NI 43-101 Technical Report

Preliminary Economic Assessment of the Santa Barbara Gold and Copper Project in Zamora, Ecuador

Prepared For:

Ecuador Gold and Copper Corp.

Qualified Persons:
Michael Short, B.E., FIMMM, C.Eng
Allen J. Maynard, BAppSC(Geolg), MAIG, MAusIMM
Phillip A. Jones, BAppSC(Geolg), MAIG, MAusIMM

Effective Date: May 19, 2015

Date of Signing: May 29, 2015

Prepared and compiled by:

GBM Minerals Engineering Consultants Limited

GBM Project Number: 0503

Other contributors:
Al Maynard and Associates Pty Ltd., Geological Consultants
Mine Development Associates, Geological and Engineering Consultants
Phillips Enterprises LLC, Metallurgical Laboratories
Resource Development Inc., Diversified Mining Consulting Services
Wardell Armstrong International Limited, Environmental and Engineering Consultancy

Technical Report under NI 43-101

This report was prepared as a National Instrument 43-101 (NI 43-101) Technical Report for Ecuador Gold and Copper Corp. (Ecuador Gold) by GBM Minerals Engineering Consultants Limited (GBM). The quality of information, conclusions and estimates contained herein is consistent with the level of effort involved in GBM's services, based on: (i) information available at the time of preparation, (ii) data supplied by outside sources, and (iii) the assumptions, conditions, and qualifications set forth in this report. This report is intended for use by Ecuador Gold subject to the terms and conditions of its contract with GBM and relevant securities legislation. The contract permits Ecuador Gold to file this report as a Technical Report with Canadian securities regulatory authorities pursuant to NI 43-101—Standards of Disclosure for Mineral Projects. Except for the purposes legislated under applicable securities law, any other uses of this report by any other third party is at that party's sole risk. The responsibility for this disclosure remains with Ecuador Gold. The user of this document should ensure that this is the most recent Technical Report for the property as it is not valid if a new Technical Report has been issued.

Important Notice

This preliminary economic assessment (PEA) report is intended to provide an initial review of the potential for Ecuador Gold's Santa Barbara Gold and Copper Project and is preliminary in nature. The PEA 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, and there is no certainty that the PEA based on these mineral resources will be realized. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

TABLE OF CONTENTS

LIST OF TABLES

LIST OF FIGURES

STANDARD TERMS AND NOMENCLATURE

The following terms and definitions are used throughout this report:

Term	Definition
Condormining	Condormining Corporation S.A.
Effective Date	The cut-off date for the scientific and technical information included in the PEA, as stated on the cover page.
FJTX	Corporacion FJTX Exploration S.A
Inferred Mineral Resources	An Inferred Mineral Resource is that part of a Mineral Resource for which quantity and grade or quality can be estimated on the basis of geological evidence and limited sampling and reasonably assumed, but not verified, geological and grade continuity. The estimate is based on limited information and sampling gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes.
Indicated Mineral Resources	An Indicated Mineral Resource is that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics, can be estimated with a level of confidence sufficient to allow the appropriate application of technical and economic parameters, to support mine planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes that are spaced closely enough for geological and grade continuity to be reasonably assumed.
Material Planned for Processing (MPP)	Means the potential material planned for processing from the Mineral Resources at the Project that would, if economically viable, potentially be mined from the Deposit in raw mineral form for further processing to extract any valuable contained metals present in the mineralized material.
Measured Mineral Resources	A Measured Mineral Resource is that part of a Mineral Resource for which quantity, grade or quality, densities, shape, and physical characteristics are so well established that they can be estimated with confidence sufficient to allow the appropriate application of technical and economic parameters, to support production planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration, sampling and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes that are spaced closely enough to confirm both geological and grade continuity.
Mineral Reserves	A Mineral Reserve is the economically mineable part of a Measured or Indicated Mineral Resource demonstrated by at least a Preliminary Feasibility Study. This Study must include adequate information on mining, processing, metallurgical, economic, and other relevant factors that demonstrate, at the time of reporting, that economic extraction can be justified. A Mineral Reserve includes diluting materials and allowances for losses that may occur when the material is mined.

Term	Definition
Mineral Resources	A Mineral Resource is a concentration or occurrence of diamonds, natural solid inorganic material, or natural solid fossilised organic material including base or precious metals, coal and industrial minerals in or on the Earth's crust in such form and quantity and of such a grade or quality that it has reasonable prospects for economic extraction. The location, quantity, grade, geological characteristics and continuity of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge.
Preliminary Economic Assessment (PEA)	Means a study, other than a Pre-Feasibility Study or Feasibility Study that includes economic analysis of the potential viability of mineral resources. The PEA is intended to provide an initial review of the potential for Ecuador Gold's Santa Barbara Gold and Copper Project and is preliminary in nature. The PEA 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, and there is no certainty that the PEA based on these mineral resources will be realized. Mineral resources that are not mineral reserves do not have demonstrated economic viability.
Probable Mineral Reserve	A Probable Mineral Reserve is the economically mineable part of an Indicated and, in some circumstances, a Measured Mineral Resource demonstrated by at least a Preliminary Feasibility Study. This Study must include adequate information on mining, processing, metallurgical, economic, and other relevant factors that demonstrate, at the time of reporting, that economic extraction can be justified.
Proven Mineral Reserve	A Proven Mineral Reserve is the economically mineable part of a Measured Mineral Resource demonstrated by at least a Preliminary Feasibility Study. This Study must include adequate information on mining, processing, metallurgical, economic, and other relevant factors that demonstrate, at the time of reporting, that economic extraction is justified.
Sulfur, Sulphate, Sulphite	Note British and international scientific community adopted spellings.
The Project	All planning, commercial, statutory, design, and construction activities associated with mining of the Santa Barbara Deposit.
The Condor/Northern Sector	Chinapintza, Los Cuyes, San Jose, Enma and Soledad deposits.
The Deposit	Potentially economic material associated with the Santa Barbara Deposit, which is made up of the Santa Barbara South and Santa Barbara North deposits.
The Southern Sector	Santa Barbara South, Santa Barbara North and El Hito deposits.

The following abbreviations are used throughout this report:

Acronym	Definition
AACE	AACE International (previously known as American Association of Cost Engineering and Association for the Advancement of Cost Engineering).
AAS	Atomic Absorption Spectrometers
AMA	Al Maynard & Associates Limited
amsl	Above mean sea level
CAPEX	Capital Expenditure
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
CIP	Carbon-in-Pulp
DINE	La Direccion de Industrias del Ejercito
ECC	Ecuador Capital Corporation
EGI	Ecuador Gold and Copper Inc.
EGX	Ecuador Gold and Copper Corporation
EIA	Environmental Impact Assessment
EMP	Environmental Management Plan
EPC	Engineering Procurement Construction
EPCM	Engineering Procurement Construction Management
ESIA	Environmental and Social Impact Assessment
FS	Feasibility Study
GBM	GBM Minerals Engineering Consultants Limited
ICP-ES	Inductively Coupled Plasma Emission Spectroscopy
IRR	Internal Rate of Return
IP	Induced Polarisation
LOM	Life of Mine
MDA	Mine Development Associates
n / a	Not applicable
MPP	Material Planned for Processing
NPV	Net present value
OPEX	Operating Expenditure
OREAS	Ore Research & Exploration Pty Ltd (ORE) Assay Standards
PEA	Preliminary Economic Assessment
RDi	Resource Development Inc.
Phillips	Phillips Enterprises LLC, metallurgical laboratories
ROM	Run of Mine
SG	Specific Gravity
TMF	Tailings Management Facility
USD	United States Dollar
VAT	Value Added Tax
WAI	Wardell Armstrong International Limited

SECTION 1 SUMMARY

1.1 GENERAL

Ecuador Gold and Copper Corp. (TSX-V:EGX) (the Company or simply, Ecuador Gold) is a British Columbia corporation that owns mineral assets in Ecuador through its wholly-owned subsidiaries, Ecuador Capital Corporation (ECC) a British Columbia corporation and Ecuador Gold and Copper Inc. (EGI) an Ontario corporation, which has a 90% ownership interest in, Condormining Corporation S.A. (Condormining) and a 99% ownership effecting full ownership and control of Corporacion FJTX Exploration SE (FJTX). Condormining and FJTX are the concession holders of a number of mineral properties including the Santa Barbara North and Santa Barbara South deposits (the Deposit) which comprise the Project. The Deposit considered in this report is the Santa Barbara Deposit which is located in south east Ecuador approximately 400 km south south-east of Quito and 40 km east of the nearby town of Zamora. This Preliminary Economic Assessment (PEA) presents the findings of a PEA of the Santa Barbara Gold and Copper Project (the Project). The Project proposes to exploit copper, gold and silver from the Deposit via open pit mining methods and to process the Material Planned for Processing (MPP) via a flotation process to recover a copper concentrate with gold credits followed by a carbon in pulp process, elution and electrowinning circuit to produce gold and silver doré from the flotation tailings.

1.2 GEOLOGY, MINERALISATION AND EXPLORATION

The Project is located within the Condor Cordillera, between the Andean Cordillera Real in the west and the Pre-Cambrian Amazon Craton. It forms part of a significant Jurassic-Cretaceous back-arc fold-thrust belt with Jurassic granitoid plutons and younger supracrustal sequences consisting of Palaeozoic and Mesozoic sediments and arc-related igneous-volcanic lithologies. This district, including the Project is known as the Zamora Copper-Gold Belt and is related to Late Jurassic magmatism (Drobe, 2013). The most extensive unit exposed in the Santa Barbara Project is a fine-grained green basaltic andesitic volcanic rock which has been assigned to the Upper Jurassic Misahualli formation. Overlying the volcanics is a sedimentary sequence comprising conglomerate, quartz sandstone, limestone and locally garnet skarn.

The main host for the gold-copper mineralisation at the Santa Barbara Project is the basaltic andesite volcanic unit. Two separate mineralised zones have been defined in the Project. Santa Barbara South is the main mineral body defined thus far. It is elongated in a north-northwest direction following the trend of interpreted faulting. As currently defined, the mineralized body extends at least 700 m, and is approximately 300 m wide. The contact is steep on the west side of the deposit and the zone dips consistently 40-50° to the east.

Exploration activities, in addition to the Phase I and Phase II drilling programs (under which 18 112 m were drilled) and the Phase III drilling program (3 939 m in seven drill holes at Santa Barbara), include mapping and sampling at Santa Barbara by Condormining geologists. Minor chip sampling and mapping was also completed at Santa Barbara North. Surface mapping and sampling are presently continuing in the El Hito and Santa Barbara areas to identify satellite deposits as potential feedstock to the Project.

1.3 MINERAL RESOURCE AND MINERAL RESERVE ESTIMATES

A Mineral Resource statement was completed based upon 36 bore holes totalling over 22 000 m that is summarised below in Table 1-1. The resource estimation work was completed by Philip A. Jones BAppSC(Geolg), MAIG, MAusIMM of Al Maynard and Associates Pty Ltd. (AMA), an appropriate independent qualified person as defined by NI 43-101 (AMA, 2014a). The effective date of that resource statement is March 24, 2014.

Table 1-1 Mineral Resource Statement (AMA, 2014a)

Resource Classification	MPP (Mt)	Grade			Au (oz.)	Ag (oz.)	Cu (lbs)
		Au (g/t)	Ag (g/t)	Cu (%)			
Indicated	364.572	0.51	0.9	0.1	5 978 000	10 080 000	0.8 billion
Inferred	177.601	0.4	0.8	0.1	2 300 000	4 625 000	0.4 billion

The Mineral Resource models were used as the basis to develop an estimated quantification of Material Planned for Processing that has the potential to be economically viable but has not been determined to be economically viable and there is no assurance that it will be economically viable. A summary of the Material Planned for Processing is shown in Table 1-2. Indicated and inferred resources were used in the LOM plan. Mineral resources that are not mineral reserves do not have demonstrated economic viability. There is no certainty that all or any part of the mineral resources would be converted into mineral reserves. Mineral reserves can only be estimated as a result of an economic evaluation as part of a pre-feasibility study or a feasibility study of a mineral project. Accordingly, at the present level of development there are no mineral reserves at the project.

Table 1-2: Summary of Material Planned for Processing

MPP (Mt)	Gold Grade (g/t)	Silver Grade (g/t)	Copper Grade (%)	Contained Gold (oz.)	Contained Silver(oz.)	Contained Copper (lbs)
98.807	0.72	0.96	0.11	2 272 000	3 036 000	233 018 000

1.4 CONCLUSIONS

1.4.1 MINING METHODS

This PEA examines the exploitation of the MPP over a 10 year life of mine, predominantly by conventional truck and shovel operations for the open pits. The open pit operations were determined to have a strip ratio of 1:0.65 for the proposed production schedule. The proposed method of MPP transit from the mine to the processing facility is transport by truck haulage.

1.4.2 RECOVERY METHODS

The proposed method of recovery is based on the developed production schedule of the Material Planned for Processing. The proposed method includes crushing and milling of the MPP prior to going through flotation. The flotation rougher concentrate would be reground and cleaned in a cleaner circuit to produce a copper concentrate with gold credits.

The flotation tailings and flotation cleaner tailings would be sent to a carbon-in-pulp leach circuit to recover gold and silver. The loaded carbon would then pass through an elution and acid wash prior to the gold and silver being recovered by electrowinning and smelted to produce doré. Interpretation of metallurgical testwork results determined a processing recovery rate of 80 % for copper, 86.7% for gold and 20% for silver, resulting in a life of mine production of 186 414 400 lb of copper, 1 971 500 oz. gold and 607 700 oz. silver.

1.4.3 FINANCIAL ANALYSIS

Capital cost and operating cost estimates, based on the proposed mining and recovery methods, were prepared to the level of a PEA, with an accuracy of -30% to +50%. A base economic model was prepared using these capital cost and operating cost estimates as presented in Section 21, the model parameters outlined in Section 22, and the production schedule outlined in Section 16. A summary of the major financial parameters and results are as shown in Table 1-3. It should be noted that these results do not demonstrate economic viability as the mine production schedule includes Inferred Mineral Resources and Indicated Mineral Resources, NOT Mineral Reserves. The results should be considered only as potential results that could be achieved if the Inferred Mineral Resources were able to be converted to Indicated or Measured Mineral Resources in the future and if further economic analysis were done to the level of a pre-feasibility study or feasibility study. Mineral resources that are not mineral reserves do not have demonstrated economic viability. There is no certainty that all or any part of the mineral resources would be converted into mineral reserves.

Table 1-3: Financial Model Parameters and Results – Base Case

Item	Unit	Value
Parameters		
Copper Price	USD/lb	3.34
Gold Price	USD/oz.	1 448
Silver Price	USD/oz.	24.66
Average Cost of Production	USD/t ore	18.78
Start-up CAPEX	M USD	598.9
Sustaining CAPEX (Life of Mine)	M USD	826.0
Results		
NPV (8 % discount rate)	M USD	47.4
IRR	%	9.5
Payback Period	annum	6.5

1.5 RECOMMENDATIONS

Based on the results of the PEA economic analysis, it is recommended that the project development should be pursued further as a pre-feasibility study, with additional drilling to upgrade the resource estimates and test work required on new drilled samples for confirmation of the copper, gold and silver recovery and process parameters for a flotation treatment with cyanidation of the tailings stream. In particular, it is recommended further investigation is undertaken:

- To upgrade the mineral resource from the Inferred category to allow determination of Mineral Reserves under a PFS to determine economic viability;
- To confirm copper, gold and silver recovery and process parameters through additional metallurgical test work; and
- To further investigate opportunities in the PFS economic analysis to improve/reduce the cost of power by investigating in more detail the new hydroelectric projects under development in the surrounding area of the Santa Barbara Project. This would include strategies and opportunities for securing a more economic cost of power to compliment the Project economics, by investing or leveraging of power infrastructure and power supply that is available from such new hydroelectric projects.

The estimated cost for the 12 month recommended work program amounts to USD 1 860 500, comprised of USD 1 160 500 for the first phase of drilling to upgrade the resource estimate and USD 700 000 for additional metallurgical test work and pre-feasibility economic analysis.

Table 1-4 below demonstrates from current economic modelling, the positive returns based on the three year trailing prices for gold, silver and copper and the positive economic returns likely to be achieved.

Table 1-4: Financial Model Sensitivity to gold price

Gold price (USD/oz.)	% of three year trailing average	IRR (%)	NPV (M USD)	Payback (years)
1 303	90	5.5	-74.4	7.4
1 448	**100**	**9.5**	**47.4**	**6.5**
1 593	110	13.4	168.9	5.6
1 738	120	17.0	290.2	4.5
1 882	130	20.5	410.7	3.0
2,027	140	23.9	532.1	2.1
2,172	150	27.2	653.2	1.7

SECTION 2 INTRODUCTION

2.1 GENERAL

This Preliminary Economic Assessment (PEA) presents the findings of a PEA of the Santa Barbara Gold and Copper Project (the Project), located in the Zamora-Chinchipe province, in south east Ecuador. The purpose of the Project is to develop the Santa Barbara Deposit (the Deposit) into a 10 million tonnes per annum open pit gold mine and produce gold and silver doré bars for export and sale to a refinery and copper concentrate with gold credits from flotation.

Ecuador Gold and Copper Corp. (TSX-V:EGX), owns mineral assets in Ecuador through EGX's wholly-owned subsidiaries, Ecuador Capital Corporation (ECC) and Ecuador Gold and Copper Inc. (EGI), which in turn holds a 90% ownership interest in Condormining Corporation S.A. (Condormining) and 99% ownership effecting full ownership and control of FJTX, both companies incorporated under the laws of Ecuador. Condormining and FJTX are the concession holders of a number of mineral properties including the Santa Barbara North and Santa Barbara South deposits which comprise the Project.

GBM Minerals Engineering Consultants Limited (GBM) is an independent firm of engineering consultants specialising in the development, design and construction of mining projects. GBM was commissioned by Ecuador Gold and Copper Corporation (EGX) to carry out the PEA. This included the development of the mine plan, process flowsheet, the processing facilities and infrastructure engineering design, the capital and operating cost estimates to an accuracy of -30% to +50%, and the compilation of this PEA report. The mine plan was developed by Mine Development Associates (MDA) of Reno, Nevada, test work programmes were undertaken by Phillips Enterprises LLC (Phillips) in Golden, Colorado and Resource Development Inc. (RDI) in Wheaton Ridge, Colorado. The first phase metallurgical test work by Phillips using 'whole ore' cyanide leaching and a two stage flotation process resulted in an 86% recovery of gold and a 64% recovery of copper. Additional test work undertaken by RDI incorporated alternative flotation reagents and fine grinding. As a result, the following metallurgical conclusions can be drawn:

- Copper and gold recoveries to flotation concentrate achievable by rougher flotation are ± 80% and ± 65%, respectively.

- Due to the very small particle size of some of the chalcopyrite, as seen in the mineralogy study, the recovery of copper is unlikely to increase without grinding the ore to an extremely fine size (i.e. ± 10 µm).

- Flotation at acidic pH conditions improved copper and gold recoveries in the rougher flotation.

- Sulfidization of the ore slightly improved copper and gold recoveries in the rougher flotation.

- The optimum process conditions for the ore are primary grind of p80 of 200 mesh and reagent suite consisting of potassium amyl xanthate (PAX), AP-208 and methylisobutal carbonal (MIBC).

Based on the metallurgical testwork to date, the Company believes that a flowsheet based on fine grinding (p80 of 200 mesh) followed by rougher flotation, concentrate regrind, cleaner and scavenger flotation to recover a copper concentrate with gold credits followed by cyanidation of the flotation tails will produce total process recoveries in the order of 80% copper, 80% gold and 20% silver.

Al Maynard and Associates Pty. Ltd. (AMA) is an independent firm of geologists who have provided geological analysis and produced the resource estimates.

Mine Development Associates (MDA) is an independent mining engineering firm that has completed the open pit design, produced the mining plan and provided capital and operating cost estimates for mining activities.

Wardell Armstrong International Limited (WAI) is a multidisciplinary engineering consultancy who have provided designs and cost estimates for the tailings management facility.

A Mineral Reserve estimate is outside the scope and consideration of this PEA as Inferred Mineral Resources form part of the resources upon which this PEA is based. Mineral resources that are not mineral reserves do not have demonstrated economic viability. There is no certainty that all or any part of the mineral resources of this project would be converted into mineral reserves. Mineral reserves can only be estimated as a result of an economic evaluation as part of a pre-feasibility study or a feasibility study of a mineral project's measured and indicated resources.

2.2 SOURCES OF INFORMATION

This report is based, in part, on data and reports supplied by Ecuador Gold, as well as published government reports and other public information. In particular, this report makes use of the following information sources:

- NI 43-101 Technical Report on the Condor Gold and Copper Project Located in Zamora, Ecuador, dated effective March 24, 2014 (Maynard et al, May 8, 2014); and

- Condor Copper-Gold Santa Barbara Deposit, Ecuador, Addendum to Tailings Dam Concept Design Report, ZS611272 REP-002 V.01 (WAI, May 2014).

GBM has used these and various other information sources which are referenced where applicable throughout this report and presented in Section 27. The information collected has been sufficient to allow the preparation of this PEA.

2.3 PERSONAL INSPECTION BY AUTHORS

Allen J. Maynard of AMA visited the site from 9 July to 11 July, 2010 and again on 21 and 22 March, 2013 to inspect the surface geology, evaluate QA/QC procedures on the property and confirm that the sampling procedures met CIM Code standards. Reference check samples were taken of selected core as detailed later in this report. Further discussions with Ecometals/Goldmarca personnel were held in Quito between 7 and 12 July, 2010 to discuss logistics as well as government permitting and concessions. An additional field trip was conducted by Mr. Maynard from 14 to 17 January, 2011 to re-visit the site and discuss the proposed exploration program with the site personnel.

Philip Jones of AMA made a site inspection field trip from 10 to 16 April, 2011 during which the site was visited to inspect the geology, evaluate QA/QC procedures and topography. Discussions were also held with the site personnel on core sampling, storage and security, resource modelling methods, CIM Code requirements, setting up a central database for the project and the proposed exploration program.

Michael Short from GBM visited Ecuador and the Project from 15 to 18 September, 2013 and more recently from 18 to 19 May, 2015. During his visit he met representatives of Condormining and EGX as well as undertaking a site visit. He discussed geology and mining and plans for proposed infrastructure and inspected the site and the main exploration camp. At the site, he inspected surface geology and topography conditions and at the camp, he discussed geography and topography conditions, inspected core and reviewed QA/QC procedures as well as the equipment used for sample preparation and testing.

Three members of the GBM study team, namely Sally Parker, Damian Hicks and Joel Lewis visited Ecuador and the Project from 9 to 17 October, 2013. They met with representatives from Condormining, government ministries and private contractors to the mining industry. An inspection of the Project and surrounding areas was also undertaken in conjunction with Condormining personnel.

2.4 FINANCIAL INTEREST DISCLAIMER

Neither GBM, AMA, nor any subconsultants employed by the report contributors have any beneficial interest in the assets of EGX, ECC, EGI, Condormining, FJTX and/or the Project.

All consultants have been and will continue to be paid fees for technical advice for this work in accordance with industry standard consulting practices.

SECTION 3 RELIANCE ON OTHER EXPERTS

The Qualified Persons have relied on all of the information and technical documents listed in the references (Section 27) and have assumed these are accurate and complete in all material aspects; however, none of the Qualified Persons can guarantee the accuracy and completeness of the relied upon information and technical reports. The Qualified Persons reserve the right to, but will not be obligated to, revise this report if additional information becomes available subsequent to its effective date.

The Qualified Persons have relied on expert opinions and information provided by EGX and Condormining and their legal counsel pertaining to environmental considerations, taxation matters and legal matters including mineral tenure, surface rights and material contracts. In particular, the Qualified Person from GBM has reviewed the taxation information provided and believes it to be correct and adequate for use in this PEA report (Tobar y Bustamante, 2015 and 2013). The Qualified Persons have relied on property ownership information provided by EGX, Condormining and a legal title opinion regarding Ecuador corporate ownership and structure provided by their legal counsel Tobar y Bustamante of Quito, Ecuador (Tobar y Bustamante, 2015), as well as Canadian corporate ownership information provided by Canadian legal counsel, Boughton Law Corporation of Vancouver, Canada (Boughton, 2015). The Qualified Persons have not researched the property title or mineral rights for the Project and expresses no legal opinion as to the ownership status of the property.

This information is believed to be essentially complete and correct to the best of the Qualified Person's knowledge and no information has been intentionally withheld that would affect the conclusions made herein.

SECTION 4 PROPERTY DESCRIPTION AND LOCATION

4.1 PROPERTY LOCATION & AREA

The Santa Barbara Project is located approximately 400 km south-southeast of the Ecuadorian capital, Quito, and approximately 40 km east of the town of Zamora. The Project lies within the Nangaritza Canton of the Zamora-Chinchipe Province, close to the Peruvian border. The approximate centroid of the Project is shown in Table 4-1. The location of the Project is shown in yellow on Figure 4-1.

Table 4-1: Santa Barbara Deposit Location

Latitude	Longitude	Northing[1]	Easting[1]
4° 7' 12" S	78° 36' 33" W	9 544 210 N	765 452 E

Note 1: UTM PSAD-56 Zone 17 S.

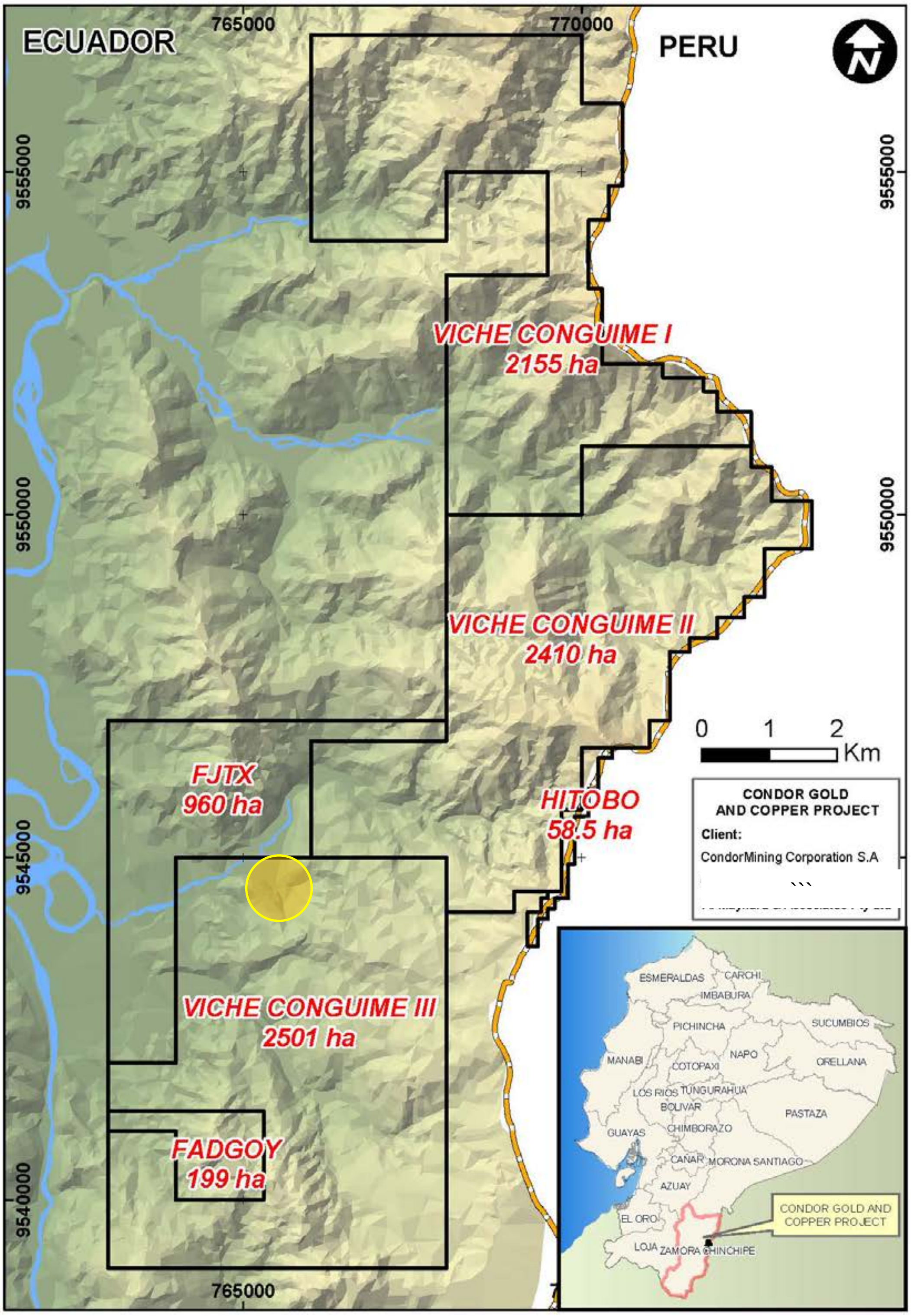

Figure 4-1: Concessions.
The approximate location of the Santa Barbara South Deposit is shown in Yellow (4).

4.2 MINERAL TENURE

The Deposit lies within the Viche Conguime II, Viche Conguime III and FJTX concessions, the details of which are presented in Table 4-2.

Table 4-2: Santa Barbara Project Concessions

Concession name	Concession code	Owner	Province	Canton	Area (Ha)	Title registration date	Expiry
Viche Conguime II	2024A	Condormining Corporation S.A	Zamora Chinchipe	Paquisha and Nangaritza	2410	21/05/2010	1/9/2021
Viche Conguime III	500802	Condormining Corporation S.A	Zamora Chinchipe	Nangaritza	2501	20/05/2010	2/4/2033
FJTX	500135	FJTX Exploration S.A.	Zamora Chinchipe	Nangaritza	960	25/05/2010	11/10/2031

Concession boundary (corner) co-ordinates are presented shown in Table 4-3.

Table 4-3: Project Concession Coordinates

Point	Coordinates		Point	Coordinates	
Viche Conguime II			Viche Conguime III		
	East	North		East	North
PP	768,000	9,550,000	PP	763,000	9,542,000
1	770,000	9,550,000	1	764,000	9,545,000
2	770,000	9,551,000	2	764,000	9,545,000
3	772,500	9,551,000	3	768,000	9,541,000
4	772,500	9,550,700	4	768,000	9,539,000
5	772,800	9,550,700	5	763,000	9,539,000
6	772,800	9,550,200	6	763,000	9,541,000
7	773,400	9,550,200	7	764,000	9,541,000
8	773,400	9,549,500	8	764,000	9,540,000
9	772,700	9,549,500	9	765,300	9,540,000
10	772,700	9,548,800	10	765,300	9,541,300

Point	Coordinates		Point	Coordinates	
Viche Conguime II			Viche Conguime III		
	East	North		East	North
11	772,400	9,548,800	11	763,000	9,541,300
12	772,400	9,548,500			
13	772,000	9,548,500	FJTX		
14	772,000	9,548,200	PP	763,000	9,547,000
15	771,600	9,548,200	1	768,000	9,547,000
16	771,600	9,548,000	2	768,000	9,546,700
17	771,300	9,548,000	3	766,000	9,546,700
18	771,300	9,547,000	4	766,000	9,545,000
19	771,000	9,547,000	5	764,000	9,545,000
20	771,000	9,546,600	6	764,000	9,542,000
21	770,000	9,546,600	7	763,000	9,542,000
22	770,000	9,545,700			
23	769,700	9,545,700			
24	769,700	9,544,500			
25	769,000	9,544,500			
26	769,000	9,544,200			
27	768,000	9,544,200			
28	768,000	9,545,000			
29	766,000	9,545,000			
30	766,000	9,546,700			
31	768,000	9,546,700			

4.3 PROPERTY, TITLE AND SURFACE RIGHTS

The Viche Conguime II and Viche Conguime III concessions are wholly owned by Condormining, a 90% owned subsidiary of EGI. Through its 99% owned subsidiary, Corporacion FJTX Exploration SA (FJTX), of which it effects full ownership and control, EGI effectively owns all interests in the FJTX concession adjacent to Viche Conguime III.

Condormining, through a number of property acquisitions, owns the majority of the surface rights corresponding to the Project. The Project area is within an unsurveyed, agricultural region, with land claims registered at INDA (Instituto Nacional de Desarollo Agrario).

All mineral concession boundaries are submitted by applicants to the relevant Ecuadorian government agency in standard documents describing the mining authorities with boundary corners specified in UTM grid coordinates. No physical pegging of boundaries is required. Exploration concessions are valid in their entirety for 10 years provided that annual reports are submitted on time, expenditure commitments are met and annual rents are paid. Once a production decision has been made, exploration concessions can be converted by application to exploitation concessions for a further period of 10 years.

The Project currently assumes additional land acquisition and surface rights will be obtained in the future, in addition to the existing surface right holdings, to accommodate proposed infrastructure such as access roads and conveyors as well as the area identified for the processing and mining facilities.

Based on information provided by the legal counsel of EGX, there are no known outstanding legal mineral claims on the Project concessions.

Owners of mining concessions must pay an annual conservation patent fee for each hectare of their concessions. Currently, the concessions fall under the advanced exploration regime which attracts a cost of USD 15.90 per hectare. Reports describing in detail all exploration work and expenditures carried out by the concession owners are required to be submitted annually to the Ecuadorian government by a set anniversary date. Copies of the 2013 reports submitted for each mining concession were reviewed by AMA. Condormining has confirmed that all the necessary reports were submitted on time and that there are no outstanding reports required.

Rental fees on the mining concessions are up to date as of the effective date of this PEA report.

Costs for holding and maintaining the licences are shown in Table 4-4.

Table 4-4: Tenement information

Concession	Annual Concession Rent (USD)
Viche Conguime II	38 319.00
Viche Conguime III	39 765.90
FJTX	30 528.00
Total	108 612.90

4.4 ROYALTIES

All mine production is subject to royalty payments to the Ecuadorian government. The relevant royalties are as follows:

- Gold and silver; minimum of 5 % and maximum of 8 % gross value of bullion produced.
- Base metals including copper, lead, zinc; minimum of 5 % gross value of metal produced.

As no minerals are currently being produced, no royalties are applicable.

There are no other royalties required to be paid on mine production or exploration rights. .

4.5 ENVIRONMENTAL LIABILITIES

Exploration activities are currently occurring under Environmental Licence No. 267 issued by the Ministry of the Environment on 24 April 2013.

A water extraction licence is required for the Project from SENAGUA, the national agency for management of water resources within Ecuador. To date, no applications have been filed for this licence.

In regards to the key permit and licence areas the following currently applies:

- EIA 2006 was submitted and approved in 2007 for the advanced mining exploration phase and a new EIA would be required for the next phase of the Project.
- Application for Water Permit Extraction Licence to extract water for use from the river, will have to be lodged with the Ministry of Strategic Solutions, for the quantities of water required for mining and operations, and it was confirmed by the authority in December 2013 that a water licence would cost approximately USD 9 000.

4.6 PERMITS

A drilling permit from the Ecuadorian Government was required before drilling could begin on the concessions and this was granted effective 1 July 2011. The permit is effective for the life of the concessions. All of the necessary environmental, and health and safety permits have also been granted for exploration activities on the concessions.

An Annual Environmental Audit Report must be submitted to the Ecuadorian government for approval before exploration in subsequent years can proceed. The environmental impact of the proposed exploration work and all remedial work that have been performed is reported. The 2013 Annual Environmental Audit Report has been submitted by Condormining while the 2014 report is being prepared for submission. The current environmental licence, Resolution Number 267, was approved on 14 April 2013 by the Minister of Environment, Lorena Tapia Núñez.

4.7 SIGNIFICANT FACTORS AND RISKS AFFECTING ACCESS, TITLE, AND ABILITY TO WORK ON THE PROPERTY

Climatic factors and ground access (described in Section 5) may affect physical access to the Project. Other than as stated above, the authors are not aware of other significant risks or factors that may affect access, title, or the right to perform work on the Project.

SECTION 5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 TOPOGRAPHY, ELEVATION AND VEGETATION

The Project is located on the western flanks of the Condor Mountain Range, the crest of which defines the Ecuador-Peru border. Elevations within the project area range from approximately 850 m amsl adjacent to the Nangaritza River to 1 750 m amsl at the Ecuador-Peru Border. The topography is very rugged and slopes are steep. The mountains are covered with tropical rain forests and dense vegetation.

The Nangaritza River flows approximately 1 500 m west of the Deposit. The floodplain varies from approximately 100 to 2 000 m in width in the vicinity of the Project site and is characterised by flat areas with moderately dense to sparse forest. Two tributaries of the Nangaritza River flow to the north (Pachikutza River) and south (Yapi River) of the Project site. There are many smaller streams and waterfalls located in the vicinity of the Deposit.

5.2 PROJECT ACCESS

Access is along paved highways except for the last 48 km, which is a dirt road through the towns of Zumbi, Paquisha, Nuevo Quito, Conguime and Pachikutza. Depending on road conditions, travel time from Catamayo airport near Loja to the property is between three and four hours. Catamayo to Zamora is approximately 92 km, and Zamora to the property is a further 83 km.

The closest commercial airport is located in Catamayo (Camilo Ponce Enríquez Airport) near Loja. Military airports are located at Zamora and Gualaquiza.

The Project site and surrounding area can be accessed by 4-wheel drive vehicles, through a number of variably maintained secondary dirt roads. These roads are subject to seasonal modification by slope instability and rainfall, which may prevent access to parts of the concessions from time to time, particularly in the rainy season between January and May.

Much of the primary road route to the Project is subject to landslides, in particular between Zamora and the Project. Slope instability also affects the low side of the roads with wash outs and damage to culverts and/or the road above, being common. Access is usually blocked for approximately 24 hours in the instance of a landslide event occurring.

5.3 PROXIMITY OF POPULATION

The largest regional centre close to the Project is Loja, with a population of approximately 170 000. Loja is an education centre and provides unskilled to skilled labour and basic equipment and supplies. Zamora, with a population of 15 000 also provides unskilled and skilled labour.

Two small villages are located in the immediate vicinity of the Project site: Pachikutza and Los Geranios, each with a population of several hundred people. The inhabitants of Pachikutza are predominantly Shuar; the predominant indigenous group found in Ecuador. Los Geranios is the closest settlement to the Project and is predominantly inhabited by non-indigenous residents. Much of the land in and around the town is privately owned. Structures comprise mainly small timber and brick cottages, with sheet metal or thatch roofs.

Other small towns around the Project include, Zumbi, Zurmi, Santa Elena and Guayzimi. Guayzimi lies to the west of the proposed Tailings Management Facility.

5.4 PHYSIOGRAPHY AND CLIMATE

The climate at the Project is typical for areas situated at this elevation along the Amazonian side of the Andes. Daily temperatures range between 18 °C and 29 °C, and average 22 °C. Rainfall is in the order of two to four metres per year, with maximum rainfalls occurring between February and April. However, heavy rainfalls can occur at any time of the year. Fog and cloud cover is typical during the rainy season. Except for disruptions that may occur as a result of unusually heavy rainfall, the Project can be operated year round.

The meteorological information detailed in Table 5-1 was obtained over a ten year span at two meteorological stations located in regional towns, 23 kilometres and 37 kilometres from the Project area. As the Project and proposed infrastructure is located at different locations and elevations, it is recommended that the assumed rainfall data is verified by establishing a meteorological station at the Project site.

Table 5-1: Regional Climate Data

Item	Unit	Value
Mean High Temperature	°C	28.6
Mean Low Temperature	°C	18.1
Daily Mean Temperature	°C	21.7
Average Annual Rainfall at Yanzatza, 30 km. distant	mm/year	2 045
Average Annual Rainfall at Paquisha, 18 km. distant	mm/year	3 724
Average Annual Evaporation	mm/year	1 237
Humidity	%	89.1
Average Wind Velocity	km/h	9.0

5.5 AVAILABILITY OF LOCAL RESOURCES

5.5.1 CONDORMINING RESOURCES

The Condormining exploration camp is located approximately 9 km north east of the Project; a one hour drive from the Project. Bottled water is brought in to the camp site for drinking and cooking while water for washing and showers is obtained via pipes from local springs or collected in rain water tanks. This camp can accommodate over 60 personnel. Electricity to the camp is provided by a government installed 22kV line to the nearby town of Chinapintza, via a transformer with two lines into the camp. These power lines are currently being upgraded by the authorities to three phase power. An 85kVA generator supplies backup power to the Condormining camp.

Due to the distance between the Condormining camp and the Project, a Santa Barbara exploration camp is located to the immediate north of the Yapi River Bridge about 3 kilometres from the Project. The camp comprises a mess facility and accommodation for approximately 30 people. The facility is leased from the local Shuar community who reside in Pachikutza.

Internet services are provided to the Condormining and Santa Barbara camps with microwave links.

5.5.2 SURFACE RIGHTS

Condormining has acquired surface rights that cover most of the immediate Project area. It is anticipated that continued good standing and communication with the community will allow for additional surface rights to be acquired for future operations.

5.5.3 POWER SOURCE

Currently, there are government owned and operated 22kV power lines running to the nearby villages of Pachitkutza, Los Gerianos and Chinapintza. The existing infrastructure will not be capable of providing the large power demand required for the Project, therefore it is proposed to connect to the national grid via high voltage transmission lines. This option has been assessed to be the most reliable and least costly in terms of capital expenditure and operating cost. The PEA assumes that there is sufficient capacity in the national grid to supply the Project.

Constructing a hydroelectric power plant in the vicinity of the Project was investigated, however it was deemed to be too capital intensive. A heavy-fuel-oil power plant was also investigated, but this was deemed to be unsuitable due to the high operating costs.

5.5.4 WATER SOURCE

It is proposed to draw water required for mining, process and other Project applications from the Nangaritza River to supplement water recycled within the process facilities. The Yapi and Pachikutza rivers, as well as the numerous small streams in the area, could also be utilised. In addition, the Project is located in an area of high rainfall, hence it may be desirable to capture and use rainwater. Significant volumes of fresh water could be captured from Project infrastructure, the open pit and the waste rock dump areas and diverted to a dedicated storm water dam, to be used for process applications.

SECTION 6 HISTORY

6.1 BACKGROUND AND CHRONOLOGY

Exploration has occurred within the concession areas since 1988. This exploration work has continued through to the present day, however a moratorium which froze all new exploration activities was imposed by the Ecuadorian government between 15 April 2008 and 12 January 2009. This moratorium was enacted to allow a revision of the country's mining laws to bring these laws more in line with those of other South American countries. After the moratorium was lifted Condormining acquired a permit to drill within the concessions which was granted effective 1 July 2011.

During the moratorium, the concession owners at the time, Goldmarca/Ecometals, were restricted to data compilation and processing data collected to date, re-logging existing core and generating new geological models to be used in future exploration programs along with limited surface mapping and geochemical sampling.

A summary of exploration work completed between 1993 and the present day by the various concession owners is provided below.

6.2 PRE–1993

Gold was discovered within and about the property now held by Condormining in 1984 by prospectors and informal miners in what became known as the Pachicutza mining camp (4). Previous studies have indicated that the presence of gold in the area was known since pre-Columbian times (4).

From 1988 to 1991, Pachicutza CEM, an association of companies including La Direccion de Industrias del Ejercito (DINE), in which Prominex U.K had a majority interest (65 %), initiated formal exploration within and about the property covering some 25 000 hectares (4). This work included regional mapping, geological reconnaissance and geochemical stream sediment sampling. The bulk of the currently known prospects and deposits (gold - polymetallic veins and porphyry breccias) were first discovered at this time (4).

In 1991, Prominex U.K. withdrew from the project and TVX Gold (TVX), under the name of Condormining, acquired the property previously held by Pachicutza CEM with DINE and the Chulapas Mining Company as partners.

.

6.3 TVX EXPLORATION 1993 TO 1999

Exploration was put on hold during the short border war between Peru and Ecuador in 1995. TVX completed 6 000 metres of trenching across the various mineralised breccia pipes and areas which had been discovered by the earlier rock chip and soil geochemical sampling. TVX also completed 10.2 line-km of ground geophysics, took 1 200 geochemical soil samples on 3 000 metres of grid, assayed 23 539 drill core samples, and took 2 800 underground samples and 3 636 rock chip samples (4). These areas included the following prospects as they were then known: La Pangui, Reina del Cisne, Los Cuyes, San Jose 1 and 2, Soledad, Bonanza, Guayas, Enma, Conguime, Santa Barbara and El Hito. Access within the property required the preparation of approximately 26 km of hand constructed roads and approximately 53 km of trails.

TVX used a variety of electronic surveying equipment for its surface and underground work, which included the Electronic Total Station GTS 13, TOPCON, Electronic Total Station Sokkia SET-6 and the Electronic Total Station Sokkia SET-3E each of which have accuracies of down to 1.0 mm. This equipment was used to create a series of triangulation stations from which second and third order points were established so that the property could be topographically surveyed and the locations of all of the trenches, drill holes, roads, etc. could be accurately located with X, Y and Z coordinates.

