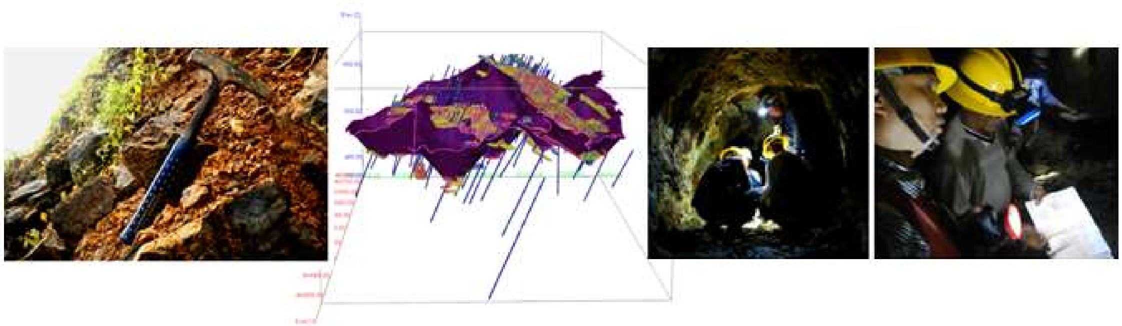
NI 43-101 Technical Report: Balabag Gold and Silver Project

Balabag, Depore, Zamboanga del Sur, Philippines



Prepared for:

TVI Resource Development (Phils.) Inc.

Report Date:

October 15, 2019

Qualified Person:

MICHAEL JAMES BUE

INDEPENDENT CONSULTING MINING ENGINEER
PROFESSIONAL ENGINEERS OF ONTARIO, CANADA NO. 5931506
CANADIAN INSTITUTE OF MINING & METALLURGY N0. 90070

CERTIFICATE OF QUALIFICATION

I, Michael James Bue, of Makati City, Metro Manila, Philippines, hereby certify that:

1. I am a Professional Mining Engineer who has worked as an Independent Consultant for a period of more than 10 years.

2. I am responsible for the review of the technical report titled "NI 43-101 Technical Report: Balabag Gold and Silver Project" located in Balabag, Depore, Zamboanga del Sur, Philippines, and dated October 15, 2019.

3. I am a Registered Mining Engineer No. 5931506 with the Professional Engineers of Ontario.

4. I am a graduate of the University of Saskatchewan (BSc Mining Engineering) and McGill University (Master of Engineering).

5. I have practiced my profession continuously for the past 42 years. I have been involved in mineral exploration, project development, mining operations and property evaluation on gold and silver properties in Canada, Indonesia, Chile, Africa, Australia, PNG and the Philippines.

6. I certify that by reason of my education, registration as a Professional Engineer, affiliation with the CIMM and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101. I am an Independent Qualified Person as defined by NI 43-101 and by the companion policy NI 43-101CP to NI 43-101.

7. This technical report is based on my personal visits to the property, review of available published data and company reports. I have spent a total of 7 days working in my office, on the property, and in the Corporate Office of TVI Resources Development Phils., Inc. in Makati, Metro Manila, Philippines working with project geologists and other technical persons of TVIRD. My visit to the property was in April 2019.

8.The resources and corresponding reserves estimates of the Balabag Gold-silver Project in my professional opinion are valid, have passed quality assurance and controls, and the estimation methods used are in accordance with NI 43-101 standards.

9. I have read NI 43-101 and Form 43-101F1. The Technical Report has been prepared in compliance with both documents.

10. I, Michael James Bue, do not own or expect to receive any interest (direct, indirect or contingent) in the properties described herein, nor in the securities of TVI Pacific or any of their affiliates. I am independent of the issuer under all criteria of Section 1.5 of NI 43-101.

11. I am not aware of any material fact or material change with respect to the subject matter of this technical report which is not reflected in this report, the omission to disclose which would make this report misleading.

12. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes. I consent to the filing of extracts from the technical report. I also consent to the inclusion of parts of the Technical Report as electronic publication on the companies' websites that are accessible to the public.

Signed in Makati City, Metro Manila, Philippines on 15 October 2019.

Mike Bue

Signature of Qualified Person

Michael James Bue, Registered Professional Engineer, No. 5931506,
Province of Ontario, Canada.

Valid until December 31, 2019

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1 SUMMARY

1.1 Location

The Balabag deposit is a low sulphidation epithermal vein system located at the municipality of Bayog, Zamboanga del Sur. The area is accessible from Manila by air transportation to Zamboanga City or, alternatively, Pagadian City. From either point, access to the property is via Zamboanga-Pagadian national highway passing through a sealed road from Imelda town to Barangay Guinoman, Municipality of Diplahan in Zamboanga Sibugay Province.

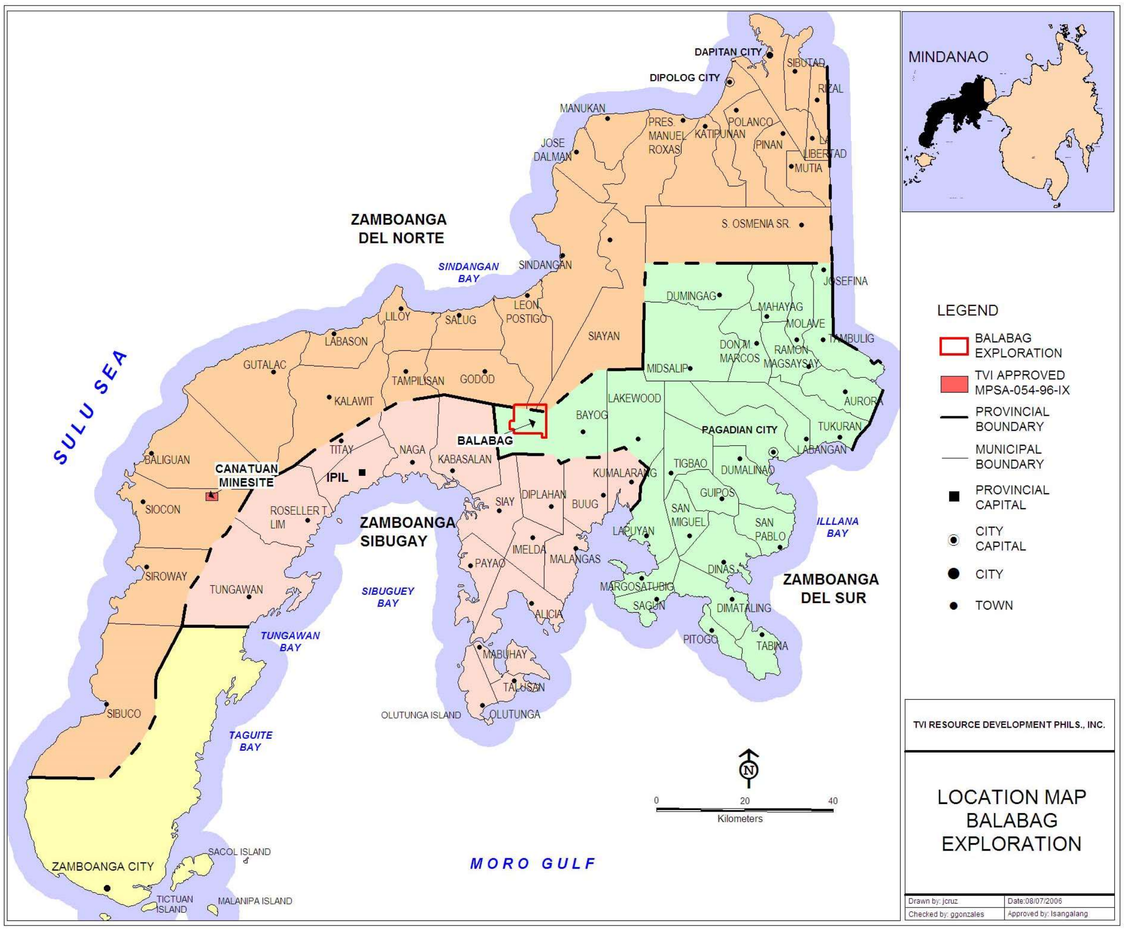


Figure 1-1 Location Map of Balabag MPSA

The property (Figure 1-1) is covered by a Mineral Production Sharing Agreement ("MPSA") that was approved on November 20, 1997, with a total area of 4,779 hectares under Zamboanga Minerals Corporation ("ZMC"). In September 2009, the MPSA was assigned to TVI Resource Development (Phils.) Inc. (TVIRD). The Balabag Project is considered TVIRD's next mine which has undergone several valuation studies and targets production in 2020.

1.2 Mineral Resource

Exploration works which include mapping, outcrop sampling and diamond drilling confirm the occurrence of gold and silver mineralization within the east-west trending epithermal vein system in Balabag Hill. The area is characterized by quartz veining and silica replacement with andesitic to dacitic volcanic and fine-grained laminated tuffs.

The veins comprise of quartz with variable proportions of pyrite and typically exhibit multiphase brecciation, crustiform banding and fine sacharoidal textures. Occasional stringered to stockworked quartz veining are also evident in the immediate walls of the veins. Hydrothermal alteration comprises silicification that grades outward to argillic and propylitic alteration.

Gold and silver are the two elements of economic interest which have been identified so far in Balabag Project. Preliminary mineralogical studies show that gold occurs as fine and coarse-grained particles, mainly high-Ag electrum with minute amounts of native gold. Silver occurs largely as acanthite and as native silver.

The mineralized units are subdivided into two major domains — true veins and quartz stockworks. The grouping is based on the general occurrence and continuity of the veins as well as grade uniformity.

The true veins which consist of banded quartz vein, massive quartz vein and quartz vein breccia have more predictable continuity and relatively uniform in grade distribution. On the other hand, the quartz stockworks are grouped as another domain since they are generally erratic in grade and in thickness.

Waste units were categorically identified among rocks that generally lack any form of significant mineralization such as andesite, andesite porphyry, dacite, volcanic breccias, phreatic breccias, hydrothermal breccias and tuffaceous sandstone.

The total undiluted mineral resources for the Balabag Gold-Silver Project are presented according to different cut-off grades in Table 1-1. The resource estimate has been validated and certified by Leo A. Sosa, Competent Person under PMRC Code.

Table 1-1 In House Mineral Resource Estimate (TVIRD, January 2014)

CLASS	DOMAIN	CUT-OFF AuEq (g/t)	Tonnage DMT	Au g/t	Ag g/t	AuEq g/t	AuEq Oz
MEASURED	DOM 1	0.0	722,749	2.09	65.04	3.39	78,835
		0.1	718,533	2.10	65.42	3.41	78,828
		0.3	700,117	2.16	67.02	3.50	78,727
		0.5	675,170	2.23	69.25	3.61	78,418
		1.0	592,301	2.47	76.96	4.01	76,340
		5.0	165,710	4.56	145.34	7.46	39,760
	DOM 2	0.0	407,445	0.36	21.00	0.78	10,187
		0.1	401,983	0.36	21.26	0.79	10,171
		0.3	354,759	0.39	23.60	0.87	9,866
		0.5	198,963	0.53	35.79	1.24	7,964
		1.0	96,915	0.74	54.34	1.83	5,691
		5.0	3,564	4.46	131.80	7.10	813
INDICATED	DOM 1	0.0	1,152,074	2.17	47.92	3.13	115,751
		0.1	1,135,173	2.20	48.61	3.17	115,718
		0.3	1,097,390	2.27	50.15	3.27	115,499
		0.5	984,487	2.50	55.28	3.61	114,126
		1.0	815,137	2.91	64.45	4.20	110,065
		5.0	197,918	6.70	154.57	9.80	62,337
	DOM 2	0.0	1,574,828	0.26	5.76	0.37	18,773
		0.1	1,518,694	0.26	5.94	0.38	18,639
		0.3	871,247	0.35	8.75	0.53	14,810
		0.5	255,501	0.62	16.99	0.96	7,910
		1.0	67,182	1.16	31.80	1.79	3,869
		5.0	1,708	6.14	12.17	6.38	350
INFERRED	DOM 1	0.0	66,972	2.06	32.69	2.72	5,847
		0.1	62,092	2.22	35.25	2.93	5,847
		0.3	61,406	2.25	35.61	2.96	5,843
		0.5	57,019	2.40	37.77	3.16	5,787
		1.0	46,854	2.81	43.49	3.68	5,536
		5.0	10,794	5.78	87.26	7.52	2,611
	DOM 2	0.0	145,731	0.26	5.50	0.37	1,723
		0.1	138,614	0.27	5.77	0.39	1,717
		0.3	81,518	0.36	8.41	0.52	1,375
		0.5	22,125	0.65	16.54	0.98	699
		1.0	7,785	1.10	25.01	1.60	402
		5.0	31	6.91	10.50	7.12	7

**Mineral resources are not mineral reserves and do not have a demonstrated economic viability. All figures have been rounded to reflect relative accuracy of the estimates.*

The breakdown of mineral resources at a cut-off grade of 0.4 g/t AuEq in Measured and Indicated classification are detailed in Table 1-2.

Table 1-2 Measured and Indicated Mineral Resources at 0.4 g/t AuEq cut-off (TVIRD, January 2014)

CLASS	ROCKGROUP	Tonnage DMT	Au g/t	Ag g/t	AuEq g/t	AuEq Oz
Measured	DOMAIN 1	686,501	2.20	68.24	3.56	78,585
	DOMAIN 2	265,302	0.46	29.43	1.05	8,922
	TOTAL	**951,803**	**1.71**	**57.42**	**2.86**	**87,506**
Indicated	DOMAIN 1	1,032,062	2.40	53.01	3.46	114,808
	DOMAIN 2	419,685	0.50	13.17	0.76	10,262
	TOTAL	**1,451,748**	**1.85**	**41.49**	**2.68**	**125,070**

**Mineral resources are not mineral reserves and do not have a demonstrated economic viability. All figures have been rounded to reflect relative accuracy of the estimates.*

1.3 Planned Mine and Mill Capacity

The Balabag ore reserves estimate stands at 1.35 M tonnes at 2.50 g/t Au and 68.25 g/t Ag. The total reserves will be mined by side cut method at a projected waste to ore stripping ratio of 10:1. Table 1-3 presents the total volumetric estimate of the mineable reserves.

Table 1-3 Summary of Balabag Ore Reserves

	Rockgroup	Volume m3	Density DMT/m3	Tonnage DMT	Au g/t	Ag g/t	AuEq g/t	AuEq oz
PROVEN	DOM 1	250,349	2.4	600,837	2.42	75.09	3.55	68,578
	DOM 2	36,241	2.5	90,601	0.65	54.73	1.47	4,274
	SUB-TOTAL	286,590	2.5	691,439	2.19	72.42	3.28	72,852
PROBABLE	DOM 1	256,317	2.4	615,160	2.95	65.75	3.93	77,802
	DOM 2	16,904	2.5	42,259	0.95	36.40	1.50	2,037
	SUB-TOTAL	273,220	2.5	657,420	2.82	63.86	3.78	79,838
TOTAL ORE RESERVES	DOM 1	506,666	2.4	1,215,998	2.69	70.36	3.74	146,380
	DOM 2	53,144	2.5	132,861	0.74	48.90	1.48	6,310
	TOTAL	**559,810**	**2.5**	**1,348,859**	**2.50**	**68.25**	**3.52**	**152,690**

The mine was defined based on the resource block model as of December 2018, with metal prices set at USD 1300/oz and USD 18/oz for gold and silver, respectively. The internal cut-off grade is 0.83 g/t gold equivalent (AuEq), which was calculated based on the assumed metal prices, combined milling, selling and General and Administrative costs and plant recovery.

Mine optimization using Whittle mining software was carried out to estimate the mineable ore reserves. The following were used as inputs to the optimization process:

Table 1-4 Parameters Used in Mine Optimization Process

	Variable		
Optimization Parameters	**Mill Throughput**	t/d	2,000
	Upper Slope Angle	deg	40
	Distance from surface	m	25
	Lower Slope Angle	deg	45
	Ore Mining Cost	$/t	2.51
	Waste Mining Cost	$/t	2.00
	Milling Cost	$/t	23.00
	Gold Price	$/oz	1300
	Silver Price	$/oz	18

The mine will be free-draining. Mining is planned such that no water will accumulate within a pit. Runoffs will be diverted through channels into the tailings storage facility.

The Balabag gold and silver plant will consist of two-stage crushing circuit utilizing primary and secondary jaw crushers to produce a coarse stockpile. Crushing is followed by grinding utilizing a SAG mill and two ball mills operating in closed circuit with cyclones. A bleed from the secondary cyclone underflow will be processed for the recovery of free gold by gravity concentration and sent to the gold room for smelting.

Bulk flotation is done to ground ore to produce silver concentrate. The silver and gold from the flotation concentrate will be dissolved by cyanidation process. Pregnant solution containing the gold and silver will be recovered by two-stage dewatering combination of the high-rate thickener and filter press; then forwarded to the Merrill-Crowe circuit for gold and silver precipitation using zinc dust.

Tailings from bulk flotation and re-pulped flotation concentrate leach residue will be processed in carbon-in-leach (CIL) tanks to extract and simultaneously recovery of residual gold and silver by cyanidation and carbon adsorption. Loaded carbon will be stripped and the precious metals will be recovered as sludge through an electrowinning process.

Final tailings will be treated in the detoxification process for cyanide destruction. Overall gold and silver recoveries on the grade to be treated are projected at 87.9% and 88.3%, respectively.

Activities related to detailed design for Balabag Project commenced during the fourth quarter of 2015. Detailed design for the buck earthworks for the plans and infrastructure sights have been completed while detailed design for the plant is on progress. The construction phase will immediately follow the detailed design. The latter will involve the construction of the two vital surface infrastructures of the project – mill plant and tailing storage facility. The mill plant will be constructed southeast of the deposit, with about 900 m of hauling distance from the mine. The tailings storage facility will be a downstream designed earthfill dam northeast of the mine. Mine pre-stripping will be conducted alongside dam and mill construction.

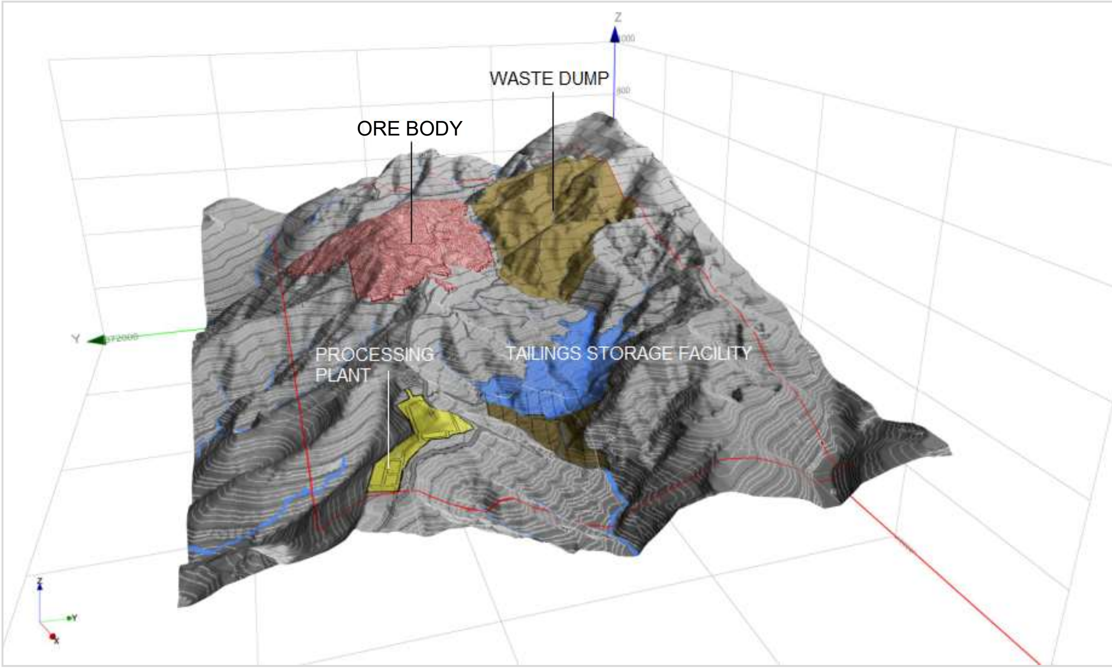


Figure 1-2 Three-Dimensional Perspective of the Balabag Project Showing Mine Facilities Relative to Topography (Looking West)

1.4 Estimated Life of Mine

The Balabag Project is expected to produce gold and silver for an estimated period of three (3) years from the mineable reserves. The anticipated dilution of 5 to 10% has been anticipated and included resource blocks and in the cost of waste movement.

Exploration is currently ongoing to further increase the resources and to convert inferred resources to measured and indicated categories, thereby increasing the mine life.

1.5 Volume of Investment

The Balabag Project's initial capital expenditure for Year 1 of operations is estimated at USD 28 million (

Table 1-5). This includes costs associated with predevelopment works, mine development and pre-stripping, construction of plant infrastructure, tailings storage facility, and a contingency of 10%. The following table summarizes the overall CAPEX for the project.

Table 1-5 Capital Expenditure for the Balabag Project

CAPEX	UOM		Year 0	Year 1	Year 2	COST (M)
Pre-Development Cost	US$		1,500,000	-	500,000	2,000,000
Mine Development & Pre-Stripping	US$		1,000,000	500,000		1,500,000
Plant & Mill Infrastructure	US$		20,900,000			20,900,000
Tailings Storage Facility	US$		5,000,000	5,000,000		10,000,000
Waste Dump	US$		-	300,000	1,000,000	1,300,000
Channel Drain	US$				1,600,000	1,600,000
Exploration	US$	400,000		1,000,000	1,000,000	2,000,000
Total	**US$**	**400,000**	**28,400,000**	**6,800,000**	**4,100,000**	**39,300,000**

From 2005 to 2011, TVIRD spent $12M (Php 591 million) on exploration-related activities in Balabag Hill. Bulk of this was spent on 199 drill holes summing up to 23,700 m that currently define the Project's mineral resource. The table below summarizes the annual exploration expenditures for Balabag Project.

Table 1-6 Balabag Prospect Exploration Expenditures, 2005-2011

Year	Cost, in Million Php
2005	15.2
2006	108.9
2007	81.3
2008	12.9
2009	7.6
2010	135.1
2011	230.3
Total	**591.2**

During the latter part of 2018, additional 12 holes with a total meterage of 1,068 m was drilled over the Lalab area, at a cost of $0.4M (Php20 million).

1.6 Market

The World Gold Council (WGC) believes that the demand for gold is likely to increase, as supported by structural economic reforms, uncertainties in the financial market, and geopolitical unrests.

Consumer demand in key markets rose in 2018 due to positive economic growth. However, from an investment standpoint, gold faced headwinds as investment demands for gold falters due to strengthening of the dollar and rising of US interest rates.

By the fourth quarter of 2018, many of these negative factors started to ease away, thus increasing the demand for gold. Consequently, the price rose comfortable above USD 1,250 / oz.

In 2019, suggestions that the US economy will experience weaker growth could slow down rising interest rates and limit dollar strength. This, along with the structural economic reforms is perceived to support the gold demand for technology, jewelry, and long-term savings. Furthermore, the increased market uncertainty and protectionist economic policies are seen to make gold increasingly attractive as a hedge.

From a global perspective, the short-term outlook is that market risk and uncertainty will continue to make gold attractive. In the long term, the performance of gold will be supported by the development of the middle-class emerging markets, gold's role as an asset of last resort, and its growing use in technological applications.

1.7 Production Costs

The successful Canatuan Gold Plant Operations and the Copper-Zinc Mine formed the basis of estimating the operating costs for Balabag with adjustments to reflect costs of the updated process flow sheet and incorporation of current pricing of major consumables.

Overhead cost estimates are also based on Canatuan Operations but were adjusted to a reduced organization, required to run a relatively small mine operation. Overhead costs are considered fixed dollar expenses and are independent of the mill throughput. The key cost assumptions are summarized in **Error! Reference source not found.**.

Table 1-7 Key Cost Assumptions

Key Cost Assumptions:	UOM	COST
Ore Mining Cost	USD/t	2.51
Waste Mining Cost	USD/t	2.00
Milling Cost	USD/t	21.84
Site Overheads	USD/day	5,000.00
G&A	USD/day	2,500.00
SDMP	% of OPEX	1.50
EPEP	USD M	6.90
EXCISE TAX	% GR	4.00
NSR ROYALTY (Zamboanga Minerals Corporation)	% GR	2.50
IP ROYALTY (ICC Subanen Tribe)	% GR	1.00
FMRDP	USD M	5.02
FOREX	USD=PHP	54.00
REFINING COSTS – GOLD	% GR	0.10
REFINING COSTS – SILVER	% GR	1.00

1.8 Employment

Balabag Project will require a total of 544 personnel for the fully operational stage. This consists of 109 staff and 435 non-staff (rank-and-file) personnel and excludes those that are employed by the mine contractors.

Department managers and senior technical staff will be hired from the local group of competent professionals with vast experience in the minerals industry. The country also has a growing population of young professionals and engineers, as Philippine universities continue to promote mining-related courses following the boom of the mineral industry in the past decade.

Skilled and non-skilled workers, as well as some administrative personnel can be recruited from a pool of qualified locals from the host and nearby communities. Balabag Hill has been occupied by artisanal miners which can provide the labor backbone for the mine and mill operations. This has been the successful experience in Canatuan Mine which can be applied to Balabag Project.

The company also has a pool of highly competent managers, technical personnel and skilled workers with previous experience in gold-silver operation (TVIRD's CIP-CIL plant).

1.9 Profitability

Initial cash flow projections using a gold price of USD 1300/oz and silver price of USD 18/oz indicate a Net Present Value (NPV) of USD 17.0 million at 8% discount rate with Internal Rate of Return (IRR) of 37%.

However, in response to the current trend in metal price and forecasts, the base case scenario for the feasibility study considered conservative gold and silver prices of USD 1250/oz and USD 15/oz, respectively. At 8% discount rate, the estimated NPV for the Project is at USD 12.0 million with a projected IRR of 30%. This demonstrates that the Balabag Project is still viable at a much lower metal price.

Sensitivity analysis was conducted to test the impact of varying economic and operating criteria to the project. Variations to gold price, stripping ratio, capital cost, gold grade and gold recovery were applied to assess the viability of the Balabag Project.

The results of the sensitivity analysis show that the project is highly sensitive to gold price and grade. The mine is still expected to generate cash and yield positive NPV even at 10% drop in any of the mentioned key economic parameters.

1.10 Sources of Financing

The capital for the initial site development will be drawn from banking institutions in the form of loans. Additional financial requirements to cover other mine and plant improvements, as well as operating expenses will be generated internally.

1.11 Risks and Mitigation Analysis

A risk assessment indicates low risk as to the tailings storage facility and security. The tailings dam is designed based on downstream construction with seismic and hydraulic factors reviewed by Knight Piesold Engineering.

The possible changes in the fiscal regime on the project economics were also considered and will not affect the viability of the project.

The open pit ban may pose a risk but this has been mitigated by designing the mine to be free draining to the tailings dam. Future developments also considered a construction of a channel at the lowest benches to allow water to drain into the tailings dam.

There were two other critical paths of the project: the securing of the mining and environmental permits, and the relocation of the illegal small-scale miners (SSM) occupying the project area. During the last quarter of 2012, all illegal miners occupying the area were removed through the serving of the Cease and Desist Order (CDO) from the Department of Environment and Natural Resources (DENR), Mines & GeoSciences Bureau. A combined team of government agencies led by MGB and the Zamboanga del Sur provincial government successfully implemented the CDO. On the other hand, the Notice to Proceed has been awarded in 2018 after securing a Declaration of Mining Project Feasibility.

2 INTRODUCTION

The technical report has been prepared as a commissioned work pursuant to the Canadian Instrument NI 43-101. The author has relied mainly on the information provided by the technical personnel of TVI Resources Development Phils. Inc. based on previous technical reports as well as the latest Mineral Resource and Reserves report prepared.

This report discloses the Mineral Resource and Reserve estimates as of December 31, 2018. The validation and assessment of Balabag Gold-silver Project was conducted by the undersigned from April to May 2019 with a mine site visit on 09 April 2019.

2.1 Units, Currency and Rounding

The units of measure used in this report are as per the International System of Units (SI) or "metric" except for Imperial units that are commonly used in industry (e.g., ounces (oz.) and pounds (lb.) for the mass of precious and base metals. All dollar figures quoted in this report refer to US dollars (US$ or $) unless otherwise noted.

The calculation of Gold Equivalent "AuEq" in the report is as follows unless otherwise noted;

AuEq = Au grade + (Ag grade x (Ag Recovery/Au Recovery x Ag price/Au price)

Frequently used abbreviations and acronyms can be found in Section 28. This report includes technical information that required subsequent calculations to derive subtotals, totals and weighted averages. Such calculations inherently involve a degree of rounding and consequently introduce a margin of error. Where these occur, the QPs do not consider them to be material.

3 RELIANCE ON OTHER EXPERTS

Sample protocols, including sample methodology, preparation, analyses and data verification have been conducted in accordance with industry standards using appropriate quality assurance/quality control procedures in all exploration activities of TVIRD direct supervision of the OIC- Exploration.

A review by **Mr**. **Leo A. Sosa**, of the procedures employed by the company in terms of field sampling, security in handling the samples, sample preparations and the QA/QC protocols concludes that all these are in accordance with sound industry best practices and is sufficient for use in performing the mineral resource estimation. He is Competent Person under the Philippine Mineral Reporting Code (PMRC) and recognized by the Philippine Stock Exchange (PSE) and currently the Operations Geology and Exploration manager of TVIRD.

The Mineral Resource Report on The Balabag Gold and Silver Project has been signed and approved by Mr. Sosa in 2014.

The estimation of the mineable reserves for the mine was conducted by **Emmanuel Puspos**, a registered mining engineer under the Professional Regulations Commission of the Philippines, and under the guidance of **Mr. Marcelo A. Bolano,** a third-party consulting mining engineer and a Competent Person under the Philippine Mineral Reporting Code**.** The key economic parameters used in mine optimization and cost estimates have been validated and approved by the Competent Person who also updated the nickel laterite resources and prepared an updated report

Mr. Jake G. Foronda (MAusIMM, Metallurgical Engineer, and VP Operations for TVIRD) – supervised and directed the development of the process design and provided technical information and economics of the Project.

4 PROPERTY DESCRIPTION AND LOCATION

4.1 Location and Accessibility

The Balabag Gold and Silver Project is located in the southern segment of the Zamboanga Peninsula, situated at Sitio Balabag, Barangay Depore, Municipality of Bayog, Zamboanga del Sur. The property is covered by a Mineral Production Sharing Agreement denominated as MPSA-086-97-IX; approved November 20, 1997.

It encompasses 4,779 hectares straddling Zamboanga del Sur, Zamboanga Sibugay and Zamboanga del Norte.

The MPSA for Balabag property was originally awarded in favor of Zamboanga Minerals Corporation ("ZMC") and was later assigned to TVIRD in September 2008. It is bounded by geographic coordinates from 7º51'30" to 7º55'30" latitude and from 122º53'30" to 122º58'00" longitude.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Access

The area is accessible from Manila by air transportation to Zamboanga City Airport, serviced daily by two airlines. From Zamboanga City, access to the property is by a 6-hour drive via Zamboanga-Pagadian national highway passing through a sealed road from Imelda town to Barangay Guinoman, Municipality of Diplahan in Zamboanga Sibugay Province. Alternative routes are via Pagadian City which is serviced daily by one airline and Dipolog City which is serviced by one airline three times a week. TVIRD in cooperation with the Local Government Unit of Diplahan constructed a 13 kilometers graveled road from Barangay Guinoman to Balabag Hill which serves as a farm to market road.

5.2 Terrain and Physiography

Generally, the topography is moderately rolling to semi-rugged. Elevation ranges from about 200 to 900 meters above mean sea level (AMSL). The highest peak is located some 3.5 kilometers directly northwest of Balabag that towers above the surrounding area at 928 meters AMSL.

According to the Modified Coronas Climate Classification Scheme used by the Philippine Atmospheric Geophysical and Astronomical Services Administration (PAGASA), the climate in Balabag and its adjoining areas is classified under Type IV. This is characterized by mild or moderate temperatures and rainfall. Rainfall is evenly distributed throughout the year. It ranges from 1,599 mm in the driest areas to 3,500 mm in the wettest ones. Temperature is relatively warm and fairly constant throughout the year varying only from a minimum of 22°C to a maximum of 35°C.

5.3 Drainage System

Kabasalan River is the main drainage system passing across the western and northern half of the tenement. The eastern and southern sections are drained by the Sibuguey River and Depore River, respectively. The creeks are mostly characterized by steep and V-shaped valleys. The drainage pattern is distinctly dendritic although some trellis and rectangular patterns were likewise observed.

5.4 Vegetation

Patches of secondary growth forests and log areas from previous commercial logging operations with only a few large mother trees now remain as the dominate vegetation cover at the northwest portion of MPSA area. Some ridges have been denuded of its forest cover and locally cultivated to rice, corn, ginger, cassava, sweet potato and other root crops, and other fruit trees. Left uncultivated particularly in Balabag area, the denuded slopes commonly host lush growths of fast-growing tree locally known as '*buyo-buyo'*.

5.5 Land Use

By and large, the area which has been occupied by small-scale miners is planted with minimal agricultural and seasonal crops. The works of the small-scale miners may have disturbed the ground soil as evidenced by some erosion due to the effect of running waters. Illegal miners extracted gold and silver through surface and underground workings, mercury-based amalgamation process, while the CIP operators recovered gold using cyanide.

6 HISTORY OF MINING CLAIMS AND MINING OPERATIONS

The MPSA for Balabag property was originally awarded to Zamboanga Minerals Corporation ("ZMC"). From October 15, 1996 to January 14, 1998, the ZMC property was put under an option agreement with Rio Tinto Exploration Philippines Corporation (RTEPC). During this period, RTEPC conducted detailed geological mapping and rock sampling, geochemical stream sediment and grid-soil sampling, and an IP-Resistivity survey.

Then, Templar Gold N.L., an Australian company, also undertook an option agreement with ZMC from February 18, 1999 to November 25, 2004; Templar Gold N.L. drilled five shallow holes to test the prospects within the area.

TVIRD's first phase of exploration within the area was concentrated on confirming the presence of gold, silver and quantifying respective values. In 2005, TVIRD conducted due diligence work which involved channel sampling of sites previously sampled by RTEPC.

During the latter part of 2005 TVIRD's geologists methodically revisited, mapped and resampled some existing small-scale tunnels and accessible workings that surround the Tinago-Miswi mineralized system.

Data gathered confirm the presence of significant gold and silver values associated with the Tinago vein at Balabag. The distribution of illegal small-scale miners also indicated that gold is also present in the other mineralized areas, at Unao Unao, Miswi Lalab and at least one other vein structure.

TVIRD commenced its own drilling program with the first hole collared on November 17, 2005. In light of the encouraging results, the company decided to further expand the program.

7 GEOLOGICAL SETTING AND MINERALIZATION

7.1 Geological Setting

The regional geology of the Philippines is dominated by Tertiary and younger plutono-volcanic sequences that are superimposed on both continental and oceanic crust. The regional geology of the Zamboanga Peninsula is less well understood than other parts of the country, and its mineralization potential remains to be quantified as the area continues to be underexplored.

7.1.1 Regional Geology

The Mindanao Island is bounded to the west by the Eurasian Plate being subducted southeastward into the Sulu Trench, and to the east by the Philippine Sea Plate being subducted westward into the Philippine Trench.

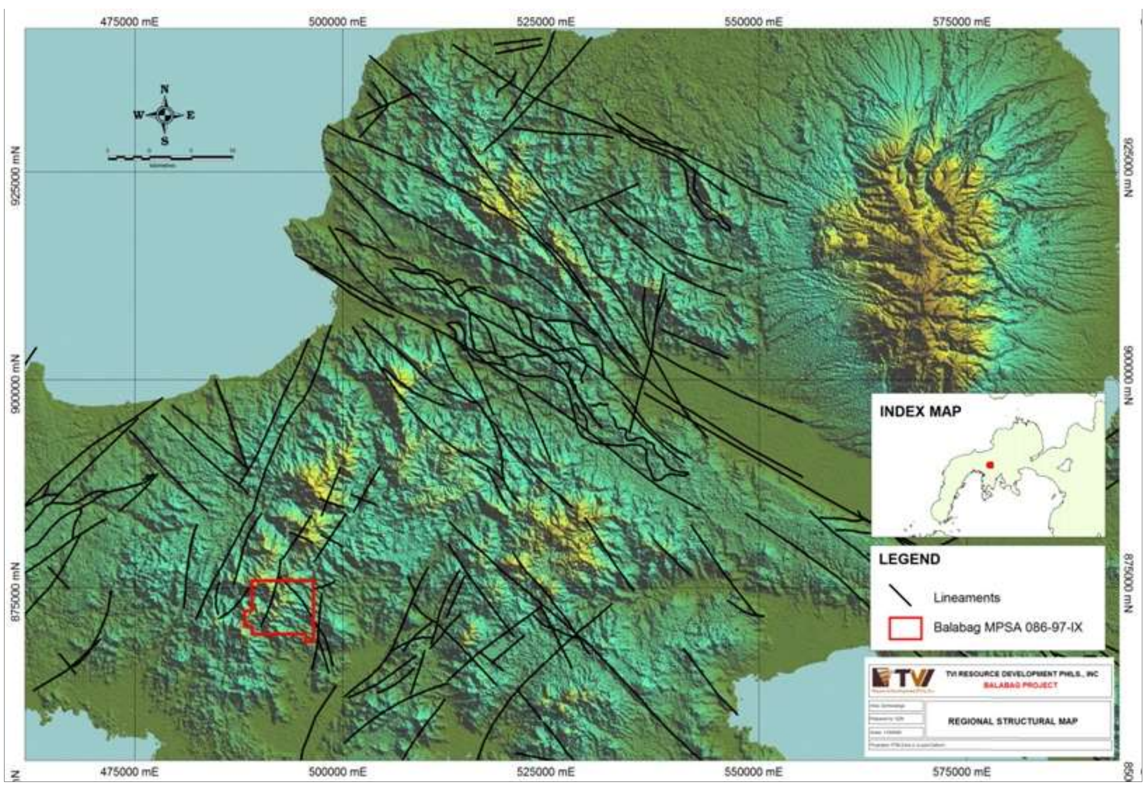


Figure 7-1 Regional Structural Map of North Central Zamboanga Peninsula

This complexity is increased in Zamboanga (West Mindanao) by the NE-subducting Cotabato trench that is expressed inland as a collision zone along the Cotabato-Sindangan Fault, which is believed to connect northwestward to the Negros Trench (Acharya and Aggarwal, 1981). This tectonic setting has produced three distinct rock stratigraphic assemblages, namely: (a)

SW-Zamboanga Zone; (b) Cotabato-Sindangan Collision Zone and (c) NE – Zamboanga Zone (Flores, 1999).

The SW-Zamboanga Zone consists of a generally NE-trending and relatively older suite of rock stratigraphic units. This includes a pre-Tertiary basement complex consisting of Triassic schists and other metamorphics, Cretaceous ultramafics and ophiolitic rocks. Paleocene to Miocene sediments and volcanics unconformably overlie the basement complex. Miocene intrusives and hypabyssal rocks intrude the preexisting rocks. The youngest sequences comprise Quaternary volcanics and finally a young cover of Quaternary sediments, alluvium and terrace gravel.

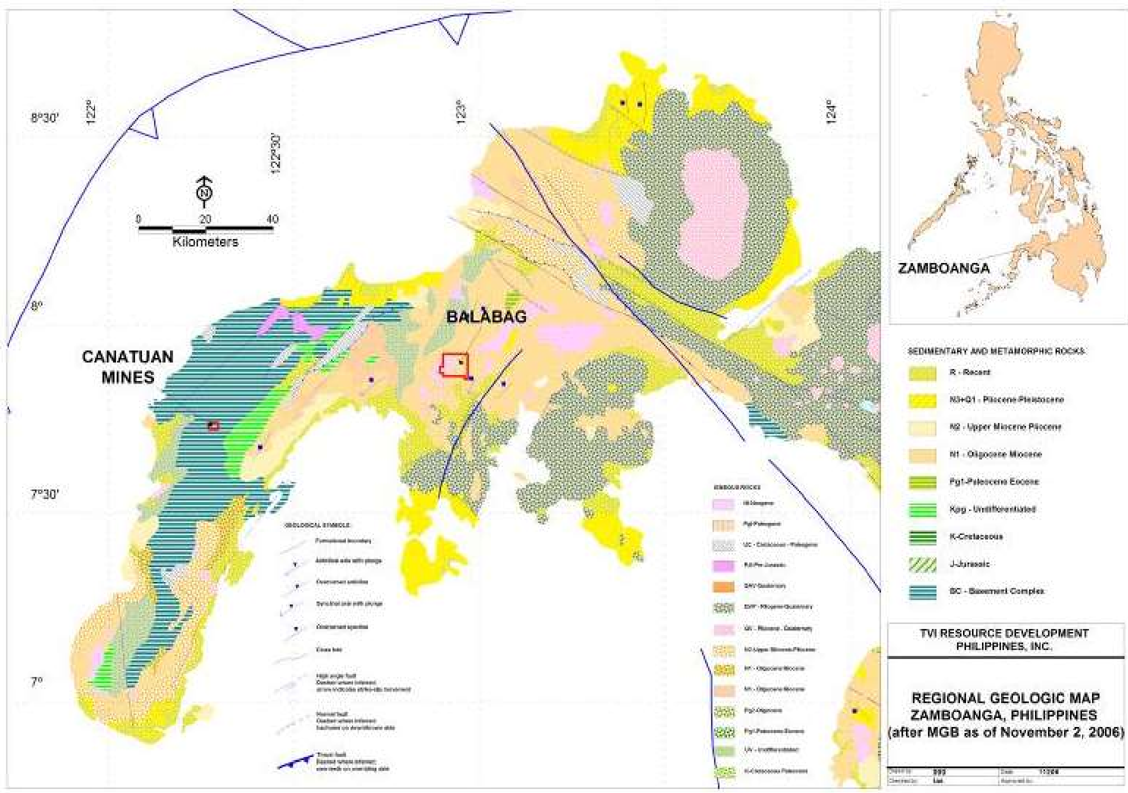


Figure 7-2 Regional Geologic Map of Zamboanga Peninsula (After MGB, 2006)

The Cotabato-Sindangan Collision Zone is characterized mostly by NW-trending braided or anastomosing sinistral faults and similarly trending lithostratigraphic units. Rock suites comprise Cretaceous ultramafics and ophiolitic rocks, Paleocene-Eocene sediments and Oligocene to Miocene volcanics and sediments, Miocene intrusive and hypabyssal rocks, Quaternary igneous sequences (both intrusive and extrusive), and alluvium comprise the youngest sequences.

