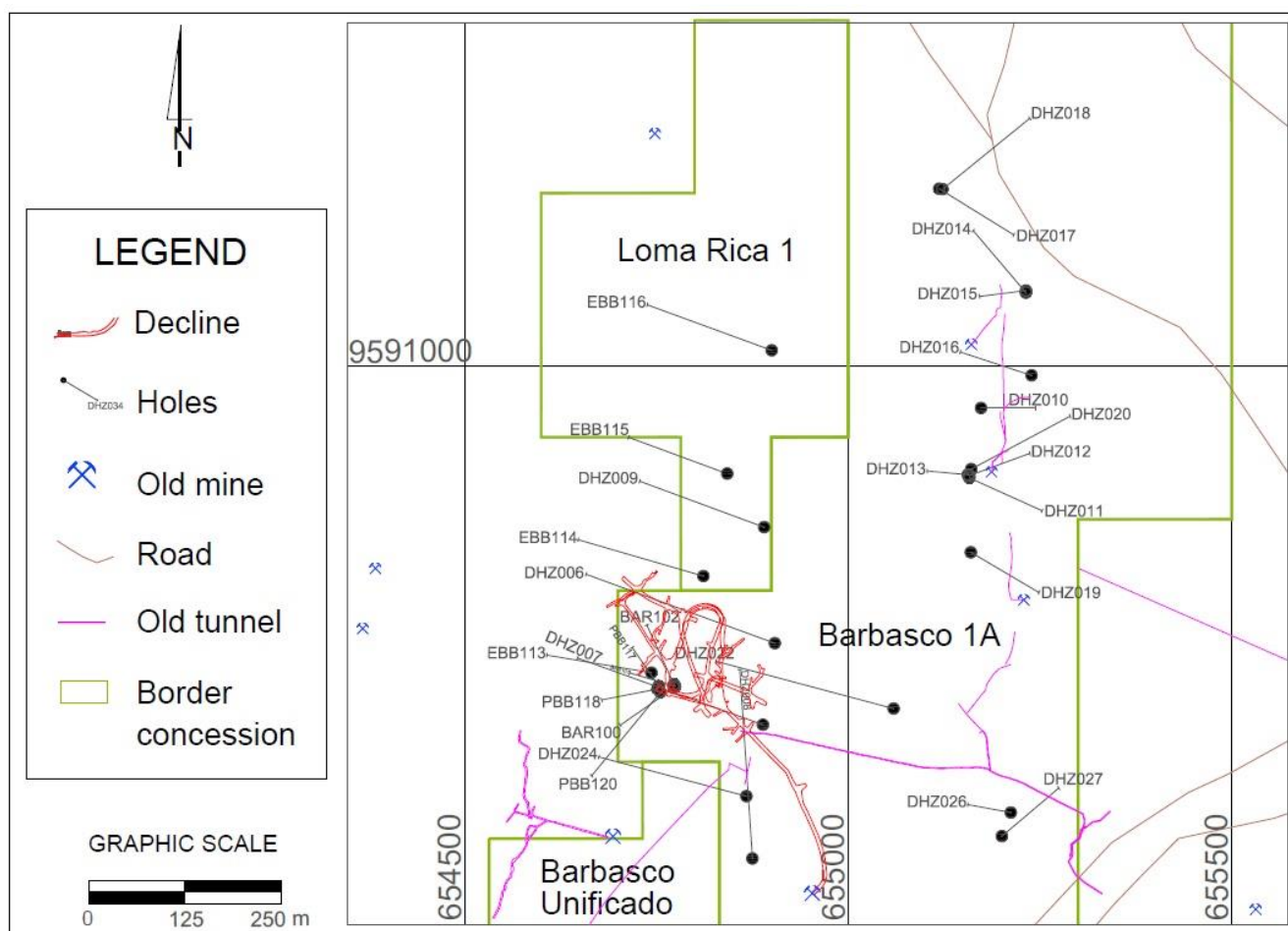


Figure 32: Drill hole locations at Barbasco.

Of the 28 drill holes completed at Cabo de Hornos, 9 were for geotechnical investigations. The exploration diamond drill holes collared from surface at Cabo de Hornos tested for new veins further west of the known mineralized trend but only narrow low grade structures were intersected to the south and southwest of the mine. A later drilling program from underground tested the vertical continuity of the Tamayo, San Ernest, Matalanga, Nudo and Soroche veins. The northern sector drilling at Soroche Unificado intersected veins with economic gold and silver values.

Twenty eight exploration drill holes were drilled at Barbasco to test the Clara, Barbasco and Barbasco West veins. Twenty two of the holes were collared at the surface and six collared underground in the mine. Eight of the drill holes intersected three major veins called Daniela, Silvia and Clara yielding promising results.

Twelve exploration drill holes were collared at the surface at Ayapamba in the Northern District and confirmed the continuity of Sucre and Tres Diablos veins to the south in an area formerly operated by CIMA Co. and SADC. The intercepts in the Tres Diablos and Bolivar veins however yielded low grades.

Five diamond drill holes to 350 m deep at Pacayurco tested gold and silver targets reported historically but no significant grades were intersected by this drilling.

Epithermal textures in the quartz veins along with lead, zinc and copper sulfide mineralization were intersected at Playitas and La Colorada in the El Derrumbo and Ramon veins from seven exploration drill holes up to 350 m deep. The gold and silver values however were poor in these three veins.

Only the vein intersections and intense hydrothermal alteration zones were sampled for chemical analysis. The samples were numbered, labelled and with the corresponding security measures were sent to BSI Inspectorate of Ecuador and/or ALS for analysis.

Hole ID	Total Depth	From (m)	To (m)	Length	Au g/t	Ag g/t	Cu, Pb, Zn
Ayapamba-Bethsabeth							
DHZ-021	183.8	121.75	122.1	0.35	7.0	131.6	
DHZ-023	241.7	195.82	195.9	0.08	40.7	285.6	
DHZ-031	347.5	195.6	196.6	1.00	5.1	106.9	
		220.8	221	0.2	10.4	71.8	
Barbasco							
DHZ-006	371.6	333.89	334.36	0.47	5.1	82.1	
DHZ-010	101.3	38.75	41.1	2.35	6.1	16.9	
DHZ-014	150	101.3	102.35	1.05	5.1	116.4	1.7%Cu
DHZ-014		126.25	126.8	0.55	5.2	8.6	
DHZ-018	206.4	47.2	48.2	1.00	2.6	9.5	
				Includes: 0.2	8.9	30.9	
DHZ-022	346.95	230.8	231.07	0.27	4.8	52.4	
DHZ-024	299.75	118.6	121.25	2.65	0.6	14.5	0.15%Zn
		125	125.25	0.25	8.1	11.7	0.24%Pb/1%Zn
EBB-113	201.6	109.95	110.5	0.55	8.54	138.5	
PBB-117	86.4	52.67	53.07	0.40	15.2		
		54.94	55.08	0.14	7.3	9.0	
Cabo de Hornos (Ana Michelle)							
DHCH 001	87.6	48.55	49.5	0.95	35.3	35.6	
DHCH 008	88	75.8	75.85	0.05	49.2	43.5	
Cabo de Hornos (Soroche Unficado)							
PCM-107	176.2	91.2	91.27	0.07	5.8	17.0	3.1%Zn
		149.35	150.55	1.2	7.3	10.7	
PCT-123	101	69.77	70.18	0.41	6.2	78.5	
PCT-124	79.05	44.5	44.75	0.25	5.5	175.5	

Table 11: Summary of Representative Drill Hole Intervals.

All the drilling, both collared at the surface and underground, is inclined at various angles as close as practical to perpendicular to the interpreted strike of the mineralization however collar sites may be located “off section” due to accessibility. The penetration angle of the drill intersection in the mineralisation therefore varies and will often be significantly longer than true thickness of the mineralisation. The resource estimation method however uses true

widths calculated using trigonometry based on the interpreted dip and strike of the mineralisation and the orientation of the drill intersection.

10.2 Sampling Method and Approach

10.2.1 Diamond Drilling

The drilling at Zaruma was all diamond drilling using wire line triple tube core recovery equipment to maximise core recovery. The first 5 drill holes the holes were drilled entirely with BQTK (hole diameter = 60.0mm, core diameter = 40.7mm) equipment with the remaining drill holes collared using HQ (hole diameter = 96mm, core diameter = 63.5mm) drilling equipment and usually from about 80 m the drilling equipment was reduced to NQ (hole diameter = 75.7mm, core diameter = 47.6mm). The drill holes have generally been located at regular intervals along the strike of the target veins at predetermined section intervals but accessibility considerations due to the steep topography may require that some holes are actually located “off section”.

The drill core recoveries are recorded by the geologists while logging the core prior to sampling. Drill control included having a drill geologist who sited in drill setups, aligned drills and visited drills throughout the day. Continuous monitoring of the drills ensured any drilling problems were noted, and helped ensure good core handling practices were maintained by drill crews. Table 12 provides a summary of the core recoveries.

Item	Number of Holes	Total depth of Drilling (metres)	Core Recovery in Waste (%)	Metres drilled in ore	Core recovery in ore (%)
Total/Average	54	12,024.6	98.4	105.9	99.4
Minimum (per hole)			84.5		95
Maximum (per hole)			100		100

Table 12: Summary of Drill Core Recoveries in Dynasty Drill Holes

10.2.2 Underground Sampling

Of the 27 principal veins at Zaruma–Portovelo, the Company has initially targeted exploration on four under-explored target areas: i.e. Sucre, Pacayurco, Tamayo and Tablon. Surface and underground sampling and mapping by Dynasty has extended the previously known strike and down dip extent of most of the veins.

The historic underground workings were sampled in 2005. All the tunnels sampled were surveyed by qualified surveyors. Samples were collected across the backs (roofs) of the tunnels from channels approximately 15cm wide by 10cm deep. Samples were differentiated according to mineral character, which determined sample lengths. Initially samples were collected at 6m spacings along the mineralised structures. Infill sampling later reduced the sample spacing to every two metres in some workings.

No factors impacting on a fair collection of the underground channel samples are apparent from the data available within the project areas. The quality of historical underground sampling provides a reasonable basis to calculate resource estimates and plan further investigations.

A total of 56 underground channel samples were taken from the Sucre Vein over widths ranging from 0.02 m to 2.1 m. Assay results ranged from 0.01 g/t to 55.62 g/t gold and 0.8 g/t to 278 g/t silver. Along with the drilling results, the strike length of Sucre has increased by 400 m and the depth of the vein increased by 90 m.

In all, 36 underground channel samples were collected from the quartz-sulphide Pacayurco Vein. Assay values ranged from 0.02 g/t to 99.57 g/t gold and 0.3 g/t to 62 g/t silver. Sample widths vary from 0.3 m to 2.4 m. Along with the drilling results, the strike length was increased by 800 m and down dip extent extended by 36 m.

Underground and surface mapping and channel sampling is ongoing at Tamayo and Tablon.

10.2.3 Adequacy of Sampling Method and Approach

The quality of sampling in epithermal quartz veins like those found at Zaruma is critical. The quartz veins are often very friable and much of the mineralisation can occur as coatings on the outer surfaces of the quartz and other mineral grains and along partings so particles of gold can be easily selectively lost resulting in biased samples. Sampling coarse nuggetty gold can also be difficult and the best way to get a representative sample where nuggetty gold is found is to take as large samples as possible, so with drilling the larger the core size the more representative the sample. The authors have concluded that the sampling methods and approach, including the drill core sample size and recoveries and sample spacing, are all appropriate and meet all the necessary mining industry standards.

11.0 Sample Preparation, Analyses and Security

11.1 Previous Owners

Limited information about sample preparation, analyses and security is available from SADCO's times, however, details of production and plant registers confirm the company's assay database totalling 103,657 assay results, which is in the possession of Dynasty. SADCO's on-site laboratory used a 3-acid digestion and gravity and/or fire-assay analysis for gold and silver determinations while base metals were determined by quantitative spectral assaying. The quality control data has not been preserved and it is likely that SADCO did not use systematic analysis control.

TVX sample databases are missing quality control samples, however, Mr Marislav Kalinaj, being one of the TVX's Zaruma Gold Project geologists, confirmed that systematic procedures were controlled by standard and duplicate samples inserted with a frequency of 2 standards, 2 field duplicates and 2 laboratory duplicates samples per hundred samples sent for assay.

Samples from surface, underground and drilling core collected by IAMGOLD were bagged on site, sealed and sent to an independent certified and registered laboratory (Chemex Labs, now ALS Geochemical) for assaying. ALS is a certified laboratory in more than 80 countries around the world and has been operating in Ecuador since 1997. Its sample preparation facilities are located in Quito. Samples are sent to Canada or Peru for analytic testing depending on the client's requirements and the complexity of the chemical tests. Blanks and standards were routinely submitted with these samples at the rate of 1 standard per 20 samples and blanks inserted at random but not less than 1 blanks per 50 samples.

The standard procedure included sample homogenization at -80# and -150#. The Au and Ag content was determined by fire assay using 50g pulps. Induced Coupled Plasma (ICP) and/or Neutron Activation (NAA) methods were employed for the multi-element analysis of 35 elements (Table 13). The procedures followed for sample preparation, identification, classification and delivery for chemical analysis are appropriate and within the norms established by the mining industry.

Element	Method	Units	Low limit	High Limit
Au50	FA-50	PPB	5	10000
Au Rew	1	PPB	5	10000
Au Rew	2	PPB	5	10000
Au Gra	v FA	PPM	0.17	999999
Au Wt	1 FA-AA	GM	0.01	999.99
Ag	ICP	PPM	0.2	200
Cu	ICP	PPM	1	10000
Pb	ICP	PPM	2	10000
Zn	ICP	PPM	1	10000
Mo	ICP	PPM	1	10000
Ni	ICP	PPM	1	20000
Co	ICP	PPM	1	20000
Cd	ICP	PPM	0.2	2000
Bi	ICP	PPM	5	2000
As	ICP	PPM	5	10000
Sb	ICP	PPM	5	2000
Hg	CV AA	PPM	0.01	50
Fe	ICP	PCT	0.01	10
Mn	ICP	PPM	1	20000
Te	ICP	PPM	10	2000
Ba	ICP	PPM	1	2000
Cr	ICP	PPM	1	20000
V	ICP	PPM	1	20000
Sn	ICP	PPM	20	2000
W	ICP	PPM	20	2000
La	ICP	PPM	1	2000
Al	ICP	PCT	0.01	10
Mg	ICP	PCT	0.01	10
Ca	ICP	PCT	0.01	10
Na	ICP	PCT	0.01	10
K	ICP	PCT	0.01	10
Sr	ICP	PPM	1	2000
Y	ICP	PPM	1	2000
Ga	ICP	PPM	2	10000
Li	ICP	PPM	1	20000
Nb	ICP	PPM	1	10000
Sc	ICP	PPM	5	2000
Ta	ICP	PPM	10	1000
Ti	ICP	PCT	0.01	5
Zr	ICP	PPM	1	5000

Table 13: Elements Assayed by IAMGOLD and Detection Limits for the ICP Method Applied for Sample Analyses at Zaruma.

11.2 Dynasty

Dynasty employed the following sampling methods, protocols and security measures for its exploration activities:

Diamond Drill Core

- After the core has been packed into core trays at the drill rig, the core trays are sealed with covers to prevent spillage and excess movement of the core and transported carefully in the back of a flat trayed pick-up along the main paved road to a core storage facility located at Dynasty's Divino Niño store 6 kms from Celica town.
- After the core has been carefully unpacked, any impurities and dirt on the core is cleaned off by careful washing.
- A qualified geologist logs the core including such parameters as: core recovery, lithology, mineralogy, vein and fracture orientations etc.
- The intervals to be sampled are clearly marked by a geologist with the ends and the cutting axis perpendicular to the vein dip clearly indicated. Dynasty sampled only the drill intercepts containing quartz veins or alteration that the logging geologist deemed likely to carry gold, silver or base metal mineralisation.
- The core is photographed both wet and dry with the sample markings clearly visible.
- The core is then cut by a trained operator with a diamond saw along the sample lines indicated by the geologist. The samples normally weigh 1.5 kg to 5 kg.
- Each sample bag is weighed and hermetically sealed with the identification code visible, which is registered with sample description, weight and depths down the hole recorded in a log book.
- Individual sample bags are put into larger sacks, up to six samples in each, and then securely transported to the BSI Inspectorate of Ecuador laboratory under supervision by Dynasty personnel after a detailed check by the Security Department before leaving the mine. At least four standard samples and blanks are included for quality control with each sample batch sent to the laboratory.
- A receipt for each sample lot including a full inventory of the samples submitted is provided by the laboratory.
- Assay results are generally received after 20 working days.

Trench and Underground Channel Samples

- A trained geologist or supervisor first evaluates the area to be sampled taking into account safety and then the geology to define the number of channel samples to take. At each sample site the geologist records in a fieldbook the technical parameters such as: sample orientation, sample width, structural character of the rock, mineralogy etc.
- Any impurities are cleaned off by chiseling to a depth of at least 3 cm over at least the width of the vein being sampled.
- Continuous channel samples are collected over the veins and mineralized structures being sampled, perpendicular to the strike of the mineralisation, using chiselled channels 5 cm – 10 cm wide and 3 cm – 5 cm deep over the total length of the vein. The sample chips are generally less than 3 cm in diameter.
- The sample is collected on a polypropylene sack laid below the sampling area. Once the sampling is complete the sample is gathered up and placed into a plastic bag identified with a unique code. The samples normally weigh 1.5 kg to 5 kg.
- Each sample bag is weighed and hermetically sealed with the identification code visible, which is registered with sample description, weight and location recorded in a mining log book.
- The sample location is located by tape and compass from a nearby survey station or using a "total station" survey instrument. The location is then recorded on all the relevant maps and cross and longitudinal sections. An aluminum tag is left at the sample site to identify the location.
- Individual sample bags are put into larger sacks, up to six samples in each, and then securely transported to the BSI Inspectorate of Ecuador laboratory under supervision by Dynasty personnel after a detailed check by the Security Department before leaving the mine. At least four standard samples are included for quality control with each sample batch sent to the laboratory.
- A receipt for each sample lot including a full inventory of the samples submitted is provided by the laboratory.
- Assay results are generally received after 20 working days.

Dynasty's samples were chemically analysed at the independent BSI Inspectorate of Ecuador laboratory located at Guayaquil, an ISO certified laboratory operating since 2001. The gold is analysed by fire assay with an atomic absorption spectroscopy (AAS) finish while the silver and 31 other elements are analysed by the Inductively Coupled Plasma (ICP) method.

11.3 Adequacy of Sampling Preparation, Analyses and Security

Sampling procedures and quality assurance have been conducted in conformance with standards applicable to the industry, including CIM standards. There are no actions or recommendations required on drill data at the time of this report.

The authors have concluded that the sample preparation methods, transport, QA/QC procedures and chemical analysis methods are all appropriate and meet all the necessary mining industry standards and the analytical results are suitable for resource estimation.

12.0 Data Verification

12.1 General

There is no information available on sampling and assay quality control in both SADC's 103,657 sample database or in the 733 samples from TVX's database.

The Company's ongoing sampling programme within the Zaruma Gold Project area included re-sampling original channel samples to verify the accuracy and repeatability of the original assay results, in particular for high-grade samples.

Kalinaj (2004) reports that sample assay results reported by IAMGOLD, a total of 17,448 samples of which only 9,335 samples were from the Zaruma Gold Project area, have an appropriate QA/QC quality control consistent with current standards in the mining industry. One of three commercially prepared standards was randomly inserted at a frequency of 1 standard per 20 samples, with a total of 213 standard samples being inserted. Blank samples were inserted randomly with not less than 2 blanks per 100 samples.

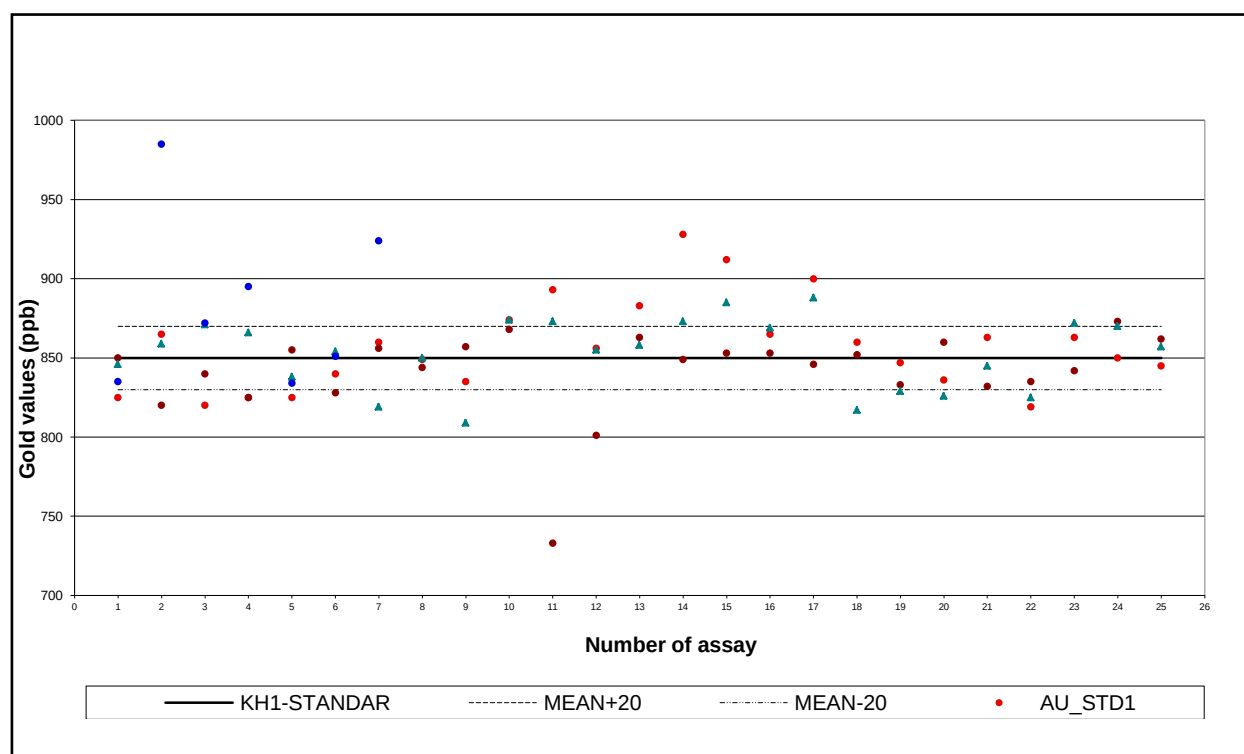


Figure 33: KH – 1 Standard Assays, IAMGOLD Database.

The behaviour of the standards demonstrates acceptable levels of dispersion with a stable performance for standard KH-1 with all results falling within the range of $\pm 15\%$ and are considered "good" (Figure 33). Standard OX-4 (Figure 34) gave higher dispersion values reaching in sample Au_STD33 up to a 80% higher value than expected and in the case of sample Au_STD20 almost 60% less.

Despite these two extremely out-of-range samples, the performance of standard OX-4 can be considered satisfactory.

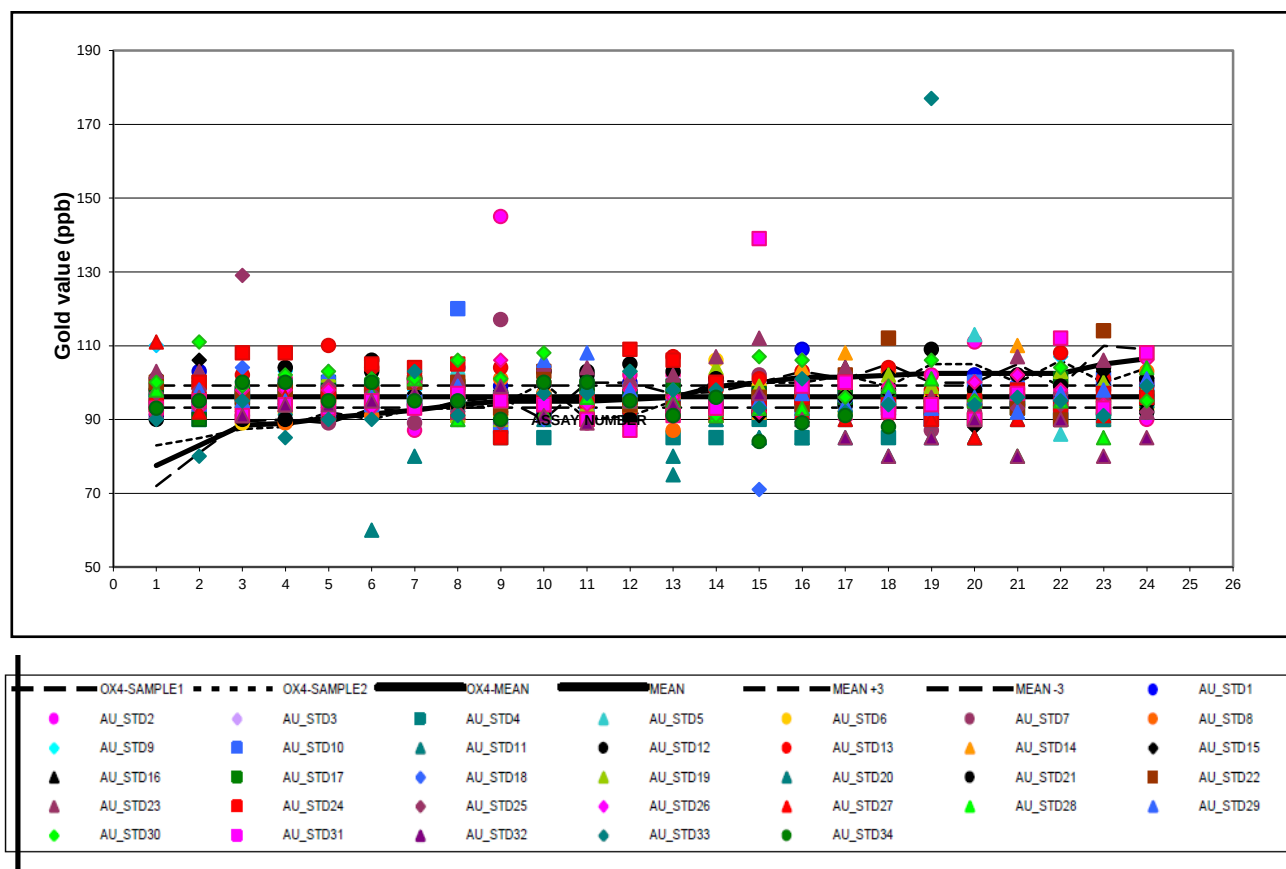


Figure 34: OX-4 Standard Assays, IAMGOLD Database.

No relevant information is available regarding procedures used with sample batches that contained the standards Au_STD33 and Au_STD20. The performance of standard OX-8 (Figure 35) is considered "very good", particularly in sample batches submitted to the laboratory in late 1999 and before June 2000 when the QA/QC charts above were completed.

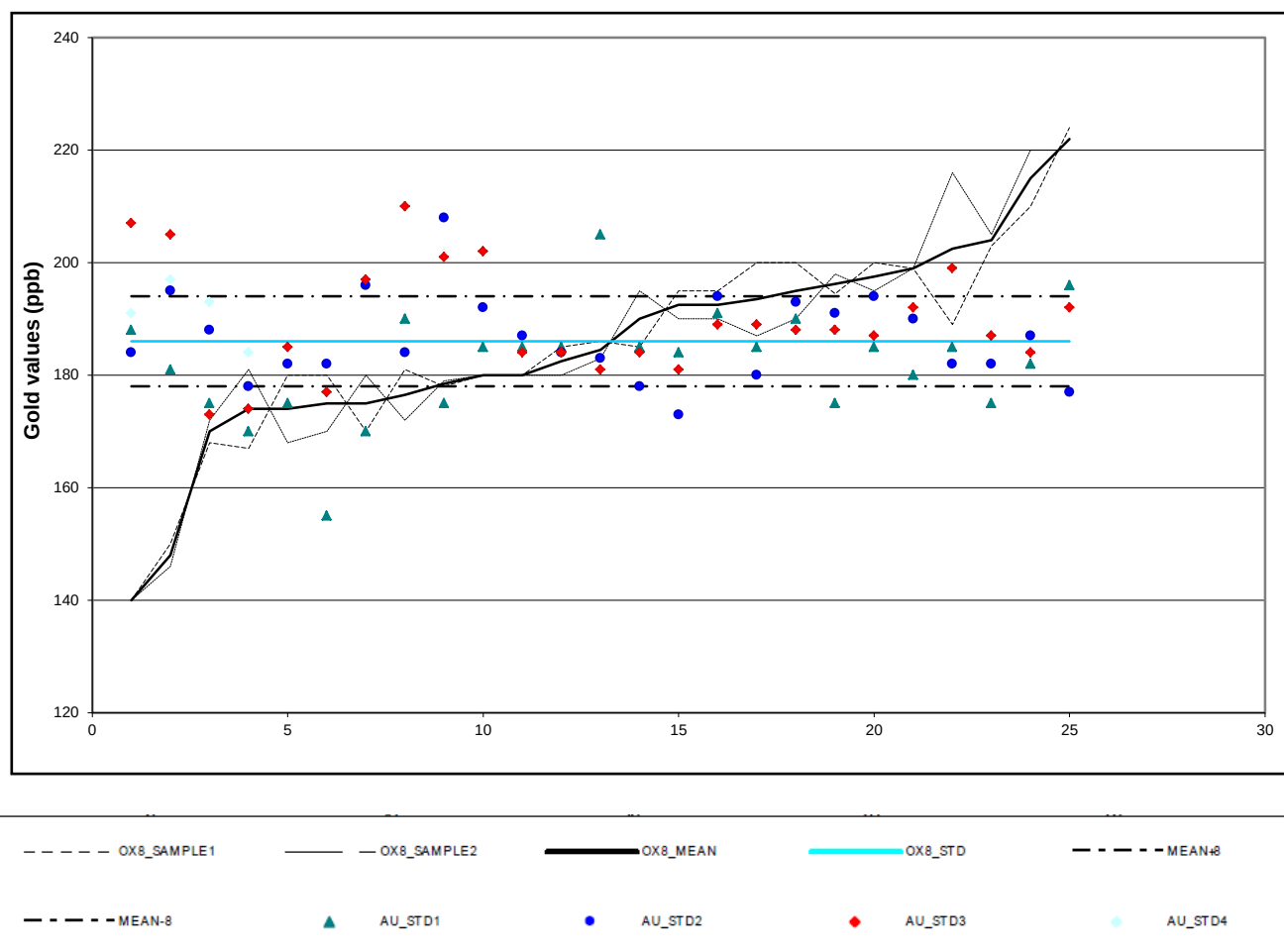


Figure 35: OX-8 Standard Assays, IAMGOLD Database.

Original laboratory certificates for IAMGOLD and TVX were not available for this report. Sources of data used for mineral resource calculations were spreadsheet files on four compact discs provided by Dynasty. Kalinaj confirmed the fidelity of the data by comparing the database with a second copy of the data obtained from the IAMGOLD office directly. Files originated with TVX dated from April 1995 to March 1996.

The IAMGOLD drill hole data contains complete results and was confirmed directly in the IAMGOLD offices in Quito. The tunnel sample data from all veins in the Zaruma-Portovelo District have been included that are within the land tenements held and/or controlled by Dynasty.

12.2 Dynasty Standards

Underground sampling by Dynasty on the Sucre and Pacayurco Veins gave values consistent with results from previous sampling.

Dynasty used certified gold standards supplied by Gannet Holdings Pty Ltd. South Perth - Western Australia. The gold values of the standards and their standard deviations are recorded as in Table 14.

CODE	GRADE Au (g/t)	Standard Deviation
ST216	0.003	0.002
ST256	0.167	0.030
ST306	1.660	0.070
ST284	3.420	0.190
ST279	7.180	0.310

Table 14: Gold Values of Gannet Standards Used at Zaruma.

Randomly selected standards were inserted at the rate of one for every 20 samples submitted for chemical analysis. The analysis results for the Standards are shown in Figures 36 to 39.

The gold analyses for all the standards fell within an acceptable range of ± 2 Standard Deviations except for STD216 which varied well outside the acceptable range, but generally too low. It is concluded that all the mineralised analyses are reliable although the waste analyses at about the laboratory detection limit are less reliable and most likely to be biased low. These very low grade samples however have no effect on the resource calculations as they are all below the lower cut-off grade.

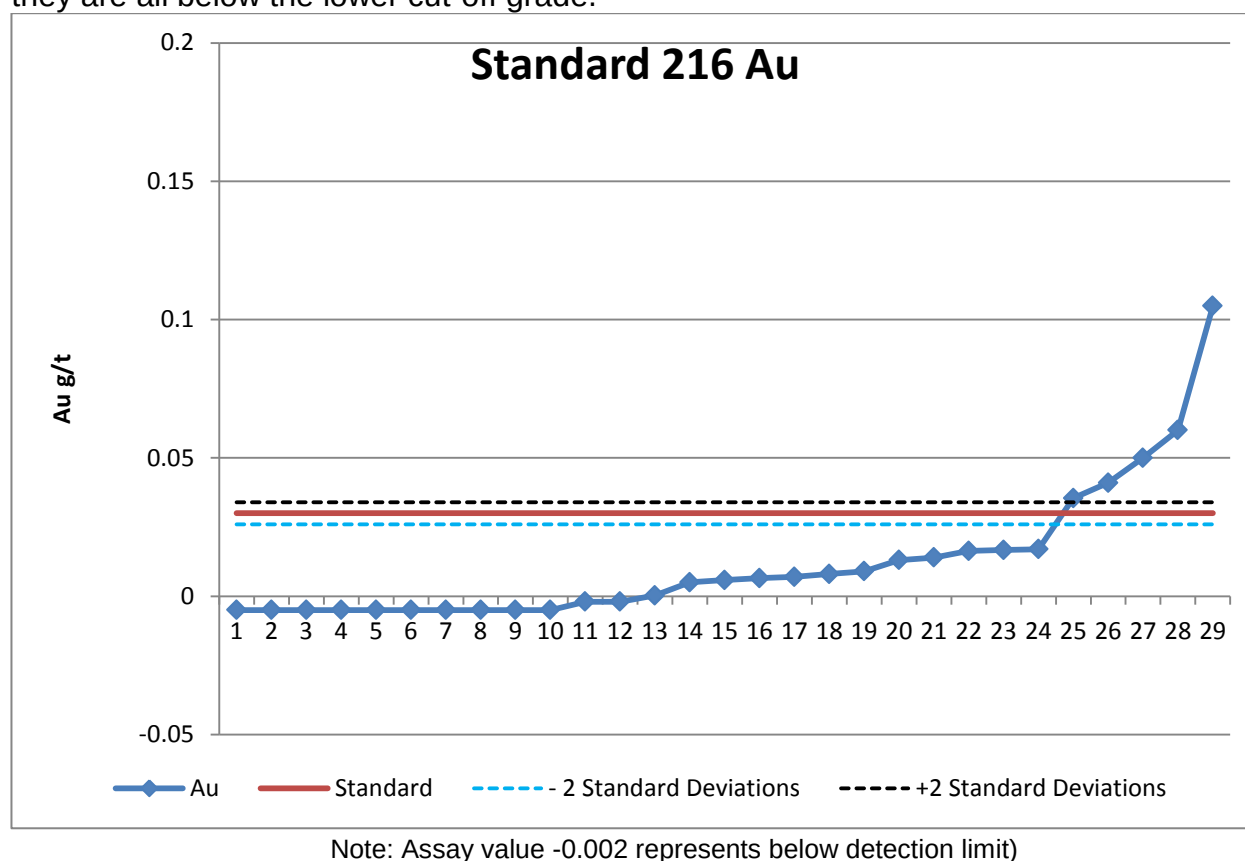


Figure 36: Plot of Dynasty's Standards Analyses (sorted by ascending grade) STD216.

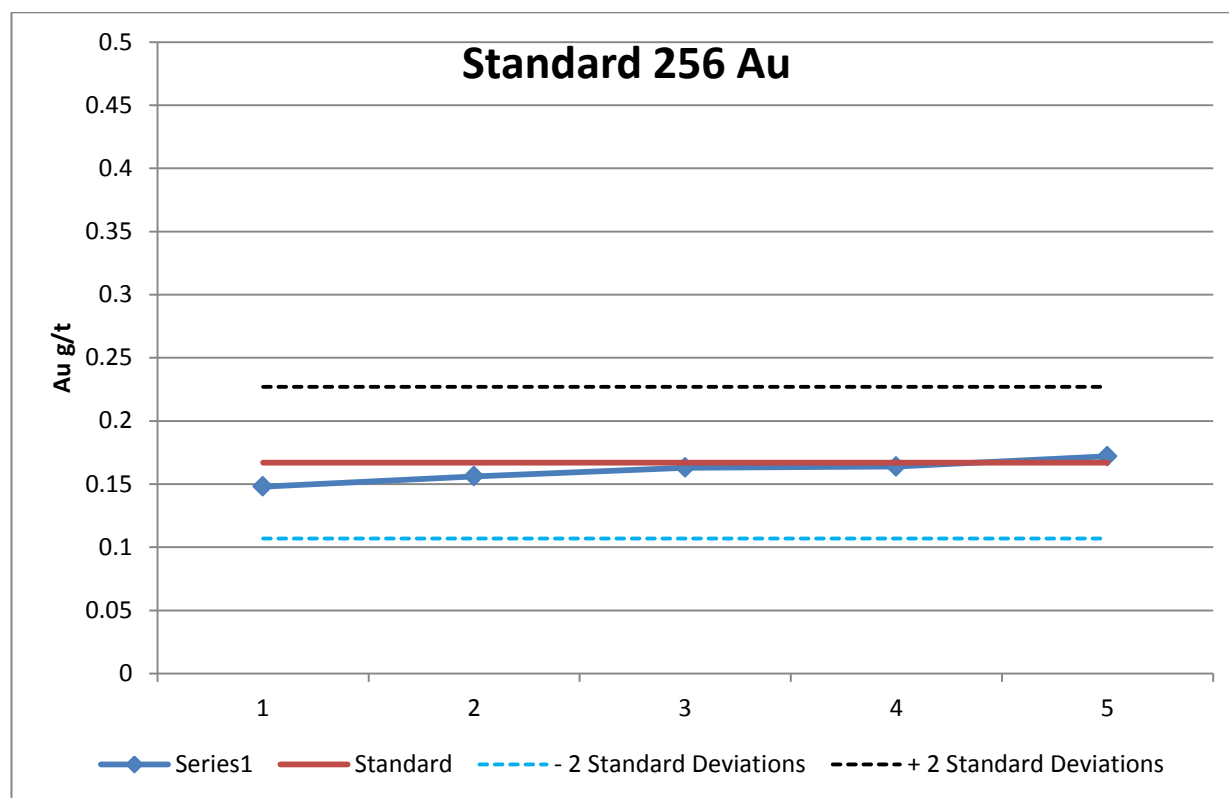


Figure 37: Plot of Dynasty's Standards Analyses (sorted by ascending grade) STD256.