In early 1998, due to the prevailing corporate situation TVX withdrew from the project to develop Chinapintza; however, it continued to conduct exploration elsewhere in the area and particularly in the Southern Sector or southern part of the concessions at the El Hito and Santa Barbara porphyry targets.

Between 1994 and 1998, TVX covered the El Hito and Santa Barbara areas with a north-east trending soil sampling grid of dimensions 250 m x 50 m. In 1997 and 1998, additional work consisted of stream silt-sediment sampling and heavy mineral panned concentrate sampling along the rivers and creeks draining the area, outcrop sampling and channel sampling of altered/mineralised outcrops. TVX also established two detailed (50 m x 50 m) geochemical grids over areas of 1 000 m x 1 500 m in the Santa Barbara North and South areas, respectively. Exploration in the Santa Barbara sector consisted of Induced Polarisation Survey (IP) (29 250 m over 14 lines), trenching (5 100 m; 514 of 3 m chip channel samples) rock chip and soil geochemical sampling, mapping and drilling (17 holes 4 296 m). The IP defined two positive chargeability and resistivity anomalies that were tested by drilling.

Nine diamond drill holes were drilled in the north-east sector to test the coincident geochemical/IP anomalies for the indicated gold-copper potential. Two holes intersected significant widths of gold mineralisation -104 m (45 to 149 m) grading 0.73 g/t Au and 44 m (0 to 44 m) grading 1.32 g/t Au.

Eight diamond drill holes were drilled in the south-west sector to test the coincident geochemical/IP anomalies for the indicated gold-copper potential. Four holes intersected significant intervals of gold mineralisation -104 m (0 to 104 m) grading 1.3 g/t Au; 84 m (140 to 224 m) grading 0.7 g/t Au, 42.3 m (224 to 266.31 m) grading 0.54 g/t Au; 6 m (104 to 120 m) grading 1.5 g/t Au and 14 m (186 to 200 m) with 0.85 g/t Au; and, 228 m (16 to 244 m) grading 1.01 g/t Au.

In 2000, TVX abandoned all exploration within the El Hito and Santa Barbara portions of the property citing inadequate potential to meet their corporate objectives.

In December 2002, Goldmarca entered into a joint venture agreement with DINE to acquire the properties which now constitute the Project, and in 2003 initiated activities which are described in Section 6.4

6.4 GOLDMARCA/ECOMETALS 2003 TO 2010

Three magnetic surveys have been conducted over the Santa Barbara Project area: TVX (1999), Goldmarca (2006) and Ecometals (2008). The Goldmarca survey repeated and expanded the smaller TVX grid area. The detailed survey by Ecometals was interrupted by the exploration moratorium, and along with logistical problems, limited the area covered.

The magnetic response and the western mineralised zone at Santa Barbara South (cf. Analytic Signal) are shown in Figure 6-1.

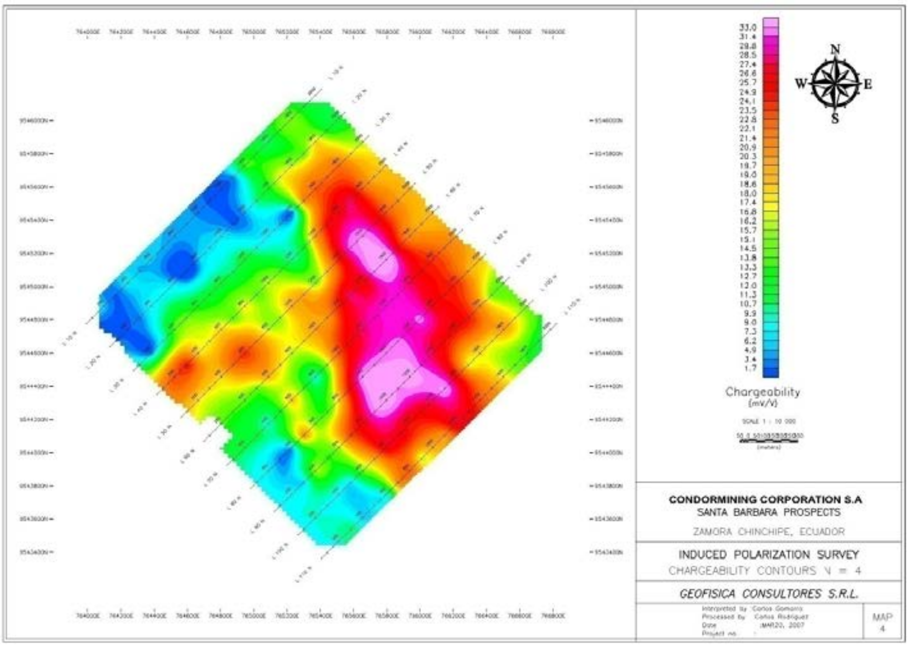

Figure 6-1: Santa Barbara IP Chargeability, N=4. (4)

6.4.1 PREVIOUS DRILLING

TVX drilling was summarised in Section 6.3. No drilling by Goldmarca/Ecometals was completed on the Project after the exploration moratorium was proclaimed in April 2008. Once the moratorium was lifted in January 2009, the project remained effectively dormant until 2010, pending the sale of Condormining to ECC.

From 2002 to 2008 a single 600 m deep hole was drilled at the Project.

6.5 HISTORICAL MINERAL RESOURCE ESTIMATES

Several generations of resource estimates have been calculated for the various mineralised areas. These include studies by TVX, Goldmarca, Ecometals and Condormining covering the Chinapintza veins, Los Cuyes deposit, Soledad and related breccias and the Santa Barbara porphyry gold deposit.

6.5.1 TVX RESOURCE STUDIES

From 1998, TVX continued exploration efforts at the El Hito and Santa Barbara porphyry targets. The following description of their activities is from the AMA Technical Report (AMA, 2014a).

A program of soil sampling, stream sediment sampling, grid and outcrop sampling and IP geophysical surveys were conducted over El Hito and Santa Barbara. TVX completed 17 core holes over coincident IP and geochemical anomalies at Santa Barbara, nine holes over the Northeastern Sector (now known as Santa Barbara North). Two holes intersected significant widths of gold mineralisation - 104 metres (45 to 149 m) grading 0.73 g/t Au and 44 metres (0 to 44 m) grading 1.32 g/t Au. TVX estimated that, using an specific gravity of 2.7, the mineralised structure contained an inferred resource of 5 M tons of material grading 0.91 g/t Au to a depth of 200 metres. This resource is a historical resource provided for informational purposes only and the authors of this report have not done sufficient work to classify the historical estimates as current mineral resources. Accordingly, the historical estimate is not being treated as current mineral resources.

Eight diamond drill holes were drilled in the Southwestern Sector (now known as Santa Barbara South). Four holes intersected significant intervals of gold mineralisation - 104 metres (0 to 104 m) grading 1.32 g/t Au; 84 metres (140 to 224 m) grading 0.67 g/t Au, 42.31 metres (224 to 266.31 m) grading 0.54 g/t Au; 6 metres (104 to 120 m) grading 1.54 g/t Au and 14 metres (186 to 200 m) with 0.85 g/t Au; and, 228 metres (16 to 244 m) grading 1.01 g/t Au. TVX estimated that the mineralised structure has an inferred resource of 21 Mt of material grading approximately 1.0 g/t Au to a depth of 200 metres. As above, the authors of this report have not done sufficient work to classify these historical estimates as current mineral resources and therefore the historical estimate is not being treated as a current mineral resource. In 2000, TVX abandoned all exploration at the El Hito and Santa Barbara prospects.

Table 6-1 details a historical mineral resource estimate completed in 1998 by TVX based on the work completed on the Santa Barbara prospect. As above, the authors of this report have not done sufficient work to classify this historical estimate as current mineral resources and EGX are not treating the historical estimate as current mineral resources.

Table 6-1: TVX Mineral Resource (Easdon, 2004)

Deposit	Resource Type (t)			Grade (g/t Au)	Total Grams Au	Total Oz Au
	Measured	Indicated	Inferred			
Santa Barbara	-	-	26 000 000	1.00	26 000 000	840 000

6.5.2 GOLDMARCA HISTORIC RESOURCES

In 2004, Goldmarca contracted a Chilean engineering group South American Management S.A. (SAMSA) to prepare resource estimates for seven separate mineral deposits identified by TVX and other previous workers. The objective of this study was to complete a resource estimate in accordance with NI 43-101 based on the data generated in large part by TVX.

In the course of this resource evaluation, Goldmarca completed a substantial amount of verification and due diligence. This work included re-logging of the available TVX core, and re-sampling of select intervals of the core using acceptable QA/QC methods. Mapping and rock sampling over the gold deposits was also conducted as well as trenching over the Chinapintza, Soledad, Enma Reina del Cisne and Santa Barbara targets. SAMSA compared the gold assay results of TVX to the re-sampling done by Goldmarca and concluded the assay results had a high correlation coefficient, indicating the assays by TVX were reliable. On the basis of this confirmed data, SAMSA calculated the resources for the various deposits using 0.4 g/t gold cutoff. Their results are presented in Table 6-2.

Table 6-2: Inferred Santa Barbara resources calculated by SAMSA (AMA, 2014a)

Deposit	Cutoff Grade (g/t Au)	Inferred Resource (t)	Grade (g/t Au)	Contained Oz Au
Santa Barbara North	0.4	5 000 000	0.9	146 000
Santa Barbara South	0.4	21 000 000	1.0	675 000

The resources calculated by SAMSA followed the CIM 2000 resource guidelines and may not conform to current CIM standards. The information here should be considered historic in nature and for historical information only. The authors of this report have not done sufficient work to classify these historical estimates as current mineral resources and EGX is not treating the historical estimates as current mineral resources.

6.5.3 EGX MINERAL RESOURCE STATEMENTS

EGX has previously reported Mineral Resource estimates for the Deposit in their March 2014 Technical Report (AMA, 2014a), October 2013 Technical Report (AMA, October 16, 2013) and July 2013 Technical Report (AMA, July 23, 2013). This latest resource estimate, after including the newer drilling results, has in effect converted a portion of the previously reported Inferred Mineral Resource to the Indicated Mineral Resource category and upgraded additional blocks to the Inferred Mineral Resource category.

6.5.4 PRODUCTION

Small-scale miners have been extracting gold bearing material from the Chinapintza veins since the 1980’s and this activity is continuing to the present. Illegal underground exploitation of the veins and processing with a variety of mill types using mercury and/or cyanide for recovery is common. However, there are no production records for the Chinapintza or Pangui camps where most of this activity has occurred. No small scale mining around the Project has been recorded. The resources on the Project are believed to be relatively unaffected by this historic mining.

SECTION 7 GEOLOGICAL SETTING AND MINERALISATION

7.1 REGIONAL GEOLOGY

The Project is located within the Condor Cordillera, between the Andean Cordillera Real in the west and the Pre-Cambrian Amazon Craton. It forms part of a significant Jurassic-Cretaceous back-arc fold-thrust belt with Jurassic granitoid plutons and younger supracrustal sequences consisting of Palaeozoic and Mesozoic sediments and arc-related igneous-volcanic lithologies as shown in Figure 7-1. With many mineral discoveries over the past twenty years in south-eastern Ecuador, this district has become an important gold and copper belt linking the world class deposits of Chile, northern Peru and Columbia. This district is host to the Mirador I and II porphyry copper deposits roughly 60 km to the north as well as the Fruta del Norte intermediate sulphidation epithermal gold deposit located 35 km to the north and the Nambija gold skarn deposit located 25 km to the west. This district, including the Project is now called the Zamora Copper-Gold Belt, replacing former names Corriente Copper Belt and the Pangui Belt. All of these deposits are related to Late Jurassic magmatism (Drobe, 2013).

Palaeozoic (Precambrian to Devonian) Isimanchi (Pumbuiza and Macuma) Formations, comprising slates, schist and quartzite form the basement in the district. These rocks are unconformably overlain by the quartzite, limestone/marbles, siltstones/slates and volcanic and volcaniclastic rocks of the Piuntza Formation of Triassic to Lower Jurassic age. These rocks are overlain by red bed sandstones, turbidite, mudstones and basaltic flows of the Chapiza Formation. The older rocks appear to be preserved as down faulted blocks and as roof pendants within the Zamora Batholith. The Project area lies at the approximate eastern limit of the Batholith. These units are in turn overlain by transgressive marine sediments comprising sandstone, mudstone and fossil-bearing limestone of the Lower Cretaceous Hollín and Napo Formations, which are themselves overlain by rhyolitic to dacitic pyroclastic volcanics of the Lower Cretaceous Chinapintza Formation. Table 7-1 shows the stratigraphic section of the Project.

The dominant geological feature in the area is the upper Middle Jurassic I-type Zamora batholith, a regionally extensive intrusive complex elongate north-northeast and parallel to the Ecuadorian Andes. The batholith is comprised of hornblende-bearing diorite and granodiorite, plus lesser granite, tonalite and monzodiorite. Related to the main intrusive are andesitic volcanic rocks and intermediate to mafic dikes and porphyries that locally intrude the batholith. The Misahuallí Formation is interpreted to be coeval with the Zamora Batholith. The Misahuallí Formation is a mix of volcanics, volcaniclastics, epiclastics and intrusives that range in composition from alkali basalt to dacite and crop out as approximately north – south aligned supra-crustal pendants within the Zamora Batholith. Recent reporting of age dates in the region by gives a Middle Jurassic age (169-164 Ma) (Drobe, 2013). The batholith was emplaced along a north-south structural feature as evidenced by contact relationships with roof pendants of Triassic-Jurassic volcanosedimentary formations, including the Triassic Santiago Formation and Early Jurassic marine sedimentary formations. The region was then intruded by coeval Late Jurassic diorite porphyry dikes and stocks which tend to be associated with the margins of the Zamora batholith, and with many of the areas of mineralisation in the region.

Breccia zones associated with the batholith are important in the Mirador copper/gold porphyry and other copper deposits of the Zamora Copper-Gold Belt located ~60 km to the north of the Project area. Felsic to intermediate pyroclastic rocks and high level porphyries preceded and/or accompanied early movement on regional fault zones within the batholith. These Late-Cretaceous rocks (160-145 Ma) are spatially associated with mineralisation in the Chinapintza portion of the Project epithermal gold–silver systems as well as the Nambija gold skarn district (Henderson, 2010).

This entire sequence of rocks was then intruded by a series of acid to intermediate, commonly porphyritic, intrusive stocks and dikes. The emplacement of these intrusives is controlled by regionally developed structures. Associated with this Middle Cretaceous igneous activity is a volcanic event with associated near surface intrusive activity and associated mineralisation.

The region is strongly structurally controlled by through going north–northeast to north-south trending lineaments cross-cut by younger, northeast and northwest sets. Jurassic magmatism and volcanism was partly controlled by the former regional lineaments which remain active.

Table 7-1: Stratigraphic Section for the Project

Age	Formation	Description
Cainozoic	N/A	Undifferentiated sediments
Mid-Upper Cretaceous	Unknown	Felsic-intermediate stocks and dykes and shallow volcanism plus mineralisation
Lower Cretaceous	Chinapintza	Rhyodacite-dacite volcanic suite
Lower Cretaceous	Hollin and Napo	Marine sediments - sandstone, limestone and mudstone
Upper Jurassic	Misahualli	Flows, breccias, pyroclastic and volcaniclastic rocks
Lower-Mid Jurassic	Chapiza	Continental red beds, turbidites, mudstones and
Lower-mid Jurassic		Igneous intrusions, notably Zamora Batholith tonalite and granodiorite (I-type tonalitic)
Triassic-Lower Jurassic	Piuntza	Siliciclastic rocks, siltstone, volcanic and volcaniclastic rocks
Unconformity		
Pre-Cambrian-Devonian	Isimanchi	Slate, schist, siliciclastic rocks

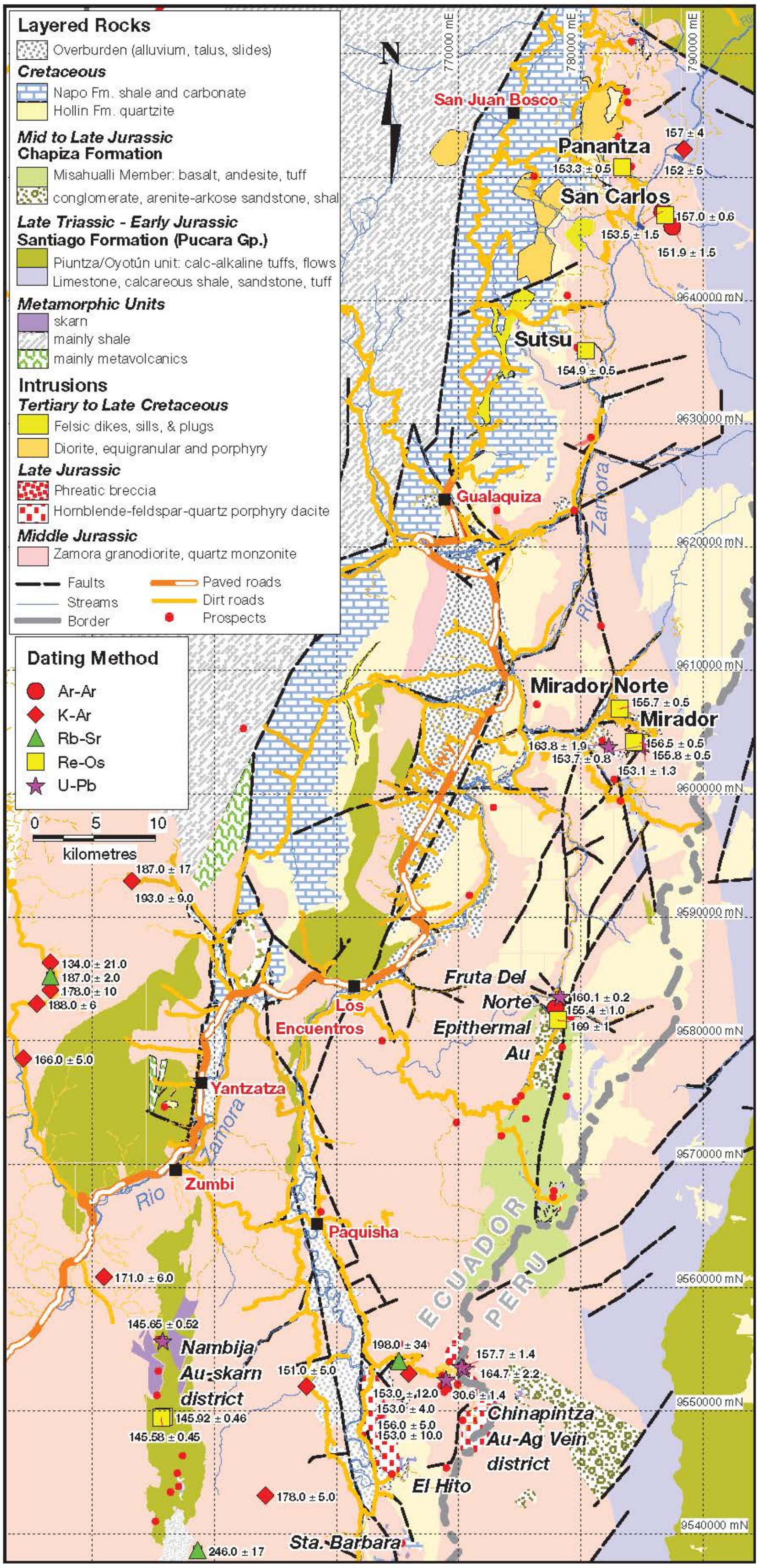

Figure 7-1: Regional Geological map of southeast of Ecuador (Drobe, 2013)

7.2 LOCAL GEOLOGY

Locally, there are three distinct and different geologic settings represented: the Chinapintza vein district; the Condor Sector epithermal gold complex; and the Southern Sector with both porphyry Gold-Copper and porphyry Copper-Molbdenum settings. Figure 7-2 shows the geological map of the Project.

The Santa Barbara - El Hito area (referred to as the Southern Sector), which is located approximately 7.5 km to the south of the Condor Sector, is underlain by the gently dipping, continental and volcaniclastic sediments (comprising coarse, ferruginous (redbed) sandstones with intercalated conglomerates, tuffs, agglomerates, etc) of the Chapiza Fm., which are overlain by the andesites and basalts (locally with pillows) of the Misahualli Fm. The Chapiza Fm. partially overlies, and in part comprises the lateral facies equivalent of the calcareous turbidites of the Santiago Fm. (Lower Jurassic) (Ecuador Ministry of Energy and Mines, 2000). The sedimentary/volcanic sequence has been intruded by the rocks of the Zamora Batholith. The bulk of the exposed rocks at Santa Barbara comprise the andesitic/basaltic volcanics of the Misahualli Fm. which have been intruded by a north-northwest trending swarm of felsic porphyry dikes and stocks which may be of Tertiary age. The Northern Sectors are separated from the Santa Barbara area by rocks of the Zamora Batholith.

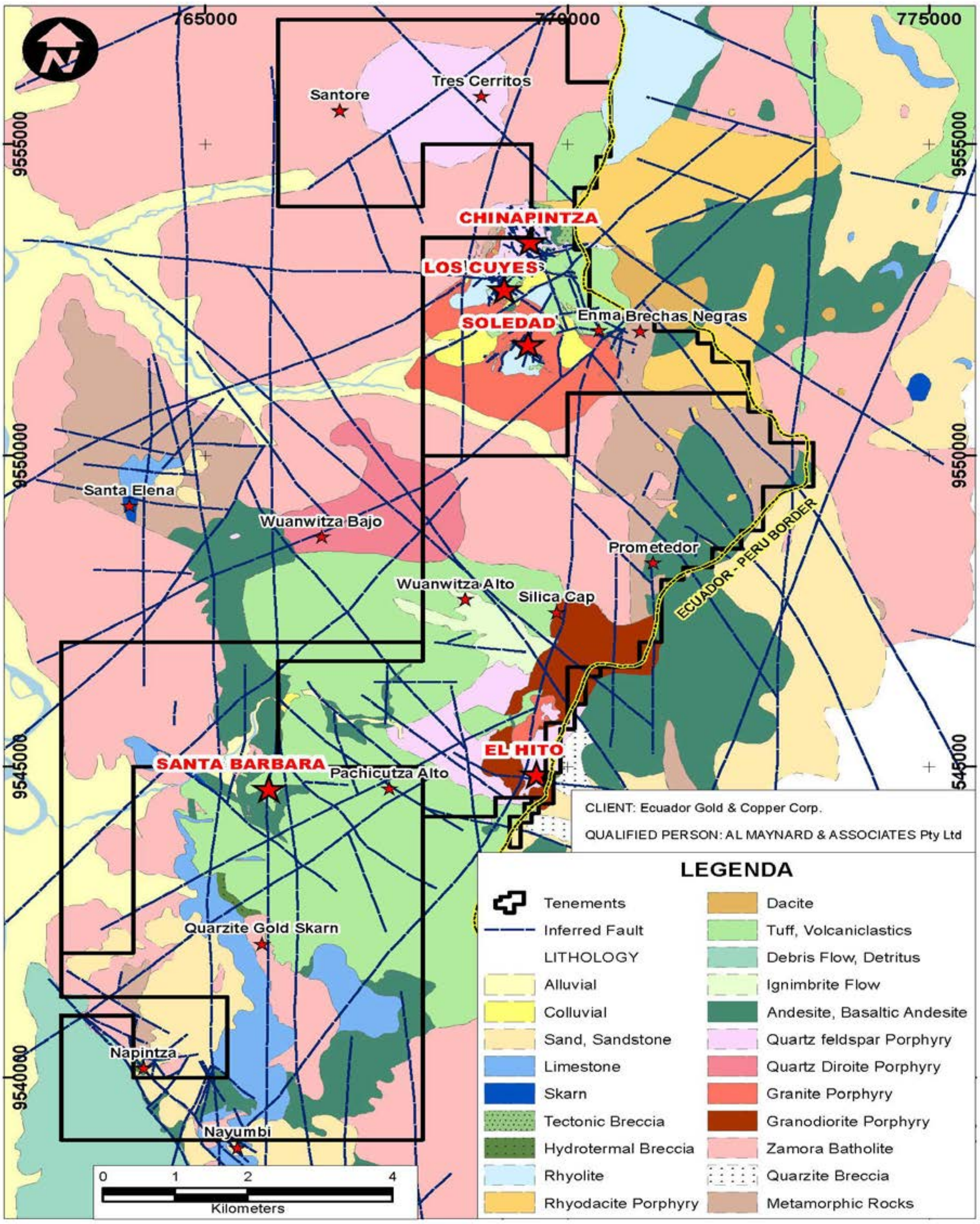

Figure 7-2: Geological map of the Project

SECTION 8 DEPOSIT TYPES

8.1 SANTA BARBARA PORPHYRY GOLD-COPPER DEPOSIT

The focus of recent exploration activity by EGX and Condormining has been the Santa Barbara porphyry gold-copper deposit, located south of the Condormining camp near the village of Pachicutza. Initially drilled by TVX, this deposit has developed into a more significant gold resource based on the results of EGX and Condormining drilling in 2012 and 2013.

8.1.1 GEOLOGY

The most extensive unit exposed in the Santa Barbara target is a fine-grained green basaltic andesitic volcanic rock which has been assigned to the Upper Jurassic Misahualli formation. Overlying the volcanics is a sedimentary sequence comprising conglomerate, quartz sandstone, limestone and locally garnet skarn. In drilling, sedimentary rocks also appear to be intercalated with the andesite volcanic noted particularly on the western margin of the drilling. Whether the sedimentary rocks are both coeval and younger than the volcanic rocks has not been determined.

The andesite and other units have been intruded by a swarm of 2-30 m wide northwest trending diorite porphyry dikes, and a large diorite porphyry stock in the northwestern potion of the prospect area. These dikes are exposed in the northeast portion of the Project. At least two types of diorite porphyry dikes have been identified on the surface and in drilling. A relatively feldspar phenocryst-rich variety apparently directly associated with mineralisation (DP1), and a post-mineral hornblende phenocryst-rich variety (DP2) have been designated. The DP2 diorite porphyry also forms the stock in the northwest portion of the property and seems to truncate a series of northwest trending DP1 dikes. Another interpretation is the large DP2 body is the primary porphyry stock with apophyses of metalliferous dikes rising from the main intrusive stock. Other types of porphyry dikes not clearly DP1 or DP2 have been encountered complicating the interpretation. See Figure 8-1 for the geology and drill hole locations of the Project.

8.1.2 MINERALISATION

The main host for the gold-copper mineralisation at Santa Barbara is the basaltic andesite volcanic unit. Mineralisation is present in other rocks, but is most developed and has better grades in the andesitic volcanic unit, often in proximity to diorite porphyry dikes. Two separate mineralised zones have been defined in the prospect. Santa Barbara South is the main mineral body defined thus far. It is elongate in a north-northwest direction following the trend of interpreted faulting (see Figure 8-1). A total of 26 holes from current and past drilling been completed in Santa Barbara South with 23 of these holes defining a gold resource detailed in Section 14. As currently defined, the body extends at least 700 m, and is approximately 300 m wide. The contact is steep on the west side of the deposit and the zone dips consistently 40-50° to the east. Mineralisation is coherent and continuous, correlating well between holes at nominal 100 m spacing. Figure 8-1 is a plan map of drill hole locations and assays for the Deposit, which demonstrates continuity of the mineralisation. It is presently open to the east and at depth, where most of the recent drilling has focused. The results of drill holes DSB-28 and DSB- 29 appear to define the southern limit of the deposit.

The initial discovery holes were drilled in Santa Barbara North, which is located to the east in contact with the DP2 diorite porphyry stock. Gold-copper intercepts in three holes defined a small resource but mineralisation remains open to the east, southeast, and at depth. Additional drilling may extend and add mineralisation in these directions.

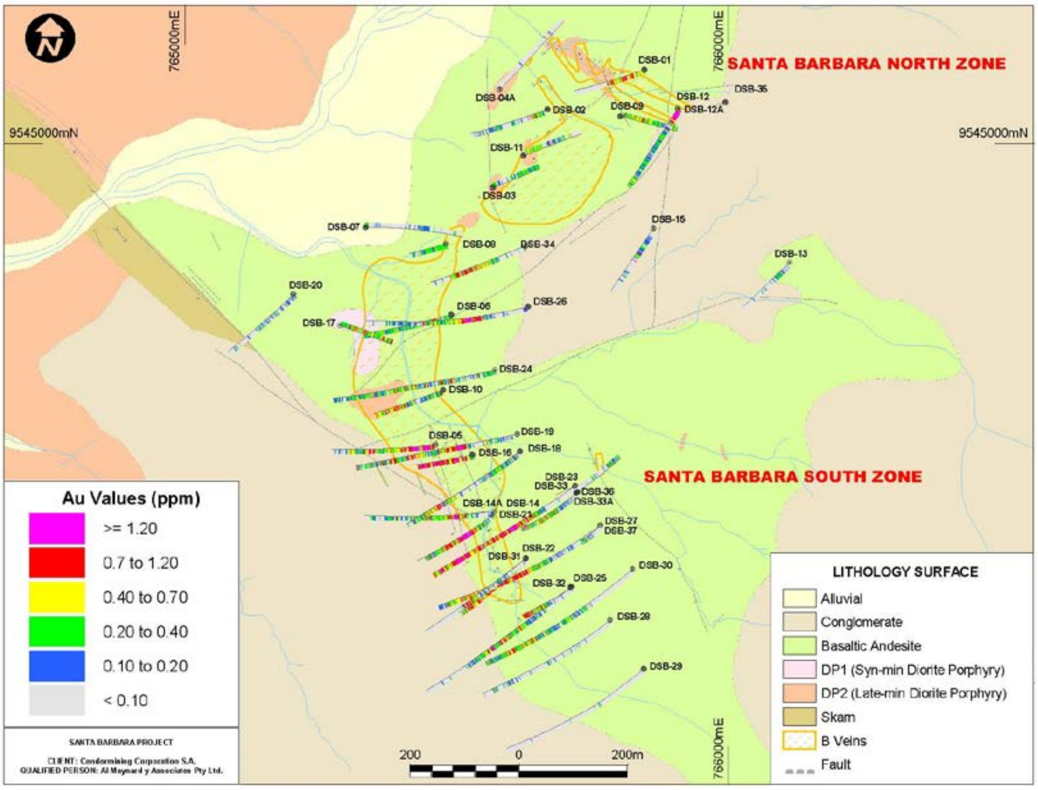

Figure 8-1: Geologic Map and Drill Hole Locations for Santa Barbara

High gold values are closely associated with chalcopyrite. Key indicators of gold-copper mineralisation are the presence of B-type quartz veins which often carry sulphide minerals, biotite alteration and disseminated pyrite. Pyrrhotite is also present in the system but serves as a negative indicator as it tends to occur outboard of the gold-copper mineralisation. There are at least two quartz vein types, present in the system; a massive deformed quartz vein suggesting high temperature ductile deformation, and later straight B-type veins with white to banded and coliform quartz typical of many porphyry deposits.

8.1.3 ALTERATION

Alteration associated with the gold-copper mineralisation is patchy to pervasive very fine-grained secondary biotite or phlogopite indicative of potassic alteration often with finely disseminated magnetite. Propylitic alteration as evidenced by chlorite-epidote and actinolite forms a halo around the potassic alteration. Alternatively, the propylitic alteration was the primary alteration possibly from contact metasomatism of the diorite porphyry intrusions or of the Zamora batholith. It has been suggested that there is also an illite alteration overprint over the potassic alteration. Late stage alteration includes minor prehnite, calcite and zeolite veins (Hedenquist, 2007).

8.1.4 DEPOSIT TYPE

Gold-rich copper porphyry deposits are a relatively recently recognized subclass of porphyry deposits. A gold-rich porphyry deposit has been defined as one containing >0.4 g/t gold (Sillietoe, 1998). These deposits show features generally similar to copper porphyry deposits, i.e., genesis, alteration features and geologic setting. The Santa Barbara Deposit belongs to this class of gold-rich porphyry deposits recognized elsewhere in the Andean chain. Other gold rich porphyry copper deposits are found in the Maricunga district in Chile where multiple deposits of this type exhibit a variety of grades and tonnages as well as mineralisation styles. Included in this district are the Lobo deposit, the Marte deposit, Verde deposit and La Pepa deposit.

A more recent gold-copper porphyry discovery in the northern Andes is the Colosa deposit located in Central Cordillera in west-central Columbia. Colosa is one of the largest deposits of this type although with a ~8 million year age, it is younger in age compared to the Maricunga deposits with Oligocene to Mid-Miocene ages (Gil-Rodriguez, 2010). The age of Santa Barbara is not known. However, recent age dating of other mineral deposits in the Zamora Copper-Gold Belt at the Mirador I and II porphyry copper deposits roughly 60 km to the north, and at Fruta del Norte gold deposit indicate a Late Jurassic age of 156 million years (Drobe *et al.*, 2013).

The Colosa deposit shares many similarities with Santa Barbara including the association of the gold mineralisation with biotite alteration, and porphyry A and B type quartz veins. Mineralogy is also similar, with the presence of pyrite, magnetite, chalcopyrite, and molybdenite in both systems. Both also have pyrrhotite peripheral to the gold mineralisation.

The Verde and Pancho gold porphyry deposits in the Maricunga district are associated with subvolcanic andesitic to dacitic intrusions emplaced into coeval volcanic rocks, a similar setting to the Santa Barbara deposit (Muntean and Eiaudi, 2000).

SECTION 9 EXPLORATION

Since the acquisition of the Project by EGX in July 2012, exploration activities, in addition to the Phase I and Phase II drilling programs (under which 18 112 m were drilled) and the Phase III drilling program (3 939 m in seven drill holes at Santa Barbara) (see Section 10.1), include mapping at Los Cuyes, El Hito and Santa Barbara by EGX and Condormining geologists. Minor chip sampling and mapping was also completed at Santa Barbara. Surface mapping and sampling are presently continuing at El Hito and Santa Barbara. The results of this work are currently being compiled.

Following the Ecuadorian government moratorium, a permit to drill in the concessions was granted. By July 2011, the geological staff of Condormining were preparing for a surface exploration program including geological, alteration and structural mapping, extensive geochemical sampling, detailed re-logging of earlier drill holes and revision of geology models in preparation for the Phase I drilling program. These programs were designed to both delineate and expand known gold resources and define new and existing gold occurrences, both in outcrop and with scout drill intercepts.

All supporting geological, geochemical and related detailed information concerning the Project is incorporated into a new centralised database.

SECTION 10 DRILLING

10.1 EGX / CONDORMINING PHASE I, II AND III DRILL PROGRAM

EGX / Condormining began Phase I exploration drilling in August 2012. Drilling was undertaken with two drill rigs and was completed on 1 June 2013 with 20 drill holes and 13 131.1 m drilled with results first reported in a Technical Report dated July 23, 2013 and most recently reported in the March 2014 Technical Report (AMA, 2014a). The Phase II program was completed on 31 August 2013 with nine drill holes (two incomplete) totalling 4 224.5 m of drilling, and the results of Phase I and II were reported together in a Technical Report dated October 16, 2013 and most recently reported in the March 2014 Technical Report (AMA, 2014a). The Phase III program consisted of seven drill holes totalling 3 939.1 m completed between October 2013 and 31 January 2014 and the results were reported in the March 2014 Technical Report (AMA, 2014a).

Two drill contractors have been employed in the Phase I program, Roman Drilling Corp SA of Cuenca, Ecuador and Hubbard Perforaciones CIA, LTDA of Cuenca, Ecuador. Hubbard Perforaciones completed all of the Phase II and Phase III drilling. All holes were drilled with diamond core using HTW size core initially and reducing to NTW as needed. Core recoveries in these programs have been very good, averaging about 93 %. Problematic zones encountered are at the surface where recoveries are complicated by lateritic weathering, and occasionally in fault zones. Other than these areas, recoveries are generally above 95 %. All holes have been surveyed using Reflex Multi-shot down hole survey equipment. Down hole survey readings are taken every 75 m. There are no drilling, sampling or recovery related issues that could materially affect the reliability or accuracy of the results.

Table 10-1 lists the location of each EGX / Condormining hole along with the orientation, dip and total depth.

Table 10-1: EGX / Condormining Phase I, II and III Drillhole Data

Hole Number	Hole Type	Target	UTM (PSAD 56)		Elevation (m)	Dip (degrees)	Azimuth (degrees)	Total Depth (m)
			Easting	Northing				
DSB-19	HQ Core	Santa Barbara	765,611.22	9,544,461.75	943.92		257	600.00
DSB-20	HQ Core	Santa Barbara	765,199.72	9,544,720.05	898.7	-65	225	400.20
DSB-21	HQ Core	Santa Barbara	765,563.48	9,544,309.94	949.25	-70	230	457.20
DSB-22	HQ Core	Santa Barbara	765,626.61	9,544,231.54	1 014.51	-75	230	676.96
DSB-23	HQ Core	Santa Barbara	765,716.92	9,544,365.24	1 002.75	-65	235	700.13
DSB-24	HQ Core	Santa Barbara	765,569.86	9,544,580.11	954	-65	255	725.42

Hole Number	Hole Type	Target	UTM (PSAD 56)		Elevation (m)	Dip (degrees)	Azimuth (degrees)	Total Depth (m)
			Easting	Northing				
DSB-25	HQ Core	Santa Barbara	765,709.92	9,544,178.50	1 046.6	-60	230	704.09
DSB-26	HQ Core	Santa Barbara	765,631.13	9,544,697.17	984.59	-65	255	701.04
DSB-27	HQ Core	Santa Barbara	765,763.66	9,544,292.62	1 056.81	-60	230	772.97
DSB-28	HQ Core	Santa Barbara	765,781.775	9,544,116.899	1 088.033	-65	230	626.67
DSB-29	HQ Core	Santa Barbara	765,843.393	9,544,026.31	1 101.925	-65	230	694.94
DSB-30	HQ Core	Santa Barbara	765,822.887	9,544,211.128	1 093.503	-65	230	771.14
DSB-31	HQ Core	Santa Barbara	765,625.858	9,544,230.908	1 014.722	-50	230	350.52
DSB-32	HQ Core	Santa Barbara	765,709.919	9,544,178.5	1 046.596	-75	230	472.13
DSB-33	HQ Core	Santa Barbara	765,717.408	9,544,365.825	1 002.752	-80	235	213.36
DSB-33A	HQ Core	Santa Barbara	765,719.785	9,544,352.427	1 009.333	-80	230	813.51
DSB-34	HQ Core	Santa Barbara	765,625.976	9,544,809.518	992.081	-65	250	453.84
DSB-35	HQ Core	Santa Barbara	765,993.922	9,545,075.007	1 096.555	-60	255	34.74
DSB-36	HQ Core	Santa Barbara	765,721.565	9544353.871	1 009.421	-80	50	591.31
DSB-37	HQ Core	Santa Barbara	765,764.363	9,544,292.868	1 056.82	-85	230	523.95
DSB-38	HQ Core	Santa Barbara	765,879.6	9,545,168	1 034.918	-60	250	500.48
DSB-39	HQ Core	Santa Barbara	766,043.5	9,545,224	1 109.269	-60	210	500.17
DSB-40	HQ Core	Santa Barbara	766,093.7	9,545,185	1 117.739	-60	210	500.02
DSB-41	HQ Core	Santa Barbara	765,911.1	9,545,234	1 044.813	-70	205	512.06
DSB-42	HQ Core	Santa Barbara	765,678.6	9,545,057	975.666	-65	105	600.45
DSB-43	HQ Core	Santa Barbara	765,883.3	9,545,313	1 054.603	-60	205	441.96
DSB-44	HQ Core	Santa Barbara	765,511.8	9,544,102	1 063.715	-80	70	883.92
TOTAL								**15 223.18**

10.2 SANTA BARBARA TARGET

At the Santa Barbara target, porphyry gold-copper mineralisation is hosted by altered basaltic andesite intruded by Jurassic intrusive stocks and dikes. Major structural orientations are exhibited by steeply dipping intrusive contacts, B-type porphyry quartz veins and faults striking northwest and northeast. Eleven drill holes were completed and assay results returned for the Phase I drill program. Seven drill holes were completed and assay results returned for the Phase II drill program. The results of the Phase I and II drilling were first reported in the Technical Report dated October 2013 and most recently in the March 2014 Technical Report (AMA, 2014a). Seven holes have been completed under the Phase III drill program with one hole suspended since the previous Technical Report and the results of these holes are reported in the March 2014 Technical Report (AMA, 2014a). A total of 20 119.3 metres have now been completed on the Santa Barbara target by EGX and others. Of this total, 15 223.2 metres have been completed by EGX/Condormining since 2012.

10.2.1 PHASE I DRILLING

DSB-19 encountered 420 metres of 0.571 g/t gold including 101 metres grading 1.017 g/t gold. DSB-20 was collared 100 metres northwest of the mineralised zone on a roadside geochemical anomaly and failed to return significant results.

DSB-21 intersected the main gold zone approximately 50 metres south of previous intercepts, intersecting almost continuous gold mineralisation over 350 metres from 68 metres to 418 metres grading an average of 0.74 g/t gold, 0.11 % Cu and 1.09 g/t silver. This drill hole is shown on Figure 10-2. Drill hole DSB-22 intersected almost continuous gold and copper mineralisation over 484 metres from 190 metres to 674 metres grading an average of 0.80 g/t gold and 0.13 % copper.

DSB-23 was drilled to 700 metres at -65 degrees, parallel to and collared approximately 70 metres northeast of DSB-22. This hole returned significant results within a continuously mineralised 496 metre interval starting at 204 metres that averaged 0.90 g/t gold and 0.12 % copper ending in mineralisation at 700 metres. The mineralised zone includes 144 metres averaging 1.03 g/t gold and 0.15 % copper and an additional 82 metres averaging 1.16 g/t gold and 0.15 % copper. A 40 metre section between 310 and 350 metres averaged 1.42 g/t gold and 0.18 % copper. Figure 10-3 shows the drill hole section with alteration patterns through DSB 22, 25, and 27.

Hole DSB-25 returned 212 metres averaging 0.55 g/t gold and 0.11 % copper. This includes 50 metres that averaged 0.79 g/t gold. Drilled to a depth of 704 metres, this hole was collared 100 metres southeast of DSB-22. DSB-24 drilled through and along intervals of lower grade gold in thick intrusive dikes but still encountered 292 metres of 0.38 g/t gold and 0.08 % copper. This hole was located 200 metres northeast of DSB-19.

Assay results from drill hole DSB-27 (drilled to 772 metres) returned a mineralised intercept averaging 0.68 g/t gold and 0.12 % copper over 476 metres from 266 metres. This includes 310 metres averaging 0.80 g/t gold and 0.13 % copper and a second zone of 144 metres at 0.89 g/t gold and 0.13 % copper. Hole DSB-26 is a step out hole drilled 150 metres north of the known resource in the Santa Barbara South Zone that has encountered 146 metres of 0.90 g/t gold and 0.12 % copper starting at 204 metres. This hole was located 275 metres northeast of DSB-19 (420 m of 0.57 g/t gold and 0.08 % copper). The Santa Barbara South zone remains open to the north, east and south.

Holes DSB-28 and DSB-29 were completed on the southern extent of the previously defined Inferred Mineral Resource (refer to Section 6). DSB-28 encountered weak mineralisation reaching 0.13 g/t gold over a 110.13 metre interval starting at a depth of 482 metres. DSB-29 encountered only very weak gold mineralisation and alteration. These holes effectively define the southern limit of the Santa Barbara South zone.

10.2.2 PHASE II DRILLING

Phase II drilling focussed on extending gold mineralisation to the south, east and northeast of the Santa Barbara South deposit, as previously defined in past drilling. Phase II drilling in Santa Barbara South totalled 3 976.4 metres and DSB-33 which had to be abandoned.

Results from the Phase II drilling include 210 metres of 0.51 g/t gold starting at 482 metres in drill hole DSB-30. DSB-30 was collared 100 metres southeast of DSB-27 (drilled to 772 metres), which contains one of the longest mineralised intercepts to date. Hole DSB-32 intersected 238 metres of 0.64 g/t from 236 metres, ending in mineralisation at 472.13 metres. This hole was terminated short of its 600 metre target depth due to ground problems.

Drill hole DSB-33A extended mineralisation at depth and to the east of the then known Inferred Mineral Resource (7), intersecting 493.51 metres of 0.53 g/t gold starting at 320 metres. Mineralisation ended at 813.51 metres, in diorite porphyry. This is the deepest hole yet drilled on the Project. Drill hole DSB-34 encountered 212 metres of 0.62 g/t gold and 0.12 % copper from 154 metres, extending mineralisation 100 metres to the north of the Inferred Mineral Resource first defined in July 2013 and most recently reported in the March 2014 Technical Report (AMA, 2014a).