The NE Zamboanga Zone is mostly covered with the Quaternary Malindang Volcanics and related lahar and alluvial deposits.

7.1.2 Local Geology

The oldest rock unit mapped in the area is a diorite intrusive inferred as pre-Oligocene (Antonio, 1951). This is overlain by late Oligocene to Middle Miocene sediments. The sediments are generally steeply dipping and unconformably overlain by volcanic rocks.

A younger intrusive suite of andesite to dacite porphyries intruded the volcanics, sediments and older diorite. Oligocene to middle Miocene in age, the sedimentary units within the Balabag MPSA is composed of indurated and bedded calcareous-sandstone-siltstone and mudstone sequence with interbedded limestone, conglomerate and greywacke. This unit is exposed at Unao-Unao creek, north of Balabag Hill.

The volcanic rocks, consisting of porphyritic andesite and dacite, pyroclastics or lithic tuff and volcanic flow breccia are the most extensive rock type in the area. They unconformably overlie the interbedded and steeply dipping (65°-70° SW) sediments. This unit constitutes the host rock of the Balabag epithermal gold veins.

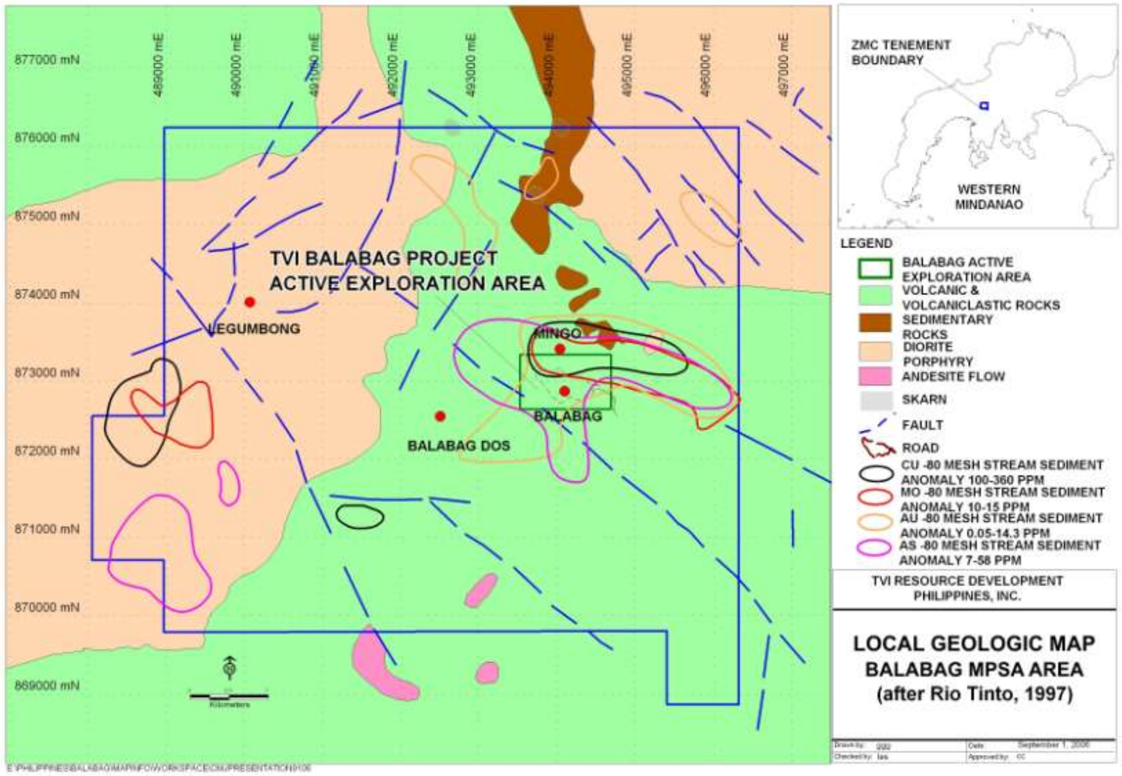


Figure 7-3 Local Geologic Map of Balabag MPSA Area (After RTEPC 1997)

Sedimentary Rocks

The sedimentary unit consists of well indurated sequence of bedded calcareous mudstone-siltstone-sandstone-conglomerate intercalated with limestone. It is locally exposed north of the Balabag hill at Unao-Unao Creek and Lower Dipili River. The mudstone is thinly laminated and brown to dark grey in color. The sandstone consists of lithic fragments cemented by a calcareous matrix. The conglomerate is poorly sorted and usually contains clasts of limestone and other sedimentary rocks. The limestones are white to grey and crystalline. The beds generally strike NW and dip 60-70° SW.

Volcanic Rocks

The volcanic rocks comprise volcaniclastics intercalated with andesitic lava flows.

Volcaniclastics

The volcaniclastics include ash tuff, lithic tuff, monomictic, and polymictic volcanic breccia. The ash tuff is usually cream to buff in color and are thinly bedded. The lithic tuff is well indurated, greenish to grayish black and is usually propylitic. The monomictic breccia are poorly sorted, matrix to clast supported and contain andesitic clasts cemented by an andesitic sandy matrix. The polymictic breccias are also poorly sorted; matrix supported and contains clasts of limestone, porphyritic andesite, aphanitic andesites, and occasional mudstone and clastic rocks.

Andesite

The andesite lava flows are aphanitic to porphyritic and occasionally contain calcite amygdules. The mineral assemblage is typically dominated by plagioclase with visible chlorite patches. Thin section of altered porphyritic andesite illustrates complete replacement of plagioclase phenocrysts (centre left, bottom right) by fine-grained dense sericite (tiny yellowish flecks), and replacement of a hornblende phenocryst (upper right) by quartz (pale yellow, white) and fine-grained dense chlorite (anomalous dull green).

Plutonic Rocks

Large intrusive bodies of quartz diorite are exposed in the western section of the Balabag MPSA. They also outcrop northeast of the tenement. Younger intrusives comprising andesite porphyry are exposed south of the tenement.

Diorite

The diorite is characterized by interlocking crystals of sodic plagioclase, quartz, biotite and fibrous hornblende. It exhibits hypidiomorphic granular to medium-grained texture.

Andesite Porphyry

The andesite porphyry occurs as narrow dikes and sills intruding the older volcaniclastic, lava flows and the calcareous sediments. It contains zone of turbid insets of plagioclase (andesine), well-formed but altered crystals of pyroxene (augite) and slender prism of hornblende.

7.1.3 Mineralization

The gold-silver mineralization in the Balabag deposit is classified as a low sulphidation epithermal vein system. It is characterized by quartz veining and silica replacement in andesitic lava flows and fine-grained laminated tuffs and andesite porphyry. The true veins variably exhibit multiphase brecciation, crustiform banding and fine saccharoidal texture.

Occasional stringers and quartz vein stockworks occur as halo of the true veins, oftentimes on the hanging wall side. The vein width in the stockworks ranges from <1 to 20 cm and have vein densities of 1 to >5 veins per meter. Hydrothermal alteration comprises argillic, chloritic, propylitic and silicic alteration.

Drilling and mapping suggests the occurrence of three major vein systems being Tinago on the north, Miswi to the southeast and Lalab to the south. The geometry of the veins and the associated stockworks are generally sub-horizontal instead of the prevalent sub-vertical nature of Philippine epithermal veins (C.A. Angeles, 2010). The Tinago veins dip toward the north-west direction at 20° to 30° and appear to locally consist of several sub parallel individual veins that vary in width from less than 1 m to more than 10 m.

The main mineralized zone is situated at the Tinago area where a relatively more continuous quartz vein system along strike has been interpreted based on drill data of 25 m spacing. The Tinago vein strikes generally ENE and appears to be traceable for several hundred meters.

Figure 7-4 Quartz Vein Breccias with Colloform/ Crustiform Banding

The thickest mineralized zone exposed at surface is located at the Tinago small-scale mining area that is locally called as Warik-Warik (Figure 7-5). Outcropping veins are generally colloform and crustiform banded with dark fine grained sulphidic bands. The banded portions are about 3 m to 10 m thick and bordered by massive silica-illite-sericite zone.

Figure 7-5 Vein Outrcrop at Warik-Warik

Other gold veins occur in the Miswi-Lalab area, located at the southern slope of the Balabag Hill. The vein system in this area is characterized by oxidized sugary quartz veins with vuggy and comb texture. Breccia texture is also common in this zone.

Gold occurs mainly as high-Ag electrum with minor amounts of native gold. It occurs as fine and coarse-grained particles. Fine-grained Au is mainly locked in non-opaque minerals, and to a lesser extent, pyrite. In some cases, large grain size and euhedral crystal form of pyrite contains high concentration of gold. On the other hand, silver occurs mainly as acanthite and as native silver. The gold-to-silver ratio in Balabag tends to be high with an average of 1:28. Open space fillings are common and include vugs, drusy cavities, cockscomb textures, crustifications, and symmetrical banding.

Types of Quartz Veining

There are two general types of mineralization in Balabag: the true veins and the quartz stockworks. The true veins have more predictable continuity and grade uniformity while the quartz stockworks are generally erratic in grade and thickness.

The true veins exhibit various textures and are further classified into three classes: the massive quartz vein (rock code: VNQ), the banded quartz vein (rock code: VNB) and the quartz vein breccia (rock code: VNX).

The massive quartz vein is milky and translucent. They are usually moderate to low-grade with assays ranging from 1.0 to 42.81 g/t Au and 2.0 to 900.42 g/t Ag and averaging about 3.5 g/t Au and 97.44 g/t Ag.

The banded quartz vein usually exhibits colloform to crustiform banding with occasional "Ginguro" bands. The "ginguro" ore contains alternating dark and light-colored layers. The dark bands are caused by the presence of fine grained black sulphides mostly sphalerite and galena. The banded veins are usually moderate to high-grade with values ranging from 1.0 to 41.55 g/t Au and 4.8 to 551 g/t Ag and averaging 4.85 g/t Au and 195.76 g/t Ag.

The quartz vein breccia contains sub-angular to angular clasts of early vein material cemented by later quartz. The vein clasts are composed of banded and massive quartz vein material and altered wall rock. This type is usually associated with very high assay values. The grade ranges from 1.0 to 104.55 g/t Au and 0.7 to 5,254.30 g/t Ag and averaging 5.84 g/t Au and 179.19 g/t Ag.

The stockworks are characterized by crisscrossing 1 cm to 20 cm veinlets. They are generally low-grade at <1.0 g/t Au and <10 g/t Ag. They are hosted by the andesite flows, volcaniclastics and the andesite porphyry. They are usually confined peripheral to the vein zone but sometimes occur independently in small and insignificant zones.

Figure 7-6 Banded Quartz Vein with Distinct "Ginguro" Bands (3.44 g/t Au)

Fluid inclusion studies were conducted by Dr. Douglas Mason of Mason Geoscience Pty Ltd and Dr. Andreas Schmidt Mumm of the School of Earth and Environmental Sciences of the University of Adelaide to determine the temperature of deposition of the ore minerals.

Fluid inclusions were analyzed using a LINKAM THM600 heating and cooling stage which is routinely calibrated to an accuracy of ±0.1°C in the low temperature range of 0-350°C (Mumm, 2010).

Two quartz vein samples (BAL-TIN-2010-02, BAL-TIN-2010-03) were used in fluid inclusion study. The two samples contained blades and laths of quartz with microthermometrically analyzable fluid inclusions. Primary fluid inclusions revealed low salinity (0.2 to 4.3 NaCl equiv. wt. %) fluids with indication for a predominance of KCl over NaCl as the dissolved electrolyte. Homogenization temperatures of 167°C to 307°C (mean=258°C) are a first order minimum estimate of the mineral forming conditions if boiling of the fluids occurred (Mumm, 2010).

Another fluid inclusion study was performed by Hakane Hagiwara of Akita University. Her findings were almost consistent with the results produced by Dr. Mumm. Based on her study the fluid inclusions in quartz are two-phase inclusions consisting of vapor phase and liquid phase. The salinity of fluid inclusions in quartz formed during the stage 1 ranges from 2 to 3 wt% NaCl, while that of the stage 2 ranges from 1 to 2.7 wt% NaCl. Salinity of fluid inclusions in quartz is lower near the center of the vein.

The relationship between the NaCl equivalent salinity and the homogenization temperature of fluid inclusions, indicates that at the time of mineralization, the temperature of fluid was 230-270°C and the depth of formation was deeper than 600 m below the paleo water table.

Alteration Type and Style

Several alteration types are observed within Balabag Hill, namely silicic (quartz+ illite+kaolinite), argillic (illite+kaolinite+quartz), propylitic (epidote+chlorite+smectite) and chloritic (chlorite+smectite). The argillic alteration type is subdivided into supergene argillic which is a result of surficial processes and argillic which is hypogene in nature. Propylitic alteration is postulated to be regional in character.

Wall Rock Alteration

In Balabag, the immediate wall rock of the vein systems is predominantly argillic and chloritic. Near surface intercepts have been characterized as supergene argillic. Hypogene argillic is more evident at depth. Argillic alteration is more pervasive to the south of the hill particularly within the Lalab-Miswi areas, compared to the Tinago side.

Chloritic alteration (chlorite+smectite) is distinguished from the propylitic zone as it is inferred to be related to the vein mineralization. It is distinguished by the absence of epidote. It appears that the chloritic alteration has overprinted the regional propylitic alteration during vein formation as the veins are enveloped by a chloritic halo at depth.

Narrow silicic alteration halos occasionally occur peripheral to vein walls. They also occur as dike-like bodies within the pervasively argillized rocks in the Miswi area.

The geometry of the argillic zone suggests that a downward regional tilting in the range between 30° and 60° wherein the Balabag veins were originally anywhere from vertical to moderately dipping, are now lying on their backsides. The southern part, Lalab-Miswi area, is the upper part of the vein system. The central part of the Tinago area is the middle part of the vein system. And the northern part of the Tinago area is the lower part of the vein system. Just like a typical vein system, the veins are numerous at the upper part and coalesce at the bottom part. So, there are more veins at the Lalab-Miswi area, lesser in the central part of Tinago area and the least in the north part of the Tinago area.

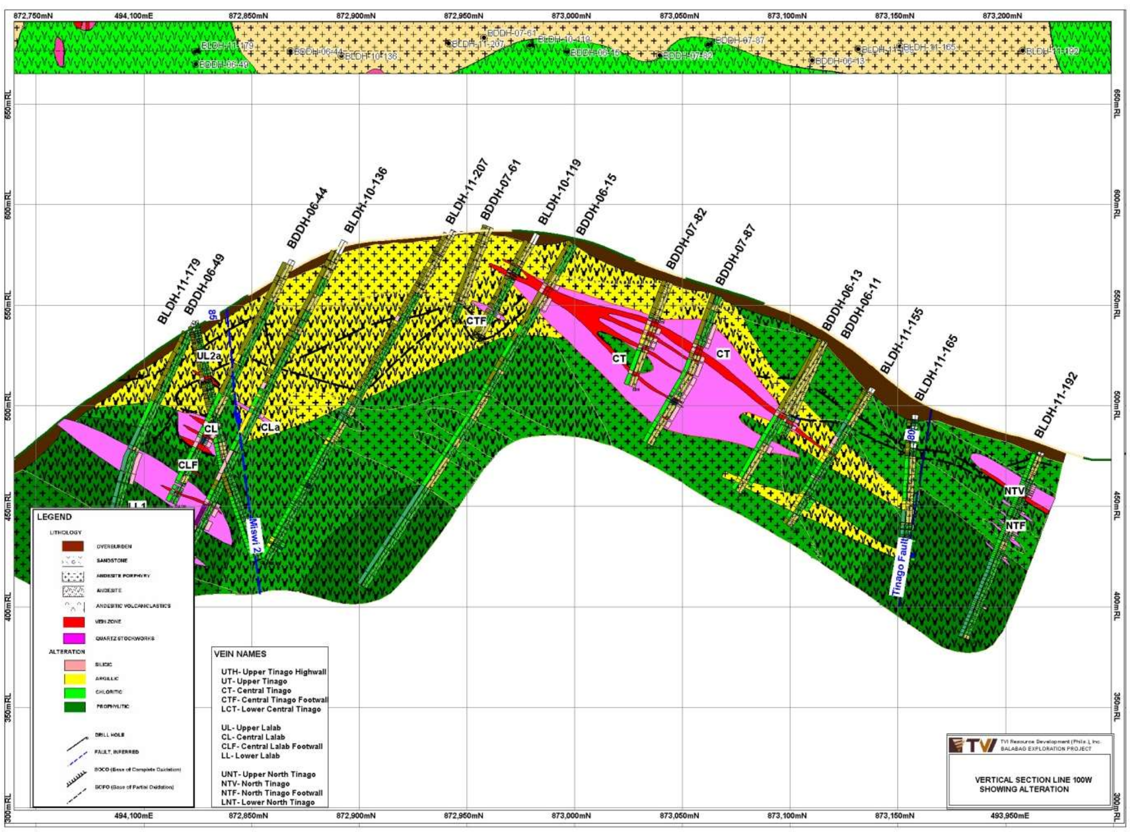


Figure 7-7 Section 100W Showing Alteration Zones

<u>Paragenesis</u>

Mason (2010) conducted mineralogical and petrological studies using petrographic methods, supplemented by optical mineragraphic observations about textural, paragenesis and assemblages in selected quartz veins of Balabag. Two stages of quartz formation were determined.

- *Quartz ± sulfides*
  Earliest quartz is commonly very fine-grained (microcrystalline) and massive. It tends to form indistinct laminae or bands, and it may be accompanied by minor sulfides (pyrite, sphalerite, chalcopyrite, and galena).

- *Quartz + sulfides ± minor carbonate, zoisite, K-spar*
  Later formed quartz is medium-grained and massive, commonly in bands with sulfides (pyrite, sphalerite, chalcopyrite, and galena).

Hakane Hagiwara, in her separate study also identified a third stage wherein the earlier formed massive and banded quartz were brecciated and recemented with later quartz. Banding in the quartz veins was observed mainly due to the difference in grain size of quartz. Her study also reveals that gold mineralization initiated at the later part of Stage 1 but most of the electrum crystallized immediately after the brecciation. These findings are consistent with the assay results of the drill core wherein the vein breccias exhibit the highest grades while the earlier formed massive veins contain lower gold values.

7.1.4 Localization of the Deposit

7.1.4.1 Lithological Controls

The quartz veins occur in all type of volcanic rocks in the area, from the andesite lava flows to volcaniclastics and the andesite porphyry dikes and sills. There appears to be no lithological control over the emplacement of the veins. No veining has been discovered within the older calcareous sediments underlying the volcanic although a skarn deposit has been reported to occur north of the MPSA area.

7.1.4.2 Structural Controls

At Warik-warik, several NW-SE parallel joints dipping steeply to the SW cut the banded vein. This suggests that the joints postdate the mineralization. The reverse and later normal E-W trending faults, i.e. Tinago and Miswi Faults, are cut by later NS to NNW-trending strike slip faults, mainly the Lalab, Warik-Warik and Tinago-Miswi Faults. The inferred sinistral Warik-Warik Fault traverses the Warik-Warik pit and displaces the Central Tinago (CT) vein. East-west trending low sulphidation epithermal gold-silver veins cut through all of the rock units. The veins are dipping to the north; about 30 to 60º.The veins are bounded in the north and south by later post-mineral un-mineralized faults, generally dipping about 70 to 80º to the south. In the north, the Tinago Fault and several subparallel minor faults trend east-west while in the south, the Miswi Fault and other faults are generally ENE-trending. These faults are presumed to be all reverse faults due to N-S compression and later some evolved to normal faults during times of relaxation. This is shown by the jostling of blocks containing veins either up or down towards the north relative to the central vein area. In the eastern portion of the drilled area, there are also NNW-trending post-mineral right-lateral strike slip (or dextral) faults which are probably later than the E-W trending faults since there are north-south displacements of the veins as well.

Structure definitely exerts the strongest control on the spatial distribution and ore shoot development of the low-sulphidation epithermal veins at Balabag. The veins are EW-trending and shallowly dipping. As said above, the sub-horizontal dip of the veins is probably due to a downward regional tilt related to thrusting in the area. Post-mineral E-W trending faults, initially reverse and later evolved to normal faults such as the Tinago and Miswi Faults, cut the veins but are sub-vertical instead of sub-horizontal. The last episode of faulting is NS to NNE-trending and strike-slip in movement, i.e. Lalab, Warik-Warik Fault and Tinago-Miswi Faults, which truncates both the E-W trending veins and faults. There is also elevation control to the mineralization as there is a mineralization bottoming of hypogene Au grades at about 425 to 450m RL. Low-sulphidation epithermal gold veins usually occur in the zone of retrograde boiling of the hydrothermal fluids, about 100-400m below the paleo-surface. Also, supergene oxidation of veins at the upper elevations produces better grade than the deeper hypogene parts of the vein (Angeles, 2012).

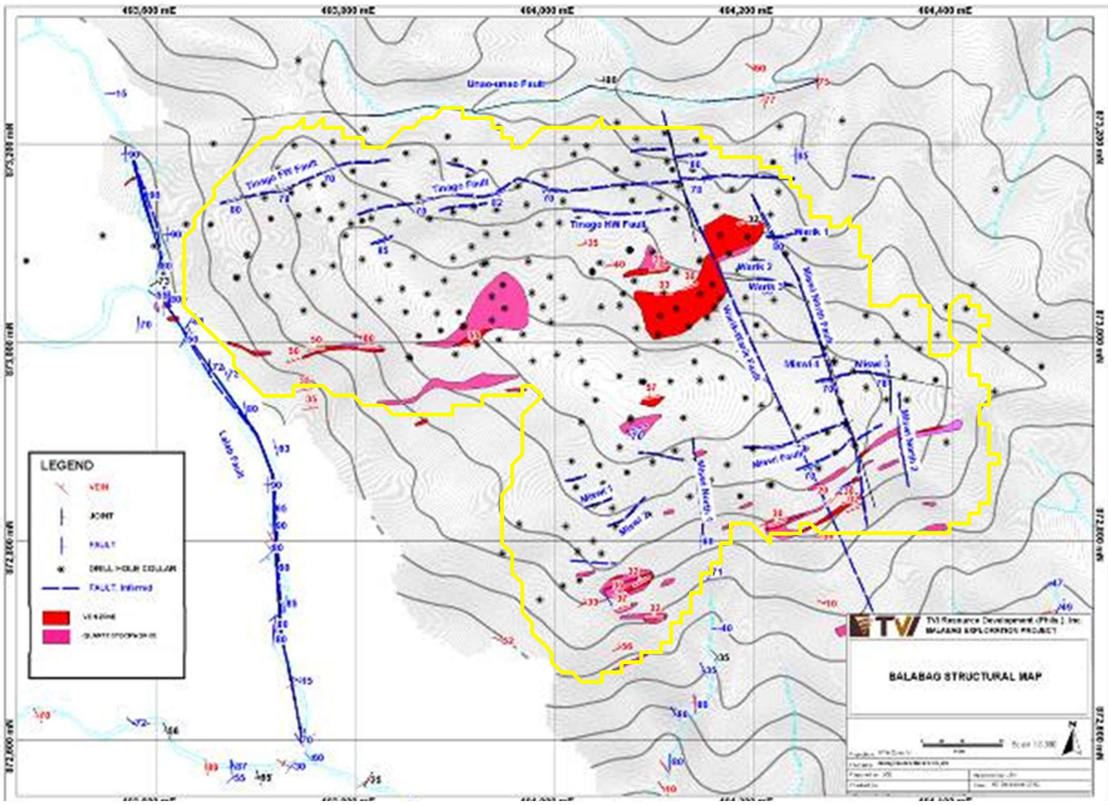


Figure 7-8 Structural Map of Balabag as Projected from Drill Hole Data showing the Outline of the Mine Pit

8 DEPOSIT TYPES

The principal copper and gold deposits of the Philippines originate as an integral part of the formation of volcanic arcs, and many copper-gold deposits are associated with intrusions (mostly diorite and quartz diorite, but also monzonites and syenites) as well as Pliocene – Pleistocene volcanism (Lepanto mine at Mankayan, Benguet).

The complex geology of the Zamboanga Peninsula is host to several styles of mineralization. The hydrothermal systems generated by the volcanic and intrusive activity, fall into one of three categories:

1. Porphyry copper-gold systems associated with high level plutonic intrusions, which are primarily dioritic in composition in the Philippines;

2. Epithermal gold systems typically associated with deep seated fault systems; or

3. Volcanogenic massive sulphide deposits associated with the submarine discharge of metalliferous hydrothermal fluids.

9 EXPLORATION

9.1 Exploration Works

9.1.1 Geologic Mapping and Sampling

In 2006 to 2007, detailed drainage geological mapping was conducted in the southeast of Balabag Hill (Genaro and Minda Creek); in the southwest-west of Balabag Hill (Depore River, Lalab and Baby Creek) and on the north (Unao-Unao Creek) with a total traverse length of 13.5 km.

The area mapped is underlain primarily by volcaniclastics and lava flows intruded by narrow porphyritic to andesite porphyry dikes. The volcaniclastics usually exhibit a dark reddish tint due to hematite staining. Underlying the volcaniclastics and the lava flows are steeply dipping sediments.

On the southeast of Balabag Hill, a major north trending fault is exposed along Minda Creek. It is characterized by a 4 m wide breccia zone. Structures along this area follow a more northerly direction compared to the previously generally observed NW-SE trend. Several silica-clay-pyrite alteration zones were identified along Minda Creek. This could represent the southeastern extension of the silicified zone/quartz veining at Balabag Hill.

On the west of Balabag Hill, the previously identified major fault along the Depore River extends northward. Structures still follow a general NW-SE trend. An almost 30 m wide andesite porphyry dike cuts across a volcaniclastic sequence along a northeasterly trend. Along the Depore creek, it appears that the silica-clay pyrite alteration is only confined to the footwall of the Vitalla vein. The upstream portion which represents the hanging wall sequence shows only propylitic alteration.

On the north of Balabag Hill, a small window of steeply-dipping sediments is exposed at Unao-Unao creek. The clastics consist of intercalated sandstone-siltstone-shale with minor conglomerate. The clastic sequence interbedded with minor beds of limestone.

Due to the security threats from illegal small-scale miners, no detailed surface mapping was conducted over the Balabag Hill. Instead, a surface geological map of the Hill was reconstructed based on drill hole data.

Surface sampling was carried out during the course of geological mapping. Grab samples from vein and altered outcrops were collected. Significant veins were channel sampled using diamond rock saws in combination with hammer and moil. The geologist together with the experienced field assistants routinely marked sample sites of rock chip and grab samples with flagging tapes. Spray paint was used to mark sites of channel samples, so they can be easily located and identified.

Standard sampling procedures for grab, rock chip, channel and trenches sampling have been systematically carried out by TVIRD field samplers with the supervision of geologists and field assistants. Most of the surface samples were collected from the so-called Warik-Warik, a “gloryhole”, in which much of the vein that was worked previously from underground by the small-scale miners is now exposed at surface due to a massive collapse. The sampled material represents remnant boulders of vein material that are not in situ but have only moved a few meters and rest on unmined portions of the same vein. Fifty-five samples, typically comprising 1 m chip channels perpendicular to the banding seen in the boulders, were collected. The average of all samples collected by TVIRD was 9.48 g/t Au, 635 g/t Ag. The values range from a low of 1 m at 0.49 g/t Au, 4.6 g/t Ag to a highest grade channel sample which returned 1 m at 62.2 g/t Au, 8,175 g/t Ag.

Trenching along the projected strike of the Tinago, Lalab and Miswi veins was also implemented to probe vein continuity. The trenches were spaced about 50 m apart and oriented almost perpendicular (NW to SE) to the general trend of the veins. A total of 33 trenches with a total length of 1066 m were excavated until the end of 2007. Channel samples were taken along the trench at 1 m interval. A total of 1189 samples were collected.

TVIRD conducted mapping and sampling of some illegal miner’s underground workings in Tinago and Miswi. Sample points selected underground were recorded with reference to distance from the tunnel’s portal.

Twenty-five samples were collected from Berong Tunnel which is located below the Warik-warik area. These were taken in the deepest portion of the stopes, representing approximately 16 m of down-dip extent of the vein. Much of the adit is little more than 1 m in height, but some 2 m composites were collected. The best intercept is 2 m at 20.3 g/t Au and 149 g/t Ag. The results range from 1 m at 2.03 g/t Au, 17.5 g/t Ag to 1 m at 21.6 g/t Au, 145 g/t Ag. The average of the 25 samples is 7.92 g/t Au, 38.5 g/t Ag. In Tunnel 32A situated at the main Tinago vein, TVIRD chip-channeled 11 narrow locations, representing approximately 6 m of down-dip extent of the vein. These samples average 4.77 g/t Au, 72.9 g/t Ag. The highest grade is 1 m at 13.6 g/t Au, 80.5 g/t Ag and range down to 1 m at 0.36 g/t Au, 27.8 g/t Ag.

Fifty-three samples were collected from the vein Tunnel 29A, and six from the hanging wall. Approximately 12 m of up-dip extent within the same plane of the vein was sampled. The mined height is typically only 1-2 m, and as a consequence the number of composites is very limited. Individual samples within the vein range from a low of 1 m at 0.53 g/t Au, 6.2 g/t Ag to 1 m at 20.2 g/t Au, 115 g/t Ag. The best composite is 3 m at 13.0 g/t Au, 125 g/t Ag. The average of the samples from the vein is 6.07 g/t Au, 91.5 g/t Ag.

The locations of underground samples were determined by the geometry of the workings. The three tunnels sampled access the veins from the hanging wall. Upon reaching the vein, a drift is driven following the hanging wall contact of the vein. After entering the vein at variable distances, and before encountering the footwall contact, the drives become inclined and subparallel to the hanging wall. These inclined drives are the active stopes. The majority of the samples were taken as 1 m chip channels in the “ribs” (or walls) to these stopes.

Samples were taken perpendicular to dip, and therefore reflect true widths. However, none of the samples represent the full width to the vein, as the footwall was never exposed. The samples in any one stope represent the same preferred, approximately planar, portion of the vein. In the wider stopes, it was occasionally possible to take composite samples, as vein width is more than a meter.

Predetermined samples sites were marked with spray paint markings. The channel cut was about six inches wide and two inches deep. Chip sampling was from bottom to top, or from “sill” of the workings to the “back”. This system eliminates contamination from top material adhering to the lower portions of the channel if sampling was taken from top to bottom. After the samples were taken, the sample locations were sprayed for easy identification. The samples were chipped manually using a heavy hammer and a steel moil/chisel. The chipped rock was caught on a canvass sheet placed immediately below the sampling area. Channel spacing on vein is 1.5 m.

The sample weight is approximately 4 to 5 kg. In order to avoid sampling bias, the samples were shipped without prior splitting. The responsible geologist logged and described each sample, with all observations written in a sample ticket logbook with a corresponding sample tag that is placed in the sample bag. The sample number was also written on the surface of the sample bag, for double checking and easy identification during the dispatch of the samples to the laboratory. Samples numbers were plotted on a sample location map, produced at the same scale as the detailed underground geological map.

Exploration Geochemistry

Geochemical surveys were conducted by Rio Tinto Exploration Philippines Corporation prior to the entry of TVIRD in the property. They implemented regional stream sediment sampling over the whole MPSA extending into their other FTAA applications. They also conducted 50 m x 25 m grid –soil geochemical sampling over the Balabag Hill deposit. A total of 213 stream sediment samples and 984 soil samples were collected during the surveys. No further geochemical survey was done by TVIRD.

Applied Geophysics

An Induced Polarization and Resistivity survey was conducted in Balabag in 1997 by RTEPC. A total of 9.6 line km consisting of six (6) lines were surveyed. No ground geophysical work was done by TVIRD.

The exploration potential of the Balabag MPSA area is discussed in Section 23 and Appendix IV-G.

10 DRILLING

10.1 Resource Definition Drilling

TVIRD started drilling the property on November 17, 2005 with the first hole BDDH-05-01 collared at central Tinago area. Two drill holes were completed during the year.

The following year, drilling accelerated and 57 drill holes were completed with a total meterage of 6,953.40. Due to the encouraging results, TVIRD embarked on an infill drilling campaign in 2007. However, drilling was suspended in April 2007 to allow independent consultants to estimate an initial resource using geostatistical estimation techniques and to prepare the NI 43-101 technical report on the project. Accumulated drilling meterage for three months of drilling was at 3,964.30 meters or a total of 101 drill holes including re-drill.

In February 2010, TVI resumed infill drilling after a three-year hiatus with the collaring of BLDH-10-100 in Unao-unao. Several subsequent phases of drilling were completed until the second quarter of 2012, including sterilization holes along the proposed sites of the mill, mine camp, waste dump and tailings storage area. Several holes were also drilled to the west of Balabag Hill to chase the possible extension of the Tinago vein system farther to the west. In April 2012, after the completion of BLDH-12-274, drilling was suspended due to security concerns.

After the successful implementation of the cease and desist order issued by the government against the illegal miners, the company resumed infill drilling in January 2013. A total of 18 holes with a total of 1220.55 meters were drilled until April 2013.

The drilling was implemented by Exploration Drilling Corporation (EDCO). Several types of diamond drill machines were used including four LY-38, two CS-1000, one Gopher-160, two Edson 150, one HYDX-150 and two EDCO-150. Core sizes are generally PQ to HQ. However, it is sometimes reduced to NQ at greater depths. As of April 2013, a total of 296 drill holes with a total meterage of 34,155.60 were completed (Table 10-1). The average core recovery was at 98.51%.

The details of drilling and core handling protocols are discussed in **Appendix IV-J**.

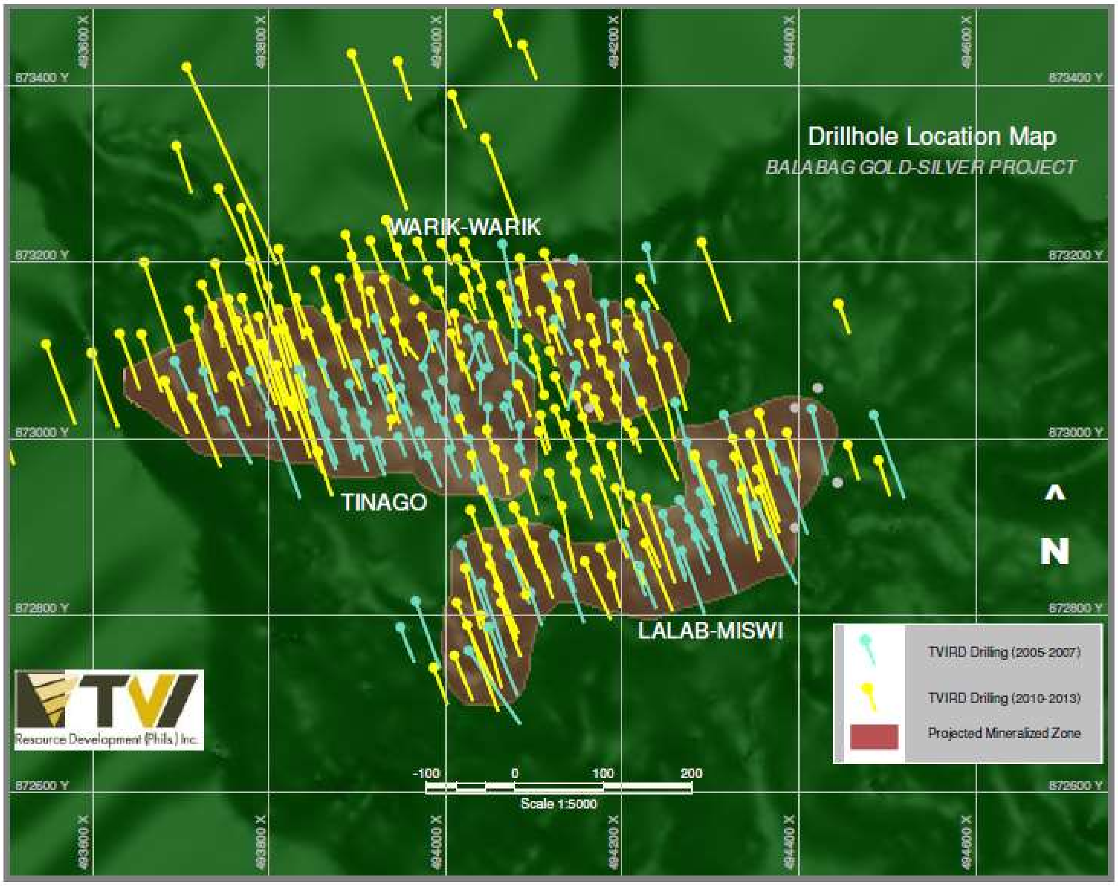


Figure 10-1 Balabag Drill Hole Location Map (as of April 2013)

Table 10-1 Balabag Exploration Drilling Statistics (as of April 2013)

Purpose	Borehole Count	Length (m)	Ave. Core Rec.
Resource Definition Drilling			
Pre-2010	101	10,974.70	92%
2010	54	7,595.15	95%
2011	90	10,975.00	98%
2012	5	1,179.60	99%
2013	18	1,220.55	98%
Subtotal	268	31,945.00	
Sterilization Drilling	16	1,706.45	99%
Geotechnical Drilling	12	504.15	99%
TOTAL	**296**	**34,155.60**	

11 SAMPLE PREPARATION, ANALYSES, AND SECURITY

11.1 Sample Preparation

Preliminary sample preparation is then carried out on site. Samples are initially hand crushed into ≤40 mm diameter fragments. The samples are then poured into two clean sample pans with the accompanying sample tag and barcode placed over the sample. The sample pan are then placed in an oven with maintained temperatures of 105°C and dried for 6-7 hours. The dried samples passed through the Boyd crusher to a final reduction of ≤2 mm.

The crushed sample is then split progressively into approximately 1 kg sample using a Jones splitter with 1' opening. The 1 kg split is then packed in plastic inside a calico bag for submission to the Canatuan Mine Assay Laboratory or the ESPF in Ipil for pulverizing. The remaining crushed split sample is properly labeled and stored at site.

At the ESPF, the 1 kg split is pulverized to -200 mesh (or 75 microns) using a disc pulverizer. Splitting of the pulverized sample is repeated until a 500 g representative sample is collected. It is then packed in a kraft envelop, sealed and transported in plastic containers to the analytical laboratory for analysis. It is ensured that the duplicate of the sample ticket is preserved and inserted in the sample until it is received by the attendants at the assay laboratory.

11.2 Analyses

From 2005 to 2006, all samples were analyzed by Mcphar Laboratory in Manila. Since 2007 until the present, all samples are analyzed by the company's internal assay laboratory at the Canatuan Mine, Siocon, Zamboanga del Norte. Drill core and rock samples are analyzed for gold using 50 g fire assay with atomic absorption spectroscopy (AAS) finish and 35 other elements such as: Ag, Al, As, B, Ba, Bi, Ca, Cd, Co, Cr, Cu, Fe, Ga, Hg, K, La, Mg, Mn, Mo, Na, Ni, P, Pb, S, Sb, Sc, Se, Ti, Sr, Th, Ti, U, V, W, and Zn using multi-element – Induced Coupled Plasma-Optical Emission Spectroscopy (ICP-OES), following HCl/HNO3/HclO4 leach on 0.5 g sample.

11.3 Security

All samples are collected by well-trained field samplers under the supervision of the geologist and geologic assistant on site. All drill core and rock and samples from the field are bagged in polyethylene. The duplicate copy of the sample number from the sample card is then inserted into the bag and sealed by a twist tie. The plastic bag is in turn inserted into a calico bag and tied using the built-in cord. The calico bag is then labeled on its side with the corresponding sample number as the sample card using a permanent marker. The samples are then arranged, packed in sacks, sealed and stored in the camp while waiting for dispatch to the Canatuan Laboratory or to the Exploration Sample Preparation Facility (ESPF) located at Ipil, Zamboanga Sibugay.

12 DATA VERIFICATION

For densities, drill cores were used to verify earlier data.

For the rechecking of the integrity of laboratory assays, the insertion of blanks, duplicates, and pulp rejects in the same batch as the mainstream samples was incorporated into the QA/QC Procedures.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

TVIRD led a comprehensive metallurgical test campaign to evaluate the feasibility of processing the Balabag gold and silver ore. As a vital part of the initial scoping phase, a bulk mineralogy and gold deportment study was initiated in order to characterize the ore and acquire some guidance into developing the basic process flowsheet. Several metallurgical test campaigns were undertaken to determine the amenability of the ore to various unit operations and concentration processes. Based on the accumulated findings from test campaigns, the optimal and most suitable flowsheet in terms of overall economics was selected.

The very first metallurgical test campaign started in 2007 when TVIRD contracted Genivar SEC (GENIVAR) to make a complete scoping study on the mining potential of the Balabag deposit focusing on the Tinago and Miswi vein deposits. Included in the GENIVAR report were the results of the metallurgical scoping study carried out by SGS Lakefield on the amenability of the Balabag ore to metallurgical processes as gravity concentration, cyanidation, and heap leaching. Grindability and settling test works were also conducted. Based on the SGS report, GENIVAR recommended a Gravity – Carbon-in-Pulp (CIP) – Merrill Crowe process combination to extract and recover the gold and silver.