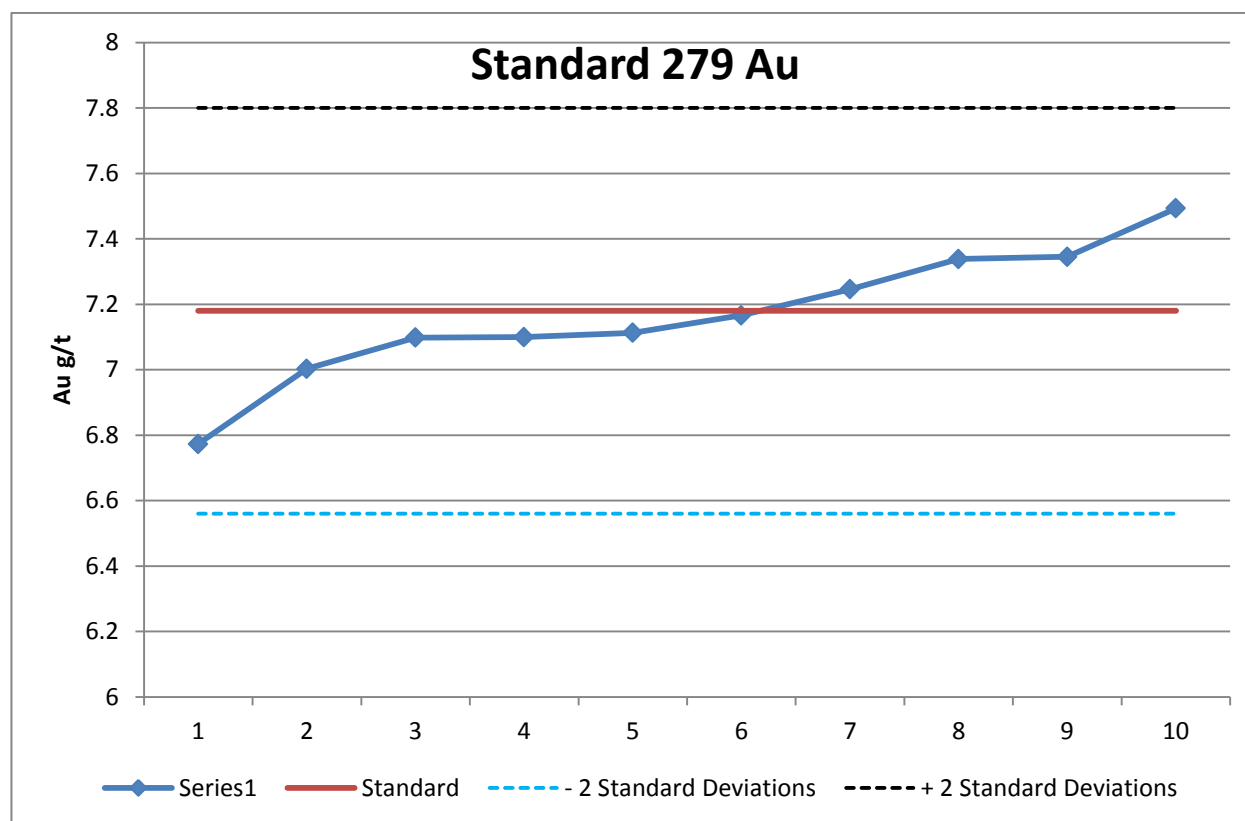


Figure 38: Plot of Dynasty's Standards Analyses (sorted by ascending grade) STD279.

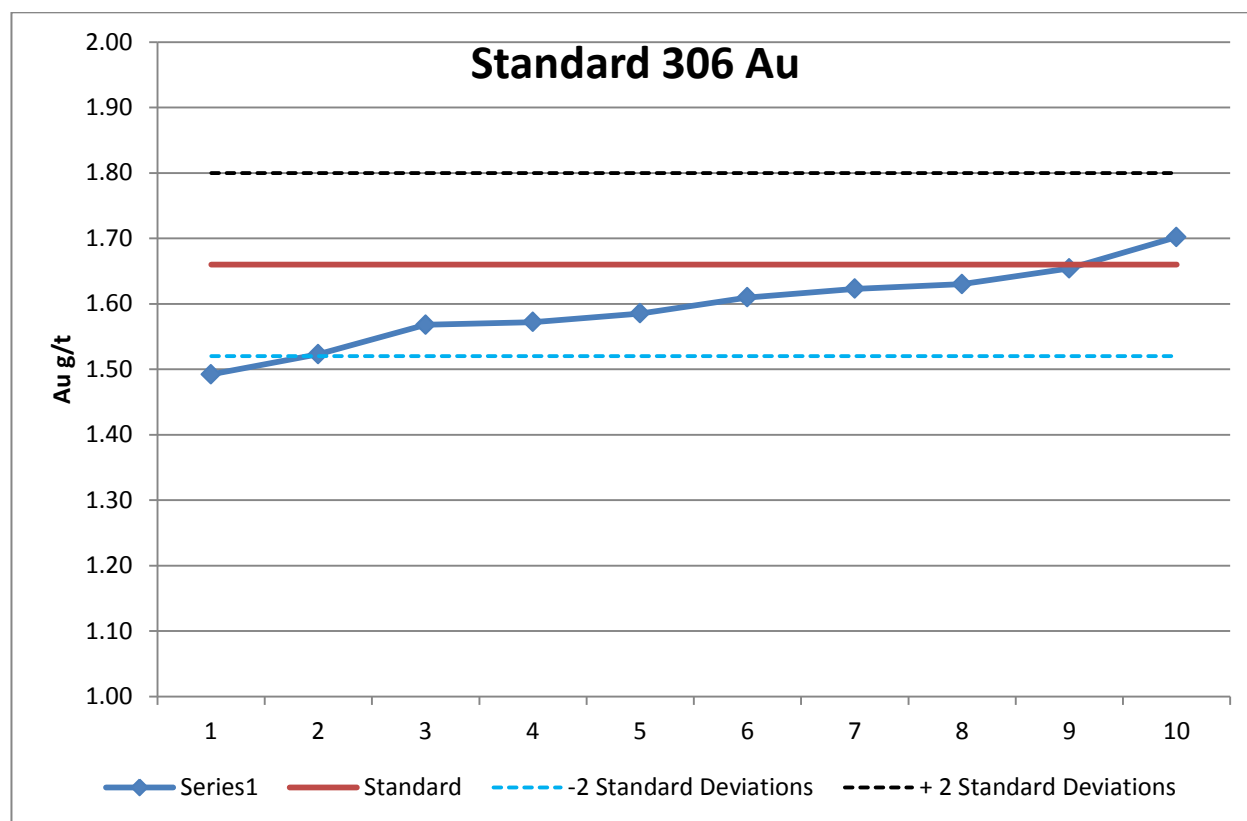
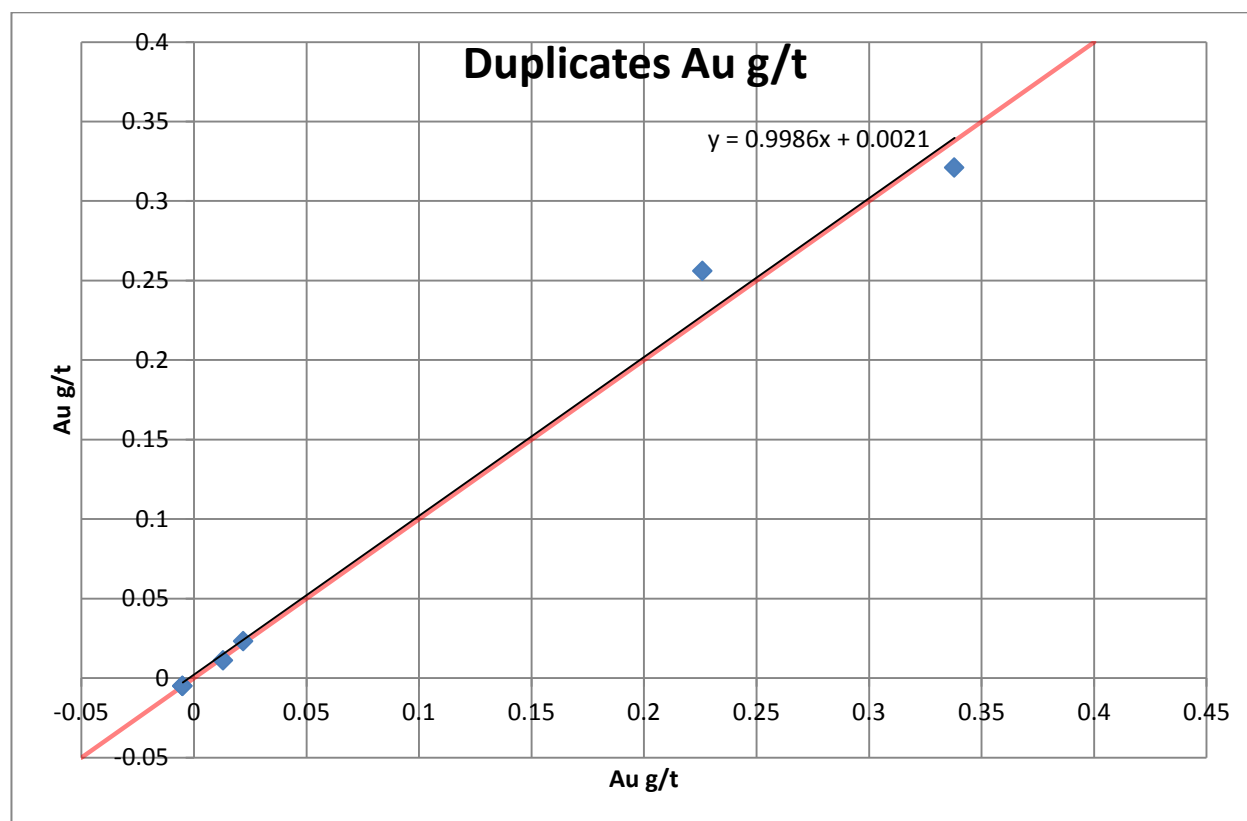


Figure 39: Plot of Dynasty's Standards Analyses (sorted by ascending grade) STD306.

12.2.1 Dynasty Duplicates

Randomly selected duplicate samples were also inserted at the rate of one for every 20 samples submitted for chemical analysis. The analyses of the duplicate samples are plotted in Figure 40. The plot indicates that the repeatability of the duplicate analyses is excellent.



Note: Assay value -0.002 represents below detection limit)

Figure 40: Plot of Dynasty's Duplicate Sample Analyses.

12.2.2 Dynasty Blanks

Blank samples, made up on site from country rock, were inserted randomly at a rate of one blank for every 30 sample sent for chemical analysis. The analyses of the blank samples are plotted in Figure 41. The plot indicates that the blank sample analyses are excellent and there is no evidence of contamination.

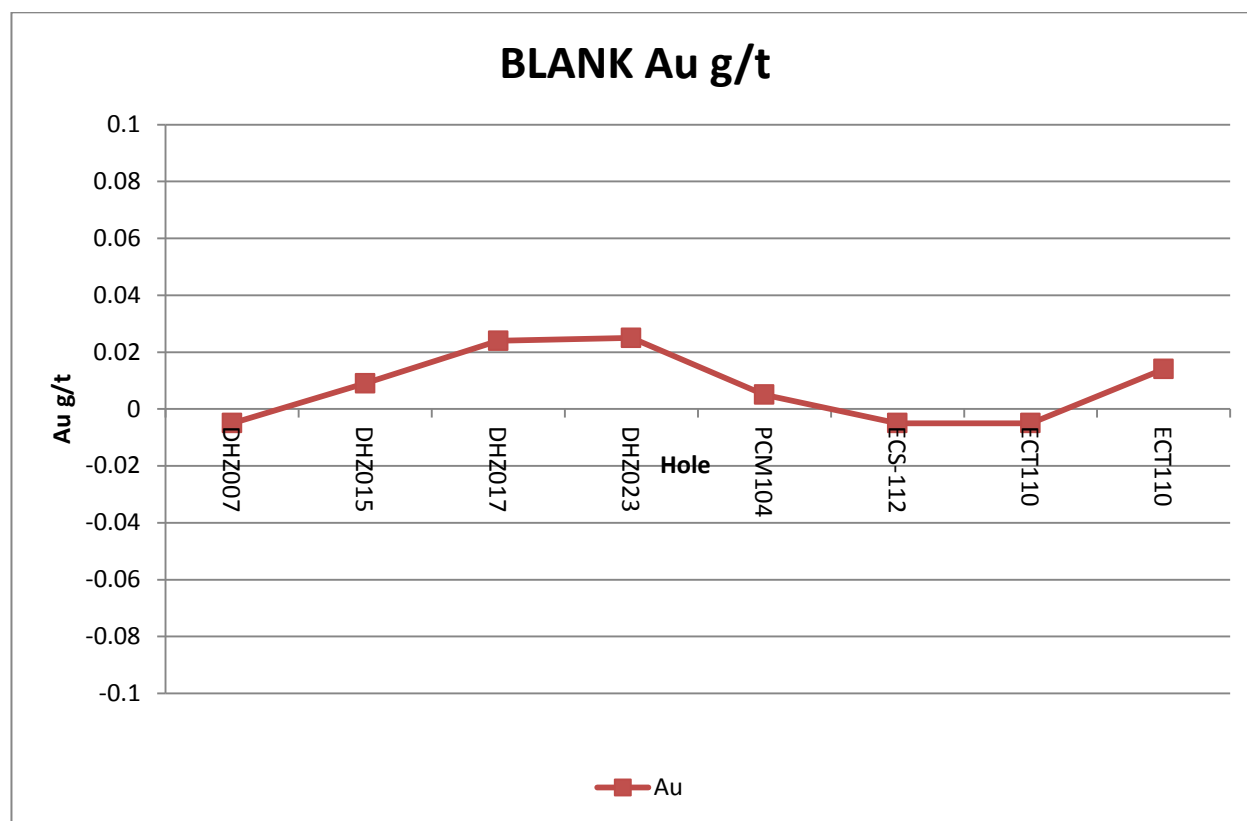


Figure 41: Plot of Dynasty's Blank Sample Analyses.

Note: Assay value -0.002 represents below detection limit)

12.2.3 Independent Check Samples

The QPs did not take independent check samples to verify any of the core or underground channel sampling since the pre-Dynasty drilling and sampling had been verified by earlier workers (Kalinaj, 2009) and the QA/QC procedures followed by Dynasty are sufficiently rigorous to ensure the assays used in the resource modelling are reliable. Mr. Maynard did visit the drill core storage and reviewed all the data from maps and stored on Dynasty's computers, and also reviewed the drill sites and assays. Underground workings were inspected by Mr. Maynard together with Dynasty's mine managers and exploration managers. Both Mr. Procter and Mr. Maynard reviewed the underground outcrops and production faces in the Soroche, Nudo and San Ernest veins. The original assay certificates were by both Mr. Maynard and Mr. Procter at Dynasty's Quito office.

12.2.4 Adequacy of Data

In the opinion of the authors, the data is sufficient and adequate for the purposes used in this Report including resource estimation.

13.0 Mineral Processing and Metallurgical Testing

Although the Project does not have a declared Mineral Reserve, it does have an operating history of some 4 years and currently the processing facility is providing an effective plant for the recovery of gold (and to a lesser extent silver) from the run-of-mine material.

The Project has an on-site laboratory facility located at the processing plant site and which carries out both fire assay and solution assays on a 3 shift basis. Currently some 40 samples (20 liquid and 20 solid) are tested daily. These data provide the facility with adequate checks for overall plant control and metallurgical balance sheet management. As well as this, the laboratory undertakes testwork with regard to the efficiency of the processing facility, whereby bench scale tests are carried out on flowsheet alternatives (such as intense cyanidation, options for flotation circuitry and leaching, absorption and tailings thickening work).

To this end, the Company has testwork equipment in its laboratory that enables it to appropriately test recovery efficacy at all stages of the flowsheet.

The processing plant is considered to be operating at a reasonably high level of metal recovery efficiency. For the 2013 calendar year, the operation achieved an overall gold recovery of 88.4% (compared to silver recovery of 41.6%). Figures 42 and 43 show the last 3 years of processing production performance. Figure 42 indicates the gold and silver produced, whilst the second provides the tonnes throughput and the metallurgical recovery for both gold and silver.

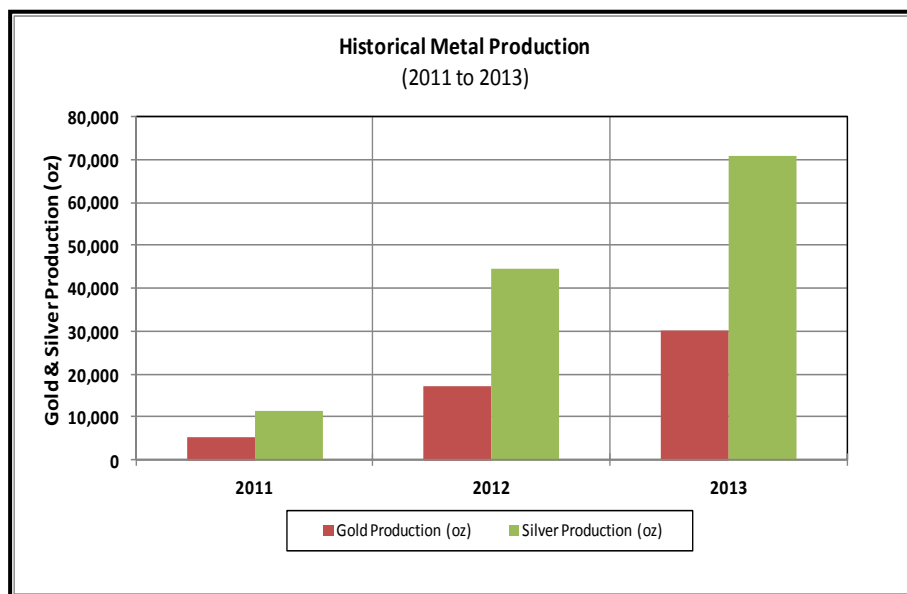


Figure 42: Historical Metal Production.

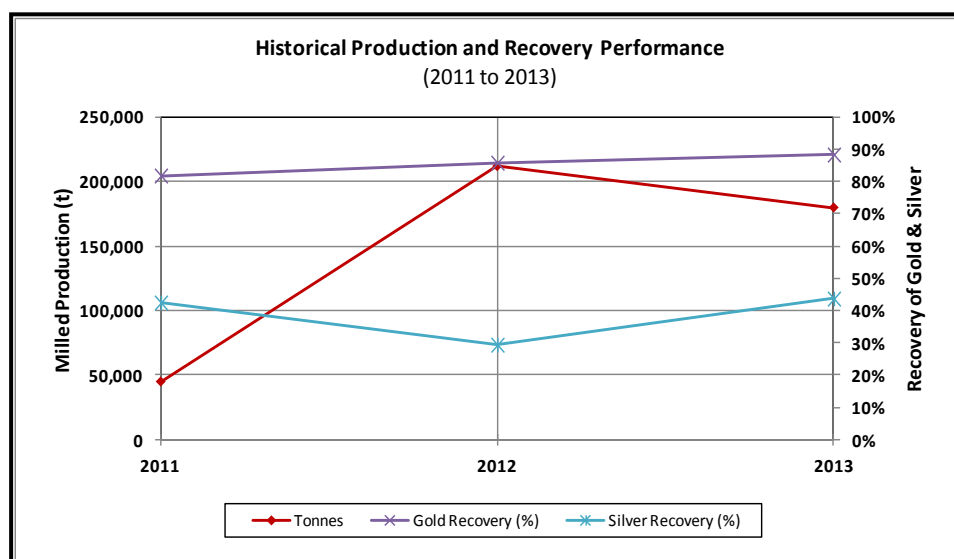


Figure 43: Historical Throughput and Metal Recovery.

Over the 3 years of operation provided in the data above, the average metallurgical recovery for gold and silver has been 86.3% and 38.3%, respectively, with silver metal output (oz) 2.4 times greater than that of gold.

More recently, production data for the last calendar year is provided in Figures 44 and 45. Although the gold grade is quite erratic (as is often the case in epithermal geological mineralised systems), it is evident that metallurgical recovery improves (albeit not directly proportional) with grade as shown in Figure 39.

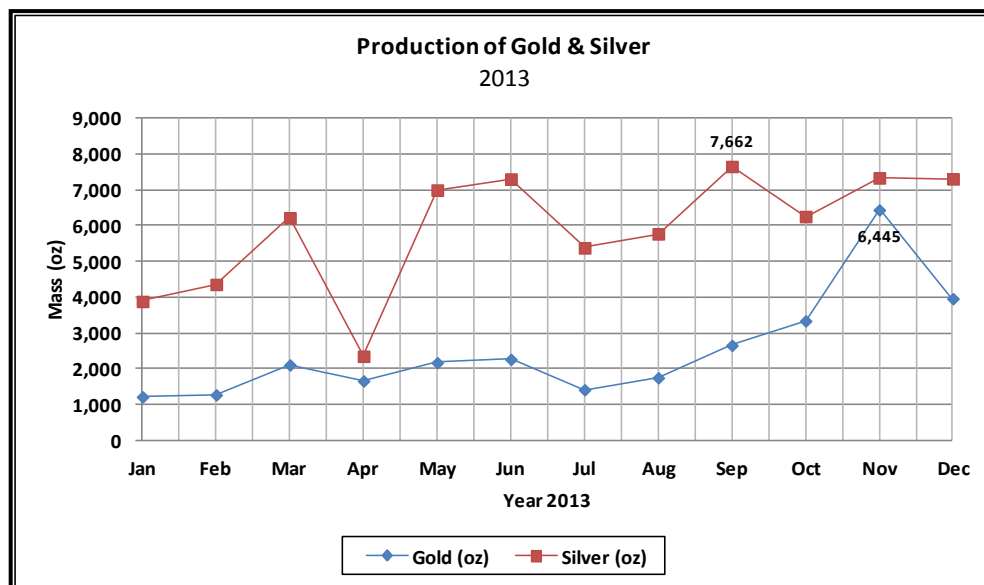


Figure 44: Precious Metal Production by Month for 2013.

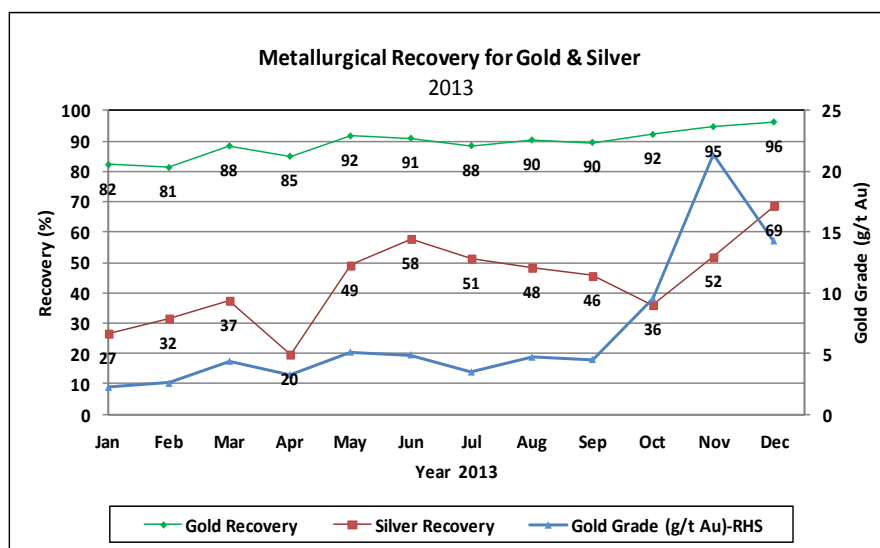


Figure 45: Metallurgical Recoveries and Gold Grade by Month for 2013.

The underground mine has a history of exploiting a number of different mineralisation types from various veins. However, due to the geology and type of mineralisation, there is no evidence of any deleterious refractory type mineralisation. The Zaruma mineral deposits are a 15 km long polymetallic epithermal vein system that lies north of the regional Pinas Fault, and the vein system is hosted by tertiary volcanics. The vein textures range from banded, massive to drusy quartz with some hydrothermal breccias. The mineralized vein portions vary in width from 0.4 m to 2.0 m (average 1.2 m) and show a zonation with chalcopyrite being the major sulphide in the west and sphalerite- and galena-dominant assemblages occurring in the east.

In summary, it is considered that gold recovery (the primary economic metal being recovered) is to a large extent dependent on the grade tenor delivered to the processing plant. Nonetheless, the processing plant's recovery performance is considered to be robust and predicting metallurgical recoveries between 85% (low) and 95% (high) is assessed to be valid. Based on the current Mineral Resource grade tenor (some 13 g/t Au in the Measured and Indicated category before the application of mining factors), a metallurgical recovery factor of 90% would appear to be a robust estimate to apply in the PEA. This value accounts for the markedly lower grade of material that has been mined over the last 3 years, versus the material expected to be delivered to the processing plant in the short term. It also considers the potential for more consistent and improved higher grade material recoveries by the Company implementing the use of intense cyanide leach processing systems.

It is also relevant that the grade to the processing plant increased markedly over the latter part of 2013 as the mining operations increased the proportion of mined material taken from actual Mineral Resource inventory displacing material previously mined from lower grade sources not included in the Mineral Resource inventory.

The underground mine is exploiting a number of veins and this material is being fed into the processing plant as ROM feed and processed without any deleterious effect on metallurgical recovery. It is therefore considered reasonable to assume that the material being delivered to the plant is representative of the various mineralisation types that will be encountered in the

medium term (possibly up to five years). However, given that the mining operation does not have a longer term (greater than 3 years) production schedule, it is not necessarily the case that in the future some type of refractory material will not be encountered.

Notwithstanding, the processing facility, which has a gravity concentrating circuit and ample leaching and CIP residence time available (35 hours), is considered adequately robust to treat multiple material types. The Company is also testing for the inclusion into the flowsheet of a flotation circuit. This would give the processing facility several recovery options for different mineralisation types. Such a flotation system would only be employed if the cost/benefit analysis clearly indicated that it was advantageous, on a gold production basis, to install.

Although there are different mineral assemblages within the various mineralisation types being treated by the processing facility, to date there is no evidence that there exists any material that would be overtly refractory and hence significantly adversely impact metallurgical recovery and therefore economic extraction of gold. However, the Company is examining two enhancements to the processing facility, both of which could have significant gold recovery advantages.

The first opportunity is to install a Gecko (or similar) intensive cyanide leach circuit. The potential immediate benefit of this system is that it will enable a very high grade stream of coarse gold (emanating from the Falcon gravity concentrator) to be separately treated and then fed directly to the electrowinning circuit (after pregnant solution recovery). This is a far superior method for high grade (coarse gold) recovery than the current system, whereby the gravity concentrate material is fed to the mill discharge sump and returned to the hydrocyclone as part of the recirculating load. This stream ultimately reports into the lower grade leach circuit, thereby undermining the benefit of the upfront extraction of the higher grade material in the gravity circuit.

The second opportunity involves consideration of a flotation circuit. This option is more complex than the first opportunity. However, given that both silver and copper minerals are present in sulphide mineralisation within some of the veins, there is merit in assessing the effectiveness of taking out a sulphide flotation concentrate, and then by differential applications recovering the precious metals and copper. Where in the flowsheet such a circuit may be positioned will need to be determined by possibly benchscale testing, but a strong case should be able to be made for its cost/benefit.

The Project is able to carry out nearly all its testwork requirements within the operating plant. This provides significant advantages as it essentially serves as a large-scale pilot plant. This eliminates many of the problems associated with bench scale testwork.

14.0 Mineral Resource Estimate

14.1 Introduction

Based on the results from geological investigations, Zaruma mineral resource estimates were completed by Dynasty geological staff under the supervision of Al Maynard & Associates who are independent Qualified Persons as defined by NI 43-101. Mr Maynard is a co-author of this Report providing verification of the resource.

The resource was estimated by reference to underground mine development and assay plans available from the period prior to 1950 when SADCO operated the field. Further investigation was carried out by way of underground sampling of lodes above the water table, and some diamond drilling by previous owners of the tenements, TVX and IAMGOLD. Dynasty has also undertaken some underground sampling. The water table is at a level such as to allow top access to some veins.

Further feasibility work will be required to determine the extent to which the estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, taxation, socio-economic, marketing, political and other relevant issues.

Although the Project has been in operation by Dynasty since late 2009, as yet the Company has not undertaken any feasibility study level engineering work and hence there is no Mineral Reserve estimate for the underground mining operation. However, the mine has active mining operations, and the elements of these operations are discussed below.

Since the mine re-started operating under Dynasty, relatively little of the material that has been mined and processed has been exploited from the Project's Mineral Resource inventory. The mining occurring outside of the known Mineral Resource boundaries is indicative of the many alternative veins available to the Project, both within and outside of known Mineral Resource areas.

The mining operation uses a cut-off grade to determine material likely to be economic based on the cut-off grade estimate calculation in Table 15. This is considered to be a reasonable conceptual basis on which to decide whether mined underground material should report to the processing plant, or sent to the waste stockpile.

Royalties	5 %	
Recovery plant	94 %	
Purity Au-Ag bullion	97.8 %	
Gold Price	1260 \$/oz	
Opex	73 \$/t	
Factor	0.87 %	
price Au	opex	cut-off
\$/oz	\$/t	g/t Au
1260	73	2.06

Table 15: Cut-off Grade Determination.

Assessing a number of scenarios for grade and gold pricing, it is considered that a reasonable cut-off grade range for the underground mine is between 2.0 g/t Au and 2.5 g/t Au. This assumes a metallurgical recovery of 94%, gold price of \$1,260/oz and a mining operating cost of \$73/t.

The data included in the resource model indicates that some of the very high grade material, for example in the Soroche sub-levels, has been encountered in the underground development. It is also relevant that the mine production grades are determined by bottle roll (direct leaching with cyanide solution) analysis, and not fire or atomic absorption spectroscopy (AAS) analysis, which may not be properly assaying coarse gold or gold included within sulphides so therefore the assay results may be somewhat conservative.

14.2 Method

Long sections were created along each vein separately with the topography as the upper limit. All the underground development within the vein was plotted on the long section. A polygon was then created around the development with a 25 m radius from the development for Measured, a further 25 m out for Indicated and another 75 m out for Inferred and the same polygon radii were used around drill intersections in each vein. The radii were based on and due to the mapped continuity both at the surface and underground. These distances were deemed appropriate by observing the continuity of veins/grades in the underground workings and comparing with drill intercepts. Average gold and silver grades and vein widths were calculated from the production face samples taken of the veins at approximately 3 m intervals along the drives. The areas of the polygons on the long sections were then multiplied by the average vein width then a bulk density of 2.65 to calculate the vein tonnes.

The bulk density of 2.65 used in the resource calculations is typical for vein quartz and was measured from a number of ore samples by SADCO, IAMGOLD and Dynasty at the National Technical University using a standard waxed sample water displacement method.

Where a drill polygon of the same resource category or lower overlaps a polygon around mine workings, the mine workings polygon takes precedence. If the drill polygon is a higher resource category than the mine workings polygon the drill polygon takes precedence.

Veins were considered for resource modelling with a minimum true vein width of 0.4 m and a conservative minimum grade of 3.0 g/t Au (see calculations in Table 15). No upper cut-off grades were applied and silver grades were not considered when delimiting a resource block.

All resource modelling was carried out by qualified local geologists supervised by either a geologist or mining engineer who are members of the AusIMM.

Resource Long Sections for the larger veins and the currently mined veins are provided: Abundancia (Figure 46), Curipamba (Figure 47), Matalanga (Figure 48), Nudo (Figure 49), Soroche (Figure 50) and Tamayo (Figure 51) veins.

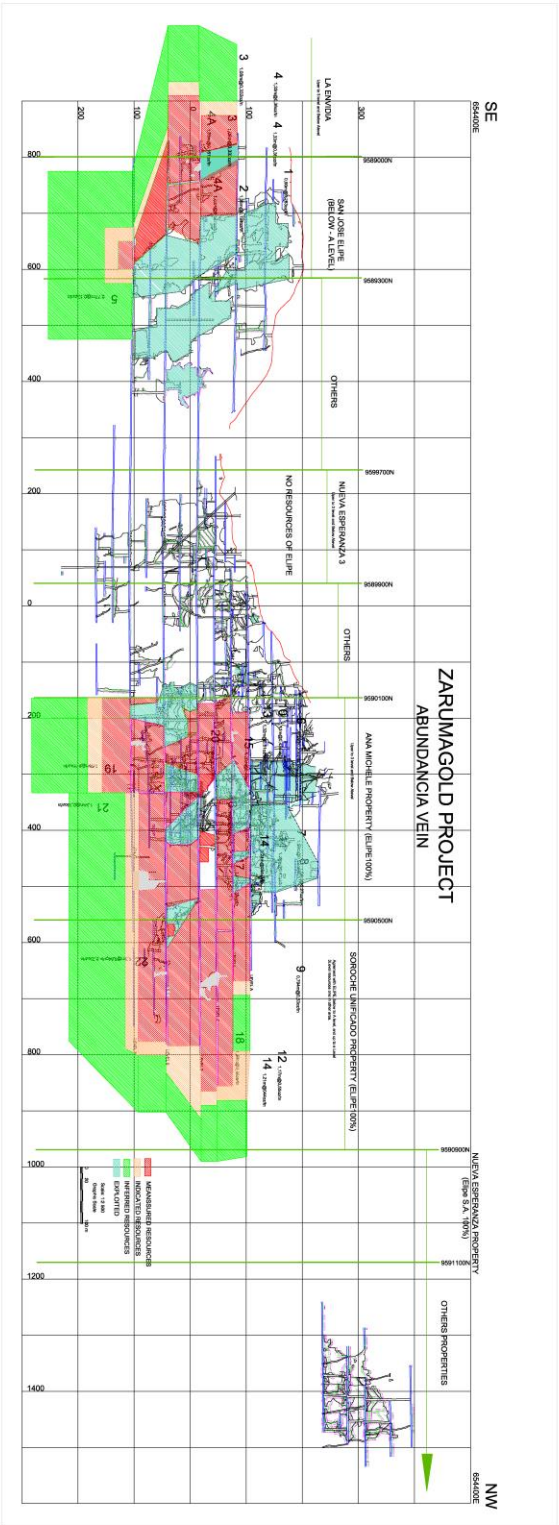


Figure 46: Abundancia Resource Long Section.

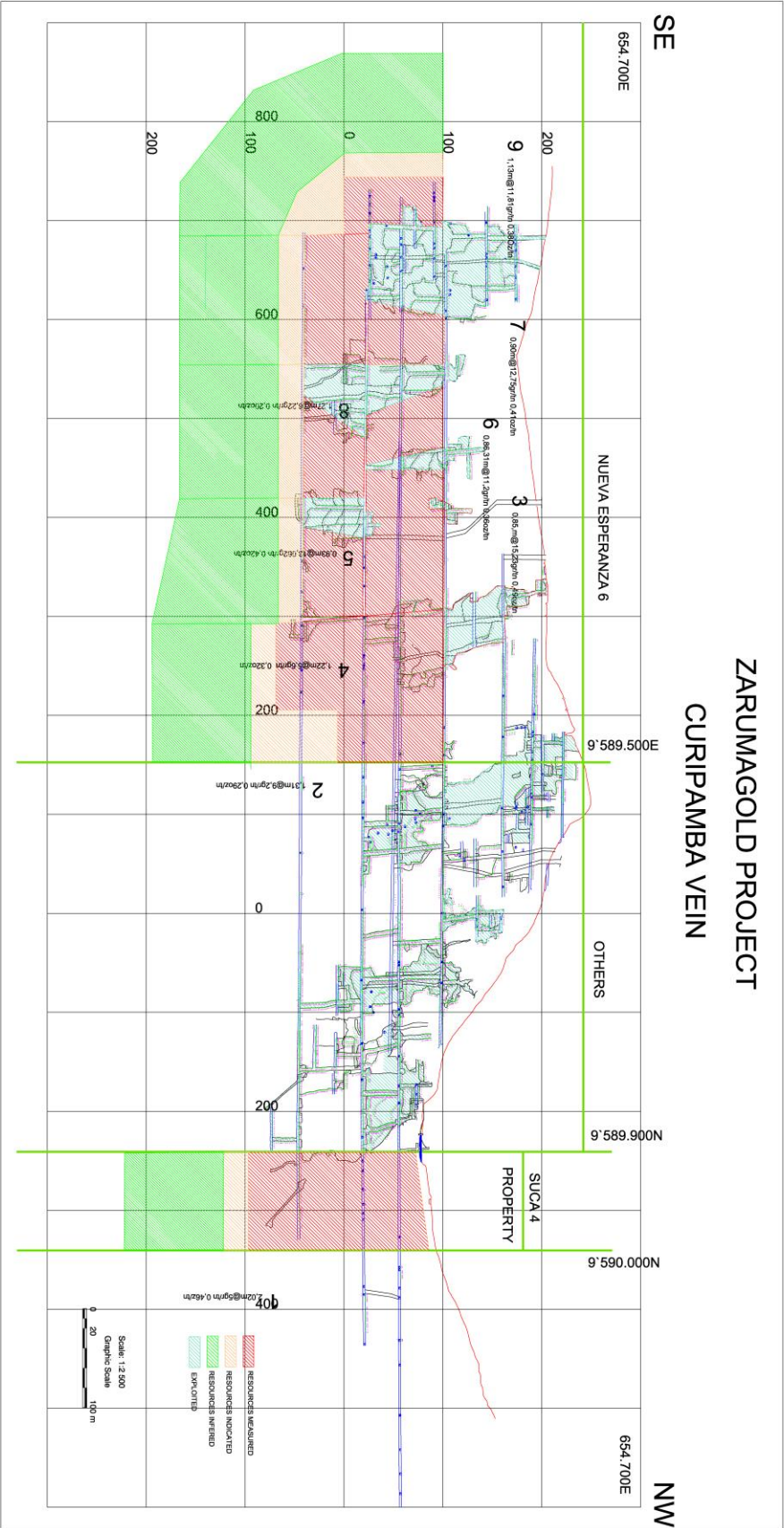


Figure 47: Curipamba Resource Long Section.