DSB-36 and DSB-37 were drilled east of the Inferred Resource (AMA, 2014a) and although lower grade, extended potential mineralisation a further 100 metres east of DSB-33A. DSB-36 encountered 159 metres of 0.28 g/t gold and 0.11 % copper starting at 432 metres, while DSB-37 intersected 182 metres of 0.23 g/t gold and 0.07 % copper. DSB-36 was oriented to the east, and was weakly mineralised until crossing a fault at approximately 420 metres, after which mineralisation increased. This suggests post-mineral faulting east of the Inferred Resource (AMA, 2014a).

10.2.3 PHASE III DRILLING

The Phase III drilling focused on extending gold mineralisation to the northeast, as previously defined in past drilling and one hole DSB-44 tested the high grade potential of the area below the abandoned drill hole DSB-32 that ended in mineralisation to the south. Phase III drilling totalled 3 939.1 metres. Total drilling by EGX and others at Santa Barbara is 20 119.3 metres as of this PEA.

The best results from the Phase III drilling in Santa Barbara North were from DSB-38 which encountered 294 m of 0.53 g/t gold from the surface, including 200 m from 94 to 294 m of 0.60 g/t gold and 38 m of 0.79 g/t gold from 256 to 294 m. Hole DSB-41 encountered 110 m of 0.42 g/t gold from 158 m depth and DSB-43 intersected a modest zone of 0.36 g/t gold from 190 to 226 m. These two holes suggest the mineralisation is weakening to the north of the previously defined Santa Barbara North target. Lack of mineralisation in holes DSB-39 and DSB-40 indicate the system is limited or cut off to the east of the previously defined mineralisation. The results of DSB-42 which was drilled from the main zone of mineralisation and was oriented to the east confirm the results of previous drilling. This hole contained 89 m of 0.69 g/t gold essentially from the surface. Hole DSB-44 was completed in the Santa Barbara South deposit to examine the southern margin of the known mineralisation. This hole successfully extended the deposit to the south, encountering 667.92 m of 0.67 g/t gold starting at 207 m depth, including a high grade zone between 298 to 434 m averaging 1.08 g/t gold.

Table 10-2 summarises assay results from Phase III drilling at the Santa Barbara target area. Figure 10-1 shows Santa Barbara North and Santa Barbara South geology and drill hole locations.

Table 10-2: Summary of Phase III EGX Santa Barbara Drill Results

Drill Hole	From (m)	To (m)	Interval (m)	Au (g/t)	Cu (%)
DSB-38	0.00	294.00	294.00	0.53	0.14
DSB-38	94.00	294.00	200.00	0.60	0.17
DSB-38	256.00	294.00	38.00	0.79	0.18
DSB-38	408.00	438.00	30.00	0.61	0.21
DSB-39	0.00	500.17	500.17	-	-
DSB-40	0.00	500.02	500.02	-	-
DSB-41	158.00	268.00	110.00	0.42	0.09
DSB-42	1.00	90.00	89.00	0.69	0.22
DSB-43	190.00	226.00	36.00	0.42	0.13
DSB-44	206.00	883.92	677.92	0.67	0.12
DSB-44	298.00	434.00	136.00	1.08	0.17
DSB-44	324.00	408.00	84.00	1.23	0.19

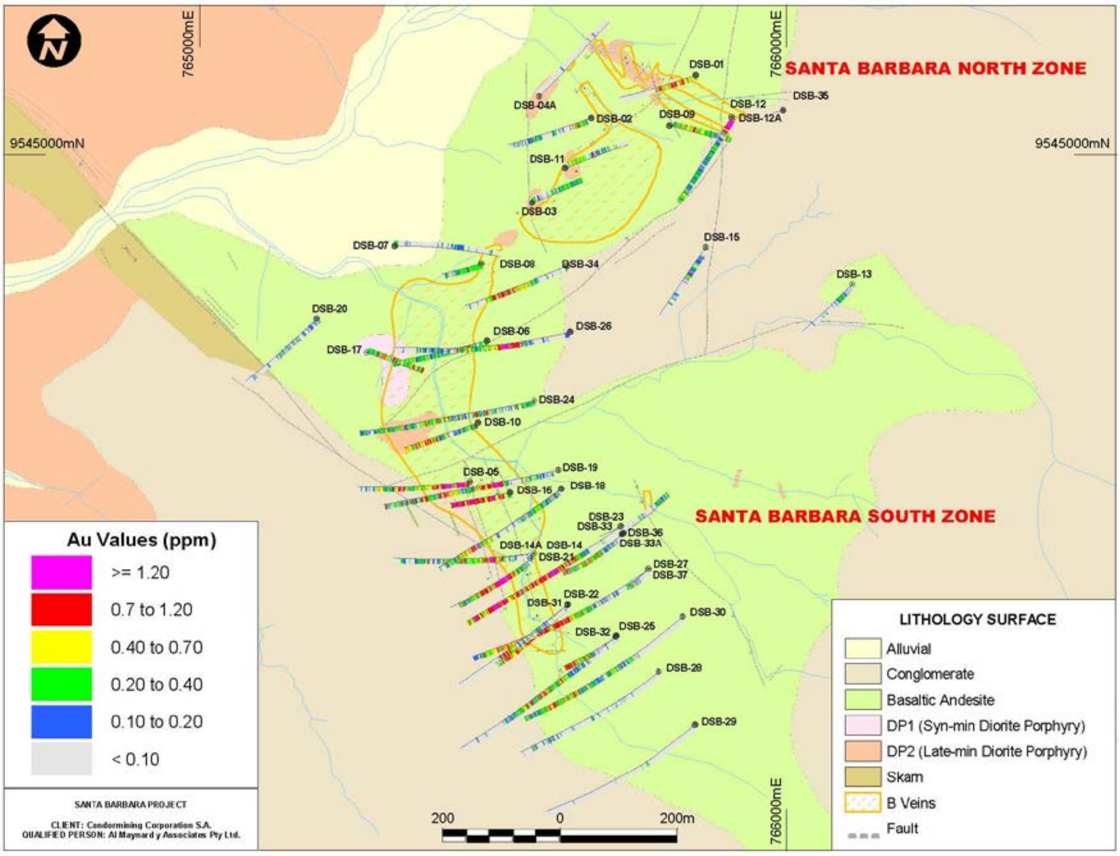

Figure 10-1: Santa Barbara Drill Hole Location with Geology

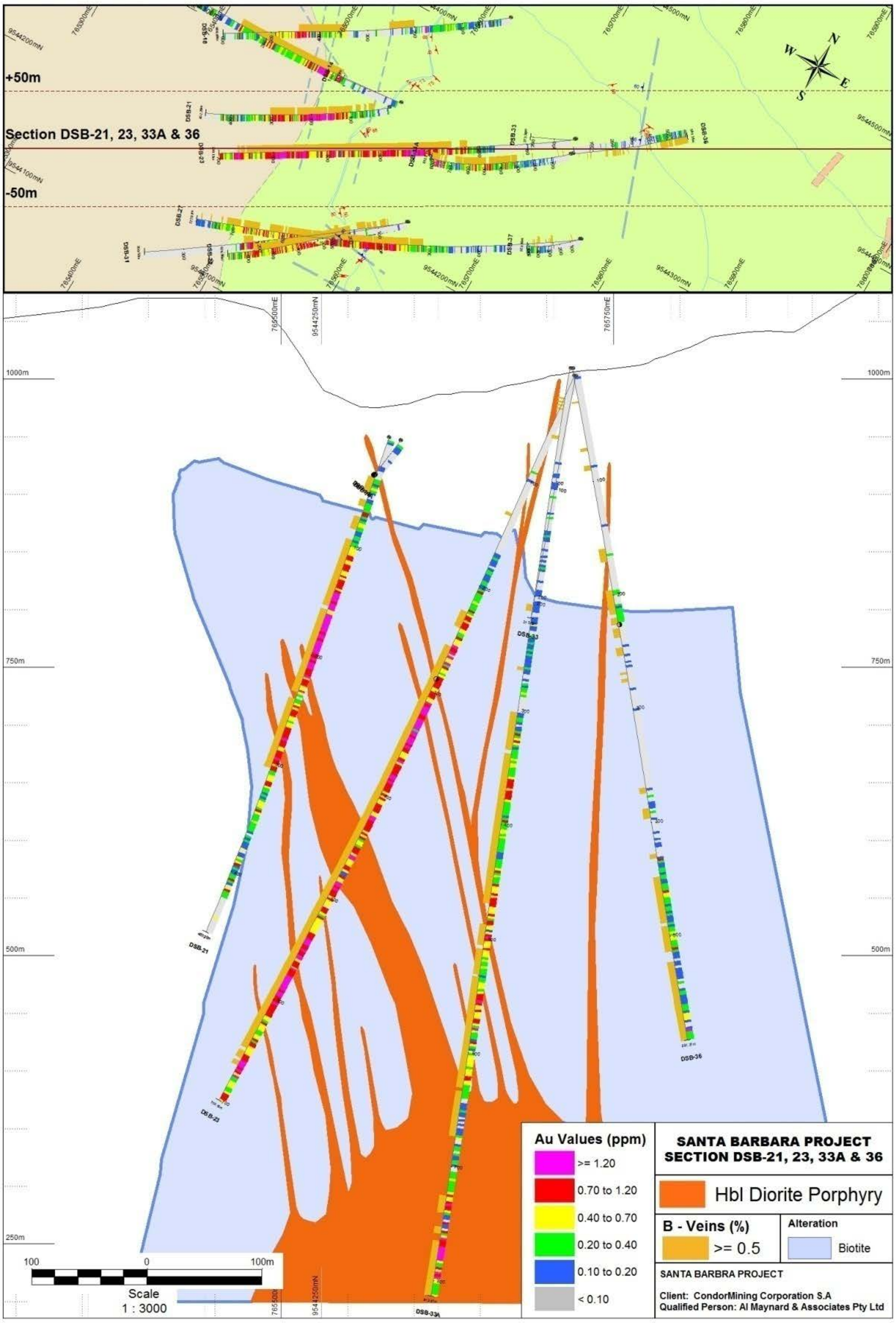

Figure 10-2: Santa Barbara Drill Hole Cross-Section DSB-21, 23, 33A and 36.

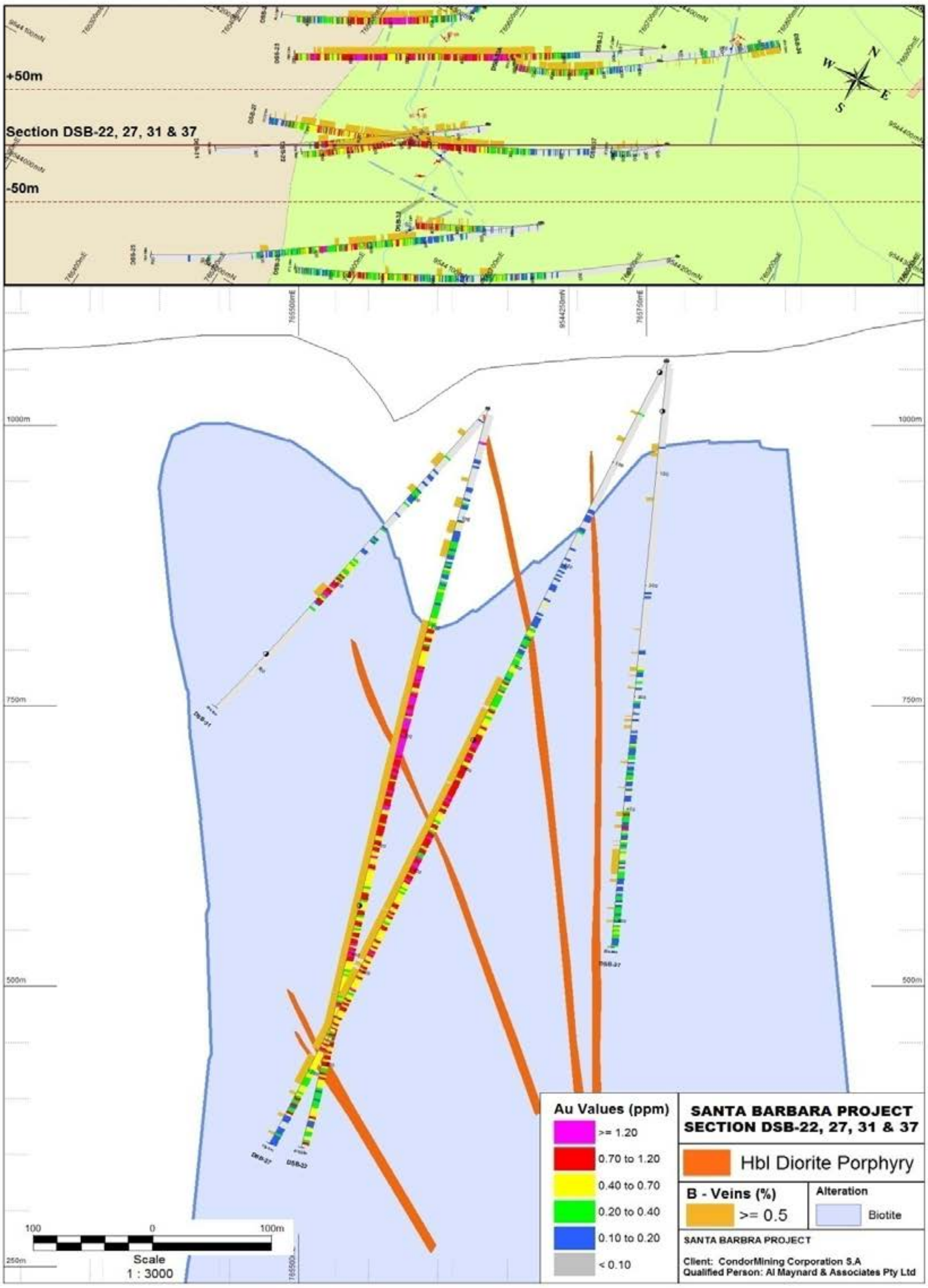

Figure 10-3: Santa Barbara Drill Hole Cross-Section DSB-22, 27, 31 and 37.

SECTION 11 SAMPLE PREPARATION, ANALYSES AND SECURITY

11.1 DRILL CORES

All diamond drill core was collected at the drill site by EGX / Condormining geologists and then sent in sealed trays to a secured central core processing and storage area. The core has been processed and stored at a number of different sites in the past but currently the core is being processed and stored in a large weather proof shed close at the Condormining exploration camp in the northern part of the concession. The stored core was inspected by AMA on each of their site visits and the core, racks, trays and depth markers are all in very good condition. The core recoveries of the core inspected corresponded with the logged core recoveries and generally the core was solid.

Prior to EGX / Condormining, almost all core was drilled with NQ size equipment with approximately 47.6 mm diameter core. Smaller BQ core, 36.5 mm in diameter, was drilled only at the bottom of a very few holes when difficult drilling conditions forced a reduction to complete the hole. EGX / Condormining drilling always begins with HTW core and only reduces to NTW core if hole conditions make it necessary.

11.2 SAMPLING METHODS AND APPROACH

Since the mineralisation at all the deposits is broad, mainly disseminated or controlled by stock-work veining with little variation in grade, the core was sampled over regular intervals commencing at the drill collar with the sample intervals varied to bracket only major geological contacts. The drill core in the earlier holes was sampled at regular 1.0 metre intervals from the hole collar but in recent years the core has been sampled over regular 2.0 metre intervals from the hole collar. To avoid volume variance effects, the core was digitally composited at 2.0 metre intervals for the resource estimation.

All core sampled was first marked at the appropriate intervals by the logging geologist with a line along the length of the core marking the top of the core to be split to ensure that the main mineralised structures were properly sampled, then photographed as a permanent record of the core prior to splitting. The core was then split by a trained technician using a diamond saw using the geologists' marks as a guide.

All samples, after splitting, were placed in sample bags marked with unique identification numbers with identically numbered tags torn from sample record books inserted inside the bags before sealing. The individually bagged samples were consolidated in sealed larger polyweave bags or large containers and despatched to the laboratory for processing and assay.

11.3 SAMPLING BIAS

Considering the very good core sample recoveries, style of mineralisation being tested by this drilling, i.e. broad zones of mainly disseminated fine grained mineralisation and stock-work veins with little variation in grade, and the general fineness of the gold particles, the sampling intervals and splitting methods used are considered to be appropriate by AMA and would provide unbiased and representative results suitable for resource estimation modelling. Core recoveries in all holes drilled were generally excellent and there was no exceptional alteration, foliation, variations in rock hardness or brecciation that could be expected to introduce a bias in the drill sampling.

11.4 FACTORS IMPACTING THE ACCURACY OF RESULTS

No factor that would impact a fair collection of samples is apparent from the data available within the Project areas.

11.5 SAMPLE QUALITY

In the author's opinion, after reading the reports previously described in this document, the quality of historical sampling provides a reasonable basis to calculate resource estimates and for planning further investigations.

11.6 LABORATORY SAMPLE PREPARATION AND ANALYTICAL PROCEDURES

All drilling and trenching since 2004 has followed sampling, assaying, sample security and QA/QC protocols, (as described in Section 11.2), which allows resource estimation to the level of confidence implied by the Mineral Resource categories discussed in Section 14.

All the drill supervision, geological logging and sampling were carried out by trained geologists who were members of the operating company at the time of this work. The sample preparation and assaying was carried out by independent commercial certified laboratories as shown in Table 11-1.

Table 11-1: Laboratories used to analyse samples from the Project

Years	Laboratory
1994 - 1996	Bondar Clegg/SGS del Peru
2000	ALS Chemex
2003 - 2007	Acme Labs
2008	Chemex
2012 - Present	Acme Labs

The sample preparation methods and QA/QC procedures followed in earlier drill programs were previously described by Burns (Burns, 2005), who subsequently carried out an informal progress report on the QA/QC procedures followed by the exploration staff. Upon investigation, he concluded that the work programs in progress were being conducted in accordance with the requirements of NI 43-101, as illustrated by the following extract (AMA, 2014a):

> *"Regarding the current drill program, consulting firm SRK was commissioned in February 2013 by EGX to conduct a "Technical Due Diligence of the Drilling and analytical Quality Assurance/Quality Control" at Condor. They noted that many aspects of the exploration and data management practices and procedures are well documented in an internal company report, titled "A Guide for Exploration Activities: Sampling Protocol in exploration activities on the Project located in Zamora, Ecuador" (CMC, 2011) dated August of 2011.*
>
> *Overall SRK were of the opinion in their report that; "the drilling methods, logging, sampling, database maintenance, and chain of custody procedures are all consistent with or above industry standards".*
>
> *SRK were also of the opinion that; "EGX is currently conducting an adequate QA/QC program with appropriate protocol in place for monitoring of failures and control sample performance. The QA/QC supports resource estimation, but some of the identified deficiencies may cause resources to be classified at a lower level of confidence. SRK is of the opinion that the QA/QC tracking and reporting is not sufficient for a consistent Measured classification of mineral resources, and that significant attention should be devoted to improving this aspect of QA/QC both in the previous data as well as going forward. The major reasons behind this conclusion are:*
>
> 1. *Total absence of QA/QC for the early TVX drilling, which comprises a large percentage of the overall drilling database;*
> 2. *High failure rates for Au CRM in the 2004 to 2008 drilling; and*

3. *High failure rates for Cu CRM and blanks in the 2008 to Present drilling.*

Given that there has been recent drilling with QA/QC, which has offset and interspersed with the TVX drilling, SRK has confidence that the analytical results are being checked with QA/QC. With some simple recommendations detailed below, there is no reason to assume that the QA/QC database could not be used for Measured, Indicated, and Inferred resources for Condor, Santa Barbara, and El Hito."

The authors of this report concur with the above SRK conclusions.

11.7 DRILL CORE SAMPLING

11.7.1 1991 TO 2004 DRILLING

There is no information available on the QA/QC methodology employed prior to 2004. There are currently 19 drill holes in the Santa Barbara database from this time period and four in the El Hito database.

11.7.2 2004 TO 2007 DRILLING

The first Certified Reference Materials (CRMs or Standards) were used on the Project in 2004. There were however, problems with these standards and the QA/QC data for this period is unreliable. SRK recommended that a statistically significant portion of the samples collected and assayed from this period be re-assayed. Blanks and quarter core duplicate samples were also inserted in these sample batches. The assays for the blanks and duplicates were generally within the expected range. SRK did recommend check assays on some of the sample batches that included outlier assays, mainly Chinapintza samples that are not part of this PEA, to determine if the variant assays are due to a nugget effect or poor analyses. Overall the results of the blanks and duplicate assays are within the expected range indicating that the sampling and assaying meets the expected standard required for resource estimation.

11.7.3 2007 TO 2008 DRILLING

A total of 20 (8 %) of the inserted Standards assays failed to produce results within the Standard assay and +/- three standard deviations. It is apparent that in some (Sillietoe, 1998) of these cases there was mislabelling of Standards rather than defective assays. SRK concluded that since the actual assay failure rate of the Standards is very low, if the apparently mislabelled Standards are accounted for, this data is suitable for resource estimation. Blanks and quarter core duplicate samples were also inserted in these sample batches. The results of the blanks and duplicate assays are within the expected range indicating that the sampling and assaying meets the expected standard required for resource estimation.

11.7.4 2008 TO 2012 DRILLING

Drilling was suspended due the moratorium imposed by the Ecuadorian government.

11.7.5 2012 TO PRESENT DRILLING

EGX / Condormining has implemented a quality assurance and quality control program to ensure that the transport, sampling and analysis of all samples are conducted in accordance with the best possible practices. Drill core is transported from the drill by employees to a secure core logging facility at the Condormining camp where, after geotechnical and geological logging, it is marked for sampling. The entire hole is sampled. Core samples are generally 2 m in length, varying at geological contacts to between 1.5 m and 2.5 m. Core is split in half by EGX / Condormining employees. One half is retained in a secure storage facility and the other half is transported by EGX / Condormining employees or a bonded courier to Acme Labs' sample preparation facility in Cuenca, Ecuador where the core sample is crushed so that 80 % passes a 10 mesh screen and a 250 g split is pulverized so that 85 % passes a 200 mesh screen. From Cuenca Acme Labs ships the samples to their laboratory in Santiago, Chile for analysis. The laboratory is ISO/IEC 17025:2005 (CAN-P-4E) certified. Where appropriate, samples are analysed for gold by 30-gm fire assay with an AAS finish and by gravimetric methods for assays over 10 ppm. Samples are analysed for silver and copper by Inductively Coupled Plasma Emission Spectroscopy (ICP-ES) after a four acid digestion. For silver assays over 200 ppm, samples are analysed using Acme Labs 7AR method consisting of hot aqua regia digestion and ICP-ES analysis. Copper assays over 10 000 ppm are re-analyzed using four acid digestion with ICP-ES finish (AcmeLabs 7TD method). Acme Labs are independent from EGX.

Currently, EGX /Condormining submits three different types of control samples as a part of their QA/QC procedures:

1. Certified Reference Material (CRM) - (Pulp OREAS, CDN);
2. Blank - (Pulp OREAS); and
3. Quarter Core duplicate (20th sample).

These are inserted at a rate of one of each control sample every 20 samples.

Under EGX / Condormining's QA/QC procedures, samples are submitted for re-analysis based on their proximity to a certified reference standard that returns a value greater than three standard deviations higher or lower than the mean value for that standard. In addition, any two consecutive reference standards falling outside the two standard deviation threshold will be considered to have failed. Since every twentieth sample is a reference standard, ten samples above and below a failed standard will be re-analysed. The same protocol will be applied to duplicate samples considered to have unacceptably divergent gold values. Periodically, random samples will be submitted to another laboratory as an external check on the results provided by the primary lab.

Based on the foregoing protocol 178 samples (3 %) have been re-analysed for copper. 102 samples (1.53 %) have been re-analysed for gold. This represents 33 copper standard failures and 9 gold standard failures.

11.8 LABORATORY SAMPLE PREPARATION AND ANALYTICAL PROCEDURES

All of the exploration samples requiring chemical analysis are submitted to Acme Labs who have ISO17025 accreditation. Their standard analytical methods are listed below in Table 11-2.

Table 11-2: Analytical methods currently in used by Acme Labs on samples submitted from the Project

Analysis Code	Method	Description	Test Wt. (g)	Lab
R200-250	Sample Preparation	Crush, split and pulverize 250g rock to 200 mesh	-	Acme Labs – Cuenca
G6	Fire Assay	Lead Collection Fire - Assay Fusion - AAS Finish	30	Acme Labs – Santiago
G6Gr	Fire Assay (over limit)	Lead collection fire assay 30G fusion - Grav finish	30	Acme Labs - Santiago
1E	ICP	4-Acid digestion ICP-ES analysis	0.25	Acme Labs – Santiago
SAN Split Pulp	Sample Split	Analysis sample split/packet	-	Acme Labs - Santiago

No problems are apparent with the sampling and assays from the current era of drilling.

11.9 BULK DENSITY

Specific gravity measurements are currently taken roughly every 10 to 12 samples to determine the density of the various rock types encountered in the Project. EGX uses industry standard specific gravity data collection and calculation procedures. Samples are selected from cut core and are designated as specific gravity samples. They are dried in an oven to +/- 105°C, weighed dry, and then weighed submerged in water. In the case of porous samples, they are weighed dry, dipped in warm paraffin wax, weighed again, then weighed submerged in water. The wax's contribution to the mass is discounted from each measurement, and the calculation is the same for each method.

The bulk densities used for the different rock types in the AMA resource estimates are included in Table 11-3.

Table 11-3: Bulk Densities used in AMA resource estimates

Code	Rock Type	Bulk Densities (tonnes/m^3)
Vdd	Dacite	2.97
Vba	Basaltic andesite	2.83
Vad	Andesite dike	2.78
Sls	Sediments	2.63
PX	Phreatomagmatic breccia	2.47
IX	Intrusive breccia	2.91
Ird	Rhyolite	2.39
Igd	Granodiorite	2.6
Idi	Diorite	2.66
DP2	Diorite (hornblende>plagioclase)	2.67
DP1	Diorite (plagioclase>hornblende)	2.62
Db	Diabase	2.84

11.10 FACTORS IMPACTING THE ACCURACY OF RESULTS

No factors that would impact on the fair chemical analysis of samples is apparent from all the data available within the Project area and therefore this assay data is suitable for resource estimation to the level of accuracy implied by the resource categories used in Section 14.

SECTION 12 DATA VERIFICATION

12.1 QUALITY ASSURANCE AND QUALITY CONTROL PROGRAMS

Information in this section was obtained largely during the property visit by the author, Al Maynard of AMA, to the Project property site and subsequently, during discussions with EGX and Condormining personnel. It refers primarily to activity prior to acquisition of the property by ECC. The EGX QA/QC program has been described above and reviewed by SRK in their 2013 report, as described above in Section 11.6.

12.2 ASSAYS

Verification of previous results is largely dependent on reliance upon checks by past companies, from TVX to Ecometals and those of the author in the Ecometals database.

Previous checks include those by Pitard (1995) (16), and with respect to the TVX core, by Easdon (2004) and AMEC in 2004.

During the site visit, the primary author carried brief field examinations of the Los Cuyes, Soledad, and Enma prospect areas, and took 16 check samples considered to represent mineralisation therein. These sites or locations with depths (where drill core was taken), were surveyed using standard GARMIN GPS equipment. The condition of trenches made re-sampling difficult and the majority of check samples came from intercepts from drilling.

The samples, including one OREAS blank and one OREAS standard, were taken under the direct supervision of the primary author and transported under his supervision to Loja, for transport to the Acme Labs preparation facility in Cuenca (by courier). The facility was not inspected at this time. The samples were sent to Acme Labs in Vancouver, an accredited laboratory.

The samples were assayed for gold, silver, copper, lead and zinc. Samples are shown in Table 12-1. The author’s samples are shown in black, original samples in red. The results largely show similar grades for all elements, save DSO-12 results, with much higher gold grades. This could be attributed to erratic, possibly visible, gold in the sections.

Table 12-1: Analysis: GROUP 7AR - 1.000 GM SAMPLE, AQUA - REGIA (HCL-HNO3-H2O) DIGESTION TO 100 ML, ANALYSED BY ICP-ES.

Element/ Sample	Location		Cu* %/ppm	Pb* %/ppm	Zn* %/ppm	Ag** g/t	Au** g/t
1-2	Lab Blank		<.001	<.01	<.01	<2	<.01
2-2	Lab Blank		<.001	<.01	<.01	<2	<.01
3004880	DDH DCU-17B 128-130		0.084 0.111	1.36 1.65	3.73 5.42	126 339	12.58 5.46
3004881	DDH DCU-17B 204-206		0.163 0.183	5.05 2.24	7.07 4.82	121 86	27.86 13.96
3004882	DDH DCU-17B 220-222		0.102 0.123	0.15 0.14	1.18 0.81	44 36	10.18 6.35
3004883	DDH DCU-17B 386-388		0.023 0.026	0.01 0.0005	0.03 0.02	6 3	1.76 2.2
RE 3004883			0.023	0.01	0.03	6	1.38
3004884	DDH DSO-12 58-60		0.01 0.029	0.03 0.06	0.46 0.52	3 7	0.78 4.906
3004885	DDH DSO-12 74-76		0.015 0.017	0.08 0.11	1.35 1.3	6 7	1.36 5.351
3004886	DDH DSJ-003 42.5-45		0.006 NA	0.02 NA	0.28 0.27	4 4	6.23 5.55
3004887	DDH DSJ-003 60-62.5		0.042 NA	0.01 NA	0.49 0.55	14 13	3.52 2.99
3004888	DDH DCU-37 90-92		0.033 0.027	0.03 0.02	0.04 0.02	6 7	1.84 2.606
3004889	DDH DCU-37 150-152		0.015 0.012	0.03 0.03	0.75 0.7	9 8	1.8 6.247
3004890	DDH DCU-37 228-230		0.018 0.014	0.01 0.01	0.02 0.02	5 4	1.3 2.086
3004891 (pulp)	OREAS std 53P		0.411	<.01	<.01	<2	0.39
3004892*	Enma Trench Sample TGM 148 way pt 36	17m 770388 3551938	0.004 48	0.07 262	<.01 116	9 2.3	0.39 0.141
3004893*	Enma way pt 37 Mineralised Zone	17M 770523 9551948	0.104 NA	0.14 NA	1.11 NA	24 NA	0.59 NA
3004894 (pulp)	OREAS BLANK 22P		<.001	<.01	<.01	<2	0.01
3004877*	5 m chip at sample Trench TGM 204-205 (approx.)		0.019 124	0.06 35	0.41 371	8 3.3	0.7 3.46
3004878*	5 m chip at sample Trench TGM 204-205 (Approx)		0.047 76	0.06 49	0.6 530	20 4.3	2.24 3.54

Element/ Sample	Location		Cu* %/ppm	Pb* %/ppm	Zn* %/ppm	Ag** g/t	Au** g/t
3004879*	Platformes Grab sample over 3 m Trench TGM 340-342		0.005 0.03	<.01 0.0005	<.01 0.0005	<2 0.005	0.02 0.02
STANDARD R-3/OxK48			0.788	1.98	3.89	201	3.5

**Cu, Pb and Zn grades shown in red as ppm and in black as percentages.*

*** Ag and Au grades shown in red as g/t and black as percentages.*

From July 2007 through the end of Ecometals' involvement with the property in 2011, Ecometals QA/QC involved the use of OREAS standards and blanks. According to Ecometals personnel, for all surface and drill sampling, the following procedure was carried out:

Insertion of blank / 6 samples / Std / 7 samples / duplicate / 6 samples / blank

A random check of individual batches of assays indicates that this methodology was not strictly adhered to through the number of blanks and standards. Approximately 1 in 10 samples for both used a different methodology, but this is considered quite adequate.

OREAS blanks were used after July 2007. Between 2004 and 2007, mine waste material was used for some standards, but analysis of this indicated high variability and these are no longer used. Duplicates were made from the pulp reject and split (Acme Labs). Table 12-2 is a summary of the OREAS and ACME Standards used from 2004 to 2008.

TVX core was re-sampled by Goldmarca and also AMEC, in 2003-2004. The full report by the latter was unavailable at time of writing, although efforts are being made to source the document.

This re-sampling comprised 358 samples. 50 g samples were fire assayed for gold. Blanks, standards and duplicates were inserted "according to the Canadian norms of the QA/QC." (Easdon, 2004). Unfortunately, the methodology and values were unavailable at time of writing.

Commencing in the third quarter of 2007, Ecometals commenced work on its database validation, covering drill and trench location, down-hole surveys, assays and drill logs. This data was subsequently used for Micromine 3-D modelling.

By mid-2008, Ecometals had completed a QA/QC report on the 2004 to 2008 sampling from the various drill and trenching programs. Samples, batch numbers, and control samples (duplicates, blanks and standards), were classified and results analysed to determine variance and overall acceptability. This covered the assay results from all the analytical laboratories used, Acme Labs Canada, Acme Labs Chile and Chemex Laboratory Canada.

A summary of the samples, blanks and hole statistics is presented in Table 12-3.

In summary, a check on the Ecometals database and sampling protocols indicates the following:

1. The ACME standards used for gold are generally high relative to the overall average gold results. The author would recommend a lower gold standard, around 1-2 ppm Au to be used for future work.
2. The ACME Blank results are considered acceptable with a 98 percentile of less than 0.02 ppm Au and a standard deviation of 0.12064. For Ag, there is an overall standard deviation for blanks of 0.4645; for copper, 0.000236; lead, 0.00075243; and zinc, 0.004728. All lie within acceptable limits.
3. Eight percent of the gold and silver results from 42 714 samples in the Ecometals database were checked for mismatches. No mismatch was found.
4. During the property visit, the author compared digital results from the assayers, Acme Labs, and those obtained directly from the laboratories. A check of 10 % of the assays indicated no difference in numbers between actual results and the Ecometals database.
5. The author could not establish precise QA/QC protocols for 2004 to third quarter 2007 sampling. It is understood that laboratory standards were inserted approximately every 15 samples with blanks every 10 to 15 samples. It is also understood that some of these internal standards were mineralised or otherwise, taken from the Project property. A check on assay variance by Ecometals indicated they were unsuitable and subsequently were no longer used.
6. From late summer 2007, new sampling protocols and QA/QC procedures were implemented. Blanks and standards were obtained from OREAS, Australia. No Project property material was processed.
7. OREAS standards were stored in the secured company warehouse. Access was restricted and the geological personnel requesting material were registered. A chain of custody existed to ensure all handling was monitored. Individual OREAS samples were put in appropriate sample bags with sample numbers under the supervision of the geologist but not handled by him to ensure there was no tampering by such personnel.
8. The 2008 QA/QC study indicates that from a population of 42 samples obtained from 2004 to 2007 trenching and drilling, standard deviations for coarse reject (duplicates) and for original samples is 1.584 and 1.587. This was attributed to either poor mixing of the sample or improper splitting.

9. Of the 671 blanks used since 2003, for gold, 64 samples (9.54 %), returned values >0.01 ppm Au. These Acme Labs blanks yielded five samples greater than one standard deviation and four samples greater than two standard deviations. OREAS blanks all passed 'tests' of one, two and three standard deviations. It was concluded that the Acme Labs Chile and Acme Labs Canada laboratories did suffer from contamination problems at the time. As a consequence, batch A570157 should be considered for exclusion from the database. Results from this batch are considered to be sufficiently anomalous to warrant exclusion, particularly as the associated standards also failed. While other batches contained 'high' blank gold numbers, the related samples are deemed to have no material impact on overall results.
10. The accuracy of 2004 to 2007 sampling was assessed by simple statistical calculations on the total number of samples and related batches plus the four gold standards used during that time. The findings indicate between 9 % and 16 % of samples exceed a one standard deviation control. As such, the inclusion of these gold results in any resource calculation and as 'guides' for future drill programs should be reviewed immediately.
11. QA/QC performed on the samples using OREAS standards indicated that in general this data set is robust; however, the findings indicated several batches would warrant re-sampling on the basis of failure at one and two standard deviations, particularly when taking into account specific relatively high gold values within each subset. This would include batches A22792, A770375, A770308, A770282, A770349, A770299, A770351, A770381, and A770428.

Table 12-2: OREAS Standards Data and the Condor Blended Standards Material (Not Used From July 2007)

Certified Reference Material (CRM) Standards							
Manufacturer	Reference No	Matrix	Au (g/t)	Au tolerance (g/t)	Au variance (%)	Cu %	Ag (g/t)
OREAS	15PC	basalt	1.61	0.001	0.06	-	-
OREAS	53P	qz monzonite porphyry	0.38	0.004	1.05	0.413	-
OREAS	53PB	Ditto	0.623	0.012	1.93	0.546	-
OREAS	2PD	sediment	0.885	0.015	1.69	-	-
OREAS	22P	quartz sand (blank)	<0.002			-	-
OREAS	62PA	epithermal meta-andesite	9.64	0.03	0.31	-	18.4
OREAS	15PA	basalt	1.02	0.03	2.94	-	-
OREAS	17PB	basalt	2.56	0.02	0.78	-	-
OREAS	61PA	andesite	4.46	0.02	0.45	-	8.54
OREAS	18PB	basalt	3.63	0.02	0.55	-	-

Certified Reference Material (CRM) Standards							
Manufacturer	**Reference No**	**Matrix**	**Au (g/t)**	**Au tolerance (g/t)**	**Au variance (%)**	**Cu %**	**Ag (g/t)**
OREAS	7PB	sediment	2.77	0.02	0.72	-	-
Inspectorate Services Peru	GEO-184 STD-1	Condor blended ore	1.05	0.11	10.48		
Inspectorate Services Peru	GEO-269 STD-2	Condor blended ore	2.23	0.21	9.42		
Inspectorate Services Peru	GEO-273 STD-3	Ditto	3.19	0.3	9.40		
Inspectorate Services Peru	GEO-309 STD-4	Ditto	3.82	0.13	3.40		

Table 12-3: QA/QC Table Summary

<table>
<tr><th colspan="8">Analytical QC Report</th></tr>
<tr><th colspan="8">October 2004-March 2007</th></tr>
<tr><td>Drilling number</td><td colspan="2">Control samples number</td><td>Control samples number total</td><td>%</td><td>Samples Number</td><td>Batches Number</td><td>Diamond drilling (m)</td></tr>
<tr><td rowspan="3">113</td><td colspan="2">473 Blanks</td><td rowspan="3">1284</td><td rowspan="3">13.50</td><td rowspan="3">9 511</td><td rowspan="3">235</td><td rowspan="3">20 002.9</td></tr>
<tr><td colspan="2">297 Duplicates</td></tr>
<tr><td colspan="2">514 Standards</td></tr>
<tr><th colspan="8">March 2007 - 30 September 2007</th></tr>
<tr><td>Drilling number</td><td colspan="2">Control samples number</td><td>Control samples number total</td><td>%</td><td>Samples Number</td><td>Batches Number</td><td>Diamond drilling (m)</td></tr>
<tr><td rowspan="3">30</td><td>Before OREAS</td><td>109 Blanks</td><td rowspan="3">665</td><td rowspan="3">14.88</td><td rowspan="3">4 468</td><td rowspan="3">43</td><td rowspan="3">8 814.7</td></tr>
<tr><td>and</td><td>202 Duplicates</td></tr>
<tr><td>After OREAS</td><td>354 Standards</td></tr>
<tr><td rowspan="3">17</td><td rowspan="3">Before OREAS</td><td>109 Blanks</td><td rowspan="3">309</td><td rowspan="3">15.52</td><td rowspan="3">1 991</td><td rowspan="3">21</td><td rowspan="3">3 778</td></tr>
<tr><td>98 Duplicates</td></tr>
<tr><td>102 Standards</td></tr>
<tr><td>13</td><td>After OREAS</td><td>104 Duplicates</td><td>356</td><td>14.37</td><td>2477</td><td>22</td><td>5 036.7</td></tr>
</table>

Analytical QC Report							
		252 Standards					
1 October 2007 - 2011							
Drilling number	Control samples number		Control samples number total	%	Samples Number	Batches Number	Diamond drilling (m)
9	After OREAS	13 Duplicates	163	9.06	1 799	21	3 778.02
		150 Standards					

12.3 SPECIFY GRAVITY DATA

For historical bulk density results this PEA uses the specific gravity data from the previous 2004 Technical Report (Easdon, 2004), extracted as follows:

"13.5 Specific Gravity: TVX performed 16 Specific Gravity (SG) tests in the field using standard techniques whereby the samples were weighed dry and the weight divided by the dry weight less its weight when submerged in water. Each sample was broken into three pieces and the process repeated 3 times and the average SG was the value accepted. Subsequently, TVX submitted 118 samples to SGS for specific gravity measurements. SGS generated values for: apparent specific gravity, true specific gravity, porosity of the sample and the volume of pore space. These tests were conducted using wax to coat the samples prior to submerging the sample in water. The SG tests were conducted on a variety of the rocks typically found in the Northern Secror and include porphyritic rhyodacite, rhyolite breccia, hydrothermal breccia, grandiorite and dacite. TVX concluded that I could use an average SG of 2.66 for the material contained within the various breccia bodies. TVX field calculated SGs from the (whole) core that was derived in their 1999 drilling program at Santa Barbara. They state that these tests returned an average SG of 2.7. No mention is made of the techniques used to derive this figure."

Specific gravity measurements are conducted routinely as part of the current EGX drill program. The bulk densities used for the different rock types in the AMA resource estimates are included in Table 11-3.

12.4 SUMMARY AND INTERPRETATION

For additional summary and interpretation of drilling results, see 14.3.

Overall, there is a less than 5 % failure rate for the total database, indicating as a whole, the database is fairly robust. It is recommended that re-sampling be carried out on samples within the abovementioned batches, plus a review of high gold results from the 2004 to 2007 drilling, with checks run on samples and bracketing using OREAS standards. In conclusion, post July 2007 QA/QC protocols match or exceed industry standards.

SECTION 13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 2008 CYANIDATION TEST

Metallurgical tests on representative mineralised material from mineralised occurrences in the Project have been conducted at several times. The 2008 tests included cyanide bottle roll gold extraction on crushed samples without grinding to simulate what may be expected in a heap leach environment by G & T Metallurgical Services Limited in Canada who stated (AMA, 2014a) that:

- *"The low grade samples, (less than 0.3 g/t gold) on average, leached very poorly. On average, 10 percent of the gold and 6 percent of the silver was extracted to solution.*
- *The medium grade composites, (0.3 to 1.0 g/t gold) demonstrated a considerable improvement in gold and leach performance. On average, 48 and 17 percent of the gold and silver were extracted from the feed.*
- *As the feed grade increased beyond 1 g/t, the leach performance improved to about 58 percent gold and 20 percent silver extraction.*
- *Of the variables investigated, gold feed grade had a marginal effect on leaching performance. Provided the samples have sufficient gold, the maximum gold extraction rate reached a plateau of about 60 percent. No correlations between gold leaching performance and sulphur feed grade were identified.*
- *Conventional grinding and carbon in pulp testing should also be considered as a means to further increase gold extraction rates, albeit at higher processing costs.*
- *Lime and cyanide consumptions were relatively low, averaging 0.8 and 0.5 kg/tonne, respectively. There was no apparent relationship between consumption and gold or sulphur feed content"*

13.2 TEST SAMPLES AND REPRESENTIVITY

Samples for metallurgical test work were selected from drill core collected from recent drilling of the Deposit. The core samples consisted of ¼ core splits from intervals already assayed during the standard core drilling, splitting, sampling and assaying reported in Section 12.

Sample selection is shown in Table 13-1.

Table 13-1: Metallurgical Sample Selection

Sample #	Drill Hole	From (m)	To (m)	Interval (m)	Au g/t	Cu %	Rock Type
120497	DSB-019	300	312	12	0.26	0.07	Diorite Porphyry
120498	DSB-021	188	200	12	1.30	0.13	Basaltic Andesite
120499	DSB-023	540	552	12	1.20	0.15	Basaltic Andesite
120500	DSB-022	438	450	12	0.66	0.10	Basaltic Andesite
120501	DSB-030	650	662	12	0.61	0.14	Basaltic Andesite
120502	DSB-027	288	300	12	0.33	0.09	Basaltic Andesite
120503	DSB-034	342	354	12	0.36	0.09	Basaltic Andesite

Specific drill holes were selected to cover the known Deposit both spatially and by depth as well as specific grade ranges and rock types for metallurgical testing. A sample interval of 12 m was selected to obtain approximately 30 kg for each sample to meet the expected requirements for metallurgical testing. The resulting samples and composites were therefore as representative as possible of the entire deposit given the relatively limited amount of drilling completed to date and the overall size of the deposit. The sample selection and resulting composites are considered adequate for this phase of project testing and reporting.

13.3 METALLURGICAL TEST PROGRAM AND PROCEDURES

13.3.1 COMPOSITE PREPARATION

The ¼ core samples were shipped to Phillips Enterprises, LLC in Golden, Colorado, USA for metallurgical testing. The samples were dried and weighed. The individual samples were then crushed to 3.66 mm (6 mesh), blended, sampled and submitted for assay. Results of this work are presented in Table 13-2.