An extensive metallurgical prefeasibility study was conducted in 2011 at the metallurgical laboratory in the Canatuan Mines in Siocon, Zamboanga del Norte, Philippines, which included optimization of critical parameters such as cyanide strength, activated carbon dosage and loading, and leach and adsorption times. The effect of oxidizing agent during straight cyanidation to improve the leaching kinetics of the silver was also investigated. Exploratory flotation tests were also conducted at varying silver head grade as preparation for elevated silver grades in the ore, which could reach up to 300 g/t Ag. For the treatment of cyanide leach tailings, detoxification tests were carried out at varying pulp densities. From the results of this metallurgical test campaign, the team came up with a Gravity – Flotation – Merrill Crowe – CIP process combination.

In January 2013, the construction of a CIP pilot plant in Canatuan Mines commenced, and on May of the same year the 1 t/d straight cyanidation pilot plant started operation. A continuous pilot test campaign was carried out for six weeks using a new batch of ore samples from Balabag. Pilot runs were conducted at varying grind sizes, cyanide strengths, and oxidizing agent dosages. Precious metals extractions and recoveries obtained from the CIP pilot plant were comparable to that of the bench-scale test results. Activated carbon loaded with gold and silver were produced as final product. However, due to the very long leaching time which was deduced to be caused by presence of carbonaceous materials, the team decided to pilot test a carbon-in-leach (CIL) process route. The CIL test led to a reduced contact time for the same gold and silver recoveries. From the pilot test findings, it was concluded that the CIL process was the better scheme than CIP process for precious metals extraction.

Another set of bench-scale test works were initiated to focus on optimizing the Flotation – Merrill Crowe – CIL process route. The test campaign involved optimization of flotation reagent dosages, flotation times, cyanide strengths, and adsorption times. Variability tests using different feed grade were also conducted to assess the flexibility and robustness of the proposed process.

Key parameters and assumptions for the process design criteria and design of the Balabag gold and silver plant were derived from the results of the pilot test campaign and all the optimization test works. The proposed Hybrid Flotation – CIL – Merrill Crowe Plant is designed to run at 2,000 t/d. There will be a two-stage crushing circuit, a SAG-Ball Mill-Pebble Crusher grinding circuit, a gravity circuit, a rougher flotation circuit, a standard CIL recovery circuit and a Merrill Crowe section that will treat the high-silver flotation concentrate.

Most of the grinding and flotation equipment are available from the Canatuan Mines. Only the leach, detox and reagent tanks and the Merrill-Crowe and associated equipment need to be purchased. The capital cost for these facilities is estimated at USD 10M.

The average metal recoveries derived from the metallurgical tests were 89% for gold and 95% for silver on samples grading 3.4 g/t Au and 84 g/t Ag.

The average grade of the deposit is 2.50 g/t Au and 68.25 g/t Ag, which is lower than the samples tested. At this grade, the recoveries are projected to be 87.9% for gold and 88.3% for silver. The operating cost is estimated at USD 22.73/t.

13.1 Mineralogy

A bulk mineralogy and gold deportment study was conducted on five Balabag head samples by Joe Zhou Mineralogy Ltd. In Canada. A comprehensive mineralogical and analytical approach including fire assay, gravity pre-concentration, ore microscopy, X-ray diffraction (XRD), and scanning electron microscopy (SEM) was used in the study. Representative digital photomicrographs were taken to show the general mineralogy and the deportment of gold.

Each ore sample was stage crushed to 80% passing 150 µm. Representative sub-samples were riffled out for assays, XRD semi-quantitative analysis, and pre-concentration for gold deportment study. Approximately 1.4 kg of each crushed sample was subjected to heavy liquid separation (HLS) to obtain a sink fraction (heavies, consisting mainly of sulfides, oxides, and heavy silicate minerals) and a float fraction (lights, consisting of silicates, or silicates with disseminated sulfide minerals). The sink fraction was further concentrated by super panning (SP) to separate liberated gold and gold-bearing minerals from the gangue minerals. A schematic of the concentration procedure is shown in Figure 13-1.

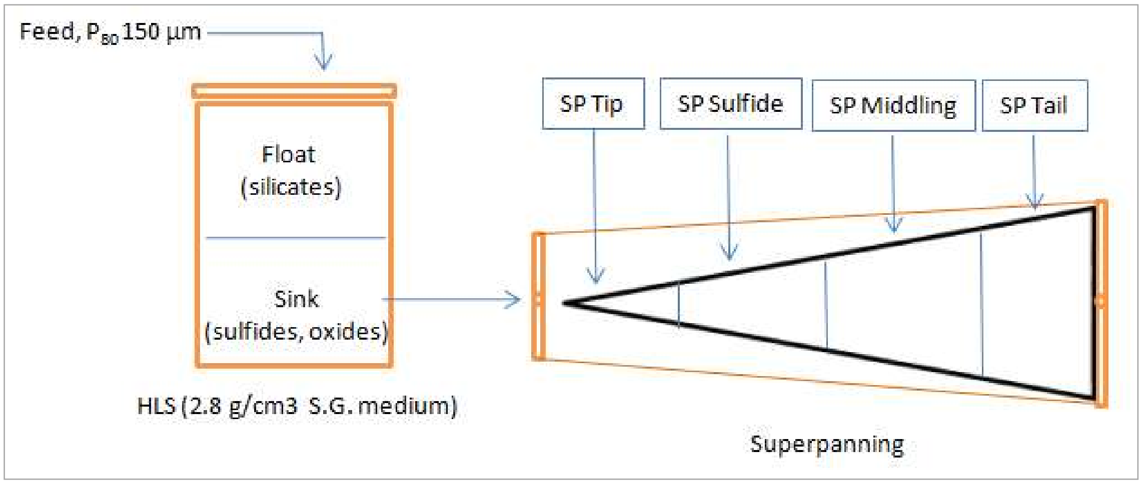


Figure 13-1 Schematic of Pre-concentration Procedure

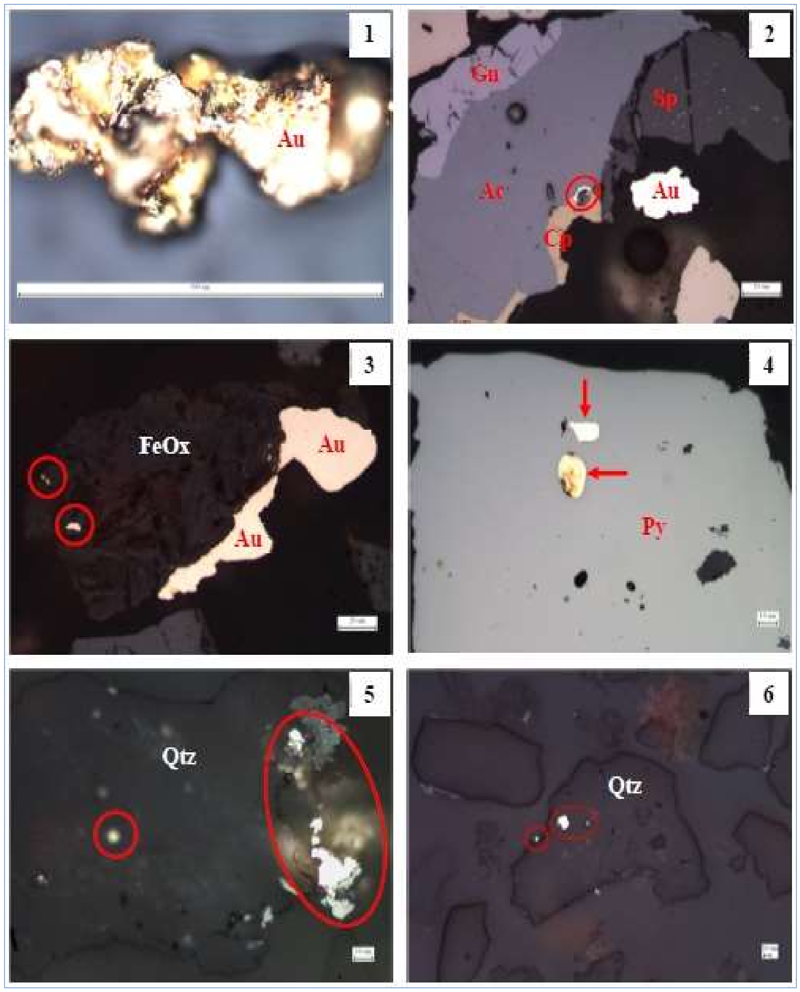


Figure 13-2 Photomicrographs showing the deportment of gold in the Balabag head samples (1) Liberated electrum (2) Liberated electrum (Au) and electrum (inside red circle) locked in between Acanthite (Ac) and chalcopyrite (Cp) (3) Electrum attached to and locked in iron oxide (FeOx) (4) Electrum (indicated by red arrows) locked in pyrite (Py) (5) Electrum (inside red circles) attached to and locked in quartz (6) Electrum (inside red circles) attached to and locked in quartz.

From the mineralogical report, bulk mineralogy is mainly composed of non-opaque minerals (>95%), with minor amount of sulphides mainly pyrite except for one head sample which is mainly composed of non-opaque minerals with minor amounts of iron oxide (mainly goethite) and trace amount of sulfate (jarosite). Gold minerals identified in the five samples include, in order of decreasing abundance, electrum, kustelite, native gold and utenbogaardtite, with high-Ag electrum being the most important gold mineral. In addition to electrum and kustelite, silver minerals include acanthite, native silver, mckinstryite, jalpaite, stephanite, pyrargyrite and polybasite. Gold minerals are closely associated with silver minerals, particularly acanthite. The concentration of gold in the five samples ranged from 2.00 g/t to 3.76 g/t.

Table 13-1 Gold Distribution in Five Balabag Samples

Sample ID	Sample ID		Gold Distribution (%)	
BAL-10-108	As-received		100.0	
	Float		68.1	Attached 20.4 Locked 47.7
	Sink	SP Tail	12.7	Liberated 30.5 Attached 1.0 Locked 0.4
		SP Middling	19.2*	
		SP Sulphide		
		SP Tip		
BAL-10-129	As-received		100.0	
	Float		44.1	Attached 14.6 Locked 29.5
	Sink	SP Tail	20.0	Liberated 54.9 Attached 0.2 Locked 0.8
		SP Middling	10.0	
		SP Sulphide		
		SP Tip	25.9*	
BAL-10-103	As-received		100.0	
	Float		54.5	Attached 18.1 Locked 36.3
	Sink	SP Tail	15.7	Liberated 42.4 Attached 0.8 Locked 2.3
		SP Middling		
		SP Sulphide	13.5	
		SP Tip	16.3*	
BAL-10-102	As-received		100.0	
	Float		39.2	Attached 3.1 Locked 36.1
	Sink	SP Tail	10.4	Liberated 53.9 Attached 5.0 Locked 1.9
		SP Middling		
		SP Sulphide	22.3	
		SP Tip	28.1*	
BAL-10-106	As-received		100.0	
	Float		66.7	Attached 49.4 Locked 17.3
	Sink	SP Tail	13.1	Liberated 32.7 Attached 0.1 Locked 0.4
		SP Middling	1.8	
		SP Sulphide	1.8	
		SP Tip	16.6*	

Only trace amount of As/Sb-bearing minerals (arsenopyrite, stephanite, and polybasite) were observed. No other minerals detrimental to processing were identified. Sulphide mineralogy of the five head samples is characterized by large grain size and euhedral crystal form of pyrite. This may indicate that pyrite (and the ore) was formed under such a mineralization environment offering sufficient crystallization time and space. Mineralogically, the oxidized ore in one of the head samples is distinguished by the presence of iron oxide with only trace amount of pyrite. Visually, oxidized ore is brownish in color while fresh ores look more grayish.

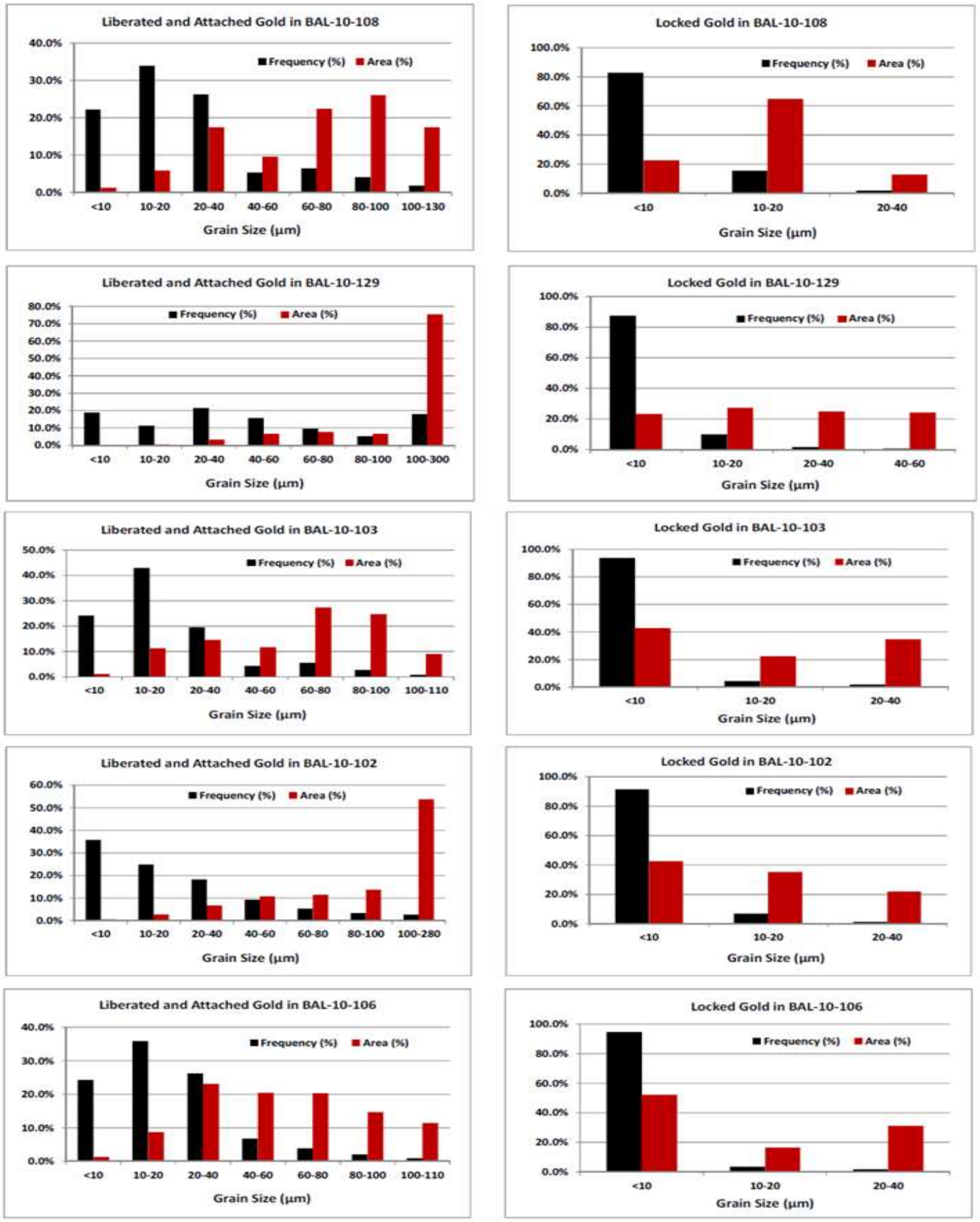


Figure 13-3 Grain Size Distribution of Gold in Balabag Samples

Important findings from the gold deportment study are presented in Table 13-1 above. Liberated (free) gold from these five samples is in the average of 39% at a 150 μm grind (P_{80}). In addition, at the current grinding fineness, gold minerals mainly occur as liberated particles in the sink fraction (oxides and sulfides) with only about 3% either attached or locked. This indicates that gold in the sulfides and oxides are easily liberated or exposed by grinding and can be recovered by gravity concentration.

In the float fraction (silicates), gold occurs as locked particles in and attached particles to non-opaque minerals. Attached gold can be recovered by direct cyanidation. To recover the locked gold (mainly in coarse non-opaque minerals), finer grinding would be required.

In terms of the grain size, gold minerals are fine to coarse with a size range from less than 1 µm to 297 µm. From the ensuing graphs, it can be seen that most of the locked gold in quartz is below 20 µm with the highest frequency below 10 µm. However, SEM analyses indicate that at (P_{80}) 150 µm, most of the quartz encapsulating the locked gold is typically greater than 200 µm. This suggests that further exposure or liberation can still be achieved at finer grinding since some has already been liberated or exposed even at 150 µm.

13.2 Metallurgical Test Program and Procedures

13.2.1 Initial Scoping Study (GENIVAR)

The metallurgical test program carried out by SGS Lakefield examined the response of the Balabag ore composite to heap leach simulation testing, gravity separation, and gravity tailing cyanide leaching as well as a preliminary thickening test.

13.2.1.1 Sample Preparation

The quarter sections were crushed to 1 inch and blended. Then 60 kg were riffled out, crushed to –¼ inch and blended, and the remainder was stored for possible future use. Six (6) 1-kg charges were riffled out of the 60-kg, -6 mm (¼ inch) material, and the remainder was crushed to -6 mesh. From this, a sufficient amount of material for a bond ball mill grindability test was riffled out, and the remainder was crushed to -10 mesh. Samples for head analysis and 1 and 10 kg charges for test work were then riffled out.

13.2.1.2 Gravity Separation

Two (2) gravity separation tests were conducted on 10 kg charges of 89 µm and 132 µm material, respectively. The tests were performed using a Knelson MD-3 concentrator. The Knelson concentrate was then upgraded further using a Mozley mineral separator.

13.2.1.3 Gravity Tailings Cyanidation

SGS used the coarser gravity tailings from previous test as feed for the cyanidation tests. The effect of regrind size (100, 75 and 50 µm) on gold and silver extractions was examined at a sodium cyanide (NaCN) dosage of 1 g/L, at 40% solids, and a pH of 10.5-11.0. After selecting the optimum grind size, a subsequent test was performed at that size with a NaCN dosage of 0.5 g/L.

13.2.1.4 Whole Ore Cyanidation

Two (2) whole ore heap leach bottle roll simulations were conducted, one with -6 mm (¼ inch) material and one with -10 mesh material. The bottles were rolled for one minute every hour for three weeks. Tests were conducted at a NaCN dosage of 1 g/L, at 40% solids, and at a pH of 10.5-11.0.

13.2.1.5 Settling Test

A settling test was conducted on the tailings of the final cyanidation test (CN 7). The initial %solids was set at 30% as can be expected at the grinding circuit cyclone overflow, and Magnafloc 351 at a flocculant dosage of 30 g/t was used.

13.2.2 Scoping Phase

The initial scoping study was conducted by GENIVAR and led to the 2011 gold scoping study at TVI's metallurgical laboratory in Canatuan Mines. With a limited sample available, a 20 kg composite sample was used for the flotation amenability and cyanidation amenability tests. Due to the unavailability of a gravity concentration laboratory equipment at the time of the study, no gravity test for assessment of free gold was conducted.

13.2.2.1 Sample Preparation

A master composite sample used for the scoping study was prepared out of the available core samples which were at hand. Based on the head grades and available amount of each component sample, a 20 kg master composite was prepared to produce the ore blend with the target head grade. Component samples were crushed and screened to -10 mesh prior to blending into a master composite sample.

13.2.2.2 Flotation Amenability Test

Two (2) sets of tests were conducted to evaluate the amenability of employing flotation as a pre-concentration step prior to cyanide leaching. For the first set of tests, three (3) varying grind sizes were tested (133, 119 and 100 µm) using a collector combination (PAX-Aerophine 3418) and frother at natural pulp pH.

For the second set of tests, the introduction of activators and regulators were evaluated to assess if improvement in recoveries can be attained. Straight 12-min rougher flotation tests on whole ore were performed for both sets of tests. If over 90% gold recovery can be achieved at a reasonable yield to the concentrate, then flotation can be deemed viable.

13.2.2.3 Cyanidation Amenability Tests

Straight cyanidation tests were carried out for scoping purposes. Three (3) bottle roll tests at varying grind sizes (120, 79 and 45 µm) were tested without any pre-treatments and two (2) CIL tests at varying conditions were considered. Grinding size range for the cyanidation tests was designed with reference to the recommendations from mineralogy study, wherein finer than (P_{80}) 150 µm was recommended to further liberate locked gold from quartz. The finest grind size (45 µm) was used for the CIL tests in order to evaluate any difficulty which may arise during carbon adsorption of gold in pregnant solution. Pulp density was set to 33% solids, cyanide strength was maintained at 0.1% NaCN and pulp pH was kept between pH 10.5 – 11.0. A 48-h total leach period in a bottle roller was considered.

13.2.3 Prefeasibility Phase

A comprehensive metallurgical test program was conducted at Canatuan Mines metallurgical laboratory which aimed to supplement the results of the initial scoping studies and to determine the most suitable and efficient process to extract the gold and silver from Balabag ore. The data obtained from the test works were used as design bases to develop a single processing plant that could treat all range of ore grade available in the mining concession that is flexible, economically viable, and requiring a minimum number of inline processing equipment.

13.2.3.1 Sample Preparation

One (1) ton of new ore samples were delivered to the metallurgical laboratory for this metallurgical test campaign. Standard procedure for ore preparation was carried out which included crushing, thorough mixing, and riffling before individual head samples from each component were taken and submitted to the assaying laboratory for elemental analysis. Based on the head grades and available amount of each component sample, two master composites having different silver head grades were prepared.

13.2.3.2 Gravity Concentration

Gravity concentration amenability tests were conducted on 10 kg ore using shaking table for first pass, then upgrading the shaking table concentrate using a laboratory scale Knelson concentrator. All concentration products were sent to analytical laboratory for gold and silver analyses.

13.2.3.3 Cyanidation and Adsorption Tests

As presented in the mineralogical study, which noted the locking of gold in coarse-grained non-opaque minerals, all laboratory cyanidation and adsorption tests were performed at (P_{80}) 75 µm. Included in the test campaign were cyanidation tests which aimed to:

1. Investigate the optimum NaCN strength and activated carbon dosage,
2. Determine the effect of using the oxidizing agent $Pb(NO_3)_2$,
3. Determine the optimum leach and adsorption time, and
4. Investigate carbon loading to estimate the silver cut-off grade

13.2.3.4 Flotation Tests

Exploratory flotation tests were undertaken to select the suitable collector combination using sulfide flotation parameters. Tests at varying collector combination were conducted using Aerophine 3418 as the first collector and either SIPX or PAX as co-collector. Different ore sample blends (416, 251 and 182 g/t Ag) were used as test samples to see how the flotation process will function under a wide range of head grades and increasing flotation time.

13.2.3.5 Detoxification Tests

Continuous cyanide destruction tests at two different pulp densities (40% vs 60%) and pulp pH (pH 9.0 vs 9.7) were conducted using accumulated cyanide leaching tails as feed. Sodium metabisulfide (SMBS) at 5,000 g/t dosage and copper sulfate ($CuSO_4$) at 800 g/t were added for the detoxification tests which were carried out below 26.7 ℃, the evaporation temperature of CN^-, and dissolved oxygen maintained above 3 ppm. Air was sparged into the test vessel to provide the oxygen requirement for CN destruction. As required by the DENR, the free CN limit in the final tailings should be no more than 0.2 mg/L.

13.2.3.6 Pilot Test Campaign

In January 2013, the construction of a crushing-grinding-CIP Balabag pilot plant in Canatuan Mines commenced, and on May of the same year the 1 t/d cyanidation pilot plant started operation. A continuous pilot test campaign was carried out for six weeks using a new batch of ore feed from the Balabag site with the main goal of confirming the bench-scale test results. Pilot runs were conducted at different grind sizes (75 µm vs 45 µm), cyanide strengths (700 ppm to 850 ppm), and oxidizing agent dosages (200, 300 and 500 g/t). Leach and adsorption times were also varied from 24 h up to 76 h. The detoxification circuit, consisting of a single air-sparged agitated tank, was used to treat the cyanidation tails stream. A total of 22 different sampling points were taken at specified time intervals in order to monitor the process parameters and ensure that the production targets are met.

When the CIL pilot tests were then conducted after the CIP tests, the leach tanks were converted to CIL tanks. All test parameters remain unchanged except for the pulp pH which was maintained between pH 10.2 and 10.8 during CIL operation.

Table 13-2 CIP Pilot Plant Test Parameters

Parameters	Targets	
Mesh of Grind (P80), µm	Feed/Static Settler	<75
% Solids	Leach Tanks 1-3	45
pH	Leach Tanks 1-3	10.5-11.0
	CIP Tanks 1-2	9.8-10.3
	Detox Tank	9.0

Parameters	Targets	
DO, ppm	Leach Tanks 1-3	>5
	CIP Tanks 1-2	>5
	Detox Tank	>4
Cyanide Strength, ppm	Leach Tanks 1-3	>700
	CIP Tanks 1-2	<300
Reagents, g/t	NaCN	1500-2000
	$PbNO_3$	300

13.2.3.7 Flotation Variability and Cyanidation Optimization Test Works

Different ore blends at varying head grades were tested to assess the versatility of the flotation and cyanidation processes and continue the optimization tests. The individual ore samples used for blending were sampled from the same areas where the ore feed from previous metallurgical tests were taken.

The flotation variability tests aim to predict the response to flotation of different ore blends with different gold and silver content and varying grind sizes. Other flotation parameters such as collector type and dosage, flotation pH, and flotation time were kept constant.

All cyanide leaching test feeds were produced from prior flotation process. Separate leaching tests, using derived optimum parameters and reagents dosages, were carried out for the accumulated flotation concentrates and flotation tails at two varying grind size (45 µm vs 75 µm). The rougher concentrates were subjected to leaching while the rougher tails were tested for simultaneous leaching and adsorption (CIL).

13.3 Metallurgical Test Results and Determination of Recoveries

13.3.1 Initial Scoping Study (GENIVAR)

13.3.1.1 Ore Characterization

A pulp and metallics assay of the composite sample was made, which showed gold values of 3.43 g/t and silver values of 83.6 g/t. Of this amount, 18.2% of the gold and 3.7% of the silver reported to the +150 mesh fraction. The composite also contained 0.1% S and 0.07% C. The ore specific gravity was also measured which resulted in a value of 2.67 g/cm^3.

13.3.1.2 Grindability Tests

Bond ball mill grindability test performed by SGS Lakefield resulted in a Bond Work Index (Bwi) of 14.7 kWh/t.

13.3.1.3 Gravity Separation

From the gravity test results, a significant amount of the precious metal values, approximately 30% Au and 4% Ag, can be recovered in the concentrate.

13.3.1.4 Gravity Tailings Cyanidation

Test results show that gold and silver extractions and kinetics are only minutely higher at the finer regrind sizes. In addition, the coarser grind resulted in significantly lower cyanide consumption. The 100 µm regrind was thus considered the most optimal and a subsequent test was performed at this grind with a NaCN dosage of 0.5 g/L. This test did not produce results as good as the same test done with a NaCN concentration of 1 g/L.

Summarized in
Table 13-3 below are the results of all tests conducted at a NaCN concentration of 1 g/L. 72-h extractions by cyanidation varied from 94% to 95% for gold and from 81% to 83% for silver. For the test performed at 0.5 g/L NaCN, 72-h gold and silver extractions were 89% and 59%, respectively.

Table 13-3 Tails Cyanidation Tests at Varying Grind Size

Test #	Grind Size, µm	Extraction, %				Residue assays, g/t	
		48 h		72 h			
		Au	Ag	Au	Ag	Au	Ag
CN 4	98	92	77	94	81	0.12	15.7
CN 5	73	93	79	95	83	0.11	14.2
CN 6	59	93	78	95	83	0.10	14.1
CN 7	101	85	54	89	59	0.25	32.2

13.3.1.5 Whole Ore Cyanidation

Test results are summarized in Table 13-4, and from the table, the -10 mesh test especially gave promising results. These show that after 21 d the leach was probably still ongoing, so if a longer test period were used the results may have been even better. If heap leach is to be investigated further, longer heap leach bottle roll simulations should first be conducted followed with column heap leach testing.

Table 13-4 Heap Leach Bottle Roll Simulation Tests

Test #	Feed Size	21 d Extraction, %		Residue assays, g/t	
		Au	Ag	Au	Ag
CN 1	-1/4 inch	81	54	0.59	37.7
CN 2	-10 mesh	93	73	0.36	24.0

13.3.1.6 Settling Test

The settling test led to an achievable underflow density of 60% solids with a corresponding settling rate of 0.46 m^2/t/d. In the mill, a thickener underflow of 50% solids should be achieved. A settling rate of 0.15 m^2/t/d was extrapolated from the results for this density.

The Balabag ore composite tested responded well to the coarse bottle roll leaching despite the limited testing. The high silver grade dictates the NaCN dosage be somewhat higher at 1 g/L than dosages found in typical gold leaching programs of 0.5 g/L NaCN. Leaching period of three weeks appeared to be insufficient. It is quite common for high silver bearing ores to require not only higher NaCN dosages but also longer leach times in order to maximize metal extractions. If further test work is to be planned, it would be recommended that this parameter be explored with longer coarse bottle rolls or perhaps some column heap leaches. After 21 d of agitation the recovery of gold and silver, for the -1/4 inch material, were 81% and 54%, respectively. For the -10 mesh material, the recovery of gold and silver were 93% and 73%, respectively.

The gravity work yielded reasonable gold recoveries of 30% and 35% while the silver recoveries were 4% and 5%. The subsequent cyanidation leaching testwork done at 1 g/L NaCN on the gravity tails yielded high recoveries for both gold and silver at all three particle sizes tested. The recovery of gold averaged at 96% to 97% leaving a residue assaying 0.10 to 0.12 g/t Au; the recovery of silver averaged at 92% to 94% leaving a residue assaying 14 to 15 g/t Ag. NaCN consumption ranged from 0.3 - 0.7 kg/t, depending on the fineness of grind while the lime consumption averaged at 1.7-1.9 kg/t. It was seen on an additional leach test performed with 0.5 g/L NaCN that the recoveries were significantly lower for the tested retention time of 72 h and thus confirmed that higher NaCN dosages will be required for the high silver bearing Balabag ore composite.

13.3.2 Scoping Phase

13.3.2.1 Head Analysis

Head analysis of the blended master composite sample used for the scoping study is shown in Table 13-5. This contains a higher silver grade compared to the feed used in the GENIVAR study.

Table 13-5 Master Composite Sample Used for the Scoping Study

Label	Au	Ag	As	Bi	Cd	Cu	Fe	Hg	Pb	S	Zn
	g/t	g/t	%	ppm	ppm	%	%	ppm	%	%	%
BL-Blend	3.7	143	0.006	<10	15.4	0.03	2.9	2.4	0.05	0.5	0.06

13.3.2.2 Flotation Amenability

Summary of the flotation tests at varying grind size and varying activators are presented in the following tables.

Table 13-6 Summary of Test Results at Varying Grind Size

Test #	P_{80}, µm	Concentrate Grade, g/t		% Recovery	
		Au	Ag	Au	Ag
0504401	133	33.0	1207.5	74.2	72.7
0504402	119	35.2	1218.7	76.9	75.0
0504403	100	39.2	1300.7	77.1	81.7

Table 13-7 Summary of Test Results Using Varying Activators

Test #	Activator	Concentrate Grade, g/t		% Recovery	
		Au	Ag	Au	Ag
0505401	$CuSO_4$	34.5	1315.4	86.1	81.4
0505402	H_2SO_4	32.9	1385.9	85.4	83.1
0505403	NaHS	35.9	1582.0	80.5	79.9

**Tests were conducted at (P_{80}) 100 µm*

From the outcome of the tests (Table 13-6), it can be observed that there was an increasing trend in gold recovery with finer mesh of grind but only to a marginal extent. The effect of finer grinding is more pronounced on silver recovery with 9% increase from 133 µm to 100 µm. This supports mineralogical study findings that there is high frequency and area of precious metals occurring in the fines. As recommended in the study, finer grinding is required to expose or liberate the gold.

From Table 13-7, there was an observed improvement in precious metals recoveries, particularly for gold, when copper sulfate ($CuSO_4$) and sulfuric acid (H_2SO_4) were added separately. The increase in acidity of the pulp brought by the said activators encouraged sulfides (pyrite) flotation and in turn the recovery of gold particles locked in it.

Greater than 90% precious metals recovery was aimed for; however, only 86% was attained at the best condition tested. Insufficient recovery of gold indicates that not all of the gold is confined to the sulfides as mineralogical study suggests. Decrease in pH to 5.5 with H_2SO_4 as regulator increased the gold recovery by about 8%.

The yield to the rougher concentrate, which is about 8%, is relatively acceptable which can be exploited in terms of lesser feed to the cyanidation circuit. This translates to lower capital cost for a cyanidation plant which will require only enough size of equipment to handle the volume of flotation concentrate. Lower overall cost for cyanide destruction can also be an added benefit.

13.3.2.3 Straight Cyanidation and Adsorption Tests

Summary of the cyanidation tests results are presented in the tables below.

Table 13-8 Summary of Cyanidation Tests at Varying Grind Size

Test Code	P_{80} (µm)	Reagents (kg/t)		%Extraction	
		Lime	NaCN	Au	Ag
T1B	120	4.80	1.47	94.77	88.84
T2B	79	4.80	1.25	94.16	93.08
T3A	45	4.40	2.75	97.08	95.52

Table 13-9 Summary of Carbon Adsorption Tests

Test Code	Activated Carbon (kg/t)	Reagents (kg/t)		%Recovery	
		Lime	NaCN	Au	Ag
T4	60	4.40	2.75	97.71	92.75
F5	100	6.00	3.21	96.06	96.88

Increasing trend in leaching kinetics for both silver and gold was observed with finer grinding. The effect of finer grinding to silver recovery is more pronounced which is in agreement to findings from flotation tests. The positive effect to recovery of finer grinding, however, is negated by higher consumption of NaCN. At the finest grind of 45 µm, 97% gold and 95% silver recoveries were attained but at 2.75 kg/t of NaCN (though free CN was maintained high at 0.1%)

From the plots of leaching kinetics in Figure 13-4, leaching rate at the onset of cyanidation is relatively slow due primarily to observed low levels of dissolved oxygen (DO) in the pulp which is well below the 5 ppm minimum requirement. A variety of minerals will remove oxygen from a leach solution including arsenic bearing minerals and reactive sulfides such as pyrrhotite. DO level was observed to be lower at finer grind.

It can also be observed that the leaching rate of silver is relatively slower than gold. This is expected to be the case since silver exists predominantly as silver sulfides which leach even more slowly than metallic silver. From mineralogical data, silver occurs predominantly as acanthite (Ag_2S) with traces of other sulfides as stephanite (Ag_5SbS_4) and pyrargyrite (Ag_3SbS_3). Moreover, oxidation products of sulfide minerals could precipitate dissolved silver.

For CIL Test T4, at 60 kg/t carbon, silver recovery was lower than gold but when carbon dosage was increased to 100 kg/t (Test F5) silver recovery improved by 4%. Gold and silver grade at the final tailings were 0.13 g/t and 3.93 g/t, respectively.

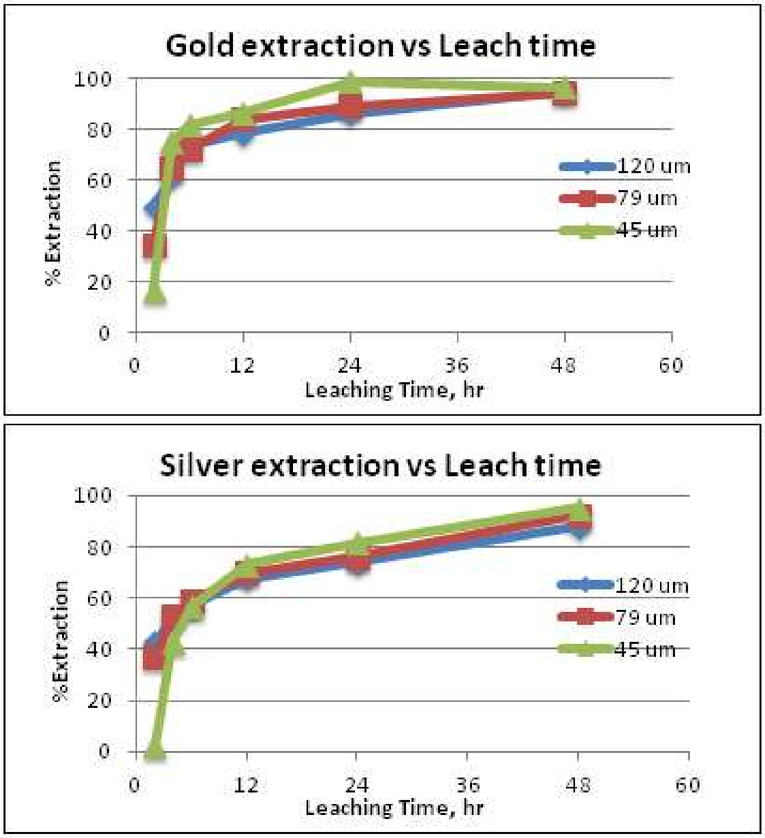


Figure 13-4 Gold (above) and Silver (below) kinetics at Varying Grind Sizes

At this stage of scoping study, it was proven that acceptable recovery can be achieved using straight cyanidation alone with 96% Au and 96% Ag recoveries. However, the relatively long retention time of 48 h to maximize precious metal recoveries and the low DO levels at the onset of leaching need to be worked out. Cyanide dosage also needs to be optimized. Out of scope studies, which include pre-aeration prior to leaching and the use of $Pb(NO_3)_2$ as catalyst, need to be taken into consideration. Moreover, gravity pre-concentration is expected to reduce the leaching time due to the high free gold content based on mineralogical data.

The benefits of employing flotation as pre-concentration stage for subsequent cyanidation of flotation concentrate can neither be argued nor ruled out of consideration as yet due to the relatively low recovery in the flotation amenability tests with maximum recovery of just 86% Au and 81% Ag.

Follow up tests need to be conducted before the pre-final flowsheet can be decided on, such as:

1. Conduct gravity concentration tests to determine gravity-recoverable gold
2. From the mineralogy, flotation of gold attached to the quartz is the greatest challenge. Selective flotation is necessary to isolate these gold bearing materials from bulk of the quartz gangue. Exploration of more selective gold collectors is necessary.
3. Explore the effect of finer grinding
4. Explore the effect of using lead nitrate, $Pb(NO_3)_2$, for leaching catalytic purposes as well as pre-aeration to improve DO levels and thus leaching kinetics.

13.3.3 Prefeasibility Phase

13.3.3.1 Head Analysis

From the outcome of the scoping study, several bench scale tests were planned all geared towards the development of a suitable process flowsheet. Shown in the table below were the head grades of the two ore blends that were used for the metallurgical test works. The first ore blend was the “milling grade” ore with nominal gold and silver assays while the second ore blend was the “ultra-high grade silver ore” which was prepared in anticipation of the elevated silver grade in the ore.

Table 13-10 Head Grade Used for the Metallurgical Test Campaigns

Ore Type	Head Assay, g/t	
	Au	Ag
Milling Grade	2.09	68.2
Ultra-high Grade Silver Ore	> 2.09	68.2 – 300.0

13.3.3.2 Comminution Tests

Using a total of 80 kg of Balabag ore feed with sizes close to 25 mm (1”), the Crushing Work Index (Cwi) was obtained with the use of a Boyd jaw crusher. The results obtained from five (5) trials gave an average Cwi of 9.5 kWh/t.

13.3.3.3 Gravity Concentration

From the results of the gravity concentration tests, at 150 μm grind size, the gravity recoverable gold ranged between 15-30%.

13.3.3.4 Straight Cyanidation and Adsorption Tests

Results of the initial leaching and adsorption tests (
Table 13-11) using the milling grade ore showed a poor extraction or leaching efficiency in the use of 0.125 %NaCN strength in cyanidation. In the succeeding cyanidation tests, 0.07 %NaCN was utilized as this concentration best conforms to a [6]:[1] ratio of NaCN:O_2, when the bottle roll test is performed at atmospheric pressure without aeration. The 48 h leach time plus the 24 h adsorption time indicated sufficient time for the comparison and sensitivity test works to proceed.

Table 13-11 Leaching (0.125 %CN) and Adsorption Tests at Varying Carbon Dose

Carbon Dosage, g/kg_{ore}	Test #					
	46 (50 kg/t), % Recovery		47 (100 kg/t), % Recovery		48 (150 kg/t), % Recovery	
	Au	Ag	Au	Ag	Au	Ag
Barren Solution	2.71	3.13	1.45	2.08	1.44	1.24
Tails	4.60	2.30	4.46	2.14	4.42	2.39
Carbon	92.69	94.56	94.09	95.78	94.14	96.37

Table 13-12 Results of Carbon Loading Tests

Test #	Carbon Dosage, kg/t ore	Head, g/t		Carbon load, g/t		Tails, g/t		Overall Recovery, %	
		Au	Ag	Au	Ag	Au	Ag	Au	Ag
46	50	1.80	65.1	33.5	1232	0.08	1.50	92.7	94.6
47	100	1.68	70.5	18.6	675	0.08	1.51	94.1	95.8
48	150	1.70	69.2	10.7	444	0.08	1.66	94.1	96.4

From Table 13-12, comparable high adsorption efficiencies were attained at 50, 100 and 150 kg/t carbon dosages. 50 kg/t was chosen as basis for calculating the theoretical cut-off grade for silver which is the limit of the feed grade to the leaching/adsorption circuit to achieve optimum recovery. The theoretical cut-off grade is also determined by the degree of carbon loading in the adsorption phase of the process.

Using 1,232 g/t Ag carbon loading (Test# 46) the estimated weekly silver metal recovery was 123,200 g or 17,600 g/d Ag. At 94% Ag overall recovery and 2,000 t/d plant capacity, the theoretical silver cut-off grade is 9.4 g/t. Evidently, the designed milling grade of 68 g/t Ag is very much higher than the calculated cut-off grade.

Adding an oxidizing agent, 300 g/t lead nitrate ($PbNO_3$), in the conditioning step prior to cyanidation, improved the leaching kinetics of the precious metals particularly silver extraction which increased by 4%, and overall silver recovery increased by 6% as shown in Table 13-13. Thus, the high silver Balabag ore is amenable to the straight cyanidation procedure with a few modifications in the conditioning stage and test parameters.