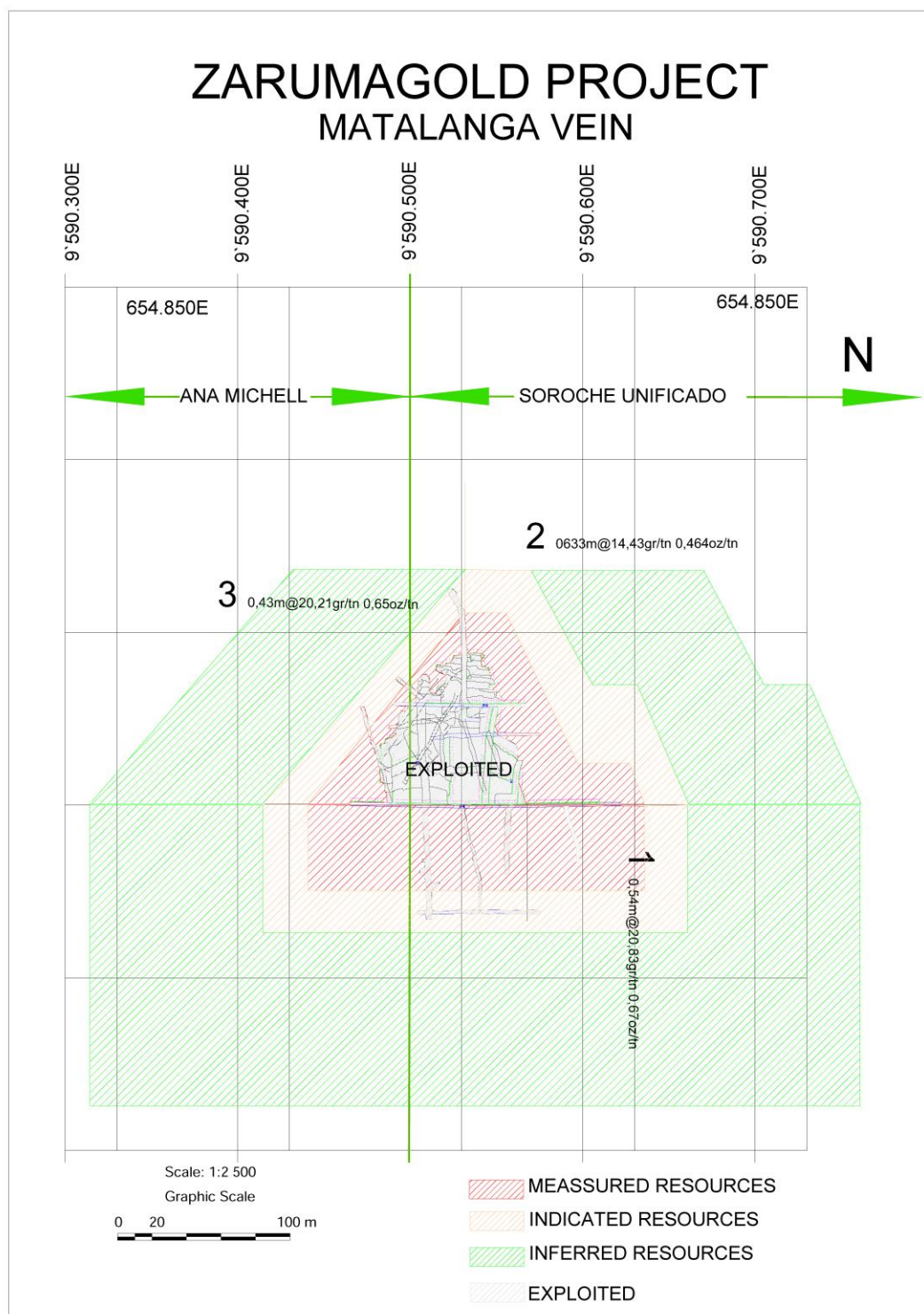


Figure 48: Matalanga Resource Long Section.

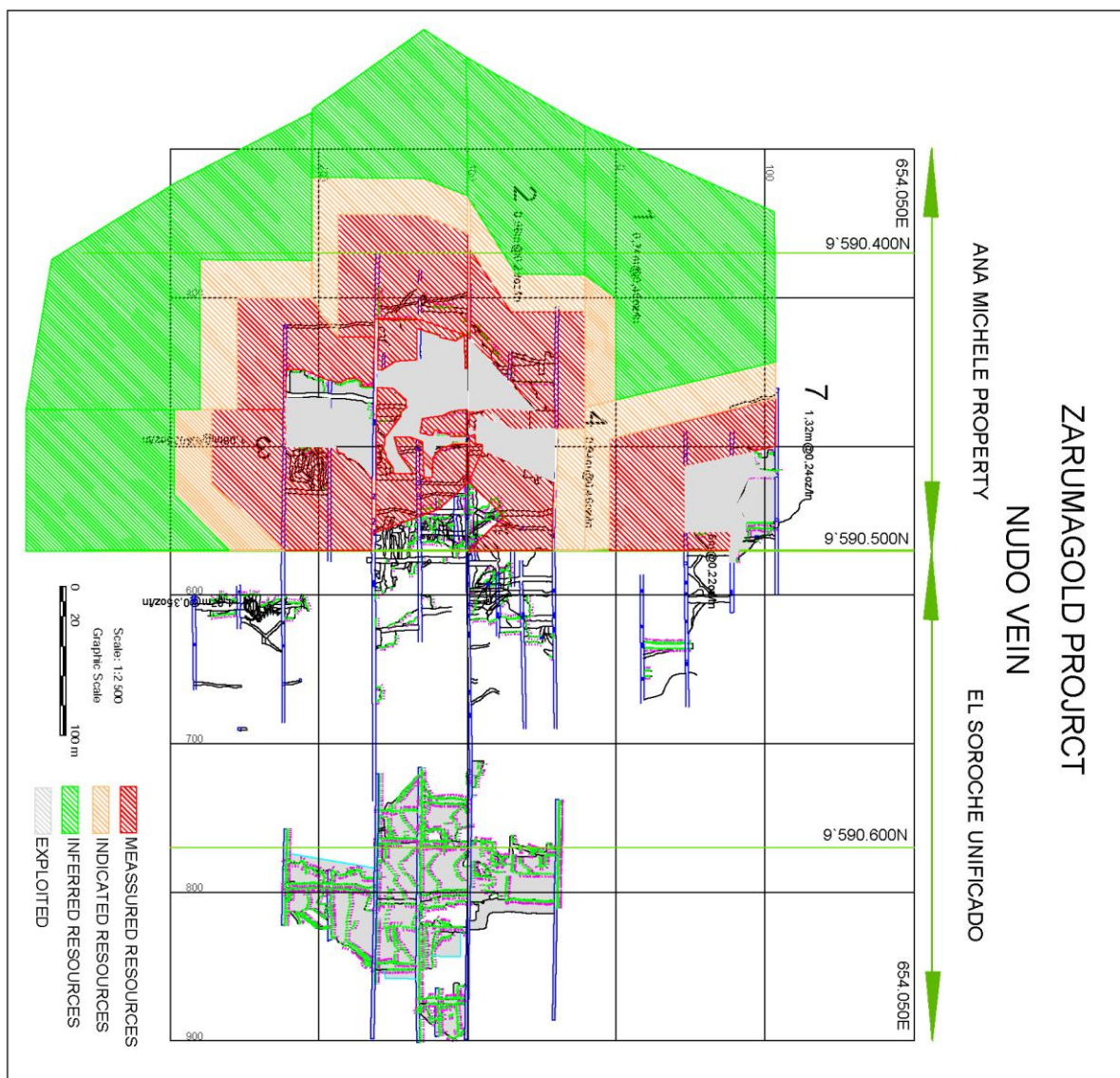


Figure 49: Nudo Resource Long Section.



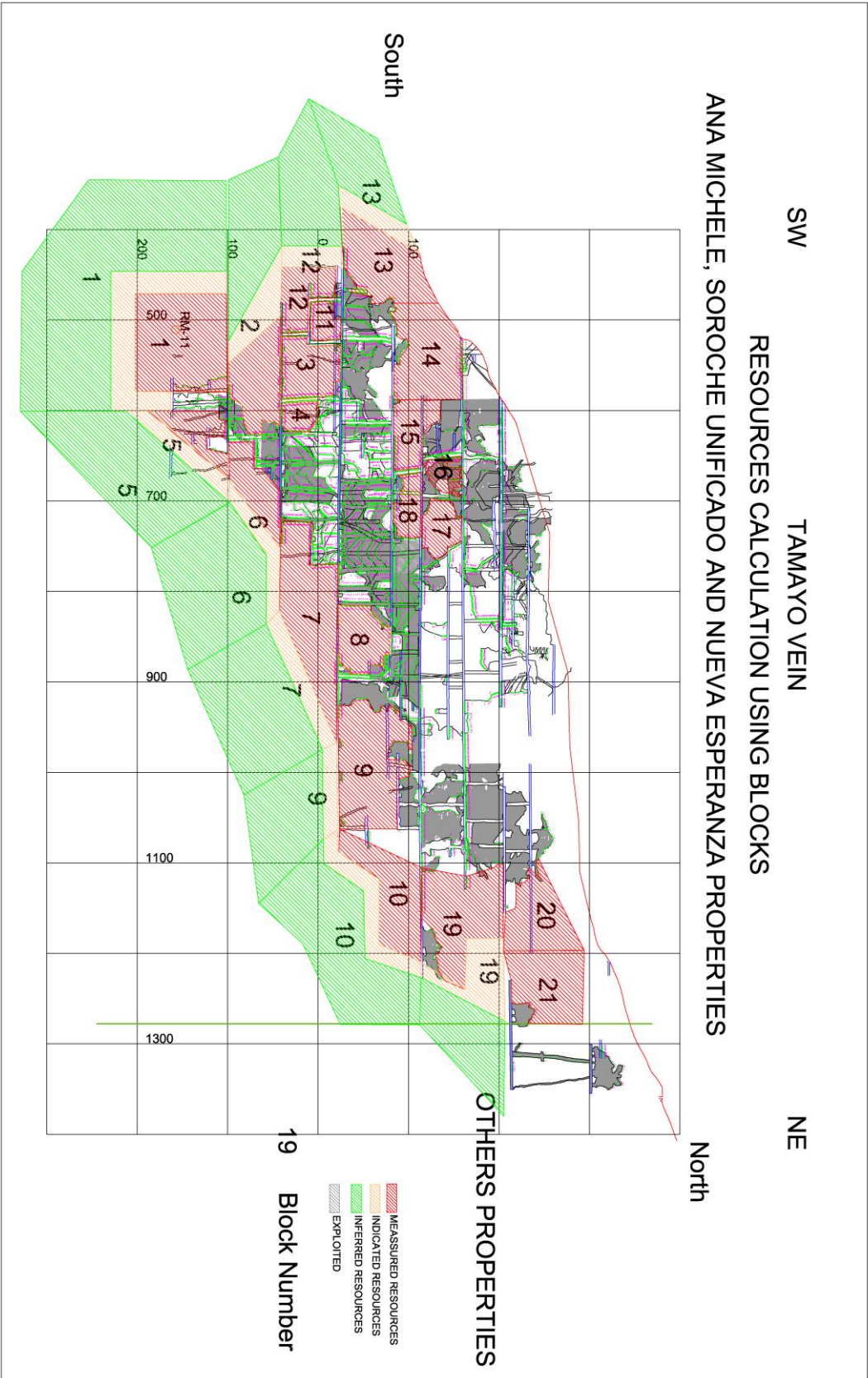


Figure 51: Tamayo Resource Long Section.

14.3 Resource Statement

The current Mineral Resource statement that has been reported in this Report has been used as the basis for the conceptual economic assessment undertaken in Section 22. The updated Mineral Resource estimate, applying a cut-off grade of 2.06, as at the date of this Report is the following:

Vein	Measured			Indicated			Inferred		
	Million Tonnes	g/t Au	Au (Ozs)	Million Tonnes	g/t Au	Au (Ozs)	Million Tonnes	g/t Au	Au (Ozs)
Ayapamba									
SUCRE VEIN	0.07	12.64	30,194	0.06	11.63	23,033	0.24	12.39	96,281
Barbasco									
BARBASCO VEIN	0.05	13.80	20,498	0.01	13.06	4,199	0.03	13.06	14,276
CAD VEIN	0.02	21.24	12,702	0.02	19.62	13,562	0.11	19.25	65,913
CLARA	0.03	5.01	5,246	0.03	4.91	5,146	0.04	4.10	4,996
JORUPE VEIN	0.06	9.67	19,400	0.01	9.02	1,856	0.04	9.02	10,585
TRES REYES	0.03	9.76	8,284	0.02	10.47	7,911	0.11	8.79	32,358
Cabo de Hornos									
26 VEIN	0.02	10.57	7,510	0.03	10.57	8,768	0.19	10.57	63,583
ABUNDANCIA VEIN	0.34	16.32	177,873	0.16	10.48	53,573	0.47	9.71	147,444
AROSEMENA VEIN	0.00	19.28	1,550	0.00	19.28	1,984	0.02	19.28	14,071
CURIPAMBA	0.18	12.76	72,203	0.08	13.55	35,418	0.28	12.63	113,129
ELENA VEIN	0.03	12.04	11,342	0.03	12.04	11,613	0.22	12.04	84,619
FORTUNA VEIN	0.09	14.48	43,761	0.05	14.35	22,145	0.14	15.36	70,717
MATALANGA VEIN	0.00	0.00	0	0.02	18.78	14,154	0.13	20.08	83,314
NUDO WEST	0.07	12.32	27,529	0.06	12.06	24,621	0.27	12.36	105,863
NUDO VEIN	0.16	15.41	81,384	0.13	12.09	51,581	0.21	15.19	103,339
POLVORIN	0.05	3.50	5,344	0.07	3.50	7,598	0.20	3.50	22,503
PORTOVELO VEIN	0.07	13.23	30,881	0.05	11.04	17,144	0.26	10.55	88,597
QUEBRADA	0.05	27.37	39,598	0.01	27.37	13,111	0.04	27.37	35,111
SOROCHE EAST	0.02	11.24	5,614	0.02	9.20	6,722	0.08	8.22	22,387
SOROCHE VEIN	0.12	11.84	46,631	0.09	12.48	35,470	0.25	11.10	90,574
TAMAYO EAST(SAN ESRNEST)									
TAMAYO VEIN	0.10	10.43	33,285	0.05	25.66	39,269	0.28	17.38	154,335
Vizcaya									
VIZCAYA VEIN	0.03	8.40	8,372	0.02	8.40	5,806	0.09	8.40	23,307
TOTALS	1.59	13.48	689,201	1.03	12.18	404,684	3.70	12.2	1,447,302

Location	Measured			Indicated			Inferred		
	Tonnes (million)	Grade Au (g/t)	Contained Au (Ozs)	Tonnes (million)	Grade Au (g/t)	Contained Au (Ozs)	Tonnes (million)	Grade Au (g/t)	Contained Au (Ozs)
Cabo de Hornos	1.30	13.99	585,000	0.86	12.46	343,000	3.0	12.3	1,201,000
Barbasco	0.19	11.05	66,000	0.09	10.81	33,000	0.3	12.13	128,000
Ayapamba	0.07	12.64	30,000	0.06	11.63	23,000	0.2	12.4	96,000
Vizcaya	0.03	8.40	8,000	0.02	8.40	6,000	0.1	8.4	23,000
Totals	1.59	13.48	689,000	1.03	12.18	405,000	3.7	12.2	1,448,000

Mineral Resource Category	Tonnes (Millions)	Au (g/t Au)	Content Au (1000 oz)
Measured	1.59	13.48	689
Indicated	1.03	12.18	405
Total Measured and Indicated	2.62	12.83	1,084
Inferred	3.7	12.2	1,448

Note: Readers are cautioned that Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

Table 16: Current Mineral Resource Estimate.

The mine's production for calendar year 2013 is shown in Figure 52. In total some 160 kt were produced grading 5.86 g/t Au. This reported production includes development material that can be processed profitably, ignoring mining costs, which would not be included in the Mineral Resource inventory. This production also largely came from outside the known resource areas as part of a further exploration in new veins, development and trial mining program.

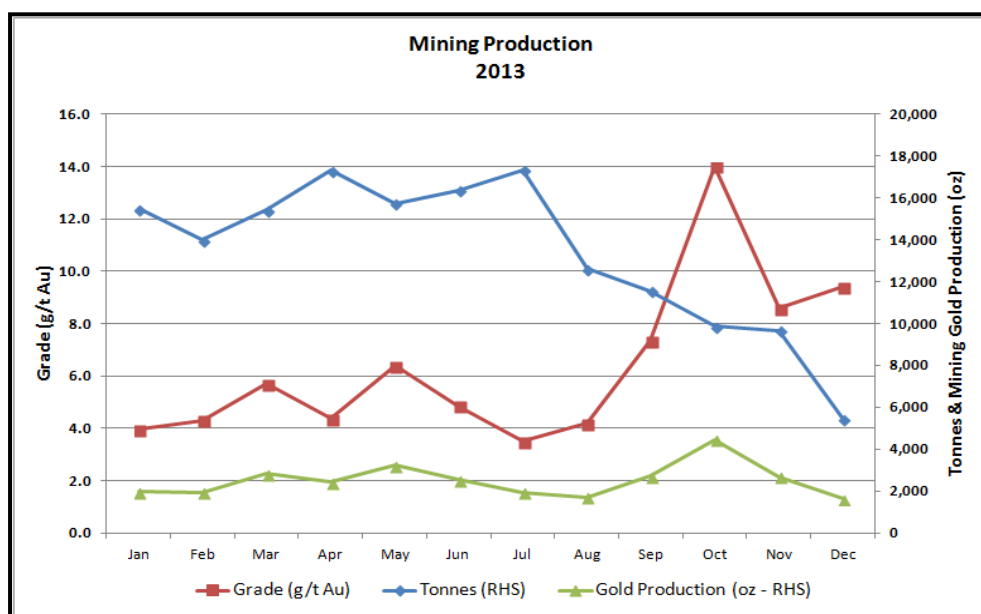


Figure 52: Mining Production for 2013.

In considering the Mineral Resource grade for use in the PEA production schedule, to arrive at a 'mined' grade and tonnage it has been assumed that the mining recovery will be 95% (i.e. the amount of in situ material that is planned to be exploited but is not mined due to practical recovery constraints), and then the grade has a 25% dilution factor applied. This factor provides for loss in grade due to: grade reduction from secondary and tertiary (in-stope development) that reports to the processing plant, unplanned increasing of mining widths (in stope and development) and barren areas that will be encountered (both along strike and down dip) and added to stoping material. There are also other engineering practicalities (including geotechnical, geological and design aspects) that are normally applied in the practical consideration of converting the Mineral Resource to a Mineral Reserve.

The Project has three major metal components: gold, silver and copper. However, in the conceptual economic assessment (Section 22), only gold and silver values are used. Here, there is conceptually a reasonably consistent ratio of in situ silver to gold grade, generally about 2:1. Further, the gold versus silver price (\$/oz) currently has a ratio of approximately 62:1 (1,300/21). Therefore in the assessment of both metals (particularly in the conceptual economic assessment), silver is accounted for on an equivalent grade basis as having a value 1/31 of that of gold (i.e. 2/62).

The authors believe that the resource for Zaruma has been calculated utilizing acceptable estimation methodology and the authors are also of the opinion that the classification of measured, indicated and inferred resources in this report meet the definitions as stated by NI 43-101 and defined by the CIM Definition Standards For Mineral Resources and Mineral Reserves adopted by the CIM Council on November 27, 2010.

15.0 Mineral Reserve Estimates

15.1 General

No Mineral Reserve estimate has been prepared for the Project to date. This Preliminary Economic Assessment does not support an estimate of Mineral Reserves, since a prefeasibility or feasibility study is required for reporting of Mineral Reserve estimates. This report is based on potentially mineable material (“mineable tonnes”).

Mineable tonnages were derived from the resource model described in the previous section. Measured, indicated and inferred resources were used to establish mineable tonnes. Inferred mineral resources are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorised as mineral reserves, and there is no certainty that the preliminary economic assessment will be realised.

Notwithstanding that there is no Mineral Reserve being reported, the underground mine is in operation and a full production route has been established, from development and stoping activities underground through to transportation of the mined mineralised material to the surface processing facility (some 7.5 km from the underground portal), where final doré is produced for onward shipment to overseas gold refineries. As part of these operations, the Company has had to meet the various statutory environmental permitting and approval requirements. These are dealt with in Section 20.

16.0 Mining Methods

The Zaruma Gold Project uses several mining methods for material extraction, and the particular method applied varies depending on the specifics of the vein being exploited. Generally, the underground conditions within the mine are considered to be highly competent, with there being little overt need for the application of full passive and/or active ground support systems to be installed. Nonetheless, the mine currently employs two principal mining methods to its veins: shrinkage (Figure 53) and cut-and-fill (Figure 54) stoping methods. Whilst these methods provide an extraction system that is reasonably safe and effective (considering the tonnage being exploited and maintaining grade with minimum dilution), in particular shrink stoping has become outdated and is essentially obsolete in most modern mines.

This has largely been driven by the move away from working under unsupported openings. The method is also a low production system that ties up significant quantities material in the stope while the stope face is being advanced.

Eventually, it would be expected that the underground operations move toward more mechanised cut-and-fill systems. This will improve productivity, consistency of tonnage delivery and potentially improve mining recovery and grade dilution factors.

The mineralised material from underground is trucked by 30 t CAT trucks to just outside the mine's portal, where the material is dumped and then loaded into 20 t dump trucks for onward road transportation to the processing plant facility.

The multi-veins available to the Project are accessed via two portals: the main access and production portal (Cabe de Hornos) and the Barbasco portal. Barbasco will be used to access the following veins: Silvia, Barbasco East (Daniela) and Mercy. Silvia and Barbasco East are considered to have good mineralisation of gold, silver and copper. Mercy appears to be a lower grade option.

The plan is to use the Barbasco portal to intersect the Jorupe, Tres Reyes, Curipamba and Clara veins. Clara is a vein that was discovered by drill holes in 2009. It is located 320 m east to the Barbasco portal. Curipamba vein is the south extension to Barbasco vein (Silvia).

Access to the main underground workings (via Cabe de Hornos portal) is by means of a 4.5 m x 5.0 m decline ramp which has been developed at 1/7 gradient. Generally, sub-level intervals are 32 m apart vertically, with internal development drives dimensioned 3 m x 3 m.

The underground operation is still relatively shallow – stoping operations are all above the 6 level (~300 m depth), the decline is developed to 9 level (~350 m depth), whilst the Mineral Resource is based on a maximum depth of 13 level (~650 m depth).

All mining operations are undertaken by the Company itself and hence there are no contractors on site.

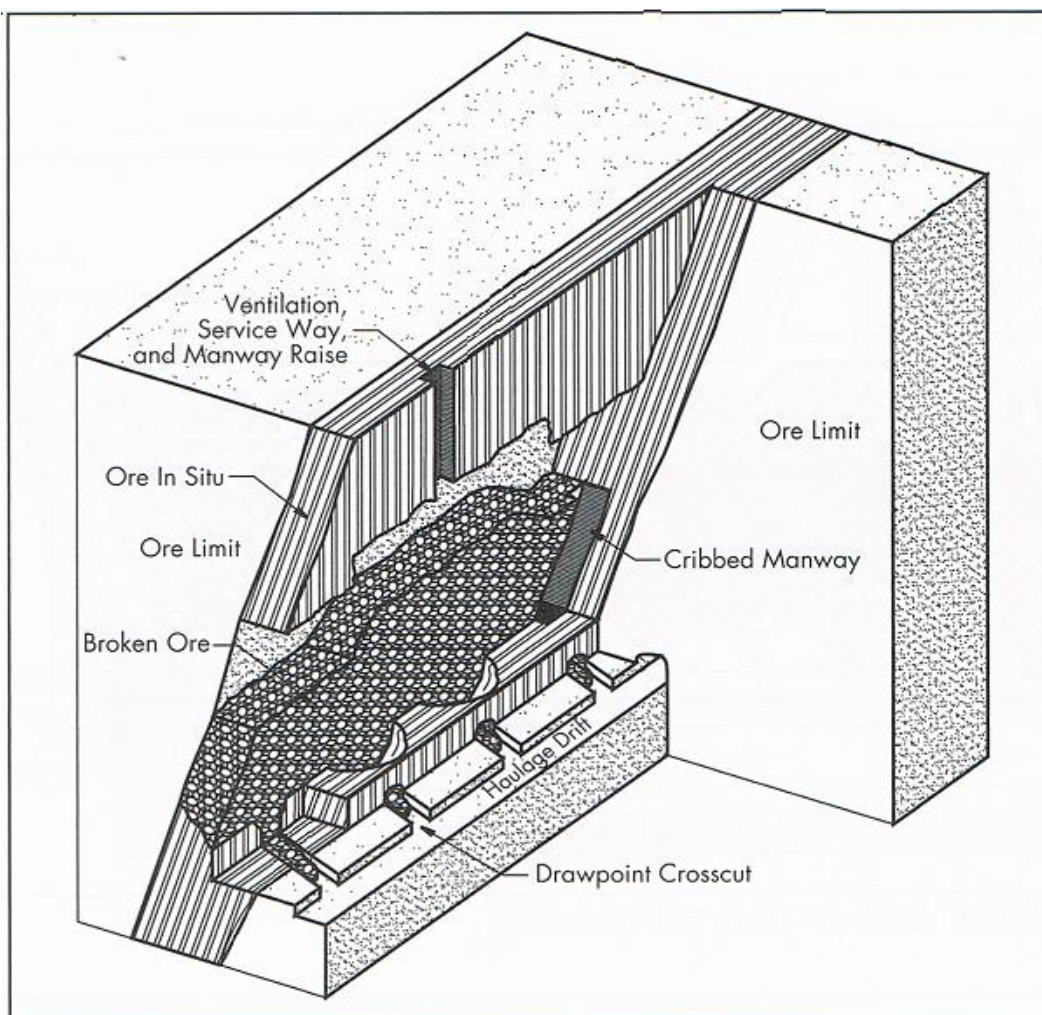


Figure 53: Schematic of Shrinkage Stopping Method.

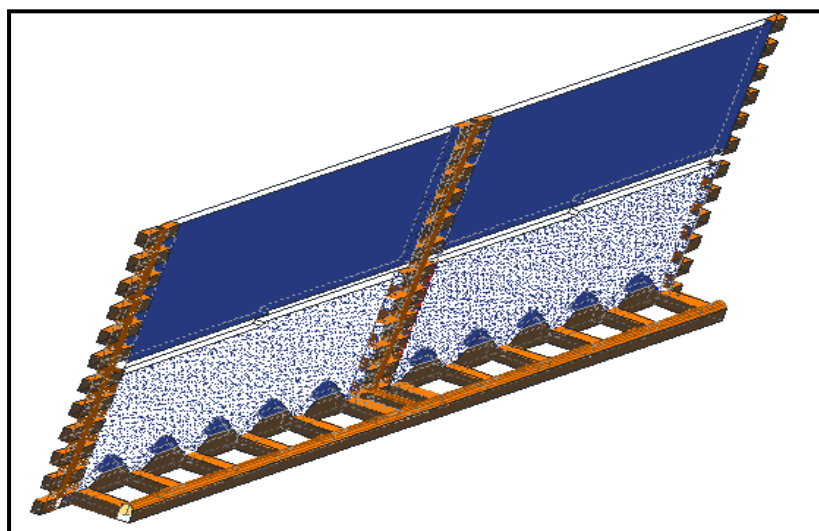


Figure 54: Schematic of Cut-and-Fill Stopping Method.

The production statistics for the 2013 year from underground development are shown in Figure 55. In Figure 55, the high degree of variability in both tonnage and grade is clearly evident.

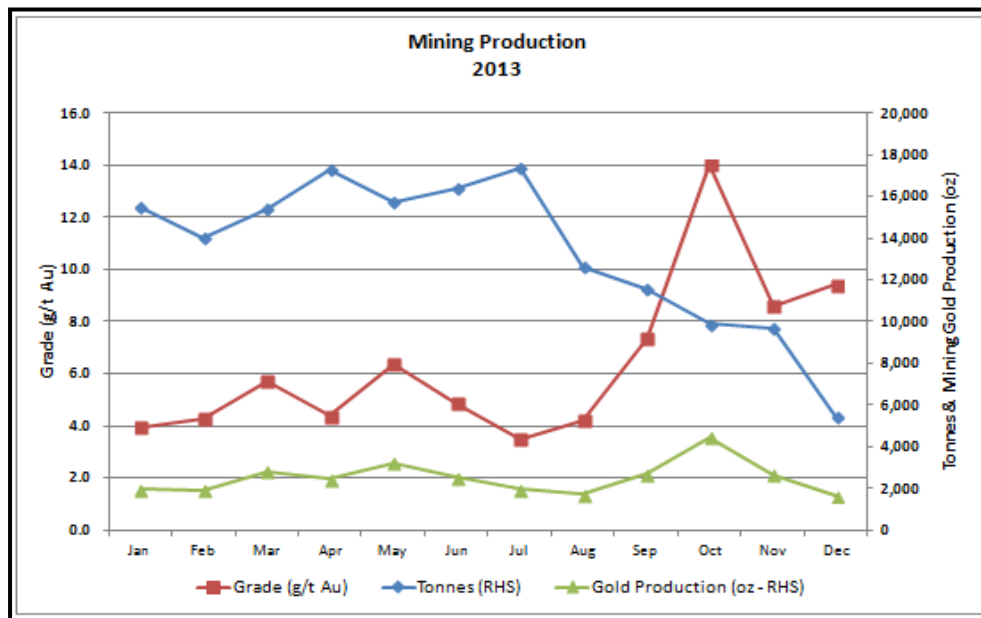


Figure 55: Mining Production for 2013.

This mining method is suitable for steeply dipping veins, with the strong and non-oxidising broken material easily feeding into the lower drawpoints by gravity. Also, given the highly competent nature of the general underground rockmass, there is little need for intensive ground support systems, thus making it a relatively inexpensive system to use. In any case, the stope backs are able to be kept close to the footwall muck pile, giving easy and ready access to the surrounding rock should additional scaling or support be required.

One of the main requirements for this mining method is to closely monitor the broken material for swelling (volume increases by between 30% and 50%), which governs the gap between the broken footwall material and re-accessing the hanging for further drilling and blasting. This issue can, to an extent, be managed by controlling the rock fragmentation through blasting practises. The mine's application of the method is, however, labour intensive, as jackleg mining is used for the drilling operation. Also, the amount that can be drawn from the drawpoints is limited by the need to be able to access the stope backs for further drilling, so there tends to be a large inventory of broken material in the stope.

This inventory of course increases until the stage the stope drilling (development) is completed, when it can be drawn empty.

The underground operations are currently developing 3 shrinkage type stopes (Soroche West, Soroche East and Matalanga 412). Soroche will actually be developed as a dual shrinkage/CAF system, with the lower section developed by shrink stoping and then the upper portion by long downhole stoping from the level above. In this mining option, the shrink stope will have a height of 25 m, with the remaining 49 m of stope height recovered by long downhole stoping. The latter method is considered to cost twice as much as the shrinkage

option, although there are likely to be recovered grade and potentially ground control benefits in it.

The underground mine is operating within a very competent volcanic 'tuff' rock type and the mineralisation is mainly associated with strong quartz stringers. There is little evidence underground of any substantial rock weaknesses, and geotechnically the mining environment is considered to be good (competent) rock. The shrinkage stopes are developed with between 8 m and 10 m sub-intervals. To start the preparation of the operation, first a centre slot is established which also aides with future ventilation requirements. Rises are established every 45 m or 60 m along strike, depending on the stoping characteristics and need for personnel movement and ventilation. Auxiliary ventilation fans are also used to force air into the working chamber. The stope itself is developed in 2.0 m height by 3.5 m in width. Currently, the mining criteria being applied are considered appropriate for the largely labour intensive methods being employed, and the skills and experience available to the Project. However, once a feasibility study process is embarked upon, the selection of mining method(s) will be a critical design and engineering aspect. It will also have ramifications for the economic performance of the Project.

The operating cycle within the stopes involves drilling, blasting, and periodic partial extraction of mineral. If horizontal holes are used, then the muck pile needs to be pulled down before each blast. Otherwise, material is left in the stope to provide a floor to work off for the next cut.

In the case of CAF stoping (Figure 54), the stope is developed with successive horizontal slices, which are then subsequently filled before the next slice is taken. The general stoping cycle entails drilling, blasting, loading and a filling cycle.

This mining method is becoming one of the most popular in small mines as it provides good support to the mining chamber, and provides the ability to maximise recovery and potentially minimise dilution. Due to mine grade tenors becoming ever lower and lower, these become very significant operational attributes. The stope fill material is usually sand and/or gravel, which is sometimes mixed with cement to provide higher strength load capacity. However, even broken waste material may also be used when appropriate. The method can also be done either in a labour intensive manner, or highly mechanised. Dynasty is currently adopting a mixed approach, whereby stope work is labour intensive but stope cleaning and hauling is partially mechanised (using scrapers and LHDs).

As mining operations develop further (at depth and along strike) at Zaruma, it is expected that the Company will tend more to the CAF type systems (of which there are numerous different options depending on the mining factors to be achieved based on criteria such as vein width, height of stope and degree of mechanisation being sought). These systems will therefore provide greater flexibility for higher and lower grade areas of exploitation. This is very applicable to the often erratic form of mineralisation associated with epithermal type deposits such as Zaruma's. It is noteworthy that Zaruma does use Alimak mining (it has two units) for raise development, and this equipment could give the operation the opportunity to also use the equipment for stoping by using long holes drilled horizontally from the Alimak equipment.

Rock breaking is carried out using ammonium nitrate fuel explosive (ANFO), and where excessive water is encountered, gel explosives are also used. Blasting patterns are carefully controlled to ensure that rock fragmentation is adequate (small enough) to enable the efficient movement of the broken material from in the stope to the ROM stockpile near the entrance to mine's portal.

Ventilation is designed so that fresh air enters the lower levels of the mine, and then moves up through the various rises and the out of the surface fan, which is a pull system. A schematic of the ventilation system is shown in Figure 56. This system has adequate capacity for the envisaged scale up of the mine, and given the likely amount of increased development over time, the overall system resistance could potentially reduce, providing more effective ventilation (volumetric flow rate) at a lower fan operating pressure.

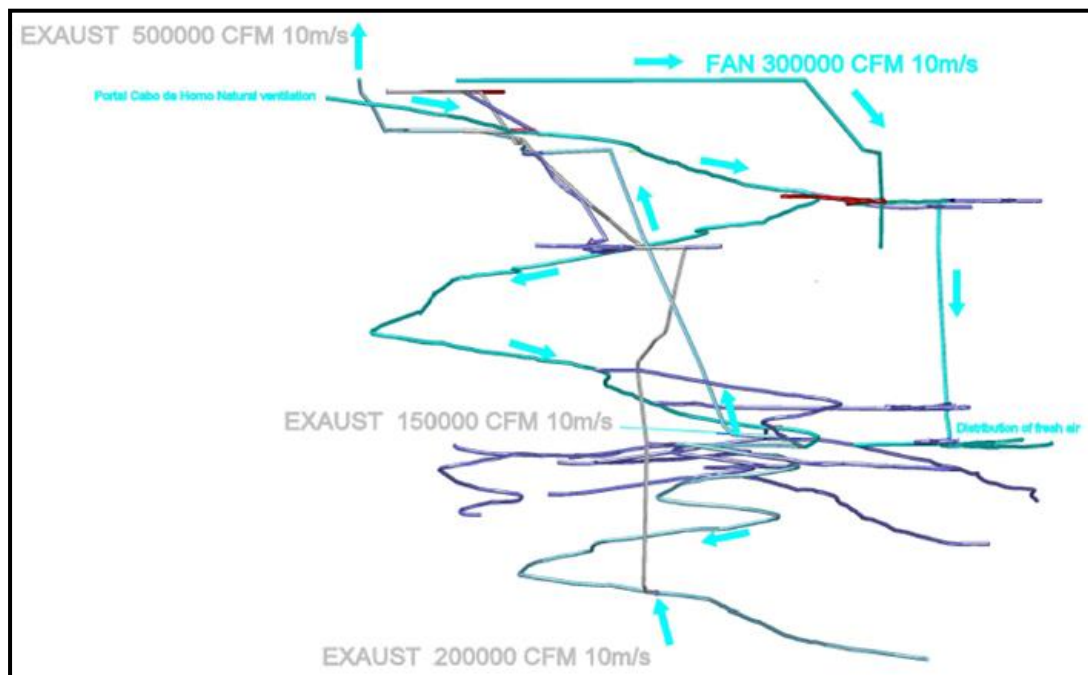


Figure 56: Schematic Presentation of Ventilation System.

For comparative purposes, based on the mine's current Mineral Resource, at a 200,000 tpa production rate, the Project will have a mine life of some 12 years, whereas at 350,000 tpa production rate, the life will be about 7 years. These tonnage data do not, however, assume a dilution factor, which is presently being modelled at 25% (at zero grade). This is conservative, because in reality, this is unlikely to be the situation and there will often be some grade associated this material. However, in general it is likely to be below the cut-off grade. Nonetheless, the assumption in the PEA's base case over the 15 year life, using the factors adopted, provides an in situ tonnage averaging 235,000 tpa and after dilution and mined material loss, averages 279,000 tpa reporting to the processing facility.

This level of dilution does significantly increase the material to be mined and processed. Application in section 1 of the modifying factor of mining recovery has been assumed at 95% (i.e. 5% mineralised material not recovered). In a vein system such as Zaruma's, the far greater risk is considered to be dilution, because of the generally narrow characteristic of the

veins. Hence, it will be much easier to recover more material (dilution) as opposed to leaving in situ mineralised material (mining recovery) behind.

A model showing the major veins underground together with the internal ramp infrastructure is shown in the 3-Dimensional (3D) model provided in Figure 57. Typical cross-sections of various veins are also shown in Figures 58 to 62. Note that these cross-sections are provided for illustrative purposes only, and may not accurately reflect current or planned development, stopping or actual Mineral Resource inventories.