Table 13-2: Core Sample Identification and Analyses

Sample #	Dry Weight (kg)	Analyses	
		Gold (g/t)	Copper (%)
120497	17.7	0.3	0.074
120498	24.9	1.2	0.119
120499	29.5	1.2	0.149
120500	30.8	0.8	0.101
120501	16.3	0.5	0.144
120502	29	0.5	0.101
120503	24.7	0.4	0.105

Based on the gold analysis of the metallurgical samples, four composites were prepared for the metallurgical test work. The composites were assembled to create high grade, average grade, and low grade composites of basaltic andesite and a low grade sample of diorite porphyry. Table 13-3 provides the components and the resulting analyses of each composite.

Table 13-3: Composites and Analyses

Composite	Core Samples			Analyses			Rock Type
	ID	% wt	Kg	Gold (g/t)	Silver (1) (g/t)	Copper (%)	
1	120498	50	23.9				Basaltic Andesite (high grade)
	120499	50	23.9				
	Total	100	47.8	1.1	2	0.136	
2	120500	50	15.3				Basaltic Andesite (average grade)
	120501	50	15.3				
	Total	100	30.6	0.7	2	0.128	
3	120502	50	23.7				Basaltic Andesite (low grade)
	120503	50	23.7				
	Total	100	47.4	0.4	2	0.104	
4	120497	100	16.7				Diorite Porphyry
	Total	100	16.7	0.3	<1	0.074	

(1) by atomic absorption analysis

The four composites were then used in subsequent testing for grindability, mineralogy, cyanidation and flotation testing.

13.3.2 GRINDABILITY TESTS

Crusher work indices, autogenous mill indices, and rod mill indices could not be determined for the samples as a result of the fine nature of the material after ¼ core splitting and crushing for analytical purposes. Ball mill indices were determined to provide a relative indication of material hardness and grinding power requirements.

Bond ball mill grindability tests were performed on Composites 2 and 3 to determine ball mill work indices on the basaltic andesite rock type. Observation of the samples and their crushing and grinding characteristics during sample preparation indicated that the samples were quite competent and hard. Bond ball mill grinding index determinations confirmed the observed characteristics. Table 13-4 presents the results of the grindability testing.

Table 13-4: Ball Mill Grindability Tests and Power Estimates

		Closing Screen	Feed	Product	Ball Mill Work Indices	
Test	Composite	Microns (mesh)	P_{80} microns	P_{80} microns	kW-h/mt	kW-h/st
1	Composite 2	150 (100)	2 520	102	24.97	22.65
2	Composite 3	150 (100)	2 560	100	22.07	20.02

13.3.3 DIAGNOSTIC LEACH TESTS

Resource Development Inc. (RDI) was contracted to perform diagnostic leach test work to determine the response of the sample to cyanidation under various conditions and treatments to determine the gold occurrence and associations.

A portion of sample 120499, high grade diorite porphyry, was provided to RDI. The sample was blended and split into a 1 kg charge for testing. The sample assay from previous analytical work was reported to be 1.2 g/t gold. The calculated head from the leach test was 1.27 g/t gold.

The test charge was stage ground to 74 microns (200 mesh) in the laboratory rod mill. The ground pulp was filtered and the wet cake split into two parts for diagnostic leach tests. The following leach tests were performed:

1. Leach Test No. 1 was conducted with carbon-in-leach conditions to evaluate the potential for preg-robbing conditions.
2. Leach Test No. 2 was performed on a separate split of the sample as a standard cyanide leach test.
3. Leach Test No. 3 was designed to evaluate if gold was tied up with arsenopyrite. The residue from Leach Test No. 2 was roasted under reducing conditions for four hours at 425°C and then leached.
4. Leach Test No. 4 was intended to evaluate the possibility of gold being tied-up with pyrite. The residue from Leach Test No. 3 was roasted under oxidizing conditions for four hours at 625°C and then leached.

All the leach tests were performed with the following conditions:

1. Slurry density was maintained at 40 % solids.
2. Sodium cyanide level was maintained at 2 grams per litre.
3. The pH was maintained above 11.0 with hydrated lime.
4. The tests were run for 48 hours.

The test results are summarized in Table 13-5.

Table 13-5: Results of Diagnostic Leach Tests

Test No.	Condition	Gold Extraction		
		Unit	Additional	Overall
LT-1	CIL	85.4	-	85.4
LT-2	Standard Leach	79.2	-	79.2
LT-3	Roast Residue LT-2, 425°C, Std. Leach	8.7	1.8	81.0
LT-4	Roast Residue LT-3, 625°C, Std. Leach	64.2	12.2	93.2

The test results indicate the following:

- The difference between carbon-in-leach and standard cyanidation leach was 6.2 % in gold extraction. This indicates the sample has moderate preg-robbing properties associated with it (Tests LT-1 and LT-2).
- Cyanidation following reducing roast at 425°C extracted 8.7 % of the remaining gold in the residue from the standard leach test (LT-2). This is equivalent to 1.8 % of the gold in the feed which is associated with arsenopyrite – a minor amount.
- Cyanidation following oxidizing roast at 625°C on the LT-3 leach residue resulted in 64.2 % of the remaining gold in the sample. This is equivalent to 12.2 % of the gold in the feed associated with pyrite.

The results indicate that carbon-in-leach will extract 85.4 % of the gold. An additional 14 % of the gold can be extracted if the feed is oxidized at 625°C prior to leaching since it is associated with sulfides. Very fine grinding could be evaluated to see if all or a portion of the gold associated with pyrite could be leached without oxidation.

13.3.4 WHOLE ORE CYANIDATION TESTS

Two programs of whole ore cyanidation tests were performed on the composites. Phillips conducted stirred tank tests on the four composites described previously under varying conditions of grind size and cyanide solution concentration. Leaching kinetic data was also collected.

A second program was conducted by RDi to investigate coarse ore leaching and methods to reduce reagent consumption on ground ore. These tests were performed in rolling bottles.

Table 13-6 presents a summary of the whole ore cyanidation leach test conditions and results.

Table 13-6: Cyanidation Test Work Summary

Test No.	Composite	Product	Grind Target	Test Duration	Head Grade			Cyanide Strength(1) g/L	Reagent Consumption(2)		Total NaCN added to test kg/t	Metal Recovery	
			P_{80} microns	Hours	Source	Au g/t	Ag g/t		Lime kg/t	NaCN kg/t		Au %	Ag %
1	Composite 1	Whole ore	106	48	Calculated	1.34	6.3	2	0.31	2.07	4.37	85	5
					Analysed	1.1	2						
2	Composite 2	Whole ore	106	48	Calculated	0.70	6	2	0.18	2.45	4.40	86	5
					Analysed	0.7	2						
6	Composite 2	Whole ore	106	48	Calculated	0.87	2.5	1	0.14	1.75	2.50	88	21
7	Composite 2	Whole ore	150	48	Calculated	0.65	2.3	1	0.06	1.39	2.25	69	13
8	Composite 2	Whole ore	150	48	Calculated	0.91	2.3	2	0.06	2.56	4.32	78	12
3	Composite 3	Whole ore	106	48	Calculated	0.65	4.2	2	0.25	2.13	4.37	85	5
					Analysed	0.4	2						
4	Composite 4	Whole ore	106	48	Calculated	0.6	2.3	2	0.25	2.1	4.34	83	12
					Analysed	0.3	1						
5	Composite 1	Rougher	106	48	Calculated	0.59	2.3	1	0.32	1.94	2.94	66	13
		flot tails			Analysed	0.5	<3						

Notes:

(1) Maintained throughout test

(2) At end of test

Four baseline cyanidation tests were performed, one on each composite ground to P_{80} = 106 microns with free cyanide was maintained at 2 grams/litre for the 48 hour test period. The pH was maintained between 10.6 and 11.1 throughout the test period with hydrated lime. These tests are Test Nos. 1, 2, 3, and 4. Gold extraction at 48 hours was in the region of 85 % for all composites. Silver extraction was below 15 %. Copper dissolution was not monitored. Cyanide consumption was approximately 2 kg/t for all tests with the test for Composite 2 slightly higher at about 2.5 kg/t. Lime addition rate was 0.2 to 0.3 kg/t for all tests.

The average grade Composite 2 was chosen to investigate the effect of a coarser grind size. Test No. 8 was performed using the same conditions as the baseline tests except the grind size was P_{80} = 150 microns. The coarser grind size reduced gold extraction by 8 % to 78 % extraction.

Composite 2 was also used to investigate the effect of reduced cyanide concentration on gold extraction and cyanide consumption. Tests 6 and 7 were both conducted at 1 g/L cyanide solution concentration and particle size P_{80} = 106 and 150 microns respectively.

By reducing the free cyanide concentration from 2 g/L to 1 g/L on Composite 2, the following results were observed:

- No significant change (~2 %) of gold extraction at a grind of P_{80} = 106 microns.
- A 9 % extraction decrease at a grind of P_{80} = 150 microns.
- A decrease in cyanide consumption of 1.17 kg/t at a grind of 150 microns.
- A decrease in cyanide consumption of 0.70 kg/t at a grind of 106 microns.

From the cyanidation tests, the following conclusions can be made:

- Gold extraction is sensitive to grind size based on the response of Composite 2. Finer grinding will permit higher extraction. It is not known if extractions will be enhanced by grinding below P_{80} = 106 microns.
- Cyanide solution strength does affect the consumption. Consumption is reduced by using lower strength solution. The magnitude of the reduction may be related to grind size. Reduction appears to be greatest at finer grinds. Cyanidation tests with free cyanide strengths maintained below 1 g/L were not conducted.
- Gold dissolves rapidly in the presence of cyanide with maximum extraction occurring between 12 and 24 hours residence time.
- Silver content in the Santa Barbara material is low and silver extraction of approximately 20 % can be expected.

13.3.4.1 RDI PROGRAM

RDI received individual composite samples left over from the Phillips testwork for further study. The testwork consisted of sample preparation and characterization, grinding study, whole ore leach, flotation and leaching of flotation concentrate and tailings.

RDI prepared a new composite from existing samples. The RDI composite weighing 64.2 kgs was prepared while retaining 2 kgs of each of the original composites in reserve. The proportion of each sample used for the composite is given in Table 13-7

Table 13-7: Make up of RDI Composite Sample

Previous Sample	New Composite kg	Reserved kg
Comp 1	9.1	2.0
Comp 2	4.5	2.0
Comp 3	27.0	2.0
Comp 4	7.0	2.0
120500	11.6	2.0
120502	3.3	2.0
120499	1.7	2.0
Total	64.2	-

Since the samples were already crushed to nominal 3.66 mm (6 mesh), the composite sample was thoroughly blended and split into 1 kg and 10 kg charges. A 1 kg charge was pulverized to 104 microns (150 mesh), blended and samples of 50 grams each were split out for head analyses. The samples were submitted for sequential copper analyses, forms of sulfur, gold and ICP-ES analyses.

The test data are summarized in Table 13-8 and Table 13-9. The test results indicate the following:

- The composite sample assayed 0.52 g/t Au, 851 ppm Cu and 0.62 %S_{Total}.
- Sequential copper analyses indicated that 4.6 % of the copper was acid soluble (oxide), 4.0 % of the copper was cyanide soluble (secondary) and the remaining 91.4 % of the copper was primarily chalcopyrite.
- Approximately 69.4 % of the sulfur occurs as sulfide sulfur.

Table 13-8: Head Analyses of RDi Composite Sample

Element	Assay
Au, g/t: Assay #1 Assay #2 Average	 0.518 0.521 0.520
Cu_{Total}, ppm	851
$Cu_{Acid\ sol}$, ppm	39
$Cu_{CN\ sol}$, ppm	34
S_{Total}, %	0.62
$S_{Sulfide}$, %	0.43
$S_{Sulfate,}$ %	0.19
Percent Sulfide Sulfur	69.4
Copper Oxides, % of Cu_{Total}	4.6
Secondary Copper, % of Cu_{Total}	4.0
Primary Copper, % of Cu_{Total}	91.4

Table 13-9: ICP Analyses of Composite Sample

Element	Assay
%	
Al	5.37
Ca	5.32
Fe	6.89
K	0.90
Mg	4.22
Na	0.81
Ti	0.70
ppm	
As	12
Ba	162
Bi	<10
Cd	12
Co	42
Cr	301
Cu	765
Mn	1 297
Mo	9

Element	Assay
Ni	236
Pb	<10
Sr	195
W	<10
Zn	139

Bottle roll cyanidation tests were performed on whole ore composite samples at five different grind sizes to evaluate the effect of process parameters on gold extraction. The process parameters examined were grind size and leach time.

A 1 kg sample was ground to the required particle size and slurried with water to a density of 40 % solids. The sample, except at a coarse grind of 3.66 mm (6 mesh), was air sparged under agitation for 45 minutes since the dissolved oxygen in the bottle roll at start was very low. The slurried sample was adjusted to a pH of 11 with lime and conditioned for four hours before adding 1 g/L of NaCN. The slurry was then bottle rolled for 49 hours. Kinetic samples were taken at 4, 24 and 49 hours. The cyanide was allowed to decay to 0.2 g/L before adjusting it back to 1g/L of NaCN. However, at 4 and 24 hours, the pH of the slurry was adjusted to 11 with lime. After 49 hours, the sample was filtered and the test residue was thoroughly washed and dried. The dried residue was pulverized and assayed for gold and copper.

The results are summarised in Table 13-10 and Table 13-11. The test results indicate the following:

- A maximum gold extraction of 42.9 % was achieved at a crush size of P_{80} of 3.66 mm (6 mesh).
- The gold extraction is size dependent. The finer the grind, the higher the gold extraction.
- Due to the low grade of the ore, the pregnant solution analyses are also low and range from 0.04 to 0.23 ppm. A 0.01 ppm assay difference which is within analytical error is equivalent to 3 % of gold recovery.
- Gold extraction was 76.6 % at P_{80} of 104 microns (150 mesh) and was 86.7 % at P_{80} of 74 microns (200 mesh). The cyanide consumption was 0.480 kg/t and 0.419 kg/t respectively at the two grind sizes for 24 hour leach time. The cyanidation leach kinetics are fast and the majority of the gold is extracted in 24 hours.

Table 13-10: Summary of Cyanidation Leach Test at P_{80} of 3.66 mm (6 Mesh) Grind

	Test 1	
Parameters	**Au**	**Cu**
Extraction %		
4 h	20.0	0.8
26 h	13.2	2.1
48 h	39.6	3.2
72 h	42.9	4.2
96 h	26.5	5.5
Residue, g/t	0.333	1 096
Cal. Feed, g/t	0.465	1 162
Reagent Consumption, kg/t		
NaCN	0.541	
Lime	0.472	

Table 13-11: Summary of Cyanidation Leach Results

Parameters	Primary Grind, P_{80} mm (Mesh)							
	208 microns 65 Mesh (Test 2)		147 microns 100 Mesh (Test 3)		104 microns 150 Mesh (Test 4)		74 microns 200 Mesh (Test 5)	
Extraction %	**Au**	**Cu**	**Au**	**Cu**	**Au**	**Cu**	**Au**	**Cu**
4 h	49.2	1.7	55.0	1.7	64.8	2.0	68.8	2.1
24 h	67.6	3.6	60.4	3.8	76.6	4.6	86.7	4.7
49 h	70.7	5.2	57.7	5.6	76.5	7.2	80.7	7.3
Residue, g/t	0.137	842	0.226	828	0.110	820	0.086	854
Cal. Feed, g/t	0.500	889	0.559	878	0.520	885	0.513	922
Reagent Consumption, kg/t								
NaCN	0.479		0.358		0.601		0.600	
Lime	0.603		0.704		0.702		0.703	

Based on the testwork, the following conclusions were made:

- The composite sample, prepared from earlier composites, assayed 0.52 g/t Au and 851 ppm Cu.
- Majority of the copper was present as primary copper, mainly chalcopyrite (91.4 %) with minor amounts of oxide copper (4.6 %) and secondary copper (4 %).
- Sulfide sulfur accounted for 69.4 % of the total sulfur in the composite sample.
- Copper dissolution in the cyanidation tests was low.

- Preconditioning for four hours with lime and aeration was successful in reducing cyanide requirements for leaching.
- Whole ore cyanidation leach tests indicate that 42.9 % of gold was extracted at P_{80} of 6 mesh. Hence, heap leach of the ore may not be a viable option.
- The finer the grind, the higher gold extraction in the bottle roll cyanidation tests. Approximately 86.7 % of gold was extracted in 24 hours at a P_{80} of 74 microns (200 mesh).

13.3.5 FLOTATION TESTS SUMMARY

13.3.5.1 PHILLIPS AND RDI INITIAL TESTS

A series of flotation tests were performed both by Phillips Enterprises and RDI in an effort to recover copper as well as gold and produce a saleable concentrate, thereby realizing additional revenue from copper. The effect of grind size, flotation time, various collectors, promoters and frothers as well as the effect of pH were investigated. Bulk rougher-scavenger flotation tests were performed using the best conditions from bench tests to produce enough concentrate to perform flotation cleaner tests.

Rougher-scavenger flotation results were below expectations with gold recovery ranging from about 50 % to 65 % and copper recovery ranging from 60 % to 75 % in a concentrate representing about 10 % of the feed mass.

Cleaner flotation tests were less encouraging with the production of a saleable copper concentrate (greater than 25 % copper) proving to be unattainable despite regrinding of the rougher concentrate to as fine as P_{80} of 28 microns. Gold and copper recovery into a low grade third cleaner concentrate was 15 % and 20 % respectively.

The very fine grained nature of the Santa Barbara material and the locking of valuable minerals prevented liberation at any reasonable grind size and resulted in the poor flotation performance. Mineralogical examination of Composites 1 to 4 and selected flotation products were examined using optical and QEMSCAN technology and results are reported in Section 13.4.

13.3.5.2 RDI CONFIRMATORY TESTS

An additional sample of Santa Barbara material was obtained from site and sent to RDI for flotation testing with the objective of maximizing gold and copper recovery into rougher flotation concentrate. The sample was composed of drill core assay rejects crushed to approximately 6 mesh (3.66 mm). The blended and prepared head sample assayed 0.892 g/t Au, 0.122 % Cu, and 2.4 g/t Ag.

Seventeen rougher flotation tests were performed at various grind sizes, flotation times and reagent suites. Grinding finer than 200 mesh (74 microns) did not show benefits to recovery so the tests on various reagent schemes were all performed at the 200 mesh grind size. The best performance resulted from a reagent suite consisting of potassium amyl xanthate (PAX), AP-208 and methylisobutal carbonal (MIBC) and pH adjustment with sulphuric acid. The resulting recoveries were 65.3 % Au, 80.6 % Cu, and 70.7 % Ag into a rougher concentrate mass of 10.9 % of the feed.

Further treatment of the rougher concentrate to produce saleable products was beyond the scope of this study. Future project development work should use this pre-concentration step as the basis for more detailed metallurgical investigations.

13.4 MINERALOGY

Detailed mineralogy of the Santa Barbara material was not performed in the metallurgical program. Selected samples were examined to help understand some of the metallurgical results, particularly the difficulty with flotation as a recovery process. Optical mineralogy including scanning electron microscopy (SEM) equipped with an energy dispersive system (EDS) was performed on flotation tailings from Composite 1. QEMSCAN bright phase searches were performed on Composites 1 to 4 and flotation tailings from Composite 1.

13.4.1 OPTICAL MINERALOGY ON FLOTATION TAILINGS

The purpose of undertaking optical mineralogy testwork on the flotation tailings was to determine bulk mineralogy, grain size of sulfides, degree of sulfide liberation and gold occurrence in scavenger flotation tails from Composite 1. The sample was prepared as a standard polished thin section for study by transmitted/reflected light microscopy and SEM equipped with an EDS. Colour photomicrographs of relevant features are shown in Figure 13-1 through Figure 13-7.

13.4.1.1 GANGUE MINERALOGY

In thin section, the dominant component is green amphibole. The amphibole occurs as stubby cleavage fragments that vary greatly in size from 1 micron up to 100 microns. Many of the fragments show mild alteration to chlorite. Chlorite is also present as small, thin plates and aggregates. Quartz occurs as liberated, angular fragments up to 100 microns. Low amounts of cloudy plagioclase and potassium feldspar are present with a grain size that does not exceed 50 microns. Oxides are present in trace amounts with magnetite as the primary type. Magnetite occurs as both liberated fragments and as small inclusions in amphibole and quartz with a grain size up to 50 microns. SEM/EDS indicates a small population of the magnetite carries low levels of titanium and chromium. Ilmenite is present as liberated fragments and inclusions in silicates with a grain size up to 50 microns. Ilmenite shows some decay to leucoxene and is commonly associated with small grains of dark rutile. Calcite generally occurs as small, liberated grains up to 25 microns as well as fine grained aggregates with chlorite.

13.4.1.2 SULFIDE MINERALOGY

Sulfide content accounts for less than 1 % of the total sample volume. Approximately 20 % of the total shows complete liberation. Chalcopyrite is the dominant type and occurs primarily as small inclusions in amphibole and quartz with a grain size of 0.5 microns up to approximately 25 microns. Rarely, some chalcopyrite occurs as small inclusions or attachments to ilmenite and magnetite. Pyrite occurs as small cubes and irregularly shaped grains that have a similar grain size to chalcopyrite and are generally seen as inclusions in silicate mineralogy. Other trace sulfides include galena and molybdenite. SEM/EDS identified a nickel iron sulfide that may represent pentlandite. Galena, molybdenite and pentlandite are only seen as inclusions with a grain size that does not exceed two microns.

13.4.1.3 GOLD MINERALOGY

SEM/EDS analysis of individual silicate and sulfide grains for gold detection was performed using backscatter imaging at magnifications ranging from 1 000 X to 50 000 X with a beam voltage of 25kV. Extensive image and chemical analysis of several hundred silicate and residual sulfide grains failed to detect gold in any form. Table 13-12 outlines individual phases and their percentages based on petrographic and SEM/EDS analyses.

Table 13-12: Minerals and their Relative Abundance

Mineral	Abundance %
Amphibole	55
Chlorite	27
Quartz	9
Calcite	6
Plagioclase	1
K-Feldspar	1
Magnetite/ilmenite	1
Trace mineralogy: Pyrite, Chalcopyrite, Galena, Iron Oxide, Pentlandite, Zircon, Molybdenite, Rutile	

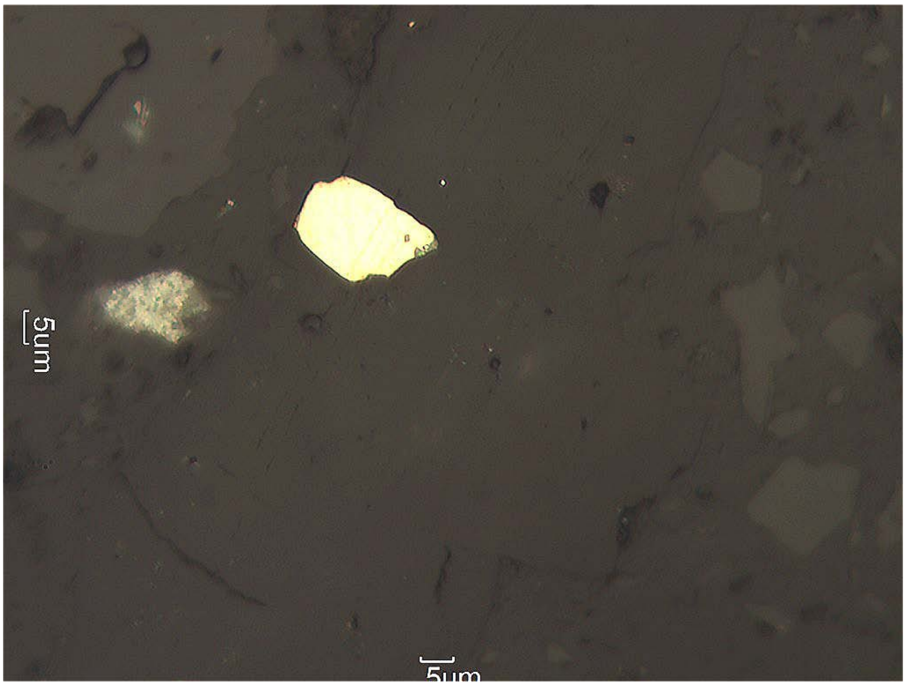

Figure 13-1: Photo 1, Scavenger Flotation Tails from Composite 1. Small chalcopyrite grain in quartz. Reflected light- 500X

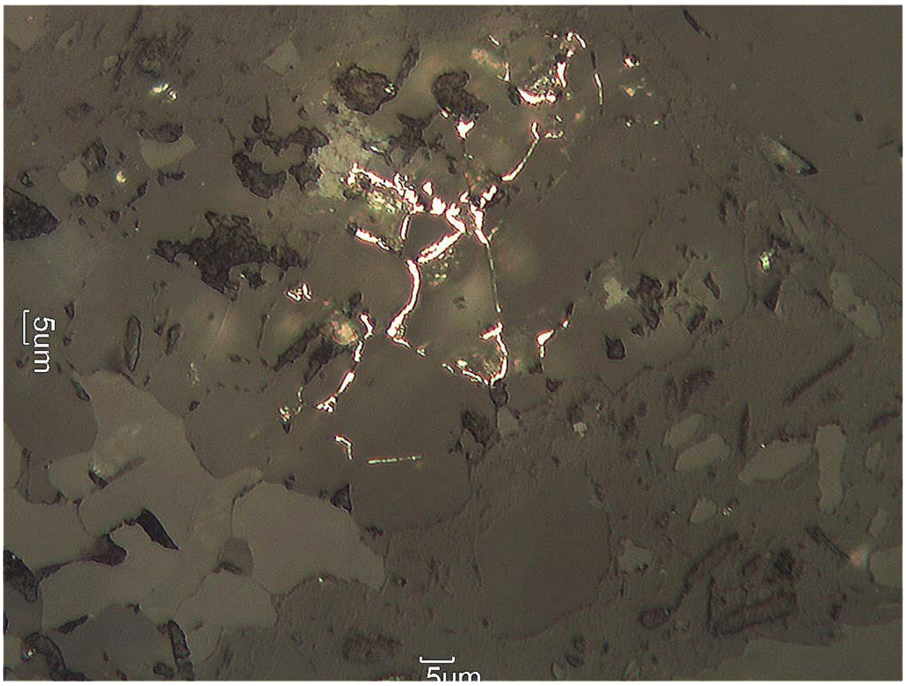

Figure 13-2: Photo 2, Scavenger Flotation Tails from Composite 1 Thin strings of pyrite rim granular quartz. Reflected light - 500X

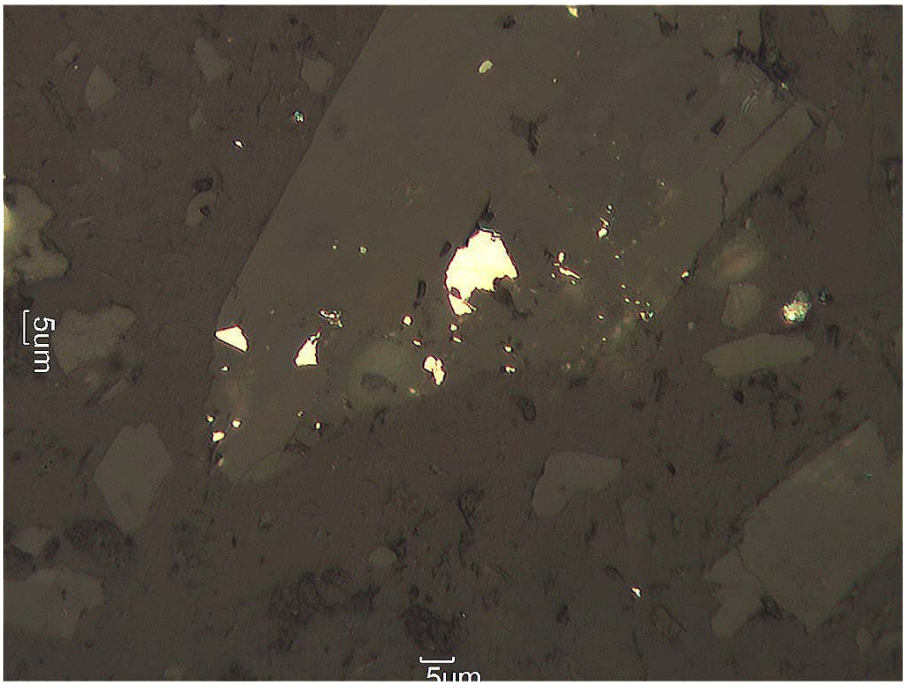

Figure 13-3: Photo 3, Scavenger Flotation Tails from Composite 1
Numerous chalcopyrite inclusions in amphibole. Reflected light - 500X

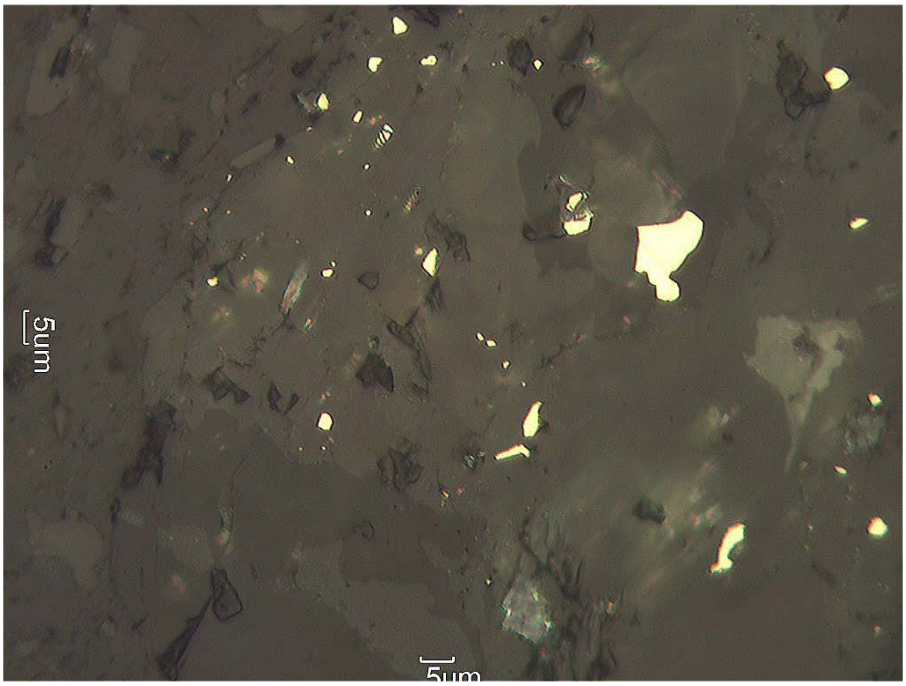

Figure 13-4: Photo 4, Scavenger Flotation Tails from Composite 1
Numerous chalcopyrite inclusions in quartz. Reflected light- 500X

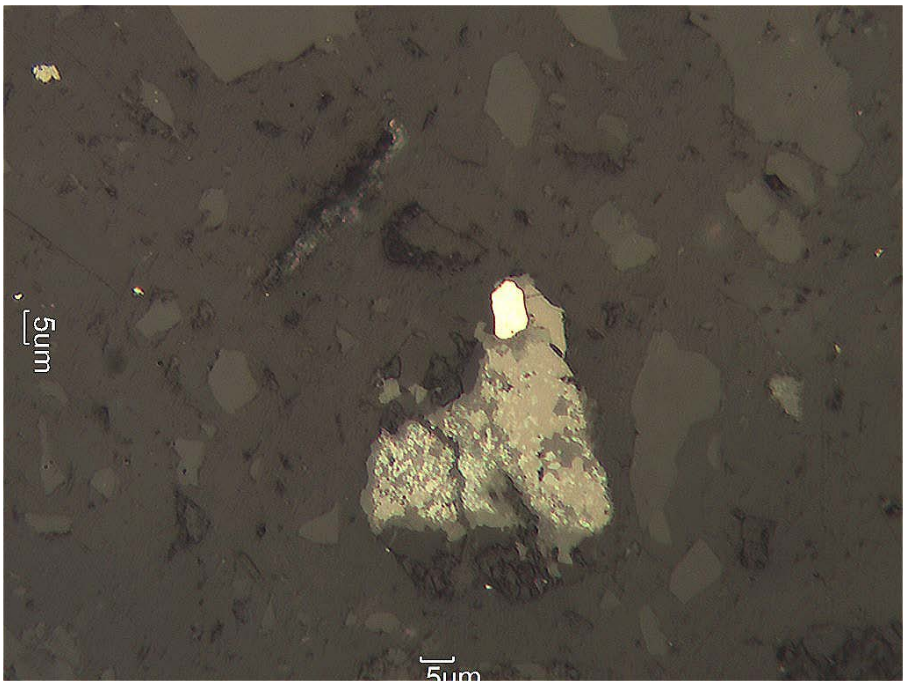

Figure 13-5: Photo 5, Scavenger Flotation Tails from Composite 1
Ilmenite with attachment of chalcopyrite. Reflected light - 500X

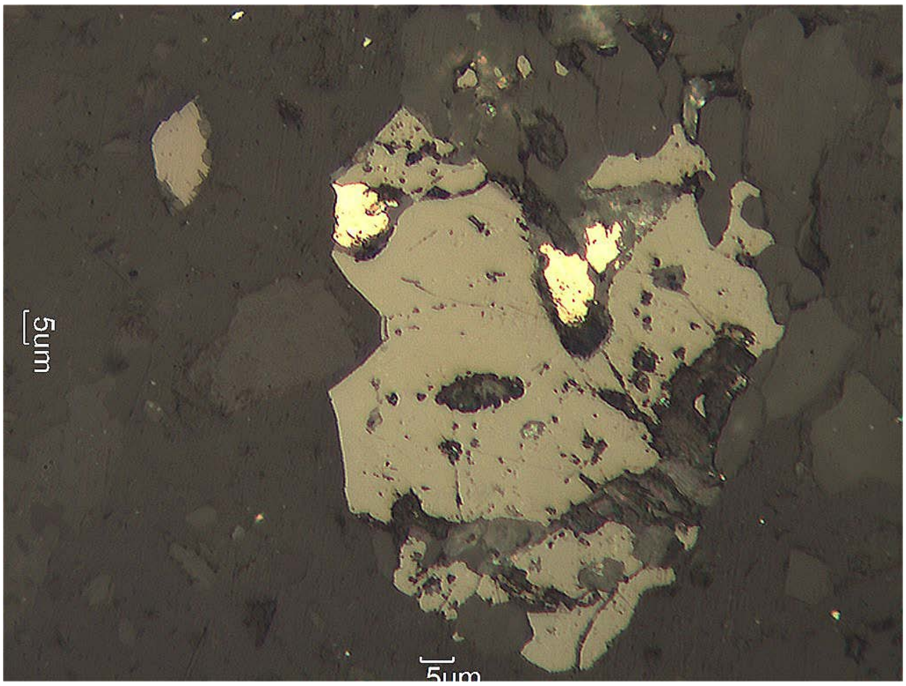

Figure 13-6: Photo 6, Scavenger Flotation Tails from Composite 1
Chalcopyrite sits in embayed areas of magnetite. Reflected light - 500X

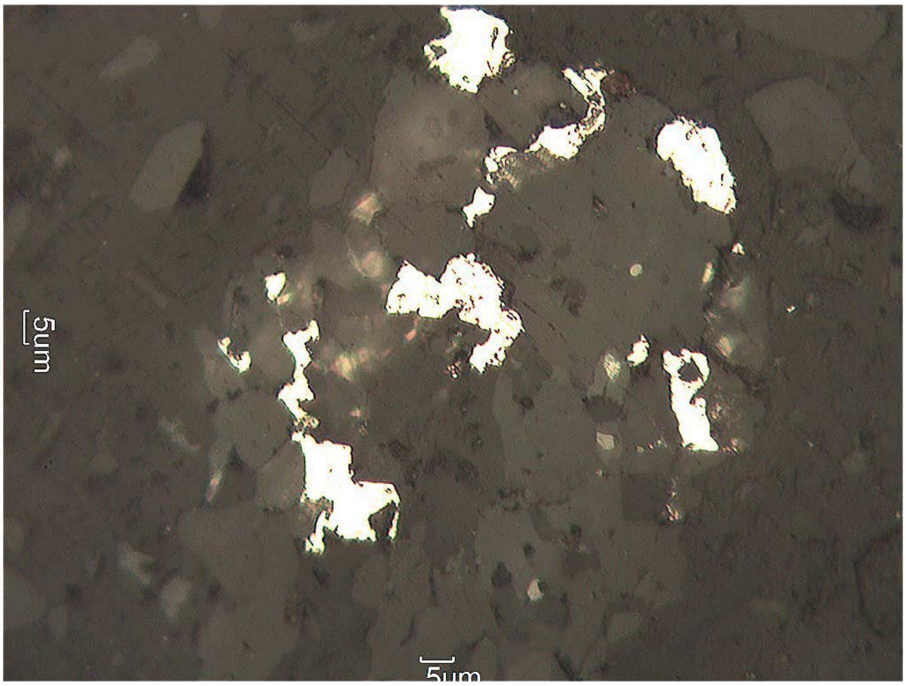

Figure 13-7: Photo 7, Scavenger Flotation Tails from Composite 1
Pyrite inclusions in an aggregate of quartz/amphibole. Reflected light - 500X

Review of the figures above indicates that of the remaining sulfides in the Composite 1 flotation tailings, there are very few liberated grains. The majority of the sulfide grains are locked within gangue minerals and are extremely fine, generally less than five microns. Recovery of these sulfides would require extremely fine grinding.

13.4.2 QEMSCAN ANALYSIS

13.4.2.1 EXECUTIVE SUMMARY

In November 2013, Phillips sent five samples to the QEMSCAN Facility at Colorado School of Mines, Golden, Colorado, for QEMSCAN analysis. The aims of the analysis were to:

- Quantify the modal mineralogy;
- Calculate the particle and grain size distribution; and
- Quantify the locking and liberation characteristics.

The samples consist mainly of feldspar, quartz, clay minerals and micas, and carbonates. Chalcopyrite, pyrite, Cr-Ni oxides, and rutile/anatase occur in minor concentrations. All other minerals occur in minor to trace concentrations.

The grain size distribution for all minerals ranges from <3 microns to 60 microns. The majority of grains are in the <3 micron to 15 micron size range.

13.4.2.2 SAMPLE PREPARATION AND MEASUREMENTS

1. The samples were split into representative aliquots using a rotary micro-riffler.
2. Sized graphite was added to mitigate particle agglomeration, preferred orientation and settling. Subsequently, the sample was mounted in a 30 mm block with epoxy-resin and left to cure.
3. The block was ground and polished to obtain a flat surface for X-ray analysis.
4. The block was carbon coated to establish an electrically conductive surface.
5. The samples were analyzed at a two micron step size in Specific Minerals Search (SMS) mode.

13.4.2.3 RESULTS

The samples consist mainly of feldspar, quartz, clay minerals and micas, and carbonates (Table 13-13). Chalcopyrite, pyrite, Cr-Ni oxides, and rutile/anatase occur in minor concentrations. All other minerals occur in minor to trace concentrations.

Table 13-13: QEMSCAN Results

Mineral Name	Comp1 Float	Comp1 HS	Comp2 HS	Comp3 HS	Comp4 HS
Feldspar	12.75	14.90	16.25	9.02	19.14
Quartz	21.20	24.92	15.85	10.71	19.10
Mica/Clay Minerals	27.95	31.49	15.38	34.85	39.45
Fe Oxide/Hydroxide	8.04	8.05	9.90	14.13	4.21
Carbonate	13.94	13.06	36.20	25.96	5.37
Sulfates/Phosphates	0.59	0.23	0.74	0.40	0.58
Cr Ni oxide	6.52	0.92	0.19	0.28	0.37
Pyrite	1.35	0.33	0.31	0.88	3.55
Chalcopyrite	5.93	3.96	3.02	1.89	5.74
Tennantite	0.03	0.05	0.01	0.01	0.01
Molybdenite	-	0.01	0.20	0.02	0.02
Sphalerite	0.22	0.51	0.16	0.05	0.14
Galena	tr	tr	0.04	0.00	0.02
Rutile/Anatase	1.48	1.56	1.74	1.79	0.78
Au phase	-	tr	-	-	-
Ag phase	tr	tr	-	tr	0.03
Wollastonite	tr	0.01	-	tr	1.50
Other Gangue	-	tr	tr	tr	0.01
tr = <0.005%					

No free gold was observed in any of the samples. A possible gold phase may have been detected in the Composite 1 (highest grade) sample reported as trace in Comp1 HS (head sample) as shown in Table 13-13. The gold occurrence must be extremely fine, less than the two micron step size used in the search.

The particle size distribution was determined by the average length and width of a particular particle. The vast majority of particles in all five samples were less than 15 microns.

Locking and liberation studies were conducted for pyrite, chalcopyrite, tennantite, molybenite, sphalerite, and galena. The results of the locking and liberation study for chalcopyrite, the primary sulfide of interest, are shown in Figure 13-8.

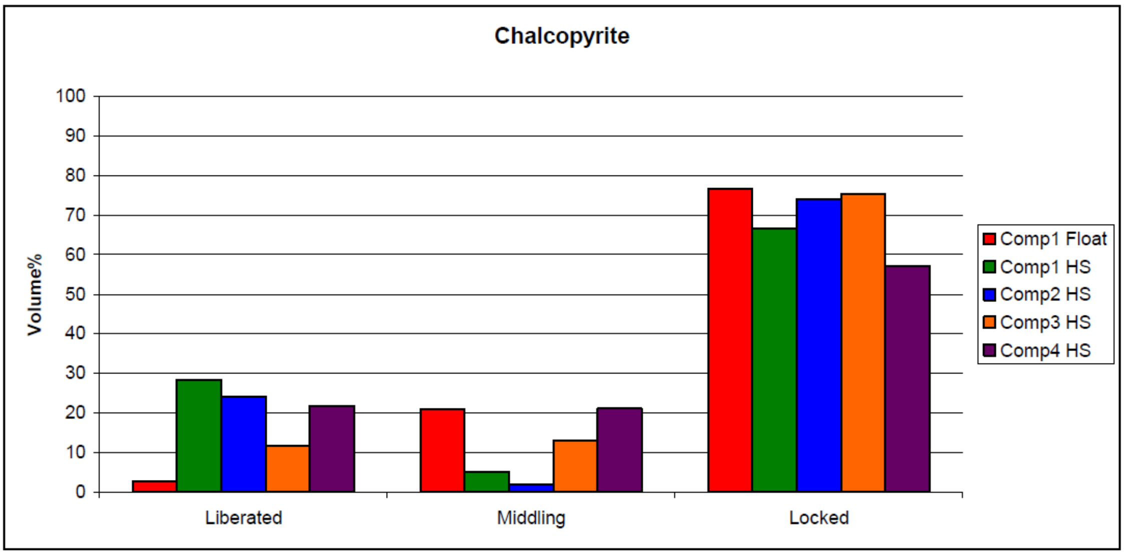

Figure 13-8: Locking and Liberation Characteristics of Chalcopyrite

Figure 13-8 illustrates the reason that flotation recovery of chalcopyrite was problematic. The chalcopyrite in all of the composite head samples as well as the remaining chalcopyrite in the Composite 1 flotation tailings was in the range of 58 % to 78 % locked even though the particle size was generally less than 15 microns. This limited the amount of chalcopyrite available to be recovered by flotation. Extremely fine grinding may improve the situation.

13.5 BASIS FOR ASSUMPTIONS FOR RECOVERY ESTIMATES

Extraction of gold and silver by cyanidation has been selected as the basis for recovery of these metals. Copper will be recovered by flotation. A coarse flotation step will be followed by ultra-fine grinding to 10 microns to liberate the copper minerals. Other hydrometallurgical methods of recovering copper cannot be justified as a result of the low grade of the ore and the limited ability to produce a suitable concentrate.

Evaluation of the metallurgical test work results indicate that gold recovery can be expected to be 86 % to 87 % in 24 hours leaching followed by carbon-in-pulp adsorption. Silver recovery was erratic during the tests but the content of silver is low and of little economic consequence. A silver recovery estimate of 20 % is expected. A grind of 75 microns to 100 microns should be sufficient to produce these results. Tests which support these estimates are Phillips Test 6, 88 % gold and 21 % silver extraction and RDI Test 5, 86.7 % gold extraction (silver not tracked in this test). Extraction of silver in the historical G&T tests at coarse size and similar head grade also returned silver recovery of 17 % to 20 %.

The rougher flotation of copper at 75 microns has been tested and the resulting recoveries were 65.3% Au, 80.6% Cu, and 70.7% Ag into a rougher concentrate mass of 10.9% of the feed. Cleaner flotation at 10 microns has not been tested but given the mineralogy and poor results achieved on sizes as small as 28 microns this finer size is clearly required to liberate the copper minerals.

A recovery of copper to final concentrate of 80 % has been assumed for this PEA. This will need to be verified in future testwork.

The cyanidation recovery of gold from the cleaner flotation tailings has been assumed to be the same as the coarser rougher flotation tailings, although it should theoretically be higher.