Table 13-13 Results of Cyanidation Tests With and Without Conditioning with $PbNO_3$ (NaCN strength = 0.07%, P_{80} = 75 μm, Carbon dosage = 100 g/kg_{ore})

Test #	49		50	
Leach Time, h	**w/o $PbNO_3$, %Extraction**		**300 g/t $PbNO_3$, %Extraction**	
	Au	**Ag**	**Au**	**Ag**
2	36.6	18.7	66.3	27.9
4	67.1	28.2	79.8	35.4
6	76.7	37.3	90.1	44.5
12	82.1	46.7	91.6	54.6
24	90.0	56.8	93.5	61.8
48	94.1	86.8	94.1	90.6
Adsorption Time, h	**w/o $PbNO_3$, %Overall Recovery**		**300 g/t $PbNO_3$, %Overall Recovery**	
	Au	**Ag**	**Au**	**Ag**
Pregnant Solution, ppm	**1.45**	**43.52**	**1.48**	**45.82**
Barren Solution	2.41	0.92	2.20	0.64
Tails	6.80	17.15	6.45	11.39
Carbon	90.79	81.93	91.36	87.97

Table 13-14 Results of Cyanidation Tests at Varying Leaching and Adsorption Time (NaCN strength = 0.07%, P_{80} = 75 μm, Carbon dosage = 100 g/kg_{ore})

Leach Time, H	**% Extraction**					
	Test # 51		**Test # 52**		**Test # 53**	
	Au	**Ag**	**Au**	**Ag**	**Au**	**Ag**
2	37.5	23.3	37.3	23.4	37.3	23.8
4	50.9	33.3	50.1	33.1	50.5	33.1
6	71.5	54.8	71.1	54.8	71.8	54.5
12	75.6	59.9	73.9	59.7	74.0	59.4
24	87.2	70.3	87.1	70.9	87.4	71.4
48	93.7	90.0	93.3	90.5	93.7	90.9
56	-	-	95.0	93.0	95.2	92.6
64	-	-	-	-	96.1	95.2
Adsorption Time	**% Overall Recovery**					
	24 h		**16 h**		**8 h**	
	Au	**Ag**	**Au**	**Ag**	**Au**	**Ag**

Leach Time, H	% Extraction					
	Test # 51		Test # 52		Test # 53	
	Au	Ag	Au	Ag	Au	Ag
Pregnant Solution, ppm	**1.39**	**44.91**	**1.42**	**44.06**	**1.50**	**45.30**
Barren Soln	1.81	0.47	4.68	2.01	7.90	3.22
Tails	5.54	11.54	3.06	11.34	4.06	12.45
Carbon	92.65	87.99	92.26	86.64	88.04	84.33

From Table 13-14 the extension of the leach time from 48 h to 64 h exhibited an increase in the extraction of the gold and silver – more pronounced on the leaching of the silver minerals. This result indicates that the silver has a refractory character of "slow rate of dissolution". On the other hand, the response of the gold showed that it is a "free milling" type and appear dominantly as electrum but having much faster rate of leaching than the silver. Carbon adsorption performance deteriorated as the adsorption time decreased. The optimum contact time was estimated at 72 h which conforms to industry practice in the leaching of refractory gold and/or silver ores.

13.3.3.5 Flotation Tests

A pre-treatment process ahead of cyanidation, either gravity concentration or flotation, is a logical remedy to recover the bulk of the precious metals. Based on the report by SGS Lakefield, overall recovery by gravity concentration resulted to 30-35% Au and 4-5% Ag. This pretreatment process is essential in gold-silver processing, particularly in recovering liberated gold particles that tend to be recirculated at cyclone underflow due to its high specific gravity in nature but the recoveries, especially silver recovery, were poor.

Hence, exploratory flotation tests were undertaken to search for a suitable co-collector and frother combination using sulfide flotation parameters. The flotation test results indicated that an Aerophine 3418-PAX collector combination in a 4:1 ratio and alkaline pH gave satisfactory recovery of at least 80% for the gold and silver as shown in Table 13-15.

The calculated flotation recovery from variable head grades with very high silver content gave a nominal recovery of 80% for gold and 73% for silver. However, the %yield of concentrates produced was high resulting to lower concentrate grade which suggests that application of one or two stages of cleaner flotation to the rougher concentrates might be needed.

Table 13-15 Flotation Tests at Varying Collector Combination

Test#	Co-collector of A3418	pH	Assay, g/t				Recovery, %	
			Concentrate		Tails			
			Au	Ag	Au	Ag	Au	Ag
1021101	SIPX	7	20.7	673	0.46	18.6	79	75
1021102	PAX	7	21.1	684	0.45	17.0	79	77
1021103	SIPX	11	22.7	710	0.47	18.9	79	75
1021104	PAX	11	18.5	582	0.38	15.1	83	80
1101701	SIPX	11	16.1	428	0.48	16.8	80	76
1101702	PAX	11	16.8	461	0.49	17.2	79	75

Table 13-16 Flotation Tests at Varying Feed Grade with Very High Ag Content

Test#	Yield, %	Assay, g/t						Recovery, %	
		Head		Concen-trate		Tails			
		Au	Ag	Au	Ag	Au	Ag	Au	Ag
1207101	18	12.7	416	60.3	1875	2.3	96.6	85	81
1209202	15	4.5	252	24.4	1232	1.0	79.8	81	73
1209201	22	3.9	183	14.1	612	1.0	63.1	80	73

13.3.3.6 Detoxification Tests

Detoxification tests were also conducted in bench-scale and shown in the table below are the results of the continuous tests at varying pulp density and target pulp pH. In this process, the toxic cyanide species, free CN and weak acid dissociable (WAD) CN complexes, are oxidized by sulfur and oxygen into cyanate ions which is a thousandth times less toxic. The governing chemical reaction is expressed in the following equation:

$$CN^- + SO_2 + O_2 + H_2O \qquad CNO^- + H_2SO_4 \xrightarrow[CuSO_4]{}$$

Efficient detoxification was achieved for all conditions having CN_{free} levels way below the DENR limit of 0.2 mg/L. Pulp pH 9.7 will be aimed for which is also the reported pH in cyanide destruction practices for gold ores (non-green gold) with moderate amount of metal sulfides hosted by quartz diorite worldwide.

Table 13-17 Continuous Detoxification Tests Using CIP Tails

Test #	Pulp density, %	pH	Composition, mg/L				
			CN_{free}	CN_{WAD}	Cu	Fe	Zn
(Feed)	45	10.1	1.80	2.40	0.84	20.0	1.30
1	45	9.0	0.05	0.13	0.23	2.0	0.28
2	45	9.7	0.01	0.14	0.22	2.0	0.61
3	65	9.0	0.06	0.09	0.47	3.1	0.64
4	65	9.7	0.06	0.11	0.88	2.1	0.39

Based on the results of the metallurgical test campaign the following conclusions were formulated:

1. A pretreatment process utilizing the flotation procedure, consisting of a rougher and two-stage cleaner, is a requisite step in the recovery of gold and silver from the Balabag ore. The flotation concentrate will be leached, while the precious metals (Au and Ag) in the pregnant solution will be recovered, either, by direct powder electrowinning with automatic washing or by the Merrill-Crowe zinc dust precipitation. The treatment process will be in a distinct cyanidation circuit. The flotation concentrates leached solid residue and the flotation tailings streams will undergo the cyanidation – carbon adsorption – pressure Zadra process. The carbon adsorption capacity is limited by the capacity of the ADR (adsorption, desorption and refining) Plant.
2. The theoretical cut-off grade of 9.4 g/t Ag for the cyanidation – carbon adsorption process measures only to 14% of the designed milling grade of the Balabag gold-silver ore based on recorded carbon loading and ADR Plant capacity.
3. The degree of "refractoriness" of the high silver ore was exhibited in its slow rate of leaching at 72 h and the occlusion of a fraction of the silver in sulfides as exhibited in the oxidation conditioning using lead nitrate.
4. Efficient destruction of cyanide in the final tailings will be conducted in two detoxification tanks using air, SMBS and $CuSO_4$ at pH 9.7 and pulp density of 45% solids. The final effluent meets the DENR requirement of cyanide concentration limit of 0.2 mg/L CN_{free}.

13.3.3.7 Pilot Test Campaign

Pilot plant feed grade ranged between 2.5 – 6.8 g/t Au and 55.0 – 270.0 g/t Ag. The important findings from pilot testing are summarized in Table 13-18 and
Table 13-19.

During the CIP pilot runs, it was observed that poor gold and silver recoveries were achieved even after 72 h, the optimum leaching period that was established from the bench scale tests. From the tables below, finer grind size (45 µm vs 75 µm) showed slight improvement in leaching kinetics at the onset of cyanidation until 48 h but had no significant effect in the overall recovery. It was the higher CN strength (850 ppm CN vs 700 ppm CN) which gave a positive effect in the precious metal recoveries wherein a 2% increase in gold recovery and 5% increase in silver recovery were realized. Adding a higher dose of lead nitrate did not seem to effect faster leaching kinetics for both gold and silver. However, an improvement in silver adsorption kinetics was observed as the lead nitrate dosage was increased. Combination of high CN strength and low lead nitrate dosage provided the best scheme for CIP process with overall recoveries of 92.8% and 88.8% for gold and silver, respectively.

Table 13-18 Summary of Pilot Plant Test Parameters and Gold Leaching and Adsoption Kinetics

						Gold Dissolution/Recovery, %				
Tests	**MOG, µm**	**Cyanide Strength mg/L**	**$PbNO_3$, g/t**	**Leach time, h**	**Ads time, h**	**24th h**	**48th h**	**72nd h**	**84th h**	**96th h**
CIP Initial Test	75	700	300	72	24	73.55	77.46	79.26	87.01	90.12
CIP Fine Grind	45	700	300	72	24	76.15	79.28	79.06	86.31	89.91
CIP High CN	75	750-850	300	72	24	73.24	77.39	83.84	87.66	92.80
CIP High CN & high $PbNO_3$	75	>750	500	72	24	75.79	74.68	75.79	78.37	90.38
CIP High CN & low $PbNO_3$	75	>750	200	72	24	80.22	86.21	82.35	89.78	89.05
CIL w/ $PbNO_3$	75	>750	200	48	48	86.32	93.18	-	-	-
CIL w/o $PbNO_3$	75	>750	-	48	72	88.48	93.79	94.65	-	-

Table 13-19 Summary of Pilot Plant Test Parameters and Silver Leaching and Adsoption Kinetics

						Silver Dissolution/Recovery, %				
Tests	**MOG, µm**	**Cyanide Strength mg/L**	**$PbNO_3$, g/t**	**Leach time, h**	**Ads time, h**	**24th h**	**48th h**	**72nd h**	**84th h**	**96th h**
CIP Initial Test	75	700	300	72	24	51.14	65.57	70.26	77.35	83.75
CIP Fine Grind	45	700	300	72	24	57.87	66.65	68.24	78.09	83.01
CIP High CN	75	750-850	300	72	24	59.56	71.70	81.18	78.89	88.81
CIP High CN & high $PbNO_3$	75	>750	500	72	24	47.50	64.51	74.74	74.52	89.49
CIP High CN & low $PbNO_3$	75	>750	200	72	24	60.06	75.11	80.86	75.12	80.81
CIL w/ $PbNO_3$	75	>750	200	48	48	78.06	89.69	-	-	-

Tests	MOG, µm	Cyanide Strength mg/L	$PbNO_3$, g/t	Leach time, h	Ads time, h	Silver Dissolution/Recovery, % 24th h	48th h	72nd h	84th h	96th h
CIL w/o $PbNO_3$	75	>750	-	48	72	76.14	87.61	89.66	-	-

The long contact time (total of 96 h) will entail several leaching and adsorption tanks which equates to high mill capital cost. A CIL pilot test campaign was initiated to see if the same precious metals recoveries can be realized at a shorter contact time. From Table 13-18 and Table 13-19, gold recovery and silver recovery of 93.2% and 89.7%, respectively, were already achieved after 48 h. Further extending the contact time to 72 h or adding lead nitrate during CIL showed very minimal to no improvement in gold and silver recoveries.

13.3.3.8 Variability Test Works

13.3.3.8.1 Head Grade Analysis

Table 13-20 below shows the resulting head grade analyses for the different ore blends used for the variability and optimization tests. Gold and silver head grades ranged between 0.8-25.7 g/t and 8.5-1014.0 g/t, respectively.

Table 13-20 Test Ore Blend for Optimization Tests

Ore Blend	Head Grade, g/t Au	Head Grade, g/t Ag
5B-323	3.1	24.6
5B-386	3.1	86.5
5B-260	2.5	62.4
BL-39347	0.8	8.5
STP-37	2.2	16.2
BL-39357	1.1	19.4
BL-39309	1.5	19.8
STP-33	3.3	26.4
BL-39367	3.1	29.6
BL-39362	1.1	20.5
BL-39354	3.7	32.2
BL-39319	2.1	35.3
STP-31	10.3	38.0
BL-39350	6.1	42.2
BL-260953	5.0	42.4
STP-25	13.2	43.9
STP-32	2.4	62.0
STP-23	8.2	78.3
BL-39360	6.4	95.3
BL-39359	4.6	101.1
STP-27	2.3	114.5
STP-13	3.0	131.0
BL-39314	4.9	150.9

Ore Blend	Head Grade, g/t	
	Au	Ag
BL-39315	5.4	182.1
STP-11	6.7	284.6
STP-16	18.1	286.1
STP-14	25.7	1041.0

13.3.3.8.2 Flotation Variability Test

From the results of the flotation variability tests (
Table 13-21) using different feed grades and grind sizes, average flotation recoveries were high at 91.0% for gold and 87.2% for silver. Using 95% confidence level, the maximum gold and silver recoveries were 92.3% and 89.5%, respectively. Moreover, the minimum recoveries at the same confidence level were 89.6% and 84.9% for gold and silver, respectively.

Correlation analysis on gold showed that the grind size had a negative correlation with mass recovery and in effect with flotation recovery. Thus, finer grind size leads to higher gold flotation recovery. No strong correlation was observed between gold recovery and head grade.

On the other hand, correlation analysis on silver showed no significant correlation between silver flotation recovery and neither grind size nor head grade. Hence, flotation of silver may be dependent on other variables such as collector dosage, flotation pH, and flotation time.

Table 13-21 Flotation Variability Tests

	Flotation Recovery	
	Range, %	Average, %
Gold	82.8 – 97.9	91.0
Silver	69.2 – 95.9	87.2

13.3.3.8.3 Cyanide Leaching Optimization Tests

The flotation-cyanidation process was simulated at two grind sizes using optimum parameters obtained from a series of optimization tests. Summarized in

Table 13-22 are the results of the comparative tests. Higher flotation recoveries were obtained at finer grind of 51 μm compared to the previously established optimum grind size of 80 μm. After leaching the individual flotation concentrates, higher gold and silver extractions were achieved at finer grind size. Likewise, higher carbon recoveries were attained at finer grind size when flotation tailings were subjected to simultaneous leaching and carbon adsorption.

As a result, overall recoveries for finer grind size were also higher reaching gold recovery of 97% Au and silver recovery of 94% Ag. However, it is important to note that the resulting recovery values for 80 µm, 96% Au and 89% Ag, were almost at par to the pilot test recovery values.

Table 13-22 Comparison of Flotation-Cyanidation Tests at Varying Grind Size

Flotation

Ore feed, g/t		MOG (P_{80}), microns	Mass Rec, %	Flot Conc, g/t		Flot Tails, g/t		Flotation Rec, %	
Au	Ag			Au	Ag	Au	Ag	Au	Ag
3.4	91.6	51	26	12	323	0.5	10	89	92
3.6	92.4	80	13	23	596	0.8	21	80	81

Flotation Concentrate Leaching

MOG P_{80}, microns	Reagents, kg/t_{ore}		Pulp density, %sol	Leach time, h	PLS, g/m^3		Residue, g/t		Extraction, %		Stage Rec, %	
	NaCN	Lime			Au	Ag	Au	Ag	Au	Ag	Au	Ag
28	0.8	0.5	33	60	4	117	0.2	16	98	95	88	87
44	0.5	0.1	33	72	9	200	0.2	62	99	89	80	71

Flotation Tails Carbon-in-leach

MOG P_{80}, microns	Reagents, kg/t_{ore}		Pulp density, %sol	Ads time, h	Loaded Carbon, g/t		Residue, g/t		Carbon Rec, %		Stage Rec, %	
	NaCN	Lime			Au	Ag	Au	Ag	Au	Ag	Au	Ag
51	1.0	0.9	45	36	14	252	0.1	1.8	89	84	9	7
76	1.5	0.3	45	48	23	394	0.1	1.0	86	93	17	18

	Au	Ag
Overall Rec, %	97	94
	96	89

13.3.3.9 Determination of Recoveries

The projected recoveries were set to be varying depending on the head grade. It was decided that a constant tailings grade will be used which were derived from the results of the metallurgical test campaigns. Final gold tailings grade was set to 0.26 g/t Au based on pilot plant test works results which is also within the range of typical tailings grade of existing gold plants. From the results of all the optimization and variability tests using different silver head grades, the silver tailings grade was derived to be equal to 3.10 g/t Ag.

Using the above established tailings grades and the average mine plan head grades, 2.5 g/t Au and 68.25 g/t Ag, the projected recoveries were 87.9% Au and 88.3% Ag.

14 MINERAL RESOURCE ESTIMATES

The total undiluted mineral resources for the Balabag Gold-Silver Project are presented according to different cut-off grades in Table 14-1. The effective date of the mineral resource estimate is January 2014.

Sample protocols, including sample methodology, preparation, analyses and data verification have been conducted in accordance with industry standards using appropriate quality assurance/quality control procedures in all exploration activities of TVIRD under the direct supervision of the OIC-Exploration.

A review by Leo A. Sosa, Competent Person under the PMRC, of the procedures employed by the company in terms of field sampling, security in handling the samples, sample preparations and the QA/QC protocols concludes that all these are in accordance with sound industry best practices and is sufficient for use in performing the mineral resource estimation. Detailed QAQC analysis is presented in **Appendix IV-I**.

Table 14-1 Mineral Resource Estimate (TVIRD, January 2014)

CLASS	DOMAIN	CUT-OFF AuEq (g/t)	Tonnage DMT	Au g/t	Ag g/t	AuEq g/t	AuEq Oz
MEASURED	DOM 1	0.0	722,749	2.09	65.04	3.39	78,835
		0.1	718,533	2.10	65.42	3.41	78,828
		0.3	700,117	2.16	67.02	3.50	78,727
		0.5	675,170	2.23	69.25	3.61	78,418
		1.0	592,301	2.47	76.96	4.01	76,340
		5.0	165,710	4.56	145.34	7.46	39,760
	DOM 2	0.0	407,445	0.36	21.00	0.78	10,187
		0.1	401,983	0.36	21.26	0.79	10,171
		0.3	354,759	0.39	23.60	0.87	9,866
		0.5	198,963	0.53	35.79	1.24	7,964
		1.0	96,915	0.74	54.34	1.83	5,691
		5.0	3,564	4.46	131.80	7.10	813
INDICATED	DOM 1	0.0	1,152,074	2.17	47.92	3.13	115,751
		0.1	1,135,173	2.20	48.61	3.17	115,718
		0.3	1,097,390	2.27	50.15	3.27	115,499
		0.5	984,487	2.50	55.28	3.61	114,126
		1.0	815,137	2.91	64.45	4.20	110,065
		5.0	197,918	6.70	154.57	9.80	62,337
	DOM 2	0.0	1,574,828	0.26	5.76	0.37	18,773
		0.1	1,518,694	0.26	5.94	0.38	18,639
		0.3	871,247	0.35	8.75	0.53	14,810
		0.5	255,501	0.62	16.99	0.96	7,910
		1.0	67,182	1.16	31.80	1.79	3,869
		5.0	1,708	6.14	12.17	6.38	350
INFERRED	DOM 1	0.0	66,972	2.06	32.69	2.72	5,847
		0.1	62,092	2.22	35.25	2.93	5,847
		0.3	61,406	2.25	35.61	2.96	5,843
		0.5	57,019	2.40	37.77	3.16	5,787
		1.0	46,854	2.81	43.49	3.68	5,536
		5.0	10,794	5.78	87.26	7.52	2,611
	DOM 2	0.0	145,731	0.26	5.50	0.37	1,723
		0.1	138,614	0.27	5.77	0.39	1,717
		0.3	81,518	0.36	8.41	0.52	1,375
		0.5	22,125	0.65	16.54	0.98	699
		1.0	7,785	1.10	25.01	1.60	402
		5.0	31	6.91	10.50	7.12	7

Mineral resources are not mineral reserves and do not have a demonstrated economic viability. All figures have been rounded to reflect relative accuracy of the estimates.

The breakdown of mineral resources at a cut-off grade of 0.4 g/t AuEq in Measured and Indicated classification are detailed in Table 14-2.

Table 14-2 Measured and Indicated Mineral Resources at 0.4 g/t AuEq Cut-Off (TVIRD, January 2014)

CLASS	ROCKGROUP	Tonnage DMT	Au g/t	Ag g/t	AuEq g/t	AuEq Oz
Measured	DOMAIN 1	686,501	2.20	68.24	3.56	78,585
	DOMAIN 2	265,302	0.46	29.43	1.05	8,922
	TOTAL	**951,803**	**1.71**	**57.42**	**2.86**	**87,506**
Indicated	DOMAIN 1	1,032,062	2.40	53.01	3.46	114,808
	DOMAIN 2	419,685	0.50	13.17	0.76	10,262
	TOTAL	**1,451,748**	**1.85**	**41.49**	**2.68**	**125,070**

The mineral resource estimate has been validated and certified by Leo A. Sosa in June 2014. The current resource model represents an update of the previous models prepared by P.J. Lafleur (GeoConseil Inc., 2007), TVIRD (Inhouse resource model, 2011) and C.P. Smyth (Georeference Online Ltd, 2012).

The above estimation did not include the resource depleted by the illegal small -scale mining activities. However, based previous authors, C.P.Smyth and Angeles, 2011 and the recent subsurface mapping conducted, have estimated that around 14,000 Au Eq oz were mined by the small- scale miners.

14.1 Previous Resource Estimates

In 2007, the first official resource model was produced together with the NI 43-101 compliant technical report prepared by P.J. Lafleur Geo-Conseil Inc. (Table 14-3). The resource estimate was based on the 66 drill hole program initially completed by TVIRD from 2005 to early 2007. This estimate covers both the Tinago and Miswi-Lalab area.

Table 14-3 Resource Table by P.J. Lafleur, Geo-Conseil Inc., 2007

Class	Grade Group (Cumulative at variable cut-off)	Tonnage MT	Au g/t	Au ounces	Ag g/t	Ag ounces	AuEq* g/t	AuEq* ounces
Indicated	>2.0 g/t Au	695,128	4.54	101,452	132.98	2,972,660	7.30	163,192
	>1.0 g/t Au	1,091,970	3.41	119,912	100.61	3,531,287	5.50	193,254
	>0.5 g/t Au	1,371,105	2.87	126,507	84.33	3,716,842	4.62	203,703
	0.0 - 0.5 g/t Au	314,820	0.28	2,834	9.25	93,665	0.47	4,779
	Total indicated	**1,685,925**	**2.38**	**129,341**	**70.31**	**3,810,507**	**3.84**	**208,482**
Inferred	>2.0 g/t Au	821,521	4.50	118,864	96.90	2,560,532	6.51	172,045
	>1.0 g/t Au	1,488,853	3.14	150,300	65.90	3,152,794	4.51	215,781
	>0.5 g/t Au	1,957,168	2.58	162,056	55.20	3,471,890	3.72	234,165
	0.0 - 0.5 g/t Au	498,254	0.24	3,845	8.33	133,411	0.41	6,615
	Total inferred	**2,455,422**	**2.11**	**165,901**	**45.69**	**3,605,301**	**3.05**	**240,780**

**AuEq @ $650/oz Au and $13.50/oz Ag*

A comprehensive scoping study was carried out by Genivar Limited Partnership in 2007 to assess the mining potential of the Balabag Project centered on its Tinago and Miswi vein deposits. Genivar recommended further infill drilling to reclassify the “inferred resources” to “measured and indicated categories”.

As the drilling in Balabag prospect progresses, an updated resource model was produced in August 2011. The model is based on 199 drill holes which include new drill data from the first and second quarter of 2011. At a cut-off grade of 0.4 g/t Au, the updated Indicated resource for the Balabag project stands at 1.69 M tonnes at 2.48 g/t Au and 75.5 g/t Ag (
Table 14-4). This is equivalent to 135,323 Au ounces and 217,842 AuEq ounces.

Table 14-4 TVIRD In-House Estimate of the Indicated Resource in Balabag Project, August 2011

Grade Group (Cumulative at variable cut-off)	Tonnage MT	Au g/t	Ag g/t	AuEq* g/t	AuEq* ounces
>2.0 g/t Au	767,743	4.18	124.10	6.67	164,536
>1.0 g/t Au	1,252,016	3.12	96.00	5.04	202,851
>0.5 g/t Au	1,601,823	2.60	79.40	4.19	215,638
>0.4 g/t Au	1,699,989	2.48	75.50	3.99	217,842
>0.3 g/t Au	1,892,343	2.26	68.70	3.63	221,052
>0.2 g/t Au	2,683,745	1.66	50.60	2.68	230,862
>0.1 g/t Au	4,429,604	1.07	32.90	1.72	245,313

**AuEq @ $1000/oz Au and $20/oz Ag*

Note:

(1) Total estimate includes both Domain 1 (true veins) and Domain 2 (qtz stockworks).

(2) Potential resource of Warik-Warik Area not included in the estimate.

(3) Estimated by Ordinary Kriging method

Lastly, an independent technical report which is NI 43-101 compliant was produced by Georeference Online Ltd. In June 2012. The parameters used in generating the inhouse resource model in 2011 were reviewed and validated by Clinton P. Smyth, P. Geo.

The author's best estimate of gold and silver resource at Balabag based on information reflecting drilling completed to the end of June 2011 is detailed in Table 14-5. The Indicated resource estimate is presented with no assumed cut-off grade. Note that the possible volume of ore depleted by the small-scale miners is not accounted in the estimate below.

Table 14-5 Resource Table by Georeference Online Ltd., 2012

Class	Tonnage MT	Au (g/t)	Cont. Au (oz)	Ag (g/t)	Cont. Ag (oz)
Indicated	1,784,555	2.34	134,262	72.3	4,148,196

Note: Measured and Inferred resources were not estimated

The recently updated inhouse resource model made by TVIRD in January 2014 is based on geological interpretations of drill hole data in the main Tinago, Miswi and Lalab which include lithological logs, vein percentages, structures and gold and silver grades. Two major geological domains were identified which separate the massive vein unit from the stockworks.

14.2 Mineral Resource Estimation Method Used

In the latest resource model, two major geological domains were created to constrain the model (Figure 14-1). The true veins which have more predictable continuity and grade uniformity were categorized as **DOMAIN 1** (Volume: 858,739 m^3). This includes the following mineralized units:

a) Banded, high Au, quartz vein with banded black sulphides (VNB)
b) Massive translucent to milky quartz vein (VNQ)
c) Quartz vein breccia with angular andesitic clasts (VNX)

A separate envelope which was tagged as **DOMAIN 2** (Volume: 960,976 m^3) was used for the quartz stockworks (QSW) and silicified andesite (VIAN-sil). The stockworks were considered as another domain due to the erratic nature of grades and thickness. This type either occurs by itself or in the walls of the main veins, usually on the hanging wall side.

The three-dimensional wireframe of the domains was generated by connecting the vein interpretations plotted in each cross section by tie lines. Solids were checked for triangulation errors using the modeling software, GEMS version 6.3.

Geological domains were validated by running basic statistics with the intersected drill hole sample points and by visual check of the lithologic logs and assay relative to the geologic interpretation in each cross section. Statistical approach includes analysis on mean, mode, median and distribution of grades in each domain.

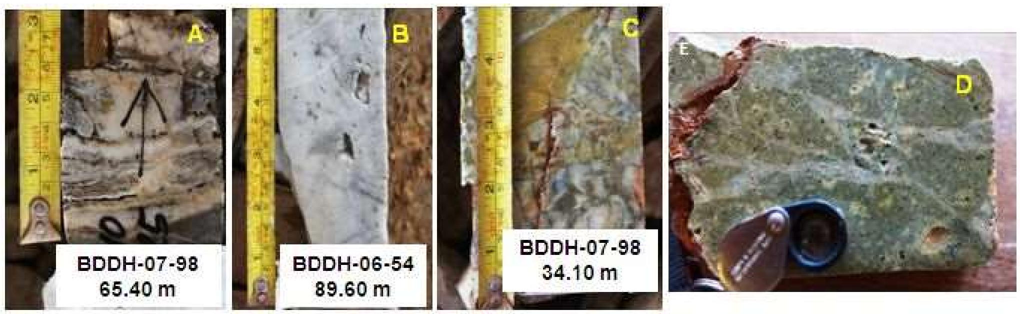


Figure 14-1 Mineralized Units: (A) Banded Quartz Vein (B) Milky Quartz (C) Quartz Vein Breccia (D) Quartz Stockworks in Propylitized Andesite

Lithological domains were refined by the statistical evaluation of assay data. A cut-off grade of 0.1 g/t Au was considered in outlining stockworks (Domain 2) while 0.5 g/t Au was used for the massive vein units (Domain 1). The resulting lithological wireframes were used as resource domains to constrain grade estimation.

Each wireframe was assigned a numerical code to facilitate identification. A rock code of 1 in the Balabag Resource block model represents wireframe characterized by massive veins or Domain 1. On the other hand, a rock code of 2 was used for stockworks or Domain 2.

Average density values of 2.4 g/cm^3 and 2.5 g/cm^3 were used for Domain 1 and 2, respectively. Figures were based on the calculated average density of the core samples.

Domain 2 exhibits higher density than Domain 1 due to the occurrence of stockworks with relatively competent andesite host rock. Meanwhile, Domain 1 rocks are observed to be generally altered and brecciated in fractured zones. It is recognized that the density, hardness and viscosity-influencing characteristics of ore-grade material will vary locally, primarily as a function of weathering and alteration. The following table shows the density values of each lithologic domain.

Table 14-6 Average Density per Domain

Geological Domain	No. of Density Determinations	Average Density
DOMAIN 1	195	2.4
DOMAIN 2	234	2.5
WASTE	363	2.5

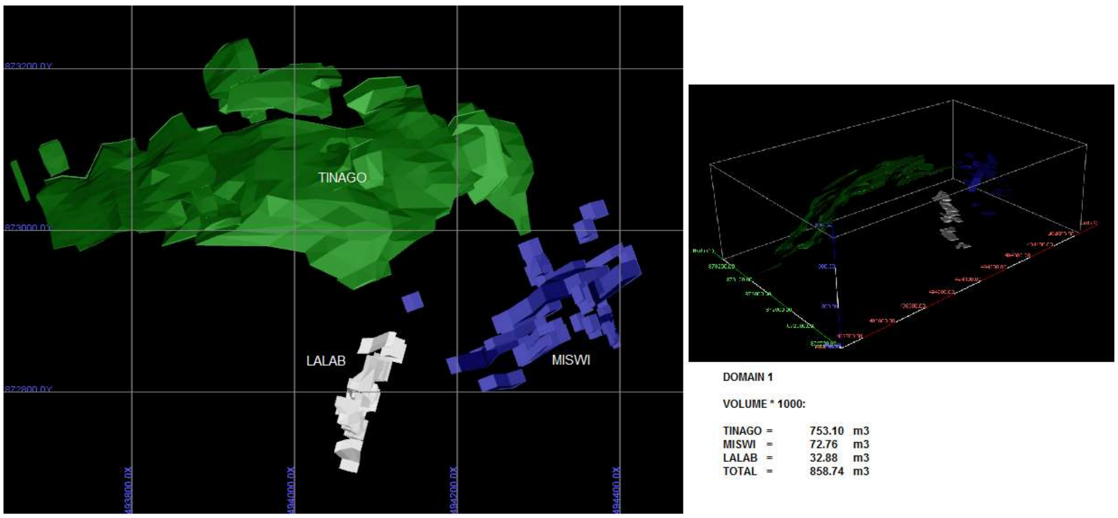


Figure 14-2 Geological Solid Model of Domain 1 in 3D View

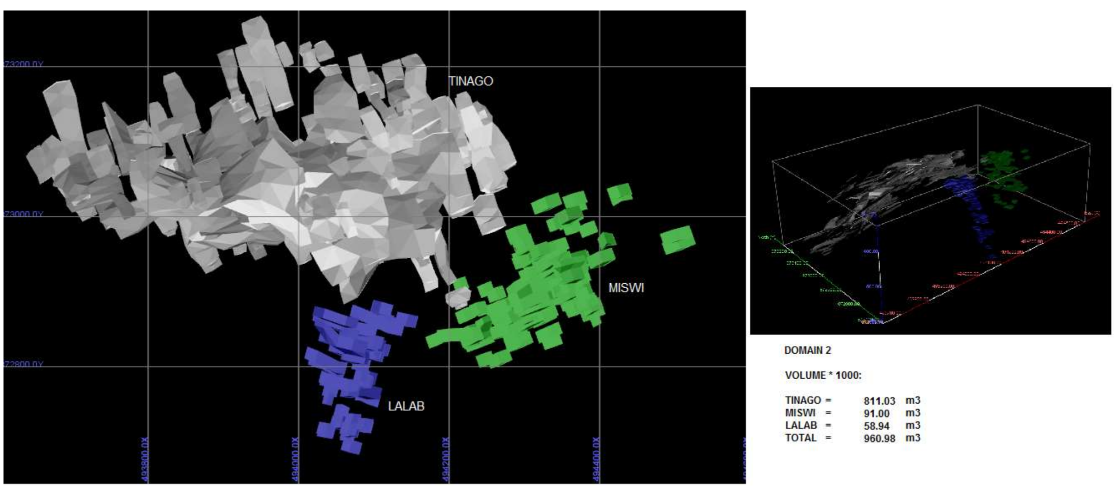


Figure 14-3 Geological Solid Model of Domain 2 in 3D View

Basic statistics on gold and interval lengths in the assay table was carried out to determine the optimum length for compositing. From the plot shown in Figure 14-4, one meter (1 m) is the optimum composite length for the core samples. Since the sampling was done every one-meter interval in average, no compositing in the assay data was necessary.

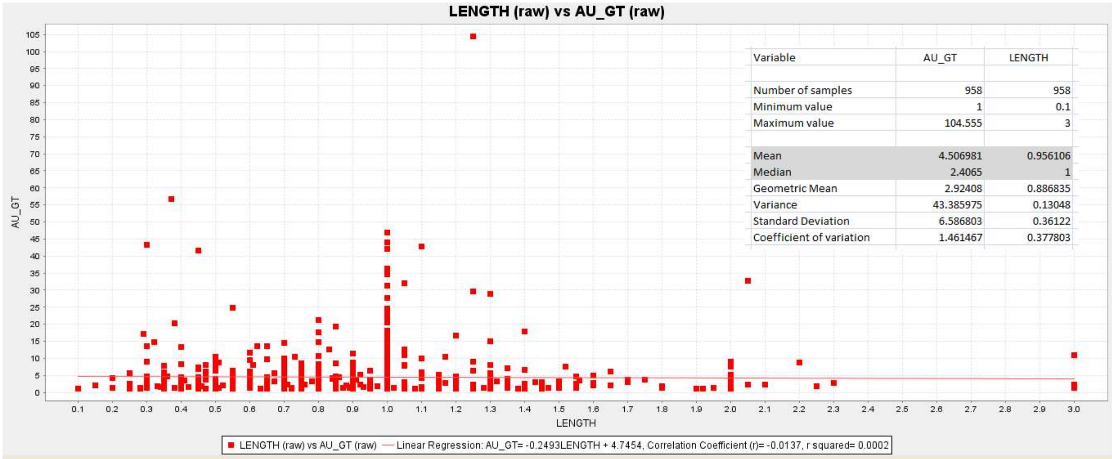


Figure 14-4 Interval Length vs. Gold Grade for Determination of Optimum Composite Length

Summary statistics and histograms for the key variables gold and silver were generated to further validate the domains. Test for correlation between gold and silver was made using scatter plots.

Statistical analysis shows that both gold and silver histograms are strongly log-normal in their distributions in both Domain 1 and Domain 2. The scatter plot also shows a very poor correlation between gold and silver. Table 14-7 shows a summary of the basic statistics of Domain 1 at different locations.

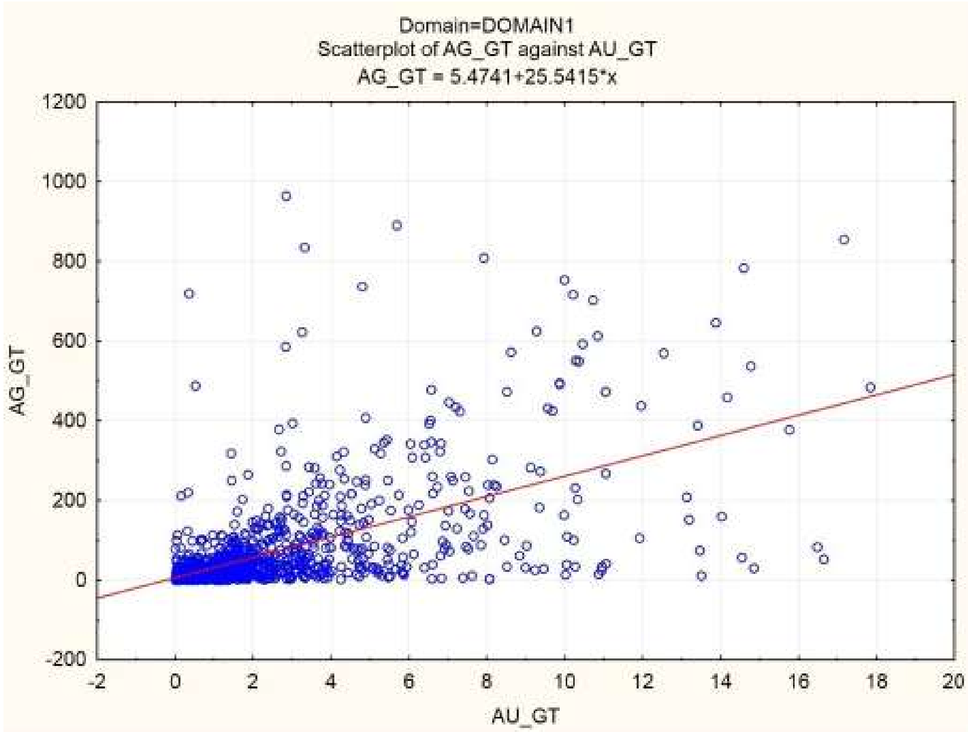


Figure 14-5 Gold vs. Silver at Domain 1

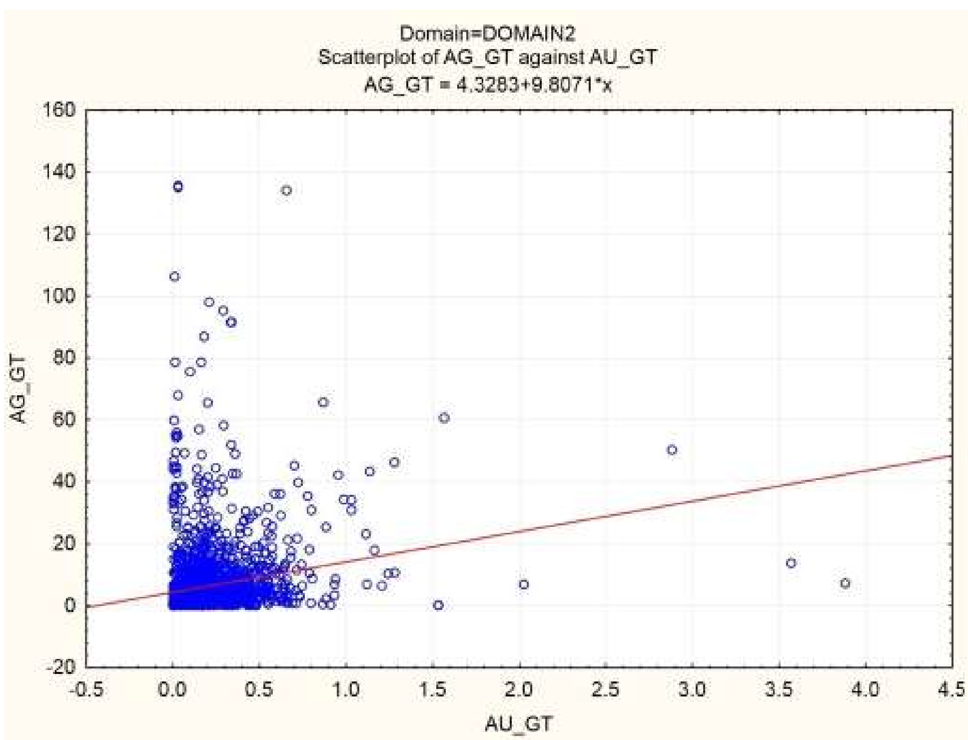


Figure 14-6 Gold vs. Silver at Domain 2

Table 14-7 Historgram Report of Samples from Domain 1

Vein	DOMAIN 1 - Au (gpt) TINAGO	LALAB	MISWI	TOTAL	DOMAIN 1 - Ag (gpt) TINAGO	LALAB	MISWI	TOTAL
Number of samples	1236	125	261	1622	1235	125	261	1621
Percentage	76%	8%	16%	100%	76%	8%	16%	100%
Minimum value	0.003	0.01	0.003	0.003	0.25	0.25	0.25	0.25
Maximum value	56.62	46.80	104.56	104.56	1,551.00	2,846.89	5,234.30	5,234.30
Mean	2.05	3.53	3.94	2.47	57.37	133.58	124.59	74.07
Median	0.76	1.48	1.19	0.89	16.24	29.80	16.07	17.05
Geometric Mean	0.65	1.34	1.20	0.76	16.59	30.34	21.24	18.08
Variance	16.84	34.48	79.03	28.77	15,658.72	92,398.52	196,370.15	51,569.50
Standard Deviation	4.10	5.87	8.89	5.36	125.13	303.97	443.14	227.09
Coefficient of variation	2.00	1.66	2.26	2.17	2.18	2.28	3.56	3.07

A capping value was determined by analyzing histograms and cumulative frequency curves of in-situ gold composites in each domain. A top-cut of 30 g/t Au and 500 g/t Ag were used in removing outliers from the histogram of the sample points used in estimation.