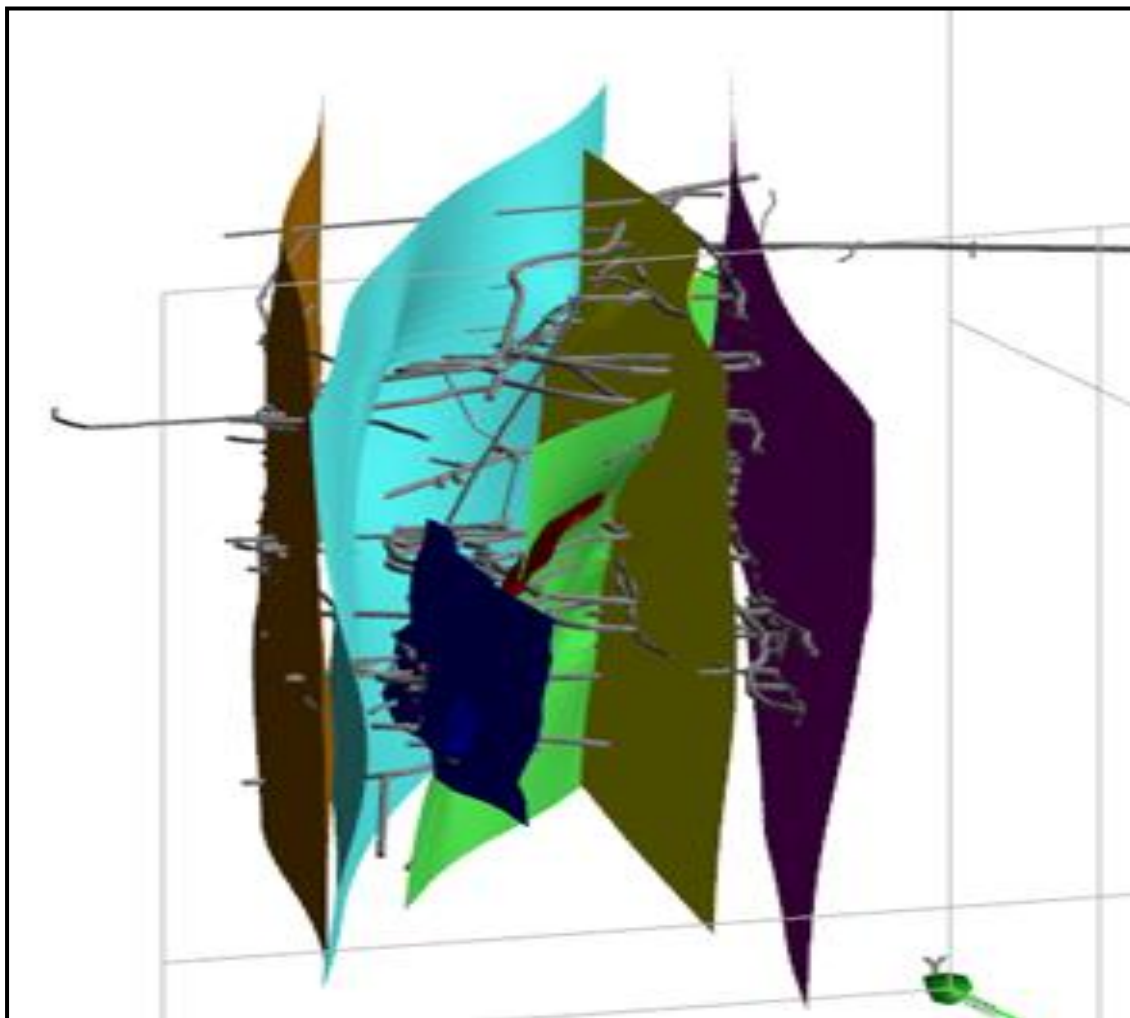


Figure 57: 3D View of Underground Veins and Ramp Infrastructure.

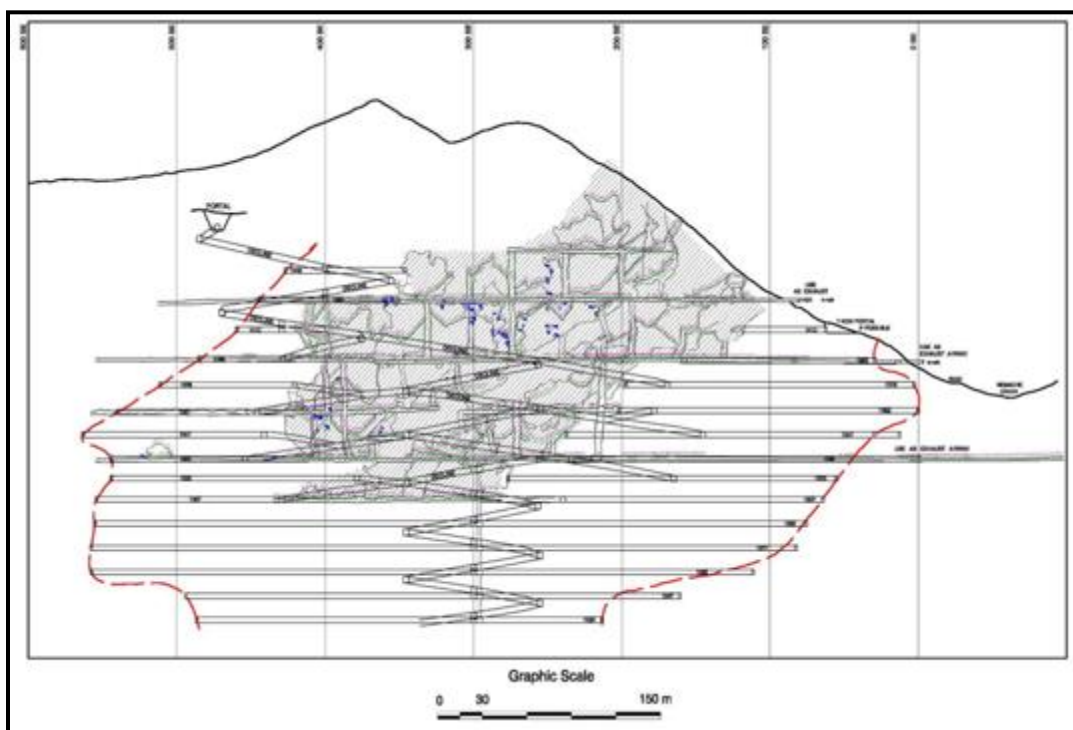


Figure 58: Typical Cross-Section of Secure Vein.

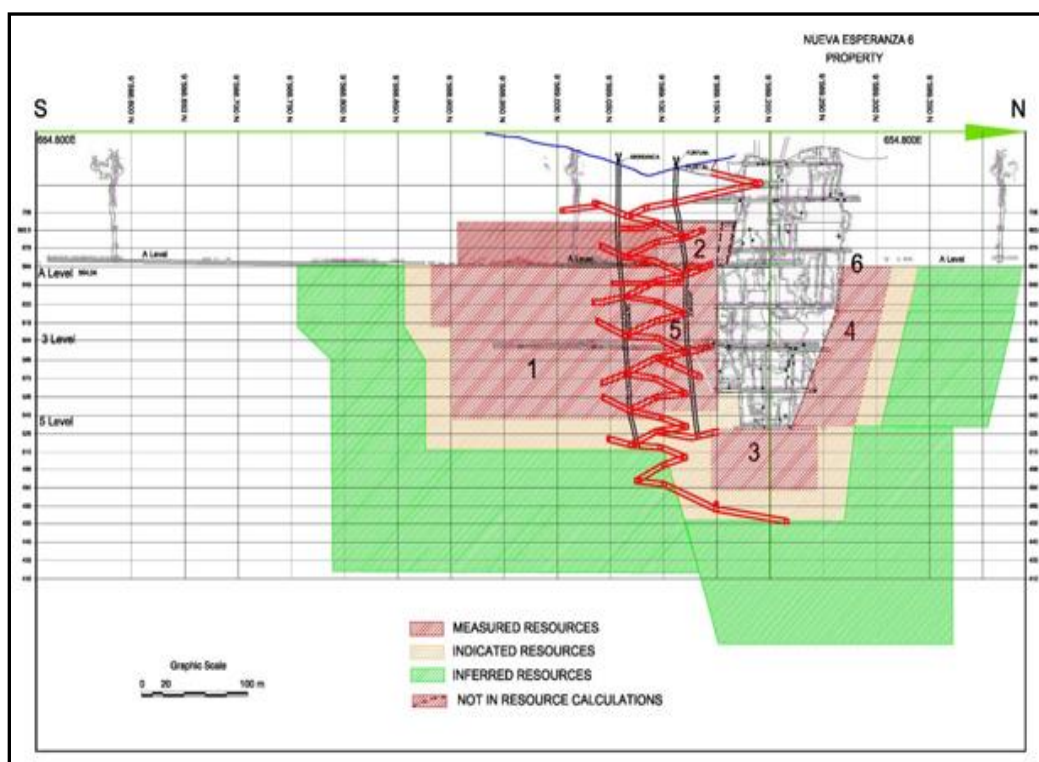


Figure 59: Typical Cross-Section of Fortuna Vein.

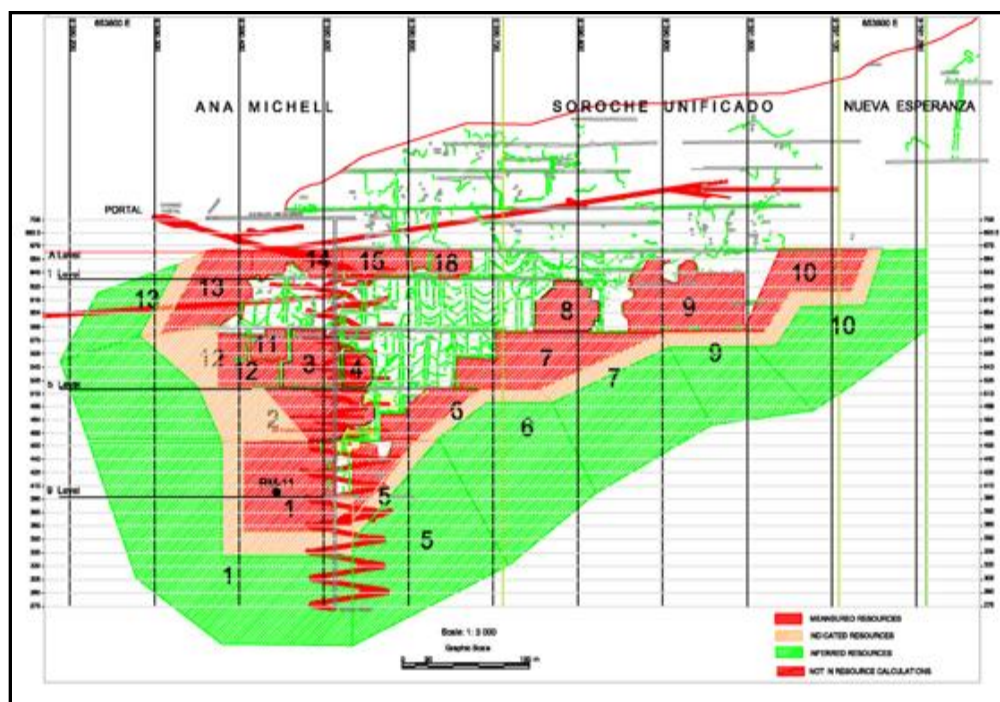


Figure 60: Typical Cross-Section of Tamayo Vein.

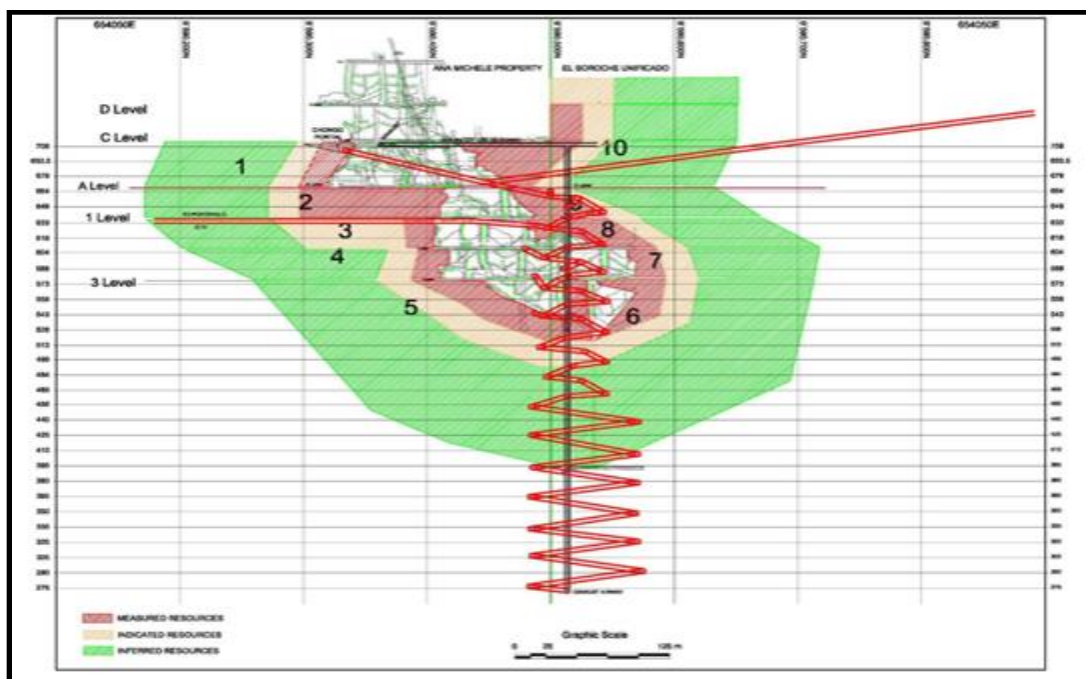


Figure 61: Typical Cross-Section of Nudo Vein.

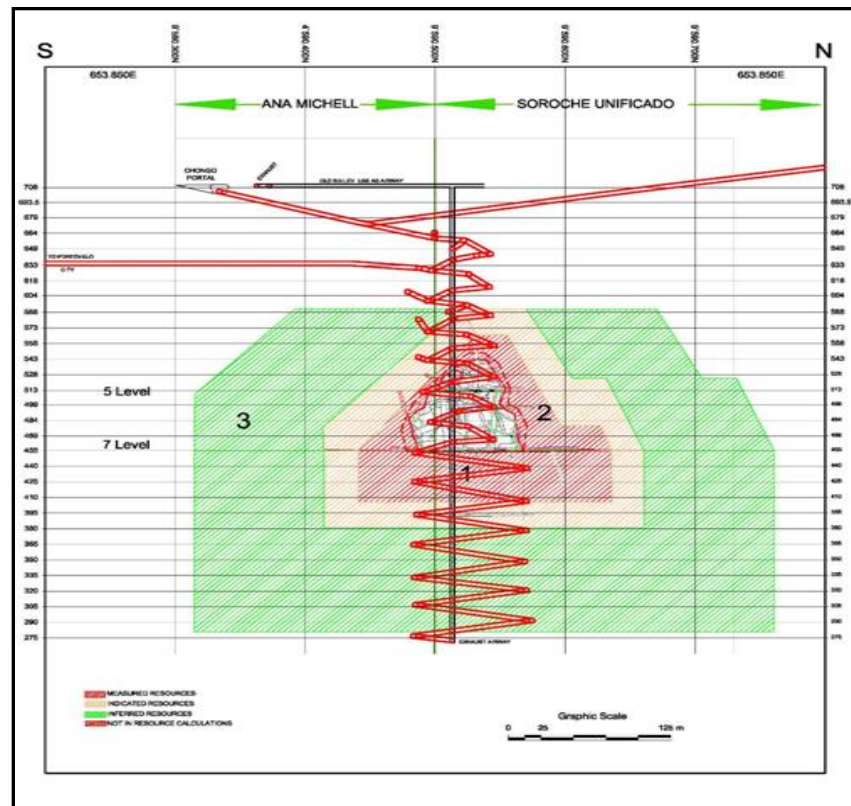


Figure 62: Typical Cross-Section of Matalanga Vein.

Mining underground at Zaruma is conventional with regard to mining methods, equipment utilisation and mining development infrastructure (ramps, raises, pillar establishment, dewatering, ventilation and safety systems and procedures). The 4.5 m x 5.0 m decline is developed tracklessly at a 1/7 decline angle, and all water made in the underground is pumped out of the mine from two major pump station levels, one being at 412 m depth and the other at 450 m depth. Both pumping stations use 110 kW Grindex submersibles for dewatering at a rate of about 800 gpm (50 L/s).

Compressed air is provided into the mine from surface compressors, which have a capacity of about 4,000 cubic feet per minute (cfm) (2,600 by diesel compressor and 1,400 cfm by electric compressor), delivered at a pressure of between 120 pounds per square inch (psi) and 135 psi (0.83 megapascal (MPa) – 0.93 MPa).

Underground development requirements for the mining methods being employed are not excessive given the scale of the operations. In 2013, the total amount of underground development was some 548 m producing about 4,804 t at a grade of about 17 g/t Au. This quantum of development grade material is a very important contribution to the mine's overall gold production output. Each stope developed will require, nominally, between 250 m and 300 m of development. At the current approximate dimensions of development, this equates to, at the lower end, about 6,000 t of material. This is almost half a month's feed requirements for the processing facility. In 2013, mining took place from inside the Mineral Resource boundaries during mainly the last quarter of the year, otherwise it was 'not in resource', meaning there was no depletion of the Mineral Resource by its extraction.

The mining operations operate on a 3 shift/day basis, with approximately 49 personnel working per shift. Shifts are nominally 8 hours each.

The mining operations are both labour intensive and mechanised, with much drilling of stope development carried out by jackhammer and cleaned by scrapers, whilst main decline (ramp) and drawpoint ends are developed by means of jumbo and LHD.

In terms of material movement, development and stoping material is loaded underground into 30 t haul trucks where it is hauled up the ramp (Figure 63) to surface. At the surface there are stockpiles for both waste and material to be processed. The material that is to be processed is loaded into 20 t dump trucks (Figure 64) where it is transported some 7.5 km (trip takes about 45 minutes) and offloaded at the processing facility ROM stockpile.



Figure 63: Portal for Underground Access.



Figure 64: Underground Material Loading into 30 t Dump Trucks.

The interface between the underground and the processing operations is at the coarse material stockpile at the mine's portal entrance.

The mine's power supply (for surface and underground uses) is provided from the regional electrical grid, which it takes off into its switchyard at 69,000 V / 11 kV and provides 4.8 MW to the operation.

The mine's current underground and surface plant and equipment is listed in Table 17. Some 40 personnel are involved in maintaining all the mining (surface and underground) plant and equipment. Personnel within the maintenance facility (Figure 65) are able to carry out most of the requisite maintenance (including equipment re-builds, motor rewinding and plant re-conditioning).

The systems and procedures for monitoring and controlling of servicing and spares (through preventative maintenance and critical maintenance scheduling) are well established by the operation and appear to be competently managed.

Only minor specialist work is out-sourced. For example, Atlas Copco rigs are provided to Atlas Copco itself, on an original equipment manufacturer (OEM) maintenance basis.



Figure 65: Entry to Underground Maintenance Facility.

Equipment Type	Make	Model	Series	Capacity*
SCOOPS	HST1-A	41779		1,25 cubic yard
	WAGNER	HST1-A	35978	1,25 cubic yard
	JCI	125-M	122695	1,5 cubic yard
	JCI	200M	56182	2 cubic yard
	JCI	250M	58888	2,5 cubic yard
	JS	220		2,2 cubic yard
	EJC	100		2,2 cubic yard
	JS	220	1540	2,2 cubic yard
	JS	220	613	2,2 cubic yard
	JS	220	612	2,2 cubic yard
	WAGNER	ST7.5Z	DZ17P0227	7,5 cubic yard
	WAGNER	ST7.5Z	DZ17P0249	7,5 cubic yard
	WAGNER	ST7.5Z	DZ17P0211	7,5 cubic yard
	JCI	250M		2,5 cubic yard
COMPRESSORS	ATLAS COPCO	GA-110	AIF 114848	132 psi
	ATLAS COPCO	GA-110	AIF 114847	132 psi
	INGERSOLL RAND	SSR-EP150	F38128U01261	128 psi
	INGERSOLL RAND	900		900 cfm
	INGERSOLL RAND	HP1600WCU	324467UILL076	1600 cfm
	INGERSOLL RAND	HP1600WCU	324691UILL076	1600 cfm
	INGERSOLL RAND	MM160	19969DYFGAG1	26 m ³ x min
	JOY	TA60RZ	BF6781	6000 cfm
TELEHANDLER	INGERSOLL RAND	VR843C	186138	Maximum height 7.5 m
	INGERSOLL RAND	VR843C	186136	Maximum height 7.5 m
	INGERSOLL RAND	VR843C	186874	Maximum height 7.5 m
	LIFTKING	LK 630	LT 0884	Maximum height 7.5 m
FRONT END LOADER	CATERPILLAR	950G	3JW00429	4.5 yd ³
	CATERPILLAR	950G	3JW01754	4.5 yd ³
	SEM	650B	3121	3 - 4.5 m ³
TRACTORS	KUBOTA	L4400D	87168	45.3 hp
	JOHN DEERE	5065E	PY3029T161258	47.8 kW
	KUBOTA	L4400D	86848	45.3 hp
	KUBOTA	L4400D	88557	45.3 hp
	KUBOTA	L4400D	87878	45.3 hp
	KUBOTA	L4400D	88228	45.3 hp
	JOHN DEERE	5065E	1PY5065ECBJ006183	47.8 kW
	JOHN DEERE	5065E	1PY5065EKBJ006186	47.8 kW
	JOHN DEERE	5065E	1PY5065ETBJ006184	47.8 kW
	JOHN DEERE	5065E	1PY5065ECCJ008078	47.8 kW
	JOHN DEERE	5065E	1PY5065EECJ008077	47.8 kW
	JOHN DEERE	5065E	1PY5065ECCJ008498	47.8 kW
JUMBOS	MTI		2793	
	TAMROCK (mini jumbo)	H103	B-20-506	
	ATLAS COPCO	BOOMER 282	AVO99C058	
	ATLAS COPCO	ROCKET BOOMER 282	AVO04C017	
	ATLAS COPCO	BOOMER 282	AVO94C015	
	MTI VEIN RUNNER		2824	
	ATLAS COPCO	BOOMER 282	AVO99C083	
BACKHOE EXCAVATOR	CATERPILLAR	420E	BLN12081	0.96 m ³
EXCAVATOR	CATERPILLAR	320C	PAB04608	1m ³
TRANSPORTER	TELEDYNE	SCT-6RD	3600-95-1	
SMALL EQUIPMENT	CATERPILLAR	252B	SCP03329	1500 lb
	BOB CAT	863	514442150	0.442 m ³
	BOB CAT	MT-50	520611486	1.5 t
CONCRETE MIXER	ATLAS COPCO		969	
LIFTING UNIT	CATERPILLAR	GP-15	ET31A60281	1 t

* cfm = cubic feet/minute; psi = pounds per square inch

Table 17: Mining Equipment & Plant List.

The underground mine's explosive magazine (Figure 66) is managed under very strict conditions, but not unusual when compared to most mining industry jurisdictions. The magazine is located approximately 3 km north-east of the underground mine portal position.

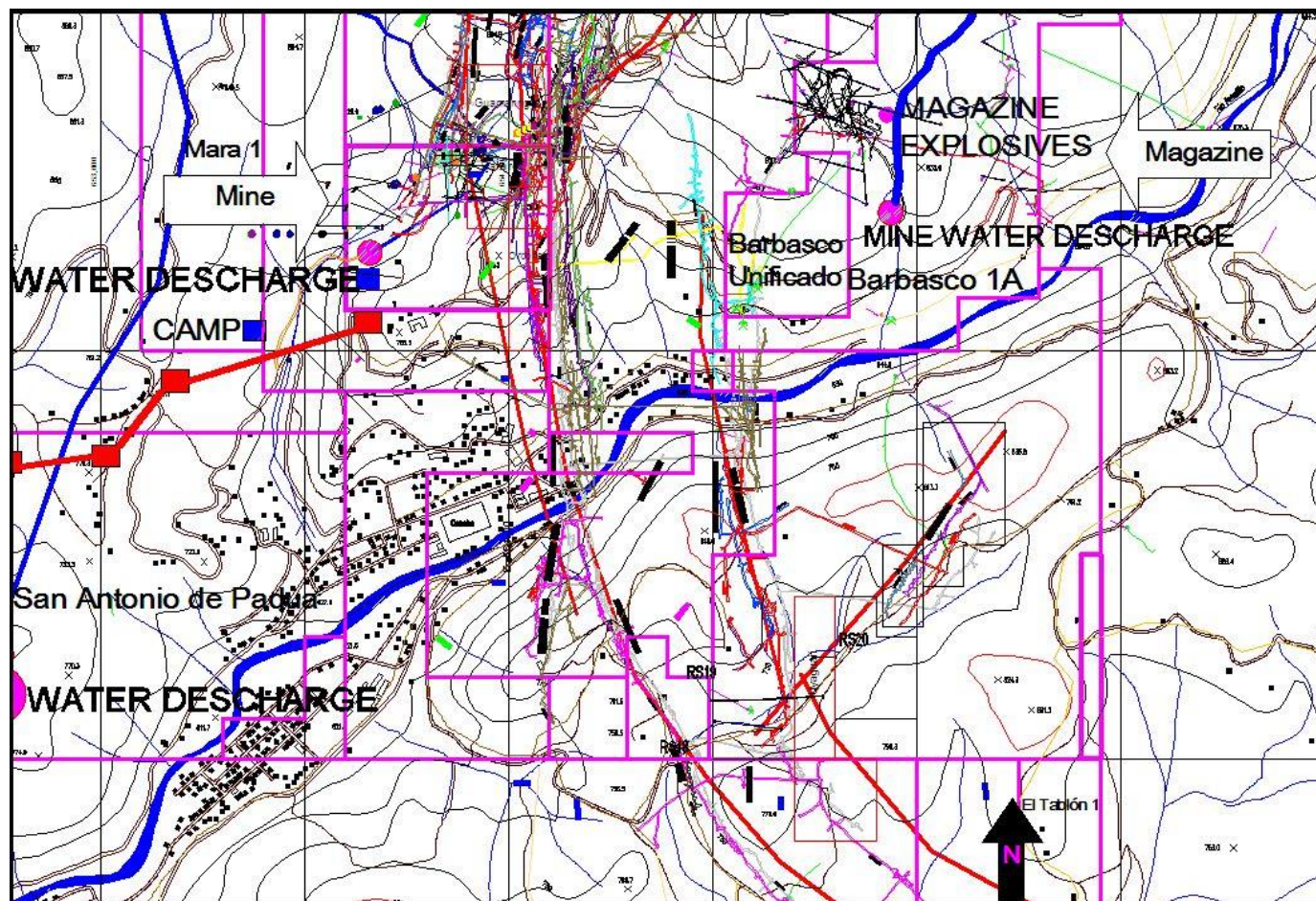


Figure 66: Explosive Magazine Location.

17.0 Recovery Methods

17.1 Processing Plant Description

The processing plant is currently operating at between 700 tpd and 800 tpd of feed, although it has the capacity of about 1100 tpd. As a consequence, it is presently producing only at some 64% of its installed capacity. This capacity shortfall is largely due to the underground mine's ability to provide additional material from its development and stoping operations.

The plant's performance over the 2013 year is shown in Figure 67, where overall gold and silver metallurgical recovery was 91.2% and 43.1%, respectively. These results were achieved from the throughput of some 180,000 t grading on average 6.4 g/t Au and 30.2 g/t Ag. The improvement in gold recovery with increasing gold grade is evident in Figure 67.

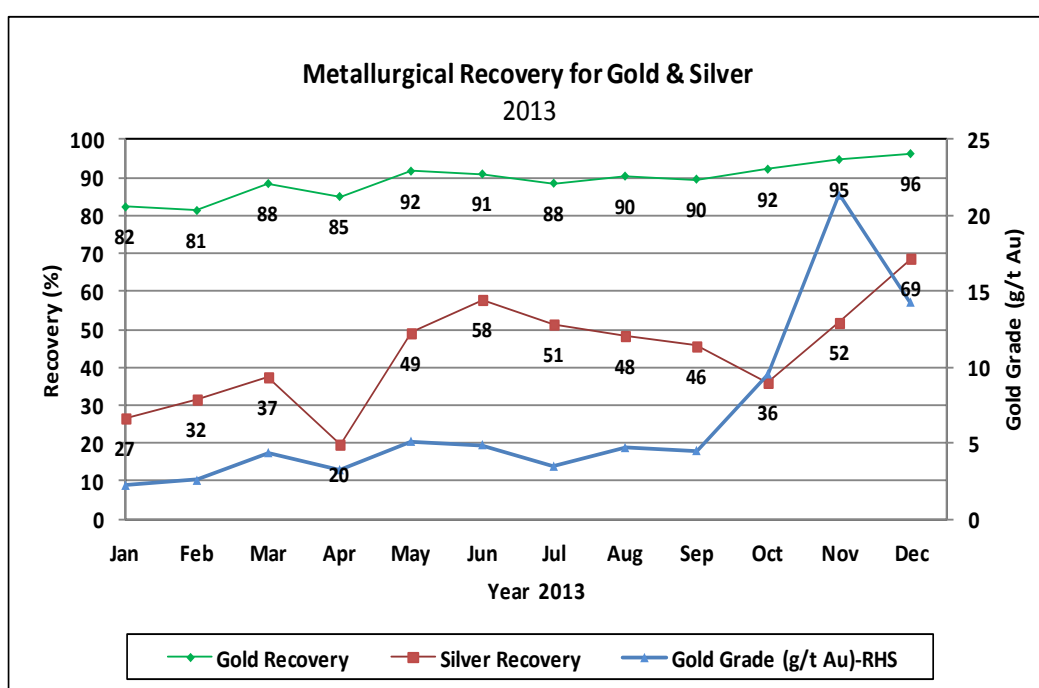


Figure 67: Gold and Silver Metallurgical Recovery – 2013.

The Project's history of production and metallurgical recovery over the past three years is provided in Table 18 and shown in Figure 68.

Year	Tonnes	Gold Production (oz)	Gold Grade (g/t Au) ¹	Silver Production (oz)	Silver Grade (g/t Ag) ¹	Gold Recovery (%) ²	Silver Recovery (%) ²
2011	44,693	5,423	3.77	11,257	7.83	81.8%	42.4%
2012	211,715	17,235	2.53	44,562	6.55	85.8%	29.4%
2013	179,675	30,260	5.24	70,874	12.27	88.4%	43.7%
Total	436,083	52,919	3.77	126,693	9.04	85.6%	38.5%

¹ Represents recovered grade

² Years 2011 and 2012 recoveries are arithmetic averages, whilst 2013 is a weighted average

Table 18: Historical (3 year) Production and Metallurgical Recovery Data.

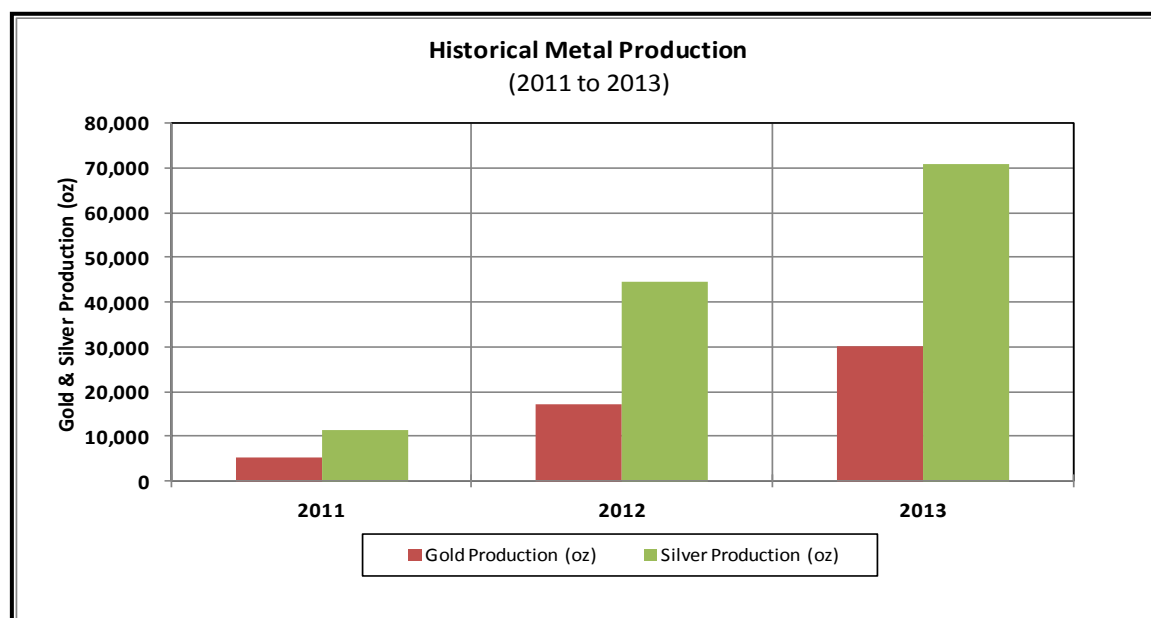


Figure 68: Gold and Silver Production (2011 to 2013).

The metallurgical recovery performance of the processing facility is considered good, given that it has increased steadily over time despite the variability of both feed grade and tonnage being treated. It is also in line with what may be expected of a non-refractory (clean) material type and conforms to the mineralogy of the vein systems (largely gold in quartz with some minor sulphides - chalcopyrite being the major sulphide in the west and sphalerite and galena-dominant assemblages in the east).

The processing facility is also considered to have a robust design in that it has a concentrate gravity recovery system as well as a leaching and CIP circuit. And, as discussed later, there is the opportunity for the circuit to be modified to add in an intense cyanide leach system, as well as a flotation circuit if warranted. The latter may be advantageous when higher copper grade material is encountered together with other sulphide minerals (e.g. galena).

The main components of the Project's processing plant circuit as currently installed and operating are described below. The overall plant flowsheet is shown schematically in Figure 69 and a summary of the Plant Design Criteria is shown in Table 19. The processing plant circuit detail, shown by operational area, is provided in Appendix 1.

The processing facility is located about 7.5 km from the underground mine portal (Figure 70). The 20 t dump trucks take approximately 45 minutes for the journey from mining operations to the processing plant. The figure also shows other elements of the mine's major infrastructure including the main rivers, road (portal to processing plant) and the town of Portovelo.

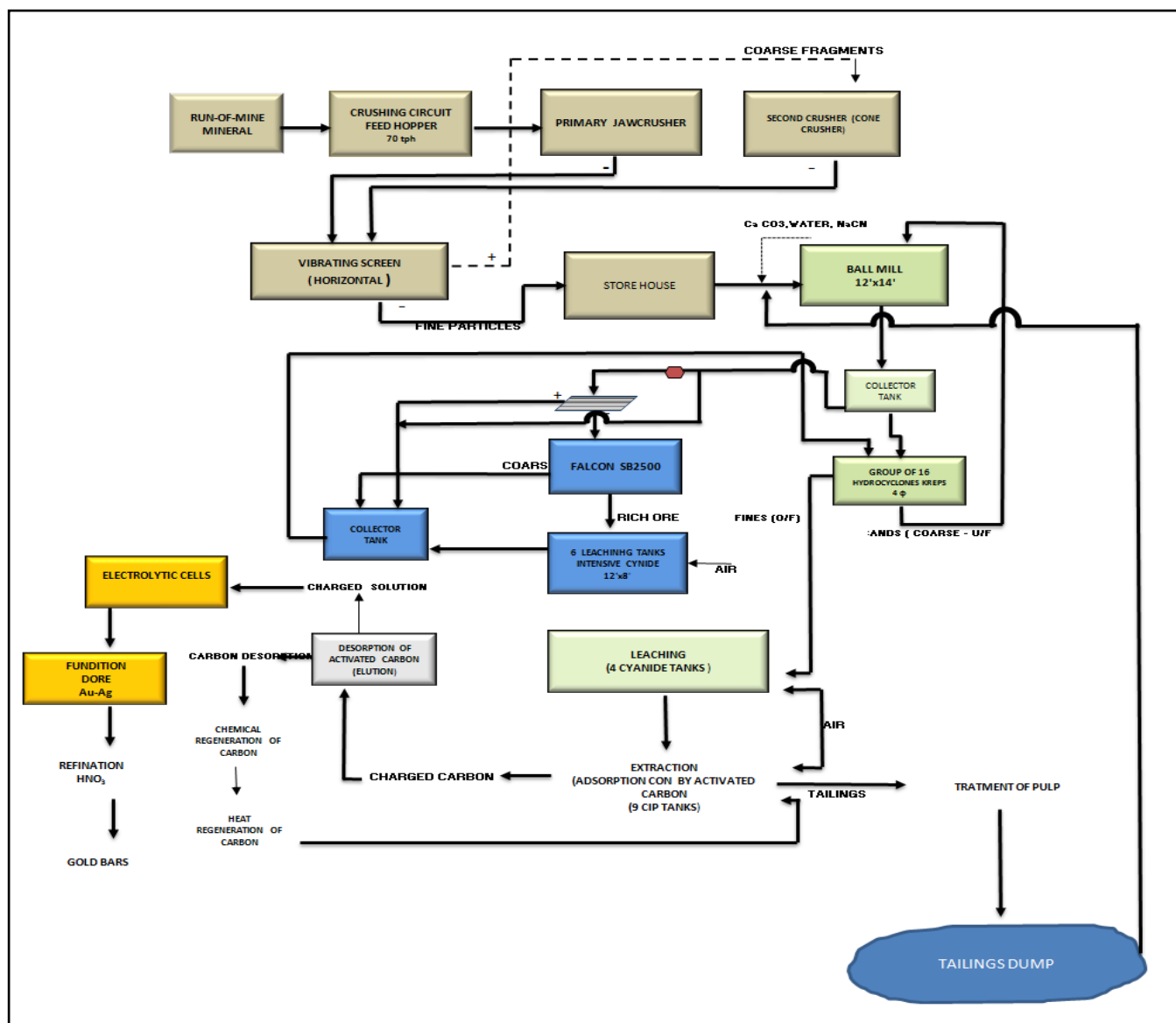


Figure 69: Schematic of Processing Flowsheet.

Parameter	Value
Ore Characteristics	
Ore Specific Gravity	2.75
Operating Work Index	27.2
Throughput	
Crushing Plant	
- Tonnes per hour - Typical	40 tph
- Tonnes per hour - Design	67 tph
Milling & CIP Plant	
- Tonnes per hour	29.2 tph
- Tonnes per day	700 tpd
ROM Pad	
Ore Delivery to ROM Pad	20t Tipper Truck
Size	75m x 75m
Stockpiles	3 Off, High Medium & Low Grade
Crushing	
Ore Delivery to Crusher	via FEL
Type	Primary Jaw & Closed Secondary Cone Crusher
ROM Size	F ₈₀ =254mm
Product Size	P ₈₀ =19mm
Crushed Ore Stockpile Capacity	5000t
Grinding	
Type	Ball Mills
Number	2
Product Size Distribution	P ₈₀ =208µm
Classification	Classification
Circulating Load	310%
Gravity Concentrator	
Type	Centrifugal Concentrator (Falcon)
Concentrate Processing	Intensive Leach
Type	Mechanical Agitated Tank with Air Injection
Tank Live Volume	28 m ³
Number Tanks	6
Leach	
Type	Mechanical Agitated Tank with Air Injection
Tank Live Volume	338.66 m ³
Number Tanks	4 Off
Tank residence	6.45 hrs
Total Residence	25.78 hrs
CIP	
Type	Mechanical Agitated Tank with Air Injection
Tank Live Volume	51 m ³
Number Tanks	9
Residence - per Tank	0.97 hrs
Residence - Total	8.74 hrs
Activated Carbon - per Tank	4 t
Elution	
Type	Zadra
Batch Size	5 t
Number of Columns	2
EW Cells Type	Stainless Steel Cathodes
Number of EW Cells	2 Series Configuration
Tailings	
Tailings Thickening	Tailings Thickener
Thickener Volume	346 m ³
Number of TSF	2
TSF Capacity (2 nd TSF)	2.5 Mt

Table 19: Plant Design Criteria.