13.6 FACTORS AFFECTING ECONOMIC EXTRACTION

The nature of the mineralisation and physical characteristics of the host rock are the primary factors affecting economic extraction:

- The rock is very fine grained resulting in the requirement to grind to relatively fine particle size to produce acceptable recovery of copper, gold and silver.
- The rock is hard and consumes a relatively high amount of power for grinding.
- Chalcopyrite is the primary copper mineral and the grains are small and locked within the gangue minerals making recovery by flotation ineffective without ultra-fine grinding.
- The gold is not refractory and can be recovered by cyanidation without the need for oxidation processes.
- Consumption of lime and cyanide are relatively low.
- Leach kinetics are rapid and the gold and silver are essentially leached in 24 hours or less.
- There are no deleterious elements present which present difficulty in processing for copper, gold and silver.

SECTION 14 MINERAL RESOURCE ESTIMATE

14.1 RESOURCES ESTIMATION

An updated resource estimate was calculated by AMA associate Mr. Philip Jones (BAppSc, MAIG, MAusIMM, Independent Consultant) for the Project gold-copper porphyry deposit and reported in the March 2014 Technical Report (AMA, 2014a), which is reproduced as a current resource estimate in this PEA. Mr. Jones, a geologist, has a degree in geology from the South Australian Institute of Technology, is a registered Member (#1903) of the Australian Institute of Geoscientists (AIG), a Member (#105653) of the Australian Institute of Mining & Metallurgy (AusIMM), and has over 30 years of continuous experience as an exploration and mining geologist that is relevant to estimating resources of the type and style described in this report and so meets the requirements of Qualified Person for resource estimation, as defined in NI 43-101.

14.2 DATA

The main data provided by EGX for this resource estimate is summarised in Table 14-1 with additional wireframes contributed by AMA that were used in the final estimate.

Table 14-1: Main data files used in AMA resource estimates

Data description	File Names	File Type
Drilling and trenching collars, surveys and assays	20130930_DataExport_SantaBarbara.xls	Excel spreadsheet
Topography	TOPO.dxf	AutoCad dxf

Data description	File Names	File Type
Geology maps and sections	12 cross sections:- • Section 0+000NZ_AuHat_Lith_Bio_BVein_BlokModel CuHat_50Win_Mit_Inter.pdf, **to;** • Section 0-500NS_AuHat_Lith_Bio_BVein_BlokModel CuHat_50Win_Mit_Inter.pdf, **inclusive.** • LongSectionNW_AuCuHat_Lith_Bio_BVein_BlokModel_NoDPt__175Win_Mit_Inter.pdf 3 plan maps:- • Plan_View_Plot_File_Geo_DH_Log_Geo_Topo_AuHat_Lith_Bio_BVein_CuHat_A3.pdf • 600RL_100W_DH_AuHat_Lit_Bio_BVein_CuHat_BlockModel_Mit_Inter.pdf • 800RL_100W_DH_AuHat_Lit_Bio_BVein_CuHat_BlockModel_Mit_Inter.pdf	Adobe.pdf
Bulk Densities		Email
Wireframes	• Biotite_Mit_Sep2013_Final.dxf • DP_Mit_Sep2013_Final.dxf • BVein_Mit_Sep2013_Final.dxf • Mineralized zone_Au_grt_0pt2_ppm.dxf • DP_Barren 3a.dxf • Steve_B Veins V3.dxf	AutoCad dxf

All the drilling data was checked for the following:

1. Negative sample lengths;
2. Very long, short or zero sample lengths;
3. Changes in dip and azimuth of over 10 degrees;
4. Hole depths in collar file less than sampling depths in assay file; and
5. Minimum detection limit indicator texts.

No transcription errors were found in the data requiring correction. All minimum detection limit text was edited to half the detection limit.

14.3 DRILLING

A summary of the drill holes comprising the database used in these resource estimates are included in Table 14-2.

Table 14-2: Drill hole and trench data used in AMA resource estimates

Prospect	Type	ID (series)	Number	Total Depth (m)
Santa Barbara	Diamond Drill	DSB	47	20 119.31
Santa Barbara	Trench	TSB	168	3 135.51
	Mineralised drill holes used in model			
Santa Barbara	Diamond Drill	DSB	37	16 513.72

14.4 RESOURCE MODELLING

The mineralisation modelled for the Project resource estimates is generally a porphyry gold-copper deposit with low grades disseminated throughout the country rock or in stockworks of quartz veinlets, except in the barren zones where post-mineralisation diorite porphyries (DP) have intruded. This style of mineralisation is well suited to mining in bulk from an open pit and or at depth using bulk underground mining methods.

It was considered appropriate to model this mineralisation using wide search ellipses that are confined by wireframes of the mineralisation controls, i.e. rock types, structures and alteration zones. Wireframes were generated separately, based on geological logging and assays of the drilling intercepts. Subsets for each wireframed zone were then extracted from the drill assay intercepts and separated as follows: a) Biotite alteration zone outside of the 0.2 g/t Au mineralised zone but including B-veins (stockwork quartz-chalcopyrite veins) and excluding Barren DP; b) within the 0.2 g/t Au mineralised zone but not including overlapping B-veins or barren DP veins; c) B-veins including overlapping Biotite and mineralised but not barren DP, and d) DP including overlapping Biotite, mineralised zone and B-veins.

The biotite and B-Veins wireframes are based on drill hole logging and interpretation of cross-sections received while the mineralised >0.2 g/t Au wireframe is based on the drill hole assays and following the B-veins and biotite zone boundaries. The barren DP zones were wireframed by modifying the original DP wireframes, moving them in to the <0.2 g/t Au grade boundary down the drill holes.

The modelling parameters used are shown in Table 14-3 and results shown in Figure 14-1 and Figure 14-2.

Grades from the drill data, confined by the wireframes, were used to extrapolate the drill and trench sample grades, composited to standard two metre lengths, in two passes. The first pass used a 500 m diameter spherical search radius to completely fill the wireframes with grades. The second pass used a 200 m x 100 m x 200 m (XYZ) search ellipse striking 348 and dipping east at 70 degrees.

Table 14-3 Project model parameters

	X	Y	Z
Max	766,200	9,546,300	1,200
Min	764,200	9,543,800	200
Cell Dimensions	20	20	10
Number	100	125	100
Search Radii 1	500	500	500
Search Radii 2	200	100	200
Strike		348	
Dip		70	

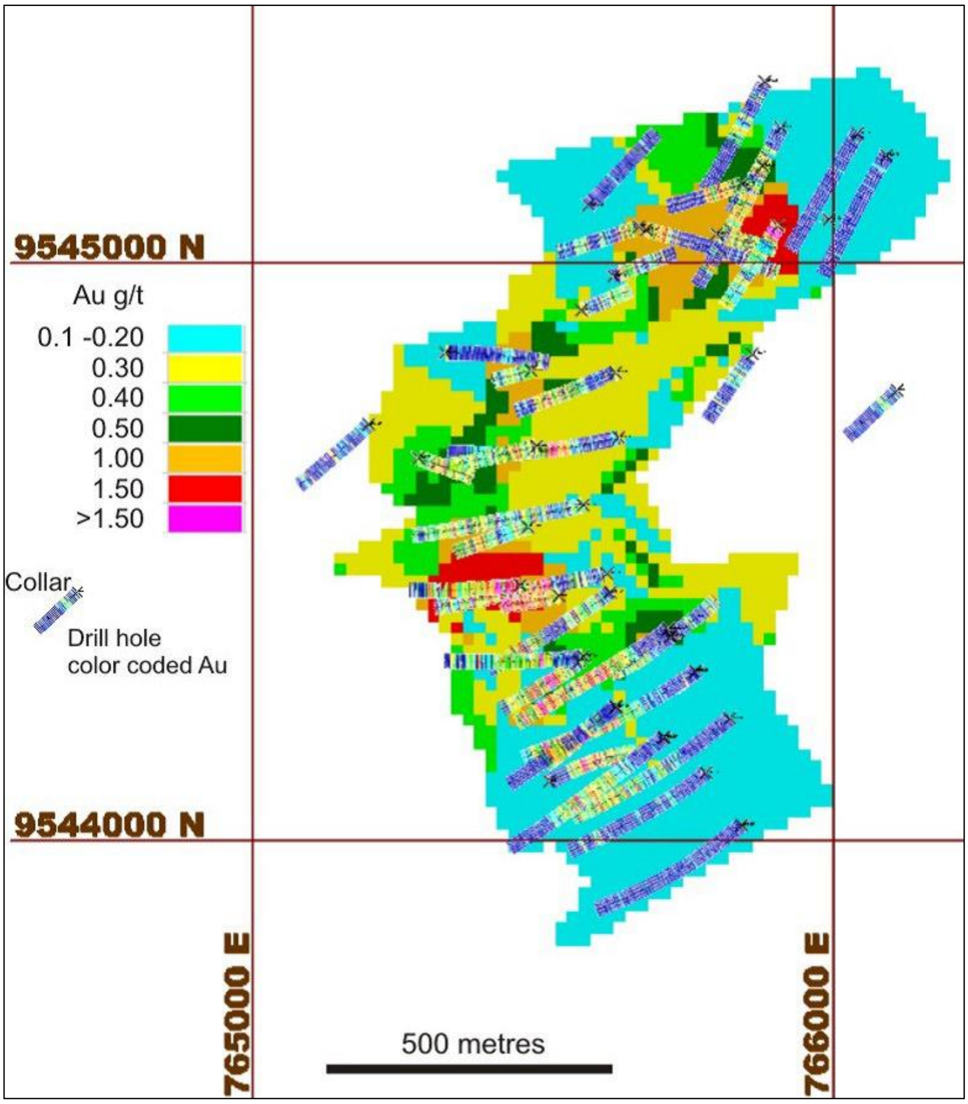

Figure 14-1: Project resource model in plan view (AMA, 2014a).

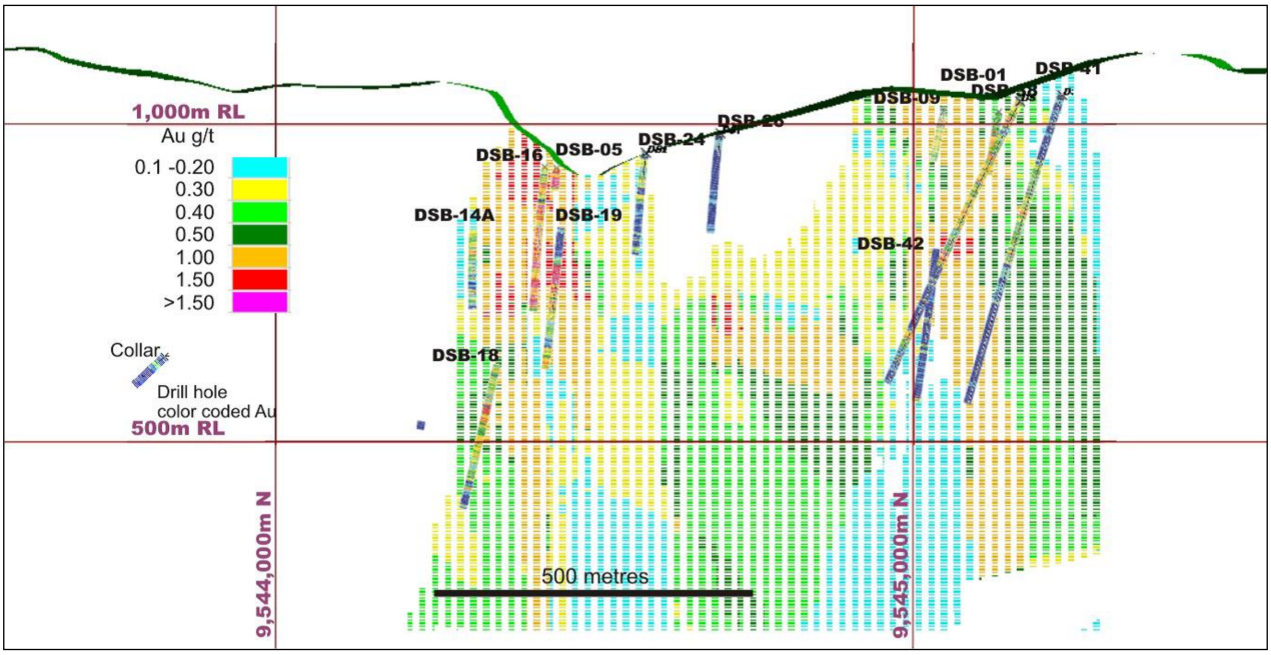

Figure 14-2: Project resource model in section view (AMA, 2014a).

14.5 RESOURCE ESTIMATES

The drill density, quality and reliability of the sampling data and the mineralisation style were all considered when categorising the Resource estimates according to the CIM code for reporting Mineral Resources.

The semi-variograms shown in Figure 14-3 and Figure 14-4 indicate that the variability of sample pairs from within the mineralised zone and within 100 m of a drill intersection is sufficiently low, along with confidence in the geological interpretation and understanding of the controls on the mineralisation, to justify categorising any resource blocks within 100 m of a drill intersection as Indicated and between 100 m and 200 m as Inferred. On the basis of the continuity and range displayed by the down-hole semi-variogram the decision was made to include the mineralised DP intersections with the andesitic basalt unit Vba inside the 0.2 g/t Au grade shell.

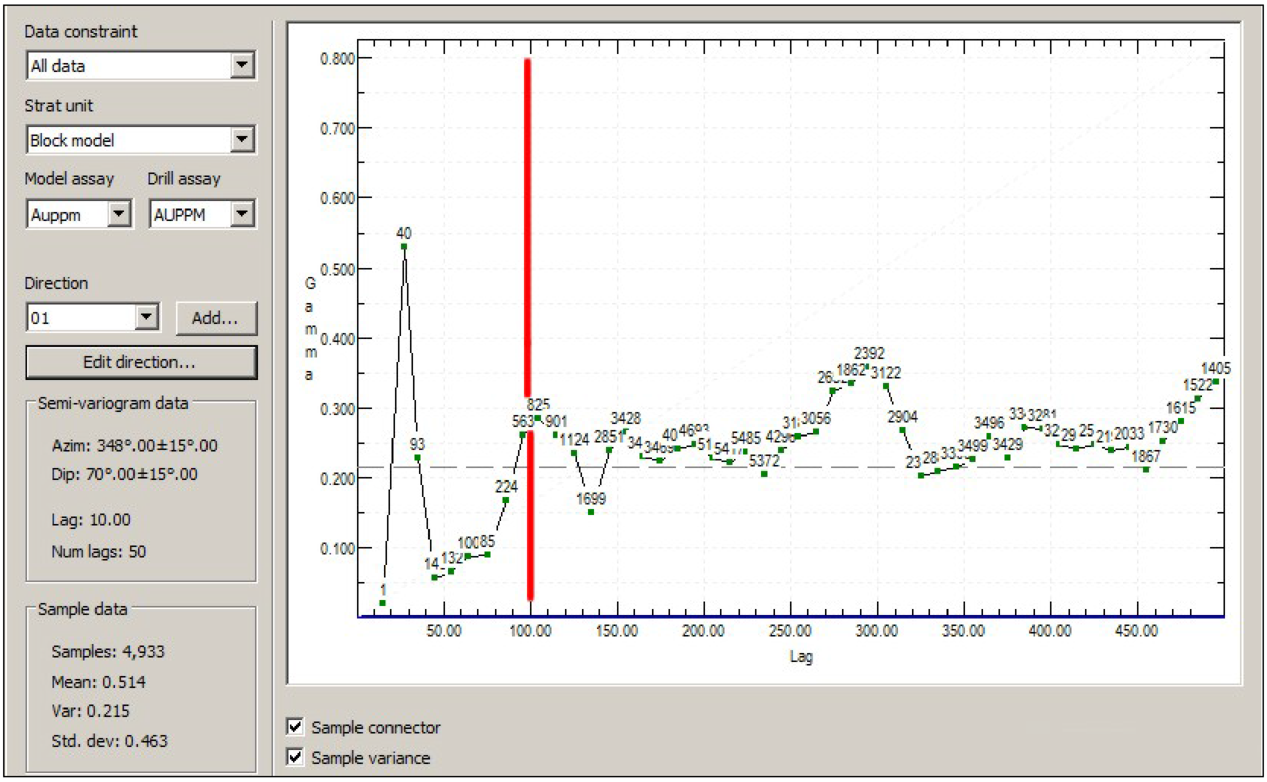

Figure 14-3 Semi-Variogram Of Drill Intersections Within Mineralised Wireframe

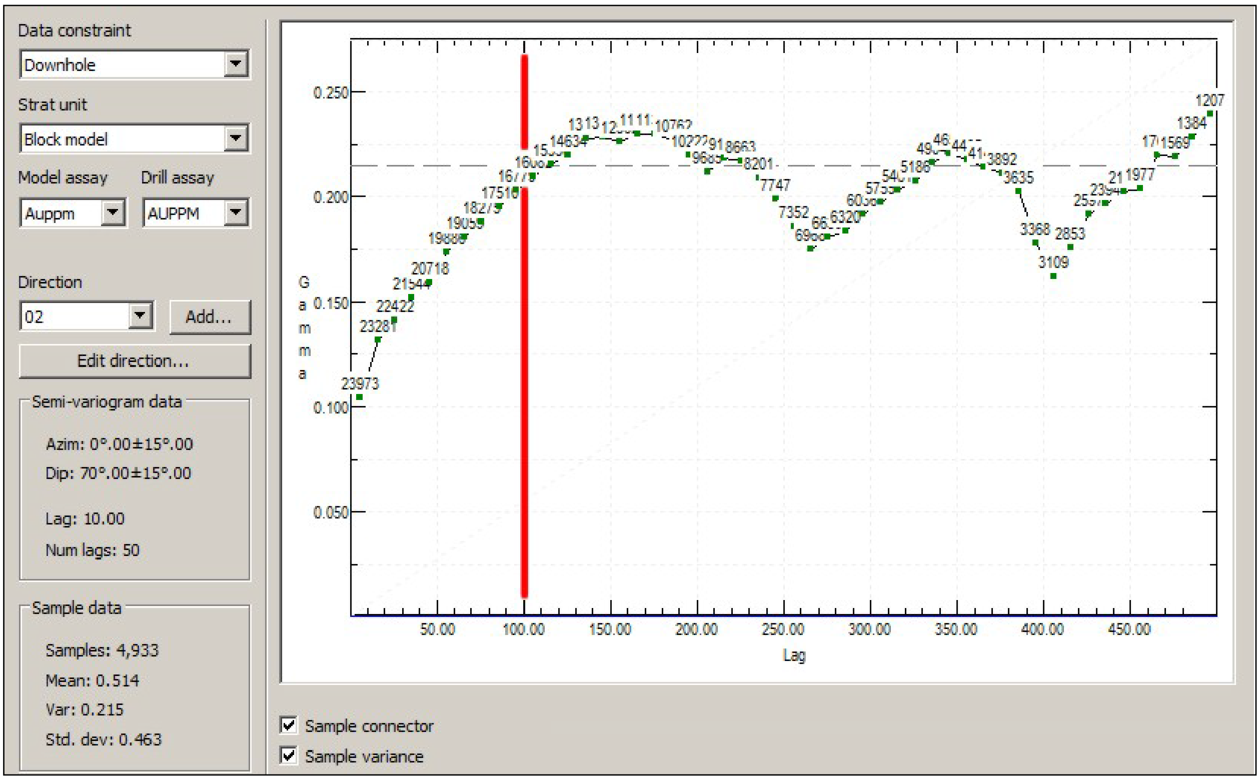

Figure 14-4**: Down-Hole Semi-Variogram Of Drill Intersections Within Mineralised Wireframe**

14.6 MINERAL RESOURCE STATEMENT

The current Mineral Resource estimate for the Project is summarised in Table 14-4. The drill density, quality and reliability of the sampling data and the mineralisation style were all considered when categorising the Mineral Resource estimates.

Table 14-4 Mineral Resource Statement

Resource Type	Tonnes (Mt)	Grade			Au (oz.)	Ag (oz.)	Cu (lbs)
		Au (g/t)	Ag (g/t)	Cu (%)			
Indicated	364.572	0.51	0.9	0.1	5 978 000	10 080 000	0.8 billion
Inferred	177.601	0.4	0.8	0.1	2 300 000	4 625 000	0.4 billion

Notes:
1. The definitions of indicated and inferred mineral resources reported here are as defined in the CIM Standards on Mineral Resources and Mineral Reserves adopted by the CIM Council, as amended.
2. Inferred resource estimates have a great amount of uncertainty as to their existence and economic feasibility. There is no certainty that all or any part of an inferred mineral resource will ever be upgraded from an inferred resource to an indicated resource category. Estimates of inferred mineral resources may not form the basis of a feasibility or pre-feasibility study but may be used in connection with a preliminary economic assessment.
3. Tonnage and grades are in metric units. Contained gold and silver ounces are reported as troy ounces, contained copper as imperial pounds.
4. Gold resources have been calculated using a 0.25 g/t Au cutoff grade. Inverse distance cubed (ID3) algorithms using wide search ellipses and confined to geological wireframe controls were used. Block size estimated is 20 x 20 x 10m.
5. Maximum search distances used to calculate indicated resources are 100m, while inferred resources were calculated using maximum distances of 100-200m from the block being estimated.

SECTION 15 MINERAL RESERVE ESTIMATE

The Project's Inferred Mineral Resources were used in the LOM plan together with the Project's defined Indicated Mineral Resources. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. There is no certainty that all or any part of the Mineral Resources would be converted into Mineral Reserves. Mineral Reserves can only be estimated as a result of an economic evaluation as part of a preliminary feasibility study or a feasibility study of a mineral project. Accordingly, at the present level of development, there are no Mineral Reserves at the Project.

SECTION 16 MINING METHODS

16.1 GENERAL

Conventional open pit mining methods were determined to be the most appropriate for the Mineral Resources in the Material Planned for Processing. The open pit optimisation parameters were designed with pit angles of 50°. Geotechnical testwork could impact on the selected slope angle which could result in more or less blocks defined as waste rock being required to be excavated to achieve the desired pit. Kinematic assessments have not been carried out, as discontinuity mapping is not available at this stage.

The level of the surrounding water table is unknown and no hydrogeological monitoring or studies have been completed to date so the presence or absence of aquifers in the vicinity is unknown. The design does not currently include dewatering facilities or other design considerations that would be required due to the presence of groundwater.

16.2 OPEN PIT MINING

The surface operations are planned as drill and blast.

Conventional truck and shovel haulage is planned within the pit, with trucks hauling the MPP to the ROM pad. Hydraulic shovels have been planned to load the material onto the trucks. In conjunction with the main operating fleet, auxiliary equipment has also been planned, which would be used to aid or supplement operations where required.

Future studies with more detailed design could see the use of contractor mining to optimise operating costs.

16.3 PIT OPTIMISATION

MDA performed a series of open pit optimisations and options in order to assess the potential final pit geometries from the Mineral Resource models provided. The pit optimisations were carried out using whittle models, and by applying conceptual economic and technical parameters.

The whittle model is considered to provide an ultimate pit outline that has the highest possible total value whilst keeping to the required pit slope parameters. An optimisation angle of 50 ° was selected for use in pit design based on the site visit in the absence of geotechnical studies.

Detailed pit designs incorporating haul roads, benches and berms have not been completed for the purposes of this PEA.

16.4 INPUT PARAMETERS

Input parameters used during the optimisation are detailed in Table 16-1. Mining loss and dilution have not been applied to the optimised shell results presented throughout Section 16.

Table 16-1: Open Pit Optimisation Parameters

Parameter	Unit	Value	Reference
Production Rate	t/a	10 000 000	Estimated
Final Pit wall Slope Angle	°	50	Section 16.3
Gold Processing Recovery	%	86.7	Section 13
Silver Processing Recovery	%	20	Section 13
Copper Processing Recovery	%	70*	Section 13
Gold Payable	%	100	Estimated
Silver Payable	%	100	Estimated
Copper Payable	%	0	Estimated
Mining Cost – Waste Mining	USD/t moved	1.70	Estimated from first principles
Mining Cost – MPP Mining	USD/t moved	1.70	Estimated from first principles
Processing Cost	USD/t MPP	11.40	Estimated
General and Administrative Costs	USD/t MPP	0.50	Estimated
Government Royalty	%	5	(Tobar and Bustamante, 2013)
Gold Product Price	USD/oz.	1 300	Estimated
Silver Product Price	USD/oz.	22	Estimated
Annual Discount Factor	%	5	Estimated

** The Open Pit was assessed at 70%. The mining model will need repeated for any future study with a new recovery once proven by test work.*

16.5 MATERIAL PLANNED FOR PROCESSING

Figure 16-1 summarises the optimisation results that have not been corrected for mining losses or dilution and should be considered as the likely best possible to achieve. The open pit figures are quoted for the contents at the selected phase pit shell.

Factors that may potentially impact on the Material Planned for Processing including mining, metallurgical, infrastructure, permitting and other factors are discussed further in the relevant sections of this report.

Figure 16-1: Potentially Mineable Resource Statement[1]

MPP	Gold	Silver	Copper	Gold	Silver	Copper
Mt	g/t	g/t	%	Oz	Oz	lb
98.807	0.72	0.96	0.11	2 272 000	3 036 000	233 018 000

16.6 MINING SCHEDULES

The Project has been scheduled to allow for a throughput of approximately 10 million tonnes per annum, which gives a life of mine of approximately 9.88 years. In addition to the feed rate, the deposits have been scheduled to mine high-grade material as early as possible in the mine life to optimise the NPV of the Project.

The mining schedule for the Project is shown in Table 16-2.

[1] Effective date 25 May 2014

Table 16-2: Mining Schedule

Material	LOM	Units	0	1	2	3	4	5	6	7	8	9	10
MPP	98 807 000	tonnes	2 000 000	8 000 000	10 000 000	9 995 000	9 997 000	9 988 000	10 000 000	10 000 000	10 000 000	10 000 000	8 827 000
Waste Rock	274 217 000	tonnes	28 000 000	22 000	45 000 000	45 005 000	45 003 000	45 012 000	22 440 000	10 923 000	5 088 000	2 932 000	2 814 000
Total Mined	373 024 000	tonnes	30 000	30 000	55 000 000	55 000 000	55 000 000	55 000 000	32 440 000	20 923 000	15 088 000	12 932 000	11 641 000
Strip Ratio	2.78	ratio	14.00	2.75	4.50	4.50	4.50	4.51	2.24	1.09	0.51	0.29	0.32
Gold Grade	0.72	g/t	0.95	0.82	0.72	0.78	0.66	0.51	0.55	0.63	0.77	0.88	0.82
Contained Gold	2 272 000	Oz	61 000	211 000	230 000	252 000	212 000	164 000	176 000	203 000	248 000	282 000	233 000
Silver Grade	0.96	g/t	1.77	1.25	1.22	1.19	0.92	0.91	0.82	0.88	0.82	0.69	0.74
Contained Silver	3 036 000	Oz	114 000	320 000	391 000	383 000	296 000	291 000	262 000	284 000	263 000	221 000	209 000
Copper Grade	0.11	%	0.15	0.12	0.12	0.12	0.10	0.10	0.09	0.10	0.11	0.11	0.10
Contained Copper	233 018 000	lb	6 520 000	21 763 000	25 751 000	25 391 000	21 628 000	21 315 000	20 529 000	22 057 000	23 980 000	23 752 000	20 332 000

N.B. Small differences between schedules and Material Planned for Processing are due to rounding.

SECTION 17 RECOVERY METHODS

17.1 GENERAL

The metallurgical process described in the following section has been developed based on the results of testwork detailed in Section 13 and the production schedule detailed in Table 16-2. The results have indicated that conventional flotation and cyanide leaching could be used to recover copper, gold and silver from the feed material.

It is proposed that the MPP be crushed and milled before entering conditioning tanks for flotation. Rougher flotation would be carried out at 75 microns, with the rougher concentrate being reground to 10 microns. The reground rougher concentrate would then go through cleaner and scavenger flotation to achieve a saleable concentrate grade.

The cleaner flotation tailings combined with the rougher flotation tailings would then enter pre-leach tanks for conditioning. Slurry would then pass into the leach tanks where cyanide would be added, after which gold would be adsorbed onto activated carbon in a series of Carbon in Pulp (CIP) tanks. The carbon would then go to elution and electrowinning, while leached slurry would be detoxified and sent to the Tailings Management Facility (TMF). This process is shown in Figure 17-1.

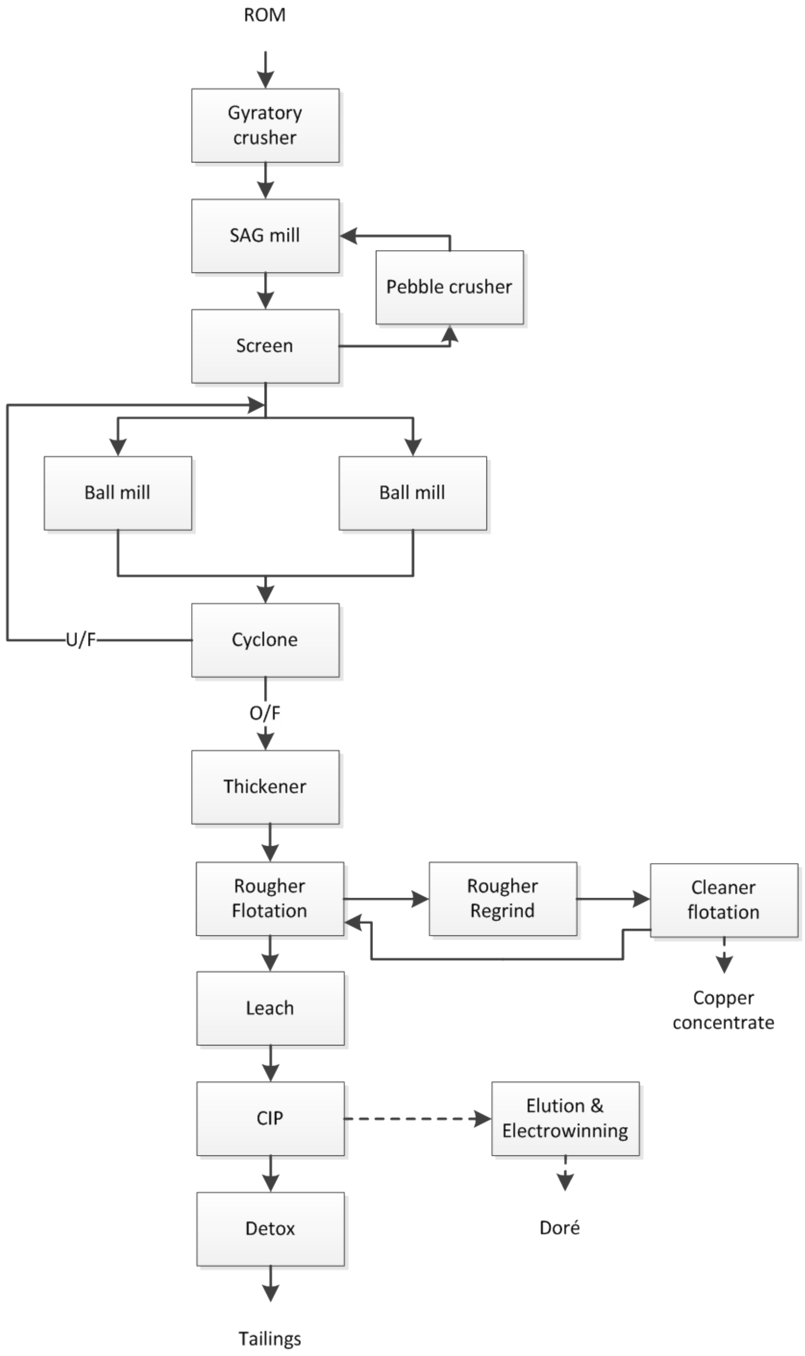

Figure 17-1: Process Block Flow Diagram

17.2 DESIGN CRITERIA

The process design criteria used as the preliminary design basis were generated based on testwork results and assumptions. These are summarised in Table 17-1.

Table 17-1: Process Design Criteria

Parameter	Value	Unit
Average annual throughput	10 000 000	t/a
Operating days	365	d/a
ROM specific gravity	2.80	t/m^3
Crushing availability	75	%
Crushing feed rate	1 522	t/h
Milling availability	92	%
Milling feed rate	1 241	t/h
Primary grind size	75	µm
Regrind grind size	10	µm
Copper recovery	80	%
Gold recovery	86.7	%
Silver recovery	20	%
Cyanide consumption	1	kg/t
Lime consumption	0.5	kg/t
Elution method	AARL	-

It should be noted that equipment was selected based on a non-optimised preliminary plant design basis and the major cost items have received most attention as is appropriate at this preliminary phase of a project.

17.3 PROCESS DESCRIPTION

17.3.1 CRUSHING AND MILLING

The MPP samples assessed showed improved leaching characteristics with reduced size and to achieve the desired 86.7 % gold recovery, a final particle size of 75 µm was selected for the Project. Therefore to reduce the assumed Run of Mine (ROM) feed from 100 % passing 600 mm to 80 % passing 75 µm, it was determined the MPP would need to undergo a three stage crushing and milling circuit. The crushing and milling circuit described is highly dependent upon the MPP characteristics. Further testwork is required to classify the MPP and its amenability to crushing and milling.

17.3.1.1 ROM BIN

It is intended that MPP would be transported from the pit to the ROM pad and haul trucks would dump the MPP into the ROM bin or on the transition ROM stockpile. In the event of crushing down time or mining schedule surplus a transition ROM stockpile was allowed for in the sizing of the ROM pad design. A front end loader would be used on the ROM pad to load the ROM bin with MPP from the stockpile for periods during which trucks are not arriving.

The primary crusher tower would be sited to suit existing topography with an approximate 15 m height difference between the ROM pad and ground level of the crusher tower. The ROM bin would allow for accepting of material directly from haul trucks or loaders. A rock breaker would be stationed at the ROM bin to break any oversize boulders such that they could pass into the crusher.

17.3.1.2 PRIMARY CRUSHER

The circuit included the feeding of MPP to a primary gyratory crusher to reduce the MPP to the desired product size of 350 mm. Crushing equipment selection is driven by the dimensions and the hardness of MPP. With little testwork of the MPP available, a qualitative methodology was applied. It was assumed that the MPP was hard and highly abrasive when compared to other crushing operations. For the desired application and capacity, sizers and arrangements of multiple jaw and cone crushers were briefly considered, however it was determined for the Project that a gyratory crusher was the most suitable due to the high throughput and large ROM top size anticipated.

17.3.1.3 MPP STOCKPILE

The coarse MPP stockpile feed conveyor would be approximately 450 metres in length and would transfer crushed MPP from the primary gyratory crusher located adjacent to the ROM pad over the Yapi River to the coarse MPP feed stockpile as shown in Figure 17-2. The point of discharge of ROM material would be the centre of the 110 m diameter coarse MPP stockpile.

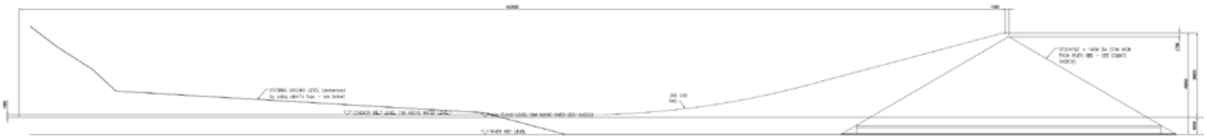

Figure 17-2: Coarse MPP Stockpile Conveyor

This intermediate stockpile would ensure the supply of crushed material to the mills is maintained, and was sized to supply a live volume for 24 hours. The crushed MPP would then be reclaimed and fed via conveyors as shown in Figure 17-3 to the SAG mill.

Three vibrating feeders would be installed under the stockpile and would discharge onto the stockpile reclaim conveyor. Each of these feeders has been sized for the full 280 t/h flow required.

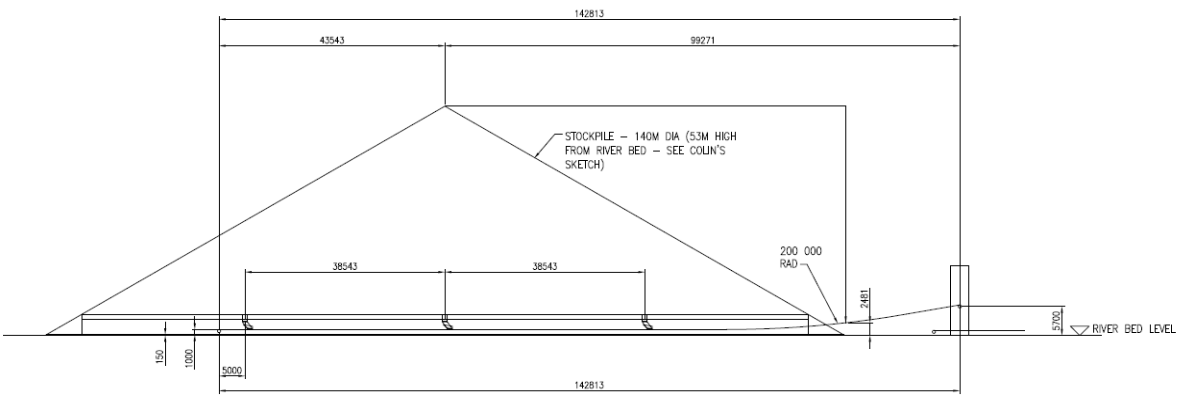

Figure 17-3: Stockpile Reclaim Conveyor

The primary crushed MPP would be fed to the milling circuit via the stockpile reclaim conveyor, which would have an approximate length of 140 m.

17.3.1.4 SAG MILL

MPP from the coarse MPP stockpile would be reclaimed and conveyed to a grinding circuit comprising a SAG mill and two ball mills. The intended 9 MW SAG mill was designed to grind material to 100 % passing 10 mm. Discharge from the SAG mill would be screened, with oversize pebbles reporting for crushing by a cone crusher. It was assumed that the pebble cone crusher throughput would be approximately 10 % of the SAG mill feed and the cone crusher was sized accordingly. The cone crusher discharge would be fed back into the SAG mill.

17.3.1.5 BALL MILLS

Cyclone underflow would feed to the ball mills for further grinding, before discharging to the cyclone feed pump boxes for reclassification. The secondary grinding circuit will consist of two 16 MW ball mills that would operate in a closed circuit with two hydrocyclone clusters that would classify to a cut size of 75 µm. The undersize would be recirculated back through the ball mills. Hydrocyclone overflow would be fed over a leach feed trash screen and the trash screen undersize from both hydrocyclone clusters would be sampled by a single leach feed sampler prior to thickening.

17.3.2 THICKENING

Overflow from the hydrocyclone would enter an 86 metre diameter concrete thickener in order to raise the solids content of the slurry ahead of flotation. Supernatant from the thickener would be returned to the process water circuit for use throughout the plant as required. The underflow from the thickener would be pumped to the first agitated flotation conditioning tank. The pump would have a variable speed drive and be installed in a duty and standby configuration.

17.3.3 ROUGHER FLOTATION

The MPP will be floated in a bank of conventional mechanically agitated flotation tanks to generate a copper concentrate. The operating conditions will be set to maximise recovery. Rougher flotation tailings will proceed to a rougher scavenger and then to the CIP leach circuit. Rougher and rougher scavenger flotation concentrate will proceed to the rougher regrind circuit.

17.3.4 ROUGHER REGRIND

The rougher concentrate will be reground in an IsaMill to a p80 of 10 microns. The regrind will be in closed circuit with a bank of hydrocyclones. To minimise overgrinding, the feed will enter at the cyclone, with only oversize material being sent to the regrind mill.

17.3.5 CLEANER AND SCAVENGER FLOTATION

The reground material will be sent to conditioning tanks and then to cleaner flotation in a bank of conventional mechanically agitated flotation tanks. The operating conditions will be set to maximise the grade of the final product. The cleaner tailings will be returned to the rougher flotation circuit.

17.3.6 LEACHING

Testwork results indicate that preg robbing is a minor issue, so rather than a carbon in leach (CIL) process, the process selected is a CIP process, comprising preleach conditioning tanks, leach tanks and CIP tanks. An added advantage, incorporation of CIP effectively reduces carbon inventory and maximizes carbon loading.

Low pressure air would be added to the preleach and all leach tanks via a sparge arrangement to assist with leach kinetics.

17.3.6.1 PRELEACH CONDITIONING

The design allows for thickener underflow to be pumped to the first two agitated pre-leach tanks prior to leaching. These tanks would be agitated and have a capacity of 3 073 m^3. Lime and low pressure sparge air would be added to condition the slurry. Preleach conditioning would be included so as to reduce cyanide consumption by the circuit based on current testwork results as discussed in Table 13-11.

Slurry would then flow to the leach circuit, comprised of six agitated tanks.

17.3.6.2 LEACHING

A leach train comprised of six tanks was proposed. Discharge from the preleach tanks would enter the first tank of the leaching train, where cyanide would be added. Lime would be added to the first and second tank to ensure pH control. All six leach tanks would be sparged with low pressure compressed air to supply the required oxygen.

Flow would cascade through open channel launders between the agitated leach tanks, each having a capacity of 3 073 m^3 and would be agitated. Launders would flow from the leach circuit to the CIP circuit.

17.3.6.3 CARBON IN PULP

Discharge from the final leach tank would be fed to a 3.5 hour, seven stage CIP circuit for the adsorption of gold from the pregnant solution onto the activated carbon. The CIP tanks would be fitted with inter-tank vibrating screen to retain the loaded carbon in the respective tanks. CIP tanks will be in a carousel configuration, whereby carbon is not transferred between tanks. Instead, carbon remains in the tank for the required time before the tank is taken offline and the slurry is discharged over a screen to recover the loaded carbon. The remaining tanks continue to operate by bypassing the tank that is being discharged. Fresh carbon is then added to the now discharged tank, and it becomes the final stage in the circuit. As such, the first and final tanks are constantly changing as tanks are bypassed to recover carbon after the required time.

Each agitated tank in the CIP circuit will have a capacity of 768.3 m^3. Each tank within the CIP circuit would be able to be by-passed. The design assumes slurry discharged would be passed over a carbon safety screen which would be a vibrating screen, to remove any fine loaded carbon and ensure no loaded or activated carbon reports to tailings in the event of a screen failure upstream. Loaded carbon from the CIP tanks would be transferred to the acid wash and elution plant.

17.3.7 GOLD RECOVERY

17.3.7.1 ACID WASH AND ELUTION

As part of the proposed design it has been assumed that the acid wash and elution columns would be supplied as a whole package with all associated equipment and would allow process flexibility in the sequencing of processes and would be fully integrated with the process plant's control system. The package is anticipated to include the following described elements and process approximately 8 t of loaded carbon per day.

The loaded carbon would be initially washed with hydrochloric acid solution, and then rinsed with treated water. The acid washed carbon would then be transferred to the elution column. The elution column would strip acid-washed loaded carbon at an elevated pressure and temperature. An eluate solution of sodium cyanide and caustic solution would be pumped through the loaded carbon.

The strip solution would consist of sodium cyanide and caustic (sodium hydroxide). The pregnant solution would circulate from the elution column to electrowinning cells in the gold room and return as barren solution.

17.3.7.2 CARBON REGENERATION

It is anticipated that the acid and elution columns package would include carbon regeneration whereby barren carbon would undergo regeneration before being fed back to the CIP circuit. This process would involve barren carbon passing through a dewatering screen, with removed water being returned to the elution circuit and the barren carbon passed through a horizontal regeneration kiln to regenerate up to 8 t of carbon per day. Regenerated carbon would then be quenched with process water and with fresh carbon passed over the carbon dewatering screen before being transferred to the last CIP tank of the circuit.

17.3.7.3 GOLD ROOM

As part of the proposed design it has been assumed that the gold room would be supplied as a whole package with all associated equipment and that the process plant's control system would provide monitoring and interlocking.

It is anticipated that the gold room package would include recovery by electrowinning, whereby pregnant solution from the elution column flows through electrowinning cells to accumulate gold on the cells' cathodes and barren solution returns to the elution column. Following which a hoist would remove the cathodes to be high pressure washed with potable water to remove the gold. Typically this process then includes the gold sludge being dewatered in a filter and dried in a drying oven, following which it is mixed with flux, smelted in an induction furnace and poured into moulds to produce gold doré. The doré would then be securely stored in a strong room.

Due to security issues it is anticipated that the gold room would be an enclosed structure and as such would require significant fans and ducts for ventilation.

17.3.8 DETOXIFICATION

The screened CIP tailings slurry would then be fed to the first of two detoxification tanks. The first detoxification tank would be dosed with sodium metabisulphite, copper sulphate and slaked lime for residual cyanide destruction and pH control.

The first and second detoxification tanks have been preliminary designed as glass lined bolted steel tanks with a capacity of 1 721 m^3 each. Two tanks are anticipated to be required for adequate residence time however this arrangement would need to be confirmed by future testwork. The tanks would be agitated and slurry would flow between the detoxification tanks through rubber-lined launders before being pumped to the TMF as describe in Section 18.7.