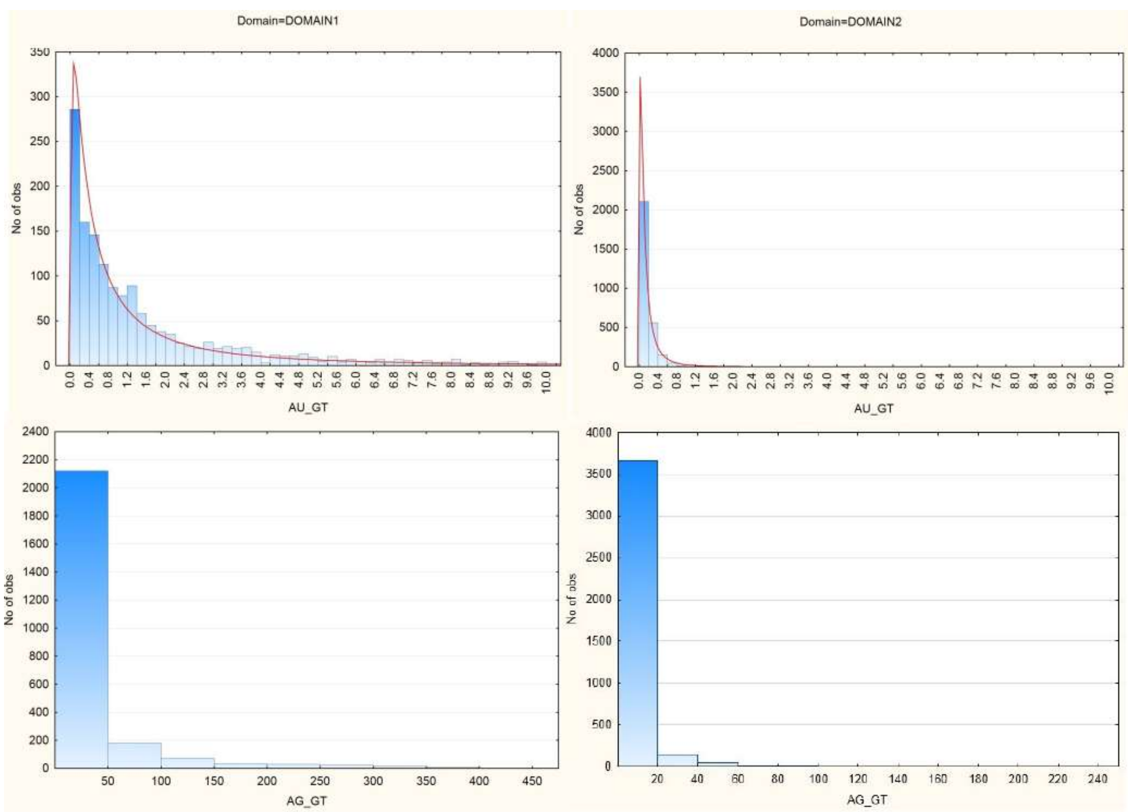


Figure 14-7 Gold and Silver Histogram for Domain 1 and Domain 2

Block Model Parameters and Estimation Method

The inhouse resource block model is designated as ‘TVIBAL2013’. Due to the presence of veins with thickness of one meter and below, a rather small block size of 5 x 2.5 x 2.5 has been considered in creating the block model workspace.

The extent of the north-south oriented block model was based on the data limits of the drill holes and geological data. Figure 14-8 is a summary of the block model geometry.

Block Model Name:	TVIBAL2013
Number of blocks	
Columns:	200
Rows:	320
Levels:	160
Origin and rotation	
X:	493600
Y:	872500
Z:	700
Rotation:	0
Block size	
Column size:	5
Row size:	2.5
Level size:	2.5

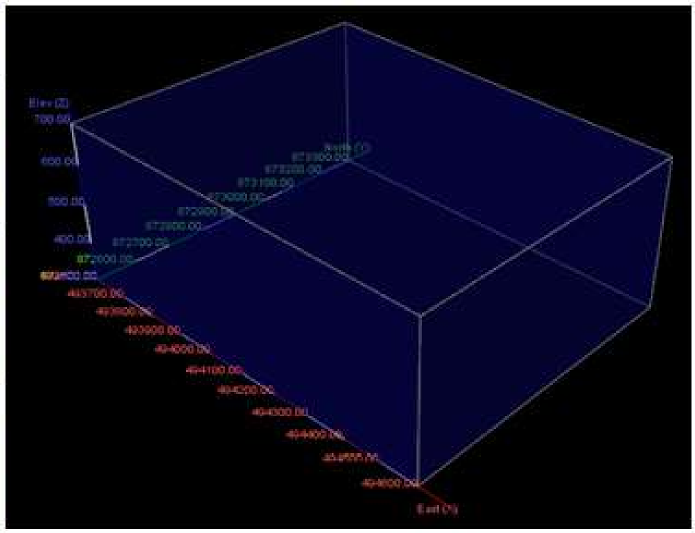


Figure 14-8 Resource Block Model Parameters (TVIBAL2013)

Due to the likelihood of having more than one mineralized material to exist in a block, partial or percent modeling was carried out. This method tends to compute for the percent volume of material intercepting a block instead of plainly assuming a 100% full block of a single material type. Two separate folders for DOMAIN 1 and DOMAIN 2 were set up to keep the data separate for different ore types. A standard folder was created to store the waste blocks as well as the air blocks. Each folder for domains contains rock type, grades, density and percent attributes for each ore type. The percent attribute represents the portion of the block volume occupied by the rock type represented in the rock type attribute in the same folder. The remaining volume of the block is associated with rock types found in the rock type attributes stored in other folders.

The mineral resources were estimated using conventional geostatistical block modeling approach constrained by mineralization wireframes. Geostatistical analysis, capping, variography and estimation were conducted on the in-situ gold and silver data.

As noted by previous authors particularly C.P. Smyth, grades at the Balabag Project are extremely erratic resulting to variograms unsuitable for use in kriging. This is somehow expected, given the non-normal distribution of gold and silver values in both domains. Hence, Inverse Distance Squared (ID^2) method was deemed more practical to use instead of Ordinary Kriging. The search criteria used in grade interpolation of blocks are detailed in Table 14-8.

Table 14-8 Search Criteria Used for Grade Interpolation in Balabag Gold Project

Area	Domain	Search Ellipsoid		Ellipsoid Orientation		
		1st Pass	2nd Pass	P. Azimuth	P. Dip	Int. Azimuth
Tinago	Domain 1	30 x 30 x 5	60 x 60 x 10	160	40	70
	Domain 2	30 x 30 x 5	60 x 60 x 10	160	40	70
Miswi	Domain 1	30 x 30 x 5	60 x 60 x 10	240	20	150
	Domain 2	30 x 30 x 5	60 x 60 x 10	240	20	150
Lalab	Domain 1	30 x 30 x 5	60 x 60 x 10	300	30	30
	Domain 2	30 x 30 x 5	60 x 60 x 10	300	30	30

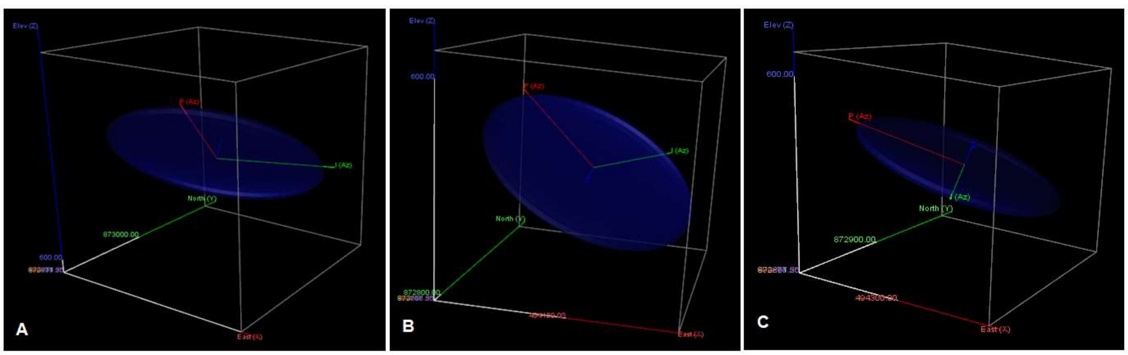


Figure 14-9 Search Ellipsoid Orientation for (A) Tinago (B) Lalab (C) Miswi. Red and lines represent the principal and intermediate axes, respectively.

The orientation of each search ellipse was based on the geological interpretation made on three major locations in the project area which include Tinago, Miswi and Lalab. Separate domains were identified to distinguish the massive vein unit (Domain 1) from the stockwork zones (Domain 2).

The model was validated visually by comparing block grade estimates to informing capped sample data on vertical sections and elevation plans. The statistics of the informing capped drill data also compared well to those of the estimated resource blocks.

14.3 Mineral Resource Categories Used

Inverse Distance Squared (ID^2) was used in the grade interpolation process with varying search ellipses to classify the resource into Measured, Indicated and Inferred. The block classification involved a two-step process. The first step was an automated classification which considered the number of points to code a block, the size of the search ellipse and the average distance to informing composites. Blocks coded during the first pass using a search ellipse with ranges of 30 x 30 x 5 were assigned as Indicated Resource. All blocks interpolated during second estimation passes with search ellipse of 60 x 60 x 10 were assigned an Inferred classification.

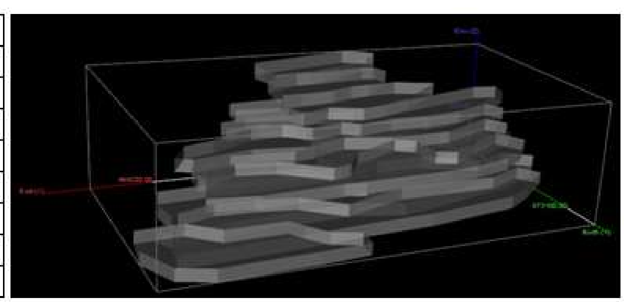
Table 14-9 Interpolation Parameters Used in Classifying Indicated and Inferred Resources

	Inferred	Indicated
Search ellipsoid	60 x 60 x 10	30 x 30 x 5
Number of samples used		
Minimum	2	2
Maximum	12	12
Calculation method	Inverse distance	Inverse distance
Inverse-distance power	2	2

A separate solid wireframe was generated to delineate a zone with tighter drill data (average of 25 m spacing) which will represent areas of higher confidence level in terms of geological interpretation and grade continuity. This was used to further constrain the blocks from first pass estimation and therefore assigning Measured classification to the blocks. The solid model was found to coincide with the initial mine design created using Whittle software during the initial mine optimization exercise in year 2011.

Table 14-10 Interpolation Parameters Used in Classifying Measured Resource and Solid Model Used to Constrain Blocks within 25 m Drill Spacing

	Measured
Search ellipsoid:	30 x 30 x 5
Number of samples used	
Minimum:	2
Maximum:	12
Calculation method:	Inverse distance
Inverse-distance power:	2
Drill spacing average:	25 m
Solid Volume:	1,180,104 cu.m.



The classification of the mineral resource is based on the 2010 CIM Definition Standards for Mineral Resources (new standard published May 10, 2014) and Philippine Mineral Resource Code (PMRC) as outlined below:

Inferred Mineral Resource

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 Resource

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.

Measured Mineral Resource

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.

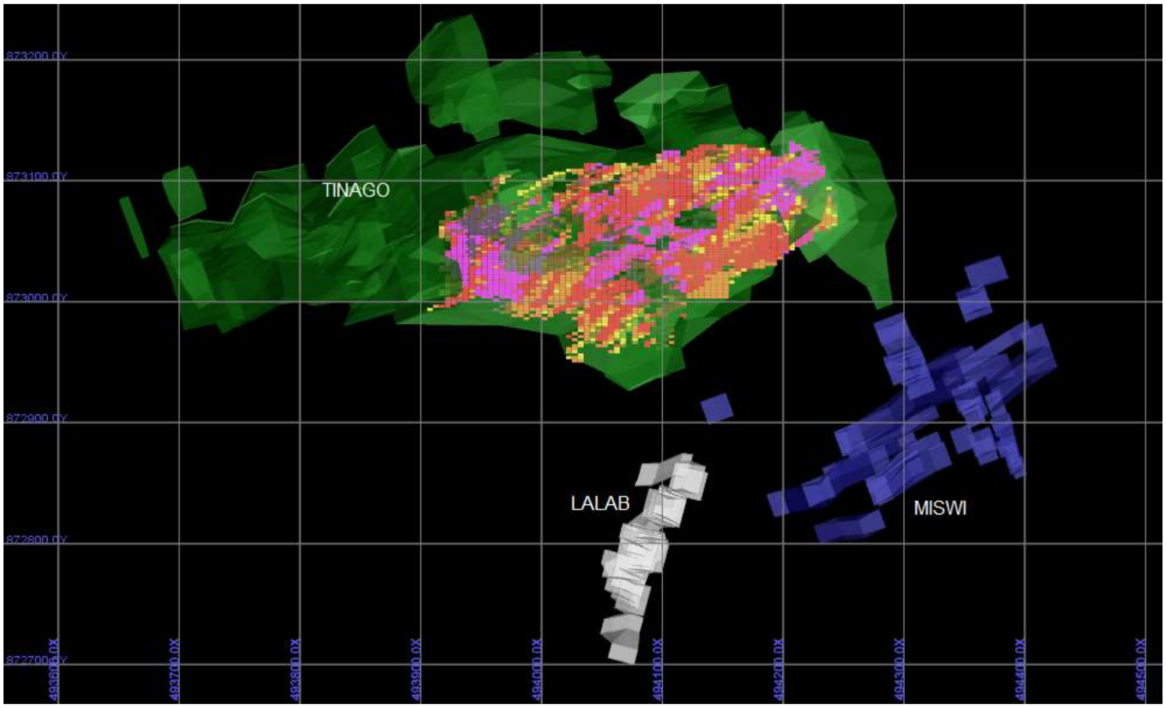


Figure 14-10 Distribution of Measured Resource Blocks Relative to the Ore Body Model (BM: TVIBAL2013)

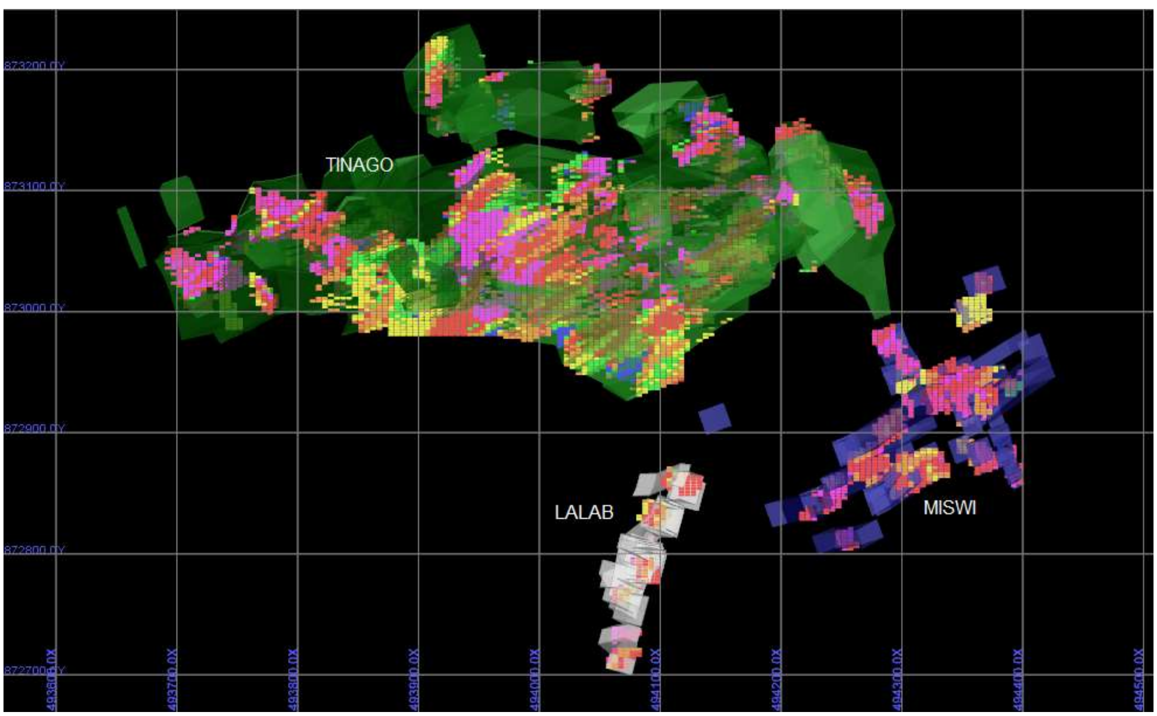


Figure 14-11 Distribution of Indicated Resource Blocks Relative to the Ore Body Model (BM: TVIBAL2013)

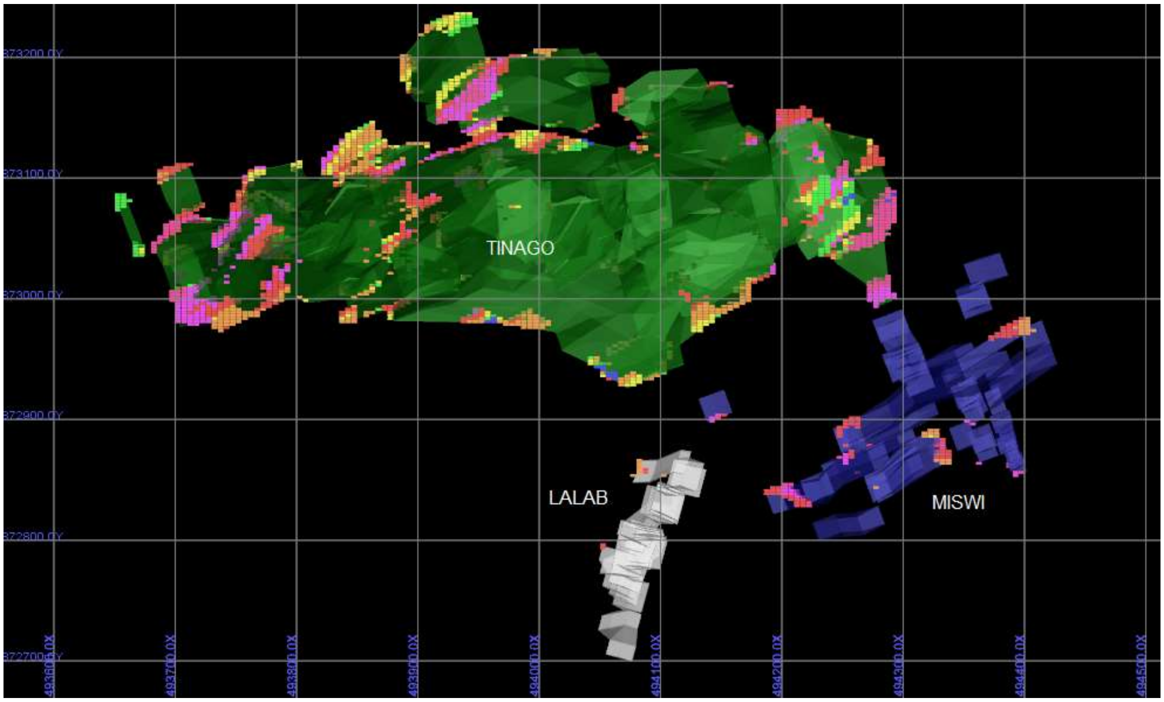


Figure 14-12 Distribution of Inferred Resource Blocks Relative to the Ore Body Model (BM: TVIBAL2013)

15 MINERAL RESERVE ESTIMATES

The orebody has an idealized shape and attitude showing distinct mineralized zones extending down dip which reflect the current geological interpretation. The upper Tinago vein is relatively gently dipping and almost outcropping to the surface with relatively thin overburden. Other parts of the deposit are rather steeply dipping but can be accessed from the slope of the topography. The details of the mineable reserves and mining operations are presented in this section.

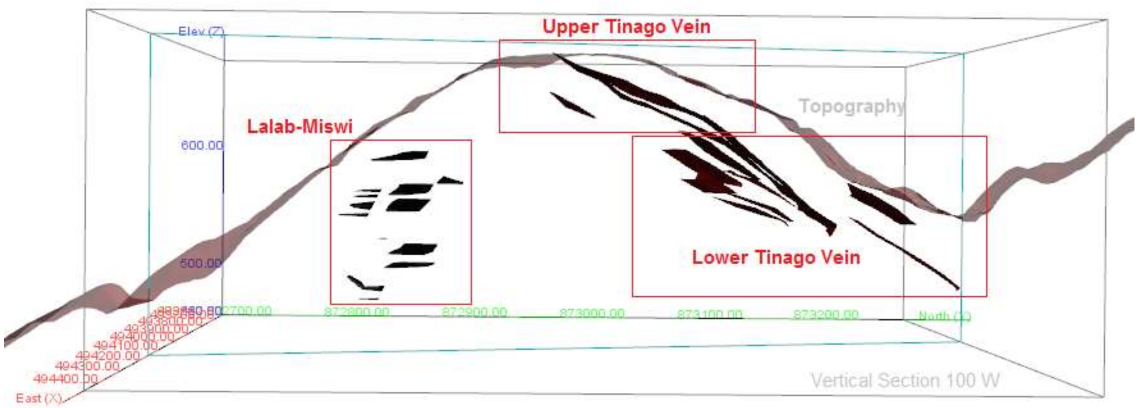


Figure 15-1 Typical Section of the Balabag Deposit (Vertical Section 100 W, looking SW)

15.1 Mineral Reserves

The mineral reserves estimate stands at 1.35M tons, at 2.50 g/t Au and 68.25 g/t Ag (Table 15-1). The current mineable reserves were defined based on the resource block model updated in January 2014, with metal prices set at USD 1300/oz and USD 18/oz for gold and silver, respectively. The internal cut-off grade for the mine is 0.83 g/t AuEq (gold equivalent), which was derived based on the assumed metal prices, combined milling, selling and G&A costs and plant recovery.

Table 15-1 Mineral Reserves of the Balabag Gold and Silver Project

	Rockgroup	Volume m3	Density DMT/m3	Tonnage DMT	Au g/t	Ag g/t	AuEq g/t	Au oz
PROVEN	DOM 1	250,349	2.4	600,837	2.42	75.09	3.55	68,578
	DOM 2	36,241	2.5	90,601	0.65	54.73	1.47	4,274
	SUB-TOTAL	286,590	2.5	691,439	2.19	72.42	3.28	72,852
PROBABLE	DOM 1	256,317	2.4	615,160	2.95	65.75	3.93	77,802
	DOM 2	16,904	2.5	42,259	0.95	36.40	1.50	2,037
	SUB-TOTAL	273,220	2.5	657,420	2.82	63.86	3.78	79,838
TOTAL ORE RESERVES	DOM 1	506,666	2.4	1,215,998	2.69	70.36	3.74	146,380
	DOM 2	53,144	2.5	132,861	0.74	48.90	1.48	6,310
	TOTAL	**559,810**	**2.5**	**1,348,859**	**2.50**	**68.25**	**3.52**	**152,690**

Note: 1) Proven and Probable ore reserves are derived from Measured and Indicated resources, respectively.

2) Cut-off grade used for ore is 0.83 g/t AuEq; everything below cut-off grade is considered waste.

3) The reserves are net of the estimated 14,000 Au Eq oz were mined by the small- scale miners.

The mineral reserve will be diluted from material below the cut-off grade so the impact on average grade mined will be minimal.

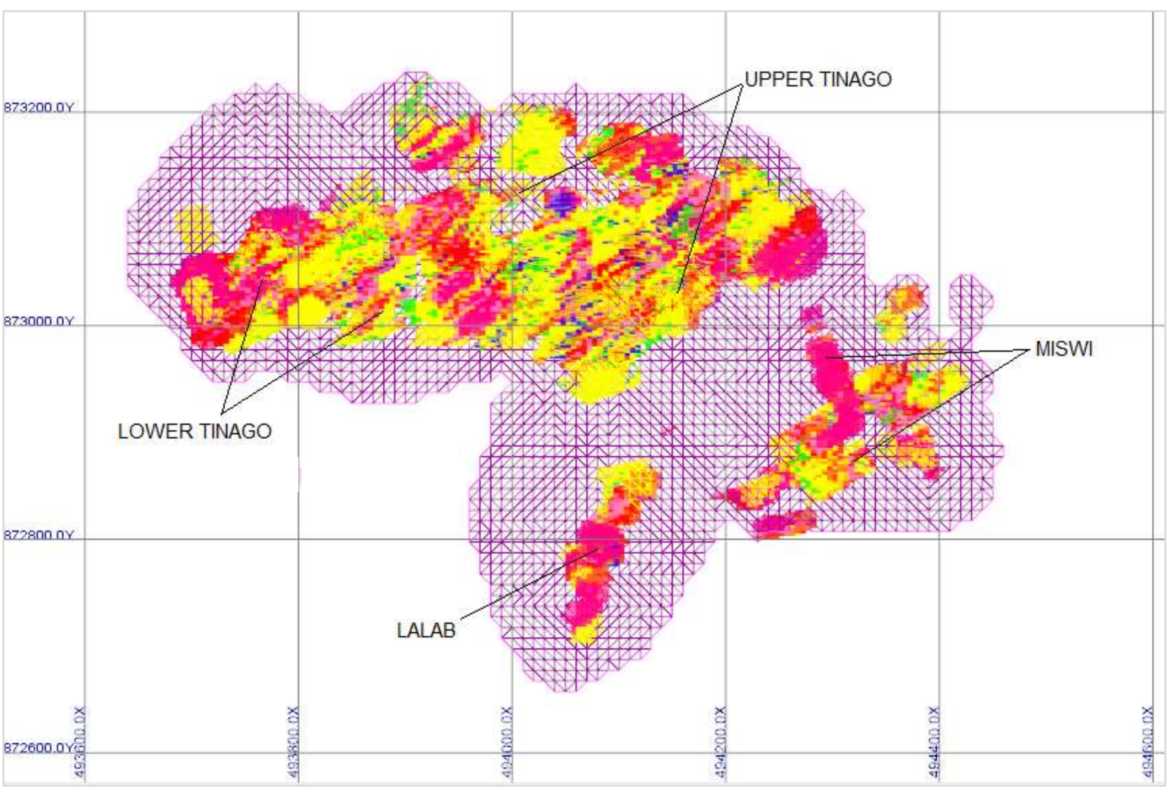


Figure 15-2 Footprint of the Optimized Mine Shell and Balabag Mine Reserves

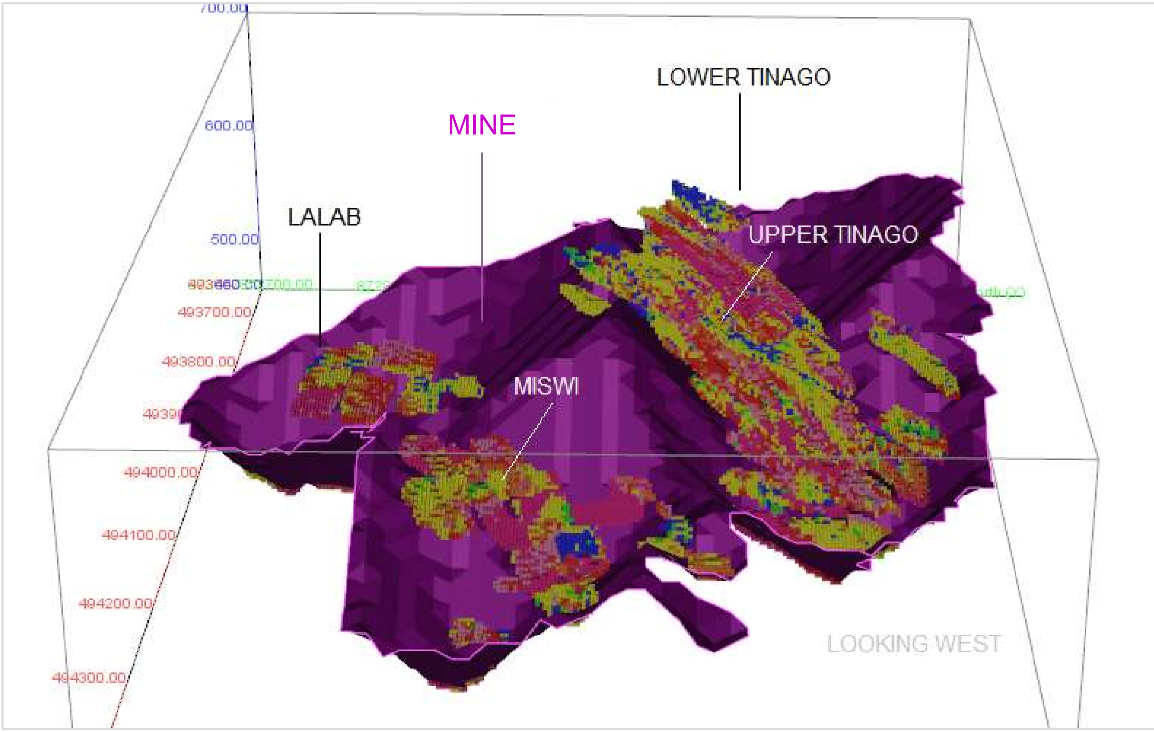


Figure 15-3 Isometric View of the Optimized Mine Shell and Balabag Mine Reserves

15.2 Mine Optimization and Cut-off Grade

The mine optimization was done using Whittle mining software, based on the resource block model of the deposit. Each block is assigned a net value depending on its particular location in the model and its correlation with surrounding blocks dictating its relative cost of mining extraction and milling.

The stages for optimization process are summarized below:

1. The ‘net value’ is calculated for each block in the resource model. The net value can be defined as the block’s revenue minus all its associated costs.
2. The software then performs multiple iterations to define the mine limits that will provide the highest value, based on the cumulative net value of each block mined. Basically, this is a trial and error process of finding the combination of blocks that can provide the highest net value.
3. Once the optimal mine limit has been delineated, the ore reserves figures are defined based on the internal cut-off grade.

Several nested mine shells corresponding to optimal limits at various gold/silver prices were generated. These nested shells were used in the selection of ultimate mine shell, and also served as guide in defining the mining stages or 'pushbacks' throughout the life of the mine.

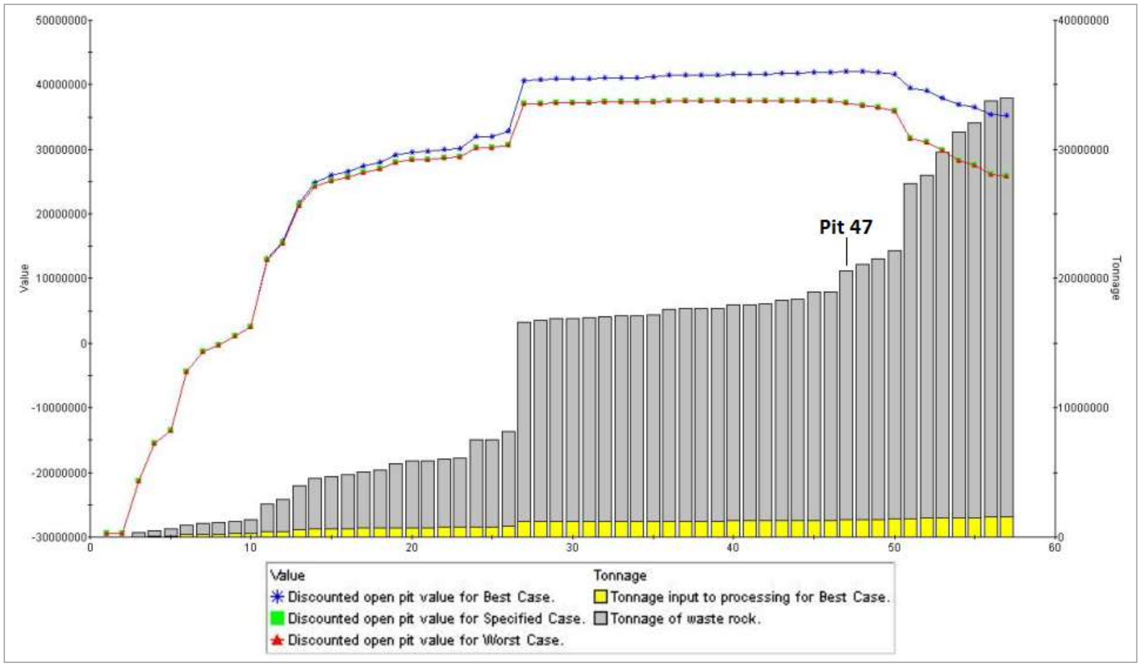


Figure 15-4 Graph for Ore and Waste Tonnage Input and Equivalent Net Value

The optimization parameters presented in Table 15-2 were established by computing capital and average operating costs from actual costs of the Canatuan Mine and with reference to other local gold projects undergoing feasibility. Slope angles were based on the geotechnical study conducted by V.M. Tumbokon in April 2015 (see **Appendix III-C**).

The selected mine shell was smoothed and adjusted to what is deemed practical for actual mining operations. The mine design criteria applied in smoothing the starter mine are provided below:

Table 15-2 Optimization Parameters and Mine Design Parameters

	Variables		
Optimization Parameters	**Mill Throughput**	t/d	2,000
	Upper Slope Angle	deg	40
	Distance from surface	m	25
	Lower Slope Angle	deg	45
	Ore Mining Cost	\$/t	2.51
	Waste Mining Cost	\$/t	2.00
	Milling Cost	\$/t	23.00
	Gold Price	\$/oz	1300
	Silver Price	\$/oz	18

Overall Mine Slope	40-45 degrees
Bench Face Angle	63.0 degrees, or 1H:2V
Bench Height	2.50 meters
Berm Width	4.0 meters/ 3 benches
Haul Road Width	10 meters
Haul Road Gradient	10%

A ramp system was not yet included in the smoothed mine limit. The final mineable reserves volume can be expected to change by 5% to 10% once the haul roads have been fully incorporated in the mine design.

The internal cut-off grade (ICOG) for the mine is 0.83 g/t AuEq. The ICOG consistent with the value used in the optimization process can be back calculated as follows:

Equation 15-2 Internal Cut-Off Grade (ICOG) Calculation

$$\frac{\text{ICOG} * \text{1MT ore} * \text{Au Recovery (Au Price} - \text{Selling Price)}}{31.1035\ \text{g/oz}} = \text{Milling Cost} + \text{G\&A cost}$$

$$\frac{\text{ICOG} * \text{1MT ore} * 89\%\ (\$1300 - \$120)}{31.1035\ \text{g/oz}} = \$23/\text{MT} + \$5/\text{MT}$$

$$\frac{\text{ICOG} * \text{1MT ore} * 89\%\ (\$1180)}{31.1035\ \text{g/oz}} = \$28/\text{MT}$$

$$\textbf{ICOG = 0.83 g/t AuEq}$$

16 MINING OPERATIONS

The deposit will be mined using the side cut mining method and will be patterned after TVI Canatuan Gossan and Sulphide Projects' model. It is conceived that the mining unit operations of drilling, blasting, loading and hauling will apply to the Balabag Au-Ag Project.

16.1 Production Schedule

The deposit will be mined using the side cut technique and will be patterned after Canatuan gossan and sulphide projects' model of contract mining. The mine will be developed in a series of pushbacks instead of applying the simple 'top-down' mining, to minimize the initial stripping requirement and associated costs. The pit will be free-draining. Mining is planned such that no water will accumulate within the pit. Runoffs will be diverted through channels into the tailings storage facility.
A production schedule had been developed for the mineable reserves presented in the previous section. The mine schedule is designed in line with the following strategies:

1. Minimize the initial stripping requirement and defer stripping to the later stages of mine development.
2. Attempt to mine higher-grade ore first, if possible.

Table 16-1 provides the annual production schedule for the mine with an average strip ratio of 10:1. The total reserve is 149,716 Au Eq at an Au and Ag price of 1250 USD/oz and 15 USD/oz.

Table 16-1 Mine Production Schedule

Year	Throughput (t/d)	ORE				WASTE	TMM	S.R.
		Tonnage (DMT)	Au g/t	Ag g/t	AuEq g/t	Tonnage (DMT)	Tonnage (DMT)	
Pre-production		-	-	-	-	517,741	517,741	-
1	2,000	410,000	2.45	90.23	3.54	3,635,268	4,045,268	8.87
2	2,000	700,000	2.46	61.96	3.21	7,251,910	7,951,910	10.36
3	2,000	290,000	2.64	52.37	3.27	2,385,081	2,675,081	8.22
TOTAL		**1,400,000**	**2.50**	**68.25**	**3.32**	**13,790,900**	**15,190,000**	**9.85**

The daily milling rate is set at 2000 t/d. Given this rate, the project is expected to operate for four years.

16.2 Mine Planning

A team of TVIRD technical personnel will plan, schedule and supervise the overall mining operations. Short-term mine plans will be developed in accordance to the long-term production goals. These plans will include mining direction, and the quantity and quality of materials to be moved.

16.3 Grade Control

16.3.1 Mining Dilution

Dilution occurs when waste (material below the cut-off grade) is mixed with ore which reduces the level of gold in the ore. While it is not simple to quantify the level of dilution, several methods were considered in running the block model to minimize the impact of ore dilution.

Partial Block Modeling

Due to the likelihood of having more than one ore type to exist in a block, partial modeling was carried out in estimating both resource and reserve. In a method called Percent Modeling, the software (GEMS) uses a technique called 'needling' to accurately estimate the percentage of the materials incorporated in a certain block. The percent value for each material is considered as factor in calculating the volume, tonnage and grades carried by each block (Figure 16-1).

This process of estimation is a practical way to somewhat allow the software to consider the proportion of dilution potential in the final resource/ reserve estimate—particularly those blocks located along the fringes of the orebody (external dilution) and those in between mineralized zones (internal dilution).

Similar method was used by TVIRD in the former Canatuan Sulphide Project. This technique helped in producing a more robust resource block model for Canatuan ore reserves; which has tremendously improved mining performance as demonstrated by better tonnage and grade reconciliation, mine recovery and mill production. The integration of Percent Modeling technique in the process allowed accurate assigning of volume/ tonnage (Figure 16-2), grades (Figure 16-3) and density to blocks which helped address issue of dilution as well as minimizing the possibility of overestimation.

Based on the Canatuan experience, the block model generated for Balabag has most likely resulted in an average of 5% dilution with a subsequent increase in diluted tonnes of ore offset by losses during mining and handling.

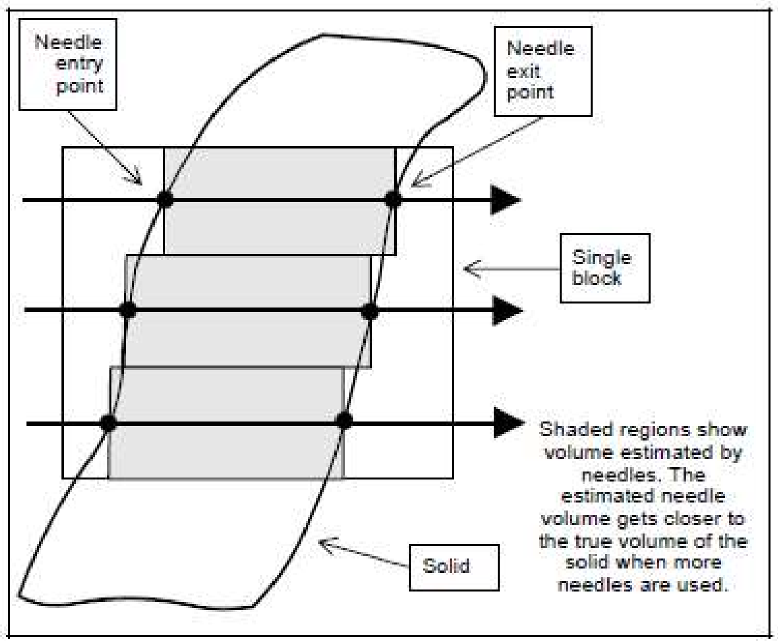


Figure 16-1 Gems calculates what proportion of each block is inside each geological solid. It reassigns the block values based on both this amount and the type of updating.

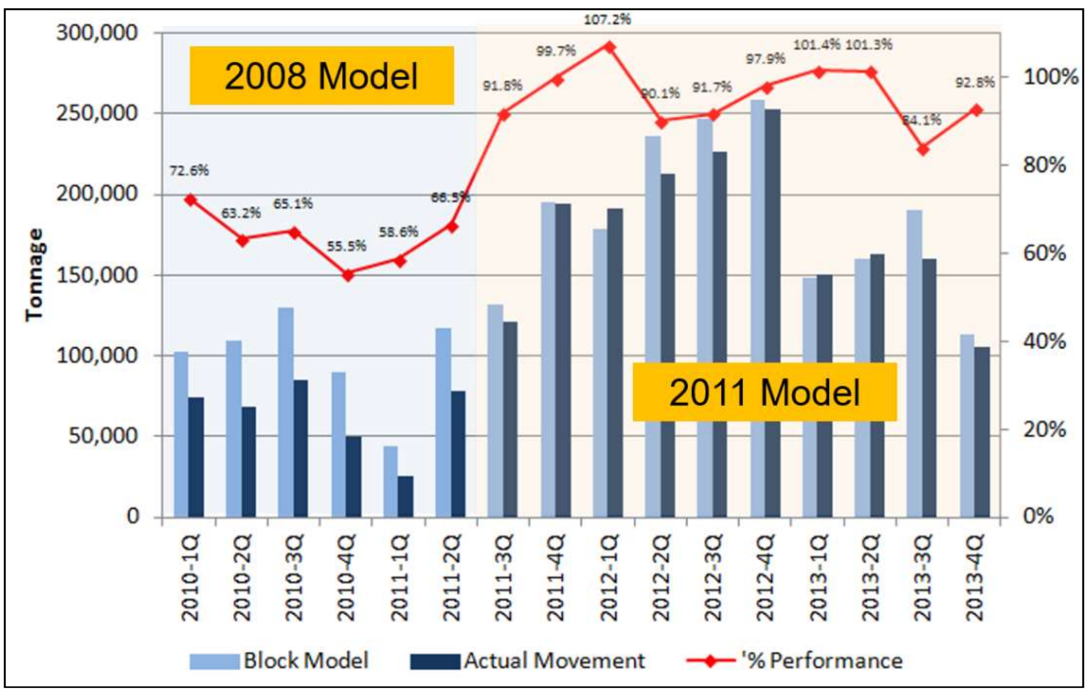


Figure 16-2 Better block model performance (tonnage) was noted in the revised Canatuan Block Model in 2011; after the integration of the Percent Modeling in the resource model.