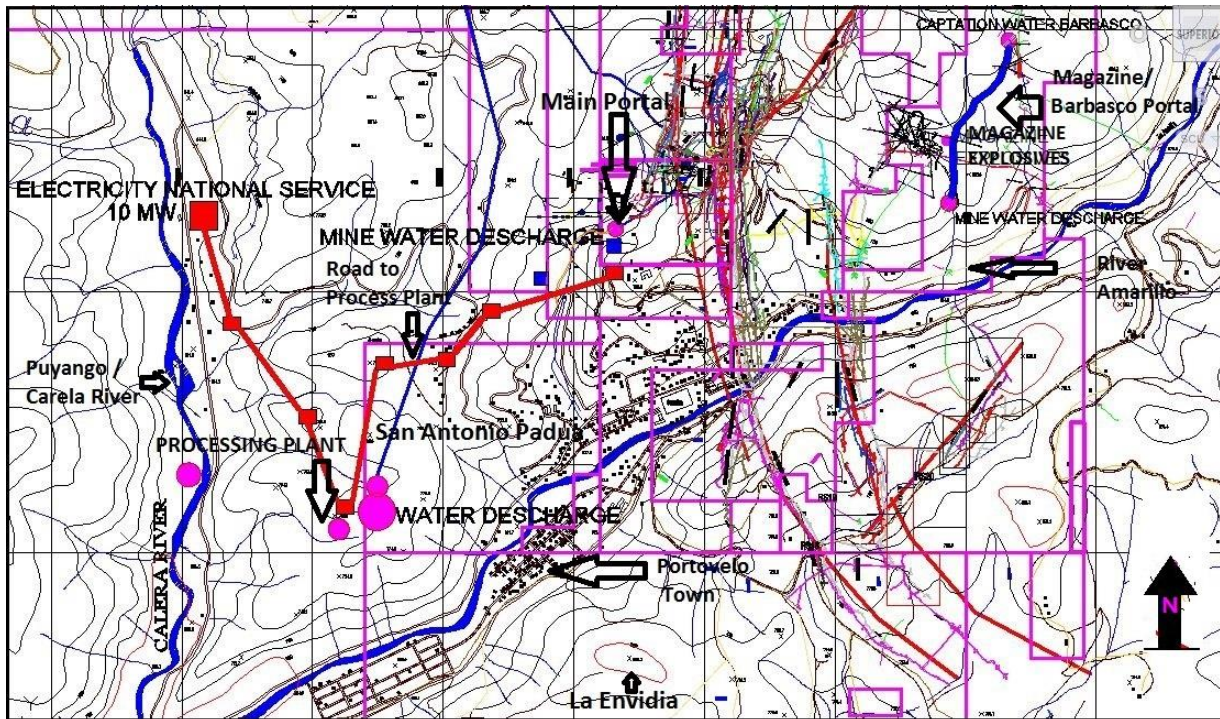


Figure 70: Map Showing Location of Mine Portal and Processing Plant.

17.2 Water Treatment

The processing facility has an abundance of water at its disposal. Notwithstanding, the plant uses several water recovery methods depending on the particular use of the recycled water. Given the current and future planned output of the plant, water usage is unlikely to increase significantly in the short to medium-term (up to about three years).

The plant has two water ponds each of which have a geomembrane coated base. They have a floor combined of cement and ceramic with a system of discharge of sediments both in series and parallel. The approximate dimensions of Pond 1 are 60 m long x 40 m wide and 6 m deep, and is connected via a goose neck with the second pond measuring 40 m long x 40 m wide and 6 m deep. The treatment carried out in the ponds limits suspended material through sedimentation, and also eliminates organic pollution dissolved through by means of an aerobic oxidation of organic matter. Water from neighbouring creeks is also deposited into these pools. The latter is then treated and used exclusively for industrial purposes.

17.3 Run-of-Mine Stockpile

The ROM area for stockpiling the mined material via 20 t tipper trucks (Figure 71) is located in proximity to the tailings dam and measures 75 m x 75 m. The material is stockpiled in three different areas depending on whether its grade is high, medium or low.



Figure 71: ROM Pad Stockpile Area.

17.4 Crushing Circuit

The ROM material is fed into the primary jaw crusher (Figure 72) via a steel feed hopper which in turn is fed by a front end loader. The receiving hopper has a capacity of 41 m³ and is fed at the rate of some 67 t/h. The ROM material (whilst much of it is quite fine there are lumps up to about 0.25 m in size) is fed via the feeder bin to the primary jaw crusher.



Figure 72: Front End Loader Discharging into Jaw Crusher Feed Hopper.

The jaw crusher is a 48 " x 36 " unit and crushes the material to -4", at a capacity rating of about 104 t/h (Figure 73).



Figure 73: Feed into 48"x 36" Jaw Crusher.

An 800 mm wide conveyor transports the primary crushed material to a double deck vibrating screen of dimensions 4.8 m x 1.5 m. The plus 2" material returns to the jaw crusher whilst the -2" +1" material is fed (after passing through an electromagnet for removal of scrap iron) to a 4 ¼' cone crusher (Figure 74), rated at 90 tph. The -1" material reports to the fine material stockpile silo (Figure 75) for delivery to the ball mill circuit via the stockpile reclaim feeders (Figure 76).



Figure 74: Cone Crusher.



Figure 75: Fine Material Stockpile Silo.



Figure 76: Fine Material Reclaim Feeders.

17.5 Milling Circuit

The fine crushed stockpile material is reclaimed from the silo via five reciprocating feeders, of which only one is used at a time. The reclaimed material is fed into the mill feed chute (Figure 77) together with lime, cyanide and water to make up a pulp density of 72% solids (including hydrocyclone underflow). The milling retention time is some 20 minutes and it is estimated that about 73% of gold dissolution takes place in the mill and gravity leaching circuit. The mill

discharge is via an 8" x 6" pump fed to a gravity circuit protected by a screen. Screen oversize is combined with gravity tails and pumped to a cluster of hydrocyclones whilst the underflow reports to the Falcon gravity concentrator (Figure 78).

After gravity concentration, the gravity tailings are pumped via the 8" x 6" pump to a nest of 16 Krebs hydrocyclones (only seven are used for the current tonnage throughput). The gravity concentrate is treated with an intensive cyanidation in six tanks. The concentrate leachate is then returned to the general circuit for recovery with activated carbon.

The hydrocyclone overflow (nominally grind size of 106 μm) is fed to the first leaching tank, at an average flow of rate of 1500 m^3/d and density of 1.32 t/m^3 .



Figure 77: Ball Mill (one of two in parallel).



Figure 78: Falcon Gravity Concentrator.

17.6 Leaching

The leaching circuit (Figure 79) comprises four tanks in series which are 7.5 m diameter and 10 m high with a capacity 384 m³ pulp each (equivalent to 9.2 hours of leaching time per tank, which makes a total leach time of 36 hours operating at 31.4 t/h). The process of cyanidation in these tanks benefits from an injection of air by means of 3 compressors and mechanical agitation. An additional 8% leaching is estimated to take place in this circuit (at a pH of 10.5).



Figure 79: Leaching Circuit Tanks (4 off).

17.7 Carbon-In-Pulp

This circuit comprises nine CIP tanks 3.5 m in diameter and 6 m high with a volume of 51 m³ each (Figure 80). Each tank has a pulp residence time of 1.22 hours which makes a total time of 11 hours in the circuit. The pulp travels continuously from the first tank through to the last tank. By this time about 95% of the gold is in solution and being adsorbed onto the activated carbon. The activated carbon is added directly to the pulp contained in the CIP tanks, the carbon being moved counter-current to the pulp flow. Movement of carbon from tank to tank is by means of air-lift system. Intertank screens are located in the direction of flow of pulp between tanks, which inhibits the passage of carbon between the tanks.

Adsorption of gold onto the carbon continues until a gold loading of about 7.5 kg Au/tonne of carbon is reached. From the highest gold loaded carbon CIP tank, the carbon is then transferred via air lift and discharged onto a loaded carbon vibrating screen. Water sprays installed on the screen assist in separating carbon from the pulp.



Figure 80: Platform of CIP Circuit Tanks.

17.8 Carbon Desorption and Electrowinning

Loaded carbon recovered from the CIP circuit is washed over a vibrating screen and fed to a 4 t stainless steel holding tank prior to be introduced to the desorption circuit (Figure 81). Subsequently, loaded carbon is transferred to the desorption towers (capacity 5 t), consisting of two stainless steel tanks of double-layer thermal insulation.

Once the tanks are loaded, cyanide is added and the solution flows through a heat exchanger. The desorption takes place at a temperature that varies between 90°C to 98°C, with a pH of 12, achieved by the addition of caustic soda at a rate of 100 kg per desorption batch. The flow of water that is required in this process is 1.25 L/s. The temperature is

controlled via a thermocouple installed in the desorption circuit, with the eluant being heated via a heater.

The pregnant solution from the desorption towers is passed through the electrowinning cells consisting of stainless steel cathodes. The cells operate at a DC current of between 500 A and 1200 A and voltages between 2.5 V to 4 V. There are two electrowinning cells, with each producing about 45 kg of precipitate (over some 48 hours), which are then collected and smelted for the production of gold doré.



Figure 81: Carbon Desorption and Regeneration Plant.

17.9 Tailings

Pulp from the last CIP tank flows into a hopper where it is pumped to a tailings thickener of total capacity of 346 m³ of pulp. This thickener is in closed circuit with the excess solution, returning to the grinding circuit by means of two 110 gpm rated centrifugal pumps. The plant has two tailings storage facilities (TSF) available. The current TSF is to be replaced with a second TSF facility close to the treatment plant and will have a capacity of approximately 2.5 Mt (of tailings).

The processing facility does not have a cyanide detoxification system, hence cyanide destruction takes place naturally within the tailings facility.

17.10 Processing Plant Distribution Water

The water from Pond 2 (Figure 82) is used in two ways: (i) some is recycled after filtration and chlorination for grinding, milling, leaching, laboratory, dining room and office areas, from where it goes to the wastewater treatment facility; and (ii) most of the water is pumped into four other streams: (a) a line goes to the leach tanks and part of this water is used as a

coolant; (b) another line goes to the mill pumps and is also used for cleaning, then to leaching tanks and finally this water goes to the tailings dam and recirculated; (c) this line also goes to the mills, then to leaching tanks and finally this water goes to the thickener and recirculated; and (d) a fourth line goes to the crushing area - this serves for cooling of the crusher, and as none of this stream gets contaminated, it is ultimately discharged to the local creeks (at a temperature of 20°C). Some of this line is also used for irrigation of the reforestation areas.



Figure 82: Processing Water Reclaim (Pond 2).

17.11 Processing Plant Equipment

The specific operating parameters for the processing plant are provided in Table 20 and processing equipment list in Table 21.

Parameter	Value
Work Index (W_i)	27.2 kWh/t
Feed F_{80}	19,000 μm
Input energy	16.5 kWh/t
Product (P_{80})	214 μm
Mill power	705 kW (each)
Pulp density	72% solids
Ball loading	82.7 t
Mill effective diameter	12.88'
Mill effective length	13.78'
Critical mill speed	21.3 rpm
Mill operational speed	15.4 rpm

Table 20: Specific Operating Parameters for Processing Plant.

Item	No.	EQUIPMENT	CAPACITY / MEASURES	POWER (hp)	STAND-BY
		CRUSHING SECTION			
1	1	CRUSHER FEED HOPPER	67 t	---	---
2	1	HYDRAULIC RECIPROCATING FEEDER	48 "x 5.30 m BRAND NEUMAC	30	---
3	1	PRIMARY CRUSHER JAW	36 "x 48" brand BIRDSBORO	150	---
4	1	CRUSHER LUBRICATION SYSTEM	BRAND LINCOLN		---
5	1	BELT CONVEYOR UNDERCRUSHERS	48 "x 15.63 m x 0 °	10	---
6	1	BELT CONVEYOR FEED SCREEN	36 "x 36.26 m x 15 °	20	---
7	1	HORIZONTAL VIBRATING SCREENING MACHINES	6' X 16' double floor BRAND TRIO	40	---
8	1	BELT CONVEYOR SCREEN OVERSIZE	36 "x 6.30 m x 0 °	7.5	---
9	1	ELECTROMAGNET	WIDTH 36 "BRAND GATOR		---
10	1	BELT CONVEYOR OVERSIZE RETURN	36 "x 36.26 m x 15 °	20	---
11	1	STANDARD CONE CRUSHER	4 1/4 ' BRAND SYMONS	200	---
12	1	CONICAL CRUSHER LUBRICATION SYSTEM	1750 rpm, 9.5 AMP	7.5	---
13	1	BELT CONVEYOR STOCKPILE	36 "x 44.30 m x 16 °	25	---
14	1	STOCK PILE	CAP 3000 t	---	---

Item	No.	EQUIPMENT	CAPACITY / MEASURES	POWER (hp)	STAND-BY
		SECTION GRINDING			
15	1	BELT CONVEYOR TUNNEL	36 "x 44.94 m x 0 °	10	---
16	1	BELT CONVEYOR FEED MILL	36 "x 42.70 m x 16 °	20	---
17	1	BELT CONVEYOR INTERMILL	36 "x 42.70 m x 16 °	7.5	---
18	5	HYDRAULIC RECIPROCATING FEEDERS	36 "x 3 m BRAND NEUMAC	20	20
19	2	BALL MILL WITH TROMMEL SCREEN	12' x 14' BRAND MARCY	2 x 750 kW	---
20	1	NEST OF HYDROCYCLONES	CLUSTER OF 16 UNITS, BRAND GMAX	---	---
21	2	BEARINGS LUBRICATION SYSTEM	BRAND NEUMAC	5	5
22	2	HORIZONTAL PUMP DISCHARGE MILLS	6 "x 4" MILLMAX BRAND KREBS	40	40
23	1	HORIZONTAL PUMP DISCHARGE MILLS	6 " x 4" MILLMAX BRAND KREBS	---	40
24	1	VIBRATING TRASH SCREEN	4' x 8' BRAND SIZETEC	2	---
25	2	HORIZONTAL PUMP FOR CYCLONES OVERFLOW	6 " x 4" MILLMAX BRAND KREBS	20	20
26	1	VERTICAL PUMP SUMP	2 1/2 "x 24" BRAND GALIGHER	20	---

Item	No.	EQUIPMENT	CAPACITY / MEASURES	POWER (hp)	STAND-BY
		SECTION CYANIDATION			
27	4	TANK AGITATION No. 1,2,3 and 4	7.5 m/day x 10 m height	---	---
28	4	AGITATOR mechanism no 1	DOUBLE HELIX 30 RPM BRAND MIXPRO	120	---
29	1	COMPRESSOR 1000 CFM	BRAND INGERSOLL RAND MODEL SSR 1000 H	271	---
30	1	COMPRESSOR 400 CFM	BRAND INGERSOLL RAND	---	100
31	1	VERTICAL PUMP SUMP	2 1/2 "x 31" BRAND HAZLETON	20	---
		CIP SECTION			
32	9	AGITATION no tanks 1,2,3,4,5,6,7,8 and 9	4 m x 5 m height	---	---
33	9	AGITATOR mechanism no 1	DOUBLE HELIX 56 rpm BRAND MIXPRO	10	---
34	2	BLOWER	BRAND QUINCY	15	15
35	1	CARBON SAFETY VIBRATING SCREEN	4' x 8' BRAND SIZETEC	2	---
36	1	TRANSFER DESORPTION CARBON HOPPER	3 t	---	---
37	1	HORIZONTAL PUMP FOR TRANSFER OF CARBON	13 m ³ /h BRAND HIDROSTAL	10	---

Item	No.	EQUIPMENT	CAPACITY / MEASURES	POWER (hp)	STAND-BY
		SECTION WASHING ACID			
38	1	HARVESTING SHALE	3' x 6' brand SIZETEC	1.5	---
39	1	STAINLESS STEEL HOPPER. WASH ACID - NEUTRALIZATION OF CARBON	BRAND 4 TM	---	---
40	1	TANK HDPE FOR HYDROCHLORIC ACID	BRAND 10 M3	---	---
41	2	HORIZONTAL PUMP FOR HYDROCHLORIC ACID	6 M3/H BRAND AMPCO	1	1
42	1	TANK HDPE FOR SODA CAUSTIC	10 m ³	---	---
43	1	HORIZONTAL PUMP FOR SODA CAUSTIC	10 m ³ /h BRAND HYGINOX	3	---
44	1	HORIZONTAL PUMP FOR TRANSFER OF CARBON	13 m ³ /h BRAND HIDROSTAL	10	---
		REAGENTS SECTION			
45	1	STORAGE OF QUICKLIME HOPPER	25 t	---	---
46	1	AUGER FEEDER	5 "dia x 3.5 m long	5	---
47	1	TANK AGITATOR FOR LIME MILK	2.5 m/day x 2.5 m height	---	---
48	1	MECHANISM AGITATOR FOR LIME MILK	SIMPLE HELIX	2	---
49	1	LIME MILK STORAGE TANK	3 m/day. x 3 m height	---	---
50	1	LIME MILK FEEDING PUMP	2 1/2 "x 2" BRAND ESPIASA	15	---
51	1	TANK AGITATOR FOR CYANIDE	2.5 m/day x 2.5 m height	---	---

Item	No.	EQUIPMENT	CAPACITY / MEASURES	POWER (hp)	STAND-BY
52	1	MECHANISM FOR CYANIDE AGITATOR	SIMPLE HELIX	2	---
53	1	LIME MILK STORAGE TANK	3 m/day x 3 m height	---	---
54	2	HORIZONTAL PUMP FOR CYANIDE	15 m ³ /h BRAND HYGINOX	7.5	7.5
		SECTION DESORPTION - ELECTRODEPOSITION			
55	2	STORAGE of harvested CARBON No. HOPPER 1	4 t	---	---
56	2	DESORPTION TOWERS	1 m DIAMETER x 10 m height	---	---
57	1	SOLUTION TANK SWEEP	3 m DIAMETER x 3 m height	---	---
58	2	PUMP FOR RECIRCULATION SOLUTION SWEEP	25 m ³ /h BRAND HYGINOX	15	15
59	1	HEAT EXCHANGER	STAINLESS STEEL. AREA 120 ft ²	---	---
60	1	SOLUTION HEATER	4 COILS OF 1 1/4 "DIA AREA 100 ft ²	---	---
61	2	DIESEL BURNER	6-13 gph 1.8 MBTU	0.5	0.5
62	2	ELECTROLYTIC CELL	STAINLESS STEEL CATHODES	---	---
63	2	RECTIFIER	DC POWER 500A 2.5V	45	
64	1	HORIZONTAL PUMP FOR TRANSFER OF CARBON	13 m ³ /h BRAND HIDROSTAL	10	---
65	1	HIGH PRESSURE CLEANER			

Item	No.	EQUIPMENT	CAPACITY / MEASURES	POWER (hp)	STAND-BY
		SECTION CASTING			
66	1	INDUCTION FURNACE	1.8 ft ³	165	---
67	1	STOVE ELECTRIC	12 TRAYS	10	---
68	1	CARRO PORTA MOULD FOR SLAG	1 PC	---	---
69	2	MOULD FOR SLAG	0.7 ft ³	---	---
70	3	MOULD FOR BARS	500 OUNCES	---	---
71	1	ELECTRONIC SCALE		---	---
		SECTION THERMAL REGENERATION			
72	1	ROTARY KILN FEED HOPPER	4 t	---	---
73	1	ROTARY KILN'S THERMAL REACTIVATION	MARK LOCHEDA HAGERTY	900	---
74	1	REACTIVATED CARBON DISCHARGE HOPPER	CHAPTER 4 t	---	---
75	1	HORIZONTAL PUMP FOR TRANSFER OF CARBON	13 m ³ /h BRAND HIDROSTAL	10	---
76		STOVE ELECTRIC	12 TRAYS	10	---
77		CARRO PORTA MOULD FOR SLAG	1 PC	---	---
78		MOULD FOR SLAG	0.7 ft ³ , 2 pcs	---	---

Item	No.	EQUIPMENT	CAPACITY / MEASURES	POWER (hp)	STAND-BY
79		MOULD FOR BARS	500 OUNCES, 3 PCS	---	---
80		ELECTRONIC SCALE		---	---
		SECTION WATER SUPPLY			
81	2	PONDS FOR WATER TREATMENT	VOLUME. 10,000 m ³	---	---
82	1	BLOWER		50	---
83	2	HORIZONTAL PUMP FOR WATER	300 gpm MARKS BERKELEY	25	25

Table 21: List of Processing Plant Equipment.

17.12 Energy, Water and Process Materials

The processing facility has a total of 7.5 MW of electrical power available to it, which is ample for its current throughput rate and would also be adequate should the facility increase production to over twice its current 700 tpd. The processing site is fed from the regional electric power grid into the Company's 5 MW @ 69,000 kV / 4,200 V and 2.5 MW @ 69,000 kV / 13,800 V switch-yard. Most of the plant equipment voltage operates at 440 V.

The processing plant has an abundance of water available to it that is sourced from both local creeks and rivers in close proximity to the plant (to which it has access and generous water rights), as well as via its well-managed water recovery/recycling systems. Its water requirements are well differentiated (dependent on particular use of various streams for either potable or unpotable applications), and treatment involves settling and filtration as necessary. Approximately 50% of the processing facilities' water is recovered and re-used within the plant.

The site's water management system is considered be mature and well controlled, and meets with all the relevant environmental permitting standards and requirements.

Typical for a conventional CIP gold processing facility, the Zaruma plant uses the normal consumables such as lime, sodium cyanide, grinding media, sodium hydroxide, oxygen etc. Such items are sourced by means of various strategically (internationally) located procurement consultants. The operation has a warehouse for storing high volume items such as cyanide.

The Company's supply line for the delivery of its processing consumables has been in operation for over 4 years now and is considered to be well established. Given that production output is not expected to increase significantly in the medium term (5 years or more), the current system of processing consumables requirements is considered to be quite adequate.

18.0 Project Infrastructure

The Project is well served for all major infrastructure elements including power, water, roads, telecommunications and logistics generally.

Both the processing plant and underground mine have dedicated electrical power switchyards fed from the regional power grid. The processing facility has a total of 7.5 MW of electric power available, which is fed from the regional power grid (Figure 83) through its own 5 MW at 69,000 kV / 4,200 V and 2.5 MW at 69,000 kV / 13,800 V switchyard.

The processing facility also has available an emergency diesel generator which would be used to keep essential plant operating (such as thickener, leach and CIP tank agitators and electrowinning cells) if the mains power is cut off for any reason. This power unit is rated at 3.25 MW and is housed within the processing facilities fence.



Figure 83: Switchyard 69,000 kV / 420 V (5 MW) and 69,000 kV / 13,800 V (2.5 MW).

The underground mine has power availability of 4.8 MW at 69,000 V / 11,000 V. This quantum of power is adequate for the surface ventilation fan, underground dewatering pumps and underground mobile electrical equipment. Distribution underground is via an 11 kV with secondary distribution at 440 V (except for lighting and other minor electrical consuming equipment units).

Water for both underground mine and processing facility is in abundance, and the operations have access to significant water rights through their water permits. The processing plant is able to draw water from nearby creeks as well as the Carela River (a major river system), although the plant does recycle some 50% of its requirements internally. The underground mine has a flowing creek next to it from which it can draw water, as well as the Amarillo River which traverses the town of Portovelo within a couple of kilometres of the site.

Both operational sites are well serviced by bitumen roads and all-weather dirt roads unlikely to be cut-off for any lengthy periods due to inclement climatic conditions.

The area is also well serviced by telecommunications, and the Company's corporate office in Quito is able to monitor via CCTV activities on site.

In general, the sites are not overly exposed to any major logistical problems. A critical area of logistics is transportation of bullion from the mine site. Security systems and procedures at the processing plant are considered to be very robust, and the checks encountered to enter the area are thorough. The plant has two high level security strongrooms: one where gold sludge from the electrowinning circuit is stored before it is smelted; and another where smelted bullion is stored prior to aerial transport from site. The bullion shipments are flown out randomly by helicopter from a pad located within a couple of hundred metres of the bullion strongroom. The bullion is transported under armed guard from this location for handover to an independent security contract firm which then delivers it to a regional airport for onward shipment.

Since the mine was commissioned in late 2009, support facilities such as water ponds, dams, water and air reticulation, tailings storage facilities and waste dump areas are all well established and conform to the Company's environmental permitting requirements.

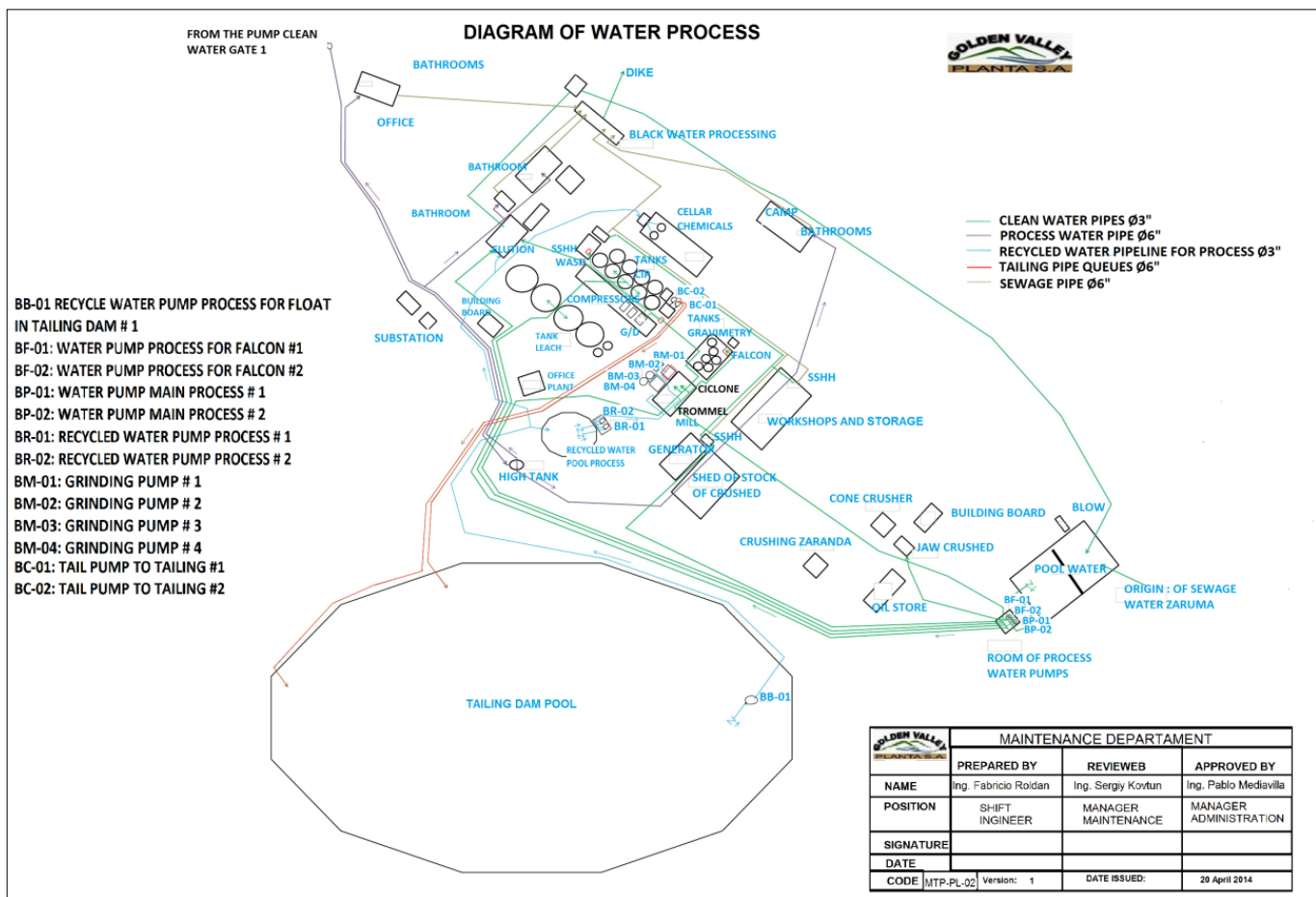


Figure 84: Water Process Layout.

18.1 Tailings

Residuals from the process of the processing plant Svetlana 1 are transported by gravity through a pipe into the tailing pools located at the Svetlana 1 plant. The pools are coated internally with geomembrane and have a system of three horizontal drain lines to help the process of sedimentation and water recovery, and two lines for overflowing water drainage communicated through a control box in the tailing level.

A neutralization plant contains four metal tanks of 44 m³ capacity each. The tailings water enters the first tank with a dosage of acid and peroxide, in which fine activated carbon sacks are found for cyanide removal and to absorb some metals.

Subsequently, the water enters the second tank or agitating tank, where flocculent is dosed so that the flocculation process of the metal oxides is carried out, which passes into the third tank (settler filter), and finally the fourth tank (settling).

The mud accumulated in these tanks are emptied from the bottom of the tank and taken to tailings dam for a final disposing, while the clarified water passes through an activated carbon filter where the rest of metals and cyanide is retained, to ultimately get a good quality of clarified water.

The company carries out quality control of water at the outlet of the filters by measuring free cyanide (by titration with silver nitrate), pH and temperature. Chemical dosing is controlled by a pH sensor placed in the second tank.

18.1.1 Installation of Exhaust Systems

The venting systems that collect the decanted water are located northeast and southwest of the tailings dam at levels 735, 745 and 755, consisting of PVC pipes with a capacity of 200 mm in diameter, which are connected to a 160 mm PVC pipes to drain the water out of the tailings dam, in the respective registry box, and then transported to the detoxification facility.

18.1.2 Internal Waterproofing

To avoid water filtration from the tailings dam in the internal ground, a coating was applied to all areas, walls and platforms, with a polyethylene geomembrane HDPE of 0.75 mm, whose life expectancy is seven years. The geomembrane strips are joined at their ends overlapped and sealed by welding.

18.1.3 Infiltration Collection System

The infiltration collector collection system in general consists of a line of underground drainage of what is captured from the water from the bottom of tailing at level 735. Independently this line discharges the water to the outside of the wall, to a registry box to be later directed via pipes of 160 mm to the detoxification system.

18.1.4 Venting System

The venting systems that collect the decanted water are located to the northeast and southwest, at levels 735, 745 and 755, and consist of PVC pipes 200 mm in diameter for collection, which will be connected to PVC pipes 160 mm diameter to be discharged outside pool tailings in their respective registry box, and then the to the detoxification system.

18.1.5 Water Treatment to be Discharged into the Environment

Wastewater before discharging it to the environment will undergo a neutralization process, in order to meet the parameters for water discharge established by current environmental regulations.

18.1.6 Auxiliary Installations for the Tailings Dam Operation

- Registry box to control water overflow and from which the water is directed to a single point for treatment and further reuse.
- Registry box to control register water under drains that will collect water from different points of out takes of the tailing, which is directed to a single point for treatment and further reuse.
- Outdoor drainage systems for rain water at the bases of the tailing.
- 12 m x 16 m warehouse, where the system for water treatment and future reuse is located.
- Three pools of 30 m x 5 m and 1.20 m in depth, which function as decanters.

18.2 Water Treatment Process Pool Area

The water collected from the Zaruma Urco creek is treated in a pools system prior to entering the metallurgical process, which has been authorized by the National Water Bureau for water use.

This system is located northwest of the processing plant, and consists of two pools with floor coating and a waterproofing geomembrane, also has a combined cement and ceramic floor with an evacuation sediment system in series and parallel.



Figure 85: Water Treatment Process System: Pools 1 and 2.

18.2.1 Black and Grey Water Treatment Area

The treatment plant for black and gray water coming from the restrooms, is located northwest of the processing plant.



Figure 86: Black and Grey Water Treatment Plant.

18.2.2 Water Storage Tanks for Domestic Use Area

The water for domestic use is taken from the Zaruma public portable water system, and is conducted to three elevated tanks where it is treated through an ozone purification system for kitchen and sink use. This infrastructure is located east of the plant.



Figure 87: Elevated Water Tanks & Ozone Purification System.

18.2.3 Industrial Use Water

The water for use in the plant processes is captured in the Zaruma Urco creek. This creek receives, apart from its natural flow, remnant instream of the mining company Bira, plus black and gray water from one sector of the city of Zaruma.

The water from the creek is represented by an intake system, designed and built for a flow rate of 35 l/sec, approved by the competent authority and that it causes no damage to the ecological flow of the creek, in which average values of 120 l/sec were registered in the dry season.



Figure 88: Water Damp System & Pressure Regulator.

This intake system has a desander made form plain concrete at 60 m downstream of the work, to the left margin of the Zaruma Urco creek. From this point, the conduction of water through the entry site to the plant is made by combining PVC pressure pipes of various diameters, having a total length of approx 3,200 m.

All conveyor pipes are buried in a trench 0.6 m wide and 1 m deep in natural terrain, except for two crosses though the Zaruma–Portovelo paved road.

Due to the large amount of sediment that is deposited in the desander (about 30 cm of sediment per week), maintenance of this system is performed every eight days. A weekly maintenance is also performed including cleaning of canals, pipes, valves and grids. Maintenance on the pressure reducing valve is also done monthly.

The collected water enters a pool treatment system located within the premises of the company, where it is subjected to a process of mud sedimentation and aerobic biological treatment. Aerobic biological treatment consists of:

- Addition of nutrients such as nitrogen, phosphorus and potassium.
- Aeration pipes by injecting air generated by a compressor.
- Control and regulation of pH using lime when the pH is less than 5.

The two pools have a floor coating and waterproof geomembrane, and a combined cement floor and ceramic with a vent sediment system in series and parallel.

The water in Pool 1 is subject to a gravity sedimentation process and aerobic biological treatment to improve water quality. The treated water from the overflow passes from Pool 1 to Pool 2, where tilapia fish were placed to act as biological indicators of water quality. Pool 1 and Pool 2 are separated by a wall of rocks 3 m wide, and connected by a narrow channel 1.2 m wide and 1.5 m deep.

When too much water comes out from the Pool 1 of the treatment plant, it is diverted through a canal into the surrounding forested areas to irrigate them (Figure 88).

18.3 Generated Waste

18.3.1 Liquid Waste

18.3.1.1 Domestic Water (Gray and Black Water)

The black and gray water is driven by an internal pipe to the treatment plant. This water is first subjected to a sedimentation process, a continuous aerobic bacteriological treatment, and finally drip chlorination to disinfect it before being discharged into the environment for irrigation. Aerobic bacteriological treatment is the addition of nutrients such as nitrogen, phosphorus and potassium, aeration pipes by injecting air, and pH control.

18.3.1.2 Waste Oils and Greases

Previously, the greases used were mixed with tar for a preparation that was used to lubricate the mill and screwing on of the pipes. The burned grease and oils are temporarily stored in 55-gallon steel drums located in the stock area intended for this purpose. For the final disposal of such hazardous waste a qualified person from the Ministry of Environment is hired.

18.3.2 Solid Waste

18.3.2.1 Domestic Waste

Organic household waste coming mainly from food preparation is placed in two plastic containers of 10 gallons each, the same that are removed daily by the staff working in the kitchen.

Other household wastes such as paper, cardboard, plastic, plastic beverage containers, among others, generated daily, are collected by plant personnel on a truck and taken to the municipal dump.

18.3.2.2 Gas Emissions

Gases from chemical reactions that occur in the smelting and refining circuits in this area are equipped with a Sorbonne filter, alkaline shower and exhaust gas and ventilation shafts. Such Sorbonne filter neutralizes the concentration of nitrous gas generated in this process by 90%.

19.0 Market Studies & Contracts

19.1 Market Studies

The Project is producing bullion (doré) and has a secure logistics set up to transport and sell its doré product. The Company is receiving its cash based on the spot price of gold on day of sale. Hence, at this stage the Company has no need for any market study.

The one and five year price performance of the Company's two principal metals being produced (gold and silver) are shown in

Figure 89, below. Silver production currently only contributes some 4% to the Project's overall revenue stream, and is extracted as a function of the gold recovery operations. Consequently, all recovery operations are targeted towards maximising gold recovery, and silver treated as a by-product.



1 Year Gold Spot Price

**5 Year Gold Spot Price****1 Year Silver Spot Price**



5 Year Silver Spot Price

Figure 89: One and Five Year Gold and Silver Spot Prices (Market Data).

19.2 Contracts

The Company has no material contracts in place. It has an association with a number of procurement professionals in several countries and has effective supply lines to provide its critical consumables requirements (namely cyanide, lime, reagents, grinding media and explosives). The Company also has developed several supply options for some of its more crucial consumables.

The company has five concessions in the exploitation phase. The concessions are Ana Michelle, Bethzabeth, Nueva Esperanza, Soroche Unificado and Barbasco 1 A. In the cases of Soroche and Barbasco, both are mining condominiums, in which Elipe S.A. owns the majority of the property. All the referred concessions are registered under the small-scale mining regime, which allows an individual production of 300 tpd of mining per concession. The small-scale regime means that the concession owner has no legal obligation to commit to a contract with the State government.

The small-scale regime is an administrative resolution issued by Ecuador's Regional Undersecretary of Mines, with the orders in favour of each company registered with the Mining Registry. The small-scale regime results in a fixed royalty in favour of the Government of 3% from the income product of the sale of minerals.

There are three main mining permit categories pursuant to Ecuadorian Mining Law:

- (i) small-scale, as summarized above;
- (ii) medium-scale, which refers to production of 301 to 1,000 tpd. No exploitation contract with the Government is required, there is no applicable windfall tax, and fixed royalty of 4% in favour of the Government applies; and

- (iii) large-scale, which refers to production of greater than 1,000 tpd. An exploitation contract is required to be entered into, a windfall tax applies, and a fixed royalty from 5% to 8% in favour of the Government applies.

If the Company chooses to increase its activities, it would first progress to medium scale mining, on the basis of a project study provided to the authorities evidencing that there is sufficient capability to produce more than 300 tpd. Normally, Mineral Reserves would have to first be demonstrated.

An exploitation agreement with the Government would be required only if production exceeds 1,000 tpd. Under such circumstances, the Company would have to negotiate: (i) a base price for the windfall tax; (ii) how royalties would apply; and (iii) the application of the sovereign adjustment for income taxes.

20.0 Environmental Studies, Permitting and Social or Community Impact

20.1 Environmental

The Project was commissioned in late 2009 and hence is considered, in terms of permitting and approvals, a mature operation. Both the mining operation, registered under the subsidiary company name of Elipe S.A, and the processing operation registered under the subsidiary company name of Golden Valley S.A, are each required to have their own statutory environmental permits and approvals.

The regulatory requirements for both Elipe S.A and Golden Valley S.A are in place and there are no approvals pending or in the process of change that are considered likely to impede the Company's operations, or planned operations, either in the short or longer-term.