17.4 CONSUMABLES

17.4.1 ENERGY

The operation would be fed with electrical power as part of a site wide reticulation system that supplies power from the hydroelectrically generated grid. The basic estimated energy consumption figures are as listed in Table 17-2.

Table 17-2: Power Requirements

Description	Requirement
Installed Power	75 MW
Estimated site power factor	0.8
Monthly energy requirement	5 000 kWh

17.4.2 WATER

A site water balance was prepared based on regional meteorological data and the mass balance. Initial estimates for water consumption focused on the process plant requirements. Other consumption on site would include the mining maintenance area, dust suppression activities and all other administration and amenity facilities on site. Only basic estimates were possible without a full set of climate data for the project site, but indications were that the operation would be a positive consumer of water due to the large catchment areas of the operation such as the pit, waste rock dumps, TMF and other infrastructure areas as well as recycling within the process plant. The latter comprises of overflow from the hydrocyclones which would be thickened, resulting in reclamation of process water for recycling to the mills. Raw water would be required to make up for water lost to the tailings stream. The required make-up raw water flow for the operation was calculated as 1 201.4 m^3/h.

17.4.3 PROCESS MATERIALS

Abrasion caused by the crushing and milling of material results in the consumption of grinding media, mill and crusher liners. In the absence of testwork with regards to the abrasiveness of the material and SAG and rod bond indexes, the consumption of liners has been accounted for within the costs as a percentage of equipment costs rather than determination of a consumption rate. SAG mill and ball mill liner machines have been included within the design to facilitate relining.

The bond ball index testwork was used to estimate a required consumption of 1.9 kilograms per tonne of MPP of grinding media for the ball mills.

Additionally, fresh activated carbon would need to be continually introduced to the circuit as make up due to losses in the regeneration process and through abrasion with the slurry. This has been estimated as 30 tonnes per day based on similar historical projects.

17.4.4 REAGENTS

It is anticipated the process would utilise the following reagents:

- Potassium amyl xanthate (PAX);
- AP-208;
- Methylisobutal carbonal (MIBC);
- Sodium cyanide;
- Lime;
- Sodium hydroxide;

- Hydrochloric acid;
- Sodium metabisulphite;
- Copper sulphate; and
- Flocculant.

The main reagents consumed in the process are discussed in the following sections.

17.4.4.1 POTASSIUM AMYL XANTHATE (PAX)

PAX is required in the flotation circuits. It is supplied as a liquid in drums and will be transferred into a dosing tank. From the dosing tank it will be supplied to the rougher, rougher scavenger and cleaner circuits. Consumption is estimated at 100 g/t.

17.4.4.2 AP-208

AP-208 is required in the flotation circuits. It is supplied as a liquid in drums and will be transferred into a dosing tank. From the dosing tank it will be supplied to the rougher, rougher scavenger and cleaner circuits. Consumption is estimated at 100 g/t.

17.4.4.3 METHYLISOBUTAL CARBONAL (MIBC)

MIBC is required in the flotation circuits. It is supplied as a liquid in drums and will be transferred into a dosing tank. From the dosing tank it will be supplied to the rougher, rougher scavenger and cleaner circuits. Consumption is estimated at 28 g/t.

17.4.4.4 CYANIDE

Cyanide would be required in both the leach and elution processes for the dissolution of gold. Sodium cyanide powder would be added via a bag breaker in a batched process to an agitated mixing tank. Once adequate mixing had occurred the solution would be transferred by gravity to the cyanide transfer tank ready for use in the process.

By conditioning the slurry prior to leaching, it is anticipated that the cyanide required for leaching would be reduced to 1 kilogram per tonne of MPP based on testwork that showed cyanide additions of 1.5 kg/t, with consumptions in the order of 0.6 kg/t.

17.4.4.5 SODIUM METABISULPHITE

The reaction in the detoxification tanks requires sulphur dioxide, supplied as sodium metabisulphite, in order to destroy residual cyanide. Similar to most reagents, the sodium metabisulphite would be supplied as a powder and added via a bag breaker in a batched process to an agitated mixing tank. Once adequate mixing had occurred, the solution would be transferred by gravity to the sodium metabisulphite transfer tank.

In the absence of testwork, it was assumed that the addition of 1.15 kilograms of sodium metabisulphite per tonne of MPP would be required based on similar historical projects.

17.4.4.6 LIME

Regulation of the pH in the leach is achieved through the addition of slaked lime to the pre-leach tanks. Slaked lime powder would be added via a bag breaker in a batched process to an agitated mixing tank. Once adequate mixing had occurred the solution would be pumped (with standby) to the lime transfer tank ready for use in the process. The transfer tank would also be agitated to maintain the lime in solution prior to its application to the elution and leach tanks.

Testwork showed 0.5 kg/t (burnt lime equivalent) was necessary to raise the pH to the required level.

SECTION 18 PROJECT INFRASTRUCTURE

18.1 SITE LAYOUT

The general site layout is shown in Figure 18-1.

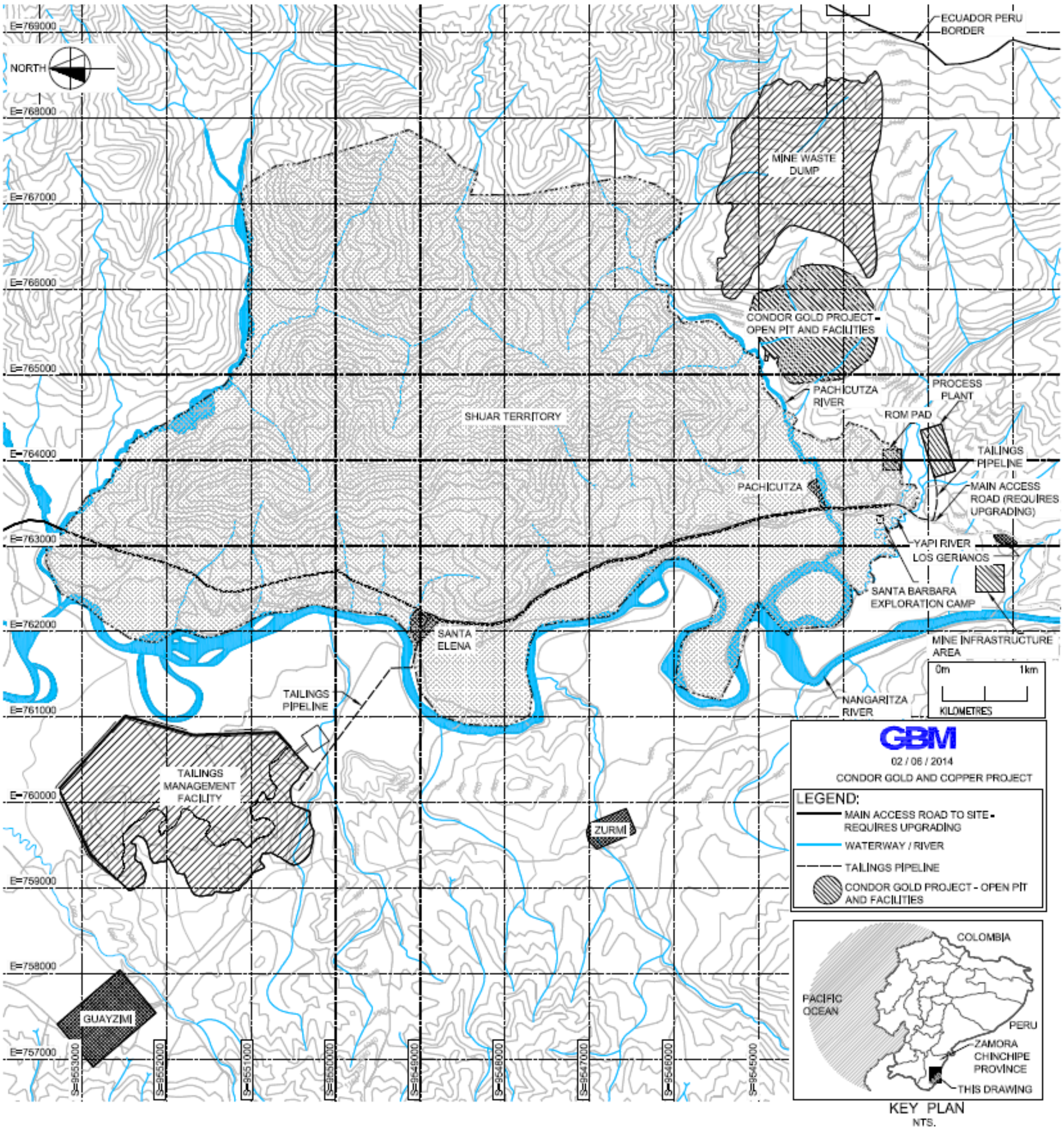

Figure 18-1: Project General Site Layout

18.2 MINE INFRASTRUCTURE AREA

The mine infrastructure area will be the base for mining operations for the open pit. It will serve as the maintenance and fuelling station for mobile mining equipment. Also, it will house mining operations administrative, professional and management staff and provide amenities for production and maintenance crews.

The following facilities are proposed:

- A six bay structural steel framed workshop suitable for CAT 797F haul trucks.
- Warehouse and external laydown for storage of consumables and spare parts.
- An administration building to provide offices for mining management, professional and administrative staff. Mess and muster facilities for mine personnel would also be provided.
- Double-skinned bunded tank farm for 1 000 000 L diesel storage.
- Double-skinned bunded tank farm for six lubricants, coolant, grease, waste oil and waste coolant.
- Combined fire water and raw water tanks and reticulation.
- Security gate house and site fencing.
- Powder and primer magazines.
- Heavy vehicle wash for cleaning of mining fleet prior to maintenance.
- Employee and mine vehicle car parks.
- Two lane access road from the existing road suitable for semi-trailers with a gross vehicle mass of less than 48 t.
- A potable water tank and pump-set to store drinking, washing and toilet flush water for the facility. Water would be pumped from the main storage tanks fed by potable water treatment plant.
- A waste water treatment plant.

18.3 PROCESS BUILDINGS

Buildings located immediately adjacent to the process plant, would house all employees and provide facilities including:

- A workshop and maintenance facility for mobile equipment.
- Warehouse and external laydown for storage of consumables and spare parts.
- An administration building to provide offices for process plant management, professional and administrative staff. Mess and muster facilities for process personnel would also be provided.
- Laboratory for process testing requirements.

- Double-skinned bunded tank farm for 10 000 L of diesel storage.
- Combined fire water and raw water tanks and reticulation.
- Employee and process light vehicle car parks.
- Two lane access road from the existing road suitable for semi-trailers with a gross vehicle mass of less than 48 t.
- A potable water tank and pump-set to store drinking, washing and toilet flush water for the facility. Water would be pumped from the main storage tanks fed by potable water treatment plant.
- A waste water pumping station to pump waste water to the waste water treatment plant.

18.4 ACCOMMODATION

The accommodation would house 25 employees and would be located in the town of Los Gerianos. It would comprise of single furnished rooms with ensuite bathrooms .The accommodation would be connected to the existing water, sewer and electricity services of Los Geranios.

Other employees would be housed in the surrounding villages or commute from one of the regional centres such as Zamora or Paquisha.

18.5 POWER AND ENERGY SYSTEMS

Power would be supplied to the Project from the national transmission network ('Sistema Nacional Interconectado') and purchased from the national electricity authority, CONELEC. The Project would utilise a proposed 230kV transmission line between the Sinincay Substation located northwest of the city of Cuenca and the Mirador Project located near the town of Tundayme. These proposed transmission lines shall be constructed by Corriente Resources Inc. and are currently in the government approvals phase. Once complete, ownership of the transmission lines shall transfer to the national electricity transmission authority TRANSELEC who are mandated to provide unimpeded access to transmission infrastructure. This presents a significant risk to the Project if the proposed 230kV lines are not built as an alternate method of power supply to the site would need to be sourced and could represent a significant capital and operating cost to the Project.

To connect to the proposed 230kV Corriente transmission lines, a 138kV transmission line would need to be constructed to the Project site. In addition, it is proposed to build a 230kV/138kV substation near Tundayme and a 138kV/22kV 90MVA substation at the Project location. The approximate route of the transmission lines and coordinates of the Mirador substation are shown in Figure 18-2. Medium voltage (22kV) power would be delivered to process, mine and other infrastructure using overhead 22kV distribution lines and substations as necessary.

Connection to the national grid will require confirmation of the availability of electricity, capacity of the existing infrastructure and capacity of the proposed 230kV transmission lines. Commercial agreements for purchase of power will also be required with CONELEC and/or private power generators. These matters will need to be taken into account as the Project progresses.

No emergency power has been allowed for.

18.6 WATER SERVICES

18.6.1 RAW WATER DAM

Two submersible pumps in a duty and standby arrangement would be used to deliver water to the process plant and 300 ML capacity raw water dam. From the raw water dam, water will be pumped to the potable water treatment plant, the process plant and mine infrastructure area.

18.6.2 POTABLE WATER TREATMENT PLANT

A single potable water treatment plant and main storage tank would supply potable water to the Project. It would be located on the hill west of the process plant. Raw water would be pumped to the treatment plant from the raw water dam.

18.6.3 WASTE WATER TREATMENT PLANT

It is proposed to install a waste water treatment plant west of the mine infrastructure area to treat sewage generated onsite. It is proposed that the plant would release treated water into the Nangaritza River downstream of the raw water intake pumps. It is envisaged that the waste water treatment plant would be supplied as a turn-key packaged plant.

18.7 TAILINGS PIPELINE

A pipeline is required to transport slurry tailings from the processing plant to the TMF. Allowance has been made for a 10 300 m long, DN450 mm carbon steel pipeline run on concrete sleepers. The pipeline will be rubber lined to prevent wear on the interior steel surface. The route of the pipeline would follow the route of the main access road from the process plant to the town of Santa Elena. Then, the pipeline will be run around the town, to the eastern bank of the Nangaritza River. A pipeline bridge will be required to span the river. From the western bank of the river, the pipeline will run directly to the corner of the TMF. The pipeline would connect into transportable spigotted pipelines to deposit tailings into the TMF.

Twelve sets of centrifugal pumps, in a duty/standby arrangement, would be required to deliver tailings slurry to the TMF.

18.8 COMPRESSED AIR

Compressed air systems would be required for the process plant. The system would include air filters, dryers, condensate removal valves and receiver tanks.

18.9 COMMUNICATIONS AND INFORMATION MANAGEMENT SYSTEMS

An optical fibre connection would be required to provide internet and telephone services to the site. Normally, an optical fibre cable would have to be installed between site and the existing national network. However, GBM understand that an optical fibre cable is installed inside the optical ground wire on all transmission lines. TRANSELEC, the national electricity transmission authority, confirmed that the optical ground wire can be used for site communications. Voice Over Internet Protocol (VOIP) would be utilised for landline telephone services.

Existing cellular networks are limited with poor reception across much of the Project site. No allowance has been made to upgrade the cellular network. Site communications would be via a two-way radio system.

18.10 FENCING AND SECURITY

The process area, mine infrastructure area and TMF would be fenced along their perimeter to protect facilities against theft, vandalism and unauthorised access. Access to the mine infrastructure area and process area would be provided via a manned guard booth and lockable gates. Fencing would also be installed in key areas such as to explosives storage, reagent mixing and the gold room.

18.11 ACCESS INFRASTRUCTURE

18.11.1 PROJECT SITE TO NATIONAL HIGHWAY E45 (LOS ENCUENTROS)

The existing road between the Project site and Los Encuentros is an unsealed gravel road of varying width. Existing bridges vary between 10 and 40 tonne capacity. There is a single crossing over the Pachikutza River. The proposed access route would require capital works to be suitable for transport of equipment, diesel fuel and consumables. These works include:

- Construction of a new 55 kilometre, two lane, 700 mm thick unsealed road suitable for heavy vehicles i.e. prime movers with semi-trailers with a gross vehicle mass of no heavier than 48 tonnes.
- Three new 48 tonne capacity bridges over the Rivers Yapi, Pachicutza and Conguime.
- Upgrades to four existing bridges between Conguime and Los Encuentros from 40 to 48 tonne capacity.
- Installation of new culverts and slope stabilisation to sections of the road identified as particularly susceptible to landslides and washouts.

The access route to the Project and proposed works are shown on Figure 18-2.

The route has been chosen based on the observed conditions of the existing roads and bridges and discussions with officials from the Ecuadorian Ministry of Transport and Public Works. Although a shorter route exists between Paquisha and Zamora it was observed to have small turning radii around bends and to be highly susceptible to landslides. The proposed route is straighter and is likely to be safer, more reliable and less costly to improve.

The Ministry of Transport and Public Works have a 35 year national transport plan that includes construction of new and upgraded roads in the Zamora-Chinchipe Canton and beyond. It is probable that some of these projects would improve travel times and road safety in the region however, they have not been taken into consideration in this study.

18.11.2 INTERNAL ACCESS ROADS

Access roads around the site would be unsealed gravel access roads; typically 500 mm thick and constructed of locally sourced material from borrow pits.

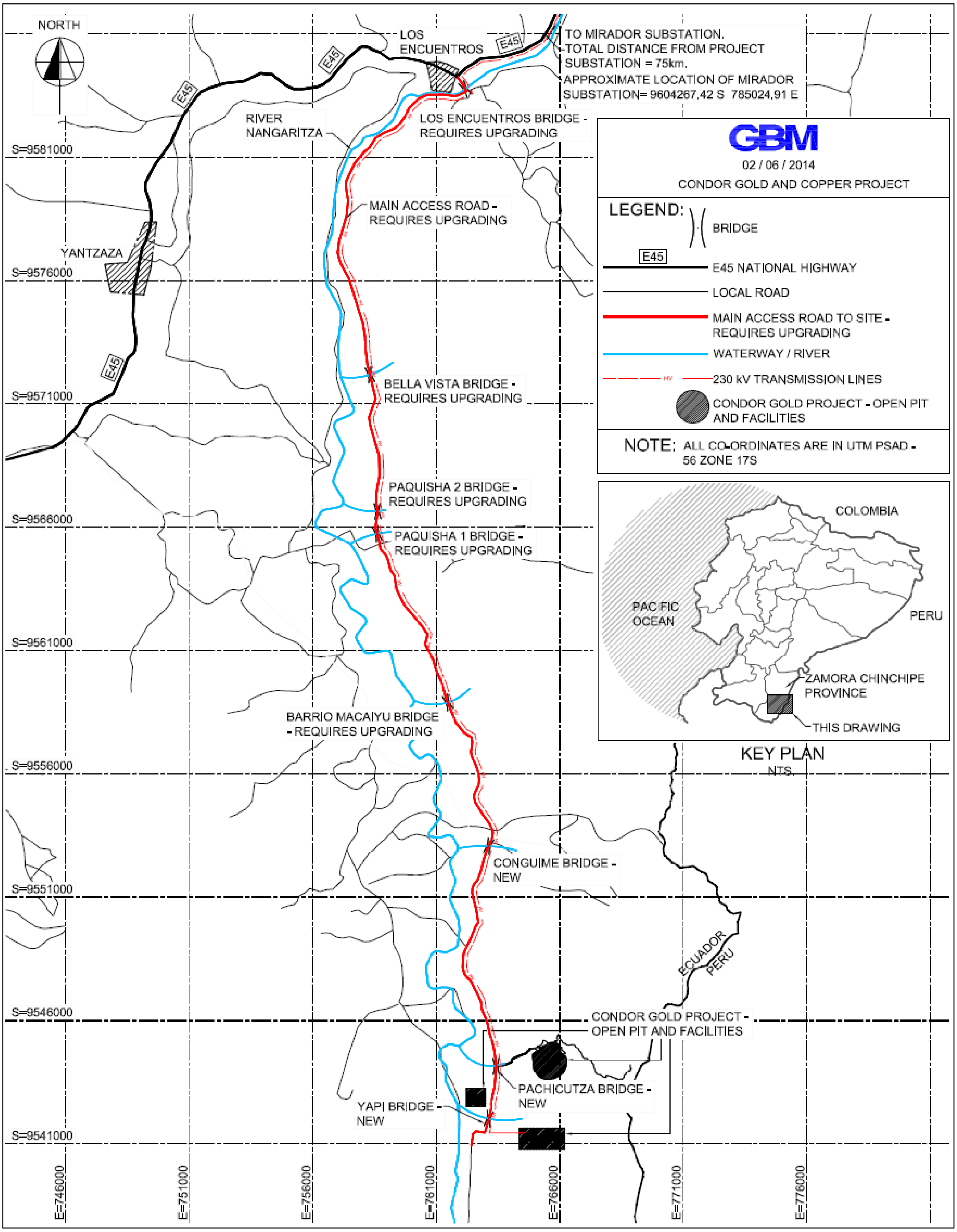

Figure 18-2: Project Access Route and Proposed Works

SECTION 19 MARKET STUDIES AND CONTRACTS

The Qualified Person has reviewed all documentation referred to in this section, and they support the assumptions in this PEA. No specific market studies have been undertaken for the purpose of this report.

The Qualified Person is not aware of contracts in force with Condormining, FJTX, EGI, ECC or EGX. It is anticipated that Condormining will establish refining agreements with third parties for refining of doré and sell bullion on the spot market by marketing experts engaged by Condormining. It is anticipated that the terms contained within the refining contracts and sales contracts would be typical and consistent with standard industry practices, and similar to contracts for the supply of bullion and doré elsewhere in the world. Discussions with the local electrical power supply authority have been undertaken but no negotiations or contract terms have been discussed to date.

Ecuador has executed various mining cooperation agreements with countries in the region and treaties to avoid double taxation with several other countries that may or may not impact on gold export (Global Legal Group, 2014). VAT is payable on goods purchased and services rendered within Ecuador or imported from abroad however is not recoverable when a non-renewable natural resource, such as gold or silver, is sold outside of Ecuador (Tobar and Bustamante, 2013).

Following sharp declines in 2013, gold and silver prices have somewhat stabilised in 2014, as shown in Figure 19-1.

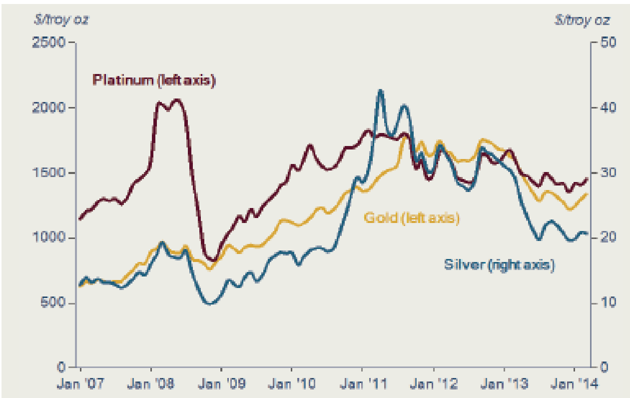

Figure 19-1: Precious Metal Prices (World Bank, 2014)

SECTION 20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 ENVIRONMENTAL STUDIES

20.1.1 ENVIRONMENTAL STUDY REQUIREMENTS

Current legislation and institutional frameworks are summarised in this section; the information is taken from the 2006 Environmental Impact Assessment (AmbienConsul Co. Ltd., 2006).

20.1.1.1 MINING ACT

The Mining Act came into force on 31 May 1992, by publication in the R. O. N ° 695 and in relation to environmental protection, in Chapter II, Article 79 it says: "The holders of mining and beneficiation plants, smelting and refining, shall perform Environmental Impact Assessments and Management Plans environmental to prevent, mitigate, control, rehabilitate and offset the environmental and social impacts resulting from its activities; studies to be approved by the Secretariat for environmental Protection, Ministry of Energy and Mining".

20.1.1.2 ENVIRONMENTAL MANAGEMENT ACT

The Environmental Management Act 1999 outlines the national environmental and administrative framework.

This Act was published in the R.O. N ° 245 of 30 July 1999 by Article 1 and sets out the environmental policy principles and guidelines, determines the duties, responsibilities, levels of participation of the public and private sector in environmental management, and indicates the permissible limits, controls and sanctions in relation to environmental management.

Article 2 of the Act states that environmental management is subject to the principles of solidarity, responsibility, cooperation, coordination, recycling and reuse of waste, use of environmentally sustainable alternative technologies and respect for cultures and traditional practices.

Article 8 provides that the national environmental authority shall be exercised by the relevant ministry, who shall act as the governing body, coordinating and regulating the National System for Environmental Management.

20.1.1.3 PREVENTION AND CONTROL OF ENVIRONMENTAL POLLUTION ACT

This Act was published in 1976 and is the legal framework that establishes the general considerations for prevention and control of environmental pollution relating to water, soil and air.

Regulations implemented under this Act that are currently in force are:

- Prevention and Control of Environmental Pollution, in relation to water, promulgated by Ministerial Agreement No. 2144 and published in the RO 204 of 5 June 1989. Article 1 regulates the activities and producing sources of water pollution.

- Prevention and Control of Environmental Pollution caused by the emission of noise, promulgated by Ministerial Agreement No. 7789 and published in the RO 560 of 12 November 1990 to regulate the activities or annoying noise sources that may contaminate or be harmful to the environment.

- Prevention and Control of Pollution in relation to soil, which was enacted by Ministerial Agreement N° 14629 and published in the R.O. 989 of 30 July 1992 which determines the control measures on activities that contribute to soil pollution.

- Solid Waste Management promulgated by Ministerial Agreement No. 14630 and published in the RO 991 of 3 August 1992, whose purpose is to regulate the storage, collection, transportation, disposal and other aspects in relation to solid waste.

20.1.1.4 GENERAL REGULATIONS THAT REPLACE THE REGULATIONS OF THE MINING ACT

There are also a number of regulations within the Mining Act, which have been replaced to strengthen and support the environment.

A regulation was issued by Executive Decree No. 1415 and published in the R. O. N ° 307 of 17 April 2001 that reforms the current Mining Act, and was introduced by the Law for the Promotion of Investment and Citizen Participation, which strengthens and consolidates mining development in Ecuador.

The regulation supports mining that is in the public and national interest, and provides that the fundamental priority is sustainable development, and a harmonious and balanced country.

It also establishes the powers of the Ministry of Energy and Mining, and details the standard procedure for the granting of mining concessions and terminating mining titles.

By Article 66 the sustainable management of mineral resources existing in the country is regulated, to allow the generation of economic, social and cultural development in the long term.

Articles 67 to 71 require Environmental Impact Assessment (EIA) studies to be submitted to the Ministry of Energy and Mining and these must be approved by the Secretariat for Environmental Protection, Ministry of Energy and Mining.

20.1.1.5 KEY ENVIRONMENTAL REGULATIONS FOR MINING ACTIVITIES IN THE REPUBLIC OF ECUADOR

Another regulation that relates to environmental management in the mining sector was issued pursuant to Executive Order No. 625 and published in the R. O. 151 of 12 September 1997. The Ministry of Environment coordinates and oversees the implementation of this regulation.

Article 10 requires holders of mining rights and public sector entities to present Draft Environmental Impact Assessments, Environmental Impact Assessments and Environmental Audits relating to mining activities to the Secretariat for Environmental Protection through the Regional Directorates of Mining Studies.

Subsequent Articles contain provisions outlining regulations applicable to exploration, mining, mineral processing and artisanal mining.

20.1.1.6 REGULATIONS OF THE SPECIAL COMMISSION ON MINING CONCESSIONS IN HERITAGE AREAS AND STATE FORESTS

These regulations were issued by Ministerial Agreement No. 039, published in the R. O. No. 571 of 8 May 2002, and are intended to regulate the operation of the Special Committee on Mining Concessions in Heritage Areas and State Forests.

20.1.1.7 INSTRUCTIONS FOR REPORTING EMISSIONS PRIOR TO APPROVAL OF ENVIRONMENTAL IMPACT ASSESSMENTS IN HERITAGE AREAS AND STATE FORESTS

This Instruction was promulgated by Ministerial Agreement No. 040, published in the R. O. No. 571 of 8 May 2002, and is intended to regulate the issuance of reports prior to the approval of Environmental Impact Studies in state forest areas, through the Ministry of Environment and the Ministry of Energy and Mining.

20.1.1.8 INSTITUTIONAL FRAMEWORK

The EGX EIA and all supporting documents were submitted to the Ministry of Mines and Petroleum in Quito, Ecuador in 2006 and were approved in 2007.

20.1.1.9 INTERNATIONAL STANDARDS

The 2012 International Financial Corporation Performance Standards provide a framework for environmental assessments. The baseline studies that were already undertaken are adequate for the revised standards.

20.1.2 ENVIRONMENTAL IMPACT ASSESSMENT

The environmental study area is located on the western flank of the Cordillera del Condor, this relief is very irregular, and has small streams that originate in the upper part of the Cordillera and travel from east to west, to the River Nangaritza which is the main axis of the river system in the area. The study area altitude varies between 850 and 1 900 m, with slopes between 50 % and 70 % which decrease as it goes downstream along the Nangaritza River. This allows for two geomorphological areas with well-defined characteristics of relief, and are relatively deep streams and in a "V" at the top. The drainage patterns are considered dendritic to sub dendritic medium density.

The EIA was approved in 2007 and was undertaken to achieve the following:

- Establish baseline and assess the current environmental conditions, from the physical, biological, socio-economic and cultural influence of the Project area.
- Identify and evaluate the environmental impacts that occur, through a review of the Project activities with consideration of the environmental characteristics found in the area of influence.
- Develop the Environmental Management Plan (EMP) in order to establish specific actions that must be taken into account by the Project, for the purpose of preventing and mitigating the environmental impacts that could occur.
- Collaborate with the Ecuadorian Government in its management to protect the integrity of the human and natural resources of the country, ensuring proper use and sustainability.
- The EIA involved preliminary studies, field research and final reporting.

The EIA was further reviewed along with the EMP that contains the mitigation actions, during an Environmental Compliance Audit in 2011 by AmbienConsul Co. Ltd. The audit covered the period from January 2008 to 2011.

EIA baseline work completed and tasks that need to be completed are summarised in Section 20.1.3 for each environmental discipline, with the details and limitations included.

20.1.3 BASELINE ENVIRONMENTAL STUDIES

This section presents the status of the on-going baseline studies that are being undertaken by AmbienConsul Co. Ltd. The environmental and social baseline studies undertaken to date are detailed in the 2006 EIA (AmbienConsul Co. Ltd., 2006). Baseline studies were completed for the biophysical and social environments.

The methodology used to complete baseline studies was to assess and analyse the environmental assessment study area, and evaluate impacts from different mining activities and different elements of the Project. This consisted of the following;

- Description of the current environmental conditions of the Project area;
- Identification of potential impacts (matrix cause and effect);
- Weighting and rating of impacts identified;
- Analysis of results; and
- Determining significance of impacts.

Baselines studies and data collection continue to be undertaken to supplement the Project EMP and on-going monitoring. The baseline studies are used as the benchmark against which to assess the potential impacts of the proposed Project. The EIA is audited for compliance with environmental legislation and standards; it is implicit that there is compliance with in-country legislation and international treaties to which Ecuador is bound. The Project is in an advanced mining exploration phase and was granted an Environmental Licence for advanced exploration Article 267 in February 2007 by the Minister for the Environment.

The EIA and the Environmental Licence for the advanced exploration mining phase 2007 form part of the Environmental Compliance Audit. The audit was first undertaken one year after the licence was issued and occurs every two years thereafter. The lead time for the last audit was March through to May 2011. The audit was undertaken by AmbienConsul Ltd and reported compliance with the elements of the EIA and the Environmental Licence conditions and requirements. (AmbienConsul Co. Ltd., 2011)

The Terms of Reference from the authorities and the Minister for Environment were completed in June 2014. Following the audit, findings of the Environmental Compliance Audit will be reported in a conformance report.

20.1.3.1 SOILS AND LAND USE

The following work has been undertaken for the soils and land use baseline:

- Soil quality was determined by undertaking a sampling and analysis program, where samples were taken from the surface area up to 20 cm. Laboratory analysis on the samples was performed and results are contained within the 2006 EIA; and
- Interpretation of the regional topography and geology from available sources, including aerial imagery.

A review of available information indicates that soil types are likely to comprise clayey, reddish brown with kaolinite and goethite in various mixtures.

- Soil results indicated the soils are acidic with pH ranging between 4.64 and 5.30 with low ranging levels of organic matter. Total nitrogen is low, as well as phosphorus, while there were moderate levels of potassium;
- Soils are classified as having low agricultural potential due to low levels of nutrients and high acidic pH; and
- The mine area is potentially suitable for the maintenance of natural forests, however there would be a thin organic soil layer required.

20.1.3.2 SURFACE WATER

The following tasks were carried out as part of the surface water baseline investigation:

- Desk study of the local rivers and waterways;
- Initial field visits and familiarisation visits to each of the rivers and streams that are likely to be affected by the proposed mine development. An assessment of surface water health (i.e. in-situ sampling) was conducted to determine the physical and chemical quality and the presence of mercury or cyanide in the water of the Robert Ruiz, Piedras Blancas, Bocamina, Fierosos and The Pangui streams, and taken to a laboratory in Quito for analysis;
- The identification of local receptors and interrogation in the field, including water users and possible surface water dependant habitats was conducted;
- Surface water drainage assessments;
- Climatology assessments were undertaken to assess the characteristics of the climate and influence of the Project; and
- Laboratory tests were conducted to establish the physical and chemical quality of the water with reference to the Eligible Quality Criteria for the Preservation of Flora and Fauna in freshwater. Full results are contained within the 2006 EIA.

Key findings emerging from the baseline appraisal are:

- The results and analysis of the streams concluded parameters such as pH are outside the permissible limits;
- The Pangui and Bocamina Rivers had pH within permissible limits however all rivers crossing the concessions were concluded to have poor physical and chemical quality for the preservation of flora and fauna;
- Water quality along the rivers crossing the concessions had turbidity, colour and total suspended solids above permissible limits. This was the baseline, even though the area was raining every day during the sampling;
- It was determined the presence of free cyanide in the waters of the streams Fierosos and The Pangui exceeded the permissible limits; and
- The streams and rivers within the vicinity of the mine site exceeded the permissible levels for mercury.

20.1.3.3 TERRESTRIAL ECOLOGY (FAUNA)

The fauna study at the site focuses on fauna biodiversity and the aim of field and desktop analysis was to determine whether any biodiversity related issues were notable during the survey and to determine whether any further studies would be required. To establish the fauna composition, habitats and significant species located in areas of influence of the Project, the following sampling techniques for mammalian fauna assessments were used:

- Fingerprint Identification and other traces, which aimed to search for and identify footprints and other traces to determine the presence of a mammalian species by other means for example, searching traces of burrows, feeding, bones, faeces, marks, urine and identifying sounds and vocalizations;
- A number of survey and sample collections were conducted in the morning, during the day and at night, with random observations being made.
- Direct Observation, which is one of the most basic techniques in terms of technical equipment required because, depending on the case, this only used binoculars and headlamps with halogen spotlights;
- Mist nets were placed in different parts of the study area, to catch flying mammals (bats), and 50 live traps in different parts of the study area were established in order to capture small mammals;
- In addition to the methodologies described, surveys were undertaken by employees working on the Project, to complement, strengthen and identify certain species not recorded during the field surveys; and

- To determine the conservation status of the species, a list of species in the area was prepared, that fall within different categories by IUCN (2008), CITES (2008) and the Red Book Mammals of Ecuador (2001).

All individual mammals captured were recorded and released at the site of capture.

During site visits 14 species of mammals belonging to 12 genera, eight families and seven orders, were discovered. Chiroptera being the most representative group with six species which is equivalent to 50 %, followed by the order Carnivora with three species of 25 %, the order Rodentia, Artiodactyla, Cingulata, Lagomorpha and Didelphimorphia which equals 8.33 %.

Among the species recorded in the study area were a common species of bat called the genus Anoura sp., Artibeussp, and also the Carolliasp and Sturnira recorded as rare species. Also recorded were squirrel Sciurus sp.

Within the study area there are species that were identified, which are recorded in different lists of IUCN, CITES and the Red Book of Mammals of Ecuador. Such as the ocelot Leopardus sp., Puma Puma concolor, as vulnerable and in Appendix II of CITES.

20.1.3.4 BIRDS

For the assessment of the avifauna (birds) sampling points were determined to take recordings of songs before dawn astronomical (05:30 - 06:30 am), and direct observation tours with the use of binoculars (8X42).

To determine the conservation status of the species on the list, species were identified that fall within different categories by IUCN (2000), CITES (2000) and Red Book of the Birds of Ecuador (2002).

There were 39 species of birds recorded, 33 genera, and 15 families belonging to six orders. The order that most species were recorded in was the Passeriformes with 21 species equivalent to 53.84 %, and the order Apodiformes which had five species equivalent to 25.64 %, and four species of Piciformes equivalent to 10.25 %, two species of Falconiforames, equivalent to 5.12 %, and finally the species Psittaciformes Columbiformes which is equivalent to 2.56 %.

Registered common species of the genus are the hummingbirds Phaethornis sp., Agaiocercus sp., Colibri sp. and Dorífera sp., and Cyanicollius Tangara tanagers and T. collophrys, and red blackbird Turdus fulviventris.

Among the species recorded as rare within the study area are the Scytalopus ass sp. and also the woodpecker Piculus rubiginosus.

20.1.3.5 AMPHIBIANS

Transects were developed, which is considered one of the most effective techniques for the study of population density of amphibians and reptiles in different habitats and different altitudes.

Transects were established 100 metres long, 2 metres wide on each side, transects were placed randomly, and sampling was conducted during the day and evening sessions.

In this study site a total of 11 individuals were recorded, corresponding to four species of amphibians and two species of reptiles. Among the amphibians recorded were the family Hylidae with four species (66.66 % of total). In reptiles the recordings identified two suborders; order Squamata, Sauria with Polychrotidae family and with the family suborder Serpentes Colubridae (16.66 % each).

In the case of amphibians and lizards, most are insectivorous, with their main food being ants.

Most of the amphibians and lizards found form part of the diet of colubrid snakes and some birds of prey.

All species reported in this study are, according to IUCN criteria, in category LC (Low Concern). None of the species are found in any of the Appendices mentioned in CITES (Convention on International Trade in Endangered Species).

20.1.3.6 AQUATIC FAUNA

Sampling was carried out between 6:00 am to 9:00 am and 6:00 p.m. to 10:00 pm, using a cast net fishing device as the main rig (2.5 m radius and mesh eye 2 cm).

When specimens were collected, the recordings of some morphometric measurements of each, such as the Standard Average Length LEP (which gives an idea of the average size of the species), were undertaken. Photographic records were kept. Definitive identification of species was then performed in the Department of Ichthyology Ecuadorian Museum of Natural Sciences, using specific taxonomic keys and reference material.

To determine how to categorize abundance of fish species, there are four known classes that exist, which depend directly on the number of individuals recorded. This consists of plenty: ten or more individuals; common: five to nine individuals; common short: two to four individuals; and rare: one individual.

Recordings of aquatic fish identified a total of seven individuals, corresponding to an order, two families and four species. Only one fish species, Characiformes, was present at the site recorded. The most abundant species was Astyanax abramis with three individuals.

Rare species recorded in the study are Astyanax cf . bimaculatus and Hoplias malabaricus, abramis and Bryconamericus Astyanax sp.

20.1.3.7 AQUATIC MACROINVERTEBRATES

During the field phase sweeps were performed every 10 m along a transect of 50 m, using a Red D net, which consists of a triangular lattice subject to a wooden handle with a 250 mesh or light, and a mouth input of about 30 cm in diameter. The collected samples were placed in ziploc bags. This sampling can trap insects such as Hemiptera, or those attached to submerged stems and leaves.

During the laboratory phase the species were cleaned using a white tray, checking details until they were completely separated and those identified as aquatic macro-invertebrates were placed in plastic bottles with their respective label.

Two sampling points were established within two of the significant streams. The results identified that there were 16 individuals, belonging to nine genera and eight families.

Both streams were found to have diversity and a medium-to low sensitivity index.

20.1.3.8 FLORA ECOLOGY

The flora study site focuses on flora biodiversity and the aim of field and desktop analysis was to determine whether any biodiversity related issues were notable during the survey and to determine whether any further studies would be required. To establish the floristic composition and structure of forest remnants located in areas of influence of the Project the following sampling techniques were used:

- Temporary plots method - To establish the floristic composition and structure of forest areas, the method was applied using plots of 0.25 ha. (50 x 50 m), logging data of all species with a diameter equal to or greater than 10 cm at breast height (1.30 m above the ground). This method allowed quantitative and qualitative determination of plant diversity, structure (horizontal, vertical), floristic composition and conservation status of forest ecosystems within the study area;

- Collections at random. At the sampling points established, random collections were undertaken, to log data and qualitatively identify fertile plants (flowering plants, fruit, and ferns);
- Observation tours - In addition to the plantation areas, grasslands, crops and orchards, observation trips were also made and recording of species in the ecosystems;
- Surveys - The information on the uses of local flora was obtained through conversations and interviews with attendees of the town at the time of fieldwork; and
- Recording of plant species - obtained through direct observation, through conversations and interviews and using chips containing the following information: blooming, fruiting, and related wildlife.

The following flora biodiversity was identified through fieldwork investigations and laboratory studies, associated with location of the Project study area:

- According to the classification the study area corresponds to the classifications of piemontano evergreen forest, whose trees consist of epiphytes found on the crests of the hills southeast.
- In the study area 71 species were recorded belonging to 31 families, emergent trees are scarce in this area, however the projecting canopy species can reach 25 m high.
- Flora families with a higher number of individuals form Lauraceae had seven species, equivalent to 9.09 % Melostomataceae families, Moraceae and Orchidaceae with five species, equivalent to 6.49 %. A complete detailed list of flora identified is recorded in the 2006 EIA.
- The resource use of flora species recorded in the study area typically have several types of uses, including construction, timber, fodder, medicinal, and mythological.

20.1.3.9 AIR QUALITY AND NOISE, VIBRATIONS

The Project area contains few traditional farming activities, and there is no permanent vehicle traffic, therefore it was recorded in the 2006 EIA that no recordings or observations of significant alterations to air quality in the area were observed.

The secondary legislation of the Ministry of Environment has a standard called the Standard Air Emissions from Stationary Combustion Sources, which was assessed for permissible limits of air emissions from stationary combustion sources at the mine site. In the 2011 Audit it was reported that the Project was compliant with air emissions for stationary combustion sources and as a preventative measure EGX calibrated every engine to improve combustion and also reduce noise.

During the site visits and 2011 Audit (AmbienConsul Co. Ltd., 2011) the noise levels of different mining activities were noted, through several noise measurements with a sound level meter, recorded at the Project site and at the camp, with the results shown in Table 20-1.

Table 20-1: Noise Measurements from the 2011 Audit (AmbienConsul Co. Ltd., 2011)

Services	First dBA 10 H 00	Second dBA 14 H 00	Third dBA 18 H 00
Administration Block	62	64	62
Kitchen and Dining Room Block	66	64	64
Dormitory Block	60	58	58
Camp and Winery Courtyard	68	68	66

These levels were compared to the maximum permitted and it was concluded that they are within the limits.

20.1.3.10 LANDSCAPE AND VISUAL AMENITY

The following landscape and visual amenity tasks have been completed to date:

- Site field visits to assess the surrounding landscape and key features including local villages and views and proximities to the mine, process plant and TMF;
- An assessment of landscape quality (scenic value), to determine the relative sensitivity of the landscape to the proposed scheme; and
- An assessment of the proposed mining activities in the landscape, and visual sensitivity that could occur.

In general, the surrounding landscape and soils of the study area are potentially suitable for the maintenance of natural forests, but the soil quality is a limiting factor for agriculture and grassland opportunities. The landscape near the top of the Project location is characterized by the presence of informal mining settlements, which for more than 15 years have been growing, with people from different parts of the country especially in the provinces of Loja, Zamora, El Oro, and Azuay. Many abandoned farms exist that had been exploited in the previous five years, through mechanized extraction using backhoes, and mechanical shovels and gutters for gold gravity separation. However in 2010 a raid by the armed forces confiscated these machines and halted operating practices that were not legal.

The proposed TMF would not be visible from villages within the study area. In most cases, dense vegetation surrounding the villages is likely to provide substantial screening, obscuring views of the facilities. The TMF is proposed to be installed in a valley which is to a significant degree hidden by surrounding mountainous slopes and vegetation. The process plant will also not be highly visible to surrounding villages due to its location and setting on a flat area of land with surrounding mountains with significant forest cover.

The most attractive parts of the landscape are considered to be the river valleys which form part of the area.

There are few settlements within the boundaries of the proposed mine pits and small settlements are unlikely to be impacted or be relocated. In landscape and visual terms no red flag issues have been identified.