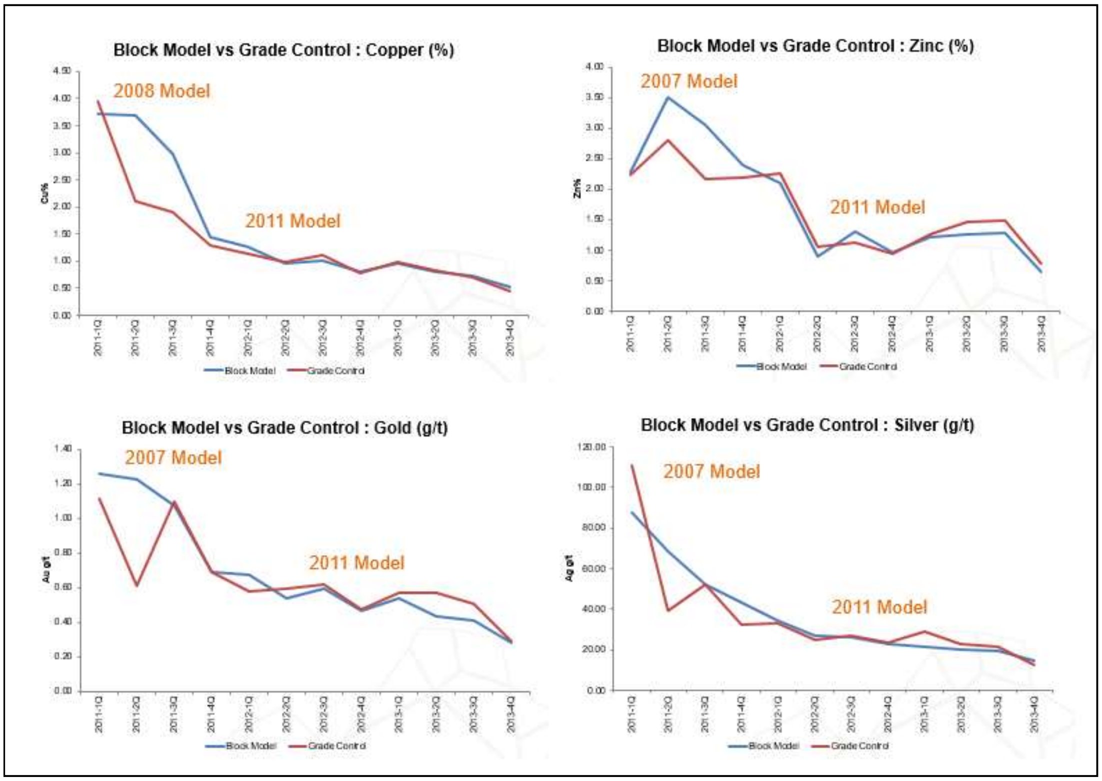


Figure 16-3 The 2011 Canatuan block model grade performance improved when Percent Modeling was initiated.

Selective Mining and Ore Control

Since majority of the veins range from 1m to 2m in average, a rather small block size of 5 x 2.5 x 2.5 has been considered in creating the block model. Smaller block size is used to relatively match the bucket size of the excavator to be used in ore extraction.

Selective mining has been practiced as well in Canatuan where complex mineralization was present. Thorough ore control must be performed to minimize dilution. Ore spotters, digger operators and pit mining supervisors must collaborate regularly to optimize digging and reduce the risk of dilution.

Operators must follow preferred digging directions to allow clean extraction of ore. Always mine by first exposing the ore by cleaning off the hanging wall waste and then start mining the ore. Along the footwall of ore block, any waste that falls off into the mining path must be pushed out of the way to avoid mixing with ore.

Dig plans must be made available on a daily basis and proper ore mark out must be performed on the ground.

A system of grade control will be incorporated in the mine. This will be implemented to ensure that the materials are properly classified as ore or waste prior to actual mining, thus avoiding dilution and ore loss. This will also provide advance information that allows the engineers to adjust mine plans, when needed, in order to keep up with production objectives. Lastly, grade control system will play a critical role in ore reconciliation and in determining the block model performance.

Grade control will be based mostly on a drilling program of closely spaced holes. Grade control holes will be laid in a staggered, 5m x 5m pattern, although actual sampling intensity may vary over time. In competent areas which will require blasting, grade control sampling will be incorporated with blast-hole drilling. All samples will be collected, logged and analyzed to create grade envelopes that will be used to delineate ore and waste. Dig limits will then be modified as necessary.

16.4 Mine Survey

A mine survey group will be put up to accommodate the survey requirements of the mine. The team will track the entire mining progress by conducting regular mine surveys and providing updates on topographic configuration of the mine – an essential part of mine planning and ore reconciliation. The survey team, in coordination with the planning group, will also provide field control stakes such as digging (and dumping) limits, elevation control and ore-waste boundaries. The team will also be tapped for other requirements, such as dam construction monitoring and volumetric surveys.

16.5 Drilling and Blasting

A contractor will be tapped to provide drilling and blasting services to the mine operations. The blasting contractor will be in charge of explosives handling and inventory and the actual drilling and blasting, while TVIRD will be in charge of the overall direction in terms of schedule, location and volume to be blasted.

The holes will be laid out in a staggered pattern, and drilling parameters will be varied according to specific material and actual experiences in the field. Blast holes will be charged with ANFO and primed with emulsion explosives. Wet holes will be loaded entirely with packaged emulsion.

16.6 Mining Fleet Selection

Fleet selection is based on a combination of criteria with consideration to production requirements, operational flexibility, minimum mining widths, economics and ability to operate continuously in light to moderate weather conditions. Main equipment requirements (excavators and trucks), cycle times and average truck speeds were determined using equipment manuals from suppliers, actual data from operating mines and operating parameters assumptions.

Dozers will be utilized for mine development, floor maintenance of active mining areas, ore stockpile and waste dump maintenance, and rehabilitation works. Road grader, vibratory compactor and water truck will be utilized for ancillary activities like maintenance of mine haul roads, the mine-ROM haulage road and all access roads on the project site.

All mining equipment will be provided by the mining contractor on a lease arrangement. Since mining operation will be subcontracted, no acquisition of the mobile and heavy equipment will be done since the mining contractor will provide all the mining equipment. Mining equipment capital costs are to the account of an equipment provider or contractor. Most of the mining equipment will be sourced out from local equipment supplier/contractor within the area on rental basis.

There is no specific equipment brand name or make (i.e. Volvo, Caterpillar, Komatsu, etc.) required for the mining operations provided the equipment are tough enough in any kind of weather conditions.

There are local equipment contractors in the area that are capable of supplying the required main mining equipment fleet and support equipment. Mobilization costs for the equipment will be less or nothing at all if sourced out from local suppliers.
The equipment contractors will also be the one responsible in providing for the maintenance and servicing of the mine equipment fleet as well as on its replacement upon reaching operating life.

An initial set of loading and hauling equipment supplemented by support maintenance equipment (i.e. dozer, auxiliary excavator and compactor) will be assigned to take care of the mine development works which will include the roads, tailings dam and site development.

Table 16-2 shows the list of equipment requirement and list of mobile and fixed equipment needed for mining to be provided by service contractors.

Table 16-2 List of Mobile and Fixed Equipment for Mining

EQUIPMENT	QTY. (UNIT)
1. Excavator, CAT 320D/PC-200 or equivalent	4
2. Excavator, CAT 336/PC-300 or equivalent	4
3. Bulldozer, CAT D6R or equivalent	3
4. Bulldozer, CAT D8R or equivalent	1
5. Dump Truck, 10-wheelers, 15 cu m	12
6. Front End Loader/Wheel Loader, CAT PL950H	1
7. Road grader, CAT 12G/Komatsu GD28	1
8. Road roller/compactor (20T)	1
9. Rock breaker, CAT 336D	1
10. Generator, 160 KW/200KVA	1
11. Generator, 30KW/ 1phase	1
12. Mitsubishi Canter/Service Truck	2
13. Boom Truck/Crane	1
14. Welding machine, engine driven type	1
15. Welding machine, transformer type	1
16. Flatbed truck with lifter, 8T	1
17. Fuel truck, 6,000 li	1
18. Lube truck	1
19. Drill Machine, Sandvik 1100	2
20. Explosive truck, Wing/Dropside	1
21. Service pick-up 4x4	10
22. Tower lights, 6Kw	10
23. Water tank truck, 15,000 liters Cap.	2

16.7 Waste Dump Maintenance

Waste rock excavated from the mine will be loaded in dump trucks and stored in the overburden stockpile (mine waste dump). The pile of waste rock will be spread and compacted by dozers and compactors stationed in the area.

Dumping areas will be filled from the bottom upwards to ensure that the materials are properly compacted. The waste dump will be raised in 5 m thick lifts in a 45-degree bench angle (1H: 1V). Each lift will have a 5-meter berm, to maintain an overall slope of 26.6 degrees (2H: 1V). In cases where possible, the topsoil recovered from the mine will we stored in a separate area and will be recovered for mine rehabilitation purposes.

17 RECOVERY METHODS

The process flow diagram for the Balabag Gold-Silver Plant can be viewed in Figure 17-1. A detailed process flow diagram can be seen in **Appendix II-D**. The mass balances for each major process route for the treatment of gold and silver for the Balabag project are presented in **Appendix II-F**. Plant water balance and requirement can be viewed in **Appendix II-G**.

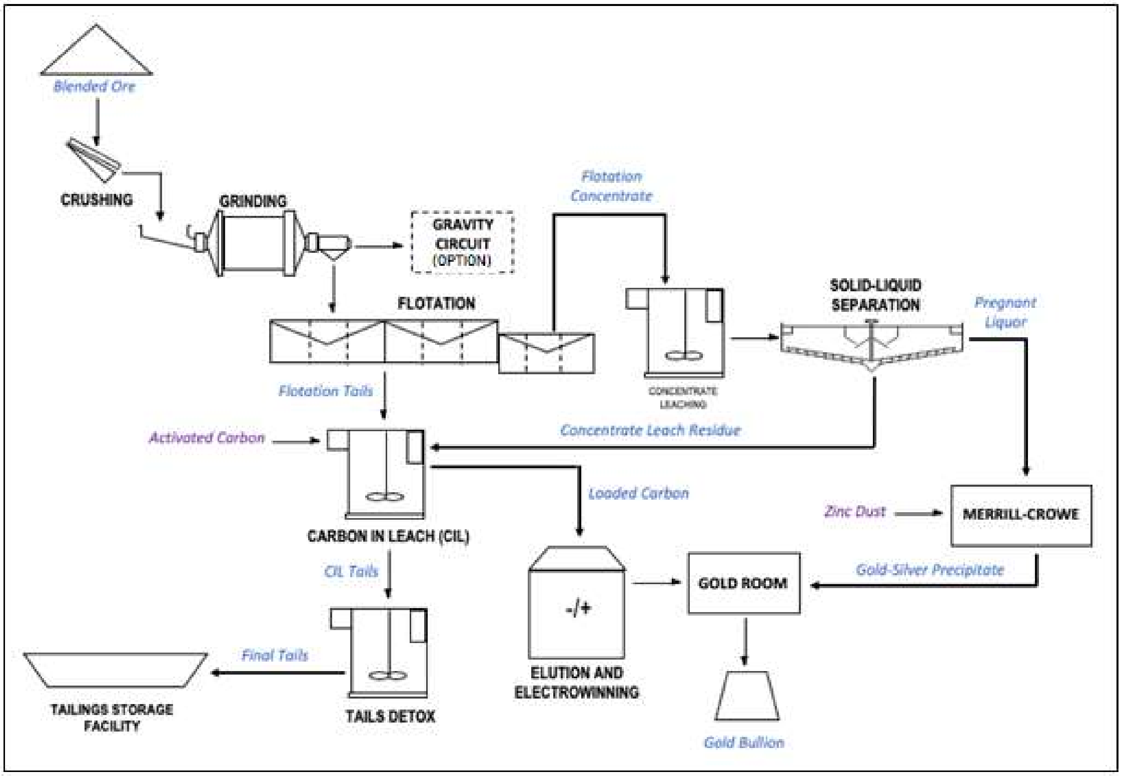


Figure 17-1. General Process Flow Diagram

The Balabag gold and silver project was designed to run at a capacity of 2,000 t/d. An availability factor of 95% was applied to the mill plant; except for the crushing circuit wherein 67% was applied. Table 17-1 shows the major components for the process design criteria (PDC) from crushing to the CIL process.

The complete list of the PDC is presented in **Appendix II-H**.
Table 17-2 shows the summary of recoveries used in the development of the mass balance.

The plant consists of a hybrid circuit of flotation, Merrill-Crowe, and CIL processes. Presented in Table 17-3 is the production schedule with the respective gold and silver recoveries.

The company will be embarking on an exploration program during operations in areas within the MPSA that are seen to have significant resource potential. It is anticipated that more ore will be developed in these continuous exploration program, thereby extending the mine life beyond four years.

Table 17-1 Process Design Criteria

DESCRIPTION	DESIGN
Crushing	
Primary Feed F80, mm	450
Primary Product P80, mm	100
Secondary Product P80, mm	50
Crushing work index, kWh/t	9.5
SAG Mill Grinding	
Feed top size, mm	100
Product P80, um	220
Bond Work Index, kWh/t	13.6
Ball Mill Grinding	
Feed F80, um	220
Product P80, um	80
Ball Mill Work Index, kWh/t	12.1
Bulk Flotation	
Flotation Time, min	12
% Solids	36
Concentrate Leaching	
Leach Time, h	56
% Solids	33
Merrill-Crowe	
Capacity, m^3/h	20
Carbon-in-Leach	
Leach Time, h	24
% Solids	40

Table 17-2 Mass Balance (For Year 2)

Stream		Grade, g/t		Recovery, %	
	t/d	Au	Ag	Au	Ag
Feed (to Mill)	2,000	2.46	61.96	100.0	100.0
Tails (from CIL discharge)	2,000	0.30	8.00	12.2	12.9
Bullion (from Gold Room)	0.003			87.8	87.1

Table 17-3 Plant Capacity and Production Schedule

Year	Annual Tonnes Milled	Feed		Recovery		Production	
		g/t Au	g/t Ag	Au	Ag	oz Au	oz Ag
1	410,000	2.45	90.23	84.5	80.4	27,333	955,666
2	700,000	2.46	61.96	87.8	87.1	48,692	1,214,301
3	290,000	2.64	52.37	88.6	84.7	21,812	413,307
Total	**1,400,000**					**97,836**	**2,583,674**

17.1 Plant Operations Description

The Balabag gold and silver plant consists of a two-stage crushing circuit utilizing primary and secondary jaw crushers to produce a coarse stockpile.

Crushing is followed by a two-stage grinding of SAG mill and ball mill closed circuit. A bleed from the secondary cyclone is processed for the recovery of free gold by gravity concentration and sent to the gold room for smelting.

Bulk flotation is done to ground ore to produce silver concentrate at 80% recovery. The silver and gold from the flotation concentrate is dissolved by cyanidation process. Pregnant solution containing the gold and silver is recovered by two-stage dewatering, a combination of high-rate thickener and filter press; then forwarded to the Merrill-Crowe circuit for gold and silver precipitation using zinc dust.

Tailings from bulk flotation and repulped flotation concentrate leach residue is processed in carbon-in-leach (CIL) tanks to extract and simultaneously recovery of residual gold and silver by cyanidation and carbon adsorption. Loaded carbon are stripped. The precious metals are then recovered as sludge through electrowinning.

The final tailings are then treated in the detoxification process for cyanide destruction.

Descriptions of the unit operation and concentration processes for the Balabag gold and silver plant are discussed in the succeeding subsections:

17.1.1 Crushing

Run-of-mine (ROM) ore with top size at 450 mm are transported by haul trucks and dumped to a stationary grizzly. Crushing is done in a two-stage circuit using jaw crushers with a feed rate of 125 t/h and a target product size of 50 mm.

Oversize feed materials undergo manual breaking while the undersize are contained in a 40 m^3 capacity ore bin. Using a variable-speed apron feeder, the coarse ore are dumped to a vibrating grizzly with a nominal 50 mm aperture. The vibrating grizzly oversize are fed to a 7 m x 10 m primary jaw crusher operating at 100 mm closed-side setting. The primary crushed product and the vibrating grizzly undersize are then dumped to a 1.2 m wide wearing conveyor going to 1.5 m x 3.6 m single deck vibrating screen. The oversize material, meanwhile, are fed into a 254 mm x 1320 mm (10”x52”) secondary jaw crusher at 50 mm closed-side setting. The crushed product of the secondary jaw crusher and undersize of the vibrating screen are then dumped to a 762 mm wide stockpile conveyor and subsequently transported to the coarse ore stockpile area.

17.1.2 Grinding

Grinding is done in a closed-circuit SAG and ball mills system wherein coarse ore at 50 mm nominal size are conveyed to the mill at a rate of 88t/h through a 610 mm wide belt feeder equipped with variable speed drive and into a 762 m wide feed conveyor with a weightometer.

Primary grinding utilizes a 3.8 m x 4.9 m grate discharge SAG mill. It operates at 75% solids, 300% circulating load and at 35% by volume ball loading to produce a product with 220 um P_{80}. SAG milling requires 23 m^3/h of dilution water in the feed and discharge to the primary hopper where an additional 78 m^3/h of water is added from the trommel spray and as a dilution water to achieve the 65% solids primary cyclone feed.

The SAG mill product is pumped to a cluster of 254 mm diameter cyclone using a 203 mm x 152 mm (8” x 6”) horizontal centrifugal pump with installed VFD. The primary cyclone overflow is then discharged to the secondary ball mill hopper and mixed together with the secondary ball mill discharge. A dilution water of 9.5 m^3/h is added to the hopper to achieve a 55% solid secondary cyclone feed. This is then pumped to a cluster of 254 mm diameter cyclones using a 203 mm x 152 mm (8” x 6”) horizontal centrifugal pump.

The secondary cyclone underflow is equally distributed to two secondary ball mills which operates at 62% solids and 250% circulating load to produce a product with 80 μm P_{80}.

17.1.3 Gravity Concentration Option

The gravity concentration circuit is being considered as an option but will not be installed initially. However, space is being allowed in the event that one is built.

It will consist of a vibrating screen with 600 μm aperture, a centrifugal gravity concentrator and two units of shaking tables. An estimated 25 m^3/h of fresh water is required for gravity concentrator fluidization. Tailings from the circuit will be returned to the secondary ball mill hopper. Gold and silver concentrates are then dried, fluxed, and smelted in the gold room.

17.1.4 Flotation

The secondary hydrocyclone overflow at 30% solids passes through a 2.6 m x 3.0 m conditioning tank for the addition of collector and frother at required dosages; and pulp alkalinity adjustment to pH 11 through the addition of lime. The slurry is then fed to a series of six 16 m^3 and three 8 m^3 capacity U-tank cells for bulk flotation. The tailings then go to a 9-m diameter thickener as feed to the CIL circuit. The high silver concentrate produced is then dewatered in a 5-m diameter thickener and fed to the concentrate leaching circuit. Total water requirement for this section amounts to 66 m^3/h of process water. The volumes of process water recovered from the overflow of the two thickeners sum up to 120 m^3/h.

17.1.5 Flotation Concentrate Leaching

From the flotation concentrate thickener underflow pump, thickened concentrate (9 t/h) at 33% solids is transferred to 6.1 m x 7.3 m aeration tank where lime is added to increase the pH to around 10.5-11.0 and 0.03 m^3/h raw cyanide solution. Leaching is conducted in six units of 6.1 m x 7.3 m agitated tanks for a total of 56 h. Make-up cyanide is dosed at the first leach tank to maintain optimum cyanide strength in the solution. All leach tanks are installed with a bypass system to enable changes in the flow of materials when necessary. Oxygen, an essential requirement to the leaching process, is supplied by low pressure, high volume, and oil-free air compressors. The leach solution is initially recovered from a 3.6 m diameter leach tailings thickener to a 3.9 m x 4.7 m unclarified pregnant solution tank. The underflow, with 60% solids, is pumped to a plate and frame filter press to recover more pregnant leach solution, which is then pumped to the unclarified solution tank. The filter cake containing 10% moisture is then repulped and pumped to the CIL circuit.

17.1.6 Merrill-Crowe Precipitation

The unclarified pregnant solution recovered from two-stage dewatering system is treated in the Merrill-Crowe Plant. The solution is pumped into two clarifiers/filters operated in parallel with one operating and one on standby so that desired process flow rate is maintained and the clarifier to be cleaned can be taken offline. The clarified pregnant solution is then deaerated to below 2 ppm O_2 in a vacuum tower. The deaerated pregnant solution is then reacted with zinc dust using a screw feeder.

A plate and frame filter press is used to dewater the precipitate. The barren solution is recycled and pumped to raw cyanide water tank for dilution purposes. The zinc precipitate containing the gold and silver will then be smelted to doré bars.

17.1.7 Carbon-in-Leach (CIL)

The bulk flotation tailings and the residue cake from the filter press are treated in the CIL tanks to further collect the gold and silver. The underflow from the tails thickener at 45% solids is pumped to a 9.3 m x 11.6 m diameter aeration tank with lime added to increase the pH to 10.5-11.0.

CIL will be conducted in six units of 9.3 m x 11.6 m agitated tanks for a total of 24 h leaching time. Cyanide is dosed in the first CIL tank to maintain strength of 0.07% CN in the solution. All CIL tanks are installed with a bypass system to enable change in the flow of materials when necessary. Oxygen is supplied by low pressure, high volume air compressors.

Carbon adsorption is done at 20 g/L carbon concentration; carbon loading is designed at approximately 5,000 g/t gold and silver. Fresh carbon is added in the last CIL tank and will advance in countercurrent flow by transfer pumps to the first CIL tank where it is harvested. The overflow discharge from each CIL tank flows by gravity to the next tank though wedge wire screens. A vibrating carbon safety screen with 500 µm aperture is placed after the last CIL tank. Any carbon recovered from the safety screen is leached in a separate tank, where the pregnant solution produced is added to the pregnant solution tank. The screen undersize flows by gravity to the cyanide destruction circuit.

A nominal 10 t/d carbon is harvested by a transfer pump from the first CIL tank to a carbon recovery screen with 300 µm aperture. The screen underflow, slurry and spray wash water returns to the first CIL tank. The screen overflow product (loaded carbon) will go to a 1.5 m diameter by 12.0 m high acid wash column.

17.1.8 Stripping and Electrowinning

A total of 10 t/d loaded carbon is transferred directly to an acid wash column. Acid washing with 3% (w/v) hydrochloric acid (HCl) follows upon complete transfer of the loaded carbon. The flow of acid wash solution continues into the column until the carbon bed is fully soaked. Acid washing is conducted to remove any acid-soluble deposits and open the pores of the loaded carbon prior to stripping. To ensure complete reaction of the acid with the carbon, the solution in the column is circulated for a period of 30 minutes. A water wash stage follows to thoroughly remove residual acid solution from the carbon and then sodium hydroxide (NaOH) solution wash to ensure no formation of hydrogen cyanide (HCN) gas occurs in the succeeding stages. Caution should be observed as HCN gas is formed in a brief period of time during the washing stage.

The washed carbon is then transferred by an eductor to a 1.5 m diameter by 12.0 m stripping column. The transfer water is flushed and recovered into the HCl and NaOH mixing tank. Strip solution at strength of 3.0% NaOH and 1.0% NaCN is then pumped from the eluant tank through a heat exchanger, solution heater, and back to the tank in a closed loop until the temperature rises to 120 °C. After meeting the required temperature, the stripping solution is then circulated from the solution heater to the stripping column; then to six electrowinning cells that require 3,352 A of current from a rectifier with a rating of 0-10 V and 1000 A/cell. The solution then returns to the eluant tank. Each cell has 11 cathodes and 12 anodes. The cathodes consist of stainless steel wires onto which the gold and silver are deposited in their metallic form. The anodes, on the other hand, are expanded stainless steel mesh plates. These electrodes are located alternately across the cells. The flow of solution is in transverse direction to the electrodes. A direct current voltage is applied between the electrodes from a rectifier.

The electrowinning cells are connected in two parallel lines. Three cells are in each line to maximize the removal of metal from the solution in each pass. The deposited gold and silver are then manually harvested from the cells after 15 h of cycle time. A high pressure, low volume water spray detaches the deposited metal from the cathode towards the bottom of the cell. The collected metal is then washed into a sludge collection hopper. The dilute sludge is then dewatered in a plate and frame pressure filter and subsequently dried in an oven. The dried sludge is then mixed with appropriate quantities of fluxes prior to charging into the smelting furnace.

The barren carbon is discharged from the stripping columns through a carbon eductor going to kiln feed hopper, where carbon fines and water are drained. Spent carbon are then reactivated utilizing a 10 t capacity vertical kiln. Reactivated carbon is discharged into a quench tank before they get recycled back to the carbon adsorption circuit.

17.1.9 Detoxification

The detoxification process is utilized for the purpose of cyanide destruction prior to disposal of effluent. About 153.1 m^3/h of barren pulp from the carbon adsorption stage are treated in two 7.3 m x 9.1 m detoxification tanks at approximately 42% solids. Sodium metabisulfide (SMBS) act as the source of sulfur, while copper sulfate ($CuSO_4$) serves as a catalyst. These are the main reagents used for the treatment. Lime is used for pulp pH adjustment. The total retention time is 1.5 h. Aside from the pH of the pulp, other critical parameters in this process are conductivity and dissolved oxygen (>3ppm). The discharge of the detoxification process should meet the DENR standard for effluents. This will be the final tails of the plant discharged to the tailings dam. Ultraviolet rays and dilution with rain further helps in the natural degradation of CN in the tailings pond.

17.1.10 Refinery

The products of the three concentration steps are combined and processed in the refinery. A fuel-fired crucible furnace operating up to 1,100 °C is used to melt the concentrates at 3 h processing time. Doré bars are casted into moulds designed to contain about 20-30 kg per ingot. Crated doré ingots are stored in the doré vault for safekeeping prior to shipment.

The doré ingots are transported to Cebu via helicopter. The custody of the products depends on the provisions of the selling contract. A similar approach was used in TVI's Canatuan Gold and Silver operations.

18 PROJECT INFRASTRUCTURE

Error! Reference source not found. shows a general overview of the overall Balabag infrastructure site. The list of process plant equipment for the Balabag project and the power requirement for the plant operation are presented in **Appendix II-I.**

The proposed process plant layout and other infrastructure as provided by 360-Global can be viewed in **Appendix II-E**. The drawing below shows the following onsite infrastructures:

A. Administration Buildings, which houses the Main Administration, Accounting, Environmental, Public Affairs, Security, and Health and Safety.
B. Warehouse Facilities.
C. Laboratory Facilities
   - Metallurgical laboratory
   - Assay laboratory
D. Site Access Roads

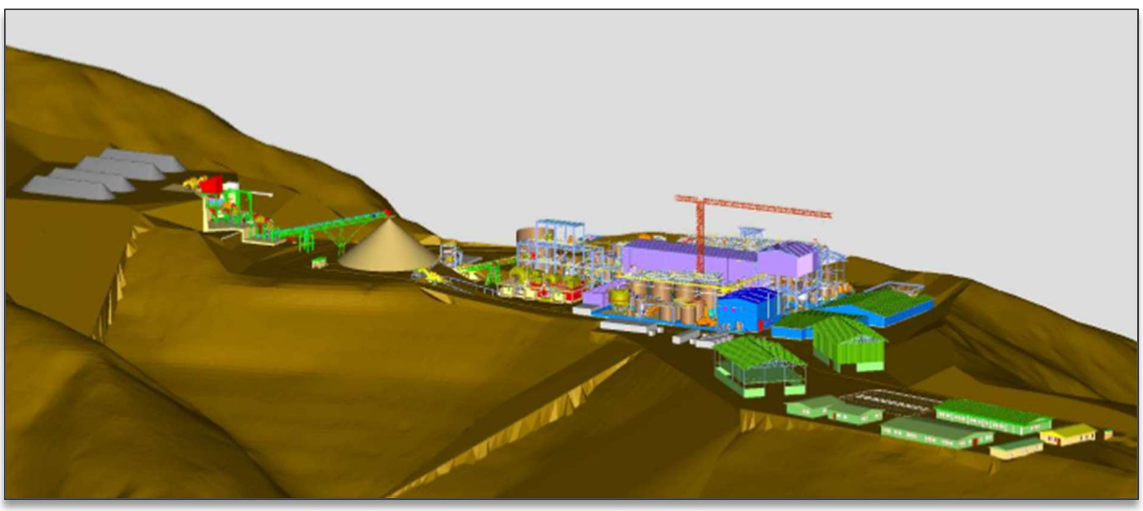


Figure 18-1 Proposed Onsite Infrastructures

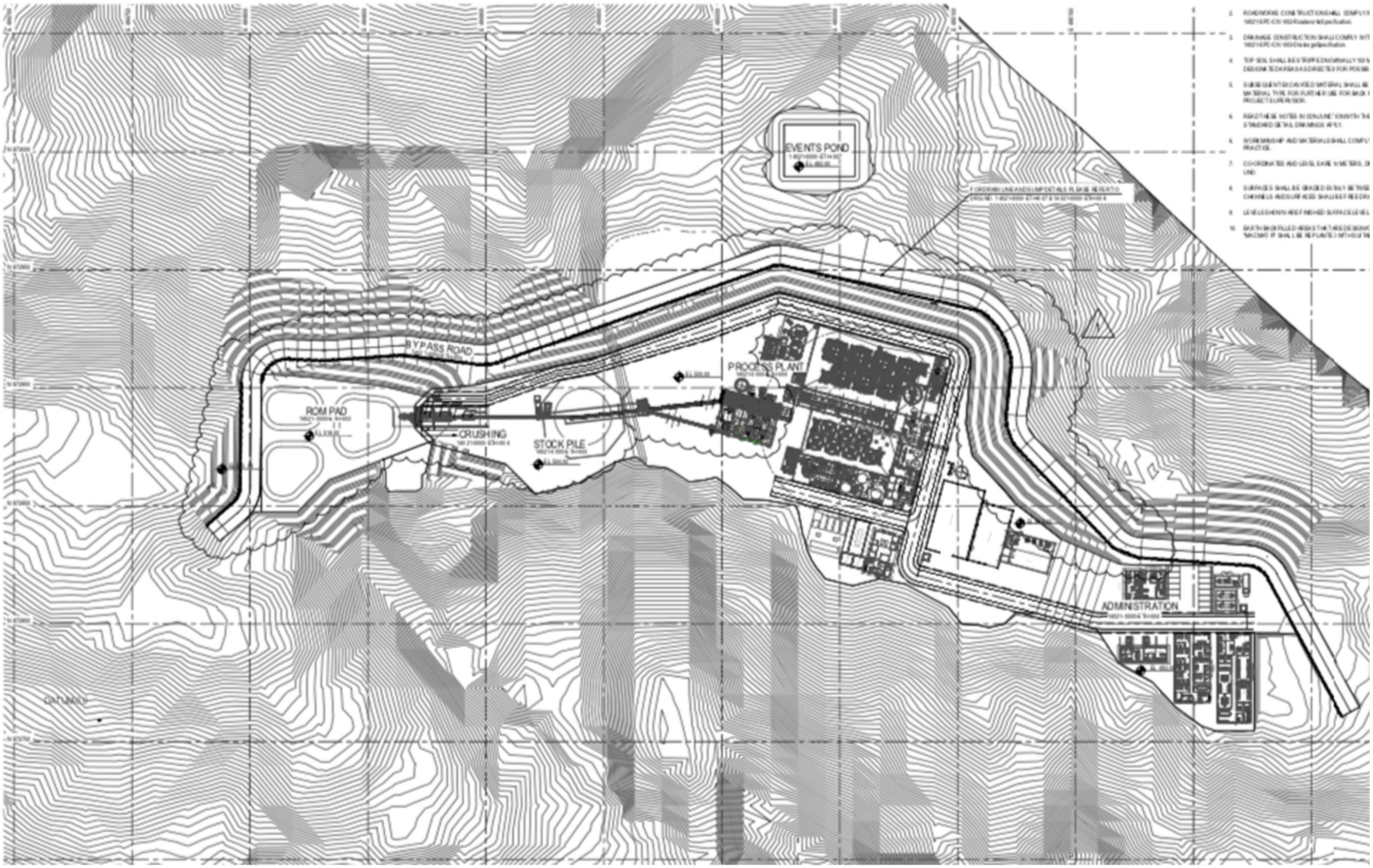


Figure 18-2 Process Plan Key Plan

19 MARKET STUDIES AND CONTRACTS

19.1 Market study

Generally, the traditional models and theories of supply and demand are not applicable to gold because this is not a consumable commodity. The concepts of deficits or excesses do not and cannot affect the market price of gold. On a marketing perspective, the commodity is basically moved from a source stockpile (the mining company) to another stockpile which is the investor. In essence, nothing has been gained or lost in terms of the commodity itself.

Gold hit an all-time nominal high of USD 1910/oz on August 22, 2011. This was during a time of financial turmoil amid speculation that the global economy was slowing. The US Federal Reserve had embarked on two rounds of quantitative easing (QE) by buying more than USD1 trilling worth of treasury securities by the end of Q2/2011. Gold was further supported in late 2012 with QE3 but in June 2013, the Feds announced a “tapering” of some if its QE policies based on signs of US economic recovery. This was when gold began its slide and has since declined by 37% to settle around ~USD 1300/oz amidst speculation that the Feds will hike interest rates.

10 Year Gold London Fix PM Daily with 60 and 200-day moving averages

Figure 19-1 Gold Price Ten-Year History

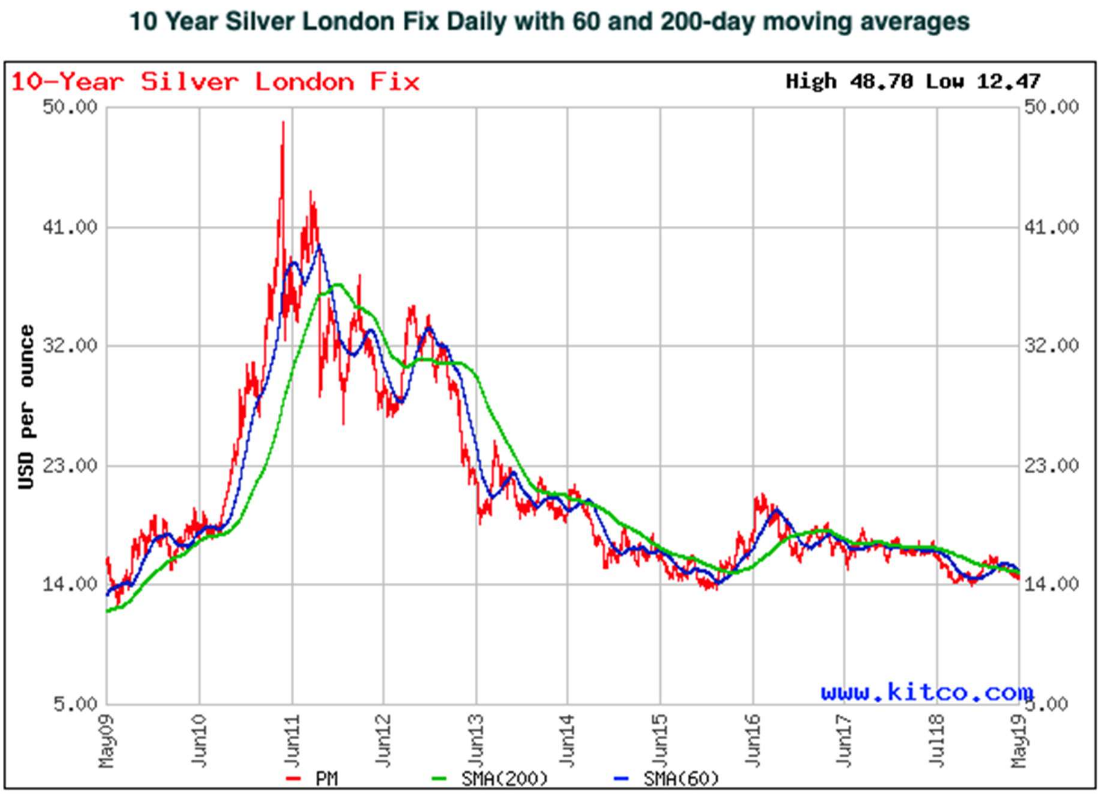


Figure 19-2 Silver Price Ten-Year History

The bases of the analysis are extracts from the World Gold Council (WGC), a leading industry resource for data and opinion on worldwide gold demand.

Key Price Drivers for Gold (https://www.gold.org/goldhub/research/outlook-2019):

1. *Financial Market Instability*

   Globally, there were net positive flows into gold-backed ETFs in 2018. While North American finds suffered significant outflows in Q2 and Q3, this trend started to shift in Q4 as risks intensified

   It is believed that in 2019 global investors will continue to favour gold as an effective diversifier and hedge against systematic risk. Higher levels of risk and uncertainty on multiple global metrics are seen:

   - Expensive valuations and higher market volatility
   - Political and economic instability in Europe
   - Potential higher inflation from protectionist policies
   - Increased likelihood of a global recession.

2. *The Impact of Rates and the Dollar*

   While market risk will likely remain high, two factors could limit gold's upside: higher interest rates and US dollar strength.

   Higher US interest rates alone are not enough to deter investors from buying gold, as seen between 2004 and 2007 or 2016 and the early part of 2018. And while higher interest rates combined with a strong dollar can dampen gold's performance, there are reasons to believe that the upward trend of the US dollar may be losing steam.

3. *Structural Economic Reforms*

   Emerging markets, making up 70% of gold consumer demand, are very relevant to the long-term performance of gold. Among these, India and China stand out.

   These two countries have begun to implement economic changes necessary to promote growth and secure their relevance in the global landscape.

   China's Belt and Road initiative, for example, is focused on promoting regional economic development, boosting commodity markets and upgrading infrastructure.

   India has been active in modernizing its economy, reducing barriers to commerce and promoting fiscal compliance. In fact, India's economy is expected to grow by 7.5% in 2018 and 2019, outpacing most global economies and showing resilience to geopolitical uncertainty.

   Given its unequivocal link to wealth and economic expansion, it is believed that gold is well poised to benefit from these initiatives. It is also believed that gold jewelry demand will strengthen in 2019 if sentiment is positive, will increase marginally should uncertainty remain.

   Similarly, efforts to promote economic growth in western markets are expected to result in positive consumer demand, as has been observed generally in the US since 2012

19.1.1 Supply

In generating supply, gold mining companies operate on every continent of the globe. This broad geographical dispersal means that issues, political or otherwise, in any single region are unlikely to impact the supply of gold. Beyond mine production, recycling accounts for around a third of all current supply. In addition, central banks can also contribute to supply should they sell part of their gold reserves. It is worth noting that after 18 years as net sellers, collectively central banks are now effectively net buyers, causing not only a significant decrease in supply but a corresponding, simultaneous increase in demand.

The defined Balabag reserves of 170,000 ounces are insignificant to the overall current supply and global production forecast.

19.1.2 Silver Supply & Demand Fundamentals

In 1900 there were 12 billion ounces of silver in the world. By 1990, the internationally respected commodities research firm CPM Group says that figure had been reduced to around 2.2 billion ounces of silver. Today, that figure has fallen to less than 1 billion ounces in above ground refined silver. As opposed to gold, it is estimated that more than 90% of all the silver that has ever been mined has been consumed by the global photography, technology, medical, defense and electronics industries.

The amount of above ground refined silver is projected to shrink to even lower levels in the coming years. Industrial demand has been outstripping mining supply for most of the last 20 years, driving above ground supply to historically low levels. Few in the investment world are aware of this important fact.

Silver production has been flat in recent years while demand has been increasing. This hasn't resulted in significantly higher prices yet because the world has been able to fill the gap from inventories and official government stockpiles

The decline in refined silver stocks, from around 2.2 billion ounces in 1990 to around 300 million ounces today means that silver stocks are near an all-time low. Very importantly, silver is very unusual as its supply is inelastic.

This means that silver production will not ramp up significantly if the silver price goes up. Supply didn't increase significantly in the 1970s when silver rose more than 35-fold in price – from USD 1.40/oz in 1971 to a high of nearly USD 50/oz in 1980. Importantly, silver is a byproduct metal and some 80% of mined silver is a byproduct of base metals. Higher prices for silver will not cause copper, nickel, zinc, lead or other base metal miners to increase their production. In the event of a global stagflationary or deflationary slowdown, demand for base metals would likely fall thus further decreasing the supply of mined silver.

There are only a handful of pure silver mines remaining – many with depleting reserves. This inflexible supply means that we cannot expect significant mine supply to depress the price after silver rises in price. It is extremely rare to find a good, service, commodity or investment that is price inelastic in both supply and demand. This is another powerfully bullish aspect unique to silver.

19.1.3 Increasing Industrial Demand

Industrial applications for silver have always been significant. Silver is used in film, mirrors, batteries, medical devices, electrical appliances such as fridges, toasters, washing machines and uses have expanded to include cell phones, flat screen televisions and many other modern devices. It is important to note that silver is heavily used in industry more than gold because the latter has much higher value.

Increasing industrial demand for silver is forecast due to economic growth in China, India, Vietnam, Russia, Brazil and other emerging economies in South America, the Middle East and Asia. This is also known as the 'healthy metal' and has many and increasing medical applications.