Cabo de Horno (Zaruma Project) has the concessions Ana Michelle, Soroche Unificado and Nueva Esperanza, approved for the exploitation phase they are currently undertaking (based on the small-scale licence held by the Company). As described above, the small-scale licence permits the Company to mine 300 tpd per concession and continue with exploration activity.

The processing plant also has a licence to operate. In the case of the mine, the environmental authority has stipulated that quarterly monitoring must be undertaken (i.e. for the mining concessions Ana Michelle, Soroche Unificado and Nueva Esperanza). For the processing plant, a semi-annual monitoring program is required, which includes water, soil and air. Testing for these aspects is carried out with a laboratory certified by the Department of Ecuadorian Accreditation, and also a biotic monitoring. Additionally, the Project reports to the environmental authority in respect of the fulfilment of its Environmental Management Plan. This is in accordance with the relevant statutory work program.

In the first year of operation, an environmental audit was carried out to verify that the Project was fulfilling its requirements and thereafter the environmental audit has, and continues to be, carried out every two years. In addition, the normal monitoring of the items mentioned above takes place.

Should there be an environmental incident or an environmental impact during the mining operations, then the Project has a term of 30 days in which to present to the authorities a contingency management plan. Whilst this process is ongoing, there is no effect on the mining and processing operations (i.e. there is no suspension of operations).

Also, as the monitoring is carried out periodically, the control is maintained during the useful life of the mine and the plant and continues until the Plan of Closure (closing down the operations) has been implemented.

In the case of Elipe S.A, it is required to have the following permits and/or approvals:

- Environmental Licence
- Environmental Management Plan
- Water Rights
- Archaeological Approval
- Community Relations Policy

Each of the above approvals is in place for Elipe S.A and is currently in good standing. These documents are held within Dynasty's Quito office. For the processing operations (Golden Valley S.A), it is required to have the following (and separate from Elipe's) permits and/or approvals:

- Environmental Licence
- Water Rights
- Environmental Management Plan
- Water Rights
- Technical Feasibility Tailings
- Agreement with Communities

Each of the above approvals are well recorded and managed from the Company's head office in Quito. The documents are currently in good standing.

Tables 24 and 25 show the various environmental controls and procedures that Dynasty undertakes to meet its statutory environmental requirements. The funding for all these programs is provided in the Company's budget expenditure plan for the relevant periods within which commitments need to be met.

20.2 Mine Closure

The Zaruma operation has carried out a process of assessing the requirements for when the stage is reached when the operations need to be closed down. This plan incorporates aspects of closure for the underground mining, infrastructure and processing facilities. These closure requirements conform to the operation's regulatory approval systems and take into account the residual environmental facets for closure. It will be necessary that as the operations (mining, infrastructure and processing) change and/or expand with time, so too will be the need to update the agreed closure plan.

As part of the closure plan, the Project has had to assess the closure costs associated with each major operating area. A summary of this cost estimate as it relates to the mine

workings, plant (including equipment, foundations and surface works, roads and tailings storage facilities) and other infrastructure items are provided in Tables 22 and 23, respectively. The total cost is estimated at \$1.88 million (\$1.18 million for the mine and \$0.70 million for the processing and plant area).

These costs take into account backfilling in any underground workings which may damage surface due to subsidence, uninstalling plant and equipment and as well as general overall site clean up including tailings facilities. The latter involves replacing topsoil and re-establishing growth on exposed areas as necessary.

	Total	Machinery & Equipment	Transport	Fuel	Labor	Materials & Tools	Laboratory	Fees	Food
1) De-commissioning of mine external surface infrastructure	\$160,912	\$79,200	\$23,100	\$21,780	\$36,832				
2) Rehabilitation of intervened areas	\$16,176	\$1,600	\$6,300	\$1,980	\$6,296				
3) De-commissioning and disposal of internal mine infrastructure	\$54,793	\$20,400	\$10,080	\$10,296	\$14,017				
4) Post-closure environmental monitoring (water, soil, air). 5 years	\$68,800						\$42,800	\$26,000	
5) Closure of mine internal chambers (sinking high risk areas)	\$530,918	\$279,000	\$75,600	\$78,705	\$77,220	\$20,393			
6) Construction of surface internal and external drainage for mine	\$71,649	\$36,000		\$8,910	\$18,671	\$8,068			
7) Rehab of roads of access	\$4,443	\$2,120	\$1,260		\$1,063				
8) Post-closure community information program	\$42,000							\$42,000	
9) Waste disposal	\$88,628	\$12,600	\$2,940	\$3,234	\$2,878		\$1,989	\$64,988	
10) Post-closure follow-up plan	\$87,914			\$12,045	\$75,869				
	\$58,277								\$58,277
TOTAL	\$1,184,509	\$430,920	\$119,280	\$136,950	\$232,845	\$28,461	\$44,789	\$132,988	\$58,277

Table 22: Summary of Mine Closure Costs.

	Total	Machinery & Equipment	Transport	Fuel	Labor	Materials & Tools	Laboratory	Fees	Food
1) De-commissioning of surface infrastructure	\$159,223	\$82,500	\$5,250	\$33,825	\$32,148	\$5,500			
2) Rehabilitation of intervened areas	\$86,045	\$48,000	\$12,600	\$12,870	\$11,675	\$900			
3) Confining tailings dams	\$144,413	\$37,400	\$9,240	\$10,164	\$11,570	\$76,039			
4) Post-closure environmental monitoring (water, soil, air). 5 years	\$32,457						\$24,457	\$8,000	
5) De-commissioning machinery	\$84,760	\$50,000	\$4,200	\$15,180	\$9,980	\$5,400			
6) Rehab of roads of access	\$15,632	\$10,480	\$840	\$1,188	\$1,094	\$2,030			
7) Post-closure community information program	\$72,000				\$54,000			\$18,000	
8) Waste disposal	\$32,641	\$2,500	\$5,250		\$568		\$1,326	\$22,998	
9) Post-closure follow-up plan	\$43,442			\$5,940	\$35,977	\$1,525			
	\$28,372								\$28,372
TOTAL	\$698,985	\$230,880	\$37,380	\$79,167	\$157,012	\$91,394	\$25,783	\$48,998	\$28,372

Table 23: Summary of Processing Plant and Infrastructure Closure Costs.

PROGRAM		SUB PROGRAM / ACTIVITIES	MONTHS												RESPONSABLE
			E	F	M	A	M	J	J	A	S	O	N	D	
7.4.1 PROGRAM FOR PREVENTION, MITIGATION AND CONTROL OF ENVIRONMENTAL IMPACTS	PARTICULATE MATERIAL	Mandatory controls daily cover material with tarps dump trucks that transport	X	X	X	X	X	X	X	X	X	X	X	X	Dept. of Safety
		Preparation of records for compliance with these activities									X				Environmental Dep
	NOISE AND VIBRATION	Staffing and review of earmuffs to workers who operate machinery and equipment and personnel working near any source of noise identified						X						X	Dept. of Safety
		Build a Noise map to determine the existing levels in different areas											X		
		Soundproofing the area where the emergency generator is										X			Environmental Dep
		Quarterly maintenance of equipment and machinery	X			X			X			X			Dept. Electrical-Mechanical
		Implementation of silencers in the exhaust pipes of vehicles										X			
	GAS EMISSIONS	Implement guidelines set out in Annex 9										X			Mine Technical Dep
	WATER	Design and construction of a water treatment system inside mine									X				Dept. Civil and Environmental Dept.
		Semester system maintenance water treatment inside the mine				X						X			
		Perform design and construction of a treatment system for gray and black water									X				
		Delete all lines containing untreated discharges into waters broken Matalanga									X				
	SOIL	Perform semiannual training for compliance with established measures		X					X						Environmental Dep
7.4.2 OIL MANAGEMENT PROGRAM		Acquisition of Mobile cubetos											X		Environmental Dept., Dept. Civil and Electrical-Mechanical Dept.
		The company will implement the measures necessary for the proper fuel storage (storage for oils).									X				
		Protect fuel storage tanks against corrosion						X						X	
		Inspect monthly storage tanks and vessels as well as pumps, compressors, and other	X	X	X	X	X	X	X	X	X	X	X	X	Environmental Dep
		Monitor the ambient air close to the facilities mentioned	X												
		Procedure should be performed for the management of oil.										X			
		The lubricant storage area will be improved to meet the specifications of the rules										X			
		Monthly maintenance storage area of fuels and lubricants	X	X	X	X	X	X	X	X	X	X	X	X	

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Table 24: Environmental Activities and Controls – Elipe S.A.

PROGRAM		SUB PROGRAM / ACTIVITIES	Month												RESPONSIBLE
			E	F	M	A	M	J	J	A	S	O	N	D	
7.2 PROGRAM FOR PREVENTION, MITIGATION AND CONTROL OF ENVIRONMENTAL IMPACTS	7.2.2 PARTICULATE MATERIAL	Monthly verification of the correct operation of stationary	X	X	X	X	X	X	X	X	X	X	X	X	Electrical-Mechanical Dept.
		Semi-annual preventive maintenance and calibration of stationary		X						X					
		Daily speed verification of volquetas transporting ore from mines	X	X	X	X	X	X	X	X	X	X	X	X	Dept. of Safety
		Mandatory controls daily cover material with tarps dump trucks that transport	X	X	X	X	X	X	X	X	X	X	X	X	
	7.2.3 GAS EMISSIONS	Perform preventative maintenance and calibration of machinery and engines fixed and mobile equipment, semi		X						X					Electrical-Mechanical Dept.
		Preventive maintenance of the Sorbonne neutralizing gas		X						X					
		Measurement of emissions of pollutants in plant generator								X					environmental Dept
	7.2.4 NOISE AND VIBRATION	Purse biannual hearing EPP staff			X						X				Dep Occupational Health and Safety
		Soundproofing the area of emergency generator			X	X									
		Semiannual monitoring of noise levels		X						X					environmental Dept
		Semi-annual preventive maintenance of equipment and machinery		X						X					
		Implementation of noise reduction systems and other equipment, machinery and vehicles			X										Electrical-Mechanical Dept.
		Mapping of noise				X									
	7.2.5 OPERATION THE SLUDGE	Quarterly inspections of tailings pond	X			X			X			X			Dep Occupational Health and Safety
		Perform pool maintenance semester tailings		X						X					
		Conduct semi-annual inspections of the river system influence the pool of tailings		X						X					Environmental and Civil Engineering Dep
		In winter make weekly inspections of drainage ditches	X	X	X	X	X								
		Check the water level in piezometers on the wall of sand					X						X		Environmental Dept., Dept. of Occupational Health and Safety, Technical Department
		Develop a contingency procedure			X										
		Weekly measure the percent solids by weight of the tailings pulp	X	X	X	X	X	X	X	X	X	X	X	X	Dep. Análisis y Control Químico
		Check monthly operational slope angle committed to the project and the width of the top of the wall of sand	X	X	X	X	X	X	X	X	X	X	X	X	
		Check monthly status line pipe tailings	X	X	X	X	X	X	X	X	X	X	X	X	Dept. of Civil Engineering
		Keep clear walls weekly sands	X	X	X	X	X	X	X	X	X	X	X	X	
		Perform monthly internal monitoring of water discharge the sludge	X	X	X	X	X	X	X	X	X	X	X	X	environmental Dept
		Perform quarterly external monitoring water quality discharge	X		X			X			X				
		Conduct annual monitoring of water quality of the receiving water drainage discharge the sludge				X									Environmental Dept., Dept. of Occupational Health and Safety, Technical Department
		Implementation of technical specifications for the different areas where oil is handled in WFP indicated	X												
	7.3 OIL MANAGEMENT PROGRAM	Acquire mobile cubetos.	X												Environmental Dept., Dept. of Civil Engineering and Dept. Electrical - Environmental dep
		The fuel storage tanks shall be protected against corrosion.	X						X						
		Monthly tanks and storage vessels as well as pumps, compressors, and others, and take necessary measures to minimize	X	X	X	X	X	X	X	X	X	X	X	X	Environmental Dept., Dept. of Civil Engineering and Dept. Electrical - Environmental dep
		Perform a procedure for handling fuels.		X	X										
		Annual monitoring of ambient air near the tanks and storage vessels as well as pumps, compressors, and other						X							Environmental Dept., Dept. of Civil Engineering and Dept. Electrical -
7.4. WILDLIFE MANAGEMENT, SITE WORKED REVEGETATION		Contents laces tree planting around pools tailings with WFP specifications					X	X							environmental Dept
		Purchase of own seedlings semi Area		X						X					
		Add nutrient and perform quarterly maintenance plants		X			X			X			X		
		Perform annual work planting (winter time)	X												
		Implementation of fire barriers at the boundaries of the land							X						
		Perform quarterly talks about the prohibitions and regulations outlined in the PMA	X			X			X			X			
		Construction of a nursery						X	X						Dept. Civil and Environmental

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7.11 COMMUNITY RELATIONS PROGRAM	Channel communication mechanisms with the locals and local authorities				X			X			X			X		
	Join in the cultural, social and sports activities make the community.				X						X					
	Training campaigns for the community.				X						X					
	Collaborate with the acquisition of basic infrastructure necessary and important to the community, together with local and regional governments and the state.							X								
	Collaborate with the purchase of computer equipment, supplies and training to primary and secondary educational institutions in conjunction with local governments and the state sectional material.											X				
7.12 MONITORING PROGRAM, MONITORING AND CONTROL	7.12.2. MONITORING AREAS RESTORED	Method contemplate direct visual monitoring spot checks		X							X					
	7.12.3 MONITORING PMA	Semi-annual follow-up concerning the measures proposed in the PMA						X						X		
		Environmental Compliance Audit Preparation									X					
	7.12.4 MONITORING OF PARTICULATE MATERIAL NOISE, AIR, WATER AND LAND	Conduct monitoring every six months (rainy season, dry season) for particulate matter		X							X					
		Semiannual monitoring noise levels				X							X			
		Semiannual monitoring emissions				X							X			
		External monitoring of the plant especially in the area of influence						X								
		Perform semi-annual monitoring of the physical and chemical quality, microbiological and heavy metals from the water from the tailings pond, the creek sn and pools of sewage treatment		X							X					
		Semiannual monitoring physical quality chemical, microbiological and heavy metals from the soil of the creek sn and pools		X							X					
	7.12.5. RECORDS AND DOCUMENTATION	Implement a system of records and files of environmental documentation		X	X	X										

Table 25: Environmental Activities and Controls – Golden Valley S.A.

The Company has well-established relations with the local community and it has systems and procedures for personnel training in matters of both health and safety. Relevantly, the Project retains four safety supervisors on site, plus it has on its payroll three nurses and a doctor to attend to personnel as necessary on site.

21.0 Capital and Operating Costs

The Zaruma Gold Project is reaching a reasonably mature mine stage, whereby its mining and processing operations are essentially funded, and hence most of the future capital expenditures required will be either for additional plant and equipment for expansion to the current operational scale, or provide funding for sustaining the operations. As there is no detailed life-of-mine (LOM) plan for the operation, it is difficult to reasonably assess the long term capital cost requirements.

21.1 Capital Costs

Two main items under consideration are: (i) a Gecko (or similar) high intensity cyanide leach system; and (ii) installation of flotation cells. The former equipment is likely to provide a marked benefit in processing gold recovery. This plant will enable the very high grade material to be extracted early in the process and leached separately, as opposed to the current circuit, where this high grade material (ex gravity concentration) is returned to the lower grade leaching circuit for eventual dissolution and absorption. The cost for this additional plant is not included. The assumption is that any capital expenditure will be offset against improved plant recovery performance.

A summary of the capital cost estimates for the 15 year life of the Project are provided in Table 26. The basis of the cost assumptions is that the underground operation will require about 2,000 m per annum of development, with an estimated cost of \$500/m. This estimate includes the development itself, utilities (power, water) and ventilation. It also assumes that 10% of main development needs to be passively supported (rockbolts and meshing). The processing facility and infrastructure are assumed to undertake improvements equivalent to 4% of installed capital cost.

Sustaining capital is estimated at 3% of \$20 million (the initial plant capital cost) and which is used as an industry benchmark. Mine closure costs are as estimated by the Company (\$1.9 million) and are included at the end of mine life. Benchmarked, these costs appear reasonable. No account is taken of any potential salvage value of plant and equipment. It is assumed that disposal costs will equate to salvage value. This is not an unrealistic position to take, plus on a discounted cash flow basis, is unlikely to have any material impact on the Project's economics.

No contingency amount has been applied to the costs, given their preliminary nature of assessment.

Cost Centre	2015	2016	2017	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	TOTAL
Mining	300	500	500	1,000	1,000	1,000	1,000	1,000	1,000	1,000	1,000	700				10,000
Plant & Infrastructure	300	500	500	1,000	1,000	1,000	1,000	1,000	1,000	1,000	1,000	700				10,000
Sustaining+Closure Costs	600	600	600	600	600	600	600	600	600	600	600	600			1,883	9,083
Total Capital Expenditure	1,200	1,600	1,600	2,600	2,600	2,600	2,600	2,600	2,600	2,600	2,600	2,000			1,883	29,083

Table 26: Summary of Capital Expenditure Estimates (\$000).

21.2 Operating Costs

Operating costs have been assumed in line with the Company's own estimates. These data are considered reasonable and are based on the current and future envisaged mining and processing methods, and scale of the operations generally.

The unit costs applied are \$90/t for mining (which includes secondary and tertiary development but do not take into account decline and major underground infrastructure components, which are accounted for in the mining capital costs). Processing costs are estimated at \$50/t treated, which has been benchmarked against similar scale and gold processing operations and is considered to be an appropriate estimate. Overhead expenses are estimated at \$25/t treated and, although high compared to most comparable operations, the scale of the operation does mean that it absorbs a large fixed unit cost over the limited tonnage treated.

Whilst greater cost definition may be improved by carrying out a detailed fixed and variable cost analysis, such an exercise is likely to only have a marginal benefit due to the very preliminary nature of the data and estimates used.

A summary of the mine's physical and operating cost assumptions for the 15 year life of the Project are provided in Table 26.

No contingency has been applied to the costs.

22. Economic Analysis

22.1 Preliminary Economic Assessment

No preliminary feasibility study has been carried out on the Project, and this PEA is preliminary in nature. It includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves. As such there is no certainty that the results of this PEA will be realised.

Further, as the results of the PEA are based upon Mineral Resources, readers should be aware that Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

The Project has a potential mine life significantly more than 15 years, at the assumed mining production rate and based on its current inventory of all Mineral Resources. However, at this stage of the Project's development, and on a discounted cash flow basis, evaluating Project life longer than 15 years is considered to result in a highly speculative result. For this reason a limit of 15 years is the extent of the Project's life used in the PEA. An alternative case is also evaluated, which eliminates all Inferred Mineral Resources from the PEA production schedule. This case provides the Project with a 10.5 year mine life (at equivalent steady state production, but 12 years in overall duration) and is reported at the end of this section.

This PEA for the Zaruma Gold Project has been conducted to determine the economic returns from a base case (taxed case) as developed for the Project. Results are also presented representing an untaxed case.

To evaluate the cases, an economic model was developed to combine mining and processing data, capital and operating costs, and details of the fiscal regime applicable to the Project. A calculation of normal measures of economic return (net present value and net cash flow) has then been carried out. As the Project is currently in operation, has no debt funding and has limited (relative to cash flow) planned or required capital expenditures in the near term, neither internal rate of return or payback period are applicable as a financial investment return criteria. They are therefore not assessed in the PEA.

The model includes sensitivity analyses, which demonstrate the effect of variations in key parameter inputs on the economic returns from the Project.

The financial analysis is undertaken in United States dollars ('USD' or '\$'), and is not constant currency based (i.e. no escalation applied). The Project has generally been assessed on both an untaxed and taxed basis, under the fiscal regime described. Unless otherwise denoted, all currency units are in USD.

22.2 Basis of Preliminary Economic Assessment

The PEA for the Project is based on the extraction of gold and silver from mineralised material (Mineral Resources) mined from a number of underground veins and processed through the processing facility. Although silver metals contribution on a revenue basis is

minor, it nonetheless is taken into account, and past performance relating to grade and metallurgical recovery are used for its assessment of contribution.

The conceptual throughput production rates are based on the current actual trend as well as a consideration of the future potential output of the Project.

In calculating the Project investment returns, the following key assumptions (of which the technical aspects have been discussed above) have been made:

- mining recovery is estimated at 95%;
- the dilution factor is estimated at 25%;
- metallurgical recovery is estimated at 90%;
- metal prices of \$1,300 per ounce gold and \$21 per ounce silver have been used for the life of the Project;
- no escalation is applied in the analysis; and
- the life of the Project is determined taking into account Measured, Indicated and a portion of Inferred Mineral Resources (to provide a maximum 15 year life of mine for evaluation).

Other assumptions used in the analysis are:

Production - it has been assumed that in the base case all Measured and Indicated Mineral Resources are consumed, as well as 27% of the Inferred category Mineral Resources. The Inferred Mineral Resources extend the mine life from some 10.5 years to 15 years, and are assumed to be treated at the end of the mine's life. The grade over this latter period is reduced by 7% to reflect the difference (lower grade) between Measured/Indicated and Inferred Mineral Resource components.

Revenue - calculation of gold doré production is based on conceptual mining and milling schedules and metallurgical recovery data detailed in other sections of this report. It has been assumed that all doré production will be sold in the period in which it is produced. Currently the Company has no gold hedging in place and all production is sold at spot prices in USD. As Ecuador has a USD based currency, exchange rate is not a factor that needs consideration.

Average gold and silver prices during February 2014 were \$1,300/oz and \$21/oz, respectively. These data have been used in the financial model as a base case. This approach is considered to be reasonable given the exceptionally poor record of consensus forecasts for commodity prices. Sensitivity analyses on gold price have been carried out.

Royalties - two royalties apply to the Project:

- Ecuadorian Government Royalty – this is calculated at 3.0% of the total gold revenue on net value after deduction of non payable metal, transport and refining charges,

which applies on a per-concession basis in respect of the five concessions for which Dynasty has received a small-scale mining licence; and

- IAMGOLD Royalty – calculated at 1.5% of net value also after deduction of non-payable metal, transport and refining charges.

Capital Costs - capital costs have been assessed for the two major cost centres: mining and processing & infrastructure, whilst allowance is made for sustaining capital as a percentage (industry benchmark 3%) of the estimated value of installed plant and equipment (\$20 million).

Contingency - due to the preliminary nature of the analysis and estimates used, no contingency has been applied to either the capital or operating costs.

Escalation - the economic model makes no allowance for escalation. This is considered to be reasonable, as escalation tends to add into the analysis a speculative component which is hard to justify given that it is a PEA level evaluation, and given that forecasts of inflation, prices and costs are notoriously poor.

Operating Surplus - operating surplus comprises Net Revenue less Operating Costs.

Taxation - there are two taxes that the Project has to meet:

- Tax 1 - an Employee & Government Participation tax at a rate of 15%; and
- Tax 2 - corporate tax which is payable at rate of 22%.

Tax 1 is treated as an offset expense in the calculation of Tax 2.

The current investment in mine, plant and equipment estimated at \$20 million is available for depreciation over 10 years, whilst the Company also has tax losses to carry forward of \$16 million. However, tax losses carried forward are limited to 25% of income tax payable in any one year.

Depreciation and amortisation allowance has been calculated for a period assuming 10 years. It has also been assumed that all capital costs, except rehabilitation costs but including the \$16 million sunk costs, are depreciable over the life of the Project.

Cash Flow - Project untaxed cash flow is calculated by deducting operating costs, government fees and charges and capital costs from net revenue. After tax cash flow has corporate and employee and government contribution taxes deducted in the taxed case analysis.

Net Present Value & Internal Rate of Return - NPV is calculated from pre- and post- tax cash flow streams, discounted to start of 2014 at a base discount rate of 10%. The appropriate discount rate to use is often highly contentious, ranging from a supposedly risk free rate to the real rate of risk of the Project. There is no 'right' answer – suffice that 10% is

considered to be a appropriate mid-range rate given the Project's location, development stage and operating risks. Sensitivity analysis has been carried out on discount rates. Internal Rate of Return is not relevant to the analysis.

Payback Period - payback period is not relevant to the analysis.

22.3 Summary of Results

The salient financial results from the PEA are summarised in Table 27 and the key Project statistics and results from the analysis for the life of the Project (taxed and untaxed cases) are provided in

Table 28. Where there is no difference (or the value is zero) between the outcome of the two cases, then only the taxed case data are shown.

- it produces an EBITDA of \$683 million;
- at a 10% discount rate, it produces a taxed NPV of \$218 million and untaxed NPV of \$322 million;
- it produces a taxed net cash flow of \$441 million and an untaxed net cash flow of \$653 million;
- it has a total unit cash operating cost of \$855/oz (taxed) and \$658/oz (untaxed);
- the overall operating costs equate to 51% of total revenue;
- total capital expenditure is estimated at \$29 million; and
- it pays \$66 million in royalties and \$213 million in taxes, over its estimated 15 year mine life.

	Unit	Value
Operating Cash Flow (EBITDA)	\$M	682.5
NPV ¹ (taxe d)	\$M	218.2
NPV (untaxe d)	\$M	321.6
Net cash flow (taxe d)	\$M	440.9
Net cash flow (untaxe d)	\$M	653.4
Unit operating cost (taxe d)	\$/oz Au	855
Unit operating cost (untaxe d)	\$/oz Au	658
Total capex	\$M	29.1
Royalties	\$M	65.6
Taxes	\$M	212.5

¹ at 10% base case discount rate

Table 27: Summary Financial Results for Tax and Untaxed Cases.

The contribution of recovered silver metal to the overall revenue amount, based on the assumptions used, is estimated at 4%. The financial model is shown in Tables 29 and 30 for the taxed and untaxed cases, respectively.

The grade has been discounted by 12.5% for the purposes of this PEA in order to provide a more conservative estimate of recovery.

		TAXED	UNTAXED	TAXED	UNTAXED
PRODUCTION	Unit	Value	Value	% of Revenue	
Mined Mineral Resources					
- Measured & Indicated	000 t	2,590			
- Inferred	000 t	935			
Mineral Resource Grade					
- Measured & Indicated	g/t Au	11.38			
- Inferred	g/t Au	10.59			
Mining Recovery		95%			
Dilution		25%			
Material Mined	000 t	4,186			
Mill Feed Grade	g/t Au	8.93			
Material Milled	000 t	4,186			
Grade	g/t Au	8.93			
Contained Metal	000 oz	1,201			
Metallurgical Recovery		90%			
Recovered Metal	000 oz Au	1,081			
OPERATING REVENUE	\$M	1,459.5			
Royalties	\$M	65.6		4%	
Net Operating Revenue	\$M	1,393.9			
OPERATING COSTS					
Mining	\$M	376.7			
Processing	\$M	209.3			
General & Administration	\$M	104.6			
Concession Fees	\$M	18.8			
Shipment & Refining Charges	\$M	2.0			
Total Operating Costs	\$M	711.4		49%	
Operating Costs	\$/oz Au	658.0			
Operating Cash Flow (EBITDA)	\$M	682.5			
Depreciation & Amortisation	\$M	42.1		3%	
Employee & Gov't Part'n Tax	\$M	96.1		7%	
Corporate Tax	\$M	116.4		8%	
Operating Costs (after tax)	\$M	923.9	711.4	63%	49%
Total Cash Operating Cost	\$/oz Au	855	658		
After Tax Cash Flow	\$M	470.0	682.5	32%	47%
CAPITAL EXPENDITURE	\$M				
Mining	\$M	10.0			
Plant & Infrastructure	\$M	10.0			
Sustaining+Closure Costs	\$M	9.1			
Total Capital Expenditure	\$M	29.1		2%	
Net Cash Flow	\$/oz Au	407.8	604.4	28%	41%
Cumulative Cash Flow	\$M	440.9	653.4	30%	45%
Project DCF (@ 10%)	\$M	218.2	321.6	15%	22%

Table 28: Key PEA Statistics and Results for Taxed and Untaxed Cases.

INPUT ASSUMPTIONS		Mineral Resources (000 t)		Metallurgical recovery		90%		Mining capital (\$000)		10,000		Gold Price (US\$/oz)		1,300		Average February 2014, Ag price \$21/oz					OUTPUTS	
		* Measured & Indicated	2,590																			
		* Inferred	935																			
(US\$ MODEL)		Grade (g/t Au)		Mining Unit Cost (\$/t)		90		Plant & Infra'sture capital (\$000)		10,000		Discount rate (%)		10		Conceptual level data					NPV (\$000) 218,158	
		* Measured & Indicated	11.38																			
		* Inferred	10.59																			
Tax Case		Dilution	25%	Processing Unit Cost (\$/t)		50		Sustaining capital (%) (pa) (\$000)		600		Royalties		4.5%		3% Ecuadorian + 1.5% IAMGold					IRR	
		Mining recovery	95%	G & A Unit Cost (\$/t)		25						Corporate tax		37%		22% Corporate + 15% Employee & Gov't Participation					Payback	
		Value	Unit	Value	Yr -2	Yr -1	2015	2016	2017	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	TOTAL
MINE PRODUCTION					NO PRE-PRODUCTION		Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	
Mined Mineral Resources																						
- Measured & Indicated		t					150,000	175,000	200,000	250,000	250,000	250,000	250,000	250,000	250,000	250,000	250,000	65,000	-	-	-	2,590,000
- Inferred		t					-	-	-	-	-	-	-	-	-	-	-	185,000	250,000	250,000	250,000	935,000
Mineral Resource Grade																						
- Measured & Indicated		g/t					11.38	11.38	11.38	11.38	11.38	11.38	11.38	11.38	11.38	11.38	11.38	11.38	-	-	-	11.38
- Inferred		g/t					-	-	-	-	-	-	-	-	-	-	-	10.59	10.59	10.59	10.59	10.59
Mining Recovery							95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%
Dilution							25%	25%	25%	25%	25%	25%	25%	25%	25%	25%	25%	25%	25%	25%	25%	25%
Material Mined		t					178,125	207,813	237,500	296,875	296,875	296,875	296,875	296,875	296,875	296,875	296,875	296,875	296,875	296,875	296,875	4,185,938
Mill Feed Grade		g/t Au					9.11	9.11	9.11	9.11	9.11	9.11	9.11	9.11	9.11	9.11	9.11	8.47	8.47	8.47	8.47	8.93
PLANT PRODUCTION																						
Material Milled		t					178,125	207,813	237,500	296,875	296,875	296,875	296,875	296,875	296,875	296,875	296,875	296,875	296,875	296,875	296,875	4,185,938
Grade		g/t Au					9.11	9.11	9.11	9.11	9.11	9.11	9.11	9.11	9.11	9.11	9.11	8.47	8.47	8.47	8.47	8.93
Contained Metal		oz					52,152	60,844	69,536	86,920	86,920	86,920	86,920	86,920	86,920	86,920	86,920	80,835	80,835	80,835	80,835	1,201,231
Metallurgical Recovery							90%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%
Recovered Metal		Au (oz)					46,937	54,759	62,582	78,228	78,228	78,228	78,228	78,228	78,228	78,228	78,228	72,752	72,752	72,752	72,752	1,081,108
Recovered Metal		Ag (oz)					93,873	109,519	125,164	156,456	156,456	156,456	156,456	156,456	156,456	156,456	156,456	145,504	145,504	145,504	145,504	2,162,216
OPERATING REVENUE		\$000					63,365	73,925	84,486	105,608	105,608	105,608	105,608	105,608	105,608	105,608	105,608	98,215	98,215	98,215	98,215	1,459,496
Royalties		\$000					2,847	3,322	3,797	4,746	4,746	4,746	4,746	4,746	4,746	4,746	4,746	4,414	4,414	4,414	4,414	65,589
NET OPERATING REVENUE		\$000					60,517	70,603	80,689	100,861	100,861	100,861	100,861	100,861	100,861	100,861	100,861	93,801	93,801	93,801	93,801	1,393,907
OPERATING COSTS																						
Mining		\$000					16,031	18,703	21,375	26,719	26,719	26,719	26,719	26,719	26,719	26,719	26,719	26,719	26,719	26,719	26,719	376,734
Processing		\$000					8,906	10,391	11,875	14,844	14,844	14,844	14,844	14,844	14,844	14,844	14,844	14,844	14,844	14,844	14,844	209,297
General & Administration		\$000					4,453	5,195	5,938	7,422	7,422	7,422	7,422	7,422	7,422	7,422	7,422	7,422	7,422	7,422	7,422	104,648
Concession Fees		\$000					1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,251	1,252	1,253	1,254	18,760
Shipment & Refining Charges		\$000					97	108	118	140	140	140	140	140	140	140	140	132	132	132	132	1,969
Contingency		%																				
OPERATING COSTS		\$000					30,738	35,647	40,556	50,374	50,374	50,374	50,374	50,374	50,374	50,374	50,374	50,368	50,369	50,370	50,371	711,409
Operating Costs/oz Au							655	651	648	644	644	644	644	644	644	644	644	692	692	692	692	658
OPERATING CASH FLOW (EBITDA)		\$000					29,779	34,956	40,133	50,487	50,487	50,487	50,487	50,487	50,487	50,487	50,487	43,434	43,433	43,432	43,431	682,498
Depreciation & Amortisation		\$000					2,120	2,280	2,440	2,700	2,960	3,220	3,480	3,740	4,000	4,260	4,400	2,440	2,280	2,020	1,760	42,100
Employee & Gov't Participation Tax		15%	\$000				4,149	4,901	5,654	7,168	7,129	7,090	7,051	7,012	6,973	6,934	7,213	6,149	6,173	6,212	6,251	96,060
Corporate Tax		22%	\$000				3,651	4,313	7,049	8,936	8,888	8,839	8,790	8,742	8,693	8,645	8,992	7,666	7,696	7,744	7,792	116,436
TOTAL CASH OPERATING COST		\$000					38,538	44,862	53,259	66,478	66,391	66,303	66,215	66,128	66,040	65,953	66,579	64,182	64,237	64,325	64,414	923,905
Total Cash Operating Cost/oz Au							821	819	851	850	849	848	846	845	844	843	851	882	883	884	885	855
CASH FLOW AFTER TAX		\$000					21,980	25,742	27,431	34,383	34,471	34,558	34,646	34,734	34,821	34,909	34,282	29,619	29,564	29,476	29,388	470,002
CAPITAL EXPENDITURE																						
Mining		\$000					300	500	500	1,000	1,000	1,000	1,000	1,000	1,000	1,000	1,000	700				10,000
Plant & Infrastructure		\$000					300	500	500	1,000	1,000	1,000	1,000	1,000	1,000	1,000	1,000	700				10,000
Sustaining (% of above)+Closure Costs							600	600	600	600	600	600	600	600	600	600	600	600			1,883	9,083
Total Capital Expenditure		\$000					1,200	1,600	1,600	2,600	2,600	2,600	2,600	2,600	2,600	2,600	2,600	2,000			1,883	29,083
NET PROJECT CASH FLOW		\$000					20,780	24,142	25,831	31,783	31,871	31,958	32,046	32,134	32,221	32,309	31,682	27,619	29,564	29,476	27,505	440,919
Net Cash Flow/oz Au							443	441	413	406	407	409	410	411	412	413	405	380	406	405	378	408
CUMULATIVE CASH FLOW		\$000					20,780	44,921	70,752	102,535	134,406	166,364	198,410	230,544	262,765	295,074	326,756	354,374	383,939	413,415	440,919	440,919
DCF (Project Net Cash Flow)		\$000					18,890	19,952	19,407	21,708	19,789	18,040	16,445	14,991	13,665	12,456	11,104	8,800	8,564	7,762	6,584	218,158

Table 29: Preliminary Economic Assessment (Taxed Case).