20.1.3.11 SOCIO-ECONOMIC

The following baseline tasks were undertaken by socio-economic consultancy, AmbienConsul Co. Ltd. (AmbienConsul Co. Ltd., 2011):

- A baseline study for the Project area around Chinapintza, La Punta, Pangui-Conguime, Santa Elena, Pachicutza and The Geraniums and along the River Cacheu corridor; and
- Surveying areas and towns by random sampling techniques, including data collected on all the villages and towns, and further assessing all demographics (age, gender, and ethnicity) data, which was tabulated and analysed.

20.1.3.12 POPULATION

The study area comprises three areas represented by informal mining associations, and a mining settlement. The first area considered corresponds to the association of informal miners in La Herradura which also includes families who work in the area known as Chinapintza.

The main area of settlement is in the north-eastern part of the concession at Puerto Mining Chinapintza and has a road connecting it to La Punta. There are approximately 300 people settled in this area.

The majority of the people within this area are migrants from Loja, Zamora and El Oro, and have come from locations where there has been a strong mining tradition.

The study area also includes small miners, The Pangui – Conguime, which was formed in 1994 and operates an area of 150 hectares, where gold mining is operating informally. This association comprises of 26 companies that perform exploitation and families consist of between one and ten people.

The third part of the study included the area of the settlement of Puerto Mining Chinapintza called The Point. This town is dedicated to the implementation of mining services and provides housing materials, housing and telephones. The creation of this town responds to the need for a storage facility and the name was established as Puerto Mining. At The Point, there is also an association of informal miners, however, this association is smaller and its operating margin is smaller.

Socio-economic surveys were undertaken by random sampling techniques, applied to the villages, and towns within the Project area of influence, and are the results are summarised in the 2006 (AmbienConsul Co. Ltd., 2006).

20.1.3.13 SOCIAL INFRASTRUCTURE

In general, populations within the vicinity of the mine site and the surrounding corridors are characterised by a traditional Ecuadorian culture.

Access to education is limited with no formal schools locally available. Children are enrolled in private schools. No health infrastructure is available in any of the villages identified in the study area. Domestic water supplies are unreliable, with many wells and pumps being obsolete and unable to provide for the needs of the communities.

The town of La Punta, is the nearest town with commercial activities that are also associated with the Project. In La Punta there are several retail stores with supplies of food, electrical goods, plumbing, and clothing being available. There are also a few restaurants, mechanics, telephone services and general supplies stores.

The town called The Point, consists of the health care centre Chinapitza, where there is a doctor and nurse service available. The most common illnesses in this area are respiratory, and also child malnutrition and intestinal issues being seen in children under five years of age.

There are shuttle services to La Punta for three companies from Zamora, Nambija and Yanzatza Union.

The towns which consist of the informal mining companies, have settled in the north and upper locations of the Project.

Within these three mining settlements, the population lives in conditions of apparent rural marginality and it is important to note that within these settlements there is a strong economic differentiation among its inhabitants, between the owners of mining operations and mining society members who generally have an improved economic position which is higher than those who work as labourers in these mining activities who generally have limited income and are part of a poor socio-economic demographic.

Within the mining settlements for these informal miners there is a private school called "Cenepa Heroes" which consists of one teacher with 25 students, and located in the area of the Horseshoe.

They have no health service, and a doctor is also transferred from Chinapintza Mining Puerto La Punta, to these settlements on an as required basis.

There is piped water however it is untreated, and no drainage. There are no solid waste management facilities.

The settlements have electricity via the national system, however there is no conventional phone service.

In relation to the appearance of housing in these settlements, structures are similar and formed mixed type viviendísticas buildings and typically have a tin roof.

Due to the proximity to the Peruvian border of the informal mining towns, many of the workers employed in mining are from Peru.

Within the informal mining settlements, there are volleyball courts, and in the case of The Pangui Camp there is a cement court and in La Herradura there is a communal house.

It is important to note that since 2006 other informal mining operations have developed in Conguime River terraces, for gold sand exploitation using heavy machinery (excavators, tractors, dump trucks) and gravity gutters to recover the ore. This situation resulted in the Ecuadorian Government forces taking action and undertaking an eviction in 2010 to remove these informal mining activities, which have caused environmental damage to the area.

Mining associations have gained access to resources to carry out works to improve living conditions in the villages where they live, whether it be a court or a communal house, which can be performed with the support of the provincial council or municipal governments and even NGOs. In the case of La Pangui is managing a project for the construction of ponds for water treatment to reduce environmental and health impacts, in the case of the horseshoe construction has been made of the water tank which serves supply for consumption.

20.1.3.14 ARCHAEOLOGY AND CULTURAL HERITAGE

A site visit to the areas and activities of the mining was assessed for either higher or lower potential to contain archaeological sites. However in the area of influence there is no information about the presence of archaeological sites or evidence of this. In the absence of clear records, sites remain undated and little can be offered regarding their present importance and cultural function.

20.2 ENVIRONMENTAL PERMITTING

Both permits and EIAs are required for a typical open pit mine operation in order to move from exploration though to operations and closure. The Project is in the advanced exploration mining phase and the Licence and EIA reflect this. Exploration requires various permits especially those which propose to use significant surface or groundwater resources during drilling, building roads, diversion of water for use. The list of permits required for these activities is identified in Table 20-2 and the three permits listed below are considered the priority permits.

- Environmental Licence (EIA and risk analysis approved);
- Licence of Forest Wood Use; and
- Water Concessions and Concession of Water Benefice Right.

Table 20-2: List of Permits

Permit	Issuing Authority	Requirements	Approximate approval period	Comments
Environmental Licence	Ministry of Environment	There are several environmental studies which depends on the activity and the phase of the mining concessions	8-12 months	All the mining activities should be licensed
Environmental Impact Study	Ministry of Environment	Determine the phase and the activities; this study is necessary in order to execute simultaneous activities on the small mining regime and exploitation in medium and general regime	6 months	
EMP	Ministry of Environment	It depends on the activities	6 months	All the environmental studies require an EMP

Permit	Issuing Authority	Requirements	Approximate approval period	Comments
Environmental Form	Ministry of Environment	Depending on the activity that will be executed	60 days	Applies to initial exploration in large scale mining or artisanal mining regime
Environmental Impact Statement	Ministry of Environment	Depending on the activity that will be executed	60 days	Applies to advanced exploration in large scale mining
Environmental Audits	Ministry of Environment	Depending on the activities	6 months	Applies to all phases
Economic Guarantees	Ministry of Environment	Depending of the cost of the EMP	N/A	Environmental guarantee depends on the environmental study approved
Water Use and treatment	National Water Secretary	Depending of the project	6 months	It depends on the project and the activities
Closure Plan	Ministry of Environment	Depending on the project and the mining regime	10 to 12 months	It depends on the project and the activities
Social Participation Process	Ministry of Environment	Depending of the project and the type of permit required	N/A	It depends on the project and the activities
Certificate of Intersection	Ministry of Environment	Depending of the geographical location of the mining concession	1 month	If the mining concession is in a protected forest area a certificate of environmental liability is needed
Environmental Control	Ministry of Environment	Depending of the approved studies	N/A	Depending of the type of licence granted
Construction of Facilities	Ministry of Environment	Depending of the project and the studies approved	N/A	Depending of the project and the studies approved
Benefit Plants	Ministry of Environment-Sectorial Ministry	Depending on environmental studies	N/A	Depending of the authorization granted
Water Concessions	National Council for Hydrological Resources	Required for water rights	Up to 180 days	
Water Concession of Benefice right	National Council for Hydrological Resources	Required for use of water and any amendment must have prior authorisation of the National Council for Hydrological Resources.	From 45 days up to 90 days; permission granted	
Licence of Forest Wood Use	Ministry of Agriculture and Ministry of Environment	Use or clearance of forest	Up to 30 days and up to 45 days when applications are submitted	In the case of the Ministry of Environment they require Environmental Plans to be approved.

20.3 ENVIRONMENTAL, SOCIAL AND COMMUNITY IMPACTS

20.3.1 STAKEHOLDER ENGAGEMENT

EIA public consultation took place when the Project was announced and meetings occurred at different levels of stakeholder engagement with representatives from EGX, the Ecuadorian Government and environmental consultants. Progress and results were reviewed in the 2011 Audit (AmbienConsul Co. Ltd., 2011).

20.3.2 IMPACTS – SOCIAL AND ENVIRONMENTAL

An Environmental and Social Impact Mitigation Plan was produced by EGX and formed part of the 2011 Audit (AmbienConsul Co. Ltd., 2011). It addresses all elements of the Project activities and the likely impacts have been assessed for significance with mitigating actions to address the impact documented as an action plan.

Environmental issues and impacts extracted from the relevant documents and from site assessments, including a site visit in October 2013, are summarised as follows:

- Water extraction from the river;
- Land use for an open pit mining operation;
- Land use impacts from tailings storage;
- Land use impacts from waste rock storage and infrastructure;
- Discharge of mine water, process water, and storm water offsite;
- Water impacts on local waterways – Yapi, Pachicutza and Nangaritza Rivers;
- Potential noise and vibration pollution;
- Loss of wildlife (fauna) and habitat;
- Loss of flora;
- Sewage treatment and discharge;
- Biodiversity loss, the Cordillera del Condor is a highly biodiverse area with many endemic and endangered species;
- Potential landslides;
- Water management, due to high rainfall;
- Issues associated with national parks and protected areas; and
- Impacts due to use of cyanide.

Community and social impacts, and associated issues identified through the exploration mining phase to be addressed in on-going consultation are outlined as follows:

- Engagement with local indigenous Shuar and Saraguro Kickwa populations and non-indigenous populations;
- Instances of local unrest and opposition due to mining projects, or territorial disputes between the borders of Ecuador and Peru;
- Proximity of local villages and impact of light, noise or air pollution;
- Impacts due to increase in demand for local resources on services such as health, police, roads or infiltration of population growth due to increased mining activities to the local villages; and
- Impacts on preservation of cultural heritage and archaeological artefacts not currently identified through absence of historical records.

The potential impacts have been addressed in the EMP which documents mitigation actions that are monitored and audited as part of on-going environmental licence requirements. The community and social impacts identified are also under on-going monitoring with on-going stakeholder engagement plans and a dedicated public relations manager and team to ensure community and social issues are identified and plans implemented to continue engagement with the local communities.

20.4 WASTE MANAGEMENT

20.4.1 WASTE MANAGEMENT PLAN

A Waste Management Plan has been developed to establish guidelines for the management and disposal of waste generated by the Project.

The Waste Management Plan covers the following aspects:

- To eliminate or minimise the impacts generated by the waste in the environment and the health of communities, contractors and employees;
- Identify the waste generated by the different activities and provide guidelines for proper handling and disposal;
- Disseminate and train staff on the proper handling, transportation, treatment and disposal of the various wastes generated in the Project; and
- Provide the tools to demonstrate the proper management of waste.

Waste categories have been identified within the Waste Management Plan with ratings of waste classified and reported in the 2006 EIA.

20.4.2 TAILINGS LOCATION

Tailings production is high at 10 Mt/a, and as the mine site is located in a mountainous and well vegetated region, there are severe topographic constraints on finding a suitable location for the TMF. Based on a review of potential candidate sites within a nominal distance of 10 km from the process plant, the site selected is the only one suitable in the area. The preferred site is located on land south of the village of Guazimi and west of the north flowing Nangaritza River, between about 6 km and 10 km to the north-northwest of the process plant. This area is reasonably favourable in terms of topography, it avoids the main drainage routes, and does not directly impact on the nearby village of Guazimi. A brief review of the alternative options for slurry tailings disposal in the area surrounding the process plant are summarised as follows:

- The mine and the process plant lie on the eastern side of the Nangaritza River in the foothills of a major mountain range. Surface levels rise from about 850 m amsl to 2 125 m amsl in this region, and the topography is generally unsuitable for slurry tailings storage.

- There is an area of relatively flat ground east of the Nangaritza River and immediately north of the processing plant site. However, two mountain streams drain into this area, and potentially a TMF constructed in this area would encroach over the open pit and process plant area.

- Topographic levels are less severe to the west of the Nangaritza River. An area of relatively flat-lying ground lies between the village of Zurmi, which is located about 6 km to the northwest of the process plant, and the Nangaritza River. However, the area available is less than the preferred site, the village would have to be relocated, and the area is fed by streams from the surrounding catchment.

The Nangaritza River is a meandering river, and abandoned channels are visible on the western banks of the current river course. It appears that the river course is currently tending to realign to the east of its historical location. There is no information available on the hydrology or meteorology of the proposed tailings storage site, or of the geology and ground conditions. For preliminary costing purposes it is assumed that there are no natural hazards at the preferred TMF location that would prevent the use of the area for the construction of the facility and the storage of mine tailings.

The TMF location has been selected to store the 10 Mt/a production. For subaerial tailings deposition and an upstream embankment raise approach, it will be necessary to limit the rate of rise of the tailings surface to about 2 m per year, maintain a dry tailings beach against the embankment, and to pursue an active dewatering strategy by decanting excess supernatant water and discharging it to the environment. The selected TMF location is considered to be the most appropriate location to meet these design criteria. The footprint area is over 400 ha, which is a rule of thumb guide to the area needed to achieve a low rate of rise for 10 Mt/a production.

Surface water will be diverted away from the storage area by interceptor ditches and stored in settling ponds prior to release to the local drainage courses. Supernatant water will be removed from the impoundment area by a series of fixed decant structures linked by causeways to the external embankment. The cells will be net accumulators of water due to the high rainfall in this area, and excess water should be removed and discharged to the environment via settlement ponds.

20.4.3 TAILINGS DESIGN

Tailings will be delivered by pipeline to the main embankment, and then routed around the main embankment structures. Tailings will be discharged from multiple spigot offtakes located at regular intervals along the crest of the embankment, so that beaching can take place and water can be segregated from the slurry. The initial modelling has just considered deposition from the embankment for the sake of simplicity, but consideration could be given to cycling the deposition points around the whole storage area. The embankment will be constructed using a cut to fill approach, whereby fill materials are derived from across the impoundment area, but in a way that does not compromise the integrity of the containment structure. The starter embankment will incorporate an internal drain and an upstream liner with key trench for seepage control, and a geomembrane on the upstream face for erosion control. It is assumed that the foundation strata beneath the embankments comprise strata of reasonable bearing characteristics, and that fill materials for construction of the starter embankment and individual raises can be located from within the impoundment area.

The starter embankment will be raised downstream in stages to the 862 m crest level, commencing with a starter dam of sufficient crest height to provide 18 months of storage. The starter dam crest level is determined from the modelling results. The predicted dry density of the deposited tailings is relatively low at 1.1 t/m^3. However, by adopting appropriate tailings management practices relating to thin layer deposition and aggressive water recovery practices, it should be possible to improve the density characteristics, and an in situ dry density value of 1.4 t/m^3 has been used for analysis.

20.4.4 TAILINGS MODELLING RESULTS

The storage characteristics of the proposed TMF have been assessed using Rift TD, proprietary tailings storage modelling software, assuming for simplicity a single embankment structure. The results are as shown in Figure 20-1. The maximum height of the starter embankment is about 11 m, and the volume of embankment fill is about 1.5 Mm^3.

The starter embankment would be raised downstream to a crest elevation of 858 m (20 m high), until the rate of rise is essentially steady – at just over 2 m/a, and then an upstream raise strategy would be followed for the remainder of the life of mine (assumed to be 10 years).

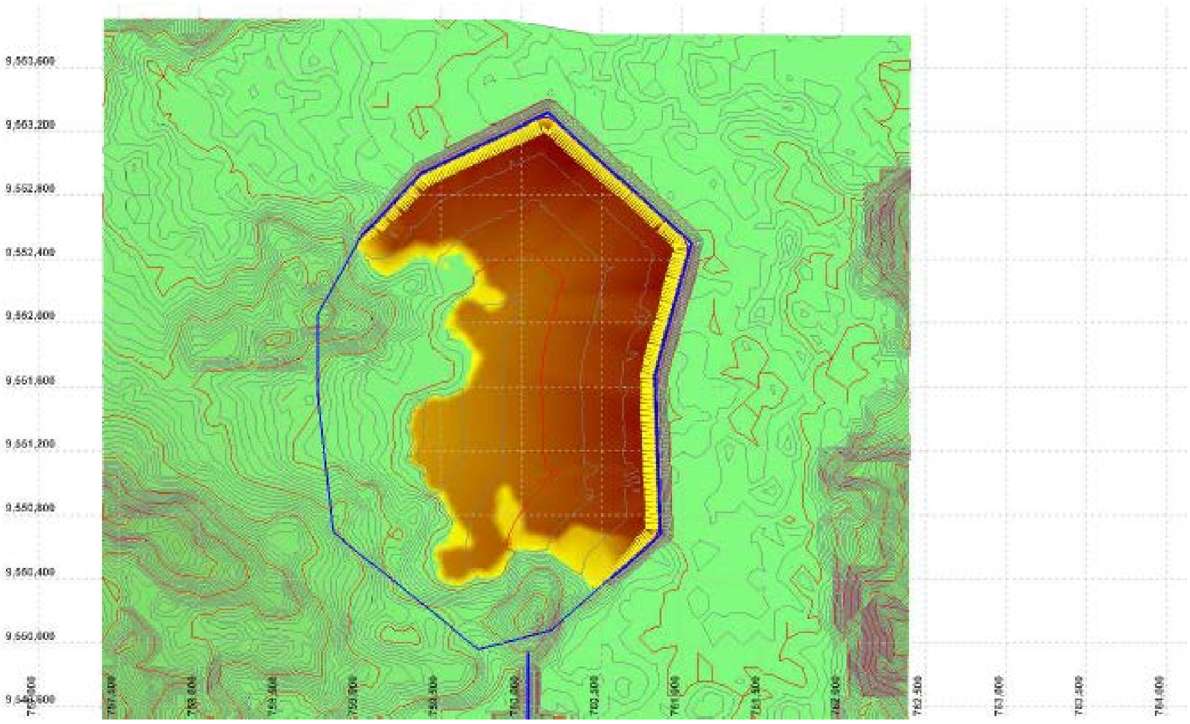

Figure 20-1: LOM Tailings Footprint Area

A nominal 2 m freeboard allowance has been assumed for the modelling exercise to accommodate extreme rainfall events and wave run up. The results of the modelling are presented as storage-elevation curves on a tonnage and volumetric basis, together with rate of rise of the tailings beach. The filling strategy is dictated by the requirement for upstream embankment raises. The main objectives with this approach are to form a beach above water onto which the embankment raises can be constructed. The rate of rise of the tailings beach must be low – preferably at about 2 m/a – to allow subaerial beaching and the formation of a dense tailings foundation to support upstream raising. The overall slope of the upstream raise embankment is set at 1(V):4(H). Individual raises have not been modelled for this exercise although for costing purposes it is assumed that 4 m high embankments would be constructed. Experience on other sites indicates that embankment slopes of this gradient or shallower are more likely to be stable under static and dynamic liquefaction scenarios for upstream raises.

20.4.5 WATER MANAGEMENT

Overall the tailings disposal operation will be a net accumulator of water due to the high precipitation in this area. Fixed decant structures set well away from the peripheral embankments near the central sections of the storage facility will be used to remove surface water. The size of pond around the decant will be determined by the settlement characteristics of the tailings. At the starter embankment, a freeboard allowance of 2 m will be maintained to cater for storm events and wave run up. The upstream embankment raises will adopt a similar freeboard allowance to determine the timing of the raises. For the purposes of this study, water will be removed from a series of internal fixed decant structures and recycled back to the processing plant via a separate settlement pond. The settlement pond will contain a spillway to permit discharge of excess water to the environment.

The storage cells will be net accumulators of water. Provision has been made for decanting surface water for recirculation or discharge to the environment, and for spillways. The design of separate settlement ponds is not covered by this report.

20.4.6 DETAILED DESIGN

Further investigations and studies are recommended to progress the design:

- A trade off study (including testwork) to evaluate the potential benefits and limitations of a Paste and Thickened Tailings disposal strategy, including updated siting study.
- Testing to quantify the potential for Acid Rock Drainage (ARD) development within the tailings.
- Geotechnical testwork on the tailings to establish settlement, strength and consolidation parameters.
- Preliminary site investigations to evaluate the ground conditions (including groundwater) at the proposed TMF site, and to locate borrow materials.
- Further studies, e.g. flood study for the Nangaritza River, water balance studies, topographic survey.

20.5 REHABILITATION AND CLOSURE

A plan to rehabilitate affected areas due to mining activities has been identified and aims to restore the environmental components affected by advanced exploration phase activities.

For the rehabilitation of vegetation cover on the mining exploration phase platform, native species from the area will be used. Rehabilitation of areas where re-vegetation is to be used to sustain grasses and legumes for soil enrichment is also addressed. The objectives of the rehabilitation and closure plan are as follows:

- To rehabilitate areas affected by advanced exploration activities; and
- To re-vegetate areas where vegetation cover was removed.

Immediately after the mining exploratory drilling phase ceases, rehabilitation is proposed to be undertaken on the platforms that were opened to rehabilitate the slope and topography, by filling with the same substrates that were extracted, and then by replacing the organic soil layer prior to re-vegetation.

Approximately 143 platforms were opened in this period of exploration drilling, which will be rehabilitated. The edges and lower slopes to the platforms will be rehabilitated by stabilizing the slopes with tables and wooden stakes or iron nails, and then planting of native seedlings to be produced in nurseries, as well as broadcast seeding native grass seed.

A full Project rehabilitation and closure plan will be developed which will detail how the open pit and associated infrastructure will be managed in terms of mine closure. It is proposed that co-rehabilitation of the open pit and TMF will be planned and executed as the operation progresses. Co-rehabilitation, which involves planning for closure from the outset and progressively rehabilitating areas as the mine matrix is worked out, has a number of benefits including:

- keeping working areas to a minimum which reduces risk of soil erosion and exposure to dangerous excavations; and
- including costs of rehabilitation into operational costs rather than increasing the funds held in a closure cost account or bank guarantee.

Measures detailed in the EMP and the Rehabilitation Plan will be implemented as mine specific closure takes place. Monitoring activities will be undertaken throughout, and will be continually updated.

A significant area of land will require reclamation to some degree. A preliminary closure management plan should be prepared in the future to better delineate estimated costs of rehabilitation and closure. The following discussion describes considerations that were used for cost estimation of rehabilitation and closure.

- The TMF is intended as a permanent long term feature. For the mine/tails areas, at the conclusion of operations, allowance has been made to cap the tailings with clay to minimise contaminated runoff.
- Dust control measures would be incorporated in the Project design. Any potential dust on and around the site may require attention during site rehabilitation.
- It is anticipated that the access roads to the site would remain in place for local community use, all other access and haul roads would be ripped and re-graded, where required, to blend in with local topography. Safety berms and drainage infrastructure will be removed or graded, where applicable.
- For the purpose of this PEA, it is assumed that open pits, and waste rock dumps will remain as permanent features with egress routes maintained in the event of entry and stormwater diversions maintained around these features. All plant, infrastructure or facilities are to be removed and ingress blocked. Limited re-grading has been allowed for to promote re-vegetation.
- It is intended that equipment be dismantled in such a manner that it may be salvaged as much as possible.
- Passive re-vegetation is proposed to promote soil stability and return of local species.

20.5.1 POST-CLOSURE MONITORING AND MAINTENANCE

An allowance for routine monitoring including personnel costs and laboratory fees is included within the estimate. A closure management plan, including monitoring plan and time frame, will allow for improved accuracy of the rehabilitation and closure estimate. The monitoring system will also provide an early warning system to identify unforeseen issues post-closure.

SECTION 21 CAPITAL AND OPERATING COSTS

21.1 COST ESTIMATE BASIS

21.1.1 METHODOLOGY

The cost estimate was generated from supporting engineering quantities and cost information derived from the following sources:

- Historical cost information sourced from in-house and commercial databases;
- Quotations from equipment suppliers;
- Factored estimates based on mechanical equipment costs, with factors from in-house databases and estimating publications;
- Rates from local service providers; and
- Client derived data from local contractor quotations.

Both capital and operating cost estimates were prepared in mixed currencies and reported in United States dollars (USD). The currency exchange rates used for the cost estimate are presented in Table 21-1. These rates were base-lined using the three month average between 1 January and 1 April 2014.

Table 21-1: Currency Exchange Rates

Currency	Rate
AUD (Australian Dollar)	1.1202
CAD (Canadian Dollar)	1.1048
EUR (Euro)	0.7263
GBP (Great British Pound)	0.6006
USD (United States Dollar)	1.0000
ZAR (South African Rand)	10.4674

21.1.2 ESTIMATE CLASSIFICATION

The prepared estimate is classified by GBM as a Class 5 estimate with a +50 % / -30 % accuracy. The respective range of the GBM Class 5 estimate compared to other classification systems is presented in Table 21-2 for comparison.

Table 21-2: Estimate Classification Comparison

GBM [based on AACE]	ANSI Standard Z94.0	Association of Cost Engineers (UK) [ACostE]	American Society of Professional Estimators [ASPE]
Class 5	Order of Magnitude Estimate -30 %/+50 %	Order of Magnitude Estimate Class IV -30 %/+30	Level 1

21.1.3 ASSUMPTIONS

The project currently assumes additional land acquisition and surface rights will be obtained in the future to accommodate proposed infrastructure such as access roads and power transmission lines. The potential costs of such acquisitions are not included within the estimate.

The mining fleet requirements were developed assuming a total of 8 640 operating hours planned per year.

Mechanical equipment was sized based on an operation of 365 days per year and 24 hours operation with 92 % availability.

21.1.4 EXCLUSIONS

This estimate has been prepared beginning from the point of project approval. Therefore, costs incurred during project development are excluded. Major project development components are:

- Metallurgical testwork;
- EIA;
- PFS / BFS engineering;
- Exploration drilling;
- Project development social and environmental programs;
- Land acquisition and right of way costs; and
- Permits, licences, bonds or legal and administrative costs associated with government mining and environmental regulations. This includes reporting requirements during operation and related administrative costs.

Additionally, no allowance has been made for:

- Cost escalation;
- Currency fluctuations;

- Currency hedging;
- Insurance;
- Container demurrage costs;
- Product transportation security;
- Containment, monitoring or treatment of waste rock in the event that acid rock drainage or metal leaching are applicable; and
- Hydrogeological monitoring, dewatering or stormwater control measures.

21.1.5 CONTINGENCY

Contingency is a cost element of the estimate used to cover the uncertainty and variability associated with unforeseeable elements not defined in the project scope. A preliminary risk identification and assessment process was conducted and significant project risks identified that may impact upon the CAPEX and OPEX estimates, included:

- Difficulties encountered in land acquisition;
- Lack of locally available skilled labour;
- Insufficient housing;
- Locally sourced materials;
- Resource grade variability;
- Grid power supply – including route, network upgrades, availability, project support by power authority; and
- Manual operations with limited standbys.

21.1.5.1 CAPEX

Contingency has been estimated based on the level of project definition and, as a Class 5 estimate, the minimum standard contingency is considered to fall between 25.00 % and 35.00 % of the total direct capital costs. The estimated contingency equates to 16.55 % of the LOM direct capital costs and 17.07 % of the initial direct capital costs.

21.1.5.2 OPEX

A contingency of 2.0 % has been applied to the process operating costs due to the level of scope definition.

21.2 CAPITAL COST ESTIMATE

21.2.1 SUMMARY

Table 21-3 shows the estimated initial capital costs for the project, including the cost breakdown for each project area.

Table 21-3: Initial Investment CAPEX Estimate

Cost Centre	TOTAL [USD]	General [USD]	Mining [USD]	Processing [USD]	Waste Manage-ment [USD]	Infrastructure [USD]
Total Capital Investment	598 867 838					
FIXED CAPITAL TOTAL	553 634 117					
DIRECT TOTAL	392 560 935	40 164 706	93 721 828	159 727 071	36 900 000	62 047 329
Civil	18 426 765		3 689 908	7 908 690		6 828 167
Control and Instrumentation	3 514 973			3 514 973		
Earthwork	54 736 534		203 389	2 636 230	34 900 000	16 996 915
Electrical	50 764 475		1 124 627	13 181 150		36 458 698
Mechanical	131 266 919	38 664 706	1 470 582	87 874 332	2 000 000	1 257 299
Mobile Equipment	80 393 000	1 500 000	77 398 000	1 495 000		
Piping	19 618 545		1 309 071	18 309 474		
Platework	13 181 150			13 181 150		
Structural	20 658 574		8 526 252	11 626 072		506 250
INDIRECT TOTAL	161 073 182					
EPCM	29 883 911					
Field	27 479 265					
Contingency	103 710 006					
WORKING CAPITAL	45 233 721					

Table 21-4 shows the estimated life of mine capital costs for the project including cost breakdown for each project area.

Table 21-4: LOM Project CAPEX Estimate

Cost Centre	TOTAL [USD]	%
Total Capital Investment	**826 004 178**	100.0%
FIXED CAPITAL TOTAL	**780 770 457**	94.5%
DIRECT TOTAL	**567 057 935**	68.7%
General	53 164 706	6.4%
Mining	211 218 828	25.6%
Processing	159 727 071	19.3%
Waste Management	80 900 000	9.8%
Infrastructure	62 047 329	7.5%
INDIRECT TOTAL	**213 712 522**	25.9%
EPCM	35 583 911	4.3%
Field	39 694 055	4.8%
Contingency	138 434 556	16.8%
WORKING CAPITAL	**45 233 721**	5.5%

21.2.2 INDIRECTS

21.2.2.1 EPCM

The Engineering Procurement and Construction Management (EPCM) costs for the LOM have been estimated at USD 35 million, the magnitude of which was verified against other similar GBM projects and is considered within the industry accepted range for a Class 5 estimate. Estimated construction costs are not included within the EPCM estimate but rather are accounted for as part of the direct CAPEX costs. EPCM was selected as the method of the project delivery due to the current level of scope definition. It should be noted that the project could be executed as an Engineering Procurement Construction (EPC) project with Condormining or their representative acting as a Project Management Contractor (PMC). Assuming all other things are equal, if the project delivery method chosen is an EPC / PMC model it may be more expensive when compared to an EPCM due to the profit margins of the EPC contractor.

21.2.2.2 FIELD INDIRECT COSTS

Field costs total 4.8 % of the LOM total capital cost. This estimate includes allowances for:

- Field labour (supervision, accounting, field engineering, staff engineering, service personnel);
- Construction support (concrete batch plants, construction supplies, construction equipment, temporary access to site during construction, temporary buildings); and
- Labour benefits (fringe benefits and construction camp).

21.2.3 WORKING CAPITAL

The working capital has been assumed as approximately three months of an averaged year's operating cost, which is within the industry standard range.

21.2.4 CAPITAL COST ESTIMATE – MINING

Direct mining costs have been categorised into six areas as outlined in Table 21-5. This estimate was completed from first principles and represents a high level estimate only.

Table 21-5: Direct Capital Cost Estimate – Mining [USD]

Cost Centre	TOTAL	Drilling and Blasting	Primary Equipment	Support Equip-ment	Ancillary	Mainten-ance	Infrastruc-ture
Civil	3 689 908						3 689 908
Earthwork	203 389						203 389
Electrical	1 124 627						1 124 627
Mechanical	1 470 582						1 470 582
Mobile Equipment	77 398 000	215 000	64 477 000	8 153 000	4 034 000	519 000	
Piping	1 309 071						1 309 071
Structural	8 526 252						8 526 252
TOTAL	**93 721 828**	**215 000**	**64 477 000**	**8 153 000**	**4 034 000**	**519 000**	**16 323 828**

21.2.5 CAPITAL COST ESTIMATE – PROCESSING

The direct processing costs, in Table 21-6, give the breakdown of the processing costs per project area. As can be seen, the major costs are in the milling area.

Costs for many items have been estimated based on a percentage of the mechanical equipment costs due to the high level nature of the cost estimate. Percentages were based on previous projects and industry standards.

Where not included, installation costs have been applied on a percentage basis derived from historical data as 25 % of the capital cost. Allowance has been made for the fact that some equipment/plant is priced on a turn-key basis.

Table 21-6: Capital Cost Estimate – Processing [USD]

Cost Centre	TOTAL	Crushing and Stockpiling	Milling	Rougher flotation	Regrind	Cleaner flotation and copper concentrate	Carbon in Pulp	Gold and Silver Recovery	Detoxification	Utilities and Infrastructure	Reagents
Civil	7 908 690									7 908 690	
Earthwork	3 514 973									3 514 973	
Control and Instrumentation	2 636 230									2 636 230	
Electrical	13 181 150									13 181 150	
Mechanical	87 874 332	8 495 202	41 061 422	4 693 808	6 311 891	3 128 091	6 671 512	5 763 941	10 288 617	831 896	627 953
Mobile Equipment	1 495 000	945 000									550 000
Piping	18 309 474								9 522 041	8 787 433	
Platework	13 181 150									13 181 150	
Structural	11 626 072									11 626 072	
TOTAL	**159 727 071**	**41 061 422**	**6 671 512**	**4 693 808**	**6 311 891**	**3 128 091**	**6 671 512**	**5 763 941**	**19 810 658**	**61 667 594**	**1 177 953**

21.2.6 CAPITAL COST ESTIMATE – WASTE MANAGEMENT

The detoxified tails from the process are intended to be pumped to the TMF which is proposed to be sited approximately 10 kilometres from the process plant. Detailed tailings design is not appropriate for this level of study. However, a location was identified as having suitable capacity and an estimate of costs is provided in Table 21-7 for reference. These areas would need to be prepared and lined prior to the placement of spent MPP, and these costs have been estimated based on a conceptual design. There is also an allowance for a treatment package to treat excess water due to precipitation and settling of the tails for environmental release which as the tailings would have gone through detoxification treatment could be limited to a series of settling ponds. The entire facility needs to be reviewed as more detailed information comes to hand such as the anticipated runoff water quality. Significant changes in CAPEX are possible based on future results of baseline studies, environmental assessment and legislation.

The tailings slurry would be transported via a DN450 pipe and 12 intermediate pumps. Detailed design may cause pumps and the slurry pipe size estimated to be different by selecting an alternate route.

Table 21-7: Capital Cost Estimate – Tailings Management

Cost Centre	TOTAL [USD]	Fraction
DIRECT TOTAL	**36 900 000**	**100 %**
Earthwork	34 900 000	94.58 %
Mechanical	2 000 000	5.420 %

21.2.7 CAPITAL COST ESTIMATE – INFRASTRUCTURE

Table 21-8 shows the breakdown of infrastructure costs, with the site power transmission from the grid to site and road upgrades comprising the bulk of the costs in this area.

The accommodation costs are based on housing approximately 25 staff. There is significant risk with this estimate that if suitable accommodation was not available for the remaining 400 or so staff, additional housing would be required to be built.

The budget cost of a 230kV high voltage line from the Mirador project to the proposed plant location would be in the order of 135 000 USD/km and the costs for substations at each end of the high voltage line and around the site are estimated to be approximately USD 15.77 M. The high voltage line required is approximately 7 km long. There is significant risk in this cost as there is uncertainty as to whether the connecting power line at the Mirador project will have been constructed and have sufficient capacity for the

Project. Inability to use this option to supply the project could significantly increase the cost magnitude of both the CAPEX and OPEX.

Access road costs include the upgrade of the main access road which links the site to the E45 Highway. Heavy vehicle traffic, which will occur during construction and operation, will be serviced by the main access road. The access road was conceptually designed as a 10 m wide road with construction of five single span reinforced concrete bridges. Road design and costing is dependent on detailed site surveying and geotechnical analysis which has not been undertaken at the time of this PEA and subsequently the cost of the road is considered to have a low level of accuracy.

The costs of the buildings which form part of the administration and maintenance areas have been calculated on a square metre rate turn-key basis at local rates for buildings of similar uses.

As an initial investment, only a small allowance for environmental capital expenditure has been allowed for in the way of site vehicles for environmental staff. It is likely further purchases will be defined in the environmental study, though there is a possibility that environmental monitoring may be performed by a consultant and therefore this expenditure will be reclassified as OPEX.

Table 21-8: Capital Cost Estimate – Infrastructure [USD]

Cost Centre	TOTAL	Water	Roads	Accommodation	Electrical Power
DIRECT TOTAL	**62 047 329**	**1 377 299**	**23 677 007**	**601 874**	**36 391 149**
Earthwork	16 996 915	120 000	16 848 840	28 075	
Civil	6 828 167		6 828 167		
Structural	506 250			506 250	
Mechanical	1 257 299	1 257 299			
Electrical	36 458 698			67 549	36 391 149

21.2.8 PHASED COSTS DISCUSSION

Initial capital costs have been spread across the first and second years of the project, assuming a two year construction period.

An allowance for rehabilitation and closure of operations is included at the end of the project including capping of tailings and monitoring. This has been inputted as a single year figure, however, it should be noted that monitoring would likely be an on-going cost post mine closure rather than a lump sum in the final year of operation. The total cost of decommissioning the project needs to be re-evaluated once an environmental study and closure management plan have been developed.

Annual sustaining capital costs include an allowance for downstream raises of the tailings embankment. Future detailed design may result in optimisation of these costs and a more accurate allocation of the sustaining capital across the years. For the purposes of the financial model these costs have been spread evenly over the three nominated phase development of the tailings facility.

The service life of the mining fleet has been estimated and sustaining capital allows for both major overhaul and purchase of new mobile equipment as required. A total mining sustaining capital cost for the Project has been estimated at USD 117.5 million.

Future studies that include greater project definition will allow for more accurate phasing of sustaining capital costs. Sustaining capital has been considered at a high level only to better define the financial model, the results of which are described in Section 22.

21.3 OPERATING COST ESTIMATE

The average operating costs estimate was prepared based on the LOM costs. This enabled the estimation of unit rates for the operating costs per tonne of MPP produced. This approach was used as the majority of operating costs were variable rather than fixed. Fixed costs included general and administration costs and labour costs with the remainder of costs being variable. The variable costs for the mining operation are driven by the total amount of material mined, including waste. The variable costs for the processing plant are a function of either the MPP processed or the gold and silver extracted.

The total operating costs were estimated as detailed in Table 21-9.

Table 21-9: Project OPEX Estimate

Area	Description	Life of Mine (LOM) Cost [USD]	Percentage of LOM OPEX	LOM average cost per MPP [USD/t]
TOTAL		**1 838 007 884**	**100%**	**18.78**
	Contingency	5 581 305.42	0%	0.23
000	**GENERAL**	**49 730 518**	**3%**	**0.50**
	Utilities	168	0%	0.00
	General and Administration	49 730 350	3%	0.50

Area	Description	Life of Mine (LOM) Cost [USD]	Percentage of LOM OPEX	LOM average cost per MPP [USD/t]
100	**MINING**	**634 140 712**	**35%**	**6.42**
200	**PROCESS**	**1 034 726 514**	**56%**	**10.47**
	Consumables	276 656 437	15%	2.80
	Maintenance and Operating Spares	108 352 521	6%	1.10
	Reagents	367 747 737	20%	3.72
	Utilities	241 775 820	13%	2.45
	Labour	40 194 000	2%	0.41
500	**INFRASTRUCTURE**	**113 828 834**	**6%**	**1.15**
	Utilities	2 408 834	0%	0.02
	Maintenance and Operating Spares	111 420 000	6%	1.13

21.3.1 GENERAL AND ADMINISTRATION

General and administration costs have been estimated from first principles based on a total workforce of 447. This figure is considered low due to the limited allowance for labour housing, catering, janitorial, etc. costs and future studies should confirm assumptions as to labour housing facilities to improve the reliability of this estimate.

21.3.2 MINING

An owner operated fleet was determined to estimate the operating cost of open pit extraction of both waste rock and MPP.

21.3.3 PROCESSING, INFRASTRUCTURE AND WASTE MANAGEMENT

The processing, infrastructure and waste management costs are captured under the following sub-areas:

- Operating spares and maintenance;
- Reagents;
- Labour;
- Consumables; and
- Electrical power.

A breakdown of these costs is presented in this section.

21.3.3.1 OPERATING SPARES AND MAINTENANCE

The cost of operating spares and maintenance was estimated as 7 % of the direct capital costs of equipment due to the anticipated abrasive nature of the material and road and building maintenance was estimated as 3 % of the respective direct capital costs.

21.3.3.2 REAGENTS

Cyanide accounts for 83.32 % of the reagents cost and would be used in the operation in two roles; firstly, as a lixiviant to leach the gold from the host MPP, and secondly, as an eluent in the elution process to strip the loaded carbon. Other reagents are used as part of the recovery, elution and detoxification processes such as sodium metabisulphite, which accounts for 10.80 % of the estimated reagent costs and is used in the detoxification process to assist in cyanide destruction.

21.3.3.3 LABOUR

The costs presented in Table 21-10 were the total costs to the employer for each employee used for the estimate and were based on rates provided by Condormining. A rotation of three gangs was assumed for these calculations, as were two shifts of 12 hours per day. A total of 190 employees were estimated to be required to operate the processing plant annually.

Table 21-10: Labour (Processing Facility)

Position	Quantity	Annual Salary [USD/person]
Mine Manager	1	250 000
Forklift Operator	1	18 000
Warehouse Supervisor	1	44 000
Security supervisor	1	44 000
Security shift control	4	18 000
Security guard	24	9 000
Plant Superintendent	1	78 000
Process Engineer	2	44 000
Mechanical Engineer	2	44 000
Electrical Engineer	2	44 000
Mill foreman	2	44 000
Metallurgist	2	44 000
Metallurgical Technician	3	18 000
Shift Boss	6	44 000
Plant Op	57	18 000

Position	Quantity	Annual Salary [USD/person]
Laboratory Operator	4	18 000
Wet Assayer	4	18 000
Sample prep	16	9 000
Refinery Chargehand	3	18 000
Chemist	2	44 000
Maintenance General Foreman	1	78 000
Mechanical Foreman	2	44 000
Elec & Instrumentation Foreman	2	44 000
Maintenance Planner	3	44 000
Mechanic	24	18 000
Electrician	8	18 000
Auto mechanic	6	18 000
Instrument Technician	6	9 000

21.3.3.4 CONSUMABLES

The estimated consumption requirements for fuel and lubricants were calculated based on the expected mobile fleet and a diesel price of 1.14 USD/L.

Grinding media for the ball mill was also estimated based on the anticipated abrasiveness of the MPP. Future testwork of the MPP as to its abrasiveness could potentially result in an increase or decrease of the estimated grinding media required. As this comprises 15.85 % of the operating cost a change in the assumed consumption rates or unit rates could have a reasonable impact upon the operating cost.

21.3.3.5 ELECTRICAL POWER

Energy costs have been calculated using tariff rates published by CONELEC (Conelec, 2013), the office responsible for planning, regulating and control of the electricity sector. As these tariffs are published by the regulatory body they can be taken as indicative (the calculations are based on the high voltage industrial rate for the southern area of Ecuador). It is anticipated that during future development of the Project alternate tariffs may be negotiated with the electricity provider. The tariffs used to determine electrical supply costs were:

- Fixed monthly fee which totalled as USD 17 per year;

- Monthly cost that covered the use of the generation and transmission infrastructure. This cost was based on the estimated maximum demand at the site and was estimated using a rate of 148.56 USD/kW/a; and
- Tariff for the estimated active energy consumed by the site and losses, which was taken as 0.0505 USD/kWh.

Additionally, CONELEC charges reactive energy tariffs and so, power factor correction equipment would be installed to increase the site power factor and avoid these additional tariffs.

SECTION 22 ECONOMIC ANALYSIS

The economic analysis presented in this section is a preliminary economic assessment that includes Indicated Mineral Resources as well as Inferred Mineral Resources, which are considered too speculative geologically to have the economic considerations applied to them to enable them to be categorized as Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. There is no certainty that all or any part of the Mineral Resources for the Project would be converted into Mineral Reserves. Mineral Reserves can only be estimated as a result of an economic evaluation as part of a preliminary feasibility study or a feasibility study of a mineral project. Accordingly, at the present level of development, there are no Mineral Reserves at the Project and caution should be used when interpreting the results presented.

22.1 ANALYSES CRITERIA

The financial model prepared was a constant dollar type model which assumed the purchasing power did not change with time. This means the CAPEX, OPEX and revenue were considered constant through time in a like-for-like manner. The model was based on full equity financing.

The criteria inputted into the financial model included those parameters listed in Table 22-1.

Table 22-1: Financial Analyses Criteria

Criteria	Value	Reference
Net-present value	8 %	Condormining
Gold	1 448 USD/oz.	Three year trailing average
Silver	24.55 USD/oz.	Three year trailing average
Copper	3.34 USD/lb	Three year trailing average
Salvage at the end of Life of Mine	No	Assumed

22.1.1 GOVERNMENT LEVIES AND TAXES

The Ecuadorian Government has various taxes, duties and levies that may or may not be applicable to future mining operations depending on the mining exploitation contract (refer to Section 22.1.1.1) established at the time of exploitation and laws in force at that time. A list of some of the taxes, duties and fees that could be applicable is listed in Table 22-2.