Silver is unique in terms of being both a monetary and an industrial metal. Figure 19-3 shows the 5-year historical silver price which indicates a sustained increase.

Figure 19-3 5-Year Historical Price of Silver

19.2 Price Forecasts for Purposes of this Feasibility Study

For the base case, the metal prices used were USD 1250/oz gold and USD 15/oz silver. Sensitivity analysis shows that the Project is sensitive to gold price but would still generate cash and positive NPV even if the price is reduced by 10%.

19.3 Availability of Raw Materials, Fuel, Power and other Service Facilities

TVIRD has operated a medium-sized commercial gold mine/plant at Canatuan, Siocon, Zamboanga del Norte, in the near proximity of the Balabag project. Logistics for the major consumables such as reagents, steel and fuel are easily available from either Zamboanga City, Pagadian City or Manila. Interisland shipping lines are available in regularity to nearby ports such as Zamboanga City, Pagadian City and Dipolog.

Power for the plant in remote places such as Balabag is initially self-generated with the possibility of connecting to the national grid. There are adequate local freshwater supply sources to support the water requirements of the plant, households and the project's auxiliary operations.

19.4 Availability of Technical and Skilled Workers

Balabag is currently populated by local natives and artisanal miners which are expected to provide the pool for skilled labor force. This has been the successful experience at Canatuan and this model will be replicated in Balabag. TVIRD has the advantage of operating the Canatuan CIP/CIL gold plant from 2004-2008 that it has developed a pool of not only skilled workers but highly competent technical personnel and managers.

20 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL IMPACT

20.1 Environmental Management Program

The overall environmental management program consists of environmental activities as identified in the Environmental Protection and Enhancement Program (EPEP) and the Fine Mine Rehabilitation and Decommissioning Plan (FMRDP). Impacts and control strategies used to identify and define the activity plans during the operations period are based on the Environmental Impact Study (EIS) previously approved by the Environmental Management Bureau (EMB). The Environmental Compliance Certificate (ECC) includes several environmental management conditions that were also included in preparation of the EPEP and the FMRDP.

The ECC was issued on October 1, 2013 with Certificate Number ECC-CO-1301-0004. Subsequent to this, the overall Project EPEP and FMRDP were prepared and submitted to the Region 9 Mine Rehabilitation Fund Committee for review. The Region 9 MRFC for the Project was assembled and convened on July 18, 2014. Presentations and discussion of the EPEP and FMRDP were held with the MRFC on August 7, 2014 in Pagadian City. The documents were subsequently endorsed to the Contingent Liability Rehabilitation Fund Committee (CLRF) in August 11, 2014. A presentation and review of the EPEP and FMRDP with the CLRF was held on December 11, 2014. Some revisions and additional information were requested by the CLRF. These were primarily focused on providing additional information and details regarding the underground mining aspects of the operations.

Since that time, the Project was re-evaluated and the proposed underground mining operations were removed from the mining plan. The EPEP and FMRDP were subsequently revised to reflect this change in operations. The revised version of the two documents is included in the overall DMPF package. Other comments and revisions requested by the CLRF have also been incorporated in both the EPEP and the FMRDP.

Since then, the Project was issued a and Environmental Clearance Certificate, a Declaration of Mining Feasibility, and finally a Notice to Proceed in 2018.

20.1.1 Environmental Impacts and Control Strategies

The key environmental impacts of the Balabag Project and consequent control strategies are focused on five environmental management sectors; land resources, water resources, air quality, noise quality and conservation values. The impacts within these five sectors will be limited to the 180-hectare impact area referenced in the ECC. Within this area, approximately

89.5 hectares will be disturbed by infrastructure construction and operations activities. This amounts to approximately 50% of the ECC designated Project area. The disturbed area distribution is shown in Table 20-1 and the disturbed areas are shown on Figure 20-1. Although the current reserves is good for 3 years, it a 7-year progressive rehabilitation program has been anticipated. This schedule of area disturbance is shown in Table 20-2.

Table 20-1 Projected Disturbed Area and Distribution

Sector	Area (ha)	Area (%)	Sector	Area (ha)	Area (%)
Mill and Processing Plant	3.0	4%	Housing and Camp Facilities	2.5	3%
Tailings Storage Facility	20.5	23%	Roads and Access	12.0	13%
Surface Mine Area	27.1	30%	Other Facilities	0.0	0%
Waste Rock and Overburden Disposal	24.4	27%	Total	89.5	100%

Notes:
1. Waste Rock and Overburden Disposal Areas are divided into four distinct disposal areas.
2. Tailings Storage Facility includes the dam embankment, spillway and impoundment.

Table 20-2 Annual Disturbed Area Schedule

Operations Area	Disturbed Area (ha)	Operating Years						
		1	2	3	4	5	6	7
Mill and Processing Plant	3.0	3.0	-	-	-	-	-	-
Tailings Storage Facility Dam and Spillway	8.0	4.0	4.0	-	-	-	-	-
Tailings Storage Facility Impoundment	12.5	3.0	4.0	2.0	1.0	1.0	1.0	0.5
Surface Mine Area	27.1	11.5	5.7	-	2.4	7.5	-	-
Waste Rock/Overburden Disposal Area 1	10.8	5.4	5.4	-	-	-		
Waste Rock/Overburden Disposal Area 2	5.4	-	-	-	-	2.7	2.7	-
Waste Rock/Overburden Disposal Area 3	2.5	-	-	-	-	-	2.5	-
Waste Rock/Overburden Disposal Area 4	5.7	-	-	-	-	-	-	5.7
Housing and Camp Facilities	2.5	2.5	-	-	-	-	-	-
Roads	12.0	7.0	2.0	1.0	-	1.0	1.0	-
Annual Disturbed Area (ha)		36.4	21.1	3.0	3.4	12.2	7.2	6.2
Cumulative Disturbed Area (ha)		36.4	57.5	60.5	63.9	76.1	83.3	89.5

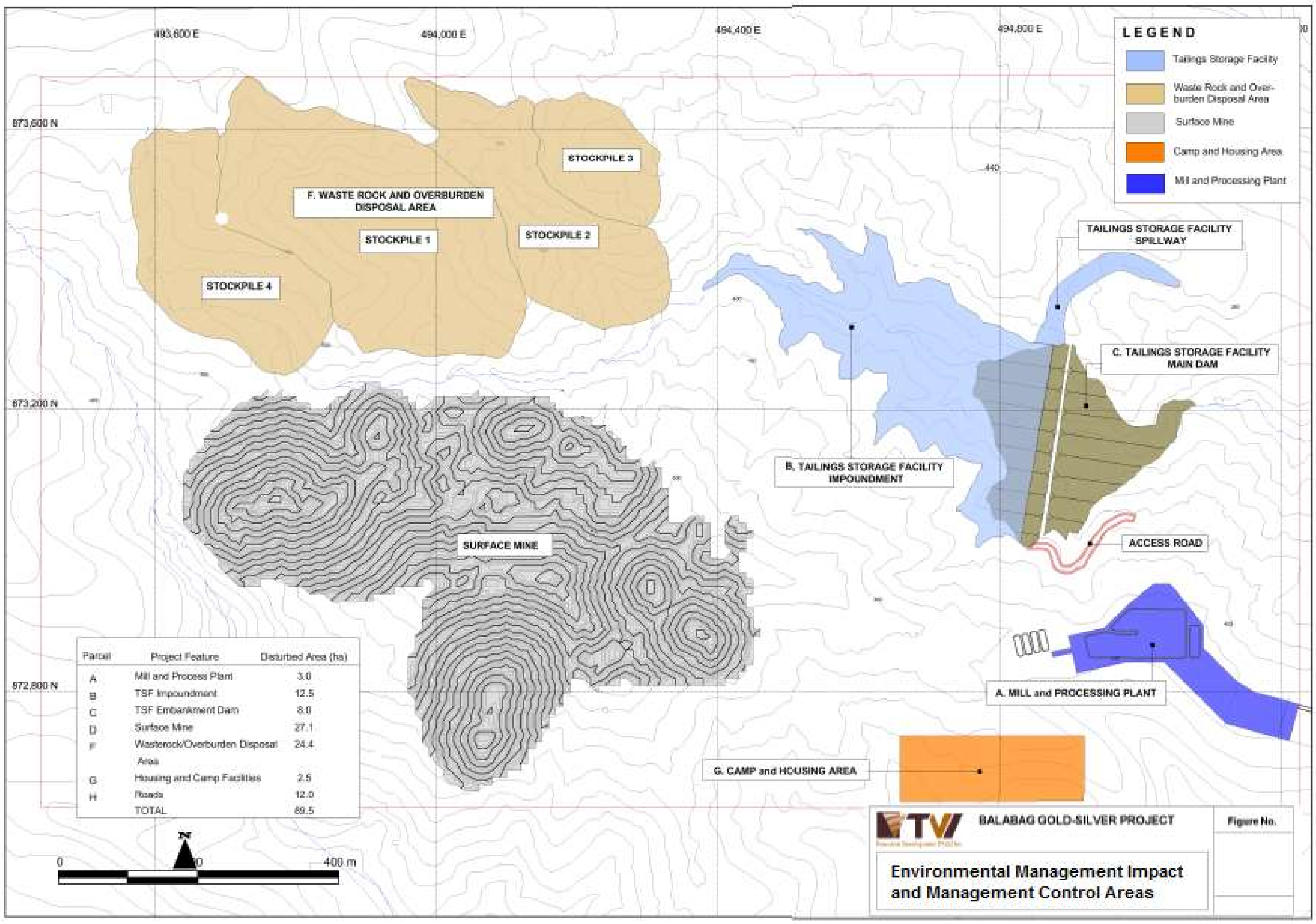


Figure 20-1 Environmental Management Impact and Management Control Areas

To address the potential environmental impacts identified for the Project the EPEP was prepared and submitted to the Region 9 MRFC, the Central Office of the Mines and Geosciences Bureau (MGB) and the CLRF. Annual EPEP's (AEPEP) will be prepared and submitted to the Region 9 MRFC prior to the beginning of each calendar operating year. The documents represent a comprehensive and strategic environmental management plan which will be implemented throughout the life of the operations stage of the Project. Following the end of the operations, the FMRDP will be implemented and will serve as the guide for the decommissioning, rehabilitation and closure of the Project.

An Environmental Monitoring Plan is also included as part of the EPEP and will be included in the AEPEP's as well. The TVIRD Environmental Management Department will be responsible for the self-monitoring programs and will, at times, be assisted by third party consultants and specialists. The multi-sectoral involvement will be primarily associated with the Multi Partite Monitoring Team (MMT) and the Mine Rehabilitation Fund Committee (MRFC). Environmental impacts and control strategies identified for each environmental management sector are summarized in the following subsections. These are based on the plans, programs and analyses developed for the EPEP.

20.1.2 Land Resources

Impacts

Impacts relative to land resources are expected to be the most significant during the operation stage wherein surface mining and processing activities will occur. These impacts include vegetation removal, loss of topsoil and overburden materials, soil erosion, changes in soil quality and fertility and changes in the landscape. These impacts are expected to be reduced as the mining activities move towards closure and the progressive rehabilitation and reforestation activities for the disturbed areas move forward. A summary of the key environmental impacts associated with Land Resources is shown in Table 20-3.

Table 20-3 Key Environmental Impacts for Land Resources

Impact Sector	Key Impacts
Land Use and Land Use Classification	Changes and/or inconsistent land use
	Encroachment in environmentally critical areas
	LGU/Indigenous Peoples land use plans (ADSDPP)
Geology and Geomorphology	Changes in surface landforms and topography
	Changes in slope stability
	Inducement of landslides and/or debris flow events
	Soil erosion
Pedology	Change in soil quality and fertility
Terrestrial Flora and Fauna	Vegetation removal and loss of habitat
	Threat to local flora and fauna species
	Hindrance to wildlife access

Nearly all Project features will result in some form of land resource impacts. The most pronounced will be the Surface Mine, Waste Rock and Overburden Waste Disposal Areas and the Tailings Storage Facility. These features will remain after the end of operations and after the end of the decommissioning and rehabilitation closure period. These impacts will be long term and irreversible.

Control Strategies

Land resource management activities will be implemented during all phases of the Project. During Project development prior to operations, environmental control activities will be focused on vegetation removal and soil excavation. During the operation phase, environmental control activities will be focused on managing and controlling the occurrence of impacts associated with the surface mining operations, waste rock and overburden waste management programs and implementation of progressive rehabilitation activities.

Progressive rehabilitation involves the staged treatment of disturbed areas during the mining operation rather than implementing one-time rehabilitation at the planned mine closure. This reduces the amount of work and cost in implementing structural improvements, slope stability measures and erosion control during closure. It also provides an opportunity for testing rehabilitation practices that may or may not work on the site. This will allow improvements to be made prior to final closure. Visual impacts will also be improved as the disturbed area footprint is reduced.

Objectives of progressive rehabilitation program are focused on slope stability, structural measures for stability and erosion control, revegetation and reforestation, and socioeconomic aspects of the rehabilitated area. These objectives will be attained through implementation of the following programs:

- Structural and nonstructural erosion controls
- Topsoil management and plant nursery operations
- Surface Preparation related to soil conditioning for revegetation and reforestation
- Revegetation related to the type and distribution of plant species
- Maintenance activities to limit plant mortality rates and encourage continued plant growth

The progressive rehabilitation program will be the key control strategy implemented during the operations.

Approximately 30 hectares (34%) of the disturbed area will be subject to progressive rehabilitation during the operations period and prior to the closure period. These areas will focus on the TSF Dam and Spillway area and the Waste Rock and Overburden Disposal Areas. The Progressive Rehabilitation Schedule is shown in

Table 20-4.

Table 20-4 Progressive Rehabilitation Schedule

Operations Area	Disturbed Area (ha)	Operating Years							Prog. Rehab	Mine Closure
		1	2	3	4	5	6	7		
Mill and Processing Plant	3.0	-	-	-	-	-	-	-	-	3.0
Tailings Storage Facility Dam and Spillway	8.0	-	-	4.0	4.0	-	-	-	8.0	-
Tailings Storage Facility Impoundment	12.5	-	-	-	-	-	-	-	-	12.5
Surface Mine Area	27.1	-	-	-	-	-	-	-	-	27.1
Waste Rock/Overburden Disposal Area 1	10.8	-	-	-	-	-	5.4	5.4	10.8	-
Waste Rock/Overburden Disposal Area 2	5.4	-	-	-	-	-	-	5.4	5.4	-
Waste Rock/Overburden Disposal Area 3	2.5	-	-	-	-	-	-	-	-	2.5
Waste Rock/Overburden Disposal Area 4	5.7	-	-	-	-	-	-	-	-	5.7
Housing and Camp Facilities	2.5	-	-	-	-	-	-	-	2.5	-
Roads	12.0	-	1.0	1.0	1.0	1.0	1.0	1.0	6.0	6.0
Annual Progressive Rehabilitation Area (ha)	89.5	-	1.0	5.0	5.0	1.0	6.4	11.8	30.2	59.3

20.1.3 Water Resources

Impacts

Impacts associated with the water resources within and downstream of the Project area are associated with changes in drainage patterns, changes in watershed base flow yields, water resource use and competition, potential water quality degradation and threats to aquatic species and habitat. Potential key impacts are summarized in Table 20-5.

Project features significantly affecting the water resources for the area will also be the Surface Mine, Waste Rock and Overburden Disposal Area as well as the Tailings Storage Facility. The impacts will be short term for the most part and can be mitigated during the operations and closure periods.

Table 20-5 Key Environmental Impacts for Water Resources

Impact Sector	Key Impacts
Hydrology and Hydrogeology	Changes in drainage patterns
	Changes in stream and river hydraulic and fluvial conditions
	Stream flow reductions or increases
	Flooding potential
	Water resource use and competition
	Reduction and or depletion of groundwater resources
	Operating rules for the Tailings Storage Facility
Water Quality	Surface water quality degradation
	Groundwater quality degradation
	Sedimentation of local streams and rivers
Freshwater Ecology	Threats to aquatic species
	Loss of important species
	Loss and/or changes in riparian habitat.

Control Strategies

Control strategies associated with impacts to the water resources of the area focus on changes to the drainage patterns, changes in water yields of the affected watersheds and water quality management. These strategies will involve specific drainage, erosion and sediment control measures including slope and surface stabilization, runoff interception and conveyance and sediment control.

Control strategies for the Waste Rock and Overburden Disposal area relative to water resource management will be incorporated in the engineering designs and will need to be constructed prior to and during placement of the waste materials. These strategies will include diversion canals, filter drains and underdrains necessary to convey surface water runoff and prevent water infiltration to the waste stockpiles.

Water resource control strategies relative to the operation of the surface mine area will be focused on the establishment of stable benches, provision of drainage canals on each bench and provision of sediment control ponds around the Surface Mine area. Four primary sediment ponds will be constructed during the life of the Project to manage and control the surface water runoff from the Surface Mine area. Smaller, short term, sediment ponds will also be constructed as needed for smaller disturbed area environmental management needs.

Water resource impacts from the operation of the Mill and Process Plant Facilities are focused on potential water contamination and water resource use and competition. Potential water contamination issues will be controlled and managed by the provision of secondary containment or bund walls around the mixing and reaction tanks containing hazardous materials. Control strategies relative to impacts to water resource and competition will be managed through the implementation of recycling programs to minimize make-up water requirements.

Control strategies relative to water resources during the construction of the TSF will involve construction of cofferdams and diversions upstream of the dam to maintain dry conditions within the dam footprint for construction. Control strategies for the TSF during operation are a function of the dam design criteria and the impoundment operating protocols. Control strategies related to water quality issues from the TSF will focus on the development of a water quality monitoring program that will cover areas within the immediate Project area and the streams and rivers downstream of the Project area.

20.1.4 Air and Noise

Impacts

Potential air pollution sources during the operation of the project will include emissions from the operation of generator sets for power generation, fumes from chemical reactions in the Mill and Processing Plant processes, emission from transportation exhausts like motorcycles, trucks and heavy equipment as well as dust generation from excavation activities and transport of vehicles.

Potential sources of noise impacts include the crushing and grinding equipment at the Mill and Processing Plant, heavy equipment used during excavation and transport, road vehicles, blasting activities and power generating equipment. These activities will be limited to the immediate Project area except for road maintenance activities. Due to the remote location of the Project, impacts to residents or commercial establishments outside the project area will be minimal if at all. A summary of the key impacts and needs for control strategies is shown in Table 20-6.

Table 20-6 Key Environmental Impacts for Air Quality and Noise

Impact Sector	Key Impacts
Meteorology	Change in local climate
	Change in regional climate
	Contribution or reduction of global greenhouse gases
Air Quality	Airshed impacts
	Local air pollution impacts
Noise	Local noise impacts

Control Strategies

Control strategies for impacts from air pollution will concentrate on dust suppression, vehicle/equipment maintenance and monitoring, and fumes management. Air dispersion modeling will be performed for the Project area before and during Project implementation.

Ventilation systems will be installed to supply fresh air, contain toxic fumes and dilute contaminants and pollutants within the mill and processing areas. Fumes management controls will be designed into the Process Plant equipment and the laboratory. Monitors will be placed within those areas subject to fumes to monitor and record the chemical characteristics. Alarm systems will also be provided and a health and safety program implemented. Fuel emissions will be addressed by providing personal protective equipment to workers and regular maintenance of vehicles and equipment. Dust generation in mine site will be managed by deploying truck sprayers to keep exposed areas moist. Natural ground wetting from rainfall will also help in dust suppression.

Impacts relative to noise will be short term and will be localized in the area of operating equipment. Control strategies will focus on provision of personal protective equipment and restriction of high noise activities to specified times and periods as well as the provision and installation of signs and warnings within areas with increased noise levels. Noise modelling will also be done both prior to and during the operation period.

20.1.5 Conservation Values

Impacts

Impacts and control strategies identified under Conservation Values focus on two sectors; Nature Issues and Visual Aesthetics. Vegetation, wildlife and aquatic ecology impacts are unavoidable impacts of mining but may be considered short term, localized and reversible. These will involve a change in the landscape and surface landforms due to vegetation clearing and stockpiling of waste and overburden materials.

Several threatened plant species were identified within the Project area during the baseline environmental studies. Most are trees forming the major component of the canopy and sub-canopy layer. Impact will be associated with the removal of these and other species as part of the development and operation of the Project.

Several threatened wildlife species were also identified within the project area. The primary impact will be associated with the loss of habitat, threat to the existence of important species and threats to the frequency and abundance of some species. Baseline studies did not indicate a significant abundance of wildlife species. This is likely due to the previous mining, logging and kaingan activities within the Project area.

Impacts to the aquatic resources will be limited due to the lack of aquatic resources. These have been significantly affected by the previous activities within the Project area. There were no fish species present within the Project area during the baseline studies. This is likely the result of the historical small-scale mining activities within the project area. Visual impacts are expected due to the mining activities and permanent alteration of the landforms. While the removal of vegetation will be a short-term impact, removal of soil, rock and topographic changes will be a permanent unavoidable impact. These will be associated with the Surface Mine, Waste Rock and Overburden Waste Area and the Tailings Storage Facility.

Control Strategies

Impact control strategies relative to the flora component of the Nature Sector will be primarily associated with implementation of an aggressive progressive rehabilitation program during the operations period and implementation of the FMRDP at the end of mining operations.
The focus of these activities is based on rapid development of the disturbed areas into stable landforms that are comparable to the natural environment. Development of these landforms will be done in a manner that provides for a beneficial land use supporting a sustainable development program by the various stakeholders within the Project area.

Disturbed areas will be reforested with diverse forest species and intercropping with cash crop species. Selected areas disturbed by the Project development and operations activities will be rehabilitated as cash crop plantation zones. The reforestation program will also incorporate Climate Change Adaptation measures.

Impact control strategies relative to the fauna component of the Nature Sector will also be associated with progressive rehabilitation and reestablishment of wildlife habitat areas. Retention of wildlife corridors will be a focal point during the development and operations period to allow migration of wildlife within the Project area.

Impact control strategies for the aquatic habitat will focus on construction of the Tailings Storage Facility (TSF) and retention of tailings materials within this structure. Water quality treatment programs will be implemented within the Mill and Process Plant to reduce potential pollutant contamination. The discharge of water from the TSF is expected to meet the applicable water quality effluent and stream standards. Visual impact control strategies will be closely tied to the progressive rehabilitation program and reforestation/revegetation of the Project area.

20.2 Environmental Management Monitoring

Environmental management monitoring will occur throughout the development, operations and closure phases of the Project. This will be done internally by the Company and externally through the different government regulatory agencies; MGB and EMB. Third party monitoring will also be done for specific environmental sectors throughout the operations and closure stages. Monitoring by the MMT and reporting of the monitoring results within the MRFC will be done on a quarterly basis once the operations begin. A summary of the monitoring programs is shown in Table 20-7. Details of the program are included in the EPEP.

Table 20-7 Summary of Environmental Monitoring Activities

Monitoring Sector	Monitoring Responsibility			Monitoring Frequency				
	Internal	Agency	3rd Party	Daily	Month	Quarter	Annual	Other
Surface Water Quality	X	X		X		X		
Ground Water Quality	X	X			X	X		
Domestic Use Water Quality	X				X			
Water Resource Supply	X				X			
Meteorology and Hydrology	X			X				
Geotechnical Stability	X		X		X		X	
Infrastructure	X					X		
Tailings Storage Facility	X		X			X	X	
Forest Assessment		X						X
Flora Assessment			X				X	
Fauna Assessment			X				X	
Aquatic Habitat Assessment			X				X	
Air Quality			X				X	
Noise			X				X	
Solid Waste Management	X				X			
Hazardous Waste Management	X		X					X
Community Health	X						X	
Progressive Rehabilitation	X					X		
MMT Events		X				X		
MRFC Events		X				X		
SHES Audit		X				X		
Self-Monitoring Reports	X	X				X		

20.2.1 Residuals Management

Residuals management consists of the disposal and management of the tailings generated by the Mill and processing plant and the waste rock and overburden stripping materials from the mining operations. Based on the mining plans, approximately 1.35 million tonnes of ore will be processed. The amount of waste rock and overburden materials requiring disposal is approximately 13 million tonnes. Discussions of the design parameters and configuration of the disposal facilities are provided below.

A key environmental management issues associated with the residuals management strategies is the potential for acid mine drainage. Acid Base Accounting tests were performed on the tailings and waste rock samples as part of the Tailings Storage Facility design. The studies indicated the tailings and the waste rock are unlikely to be acid generating.

20.2.2 Tailings Storage Facility

A Tailings Storage Facility (TSF) will be constructed for disposal and impoundment of the tailings produced by the Mill and Process Plant. TVIRD engaged the services of an international engineering company, Knight Piesold Consultants, for the design of the TSF and preparation of construction plans and specifications.

The TSF will be located east of the surface mine within the Unao-Unao Creek watershed. It will encompass approximately 20.5 hectares, with a tributary drainage area of approximately 84.5 hectares. Features of the TSF will include a zoned earth embankment dam, impoundment, spillway, and a seepage collection pond. All the tailings will be conveyed from the processing plant to the TSF by pipeline and discharged within the impoundment using a series of pipeline spigots.

The TSF Dam will be 65 meters in height at the centerline with an embankment volume of 1.33 million cubic meters. Construction may be completed in two stages depending on the weather conditions and the operations schedule. Should this occur, the first stage will be approximately 42.5 meters in height. The second stage construction will add an additional 22.5 m to the dam height with a final dam crest elevation of 385 meters. The final spillway crest elevation is at 380.4 meters which results in a 4.6-meter freeboard. The maximum storage capacity at the spillway crest is estimated to be 1.57 million cubic meters.

Based on the mill process studies, the in-place density of the tailings is estimated to be 1.3. This results in a tailings volume of 1.04 million cubic meters. This leaves approximately 530,000 cubic meters available for watershed sediment storage.

Alternatively, the TSF may be constructed in one stage with the same final dam height and elevations identified above. Operations of the mill and discharge of tailings however would not commence until the first stage elevation of the dam is completed.

The dam will be a zoned earth-fill and rock-fill embankment with an overflow spillway located within the left abutment of the dam. Design and operating criteria used to develop the Tailings Storage Facility are based on international standards adopted by the Canadian Dam Association. Geotechnical design was based on a minimum static factor of safety of 1.5. The dynamic design conditions were based on the Maximum Credible Earthquake (MCE) having a horizontal ground acceleration of 0.40 g (approximate intensity of 8.0). The spillway and flood control designs were based on the 100-year flood event during operation and the Probable Maximum Flood (PMF) for post operation. Geotechnical and hydrologic design parameters are summarized in Table 20-8 as well as the dam design features.

Table 20-8 Design Parameters of the Tailings Storage Facility

Design Parameter	Value	Design Parameter	Value
Watershed Area (ha)	84.5	Maximum Dam Height (m)	65
24-hr Rainfall 100 Year Event (mm)	360	Overall Embankment Slope	2h:1v
24-hr Rainfall PMP Event (mm)	1,635	Crest Width (m)	10
Spillway Discharge 100 Year Event m3/sec)	68	Max Storage Volume (million m3)	1.57
Spillway Discharge PMF Event (m3/sec)	245	Tailings Storage (million m3)	1.04
Design Seismic Event MCE (Magnitude)	8.0	Static Factor of Safety	1.59-1.97
Peak Ground Acceleration for MCE	0.4g	Max Deformation under MCE (cm)	7-10

Notes:
1. The parameters shown are for the final dam construction phase.
2. The embankment slope shown is for the upstream and downstream shell.

20.2.3 Waste Rock and Overburden Waste Disposal Area

As indicated, approximately 13 million tonnes of waste rock and overburden materials will be generated during the operations. All of these materials will be placed within the immediate area of the Surface Mine and the TSF. Four areas have been identified for disposal and are all located upstream of the TSF and within the Unao-Unao Creek watershed. Each disposal area will be developed using compacted benches and constructed form the lower elevation to the higher elevation. The overall slope of the waste stockpile will be 2.2 to 2.5 horizontal to 1 vertical. Each bench will have a maximum slope of 1 horizontal to 1 vertical with a maximum lift height of 2.55 meters. The bench width will be 5 meters and constructed with drainage controls and conveyance facilities. The size, capacity and operations period for each are shown in Table 20-9.

Table 20-9 Waste Rock and Overburden Disposal Area Characteristics

Facility Designation	Disturbed Area (ha)	Capacity (tonnes)	Operations Year Disposal (million tonnes)		
			1	2	3
Stockpile No. 1	10.8	8.0 million	3.0	5.0	
Stockpile No. 2	5.4	2.0 million	0.6	1.2	0.2
Stockpile No. 3	2.5	1.0 million		1.0	
Stockpile No. 4	5.7	2.0 million			2.1

A significant impact resulting from the development and operation of the Waste Rock and Overburden Disposal Areas relative to Land Resources will be associated with vegetation removal. Prior to placement of waste rock materials, vegetation will be cleared, and the cut trees will be recovered for future use. A Tree Cutting Permit has been secured from the DENR and tree cutting has commenced. Tree inventory and validation studies by the CENRO and Forest Management Bureau in support of the permit have been completed.

A secondary Land Resource impact will involve a change in topography of the area due to placement of the waste materials. A higher landform will be developed, and a valley landform will be lost. Visual aesthetic impacts will be mitigated once progressive rehabilitation activities commence and reforestation of the area becomes more evident.

Erosion potential, geotechnical stability and safety risk are also impacts that may be anticipated during and after operation. These impacts will depend on the geotechnical characteristics of the material, design specification of the fill placement and benching activities and implementation of the design criteria during operation. Slope stability and erosion potential impacts will be addressed by appropriate engineering protocols prior to and during the actual placement activities.

Erosion and sedimentation from surface water runoff is anticipated during the development and operation of the disposal areas. Using watershed sedimentation data gathered from the Canatuan operations, approximately 750 to 1,500 tonnes of sediment per hectare may be generated annually from the waste disposal areas and other portions of the upstream watershed during the operations period. The sediment will be captured and stored within the Tailings Storage Facility. Over the 7-year operation period the amount of sediment that may be deposited within the TSF is estimated to be on the order of 500,000 tonnes (400,000 cubic meters). This can be accommodated by the remaining storage volume within the TSF after subtracting the tailings deposition. Sedimentation rates are anticipated to be reduced once erosion control facilities are constructed and revegetation activities commence. Sediment control facilities are also planned downstream of the waste rock disposal areas and other disturbed areas within the watershed upstream of the TSF. Sediment recovered from these facilities will be stored and used for topsoil replacement during the progressive rehabilitation and final closure activities.

Control strategies for the Waste Disposal area relative to water resource management is incorporated in the engineering design and will be constructed prior to and during the waste materials placement. These strategies will include diversion canals, filter drains and underdrains necessary to convey surface water runoff and prevent water infiltration to the stockpile.

Underdrains shall consist of a combination of rock and perforated pipe systems placed at the bottom of the fill. The rock materials will be sourced from nearby sources. The underdrain system will be designed to convey the anticipated streamflow from the tributary watershed areas upstream of the disposal area.

As part of the TSF design, a geotechnical analysis of the waste rock and overburden waste stockpiles was done by Knight Piesold. This was based on the waste dump configuration discussed previously. The static factor of safety ranged from 1.5 to 2.0 and the maximum deformation under the MCE was 10 to 50 centimeters.

20.3 Final Mine Rehabilitation and Decommissioning Plan

The Final Mine Rehabilitation and Decommissioning Plan (FMRDP) is included in the EPEP Report submitted to and reviewed by the MGB and the CLRF. This plan identified the activities and programs to be implemented for the decommissioning, rehabilitation, reclamation, monitoring and other environmental management procedures during the mine closure phase.

The current FMRDP covers a 10-year period beginning on the last operating day of the Project. The initial 2-year period is identified as the active closure phase which is then followed by a

2-year passive closure period. All decommissioning and rehabilitation programs will be completed during this 4-year period. The remaining 6 years will focus on care, maintenance and monitoring.

The active phase will focus on decommissioning, reclamation and closure activities. The passive phase will focus on monitoring, maintenance and validation of the viability and long-term sustainability of the rehabilitation programs. The Internal and third-party monitoring programs will continue during all phases with the goal of preparing the Project area for turnover to the concerned stakeholders and to secure a Certificate of Relinquishment of the Project area.

20.3.1 Final Land Use Objectives

The Comprehensive Land Use Plan of the host community identifies the project area as a Mining Reserve area. This will remain the same throughout the operations period. Land use following the mine rehabilitation and closure will focus on supporting and sustaining the terrestrial and aquatic habitat and reinforce long term sustainable livelihood programs initiated during the operations period. This will be done through development of an agriculture-forest land use program. Land use goals are summarized in Table 20-10.

Table 20-10 Post Mine Land Use Objectives

Land Use	Area	Land Use	Area
Plantation- Cash Crops	25.6	Open Areas/Agriculture-Forestry	41.2
Lakes and Ponds	7.6	Special Land Use	0.6
Retained Infrastructure	4.2		
Retained Forest	100.8	Total	180

These land use areas cover the entire 180-hectare Project Area and have been developed to retain as much of the existing forest as possible outside the 89.5-hectare disturbed area. The land uses and identified areas may be revised over time as further input and requests from the stakeholders are discussed and evaluated.

20.3.2 Implementation of the FMRDP

Approximately 34% of the Project area disturbance (30.2 hectares) will undergo progressive rehabilitation during the operations phase. The remaining 73% (59.3 hectares) will undergo decommissioning and reclamation activities during mine closure period.

Each of the areas disturbed during the project development and operations phases have been identified as distinct rehabilitation areas and characterized based on specific factors. These include land use, topography, drainage, progressive rehabilitation programs, decommissioning needs, etc. The plans for rehabilitation and closure were then developed based on these characteristics and community requests. Details of the programs are included in the FMRDP.

20.4 Mine Safety and Health Plan

TVIRD will establish a safety program designed to provide a safe working environment and promote the culture of health and safety among its employees and stakeholders.

A Safety Department will be tasked to oversee the implementation of policies, monitor the day-to-day operations of the Project, and consolidate pertinent related statistics and reports. The department will also be the forefront in conducting trainings and IEC programs for the Project's community.

The safety program will be grounded on the company's policy, that is:

> *"To conduct its operation with good stewardship in the protection of human health and the natural environment, prioritizing the health of its employees and its host community.*
>
> *TVI shall take measures to deal with risks of incidents by diligent application of technically proven and economically feasible protective measures throughout the process, with the aim of not only meeting the normal standards but surpassing what is legislated by law."*

It will adhere to the following general objectives:

1) To promote a culture of safety and health by providing an effective monitoring and investigation system
2) To promote the strict implementation of the Safety and Health Rules
3) To promote Safety and Health training, thereby attaining its objective of total employee development
4) To comply with the various government legislations regarding Safety and Health

Balabag Project's safety program will follow that of Canatuan's effective system, which revolves upon the following:

- Top management and employee involvement – The management takes an active role not only in promoting awareness, but also in implementing safety policies. The Safety Department conducts safety meetings with managers and departments, as well as with the mine contractors, to ensure that policies and information are effectively disseminated in all levels.

- Hazard prevention and control – The company takes engineering and administrative controls to minimize the risks of incident. Employees are provided with personal protective equipment (PPEs). The Safety Department also conducts thorough accident/incident investigation as needed.

- Trainings – The Safety Department conducts various trainings which are designed to promote safety awareness and educate the employees on how to identify and react to unsafe work environment or instances. The programs include (but are not limited to): First Aid, Emergency Preparedness Accident Investigation, Job Hazard Analysis, Occupational and Off-the-job Safety.

Likewise, the company will put up a health program that will service, not only the employees, but also the host and nearby communities. This will involve the establishment of a clinic to be manned by qualified medical personnel. The Company, through its medical team, will promote good health practices, sanitation and hygiene. Annual Physical Examination (APE) shall also be provided to the employees.

20.5 Social Development

Community development programs for the Balabag Project will be carried out through the Social Development Management Program (SDMP), in accordance with the provisions of Republic Act no. 7942 (The Mining Act of 1995) and the DENR Administrative Order no. 2010-21 (The Consolidated DENR Administrative Order for the Implementing Rules and Regulations of R.A. no. 7942).

The Company, through its SDMP, aims for the Project's stakeholders to enjoy sustainable community development through the following objectives.

1) Enhance existing and develop knowledge, values, and skills in support of sustainable community development;
2) Build infrastructures relevant to community needs in partnership with the community and the local government unit;
3) Fill the vacuum in terms of basic social services (health, education, and livelihood), employment, community development, and care for the environment; and
4) Improve the household living condition of the local residents, especially the Indigenous Cultural Community, who became dependent on the small-scale mining industry in the locality.

20.5.1 Coverage

The SDMP will benefit the Project's impact communities, including areas that are (or *will be*) directly or indirectly affected by mining operations, in terms of social, economic, political, and environmental aspects. For this Project, the impact communities have been defined as follows:

- *Host Communities* – are the areas directly affected by the mining operations. For the operation as defined in this Declaration of Mining Project Feasibility (DMPF), the host will be Sitio Balabag of Barangay Depore.

- *Neighboring Communities* – are the barangays that are within proximity of the coverage of the MPSA and primary impact areas of mine operations.

- *Secondary Communities* – are the stakeholders that may experience indirect impact from the Balabag Project. These may include communities at the route of transport, environmental impact areas, the municipal capital, and key areas determined by the Indigenous Cultural Communities (ICC). Secondary barangays must undergo a resource appraisal before the intended projects can be implemented. This is to assure that the programs are sustainable and indicator-driven.

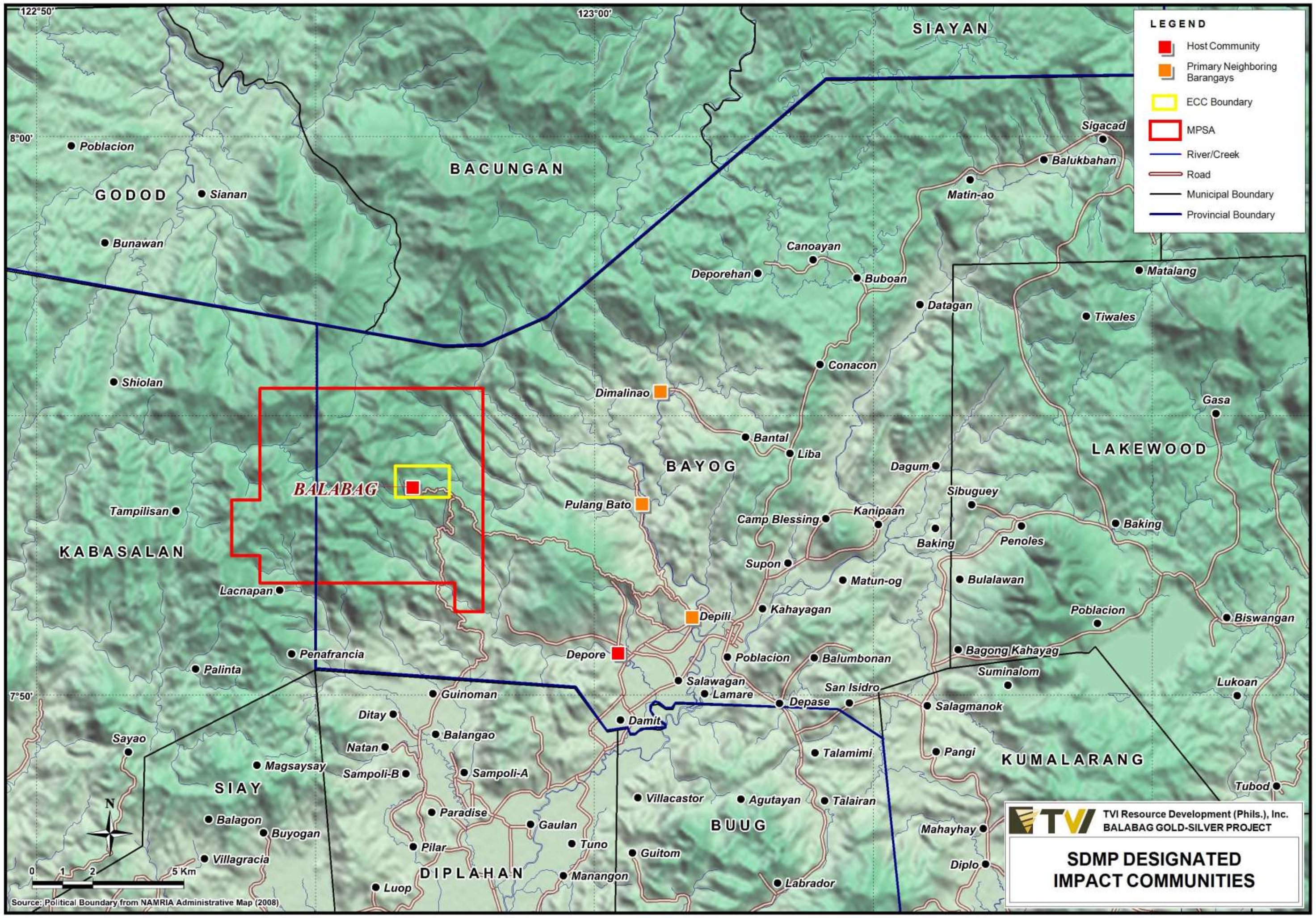


Figure 20-2 Balabag Social Impact Areas

20.5.2 Framework

The formulation of the five-year SDMP utilized an assets-based approach to poverty which allows it to make an inventory of what the poor have rather than what they do not have and look at poverty eradication as a multidimensional process of enabling the poor to acquire, develop, build, utilize, and maintain assets that will enable them to achieve quality life.

This approach, if integrated with appreciative inquiry will focus efforts towards taking stock of successful efforts of the poor, organizations, and institutions that enabled them to acquire, develop, build, utilize, and maintain assets, rather than describing the conditions that promoted deprivation and exclusion. This approach bridges the link between the micro and macro, the individual, cultural, and the structural.