INPUT ASSUMPTIONS		Mineral Resources (000 t)		Metallurgical recovery		90%		Mining capital		(\$000)		10,000		Gold Price (US\$/oz)		1,300		Average February 2014, Ag price \$21/oz		OUTPUTS	
		* Measured & Indicated		2,590																	
		* Inferred		935																	
(US\$ MODEL)		Grade (g/t Au)		Mining Unit Cost (\$/t)		90		Plant & Infra'sture capital		(\$000)		10,000		Discount rate (%)		10		Conceptual level data		NPV (\$000) 321,608	
		* Measured & Indicated		11.38																	
		* Inferred		10.59																	
Untaxed Model		Dilution		25%		Processing Unit Cost (\$/t)		50		Sustaining capital (%) (pa)		(\$000) 600		Royalties 4.5%		3% Ecuadorian + 1.5% IAMGold		IRR		N/A	
		Mining recovery		95%		G & A Unit Cost (\$/t)		25						Corporate tax		22% Corporate + 15% Employee & Gov't Participation		Payback		N/A	
		Unit	Value	Yr -2	Yr -1	2015	2016	2017	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	TOTAL
MINE PRODUCTION				NO PRE-PRODUCTION		Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	
Mined Mineral Resources																					
- Measured & Indicated		t				150,000	175,000	200,000	250,000	250,000	250,000	250,000	250,000	250,000	250,000	250,000	65,000	-	-	-	2,590,000
- Inferred		t				-	-	-	-	-	-	-	-	-	-	-	185,000	250,000	250,000	250,000	935,000
Mineral Resource Grade																					
- Measured & Indicated		g/t				11.38	11.38	11.38	11.38	11.38	11.38	11.38	11.38	11.38	11.38	11.38	11.38	-	-	-	11.38
- Inferred		g/t				-	-	-	-	-	-	-	-	-	-	-	10.59	10.59	10.59	10.59	10.59
Mining Recovery						95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%
Dilution						25%	25%	25%	25%	25%	25%	25%	25%	25%	25%	25%	25%	25%	25%	25%	25%
Material Mined		t				178,125	207,813	237,500	296,875	296,875	296,875	296,875	296,875	296,875	296,875	296,875	296,875	296,875	296,875	296,875	4,185,938
Mined Grade		g/t Au				9.11	9.11	9.11	9.11	9.11	9.11	9.11	9.11	9.11	9.11	9.11	8.47	8.47	8.47	8.47	8.93
PLANT PRODUCTION																					
Material Milled		t				178,125	207,813	237,500	296,875	296,875	296,875	296,875	296,875	296,875	296,875	296,875	296,875	296,875	296,875	296,875	4,185,938
Grade		g/t Au				9.11	9.11	9.11	9.11	9.11	9.11	9.11	9.11	9.11	9.11	9.11	8.47	8.47	8.47	8.47	8.93
Contained Metal		oz				52,152	60,844	69,536	86,920	86,920	86,920	86,920	86,920	86,920	86,920	86,920	80,835	80,835	80,835	80,835	1,201,231
Metallurgical Recovery						90%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%
Recovered Metal		Au (oz)				46,937	54,759	62,582	78,228	78,228	78,228	78,228	78,228	78,228	78,228	78,228	72,752	72,752	72,752	72,752	1,081,108
Recovered Metal		Ag (oz)				93,873	109,519	125,164	156,456	156,456	156,456	156,456	156,456	156,456	156,456	156,456	145,504	145,504	145,504	145,504	2,162,216
OPERATING REVENUE		\$000				63,365	73,925	84,486	105,608	105,608	105,608	105,608	105,608	105,608	105,608	105,608	98,215	98,215	98,215	98,215	1,459,496
Royalties		\$000				2,847	3,322	3,797	4,746	4,746	4,746	4,746	4,746	4,746	4,746	4,746	4,414	4,414	4,414	4,414	65,589
NET OPERATING REVENUE		\$000				60,517	70,603	80,689	100,861	100,861	100,861	100,861	100,861	100,861	100,861	100,861	93,801	93,801	93,801	93,801	1,393,907
OPERATING COSTS																					
Mining		\$000				16,031	18,703	21,375	26,719	26,719	26,719	26,719	26,719	26,719	26,719	26,719	26,719	26,719	26,719	26,719	376,734
Processing		\$000				8,906	10,391	11,875	14,844	14,844	14,844	14,844	14,844	14,844	14,844	14,844	14,844	14,844	14,844	14,844	209,297
General & Administration		\$000				4,453	5,195	5,938	7,422	7,422	7,422	7,422	7,422	7,422	7,422	7,422	7,422	7,422	7,422	7,422	104,648
Concession Fees		\$000				1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,251	1,252	1,253	1,254	18,760
Shipment & Refining Charges		\$000				97	108	118	140	140	140	140	140	140	140	140	132	132	132	132	1,969
Contingency		%																			
OPERATING COSTS		\$000				30,738	35,647	40,556	50,374	50,374	50,374	50,374	50,374	50,374	50,374	50,374	50,368	50,369	50,370	50,371	711,409
Operating Costs/oz Au						655	651	648	644	644	644	644	644	644	644	644	692	692	692	692	658
OPERATING CASH FLOW (EBITDA)		\$000				29,779	34,956	40,133	50,487	50,487	50,487	50,487	50,487	50,487	50,487	50,487	43,434	43,433	43,432	43,431	682,498
Depreciation & Amortisation		\$000				2,120	2,280	2,440	2,700	2,960	3,220	3,480	3,740	4,000	4,260	2,400	2,440	2,280	2,020	1,760	42,100
Tax		\$000																			
TOTAL CASH OPERATING COST		\$000				30,738	35,647	40,556	50,374	50,374	50,374	50,374	50,374	50,374	50,374	50,374	50,368	50,369	50,370	50,371	711,409
Total Cash Operating Cost/oz Au						655	651	648	644	644	644	644	644	644	644	644	692	692	692	692	658
CASH FLOW AFTER TAX		\$000				29,779	34,956	40,133	50,487	50,487	50,487	50,487	50,487	50,487	50,487	50,487	43,434	43,433	43,432	43,431	682,498
CAPITAL EXPENDITURE																					
Mining		\$000				300	500	500	1,000	1,000	1,000	1,000	1,000	1,000	1,000	700					10,000
Plant & Infrastructure		\$000				300	500	500	1,000	1,000	1,000	1,000	1,000	1,000	1,000	700					10,000
Sustaining (% of above)+Closure Costs						600	600	600	600	600	600	600	600	600	600	600				1,883	9,083
Total Capital Expenditure		\$000				1,200	1,600	1,600	2,600	2,600	2,600	2,600	2,600	2,600	2,600	2,600				1,883	29,083
NET PROJECT CASH FLOW		\$000				28,579	33,356	38,533	47,887	47,887	47,887	47,887	47,887	47,887	47,887	47,887	41,434	43,433	43,432	41,548	653,415
Net Cash Flow/oz Au						609	609	616	612	612	612	612	612	612	612	612	570	597	597	571	604
CUMULATIVE CASH FLOW		\$000				28,579	61,936	100,469	148,357	196,244	244,132	292,019	339,907	387,794	435,682	483,569	525,003	568,436	611,867	653,415	653,415
DCF (Project Net Cash Flow)		\$000				25,981	27,567	28,951	32,708	29,734	27,031	24,574	22,340	20,309	18,463	16,784	13,202	12,581	11,437	9,946	321,608

Table 30: Preliminary Economic Assessment (Untaxed Case).

22.4 Sensitivity Analysis

Sensitivity analyses have been conducted at nominal ranges as shown below:

	Low	Base	High
Gold price	70%	100%	130%
Grade	70%	100%	130%
Dilution	70%	100%	130%
Met recovery	93%	100%	107%
Mining opex	70%	100%	130%
Processing opex	70%	100%	130%
Mining capex	70%	100%	130%
Capex	70%	100%	130%

The analysis determines the effects of changes in key parameters (gold price, dilution, mining and processing operating costs, metallurgical recovery and capital expenditure) on NPV and net cash flow (tax and untaxed cases). In assessing the sensitivity of the Project returns, each of these parameters is varied independently of the others. Combined beneficial or adverse variations in any of these parameters will have a non-linear effect on the economics compared to the individual variations considered.

The sensitivity analysis has been conducted assuming no change to the mining schedule (i.e. tonnes and grade output is kept constant, except the grade is lowered proportionally as per the Mineral Resource statement when Inferred Mineral Resources are treated at the end of the base case life of mine). It is also noteworthy that a 30% absolute change in gold price has an equivalent change (i.e. has the same leverage on cash flow and hence NPV) in grade of material mined and metallurgical recovery.

22.5 Sensitivity Results

The PEA has not been tested for large increases in project scale. Although the Project will have reasonable sensitivity to scale (considering the discounted cash flow effects), at this stage, and without further feasibility study work on which to base such an evaluation, this scenario is considered to be speculative.

The results of the sensitivity analysis for NPV are given in Figures 90 and 91 (taxed untaxed, respectively) and for net cash flow in Figures 92 and 93 (taxed and untaxed, respectively). In the sensitivity figures following, the abbreviations are: 'Mining opex' is mining operating costs, 'Processing opex' is processing operating costs, 'Met recovery' is metallurgical recovery and 'All capex' is all capital expenditure.

As expected in terms of key input assumptions, the Project is highly sensitive to changes in price and metal recovery factors (particularly grade of material which is not shown in the charts but has the same effect as price for a similar percentage change, and similar for metallurgical recovery). The next ranking of sensitivity is associated with mining operating costs and to a lesser extent processing operating costs. Changes in capital expenditure have

no material impact on the Project due to their relatively small cost contribution. This is because the underground mine is already in operation and the processing facility has been constructed and is also operating, so the future capital cost impost is assumed to be limited.

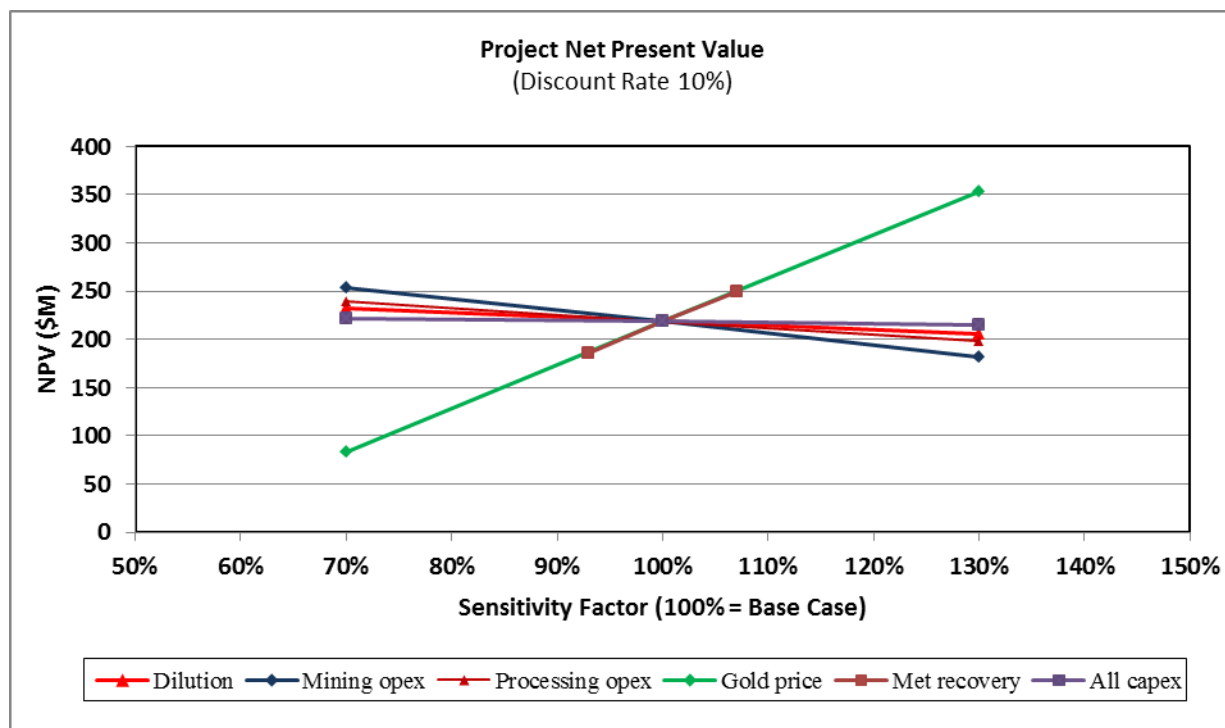


Figure 90: Net Present Value Sensitivities (taxed).

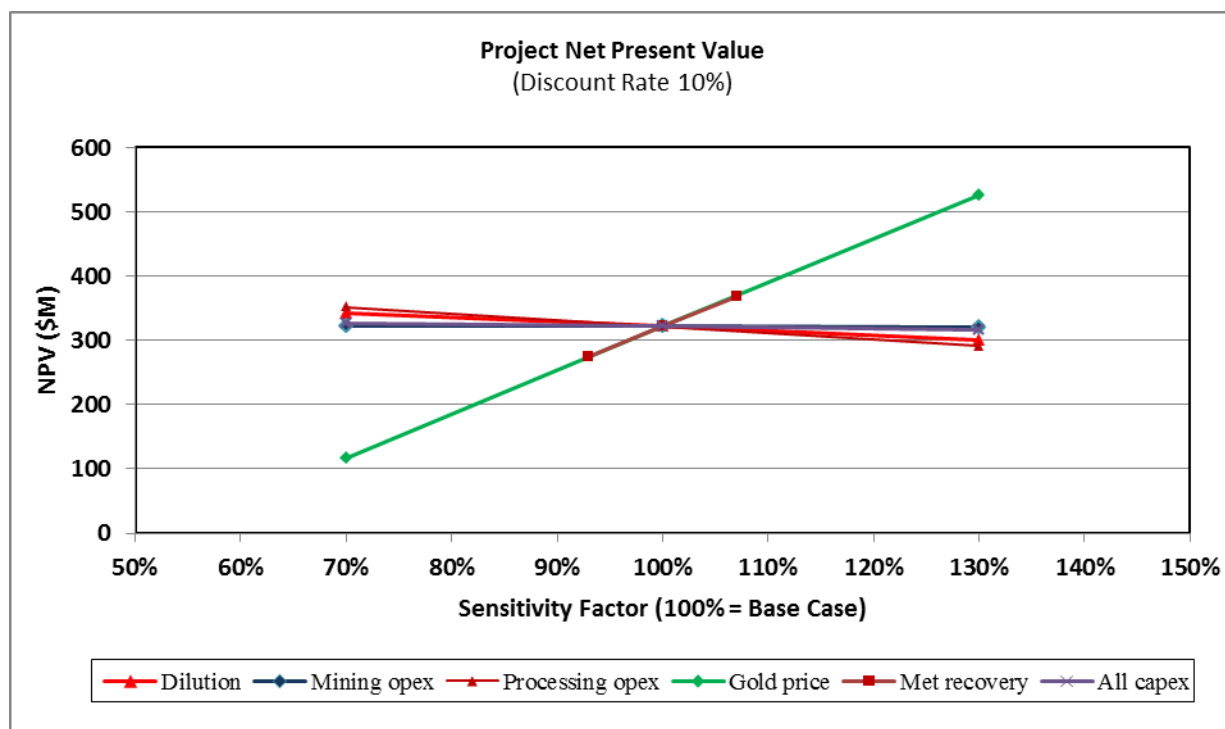


Figure 91: Net Present Value Sensitivities (untaxed).

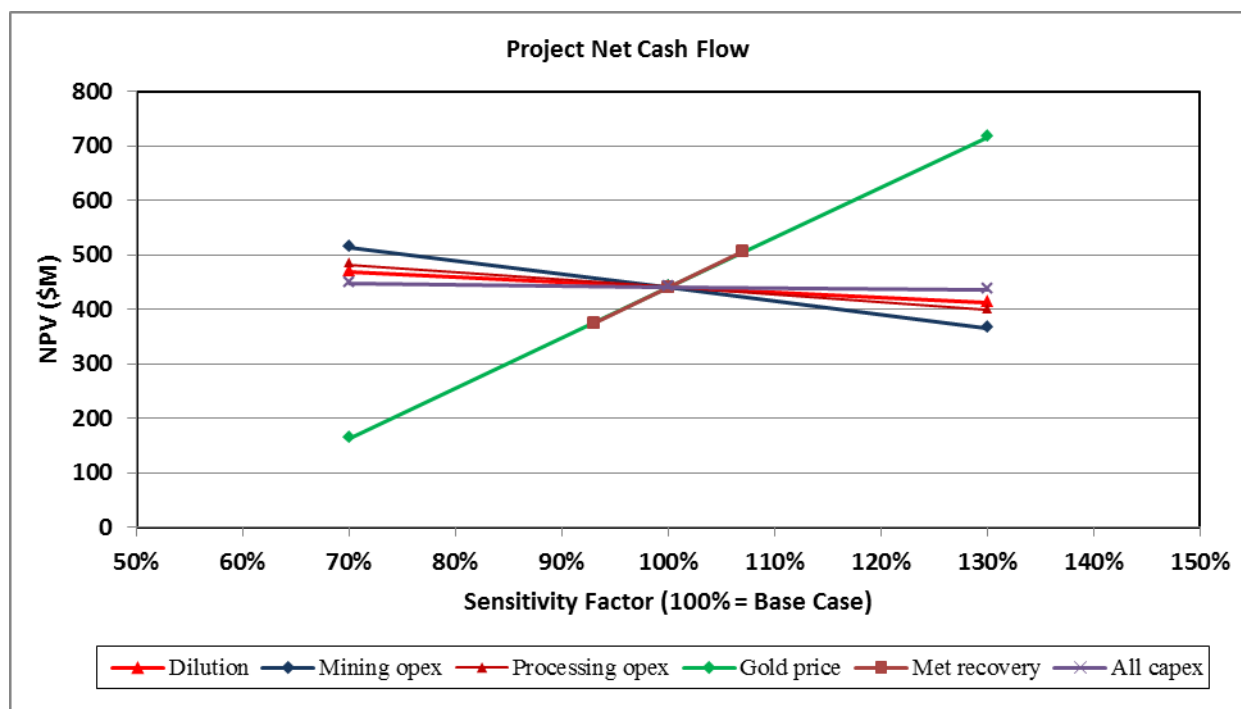


Figure 92: Net Cash Flow Sensitivities (taxed).

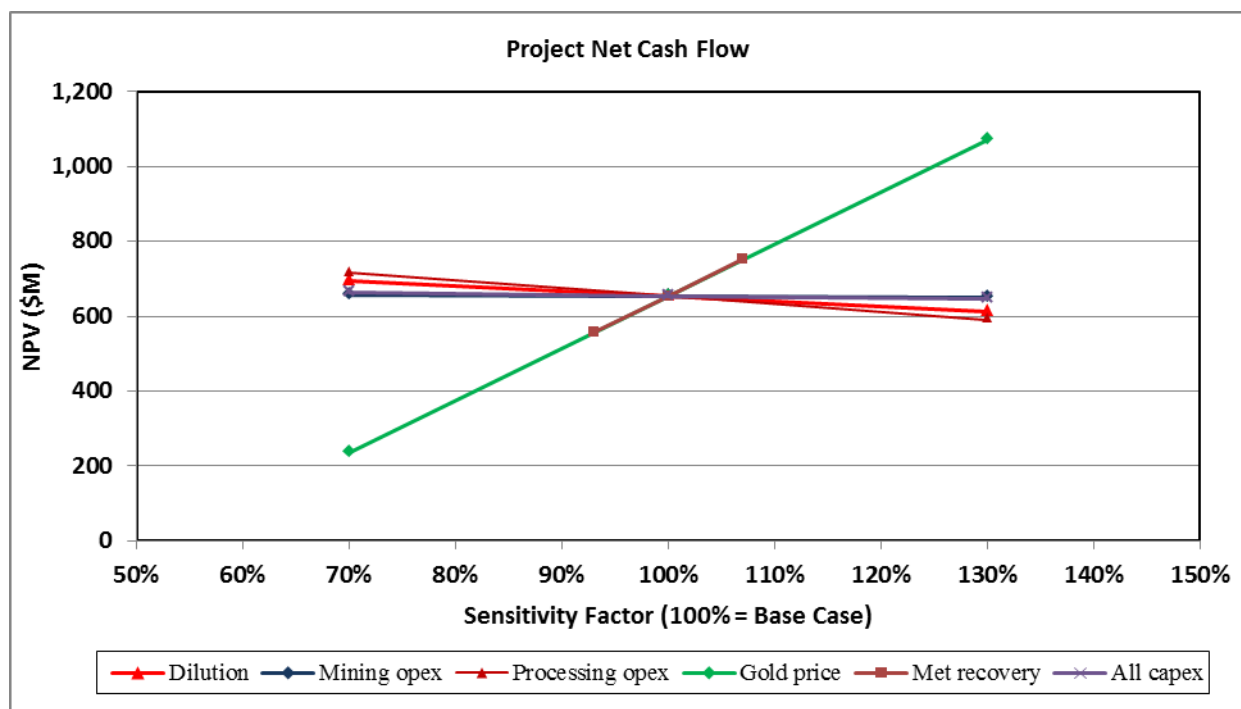


Figure 93: Net Cash Flow Sensitivities (untaxed).

The sensitivity of the Project's NPV (taxed and untaxed cases) to discount rate and gold price are shown in Table 31 and 32, respectively.

It is notable that under the assumptions used, and at a gold price of \$1,000/oz and a discount rate of 20%, the Project still provides an NPV of \$69 million and \$97 million for the taxed and untaxed cases, respectively.

		Discount Rate (%)			
		20	15	10	5
Gold Price	\$ 1500/oz	170,216	216,695	286,746	397,443
	\$ 1400/oz	149,461	190,249	251,709	348,804
	\$ 1300/oz	129,953	165,162	218,158	301,796
	\$ 1200/oz	108,991	138,490	182,874	252,886
	\$ 1100/oz	88,833	112,733	148,635	205,178
	\$ 1000/oz	69,179	87,408	114,738	157,695

Table 31: NPV (\$000) Sensitivity (taxed) - Discount Rate versus Gold Price.

		Discount Rate (%)			
		20	15	10	5
Gold Price	\$ 1500/oz	252,370	321,854	426,598	592,138
	\$ 1400/oz	221,387	282,302	374,103	519,144
	\$ 1300/oz	190,404	242,749	321,608	446,150
	\$ 1200/oz	159,420	203,197	269,114	373,157
	\$ 1100/oz	128,437	163,644	216,619	300,163
	\$ 1000/oz	97,454	124,091	164,124	227,170

Table 32: NPV (\$000) Sensitivity (untaxed) - Discount Rate versus Gold.

On a NPV breakeven basis, the following results are achieved for both the taxed base case and the taxed (without Inferred Mineral Resources) scenarios:

	Unit	Base Case	Without Inferred
Gold Price	\$/oz	686	683
Grade	g/t Au	6	5.5
Operating costs	\$/oz	1,278	1263

Table 33: Breakdown Indicators Without Inferred Resources.

These data show that due to the cash flow discounting effect, the elimination of the Inferred Mineral Resources does not have a major impact on the key breakeven indicators.

22.6 Conclusions

The key conclusions drawn from the PEA are the following:

- as expected the Project's economic performance is highly leveraged to the gold price (and hence grade of material, which for the same absolute percentage change has the same leverage as price);
- metallurgical recovery has a similar leveraging effect on cash flow to grade and gold price;
- mining operating costs have greater leverage than the same change in processing costs, but is still much less sensitive than gold price,
- changes in Mineral Resource dilution has the least sensitivity apart from capex, which based on the assumed amount of capital expenditure required, essentially has no impact on the economics, and
- even in the taxed case the PEA still provides a positive NPV when discounted at a high rate of 30% based on the adopted operating parameters and assumptions used.

22.7 Alternative Case (No Inferred Mineral Resource)

This case evaluates the Project by taking out from the PEA's production schedule all Inferred Mineral Resources. Hence only Measured and Indicated Mineral Resources are analysed for economic returns in the PEA's production schedule. Inferred Mineral Resources represent 27% of the total Mineral Resource in tonnes and 25% of the contained gold metal in the base case schedule. The result of this analysis is provided in Table 34. Table 34 shows the economic comparison between the cases, with and without Inferred Mineral Resources, in the model's production schedule.

The NPV and net cash flow sensitivity of this scenario to key input parameters (same % changes applied as those for the base case) are shown in Figures 94 to 97 for the taxed and untaxed cases, respectively. The sensitivity of these data to discount rate and gold price are shown in Tables 33 and 34 for the taxed and untaxed cases, respectively.

Salient information about the Inferred Mineral Resource used in the PEA is that, if it is eliminated from the production schedule, then in comparison to the base case:

- EBITDA is decreased by 24% to \$521 million;
- the NPV in the taxed and untaxed cases are decreased by 15% and 14%, respectively;
- net cash flow in the taxed and untaxed cases are decreased by 25%;

- it produces a taxed and untaxed NPV at a 10% discount rate of \$187 million and \$277 million, respectively;
- royalties payable are decreased by 25%, whereas taxes payable are decreased by 23%;
- the production schedule is reduced in tonnage by 27% and the contained gold by 25%;
- the overall production grade increases by 7%; and
- mine life decreases to about 10.5 years (assuming the same production profile as the base case).

		With Inferred	Without Inferred	Difference
	Unit	Value	Value	%
Operating Cash Flow (EBITDA)	SM	682.5	520.9	-24%
NPV ¹ (taxed)	SM	218.2	186.5	-15%
NPV (untaxed)	SM	321.6	277.1	-14%
Net cash flow (taxed)	SM	440.9	329.8	-25%
Net cash flow (untaxed)	SM	653.4	491.9	-25%
Unit operating cost (taxed)	S/oz Au	855	849	-1%
Unit operating cost (untaxed)	S/oz Au	658	647	-2%
Total capex	SM	29.1	27.1	-7%
Royalties	SM	65.6	49.2	-25%
Taxes	SM	212.5	164.1	-23%

¹ at 10% base case discount rate

Table 34: Summary Financial Results for Taxed and Untaxed Cases – No Inferred.

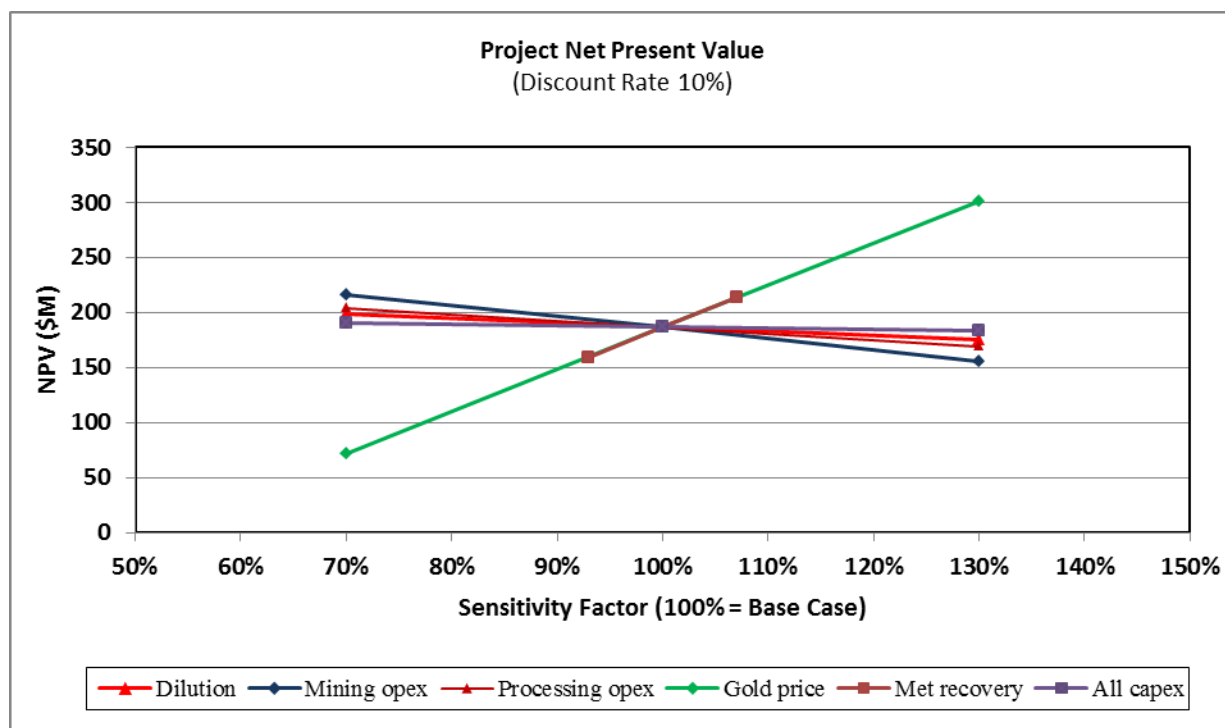


Figure 94: NPV Sensitivities (taxed).

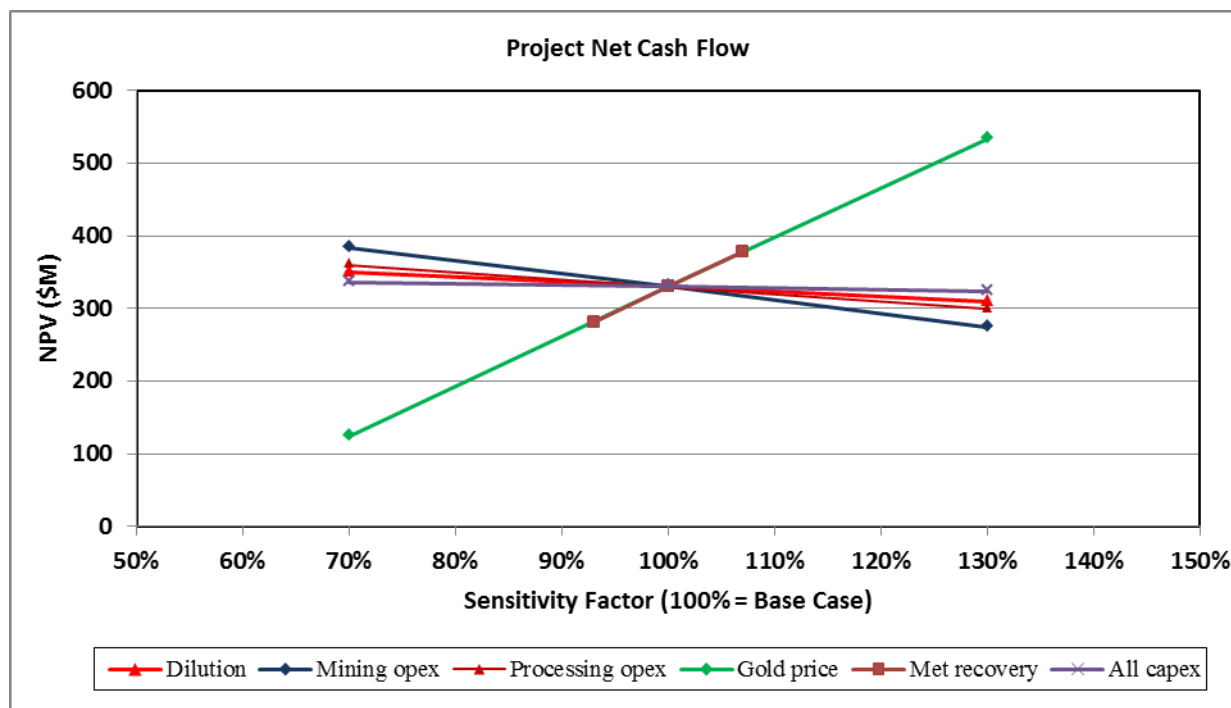


Figure 95: Net Cash Flow Sensitivities (taxed).

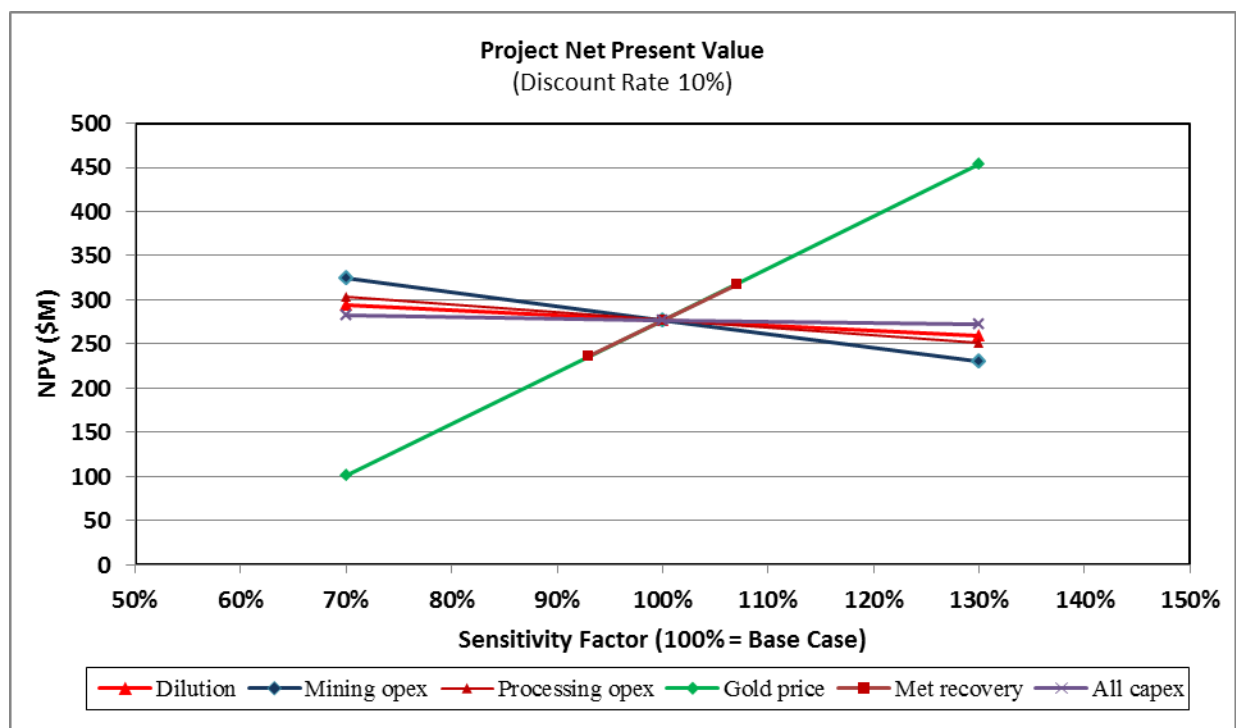


Figure 96: NPV Sensitivities (untaxed).

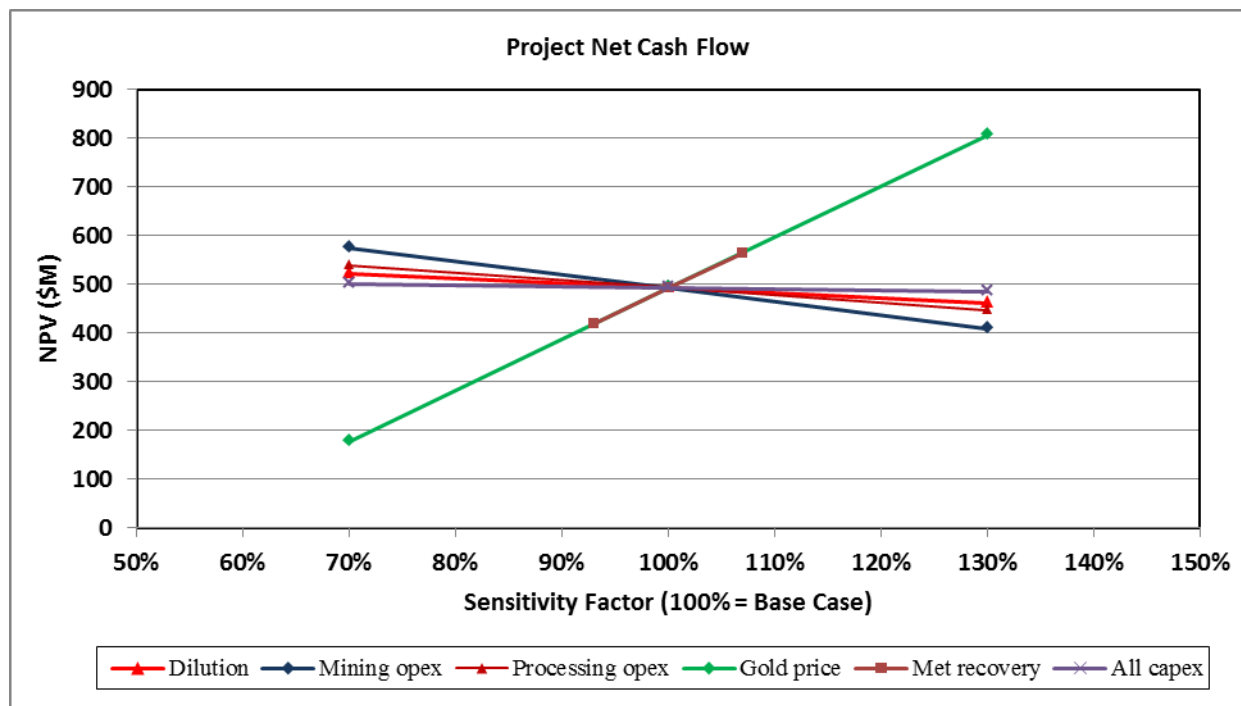


Figure 97: Net Cash Flow Sensitivities (untaxed).

		Discount Rate (%)			
		20	15	10	5
Gold Price	\$ 1500/oz	156,141	192,773	244,491	319,870
	\$ 1400/oz	137,136	169,304	214,715	280,896
	\$ 1300/oz	119,437	147,256	186,493	243,626
	\$ 1200/oz	100,215	123,551	156,459	204,369
	\$ 1100/oz	81,835	100,802	127,518	166,367
	\$ 1000/oz	63,983	78,506	98,934	128,602

Table 35: NPV (\$000) Sensitivity (taxed) - Discount Rate versus Gold Price

		Discount Rate (%)			
		20	15	10	5
Gold Price	\$ 1500/oz	233,975	289,302	367,383	481,109
	\$ 1400/oz	205,279	253,800	322,261	421,951
	\$ 1300/oz	176,583	218,298	277,140	362,792
	\$ 1200/oz	147,887	182,796	232,018	303,634
	\$ 1100/oz	119,192	147,294	186,896	244,476
	\$ 1000/oz	90,496	111,792	141,775	185,318

Table 36: NPV (\$000) Sensitivity (untaxed) - Discount Rate versus Gold Price.

23.0 Adjacent Properties

There are no active mines or formerly active mines located adjacent to the Zaruma Project not already discussed in this report that could provide relevant information that may affect the understanding of the Zaruma Project.

24.0 Other Relevant Data and Information

The author has endeavoured to include all relevant information necessary to make this report understandable and not misleading. The author is unaware of any further data or information that would enhance the understanding of this report.

The primary objective of the report is to provide a PEA of the economic potential of the Mineral Resources reported herein. It is emphasised that this report is not a pre-feasibility study nor is it a feasibility study as defined by NI 43-101 (Part 1.2).

25.0 Interpretation and Conclusions

The principal conclusions from the assessment are as follows:

25.1 Operations

The Company has established underground mining and processing operations at the Zaruma Gold Project. The underground portal is some 7.5 km from the processing facility where last year (2013), the facility treated some 179,000 t for the recovery of approximately 30 koz gold and 71 koz of silver at metallurgical recovery rates for gold and silver of 88.4% and 41.6%, respectively. Whilst the underground mine still requires a great amount of development to reach a stage where it is able to produce at a steady state, the processing plant has over the last year stabilised and is performing well (subject to the varying tonnage and grade delivered from the underground mine). In 2013, the underground mine produced 161 kt of material grading 6.37 g/t Au (including 4800 t grading 16.6 g/t Au from development headings).

The Project has numerous underground veins available to it for exploitation, and it is considered not unreasonable to assess the Project as having at least a 15 year mine life based on current operating and financial metrics and the quantity of Measured, Indicated and Inferred Mineral Resources in the Project's inventory. The underground veins tend to be narrow (mostly <1.5 m), so generally the rate of production will be limited due to the relatively small size of stopes and the amount of development required to bring stopes into production.

The processing plant (a conventional crush, mill, leach, CIP, elution and electrowinning circuit) is well established. Some 25% of the gold is also taken out of circuit via the gravity concentration system. There are available other processing flowsheet options which the Project could embark upon to further increase both throughput and precious metal recoveries.

A preliminary estimate of capital expenditure requirements for the Project for the 15 years life for which it is assessed in the PEA is \$29 million, which comprises \$10 million for each of the underground mine and processing facility areas, and \$9 million for sustaining capital and Project closure costs combined.

25.2 Mineral Resource Estimate

The estimated mineral resources for the Zaruma Gold Project as of the date of this Report are as follows:

Location	Measured			Indicated			Inferred		
	Tonnes (million)	Grade Au (g/t)	Contained Au (Ozs)	Tonnes (million)	Grade Au (g/t)	Contained Au (Ozs)	Tonnes (million)	Grade Au (g/t)	Contained Au (Ozs)
Cabo de Hornos	1.30	13.99	585,000	0.86	12.46	343,000	3.0	12.3	1,201,000
Barbasco	0.19	11.05	66,000	0.09	10.81	33,000	0.3	12.13	128,000
Ayapamba	0.07	12.64	30,000	0.06	11.63	23,000	0.2	12.4	96,000
Vizcaya	0.03	8.40	8,000	0.02	8.40	6,000	0.1	8.4	23,000
Totals	1.59	13.48	689,000	1.03	12.18	405,000	3.7	12.2	1,448,000

Table 37: Current Mineral Resource Estimate.

25.3 Economic Assessment

As required by NI 43-101, it should be noted that these evaluations are preliminary in nature and that the Inferred Mineral Resources in particular are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as Mineral Reserves. As such there is no certainty that the Preliminary Economic Assessment will be realised. Further, as the results of the PEA are based upon mineral resources, readers should be aware that mineral resources that are not Mineral Reserves do not have demonstrated economic viability.