Table 22-2: Ecuadorian Taxation and Fees

Description	Value	Reference
Government Royalty	5 % of Revenue (pre-tax)	Condormining / Section 4
Corporation Tax Rate	22 %	(18)
Depreciation Method	Straight Line	(25)
Depreciation Rate – Buildings	5 %	(25)
Depreciation Rate – Plant and Machinery	10 %	(25)
Depreciation Rate – Mobile Equipment	20 %	(25)
VAT	12 %	(18)
Profit tax rate	15 %	(18)
Windfall tax rate	70 %	(18)
Mining Conservation Patents	USD 15.90 per hectare	Section 4
Currency Exit Tax	5 % of sum sent out of the country	(18)
Municipal Fees	Maximum USD 5 000	(18) / Section 4
Municipal Tax	0.15 % Total Assets	(18) / Section 4
Rural Property Tax	0.1 % Income (pre-tax)	(18)

22.1.1.1 MINING EXPLOITATION CONTRACT

A mining exploitation contract establishes the terms and conditions for the construction, mounting, extraction, beneficiation, transportation and commercialisation of the minerals obtained within the area of a mining concession (Ecuador Government, 2008). The mining exploitation contract also provides applicable taxes, duties, royalties and rights of the concessionaire and state. The conditions of these mining exploitation contracts, including the duration of their applicability, may vary depending on many factors, such as:

- Proposed investment value;
- Proposed LOM;
- Municipality regulations;
- Laws in force at the time of agreement formation; and
- Mineral to be mined (currently no operating mines in Ecuador).

The base case selected only provides for corporation tax of 22 %, government royalty of 5 % and straight line depreciation of vehicles, equipment and buildings at the rates stated in Table 22-2.

22.1.2 EXCLUSIONS

No allowance in the model was made for:

- Reconciliation;
- Cost escalation;
- Tax regimes that are typically subject to future mining conventions, such as VAT, energy tax, windfall tax, customs duties, etc.;
- Currency fluctuations; and
- Required permits, licences or legal and administrative costs associated with government mining and environmental regulations. This includes reporting requirements during operation and related administrative costs.

22.2 FINANCIAL MODEL

The results of the base case model uses Material Planned for Processing based upon both Indicated Mineral Resources as well as Inferred Mineral Resources and therefore cannot be considered to demonstrate economic viability, indicate that the Project, would achieve an IRR of 9.5 %, realise an NPV of USD 47.4 M and pay back in 6.5 years as shown in Figure 22-1 and Figure 22-2.

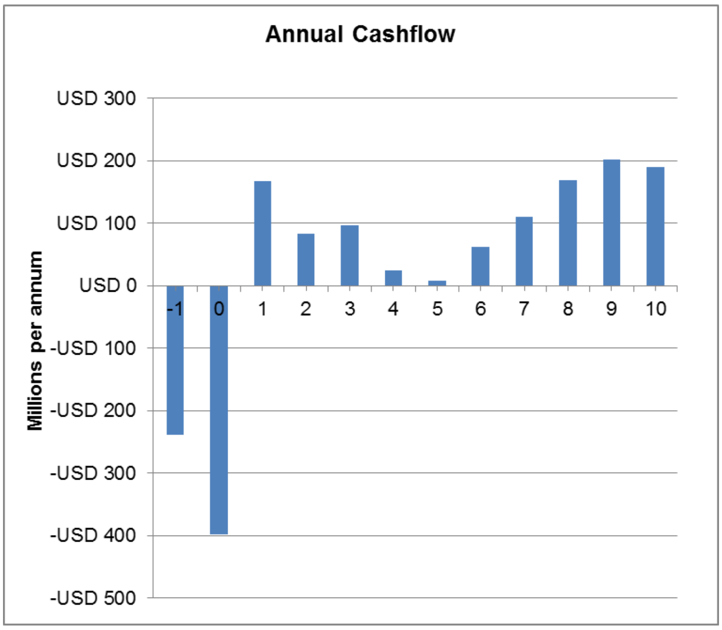

Figure 22-1: Pre-Tax Annual Cash Flow[2]

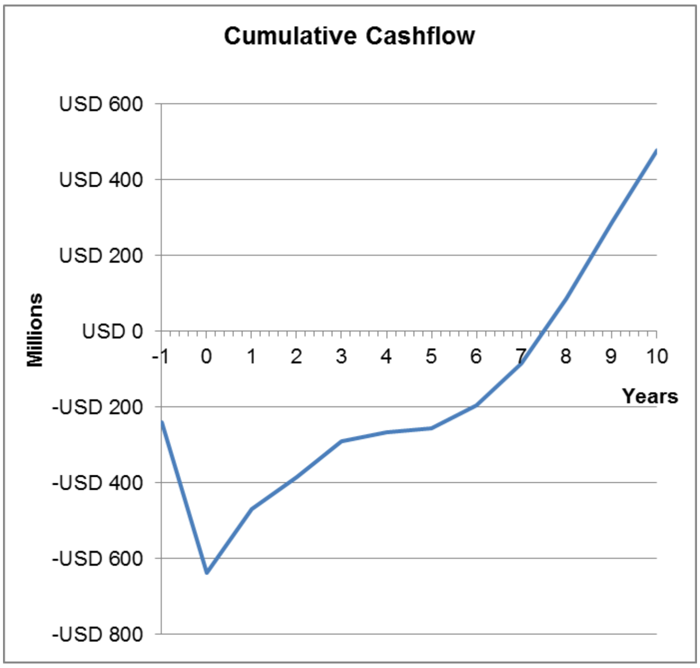

Figure 22-2: Cumulative Annual Cash Flow[3]

[2] Construction is completed in Years -1 and 0 with Operations commencing in Year 1.

22.3 SENSITIVITY AND SCENARIO ANALYSES

A major uncertainty of the Project is due to the inclusion of Inferred Mineral Resources. Additionally there are significant risks associated with the Project such as power supply, site weather conditions and mineral recovery. Subsequently a sensitivity analyses has been completed on key variables as listed in Table 22-3.

Table 22-3: Sensitivity Variables

Variable	Base	High	Low
CAPEX [% change from base]	0	-10	+20
OPEX [% change from base]	0	-10	+20
Gold Grade [% change from base]	0	+20	-10
Process Gold Recovery [%]	86.7	95.4	69.4
Gold Price [% change from base]	0	-30	+30
Discount Rate [%]	8	5	15

22.3.1 DISCUSSION OF SELECTED SENSITIVITIES

22.3.1.1 CAPEX AND OPEX

Capital and operating cost sensitivities were examined on a total basis whereby the base case cost was multiplied by a factor to increase or decrease the cost according to the scenario being examined. The high and low cases for the CAPEX and OPEX were selected as being representative of the limits of the expected costs that could be realised for the execution of this project, based on the engineering and cost estimating confidence.

22.3.1.2 MPP GRADE

Given this financial analysis includes Inferred Mineral Resources, there is significant uncertainty as to the likely grade of material that could be exploited.

22.3.1.3 RECOVERY

Limited testwork to the proposed extraction method has been completed to date, comprising bottle roll, diagnostic leach and ball work index testwork. Duplicates have not been completed on all tests. This presents a project risk, and thus the extraction and recovery efficiency of the Project is subject to uncertainty.

[3] Construction is completed in Years -1 and 0 with Operations commencing in Year 1.

22.3.1.4 GOLD PRICE

The three year trailing average price for gold is 1 448 USD/oz. This was used as the basis for the Base Case financial model. Additional discussion of the gold price is provided in Section 19.

22.3.1.5 DISCOUNT RATE

The discount rate is used during project financial analysis to calculate the present value of future cash flows, thereby facilitating investment decisions. The discount rate applied typically varies with project risk and also reflects the cost of capital. As the rate is a reflection of risk, it may be qualitatively determined, and thus is appropriate for sensitivity. A high and low sensitivity of 5 % and 15 % were selected as these rates are commonly used for a more advanced level study, and this level of study, respectively.

22.3.2 SCENARIOS

The results of the sensitivity analyses are presented in Table 22-4 and Figure 22-3.

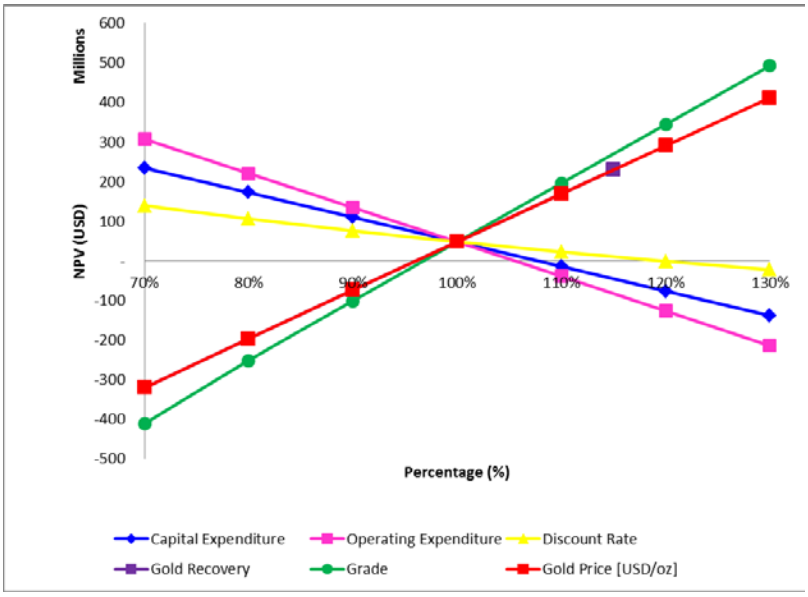

Figure 22-3: Graphical Results of Sensitivity Analysis

Table 22-4: Sensitivity Analysis Results

	Value	%	IRR (%)	NPV (USD)	Payback Period (years of operation)	Gradient
BASE	**Base Case (100% CAPEX; 100% OPEX; 0.745 g/t Au grade; 86.7 % recovery; 1 448 USD/oz.; DR 8 %)**	**100**	**9.5%**	**47 416 244**	**6.5**	**n/a**
CAPEX	-10 %	90	11.8	109 540 972	6.1	13.1
	+20 %	120	5.8	-76 833 211	7.3	
OPEX	-10 %	90	12.3	133 921 274	5.8	18.3
	+20 %	120	3.8	-127 002 654	7.9	
Gold Grade	-10 %	90	4.6	-101 390 957	7.7	31.8
	+20 %	120	18.6	343 853 992	3.9	
Gold Recovery	69.4 %	80	1.2	-196 949 046	8.6	25.6
	95.4 %	110	13.4	168 683 726	5.6	
Gold USD/oz.	-30 (1 014 USD/oz.)	70	-3.3	-320 901 705	never	25.7
	+30 (1 882 USD/oz.)	130	20.5	410 985 918	3.0	
Discount Rate	5.6 %	70	9.5	139 026 504	6.5	5.7
	15.2 %	190	9.5	-119 538 302	6.5	

22.3.3 RESULTS DISCUSSION

Gold recovery, grade and price are the variables to which the project is most sensitive of those selected to be analysed. As shown in Table 22-4 and Figure 22-3, all have a similar gradient of sensitivity as each other.

As shown in Table 22-4, the gradient of the CAPEX is less than the OPEX which indicates that the Project is more sensitive to OPEX than CAPEX. A change in project operations could impact this sensitivity for example the introduction of contract mining could cause the Project to be far more sensitive to OPEX and the incorporation of more standby duty arrangements and insurance spares could on the converse see the CAPEX increase to see it similarly sensitive to the Project.

Although discount rate is not the Project's most sensitive variable, as the confidence in its adequacy is difficult to determine and the magnitude of difference of the resultant NPV due to a given discount rate may be up to USD 60 M, its impact should not be disregarded.

SECTION 23 ADJACENT PROPERTIES

Concessions and their holders surrounding the Project concessions are presented in Figure 23-1. Several artisanal mining operations are located in the vicinity of the Deposit, but no production details have been reported. A large alluvial mining operation called “Tsamaraint” is located near the mouth of the Conguime River.

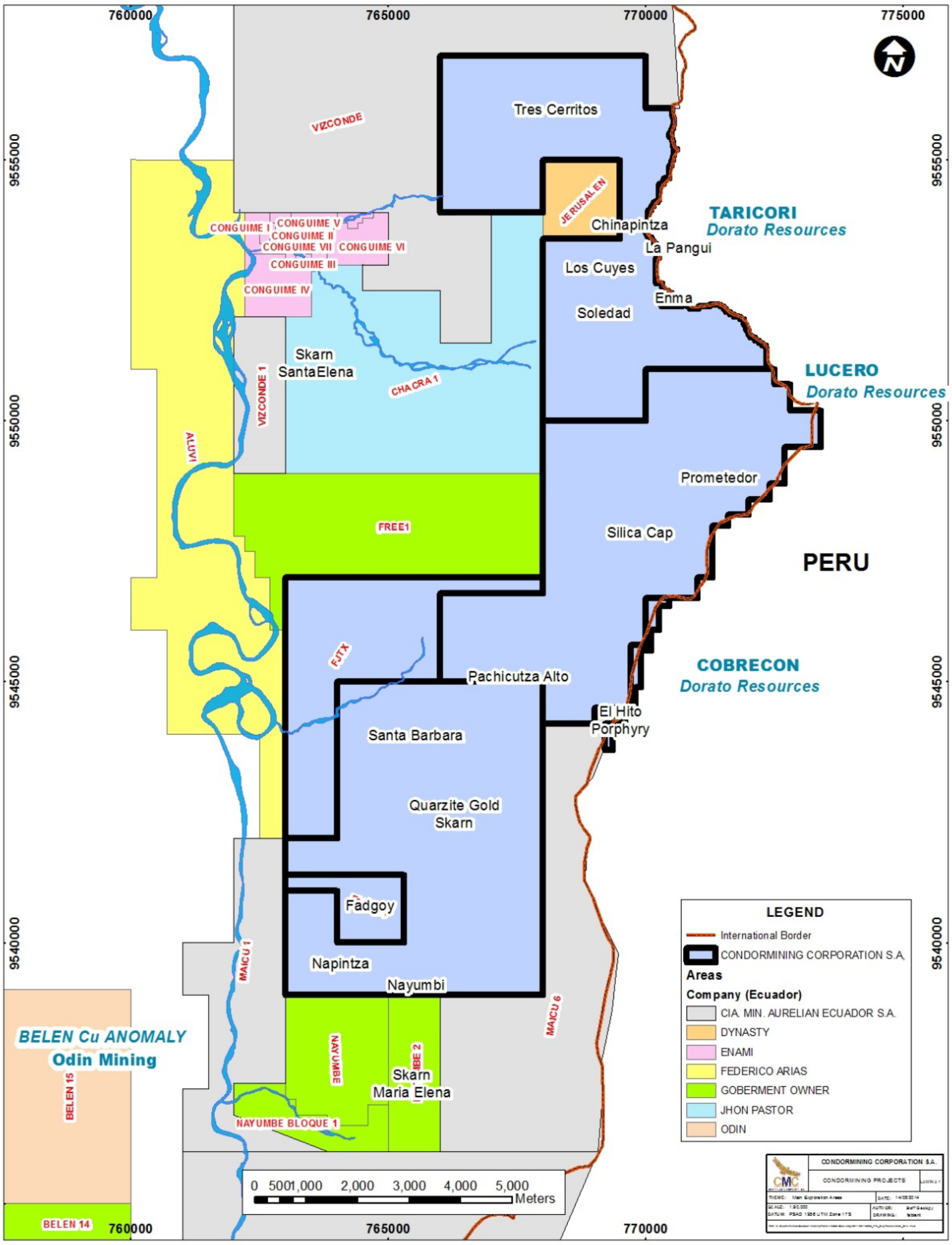

Figure 23-1: Regional Concession Owners

Dynasty Metals and Mining Inc. (TSX: DMM) ("Dynasty") owns 100 % of the Jerusalem concession, located immediately adjacent to the Condormining owned Viche Conguime I concession. The concession covers 2.25 km^2 at altitudes ranging from 1 400 m to 1 900 m, as shown in Figure 23-1, above. A technical report in 2014 identified the Mineral Resources for the Jerusalem concession, as listed below in Table 23-1 (AMA, 2014b).

Table 23-1 Dynasty Mineral Resource Statement (AMA, 2014b)

Classification	Estimated Resource (t)	Gold Grade (g/t)	Silver Grade (g/t)	Contained Gold (oz.)	Contained Silver (oz.)
Measured	379 200	14.2	76	173 234	925 980
Indicated	576 300	13.5	81	249 255	1 495 401
Inferred	1 775 000	15.0	98	856 510	5 569 949

Note:
1. Lower cut-off grade of 2.0 g/t Au.

The above Mineral Resources estimates were estimated by the Qualified Person, Allen Maynard, BAppSc (Geol), MAIG, MAusIMM, in a technical report prepared for Dynasty (AMA, 2014b), which were current as of the date of that report on October 24, 2014. However, the Qualified Person has not recently visited the Jerusalem concession site to confirm there are no changes or additional work on that property that would render the estimated mineral resources as non-current. Accordingly, the Qualified Person has not done sufficient work to classify the above resource estimates as current mineral resources as of the date of this PEA, and EGX is not treating those estimates as current mineral resource estimates, which are not part of the Santa Barbara Project, in any event.

In addition, the author cautions that the information in this report about mineralization that is not on the Santa Barbara Project is not necessarily indicative of the characteristics of any mineralization that is known or may be found on the Santa Barbara Project.

No other publicly available information could be found relating to other relevant adjacent properties.

SECTION 24 OTHER RELEVANT DATA AND INFORMATION

There is no other known relevant data available about the Santa Barbara Project that has not been included or discussed elsewhere in this report.

SECTION 25 INTERPRETATION AND CONCLUSIONS

25.1 MINERAL RESOURCES

The Mineral Resource estimates are based on geological investigations and drilling over a period spanning almost 20 years. Over this period, international standards, including the CIM guidelines, other standards relevant to NI 43-101, and general exploration practices have changed considerably. Most of the information compiled for this PEA, and the drilling data used for Mineral Resource estimation, is a result of exploration programs conducted since 2005. Upon investigation by the Qualified Person, all of the data was found to be reliable and of a high standard and compliant with the current CIM requirements. The Indicated and Inferred Mineral Resources are based on sample spacings that are too far apart to ensure that the actual tonnage, densities, shape, physical characteristics, grade and mineral content can be estimated with a high level of confidence. Further drilling and sampling on a regular and close spacing is required before the level of confidence in the Mineral Resource estimates could be considered high enough to be further categorised as Indicated or Measured Mineral Resources.

The Project includes a number of highly prospective exploration targets including mineralised porphyries, breccia pipes and epithermal veins. It is the Qualified Person's opinion that the recommended extensive exploration program outlined in Section 26 is entirely warranted.

The reliability or confidence in the exploration information and Mineral Resource estimates contained in this report may be affected by variances in sub-surface mineralisation, ground conditions and sampling QA/QC procedures and protocols. Occasionally, projects yield higher than actual results regarding the grades of precious metal minerals when exploring and assessing sub-surface mineralisation such as the mineralisation in the Project. Likewise, there can be no assurance that the exploration results will continue to exhibit good results due to natural variation of ground conditions where sometimes there is much less mineralisation than expected and other times there is more.

While the post 2007 QA/QC sampling protocols meet or exceed industry standards, statistical analysis of the accuracy of 2004 to 2007 sampling indicated a broader variance warranting additional sampling and review. On the other hand, the analysis of sampling revealed a less than 5 % failure rate for the entire database, indicating a fairly robust database. However, there is room to improve the variances. The Project's potential economic viability would be overstated if the variances in sub-surface mineralisation, ground conditions and/or sampling led to higher than representative resource estimates. However, it is the Qualified Person's firm opinion that the variances are in the acceptable range for the determination of potential economic viability, which warrants our recommendations of further exploration as described Section 26.

25.1.1 MATERIAL PLANNED FOR PROCESSING

The average run of mine grade of the Material Planned for Processing that forms the basis for this assessment is 0.72 g/t gold, 0.11 % copper and 0.96 g/t silver which may support the feasibility of alternate processing options. No Mineral Reserves could be determined. It should be noted that the Material Planned for Processing was based upon Indicated Mineral Resources as well as Inferred Mineral Resources, which cannot be used under economic analysis to determine a Mineral Reserve. A risk exists that due to the geological uncertainty associated with Inferred Mineral Resources it cannot be assumed that any part of an Inferred Mineral Resource will ever be upgraded to a higher Mineral Resource category. At the present, none of the EGX properties have Proven or Probable Mineral Reserves and the proposed programs are an exploratory search for Proven or Probable Mineral Reserves. Substantial expenditure will be required to establish Mineral Reserves through further drilling and supporting testwork. No assurance can be given that minerals will be discovered in sufficient quantities or having sufficient grade to justify commercial operations or that funds required for development can be obtained on a timely basis. The economics of developing gold and other mineral properties is affected by many factors including the cost of operations, variations of the grade of material mined, fluctuations in the price of minerals produced, costs of processing equipment and such other factors as government regulations, including regulations relating to environmental protection. In addition, the grade of mineralisation ultimately mined may differ from that estimated by drilling results and such differences could be material.

25.2 METALLURGICAL CONSIDERATIONS

The gold occurrence in the Santa Barbara Deposit is “free milling” in that it is readily recoverable using sodium cyanide as a lixiviant. Improved project economics have been demonstrated on the flotation process with flotation rougher concentrate to obtain copper credits and gold and silver revenue streams. The revised flowsheet for flotation with cyanidation on the tails stream has shown a positive NPV result and sensitivity analysis. Therefore flotation followed by treatment of the rougher concentrate is the most promising option for consideration in future project evaluations.

25.3 COST ESTIMATES

The capital cost estimate has been prepared to the level of a preliminary economic assessment study with an accuracy of -30 % to +50 %.

25.3.1 CAPEX

The estimated initial investment was determined to be USD 598.9 million including a contingency of USD 103.7 million with a total capital cost estimate for the LOM of USD 826.0 million including a contingency of USD 138.4 million.

25.3.2 OPEX

The average operating cost calculated based on the LOM costs were USD 18.78 per tonne of Material Planned for Processing.

The most significant operating costs were identified as:

- Mining machinery operation and maintenance (35 %);
- Reagents (20 %, with 15 % of the total OPEX from cyanide alone);
- Consumables (15 % with 13 % of the total OPEX from grinding media alone); and
- Electrical power (13 %).

When combined, these four items accounted for approximately 83 % of the total operating cost. Thus risks, such as the assumed electrical supply source and method, and the limited geotechnical information (which significantly influences the mining costs), have significant potential to impact upon the operational cost estimate.

25.4 FINANCIAL ANALYSIS

The flowsheet for a treatment process incorporating flotation followed by treatment of the flotation rougher concentrate to obtain copper and cyanidation on the tails to recover both gold and silver, have demonstrated favourable economic results based on the financial sensitivity analysis undertaken.

Since Inferred Mineral Resources cannot directly be converted into Mineral Reserves and due to their uncertainty of existence it cannot be assumed that any part of an Inferred Mineral Resource will ever be upgraded to a higher Mineral Resource category.

Further Project definition and technical development will enable a more refined and accurate estimate of the Project's economics for flotation followed by flotation of rougher concentrate and cyanidation of the tails stream. Particular attention must be paid to determination of the grade, process recovery rate given the sensitivity of the Project's economics to these variables. The results of the base case financial analysis were an IRR of 9.5 %, an NPV of USD 47.4 million and a payback period of 6.5 years.

SECTION 26 RECOMMENDATIONS

26.1 FUTURE PROJECT DEVELOPMENT

Based on the results of the PEA, the Project is recommended for further development with further drilling and metallurgical testwork being completed so that a higher confidence in mineral content and recovery results can be achieved in order to complete a PFS.

Further opportunities in the PFS economic analysis to make reduce power costs may be possible by investigating in more detail the new hydroelectric projects under development in the surrounding area of the Santa Barbara Project. This would include strategies and opportunities for securing a more economic cost of power to compliment the Project economics, by investing or leveraging of power infrastructure and power supply that is available from such new hydroelectric projects.

26.2 DRILLING

Further drilling is required to produce a more confident estimate of the Mineral Resource. It should be noted to upgrade Mineral Resource categories so as to assess project feasibility, additional testwork such as geotechnical and metallurgical testwork are also required. Further drilling is also required to provide further samples for metallurgical testwork if it is to be undertaken. This will form part of a PFS.

As a starting point, it is recommended that detailed surface geology, alteration and structural mapping and extensive geochemical sampling continue to be conducted to improve understanding of the controls on the mineralisation at this Project. That initial work should then be followed up with the drilling to further improve the categories and confidence of the estimated resources.

As part of the PFS, it is recommended that a surface metallurgical and drill exploration program be initiated and carried out over approximately the next 12 months in a two-phase program beginning with the drilling and resource upgrade work. The recommended work program should include surface geologic, alteration and structural mapping, extensive geochemical sampling, and detailed re-logging of earlier drill holes taking into consideration the latest geological interpretations of the structures, rock-types and controls on the mineralisation. This warrants an estimated 3 000 m of new drilling to both improve the categories of estimated resource areas and to attempt to identify new resource occurrences both in outcrop and in isolated drill intercepts. Drilling has the objective of extending known mineralisation as well as increasing confidence in the known Mineral Resource.

The primary focus will be to explore on the surface and drill high grade gold bearing structural zones, structural intersections, mineralised breccias and hydrothermal breccias. However, in addition, significant mid to lower grade gold mineralisation exists both as disseminated and fracture controlled sulphide mineralisation in tuffs, rhyolites and quartz feldspar porphyries, which are also to be targeted as significant zones to be drilled for expanding current known gold resources. Numerous other drill targets also exist and are proposed for drilling that are typical for gold or gold/copper related porphyry systems. It should be noted that significant amounts of silver, zinc, copper, lead and manganese occur in many of the existing gold resources and drill targets and these metals have the potential to add significant additional resource value in the future drilling. It is therefore recommended that all future resource models include these minerals in any evaluation.

26.3 TESTWORK

Test work has demonstrated copper recovery can be achieved by rougher flotation. Test work is recommended in a future PFS phase to confirm the recovery of copper that is achievable. It is recommended that if a PFS is pursued, the following testwork be completed:

- Test work will concentrate on further maximising recovery of copper, gold and silver into a cleaner flotation concentrate by investigating finer primary grinding and optimised reagent schemes.
- Rougher concentrate will be reground and cleaned using multiple stages of flotation to produce a saleable concentrate (locked cycle tests). Cyanidation of cleaner tailings for gold recovery will be investigated.
- Hydrometallurgical treatment of rougher concentrate may be an option to produce copper metal on site. This will involve some type of oxidation technology such as pressure oxidation, bio-oxidation, or others followed by solid/liquid separation, solvent extraction, and electrowinning. Tailings from this process could be neutralized and cyanide leached for gold.
- If a review of the previous potential for heap leaching is undertaken (low cost processing but does not provide any copper recovery), recommended bottle roll tests at coarse size would be conducted to give an indication of what might be expected from heap leaching at various crush sizes.
- Bottle roll tests on ground material would be conducted for comparison purposes with the previous test results.

The PFS metallurgical test work should be carried out sequentially. The first testing would include bench flotation tests followed by flotation cleaner tests at various concentrate regrinding particle size. Then, additional testing would incorporate the rougher concentrate oxidation/leaching tests.

The most promising option for future PFS investigation and further scope for test work to confirm copper, gold and silver recovery rates, is flotation followed by cleaner flotation of the rougher concentrate for copper recovery followed by cyanidation of the tails stream to recover gold and silver.

26.4 COST ESTIMATE

GBM recommends that the Project be advanced to the next level of study, a pre-feasibility study (PFS), in two phases beginning with the drilling and upgrading the mineral resource estimate followed by further metallurgical test work and pre-feasibility economic analysis contingent on a satisfactory updated mineral resource estimate. As noted above, the estimated Mineral Resources will have to be drilled more extensively in order to convert Inferred Mineral Resources to Indicated or Measured Resources. After drilling, sampling and assaying, a new mineral resource estimate will be determined. Following determination of a satisfactory upgraded Mineral Resource estimate, additional metallurgical test work should be carried out together with pre-feasibility economic analysis to determine Mineral Reserves and economic viability for the Project.

The estimated cost for the 12 month recommended work program amounts to USD 1 860 500, comprised of USD 1 160 500 for the first phase of drilling to upgrade the resource estimate and USD 700 000 for additional metallurgical test work and pre-feasibility economic analysis, as listed below in Table 26-1.

Table 26-1 Cost Estimate

Item and Description	USD
Phase 1: Drilling and Resource Upgrading	
In-country/camp staff and consultant salaries	300 000
Field Program – Prospecting & Drilling General Expenses	75 000
Drill Program – 3 000 m @ $136 per meter	408 000
Assay analysis – 3 000 m @ $20 per meter	60 000
Field supplies 3 000 m @ $10 per meter	30 000
Vehicle hire – 1 vehicle @ $2,000 per vehicle per month for 12 months	24 000
Fuel and Oils – 19 kl @ $1.00 per litre	19 000
Contractor – Mob / Demob	72 500
Camp operational expenses	45 000
Camp capital costs – building, core shed and other equipment	12 000
Environmental and permitting for exploration program	26 000
Community relations / communications with local residents and government	45 000
Communications and IT	20 000
Expenses and supplies	24 000
Subtotal for Phase 1:	***$1 160 500***
Phase 2: Metallurgical Testwork and PFS Economic Analysis	
Metallurgical Test Work	250 000
Pre-feasibility study economic analysis	450 000
Subtotal for Phase 2:	***$700 000***
Total cost for recommendations:	**$1 860 500**

SECTION 27 REFERENCES

Ambien Consul Co. Ltd., Jan. 2008 Auditoría Ambiental 2007 “Proyecto Cóndor Gold”. Operadora Goldmarca Mining Ecuador S.A. Enero-2008.

Ambien Consul Co. Ltd., 2006, Evaluacion de Impacto Ambiental.

Ambien Consul Co. Ltd., 2011, Audit Report.

Boughton Law Corporation, 2015, Email re Ecuador Gold – Title Opinions (Canadian Ownership Structure), dated May 19, 2015.

Briggs, D F. 2006, LIST OF ORE DEPOSIT TYPES, http://www.geomineinfo.com/Complimentary%20Downloads/Types%20of%20Ore%20Deposits.pdf

Burns, P. 2005: A Summary of Progress on the Condor Gold Project, Pachicutza Mining Camp, Zamora-Chinchipe Province in Southeastern Ecuador. (Internal Company report for Goldmarca Ltd.)

CIMM, 1995. Final Report, Project 934135, Metallurgical Laboratory Testwork on the San Jose Gold Ores.

Conelec., 2013, Pliego Tarifario Para Empresas Electricas. [Online] http://wwwconelec.gob.ec/images/documentos/doc_10332_PLIEGO_TARIFARIO_2013.pdf.

Cueva, A, and Nagua, S. Annual Report Water Sampling.Ecometals Internal Report, August. 2008.

Drobe, John, Lindsay, D., Stein, H., Gabites, J., 2013, Geology, Mineralization, and Geochronological Constraints of the Mirador Cu-Au Porphyry District, Southeast Ecuador: *Economic Geology,* v. 108, pp. 11–35.

Easdon, M Qualified Person October 28, 2004, National Instrument 43-101F1 Technical Report on the ¨Condor Project Southeastern Ecuador UTM Coordinates 9,552,000N and 770,000E for Goldmarca Limited¨.

Ecuador Government, 2008, Mining Law.

Gil-Rodríguez, Javier, 2010, Igneous Petrology of the Colosa Gold-Rich Porphyry System (Tolima, Colombia): Univ Arizona Master’s Thesis, 51p, University of Arizona.

Gonzalez, R. 2000. Final Report on the Santa Barbara-El Hito Projects, Ecuador. Joint Venture between TVX Gold and Valerie Resources.

Global Legal Group, International Comparative Legal Guides, 2014, Ecuador Chapter – Mining Law 2014. [Online], 2014. [Cited: 12 May 2014.] http//www.iclg.co.uk/practice-areas/mining-law/mining-law-2014/ecuador.

Hedenquist, J. W., 2007 Mineralization aspects of the Pachicutza epithermal and Santa Barbara porphyry prospects, Condor project, Zamora Chinchipe Province, southeastern Ecuador: Internal report for Goldmarca Ltd., 23p.

Henderson, R.D., 2009, Fruta del Norte Project, Ecuador: National Instrument 43-101 Technical Report, prepared for Kinross Gold Corporation,

HRG Byers, Robin Slaughter, Jorge Diaz, Alfonso Postigo, Carlos Rodriguez, Jose Gonzales, Edgar Melendez, Keith Dodd, Eduardo Bringas: Feasibility Study August 2005, Condor Gold Project, Goldmarca Mining Ecuador Limited.

Jones, PA. 2011, Mineral Resource and Reserve Estimates January 2011, Unpublished internal report, Enterprise Capital Corporation

Lahti, H., 2007 Progress Report, December 7, 2005-12-07, El Condor Gold Project.

Landauro, A. , Gonzales, J. 2004. Estudios Metalurgicos Vetas Chinapintza-Ecuador Proyecto Condor Gold. For Goldmarca Mining Peru S.A.C.

Landauro, A., Gonzales, J. 2004. San Jose Breccias, Ecuador, Direct Cyanidation Testwork, prepared for Goldmarca Mining, Peru. S.A.

Lang, J. R. and Baker, T., 2001, Intrusion-Related Gold Deposits: the Present Level of Understanding: Mineralium Deposita, v. 36, p. 477-489.

Lenzi, J., 2004. Los Cuyes Breccia Complex: Geology, Drilled Resources and Exploration Potential; Internal Goldmarca Ltd. Report.

Litherland, M., Aspen, J.A., and Jemielta, R.A., 1994. The metamorphic belts of Ecuador: Overseas Memoir of the British Geological Survey 11, p. 147.

Maynard, A., Jones, P., and Suda, R., 2014a. NI 43-101 Technical Report on the Condor Gold Project located in Zamora, Ecuador, prepared for Ecuador Gold and Copper Corp., dated effective March 24, 2014 and published May 8, 2014.

Maynard, A., 2014b. Form 43-101F1 Technical Report, Jerusalem Gold Project, Zamora Chinchipe, Ecuador, prepared for Dynasty Metals & Mining Inc., dated October 24, 2014.

MEM, 2000 (Ministerio de Energia y Minas, Dirección Nacional de Geologia); Evaluación de Distritos Mineros de Ecuador, Depositos Porfidicos y Epi-Mesothermales Realcionados con Intrusiones de la Cordillera El Condor Vol. 5; Campo Mineral Pachicutza, pp 145-171.

Morrison, G. 2007b. Chinapintza Update, 23rd September, 2007.

Morrison, G. 2007. Ore Controls and Exploration Potential at the Condor Gold project, Ecuador. 27th March 2007. Internal Report for Ecometals Ltd.

Muntean, J. L., and Einaudi, M. T., 2000, Porphyry gold deposits of the Refugio district, Maricunga belt, northern Chile: ECONOMIC GEOLOGY, v. 95, p. 1445-1472.

NCL Engineers, Dec. 1996. Proyecto Pachicutza. Estudio de Fatibilidad, Mineria Subterranea. NCL Ingeneria y Construccion “Pachicutza Project, Condor Feasibility Study Review”.

Oviedo, L. (South American Management S.A.), 2004 Informe Recursos y Potencial Geológico, Proyecto Condor, Goldmarca Ltda.

Pitard, F. et al., 2004. Estudio de Heterogeneidad; prepared for Condor Mines S.A. (DINE-TVX Gold Inc. Joint Venture).

PWC, 2014, Ecuador – Corporate Deductions.

Ronning, P. 2003. Review of the Jerusalem Project, Ecuador. Technical Report for Vendin One Capital Corp.

Sillitoe, R.H., 1998, Epochs of intrusion-related copper mineralization in the Andes: Journal of South American Earth Sciences, v.1, p. 89–108.

Slaughter R, Carlos Rodriguez, Alfonso Postigo. Goldmarca Mining, Panama City, April 2004. Report of Activities, Condor Gold Project. English Translation of Internal Report.

Slaughter R, Goldmarca Limited 2006, News Releases Dated; February 24, 28; March 1, 14; May 31; June 6, 7, 8; July 10, 12, 31; August 10, 21; September 21; October 5, 17, 27; November 9, 10.

Slaughter R, Goldmarca Limited 2007, News Releases Dated: January 16, 26, 27; February 26; March 8, 21, 27; May 15, 18, 23, 27.

Slaughter R, Hydromet Technologies Ltd., 2002. Pachicutza Gold Project, Ecuador, Press Release.

Slaughter, R. Goldmarca Limited 2005, New Releases Dated; February 24,28; March 1, 14, 30; May 9; June 14,30; July 7,11,19,28; August 22; October 5, 13,25,27; November 14; Dec 6, 8, 21, 28.

Tobar y Bustamante, 2013, Financial and Tax Obligations in the Exploitation Contract.

Tobar y Bustamante, 2015, Legal Opinion of the Good Standing of Condormining Corporation SA, Corporacion FJTX Exploration SA companies and of the “Viche Conguime II”, “Viche Conguime III”, and “FJTX” Mining Concessions dated May 19, 2015.

Wardell Armstrong International Limited, 2014, Condor Copper-Gold Santa Barbara Deposit, Ecuador, Addendum to Tailings Dam Concept Design Report, ZS611272 REP-002 V.01, May 2014.

Williams, H. and Escudero P., 2008. Ecometals Internal Report, Jan. 2008.

Williams, H. and Escudero, P. Proposal for deep drilling at Los Cuyes. Jan. 2008. Internal Ecometals Document.

Winter, L.S. et al. A step in the formation of the Huancabamba Deflection in the Andes of Peru and Ecuador.

World Bank, 2014. Global Economic Prospects – Commodity Market Outlook.

Zaldumbide, J.P., Carcelen, J., 2014, Latin Lawyer [Online] 10 June 2013, [Cited: 15 May 2014.] http://latinlawyer.com/reference/topics/46/jurisdictions/32/ecuador/.

CERTIFICATE OF QUALIFIED PERSON

ALLEN J. MAYNARD

As the individual who has authored or supervised the preparation of Sections 1.2, 6, 7, 8, 9, 10, 11, 12, and 23 of the technical report prepared for Ecuador Gold and Copper Corp. (the "**Issuer**") entitled "NI 43-101 Technical Report Preliminary Economic Assessment of the Santa Barbara Gold and Copper Project in Zamora, Ecuador" dated effective May 19, 2015 (the "**Technical Report**"), I hereby certify that:

1. I am a principal of Al Maynard and Associates Pty Ltd., 9/280 Hay Street, Subiaco Wa, 6008, Australia.
2. I am a graduate of Curtin University, Western Australia, Australia, with a BAppSc(Geol) in 1978 (Certificate #10534).
3. I am a registered Member (#2062) of the Australian Institute of Geoscientists (AIG), a Corporate Member (#104986) of the Australasian Institute of Mining & Metallurgy (AusIMM), and I became a member of AIG in 1990 and AusIMM in 1978.
4. I have over 35 years continuous experience as a geologist in mineral exploration, resource modelling and surface and underground mining for a range of commodities including precious and base metals (Au, PGE, Li, Nb, Ni, Cu, Ag-Pb-Zn, Fe, Mn, Mo, Sn, Ta, W, U, V), REE minerals, industrial minerals (phosphate, potash, coal, mineral sands), precious and semi-precious gemstones (diamond, ruby, emerald), project generation and evaluation, as well as independent technical valuation of mineral properties in Australia, Africa, North America, South America, Western Europe, Central & Southeast Asia, China and Greenland.
5. I do, by reason of education, experience and professional registration, fulfil the requirements of a Qualified Person as defined in National Instrument 43-101 ("**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. My most recent inspections of the Santa Barbara Gold and Copper Project were on July 9th to 11th, 2010 and again on March 21st to 22nd, 2013 and previously on January 14th to 17th, 2011, in addition to a visit to Quito, Ecuador on July 7 to 12, 2010 to meet with personnel. I have also independently verified that since my last site visit, no material work has been done on the project that would change the disclosure or cause the disclosure not to be current for the sections of the Technical Report that I am responsible, as confirmed to me by the independent qualified person, Michael J. Short, BE, FIMMM, CEng of GBM Minerals Engineering Consultants Limited, who visited the site on May 18-19, 2015.
7. I have prepared, or have supervised the preparation of, and take responsibility for Sections 1.2, 6, 7, 8, 9, 10, 11, 12, 13, and 26.2 of the Technical Report.
8. I am independent of the Issuer applying the test set out in Section 1.5 of NI 43-101.
9. I have no prior involvement with the property that is subject of the Technical Report.
10. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.
11. To the best of my knowledge, information, and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 29th day of May, 2015

Original Document signed by
Allen J. Maynard, BAppSC(Geolg), MAIG, MAusIMM

"Allen J. Maynard" (signed)

Allen J. Maynard, BAppSC(Geolg), MAIG, MAusIMM

CERTIFICATE OF QUALIFIED PERSON

MICHAEL JOHN SHORT

As the individual who has authored or supervised the preparation of Sections 1.1, 1.4, 1.5, 2, 3, 4, 5, 13, 15, 16, 17, 18, 19, 20, 21, 22, 24, 25.2, 25.3, 25.4, 26, and 27 of the technical report prepared for Ecuador Gold and Copper Corp. (the "**Issuer**") entitled "NI 43-101 Technical Report Preliminary Economic Assessment of the Santa Barbara Gold and Copper Project in Zamora, Ecuador" dated effective May 19, 2015 (the "**Technical Report**"), I hereby certify that:

1. I am the Managing Director of GBM Minerals Engineering Consultants Limited ("**GBM**") of Regal House 70 London Road, Twickenham, Middlesex, TW1 3QS, England.

2. I am a graduate of the University of New South Wales, Kensington, NSW, Australia, with a Bachelor of Engineering (Civil Engineering).

3. I am a Fellow of the Institute of Materials, Minerals and Mining (C.Eng). I have worked as a Professional Engineer in the construction and mining industries for a total of 37 years since my graduation. My relevant experience for the purpose of the Technical Report is Civil and Structural Engineering, Construction Management and Logistics.

4. I have read the definition of "qualified person" set out in the 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.

5. I have visited the property, which is the subject of the Technical Report during the period from May 18 to 19, 2015 and previously during the period from September 15 to 18, 2013.

6. I have supervised the work carried out by other GBM professionals for GBM's contribution to the Technical Report, and take responsibility for Sections 1.1, 1.4, 1.5, 2, 3, 4, 5, 13, 15, 16, 17, 18, 19, 20, 21, 22, 23, 24, 25.2, 25.3, 25.4, 26, and 27 of the Technical Report.

7. I am independent of the Issuer applying the test set out in Section 1.5 of NI 43-101.

8. I have no prior involvement with the property that is subject of the Technical Report.

9. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

10. To the best of my knowledge, information, and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 29th day of May, 2015

Original Document signed by
Michael J. Short, BE, FIMMM, CEng

"Michael Short" (signed)
Michael J. Short, BE, FIMMM, CEng

CERTIFICATE OF QUALIFIED PERSON

PHILIP A. JONES

As the individual who has authored or supervised the preparation of Sections 1.3, 14, and 25.1 of the technical report prepared for Ecuador Gold and Copper Corp. (the "**Issuer**") entitled "NI 43-101 Technical Report Preliminary Economic Assessment of the Santa Barbara Gold and Copper Project in Zamora, Ecuador" dated effective May 19, 2015 (the "**Technical Report**"), I hereby certify that:

1. I am a consulting geologist for Al Maynard and Associates Pty Ltd., 9/280 Hay Street, Subiaco Wa, 6008, Australia.

2. I am a graduate of South Australian Institute of Technology, South Australia, Australia, with a B.App.Sc. (Applied Geology) in 1974.

3. I am a registered Member (#1903) of the Australian Institute of Geoscientists (AIG), a Member (#105653) of the Australasian Institute of Mining & Metallurgy (AusIMM), and I became a member of AIG in 1985 and AusIMM in 1983.

4. I have over 30 years continuous experience as a geologist in 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 do, by reason of education, experience and professional registration, fulfil the requirements of a Qualified Person as defined in National Instrument 43-101 ("**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. My most recent inspections of the Santa Barbara Gold and Copper Project were on April 10 to 16, 2011. I have also independently verified that since my last site visit, no material work has been done on the project that would change the disclosure or cause the disclosure not to be current for the sections of the Technical Report that I am responsible, as confirmed to me by the independent qualified person, Michael J. Short, BE, FIMMM, CEng of GBM Minerals Engineering Consultants Limited, who visited the site on May 18-19, 2015.

7. I have prepared, or have supervised the preparation of, and take responsibility for Sections 1.3, 14, and 25.1 of the Technical Report.

8. I am independent of the Issuer applying the test set out in Section 1.5 of NI 43-101.

9. I have no prior involvement with the property that is subject of the Technical Report.

10. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

11. To the best of my knowledge, information, and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 29th day of May, 2015

Original Document signed by
Philip A. Jones, BAppSc(Geol), MAIG, MAusIMM

"Philip A. Jones" (signed)
Philip A. Jones, BAppSc(Geol), MAIG, MAusIMM