The base case Project PEA provides the following taxed and untaxed results:

	Unit	Value
Operating Cash Flow (EBITDA)	SM	682.5
NPV ¹ (taxed)	SM	218.2
NPV (untaxed)	SM	321.6
Net cash flow (taxed)	SM	440.9
Net cash flow (untaxed)	SM	653.4
Unit operating cost (taxed)	S/oz Au	855
Unit operating cost (untaxed)	S/oz Au	658
Total capex	SM	29.1
Royalties	SM	65.6
Taxes	SM	212.5

¹ at 10% base case discount rate

Table 38: Pre and Post Tax Results.

This analysis indicates that the Project is most sensitive to gold price (absolute percentage changes in gold price and grade are equivalent in sensitivity). This is as expected. The remaining key sensitivities assessed are ranked in accordance with absolute percentage changes: mining operating cost, then processing operating cost and the least sensitive is the dilution factor. The amount of Inferred Mineral Resource used in the PEA was 935 kt Mt grading 9.21 g/t Au. This equates to some 27% of the total Mineral Resource (in tonnes) and 25% in gold content used in the base case production schedule. The grade difference between Measured and Indicated and the Inferred Mineral Resources is some 7%, notwithstanding the difference in confidence levels between the categories.

Based on these results, and taking into account the preliminary nature of the analysis, the Project is considered to be robust. However, a feasibility study needs to be undertaken, with its concomitant level of detail, to be able to provide an assessment that better determines, to an appropriate level of confidence, the Project's cash flows and hence likely economic performance.

25.4 Critical Risk Factors

The most critical risks associated with the Project are as follows:

- Mineral Resource (moderate risk) – tonnage and grade uncertainty (particularly the latter) can be a significant issue in epithermal type deposits, where the mineralised veins tend to be of erratic grade over short spans within the workings. Generally, in converting such a Mineral Resource into a Mineral Reserve, there can be a significant grade reduction between the Mineral Resource and the Mineral Reserve. This is principally due to (i) the additional development material (which is usually at a lower grade tenor than the stope material but still above the cut-off grade) reporting to the treatment facility and (ii) the engineering design factors that are applied to achieve the practical extraction of the stope material.
- Dilution (moderate risk) – the veins being exploited underground are relatively narrow (generally less than 1.2 m and in many cases much less), which means that in mining the mineralised material, it will often be very difficult to limit overbreaking into the waste material and sending it to the processing plant for treatment. However, this risk is mitigated to an extent due to the Company effectively mining to vein widths of less than 1.0 m by the application of hand-held equipment.
- Production (moderate-high risk) – narrow vein deposits with limited tonnes per vertical metre of mineralised material tend to be production rate constrained. They also tend to need more development metreage done in advance due to the lower metreage of development required per tonne of stope material delineated. The smaller scale also means higher unit operating costs. The Zaruma underground operation, in the foreseeable future, will probably be limited to a steady state mining production rate of nominally 250 k tonnes per annum (tpa) to 300 ktpa (before dilution). The absence of a preliminary (or full) feasibility study means the underground mine is operating at a low level of production certainty (tonnes and grade). An appropriate level (e.g. feasibility study) of engineering and design is required to mitigate this production risk.

- Mining Method (moderate-high risk) – the mine's production capacity, productivity and hence costs are very much determined by the mining methods adopted. The mining methods currently in use are labour intensive (hand-held underground mining), particularly the shrinkage system, and are generally considered as obsolete mining methods in more modern mines.
- Development (moderate risk) – narrow vein deposits require higher rates of development advance per tonne of material mined compared to more massive deposits where mineralisation is more disseminated and geometrically broad allowing access to larger volumes of mined material per metre of development. In Zaruma's case there will be pressure on development to ensure adequate numbers of stopes are pre-prepared to enable the operation to both sustain a steady rate of feed to the processing plant, and schedule production so that feed grade does not fluctuate between extremes of high and low grade. This outcome can have severe adverse consequences on processing plant metallurgical recovery.
- Mining cost (moderate risk) – the mining operations are in a very favourable environment in that grades tend to be high (and very high in certain areas), ground conditions are good and the veins are steeply dipping. This generally rates as a low cost mining environment. But, the narrow veins and limited production rate are factors that will tend to increase the risk to costs.
- Metallurgical recovery (low risk) – the processing plant has been in operation for several years and there are detailed records on the plant's ability to recover precious metals, gold in particular. There are also some alternative circuit options for the plant's recovery performance to be improved (e.g. installation of a Gecko or similar system for intense cyanide recovery and/or the addition of a flotation circuit). Notwithstanding, metallurgical recovery tends to be proportional to the feed grade tenor (not unexpected). This is a key reason for the mining operations to be able to provide the plant with a schedule of less fluctuating grade.
- Metallurgical cost (low risk) – the processing facility is well established and operations are generally at a steady state. Although tonnages (and grade) from the mine do fluctuate, the most cost exposure the plant is likely to have is where the underground operations provide very limited tonnages. In this case the plant's overhead cost will increase (on a unit cost basis).
- Infrastructure (low risk) – the operations are generally well served for all utilities and logistics. There appears to be no situation where sustainable production would be interrupted due to power, water or road facilities being cut-off or not available.
- Health, Safety, Environmental and Community (low risk) – the Project has systems and procedures in place for addressing health and safety matters: employees are trained as appropriate and the mine employs both nurses and a health doctor for its operations. Environmental management plans are well established (and controlled from Quito) and the Company has strong community engagement with well established relations and commitments to the community.

- Organisation (moderate risk) – the ability of the Company to recruit and retain the appropriate skilled and experienced personnel for carrying out safe and effective operations is critical, especially given the underground nature of the operations. As previously discussed, mining operational factors, if not well managed, have the potential to significantly impact the cash flow of the Company. And although there is a strong mining culture in the area in which the Project is situated, the surrounding mine workings are associated generally with smaller and less developed operations. These may be considered low technology and labour intensive, and which have little to contribute to the requirements of a modern mining concern.
- Commodity price (high risk) – the Project is similar to most mining organisations in that its cash flows are most sensitive to commodity pricing (noting that a 20% change in price is equivalent to a 20% change in material grade in absolute percentage terms). Although the Project is a higher grade operation when compared to the global average, a rapid and significant fall in gold price (say 30%), will have a dramatic adverse effect on the Project's cash flows. This is especially the case given that the Company is completely unhedged to gold price, and operates in a USD denominated currency environment.
- Sovereign and Country (low risk) – the Project is in a stable political environment as evidenced by general lack of any major political 'unrest' demonstrations since the 'Mining Moratorium' ended and Ecuador has a stable to positive currency rating from the major credit ratings agencies (Moody's, Fitch and S&P). Also, the country's currency is denominated in USD, which is beneficial for Dynasty given that the Company is a gold producer and can sell its output internationally. The mining industry in the country also has an established fiscal framework with which to work. However, Ecuador has relatively punitive mining tax laws, and in the past has been assessed as one of the least attractive jurisdictions for mining. These tax laws encompass a 70% windfall tax impost and this has discouraged many larger mining companies from operating in the country. Several companies, including Kinross Gold Corporation and IAMGOLD, recently left the country, wrote down their Ecuador assets, and took their capital elsewhere. However, in 2013 the Ecuadorian legislature revised the mining laws (somewhat), and whilst still quite punitive, provided much relief for medium and small-scale operators. These categories encompass underground operations under 1,000 tonnes per day (tpd); open-pit operations under 2,000 tpd; and alluvial operations under 3,000 tpd. Under these daily mining rates, mine developers will not need to sign an exploitation contract with the Ecuadorian government, an action that would otherwise trigger the 70% windfall tax. Instead, under this category, a 3% royalty applies to small-scale operators, and 4% to medium-scale operators, per concession - along with a universal (and previously existing) rule: that the state gets 50.1 percent of the benefits from any mining project, which includes the sum of royalties, income tax and others contributions to areas such as social security.

In summary, whilst this PEA is conceptual in nature, a feasibility study and its associated investigation (technical, operational and commercial) will provide far greater weight to the risk assessment process; provide a basis for their rating, as well as define a management plan to either mitigate or ameliorate them. In time, it is expected that many of these risks will be reduced as the mine reaches a steady state of capacity, and the operations stabilize and

become more predictable. The indication is that the Project has the potential to become a high margin producing gold operation.

26.0 Recommendations

26.1 Feasibility Study

The Company has made significant progress, albeit without a prefeasibility or feasibility study with which to guide its development activities. Notwithstanding, the Zaruma Gold Project has now reached the stage where it is operational, and has managed to delineate adequate underground development and stope workings to potentially feed higher tenor grade (>6 g/t Au) material through the processing facility on a continuous basis, produce gold and hence achieve cash flow. It is also significant that all necessary plant and equipment is in place to achieve this, and the Project currently carries no debt.

Progress made by the Company since the last NI 43-101 report (the 2006 Report) has to an extent reduced the overall Project risk increasing the potential to become a longer-term established mining concern. This means that the Company is now in a position where it can work towards developing an Mineral Reserve estimate. This will require a life of mine plan providing greater confidence in the mine's production schedule (tonnes and grade), material processing performance, project resource requirements and is aimed to give an improved prediction of cash flow and therefore assessment of investment performance.

It is considered imperative that the feasibility study pays significant attention to mining method selection. This underground operation, being relatively small-scale, increases the risk of mining inefficiencies and ineffectiveness (such as mining dilution, mining recovery and cost per tonne stoped), and which can significantly impact on the Project's cash flow ability if not well managed.

It is therefore recommended that Dynasty proceed towards the development of at least a preliminary, if not a full, feasibility study, to which much of the progress made over the last few years will have an important contribution. This is particularly so with regards to information and experience arising from underground development grade and heading advance, stope development, ground support systems, ventilation systems, hauling performance, processing plant and equipment operations and precious metal flowsheet and circuit recovery performance. The Project will also glean valuable data about its longer-term infrastructure requirements and overall organisational requirements effectiveness.

In terms of cost, given the current maturity of the operation, a feasibility study is estimated to cost about \$1 million (approximately 5% of implied capital expenditure of some \$20 million). This cost is subject to the current Mineral Resources inventory largely being used for an approximately 15 year mine life production schedule, and therefore no extensive additional drilling being required to increase the Mineral Resource base.

Any feasibility study work should be able to be undertaken without affecting the day-to-day operations of underground mining and processing of material. However, given the Company's current organisational resources, the study will most likely need to be contracted out to an appropriately credentialed engineering company. This will also provide a higher level of confidence in the study's output. A key benefit of this outcome is that it will help the Company to hedge its gold production in the future should it want to do so.

26.2 Operations

There are considered to be two main operational areas where the Project could potentially significantly benefit. The first is underground mine development. Currently, the underground mine has very few stopes developed. The further the operation progresses with its development, providing greater flexibility with regards to headings developed ahead of stopes, and number of stopes available for exploitation, the higher the confidence will be of providing the processing facility with predictable tonnes and grade. It will also reduce the risk of lost production from unplanned inundation of water, which is a possible event.

The second relates to the processing facility, where presently high-grade material from the gravity concentration circuit is returned to the lower grade tenor leaching circuit. It is evident the Company is fully aware of the benefits of changing this system so that the gravity concentration product is separately cyanide leached via a proprietary intense cyanide leaching process. The payback for the capital expenditure for the installation of such a system would likely be very rapid, as the additional revenue due to increased gold recovery should be significantly greater than the cost of the plant to achieve it.

26.3 Exploration

- It is recommended that exploration be continued to determine the full extent of the mineralisation within the Zaruma concessions and to explore for new mineralised bodies.
- Surface mapping and sampling along with targeted drilling below the known mineralised outcrops to determine the grade characteristics of the veins with depth should continue.
- Underground and surface oriented drill holes spaced 50m apart along the vein strike should be applied to aim for the Inferred Resources to be upgraded to Indicated and/or Measured. The same platform may then be used for two or more oriented drill holes with different angles (dips).

Costs to complete the advanced exploration work and feasibility study are currently estimated to be as follows:

- Underground mapping	\$ 50,000
- Surface trenching and mapping	\$ 50,000
- Drilling (approximately 8,100m)	\$ 1,650,000
- Sample assays	\$ 70,000
- Environmental control	\$ 50,000
- Social	\$ 70,000
- Feasibility Study.....	<u>\$ 1,000,000</u>
Total.....	\$ 2,940,000

27.0 References

Baldock, J (Coordinator), 1982: "Mapa Geologico Nacional Ecuador". Instituto Geografio Militar.

Billingsley, P., 1926, "Geology of the Zaruma Gold District of Ecuador". American Institute of Mining and Metallurgical Engineering, v.74, p. 255-275.

Enadimsa, 1977: "Estudio Geologio – Minero del Area “La Zanja”. Mision Geologico-Minera de Espana en Ecuador.

Holly, WJ, & Maynard A.J., 2006, "Independent Preliminary Assessment - Zaruma Project - El Oro Province, Ecuador." Company Technical Report.

Jemielita, R. and Bolaños, J., 1993, "Mineralization, Mineral Potential, and Metallogenesis of the Cordillera Real of Ecuador." CODIGEM – Mission Britanica, Quito.

Kalinaj, M., 2004, "Zaruma Project, Ecuador - Technical Report." Company Technical Report.

Litherland and Zamora, 1991: "A Terrane Configuration for the Northern Andes". Comunicaciones Departamento de Geologia Universidad de Chile No. 42.

Mapa Geológico de La República del ECUADOR, 1969. National Service of Geology and Mining of Ecuador and Bureau d'Etudes Industrielles et de Cooperation de Institut Franç du Petrole

Maynard, A.J., 2005, "Independent Geological Evaluation - Zaruma Project, El Oro Province, Ecuador". Company Technical Report.

Pilatasig, L and Gordon, D, 2004: "Mapa Geologico Region Sur del Ecuador". Ministerio de Energia y Minas. Edicion Provisional.

Quiroga, G.J., Visit to Portovelo Minino District, El Oro Province Southern Ecuador. Unpublished Internal report for TVX Gold Corporation, March 7, 1995, 8 p.

Schenk, Viger, and Anderson, 1997: "Maps Showing Geology, Oil And Gas Fields, And Geologic Provinces Of The South America Region". United States Geological Survey OFR 97-470D.

Spencer, R.M., et al, 2002, "The Portovelo-Zaruma Mining Camp, Southwest Ecuador. Porphyry and Epithermal Environments." SEG Newsletter, Society of Economic Geologists, 2002, Number 49, 14 p.

Sillitoe, R.H., 2000: Comments on the Portovelo-Zaruma vein gold district and environs, Ecuador, Unpublished report, IAMGOLD Corp., 6 p.

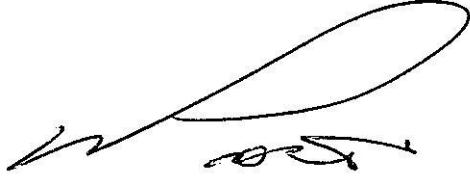
Van Thournout, Salemink, J., Valenzuela, G., Merlyn, M., Boven, A. and Muchez, P., 1996, "Portovelo: A Volcanic-Hosted Epithermal Vein-System in Ecuador, South America." Mineralium Deposita 31, p. 269-276.

Vikentyev, I., Banda, R., Tsepin, A., Prokofiev, V., Vikentyeva, O., 2005, Mineralogy and formation conditions of Portovelo-Zaruma goldsulphide vein deposit, Ecuador, Bulgarian Academy Of Sciences Geochimistry, Mineralogy And Petrology, Sofia, Au-Ag-Te-Se Deposits 43 pp 148-154

United Nations, 1972: "Exploration for Metallic Minerals in Southern Ecuador". Technical Report No. 14, UN Development Programme.

28.0 Date and Signature Page

This report, effective as of September 17, 2014, was prepared and signed by the following authors:

A handwritten signature in black ink, appearing to be 'R. Procter', with a large, sweeping loop at the top.

Richard L Procter BSc (Eng), MBA, MIMMM, CEng

A handwritten signature in black ink, appearing to be 'A. Maynard', with a stylized, cursive script.

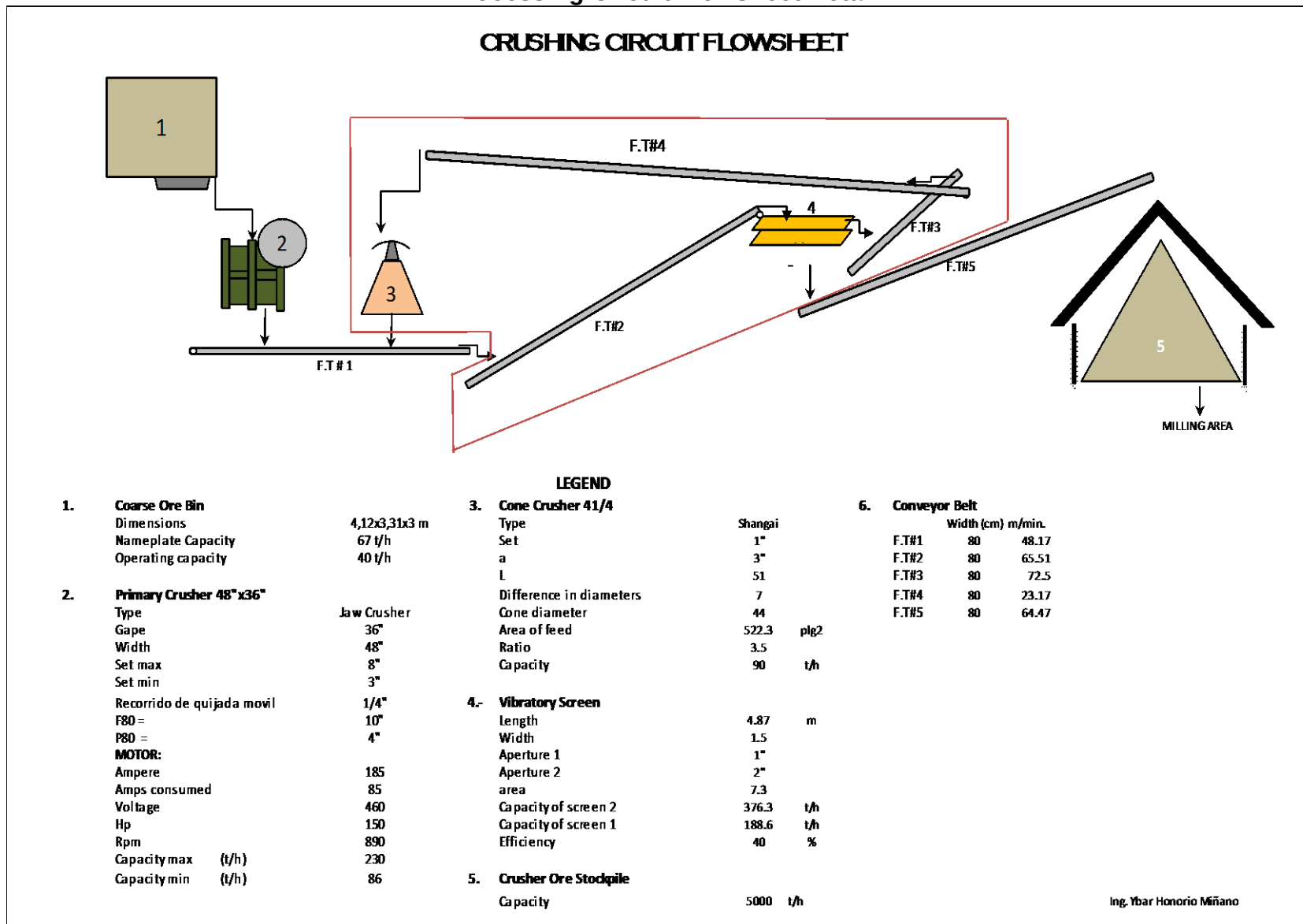
Allen J Maynard BAppSc (Geol), MAIG, MAusIMM

A handwritten signature in blue ink, appearing to be 'P. Jones', with a large, stylized 'P' and 'J'.

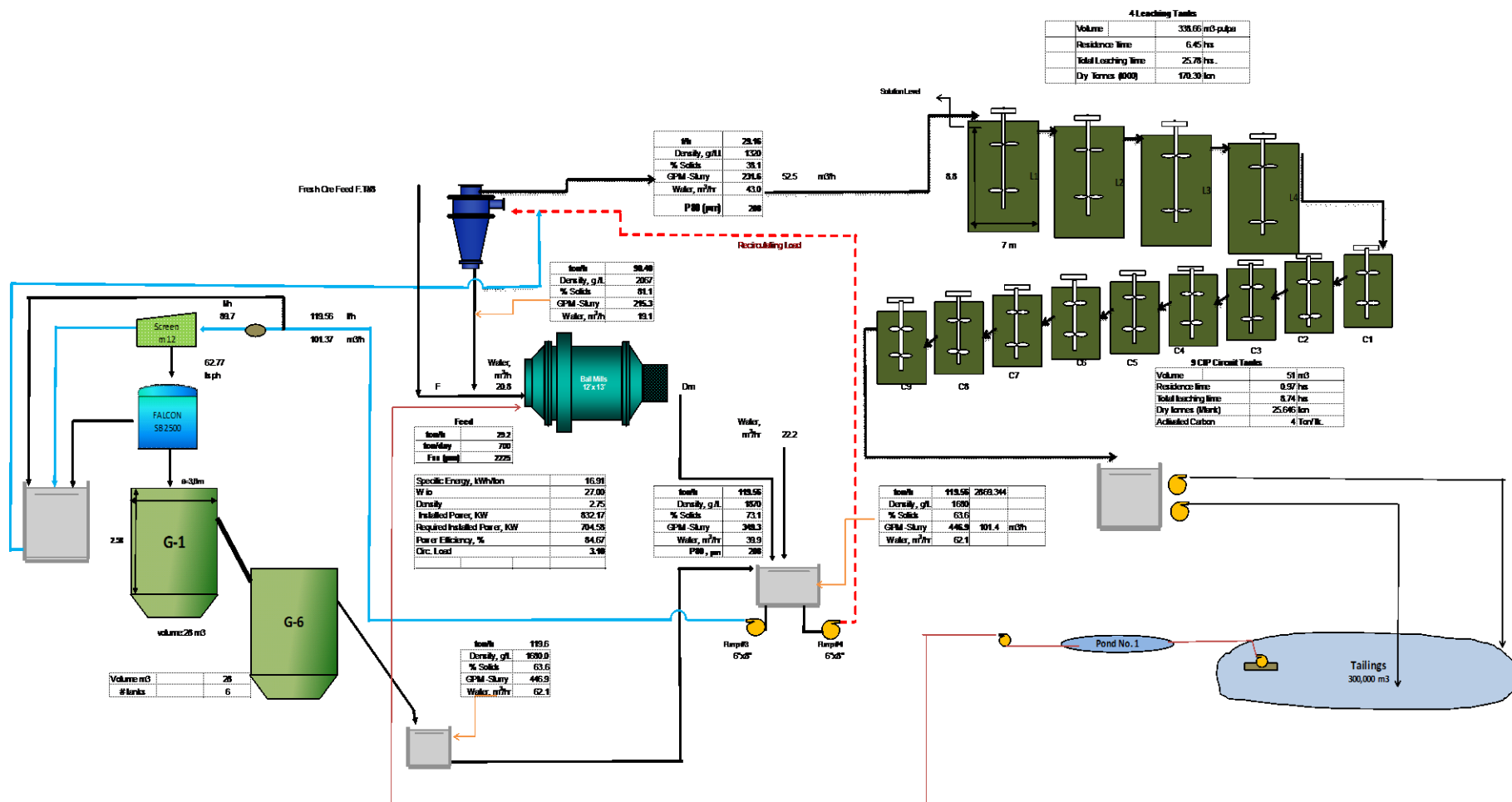
Phillip A Jones BAppSc (Geol), MAIG, MAusIMM

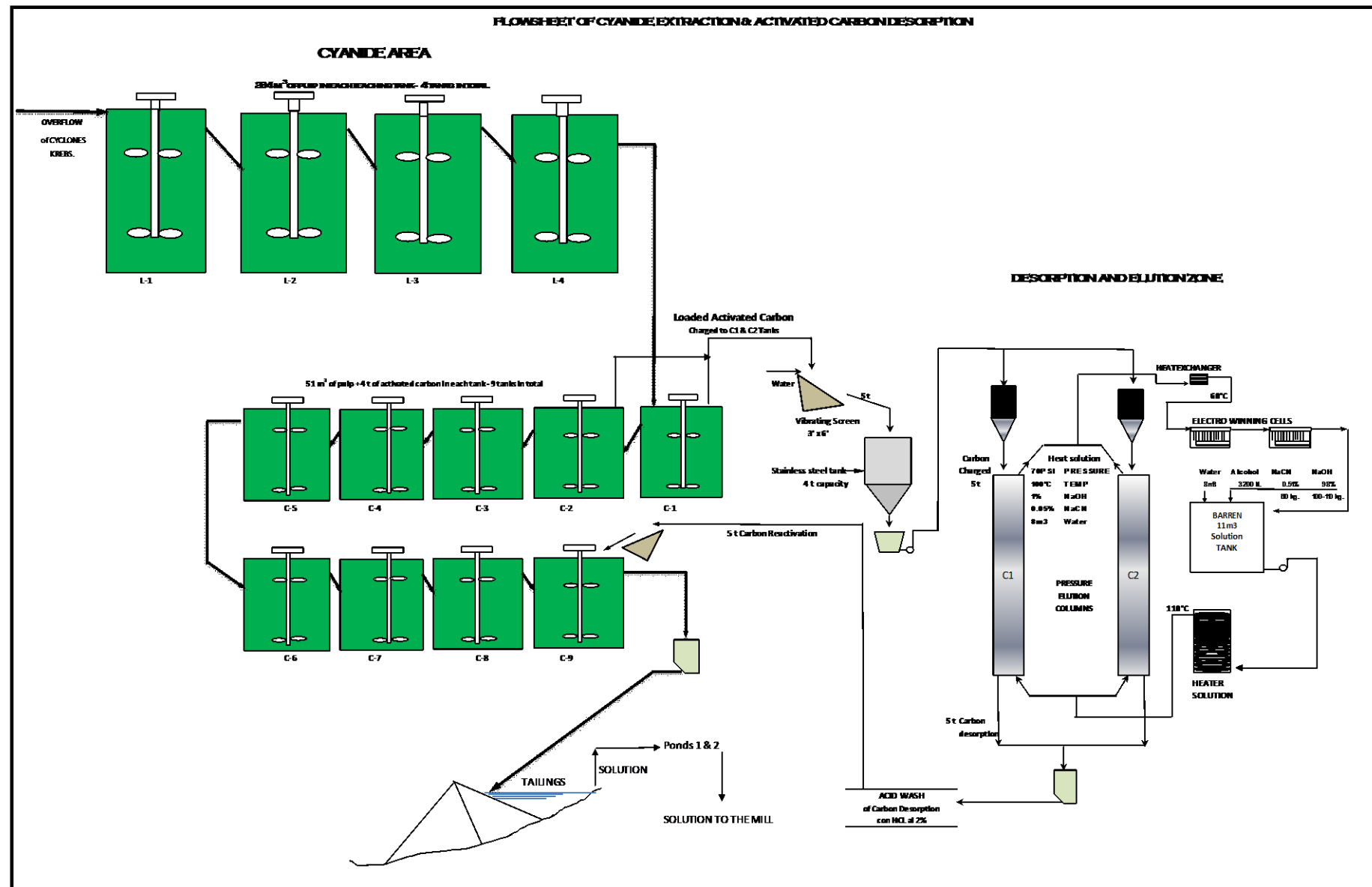
Appendices

Appendix 1 Processing Circuit Flowsheet Detail



MILLING & GRAVITY FLOWSHEET

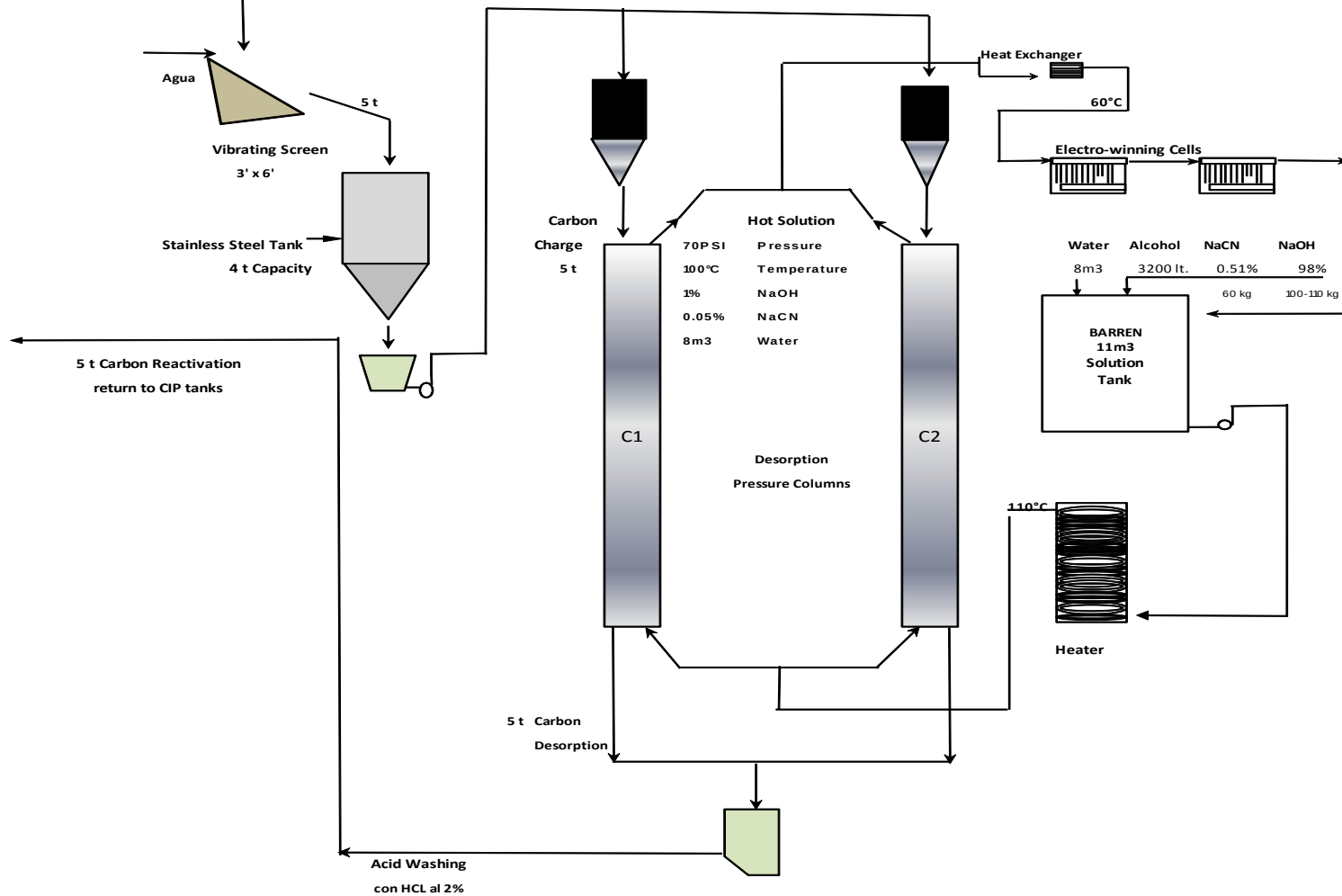




DESORPTION AREA FLOWSHEET

Recovery from Activated Carbon

Loaded Carbon from Tanks 1 & 2



Qualified Person Certificates

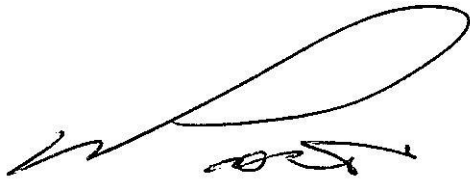
Richard L. Procter

As a qualified person responsible for sections of the report titled "Independent Preliminary Assessment - Zaruma Gold Project, El Oro Province – Ecuador" dated 17 September 2014 (the "Technical Report") to which this certificate applies, I, Richard L Procter, CEng, MIMMM, BSc (Eng), MBA, do hereby certify that:

1. I am a Consulting Engineer, with an address at Suite 1, 64 Thomas Street, West Perth, Western Australia, Australia 6005.
2. This certificate applies to the Technical Report.
3. I am a graduate of the University of the Witwatersrand, Johannesburg, South Africa (BSc, Mining Engineering) and Cape Town University (MBA), South Africa.
4. I am a Chartered Engineer and a Professional Member of the Institute of Materials, Minerals and Mining, United Kingdom.
5. I have worked as a mining engineer internationally for over 35 years in the fields of mining consulting, mine development, operations, management and general management (operational and corporate) and finance. I have had experience with mining assets and projects in Australia, New Zealand, South Africa, Zimbabwe, Zambia, Mozambique, Ghana, Botswana, Mali, Brazil, Chile, Peru, England, Ireland, France, Portugal, Austria, Germany, Saudi Arabia, Malaysia, Philippines, Indonesia, China, Mongolia, Canada, Russia and Kazakhstan. My mineral commodity exposure covers principally gold, silver, platinum, copper, lead, zinc, nickel, coal, vanadium, kaolin, uranium, diamonds, iron ore, chromite, potash, tungsten, molybdenum, asbestos and mineral sands.
6. 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 fulfil the requirements to be a "Qualified Person" for the purposes of NI 43-101.
7. I am responsible for the preparation of the Executive Summary and sections 14, 15, 16, 17, 18, 19, 20, 21 and 22 of the Technical Report. I visited the property from 22 February to 25 February 2014.
8. I have had no prior involvement with the property that is the subject of the Technical Report.
9. As of the date of the Technical Report, to the best of my knowledge, information and belief, the Executive Summary and sections 14, 15, 16, 17, 18, 19, 20, 21 and 22 of the Technical Report contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
10. I am independent of Dynasty applying all of the tests in section 1.5 of NI 43-101.

11. I have read NI 43-101 and Form 43-101F1, and the Executive Summary and sections 14, 15, 16, 17, 18, 19, 20, 21 and 22 of the Technical Report have been prepared in compliance with that instrument and form.
12. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible.

Dated this 17th day of September 2014.

A handwritten signature in black ink, appearing to read 'R. Procter', with a large, sweeping loop at the end.

Richard L Procter

Allen J. Maynard

As a qualified person responsible for some sections of the report titled "Independent Preliminary Assessment Zaruma Gold Project, El Oro Province – Ecuador" dated 17th September, 2014 (the "Technical Report") to which this certificate applies, I, Allen J. Maynard, BAppSc (Geol), MAIG, MAusIMM, do hereby certify that:

1. I am a consulting geologist, conducting business under the registered name of Al Maynard & Associates Pty Ltd (ABN 95 336 331 535). My address is Suite 9, 280 Hay Street, Subiaco, WA, 6008 Australia.
2. I graduated with a Bachelor of Applied Science degree in Geology from WA Institute of Technology (Curtin University) in Western Australia in 1978.
3. I am a registered member in good standing of the AIG and a Corporate Member of the AusIMM, and I became a member of AIG in 1990 and AusIMM in 1978.
4. I have worked as a geologist in the field of mineral exploration and development continuously for the last 35 years, with experience in mineral exploration/resource modelling and underground mining for a range of commodities including precious metals (Au, PGM); precious and semi-precious gemstones (diamond, ruby, emerald) and base metals (Cu, Ag, Pb, Zn, Fe, Mo, Mn, Sb, Sn, W, U, V); industrial minerals (coal, phosphate, potash, mineral sands) and REE minerals. I have also had experience with project generation and evaluation as well as independent technical valuation of mineral properties in Australia in eight African countries, five South American countries, central USA, Central and southeastern Asia, China, Armenia, Greenland, Portugal, Serbia, Turkey, UK, Wales and Ireland.
5. I have read the definition of "qualified person" set out in National Instrument 43-101 – *Standards of Disclosure for Mineral Projects* ("NI 43-101") and confirm that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a "qualified person" for the purposes of NI 43-101. My work experience includes the management and performance of numerous technical studies relating to mineral exploration and surface and underground mining, audit, evaluation and valuation of projects and operating mines in many parts of the world.
6. I inspected the Zaruma Gold Project during a site visit from Mar. 19, 2013 to and including Mar. 22, 2013, for a total duration of 4 days. Previously, I inspected the Project in March 2005, January 2008 and January 2010, as well as several site visits prior to 2005.
7. I am responsible for the Executive Summary and sections 1, 2, 3, 4, 5, 6, 7, 8, 9, 10, 11, 12, 13, 23, 24, 25 and 26 of the Technical Report.
8. I am independent of Dynasty applying all of the tests as described in section 1.5 of NI 43-101.

9. I have had a prior involvement with the Zaruma Gold Project as described within the report, in particular, by earlier site visits as described at Item 6 above, as well as being the author of the 2005 Report and co-author of the 2006 Report, each as defined in the Technical Report.
10. I have read NI 43-101 and the Technical Report has been prepared in compliance with NI 43-101.
11. As of the date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
12. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible.

Dated this 17th day of September 2014.



Allen J. Maynard

Philip A. Jones

As a qualified person responsible for some sections of the report titled "Independent Preliminary Assessment Zaruma Gold Project, El Oro Province – Ecuador" dated 17th September, 2014 (the "Technical Report") to which this certificate applies, I, Philip A. Jones, BAppSc (AppGeol), MAIG, MAusIMM, do hereby certify that:

1. I am a consulting geologist to Al Maynard & Associates (ABN 95 336 331 535) whose address is Suite 9, 280 Hay Street, Subiaco, WA, 6008 Australia.
2. I graduated with a Bachelor of Applied Science degree in Applied Geology from SA Institute of Technology in South Australia in 1974.
3. I am a registered member in good standing of the Australian Institute of Geoscientists (AIG) and a Member of the Australasian Institute of Mining & Metallurgy (AusIMM), and I become a member of AIG in 1985 and AusIMM in 1983.
4. I have over 30 years continuous experience as a geologist in the fields of mineral exploration, resource modelling and surface and underground mining for a range of commodities including precious and base metals (Au, Ni, Cu, Ag-Pb-Zn, Fe, Sn, Ta, Nb, W, U), industrial minerals (phosphate, silica, coal, mineral sands), project evaluation, as well as technical valuation of mineral properties in Australia, Africa, South America, Central & Southeast Asia, China and Greenland.
5. I have read the definition of "qualified person" set out in National Instrument 43-101 – *Standards of Disclosure for Mineral Projects* ("NI 43-101") and confirm that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a "qualified person" for the purposes of NI 43-101. My work experience includes the performance of numerous technical studies relating to mineral exploration and surface and underground mining, audit, evaluation and valuation of projects and operating mines in many parts of the world.
6. I have not carried out a site inspection of the Zaruma Gold Project.
7. I am responsible for the Executive Summary and section 13 of the Technical Report.
8. I am independent of Dynasty applying all of the tests as described in section 1.5 of NI 43-101.
9. I have had no prior involvement with the Zaruma Gold Project.
10. I have read NI 43-101 and the Technical Report has been prepared in compliance with NI 43-101.
11. As of the date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

12. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible.

Dated this 17th day of September, 2014.



Philip A. Jones