In the promotion and initiation of various development endeavors embedded in the five-year SDMP, the framework for sustainable development will act as the guiding principles in formulating and adopting the strategies to be undertaken as such:

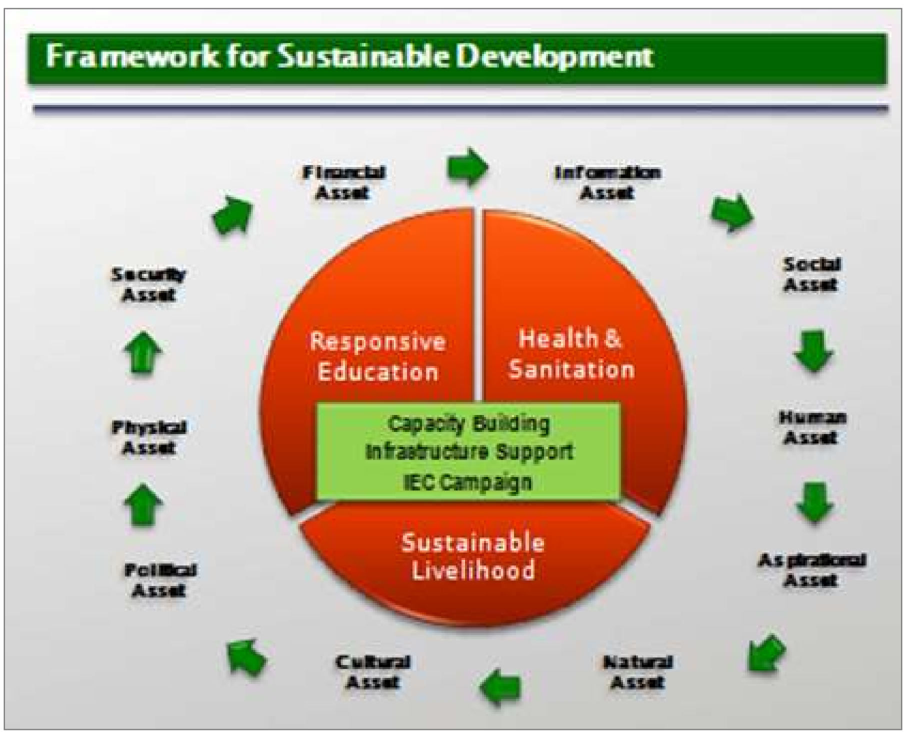


Figure 20-3 TVIRD Social Commitments Framework

The SDMP was prepared with consideration of the communities' development and investment plans. This is to ensure that the programs will be relevant and will have synergy with local government efforts. The Company identified three focus areas to which the social programs and support initiatives will be anchored:

1) **Responsive Education –** TVIRD believes in the principle that education is essential in advancing human rights, gender equity, social justice, and a healthy environment in communities that host the Company's operations. There can be no sustainable future without a responsive and inclusive education system. In implementing its Responsive Education initiatives as a major component of its Areas of Development, TVIRD partners with other sectors of society to ensure that programs for the advancement of education are relevant to beneficiaries.

    - *Goals and Strategies:* Ensure the quality of education by promoting improvement of the educational system in the community, provision of support to the schools and to competent and deserving students of indigent families in the community.

2) **Sustainable Livelihood –** TVIRD believes that livelihood can be sustainable if it can cope with and recover from stress – such as, for instance, the end of the mine life – maintain and enhance its capabilities and provide sustainable opportunities for the next generation.

    - *Goals and Strategies:* Provide sustainable livelihood and increase the household income of through the promotion of agro-forestry programs, sustainable and improved agriculture, and enhanced commerce.

3) **Health and Sanitation –** Human dignity is related to health, which, in turn, is not possible without sanitation. By providing both health and sanitation facilities to host and neighboring communities that previously had no access to these basic services, TVIRD aims to create a positive impact on the well-being and general economic productivity of its beneficiary community.

    - *Goals and Strategies:* Ensure the quality of health conditions in the community through increasing the level of awareness on appropriate health practices and preventive health care programs, through the implementation and/or improvement of existing health programs and skills of health workers; and upgrade the existing facilities and cultivate awareness on the importance of proper nutrition and sanitation practices for all ages.

20.5.3 Support Components to the Areas of Development

TVIRD is also committed to contributing to components outside of three aspects for sustainable development. These are Capacity Building and Infrastructure Support. These components are ingrained in each of the three aspects, holding them together.

***Capacity Building* –** This was established to improve the skills, enhance the capabilities, and develop the potentials of all stakeholders. This is geared towards setting the foundation for the community to be able to sustain development projects even after mine operations ceases.
***Infrastructure Support* –** It accelerates the delivery of all the services to the community to attain a level of development through the improvement and constructions of basic infrastructures that answer the socioeconomic and environmental needs of the communities.

Apart from the "physical" SDMP, a fraction of the funds will also be allotted to the following activities:

1) **Information/Education/Communication (IEC) Programs –** The Company will facilitate IEC programs, which are designed to raise the level of awareness of the community and the popularization of the concept of Responsible Mining and Sustainable Development, through TVIRD's Corporate Social Commitments and Environmental Management.

2) **Development of Mining Technology and Geosciences (DMTG) –** The Department of Environment and Natural Resources (DENR), through its Administrative Order No. 2010-13 (DAO 2010-13), mandates that a fraction of the funds be allocated to DMTG programs. Such may be in the form of, (1) basic and applied research on mining technology and geosciences, (2) provision of scholarship for mining and geosciences-related subject, or (3) providing assistance to institutions which serve as a venue for developing mining technology and geosciences.

20.5.4 Methodology

The main challenge faced by TVIRD's social and economic development programs is sustainability and resiliency. A sustainable community exhibits the following traits:

- ✓ Leaders are accountable, transactions are transparent and community members have a voice in planning and decisions.
- ✓ Basic social services are provided by local institutions, whether tribal or local government.
- ✓ Local projects are effectively and profitably managed by the community organizations.

It has been the thrust of TVIRD, through the CReDO, to wean the host and impact communities in its project areas from the dole-out mentality.

In some cases, a company goes through a “period of appeasement” wherein development interventions are driven by the primary objective of winning the support (“hearts and minds”) of the community members and their leaders. Necessarily, this will entail some compromises on sustainability considerations, amidst the strategic value to the company of responding positively to almost all requests (even demands) made, for fear that the company may jeopardize the tenuous community support. To avoid this cycle of dependence, TVIRD redirected its effort toward specific activities and interventions that are meant to develop community empowerment through participation.

The Participatory Resource Appraisal (PRA) that TVIRD implemented in the formulation of the five-year SDMP triggers a new phase of community development approach that is more focused on sustainability. The CReDO is redefining rules of engagement when it comes to dealing with communities relative to social and livelihood development projects. The PRA driving principles are:

- *Sustainability – the project should be able to continue on its own and managed by the locals*
- *Community participation – project inception, planning, implementation and assessment should be with community participation*
- *Data based, indicator driven – projects should be based on actual community needs as determined by social development indicators*

This TVIRD approach to community development is a departure from dole-out mode as tool for winning the hearts and minds of the community. The idea is that having the communities participate and empowered increases the chance of sustaining the investments made by the programs and projects initiated by the company.

The CReDO organized the conduct of the PRA in the four neighboring communities of the Balabag Project. The team of facilitators was divided into two groups so two barangay activities can be done simultaneously at the same dates.

The PRA for Barangay Depore, the host community, was held 19 to 21 February 2013. Barangay Dipili completed theirs in the same dates but was handled by a separate team. The workshops for the barangays of Pulangbato and Dimalinao were conducted on 23 to 25 February 2013.

Each barangay was represented by personalities from different sectors; teachers, health workers, barangay officials, people’s organizations, and local IP/community leaders. Number of participants in each day for every barangay ranged from 20 to 30 individuals.

Generally, the CReDO saw the community to be very participative since the main purpose of the activity is to capacitate the barangay in formulating their Barangay Development Plan (BDP) through the Participatory Resource Appraisal process. The community is then asked, given their BDP, which organizations can assist them finance the projects that cannot be accommodated by the barangay Internal Revenue Allotment (IRA).

It was revealed through this activity that the communities are already aware of the organizations they can tap to assist them in funding some of their barangay projects. This shows great potential for the four communities as the culture of dependence may be prevented from emerging and developing.

Figure 20-4 SDMP Planning Workshops at Four Primary Impact Communities of the Balabag Project – (A) Brgy. Depore on February 19-23, 2013; (B) Brgy. Dipili on February 19-21, 2013; (C) Brgy. Dimalinao on February 23-25, 2013, and (D) Brgy. Pulangbato on February 23-25, 2013.

21 CAPITAL AND OPERATING COSTS

21.1 Project Capital Costs

The total capital estimate for the project is USD 39.30 million, with an initial capital expenditure of USD 28.40 million. This includes the cost of predevelopment works, mine development and pre-stripping, and the construction costs of the processing, treatment, and tailings storage facilities.

The major capital components of the project are enumerated in the following table:

Table 21-1 Estimate of Capital Cost for the Balabag Mine

CAPEX	UOM		Year 0	Year 1	Year 2	COST (M)
Pre--Development Cost	US$		1,500,000	-	500,000	2,000,000
Mine Development & Pre-Stripping	US$		1,000,000	500,000		1,500,000
Plant & Mill Infrastructure	US$		20,900,000			20,900,000
Tailings Storage Facility	US$		5,000,000	5,000,000		10,000,000
Waste Dump	US$		-	300,000	1,000,000	1,300,000
Channel Drain	US$				1,600,000	1,600,000
Exploration	US$	400,000		1,000,000	1,000,000	2,000,000
Total	**US$**	**400,000**	**28,400,000**	**6,800,000**	**4,100,000**	**39,300,000**

Breakdown of the estimated costs for the predevelopment works, mine development and pre-stripping are presented in the following table:

Table 21-2 Predevelopment Works, Mine Development, and Pre-Stripping Costs

CAPEX	Cost, USD
Pre-development Works	**1,500,000**
Management team and pre-production personnel	445,191
Metallurgical optimization tests	22,000
Geotechnical and survey works	90,900
Infrastructure (power, water, buildings, accommodation, housing, etc.)	331,000
Insurance costs	10,000
Community development / Environment Management Plan	10,000
Contingency (10%)	90,909
Permitting Expenses	500,000
Mine Development and Pre-Stripping	**1,000,000**
Mine Ancillary Activities (SP area prep, WD prep, additional clearing)	200,364
Preproduction stripping	472,727
Mine access road (approx. 10km)	236,000
Contingency (10%)	90,909

Table 21-3 shows the details of the plant and mill infrastructure capital costs. The amount of equipment from Canatuan and the rehabilitation costs are also included.

Table 21-3 Plant and Mill Infrastructure Capital Costs

CAPEX	Cost, USD	
Equipment (Existing)	5,008,000	-
Equipment (New)	6,017,550	6,017,550
TOTAL EQUIPMENT COST	**10,739,000**	**6,017,550**
Equipment Installation		2,018,000
Piping Installation		752,000
Electrical Installation		1,611,000
Instrumentation		322,000
Buildings		1,503,000
Site Work		322,000
EQUIPMENT & INSTALLATION COST		**12,608,550**
Detailed Engineering		591,000
TOTAL INSTALLED COST		**13,199,550**
Equipment & Reagent Transfer (Canatuan to Balabag)		185,000
Existing Equipment Rehabilitation Cost		290,000
Contingency (10%)		660,000
VAT and Other Duties		1,354,949
TOTAL PLANT COST		**15,688,499**

Table 21-4 Estimated Construction Cost of the Balabag Tailings Storage Facility

Item	Units	Unit Cost Year 2019	Project Cost (USD)
Equipment	USD/m^3	3.03	4,034,307
Materials and Supplies	USD/m^3	1.43	1,905,093
Labor	USD/m^3	0.21	275,395
Quality Control Testing	USD/m^3	0.05	60,127
Consultancy Fee	USD/m^3	0.62	825,754
Monitoring Instrumentation	USD/m^3	0.02	33,030
Ancillary Facilities	Lump Sum	-	227,273
Spillway Construction	Lump Sum	-	573,864
Total			7,934,843

The capital for the initial site development will be drawn from banking institutions in the form of loans. Additional financial requirements to cover other mine and plant improvements, as well as operating expenses will be generated internally. The breakdown of the source of financing is detailed in Table 21-5:

Table 21-5 Source of Financing

Debt financing	28,500,000
Internally generated fund	5,200,000
TOTAL	**34,000,000**

Year	2019	2020	2021	2022
Balance	28,500,000	28,500,000	21,375,000	7,125,000
Payment	1,113,875	8,895,266	15,164,969	7,204,563
Interest	1,113,875	1,770,266	914,969	79,563
Principal	-	7,125,000	14,250,000	7,125,000

Loan arrangement:
Annual Interest of 6.7%
Grace period of 18 months from 1st Drawdown
Payable in 45 months
Quarterly Payment of Principal

21.2 Mining and Milling Costs

The mining and milling costs were generally patterned on the successful Canatuan gold plant operations and the copper-zinc mine with adjustments to reflect anticipated changes in the process flow sheet and incorporation of current pricing of major consumables.

21.2.1 Mine Operating Cost

The mining cost covers the cost of moving the materials from the mine to the run-of-mine stockpile for ore processing or waste dump including ancillary mining activities.

Equipment productivity has been based on load/haul simulations, assuming equipment performance parameters from equipment suppliers, gathered information or actual data from existing mining operations, and material properties of the deposit. Parameters and assumptions for the estimation of the loading and hauling unit productivity are shown in Table 21-6 and Table 21-7, respectively.

Table 21-6 Loading Unit Productivity

Parameters	Unit	Loading Unit	
		Excavator	Loader
Make/Model		**CAT 336D**	**CAT PL950H**
Bucket Capacity	m^3	1.50	2.85
Bucket Fill factor	%	80%	50%
Bucket Load per Pass	m^3	1.20	1.43
Swell Factor	%	10%	10%
Density	mt/m^3	2.50	2.50
Bucket Load per Pass	mt	2.70	3.21
Production Efficiency	%	83%	83%
Hauling Unit		**Dump Truck**	**Dump Truck**
Truck Make/Model (Dump Truck)		**10W DT**	**10W DT**
Rated Truck Capacity (by weight)	mt	18	18
Rated Truck Capacity (by volume)	m^3	15	15
Truck Fill Factor	%	80%	80%
Truck Load Capacity (by volume)	m^3	12.00	12.00
Number of Bucket Pass per Truck	n	6	5
Truck Payload (Truck Factor)	mt	14.00	14.00
Truck change/Transition Time	min	0.50	0.50
Loading Cycle Time per Pass	min	0.50	0.50
Total Loading Time	min	3.50	3.00
Total Loading Production per Hour	mt/h	200	230

Table 21-7 Hauling Unit Productivity

Parameters	Unit	Average
Haul Distance (Mine to ROM Stockpile/Waste Dump)	m	1,300
Average Speed (Loaded)	kph	15
Average Speed (Empty)	kph	15
Loading Time	min	3.50
Production Efficiency	%	83%
Travel Time (Loaded)	min	5.20
Truck Dump Time	min	0.50
Travel Time (Empty)	min	5.20
Total Cycle Time	min	14.40
Total Trips per Hour	trips	4.20
Production per hour	mt	48.95

Mine production cost estimates have been developed and estimated using information from several sources, including data from equipment suppliers and local equipment contractors, gathered information from past and existing mining operations particularly of similar sized mining operations in the Philippines or in Southeast Asia. Other cost parameter assumptions were estimated inhouse.

Equipment hourly operating costs (exclusive of internal supervision and labor costs) for major equipment are shown in Table 21-8.

The diesel price assumed for this study is USD 0.90 per liter.

Table 21-8 Equipment Rental Costs and Diesel Consumption

Equipment	Hourly Rental Rate USD/h	Diesel Consumption L/h
Excavator, CAT 320D or equivalent	25	18
Excavator, CAT 336 or equivalent	40	25
Front End Loader, CAT PL950H	30	25
Bulldozer, CAT D6R or equivalent	43	20
Bulldozer, CAT D8R or equivalent	97	40
Dump Truck, 10-wheeler, 15 cu m	27	
Motor Grader, CAT 12G or equivalent	28	13
Rock Breaker, CAT 336D	40	25
Road Roller/Vibratory Compactor, 20MT cap	23	10
Water Truck, 15,000 liters cap	35	
Fuel Truck 6,000 liters cap	29	
Workshop Crane	27	
Service Truck	32	

Free-digging will be applied to areas that can be developed readily by the excavator and dozer. For relatively competent rocks, blast holes will be drilled by 89mm bit diameter top-hammer drills. An explosives truck will be used to load and unload explosives for blasting.

During production stage, drilling will be conducted in ore and waste areas. Ore holes will be on a 3m burden x 3m spacing interval while waste holes will be drilled on a 5m burden x 5m spacing interval. This will facilitate advance drill holes assays prior to excavation. A subgrade of 0.5m will be implemented to minimize hard toes. Drill and blast activities will be carried out on a contract basis.

The following drill and blast assumptions were applied for cost estimation in Balabag Project:

- Eighty-five percent (85%) of the ore will require blasting. This assumption is based on the fact that the valuable metals are contained in silica-rich rocks, which are relatively hard to break and excavate.
- All of the ore will be drilled for grade control purposes.
- Sixty-five percent (65%) of the waste will be drilled and blasted. No further provision for grade control drilling on waste materials will be made.

The drilling and blasting unit costs based on Canatuan sulfide mine operations were likewise applied to Balabag Project.

Drilling and blasting cost per volume	*USD 2.73/m^3*
Grade control drilling cost per meter	*USD 4.00/m*

Shown below is a summary of the mining cost estimate.

Table 21-9 Summary of Mining Costs

Particulars	ORE Cost/MT (USD)	WASTE Cost/MT (USD)
Excavation & Loading	0.3	0.3
Hauling (ROM Stockpile/Waste Dump)	0.58	0.58
Rock Breaking	0.06	0.06
Dozing (Rock Breaking/Ripping)	0.05	0.05
Dozing (Ancillary/Waste Dump Maintenance)	0.02	0.27
Road Grading/Compacting (Ancillary)	0.02	0.02
Crusher Feeding	0.48	0
Drilling & Blasting Cost	0.93	0.71
Grade Control Drilling Cost	0.06	0
Total	**2.51**	**2**

21.2.2 Milling Cost

Details of the mill operating costs are presented in Table 21-10:

Table 21-10 Milling Cost Estimates and Assumptions

Particulars	kg/t, kWh/t	Unit Price, US$/kg, US$/kW	US$/a	US$/t
Labor	-	-	750,244	1.03
Power	41.88	0.22	6,624,723	9.07
Steel				
Primary crusher liners	0.01	2.82	23,045	0.03
SAG mill liners	0.07	1.82	95,205	0.13
Ball mill liners	0.05	1.82	62,522	0.09
SAG mill grinding media (balls)	1.14	1.38	1,142,525	1.57
Ball mill grinding media (balls)	1.01	1.43	1,051,790	1.44
Reagents				
Cyanide	1.24	2.04	1,843,926	2.53
Sodium Metabisulfite	0.50	0.43	154,653	0.21
Lime	1.26	0.15	135,957	0.19
Flotation Reagents	-	-	271,307	0.44
Merrill-Crowe Reagents	-	-	481,131	0.59
CIL Reagents	-	-	668,195	0.92
Maintenance	-	-	1,028,190	1.41
Assaying & Others	-	-	182,364	0.25
TOTAL COST			**14,515,776**	**19.88**

21.3 General Administration and Overhead Costs

Overhead cost estimates are also based on Canatuan operations but adjusted to a reduced size of operation. For cash flow calculation purposes, overhead costs are reflected as fixed dollar expenses independent of the mill throughput.

For base-case cash flow purposes, the following cost assumptions were considered:

Site overheads:	*5,000 USD/day*
General administration:	*2,500 USD/day*

Details of the overall general administration and site overhead costs per year are presented in the following tables:

Table 21-11 Total Site Overhead Cost (USD)

Particulars	2020	2021	2022
Business Travel and Accommodations	153,650	153,650	153,650
Business Travel and Accommodations	*18,625*	*18,625*	*18,625*
Accommodation Meals	*6,424*	*6,424*	*6,424*
Contracted Services	*128,601*	*128,601*	*128,601*
Consultants & Contractors	18,360	18,360	18,360
Freight, Delivery & Duty	2,518	2,518	2,518
Taxes & Licenses	70,779	70,779	70,779
Net Materials and Supplies	77,345	77,345	77,345
Promotion & Advertising	54,883	54,883	54,883
Rent & Utilities	1,756	1,756	1,756
Telephone & Communications	4,523	4,523	4,523
Wages & Salaries	549,484	549,484	549,484
Employee Benefits	161,681	161,681	161,681
Overtime	*16,908*	*16,908*	*16,908*
Custody Allowance	*50,631*	*50,631*	*50,631*
Casual & Temporary Labor	*5,540*	*5,540*	*5,540*
Vacation/Sick Pay	*4,052*	*4,052*	*4,052*
13th Month Pay	*30,202*	*30,202*	*30,202*
SSS Premium	*30,261*	*30,261*	*30,261*
De Minimis Benefits	*7,820*	*7,820*	*7,820*
HDMF Contributions	*8,850*	*8,850*	*8,850*
Philhealth	*5,960*	*5,960*	*5,960*
Miscellaneous Benefits	*1,457*	*1,457*	*1,457*
Total Site Overhead	**1,095,000**	**1,095,000**	**1,095,000**

Table 21-12 Total General Administration Cost (USD)

Particulars	2019	2020	2021	2022
Accounting & Legal	6,729	6,729	6,729	6,729
Bank Charges	126	126	126	126
Business, Travel & Accommodation	36,326	36,326	36,326	36,326
Business, Travel & Accommodation	*25,077*	*25,077*	*25,077*	*25,077*
Accommodation & Meals	*11,249*	*11,249*	*11,249*	*11,249*
Contracted Services	46,373	46,373	46,373	46,373
Freight, Delivery & Duty	348	348	348	348
Insurance	3,878	3,878	3,878	3,878
Dues & Subscription	4,375	4,375	4,375	4,375
Taxes & Licenses	487	487	487	487
Taxes & Licenses	*157*	*157*	*157*	*157*
Documentary Stamp Tax	*330*	*330*	*330*	*330*
Materials & Supplies & Services	4,798	4,798	4,798	4,798
Promotions, Representation and Donations	24,955	24,955	24,955	24,955
Provision for unrecoverable input VAT	16,237	16,237	16,237	16,237
Repairs & Maintenance	1,154	1,154	1,154	1,154
Telephone & Communications	7,148	7,148	7,148	7,148
Utilities, Power & Water	3,324	3,324	3,324	3,324
Wages & Salaries	124,603	124,603	124,603	124,603
Consultants & Contractors	15,294	15,294	15,294	15,294
Employee Benefits	66,605	66,605	66,605	66,605
Vacation/Sick Pay	*12,796*	*12,796*	*12,796*	*12,796*
13th Month Pay	*10,384*	*10,384*	*10,384*	*10,384*
SSS Contributions	*1,514*	*1,514*	*1,514*	*1,514*
Rice Subsidy	*1,876*	*1,876*	*1,876*	*1,876*
De Minimis Benefits	*2,831*	*2,831*	*2,831*	*2,831*
HDMF Premiums	*126*	*126*	*126*	*126*
Philhealth Contributions	*466*	*466*	*466*	*466*
Pension/Retirement Benefit	*32,016*	*32,016*	*32,016*	*32,016*
Miscellaneous Benefits	*4,596*	*4,596*	*4,596*	*4,596*
Education & Training	2,240	2,240	2,240	2,240
Total General and Administrative Overhead	**365,000**	**365,000**	**365,000**	**365,000**

21.4 Estimated Environmental Management Costs

Estimated costs for implementation of the environmental management programs during the operations period are identified in the EPEP. The costs are divided into three categories: Progressive Rehabilitation, Environmental Management and Environmental Monitoring. These represent internal operating costs. These costs will also form the basis for development of the Annual EPEP documents. The estimated costs are identified in Table 21-13. These costs exceed the minimum of 3% of operating costs guideline recommended by the MGB.

Table 21-13 Estimated EPEP Costs

Activity Sector	Total Cost M USD	Year		
		2020	2021	2022
Progressive Rehabilitation	1.62	0.47	0.81	0.33
Environmental Management	4.51	1.32	2.25	0.93
Environmental Monitoring	0.77	0.23	0.39	0.16
Annual Total	6.90	2.02	3.45	1.43

Estimated Costs for implementation of the FMRDP are also internal costs but will be set aside in the FMRDP Fund for future use under the direction of the Region 9 MRFC and the Company. Deposits to the FMRDP Fund will be made on an annual basis beginning the 1st year of operations. The total estimated cost for implantation of the FMRDP is 5.02 M USD. Details and information used in developing the costs are included in the FMRDP. The annual deposit schedule based on a 3-year mine life is shown in Table 21-14. However, for a more conservative approach, the project takes into account paying the FMRDP deposit in year one.

Table 21-14 FMRDP Fund Deposit

Deposit	Total Deposit M USD	Operations Year Deposit (M USD)		
		2020	2021	2022
Annual Fund Deposit	2.51	2.51	2.51	-
Cumulative Fund Deposit	5.02	2.51	5.02	5.02

Although the TSF cost is not included as part of the EPEP costs as mandated by the MGB, it does serve as a key environmental management feature of the Project. The estimated cost for the facility is USD 10.0 million without contingencies. Approximately USD 5.0 million will be required in Year 1 prior to operations for Stage 1. The remaining USD 5.0 million is allocated to the Stage 2 cost in Year 2.

Similarly, costs associated with the preparation and operation of the Waste Rock and Overburden Waste Disposal Areas can also be considered as environmental management costs. The initial cost for Stockpile preparation prior to operations is included in the pre-stripping budget of the Project's CAPEX. Costs for preparation and development of the subsequent Stockpiles are estimated to be USD 0.6 million. This will be used primarily for surface preparation, drainage controls and water management.

21.5 Community Development Costs

The Social Development Management Program (SDMP) for the Balabag Project was approved by the Mines and Geosciences Bureau (MGB) Region-9 Office on May 24, 2014.

The total operating cost (TOC) used for the formulation of the approved five-year SDMP was taken from the initial mining feasibility study drafted in early 2014. This annual SDMP program will be amended based on actual mining plans.

The TOC is defined in accordance to Section 134 of the DENR Administrative Order no. 2010-21 (The Consolidated DENR Administrative Order for the Implementing Rules and Regulations of R.A. no. 7942) as stated below:

> *"All costs and expenditures related to mining/extraction and treatment/processing (inclusive of depreciation, depletion and amortization), exploration activities during operation stage, power, maintenance, administration, excise tax, royalties, transport and marketing, and annual progressive environmental management"*

The projected TOC for the Balabag Project is USD 134.4 million. Therefore, USD 2.0M was the computed equivalent of the 1.5% allotment for the SDMP implementation.

There are four barangay that are determined to be impacted by the Balabag Project. Other communities may also be included as beneficiary communities but will be on a case-to-case basis only. The computed allocation per impact community is based on the implementation scheme as indicated in the approved five-year SDMP plan, as follow:

- 20% of the total SDMP allocation will be for the host community, Barangay Depore;
- 45% for the primary neighboring barangays, Pulangbato, Dimalinao, and Dipili; and
- 35% for the secondary neighboring stakeholders.

Community development programs will be provided to the secondary stakeholders of the Balabag Project but will have to go undergo evaluation first by the Community Relations and Development Office (CReDO). Projects for the affected Indigenous Cultural Community (ICC) as determined in the Memorandum of Agreement executed last August 20, 2014 may also be charged to this allocation.

The CReDO will take the lead in the Project's exercise of its responsibility as mandated by law:

> *"(creating) responsible, self-reliant and resource-based communities capable of developing, implementing and managing community development programs in a manner consistent with the principles of people empowerment and sustainable development"*

The planning, implementation, and periodic monitoring of the SDMP projects will be in coordination with the Barangay Technical Working Groups (BTWG) and the MGB Region-9 Office, through the conduct of the IEC activities of the CReDO in accordance to Section 136 of DENR Administrative Order no. 2010-21.

As per an initially approved five-year SDMP plan, a total of USD 913,500 will be allotted for the Development of the Host and Neighboring Communities (DHNC); USD 182,700 for the Information, Education, and Communication (IEC) activities; and USD 121,800 for the Development of Mining Technology and Geosciences (DMTG) activities. The CReDO will integrate Infrastructure Support in the DHNC allocation. Capacity Development activities, on the other hand, will be charged to the IEC activities budget.

The annual budget is further allocated according to the following projected proportions as per the approved SDMP plan indicated in Table 21-15 to Table 21-19.

Table 21-15 Balabag Project SDMP Calculation

(Based on latest mine plan)

Year		2020	2021	2022		
Total OPEX	**USD**	40,859,381	66,754,133	26,756,380		
SDMP Cost (projections)	**USD**	612,891	1,001,312	401,346		

(Based on approved plan)

Year		2020	2021	2022	2023	2024
Total OPEX	**USD**	19,500,000	24,400,000	11,400,000	12,900,000	13,000,000
SDMP Cost (approved)	**USD**	292,500	366,000	171,000	193,500	195,000

Table 21-16 Balabag Project SDMP Allocation Chart (Based on latest mine plan)

Year	Total Operating Cost, USD	SDMP Total Budget 1.50%	SDMP Brgy Depore (20%)	SDMP Brgy Pulangbato (15%)	SDMP Brgy Dimalinao (15%)	SDMP for Brgy Dipili (15%)	SDMP Secondary Stakeholders (35%)
2020	40,859,381	612,891	183,867	91,934	91,934	91,934	214,512
2021	66,754,133	1,001,312	300,394	150,197	150,197	150,197	350,459
2022	26,756,380	401,346	120,404	60,202	60,202	60,202	140,471
TOTAL	**134,369,894**	**2,015,548**	**604,665**	**302,332**	**302,332**	**302,332**	**705,442**

Table 21-17 Balabag Project SDMP Allocation Chart - Host Community (based on approved plan)

PROGRAM / PROJECT / ACTVITIES (PPA)	SDMP BUDGET, USD (YR1-YR5)	OPERATION YEAR				
		Year 1	Year 2	Year 3	Year 4	Year 5
Responsive Education	304,500	73,125	91,500	42,750	48,375	48,750
Health and Sanitation	304,500	73,125	91,500	42,750	48,375	48,750
Sustainable Livelihood	304,500	73,125	91,500	42,750	48,375	48,750
Total Physical SDMP (75%)	**913,500**	**219,375**	**274,500**	**128,250**	**145,125**	**146,250**
IEC (15%)	**182,700**	**43,875**	**54,900**	**25,650**	**29,025**	**29,250**
DMTG (10%)	**121,800**	**29,250**	**36,600**	**17,100**	**19,350**	**19,500**
TOTAL	**1,218,000**	**292,500**	**366,000**	**171,000**	**193,500**	**195,000**

Table 21-18 Balabag Project SDMP Allocation Chart - Primary Stakeholders (based on approved plan)

PROGRAM / PROJECT / ACTVITIES (PPA)	SDMP BUDGET, USD (YR1-YR5)	OPERATION YEAR				
		Year 1	Year 2	Year 3	Year 4	Year 5
Responsive Education	304,500	73,125	91,500	42,750	48,375	48,750
Health and Sanitation	304,500	73,125	91,500	42,750	48,375	48,750
Sustainable Livelihood	304,500	73,125	91,500	42,750	48,375	48,750
Total Physical SDMP (75%)	**913,500**	**219,375**	**274,500**	**128,250**	**145,125**	**146,250**
IEC (15%)	**182,700**	**43,875**	**54,900**	**25,650**	**29,025**	**29,250**
DMTG (10%)	**121,800**	**29,250**	**36,600**	**17,100**	**19,350**	**19,500**
TOTAL	**1,218,000**	**292,500**	**366,000**	**171,000**	**193,500**	**195,000**

Table 21-19 Balabag Project SDMP Allocation Chart - Secondary Stakeholders (based on approved plan)

PROGRAM / PROJECT / ACTVITIES (PPA)	SDMP BUDGET, PHP (YR1-YR5)	OPERATION YEAR				
		Year 1	Year 2	Year 3	Year 4	Year 5
Responsive Education	304,500	73,125	91,500	42,750	48,375	48,750
Health and Sanitation	304,500	73,125	91,500	42,750	48,375	48,750
Sustainable Livelihood	304,500	73,125	91,500	42,750	48,375	48,750
Total Physical SDMP (75%)	**913,500**	**219,375**	**274,500**	**128,250**	**145,125**	**146,250**
IEC (15%)	**182,700**	**43,875**	**54,900**	**25,650**	**29,025**	**29,250**
DMTG (10%)	**121,800**	**29,250**	**36,600**	**17,100**	**19,350**	**19,500**
TOTAL	**1,218,000**	**292,500**	**366,000**	**171,000**	**193,500**	**195,000**

21.6 Exploration Costs

From 2005 to 2011, TVIRD spent Php 591 million on exploration-related activities in Balabag Hill. Bulk of this was spent on 199 drill holes summing up to 23,700 meters that currently define the current resource.

In order to further assess the resource potential of the Balabag MPSA, TVIRD intends to continue exploration in the area. The proposed exploration work program (Phase 1) to be initiated, six months after the start of production shall consist of the following activities: 1) underground mapping of existing tunnels; 2) soil grid geochemical survey over the eastern and southern Balabag hill coupled with detailed surface geological line mapping; 3) trenching and test pitting with incidental production and 4) survey works. In the early stages of mine operations, it is also proposed to locate, assess and sample systematically some old tailings which may contain high grade materials that can be reprocessed.

It has been proven that the use of soil geochemistry in delimiting the vein zones is very effective in Balabag. The initial soil grid in Balabag Dos identified gold anomalous zones that turned out to be coincidental to vein occurrence in Vitalla Prospect. The TVIRD Exploration team has identified several prospects near Balabag Hill which also host gold and silver mineralization.

If Phase 1 exploration yields encouraging results, definition drilling and resource estimation will be carried out for Phase 2 exploration. Once fully delineated, these potential resources may be converted as additional ore reserves in the future.

The proposed budget for Phase 1 exploration is approximately 300,000 USD (Table 21-20). This shall include the capital costs and operating costs for three years of low key exploration works. Funds will be internally generated and will be sourced from the cash flow of Balabag operations.

Table 21-20 Proposed Budget for Stage 1 Exploration Within Balabag MPSA (in USD)

	Year 1	Year 2	Year 3	Total
Capital Cost				
1. Infrastructure	3,000.00			3,000.00
2. Camp Equipment	1,000.00			1,000.00
3. Office Equipment	3,000.00			3,000.00
4. Communication Equipment	2,800.00			2,800.00
5. Field Equipment	15,000.00			15,000.00

	Year 1	Year 2	Year 3	Total
Subtotal	**24,800.00**			**24,800.00**
Operating Cost				
6. Wages	30,000.00	60,000.00	30,000.00	120,000.00
7. Fuel and Lubricants	2,500.00	5,000.00	2,500.00	10,000.00
8. Spare Parts and Maintenance	1,000.00	2,000.00	1,000.00	4,000.00
9. Food and Accommodations	10,000.00	20,000.00	10,000.00	40,000.00
10.Transportation Expenses	2,000.00	4,000.00	2,000.00	8,000.00
11. Supplies and Materials	5,000.00	10,000.00	5,000.00	20,000.00
12. Assay Costs	6,000.00	12,000.00	6,000.00	24,000.00
13. Contracted Services	4,000.00	8,000.00	4,000.00	16,000.00
14. Miscellaneous Expenses	1,500.00	3,000.00	1,500.00	6,000.00
Subtotal	**62,000.00**	**124,000.00**	**62,000.00**	**248,000.00**
Total Expenses	86,800.00	124,000.00	62,000.00	272,800.00
Contingencies (10%)	8,680.00	12,400.00	6,200.00	27,280.00
GRAND TOTAL	**95,480.00**	**136,400.00**	**68,200.00**	**300,080.00**

21.7 Employment

21.7.1 Number, nationality and positions

Balabag Project will require a total of 544 personnel for the fully operational stage of. This consists of 109 staff and 435 non-staff (rank-and-file) personnel and excludes those that are employed by the mine contractors. Department managers and senior technical staff can be drawn from the local group of competent professionals with vast experience in the minerals industry. The country also has a growing population of young professionals and engineers, as Philippine universities continue to promote mining-related courses following the boom of the mineral industry in the past decade.

Skilled and non-skilled workers, as well as some administrative personnel can be recruited from a pool of qualified locals from the host and nearby communities. Balabag hill is currently occupied by artisanal miners which can provide the labor backbone for the mine and mill operations. This has been the successful experience in Canatuan mine and such practice can be applied to Balabag Project.

The company also has a pool of highly competent managers, technical personnel and skilled workers at Canatuan site – some with previous experience in gold-silver operation (TVIRD's CIP-CIL plant). The manpower requirement for the project can be grouped into four main divisions:

A. Mine Operations

- Mine Management
- Mining Contractor

B. Mill Operations and Maintenance

- Mill Operations
- Mill Maintenance

C. Assay Laboratory

- Assay Laboratory

D. Support Services

- Community Relations
- Environment
- Public Affairs
- Civil Works and Engineering
- Human Resource and Administration
- Security
- *Materials Management*
- *Finance*
- *Metallurgical Laboratory*

The table below summarizes the manpower requirement for each department.

Table 21-21 Summary of Manpower Requirement for the Balabag Project

Department	Staff	Non-Staff	Total
Overall Project Management	3	-	3
Mining	21	28	49
Mill Operations	12	109	121
Mill Maintenance	15	44	59
Assay	8	19	27
HR& Admin	17	51	68
Mobile	4	20	24
Environment	4	9	13
Materials Management	5	13	18
Civil Engineering Services	5	18	23
Credo	7	-	7
Security	6	120	126
Safety	2	4	6
TOTAL	**109**	**435**	**544**

21.7.2 Organizational chart

In the proposed table of organization ten (10) department heads will be reporting to the General Manager plus one executive assistant and public affairs manager. The hierarchy of staff to operate Balabag Project is presented below:

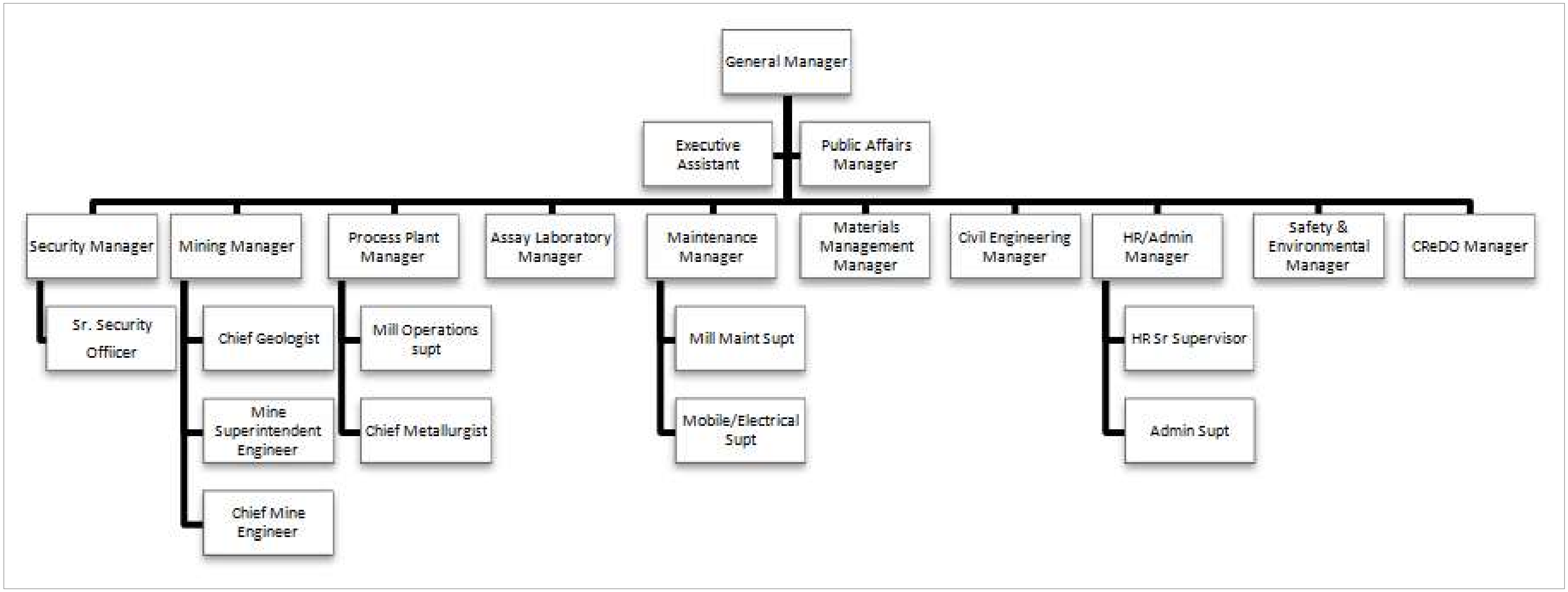


Figure 21-1 Generalized Organizational Setup for the Balabag Operations

21.8 Summary of Key Operating Costs

The total operating costs are summarized in the table below:

Table 21-22 Unit Operating Costs

Key Cost Assumptions:	UOM	COST
Ore Mining Cost	USD/t	2.51
Waste Mining Cost	USD/t	2.00
Milling Cost	USD/t	21.84
Site Overheads	USD/day	5,000.00
G&A	USD/day	2,500.00
SDMP	% of OPEX	1.50
EPEP	USD M	6.90
EXCISE TAX	% GR	4.00
NSR ROYALTY (Zamboanga Minerals Corporation)	% GR	2.50
IP ROYALTY (ICC Subanen Tribe)	% GR	1.00
FMRDP	USD M	5.02
FOREX	USD=PHP	54.00
REFINING COSTS – GOLD	% GR	0.10
REFINING COSTS – SILVER	% GR	1.00

22 ECONOMIC ANALYSIS

22.1 Project Economics

During the reserves optimization stage of the feasibility study, the gold and silver prices average at USD 1300/oz and USD 18/oz, respectively. Hence, these values were input as parameters in estimating the mineable reserves. Initial cash flow projections using these figures indicate a Net Present Value (NPV) of USD 17.0 million at 8% discount rate with Internal Rate of Return (IRR) of 37%.

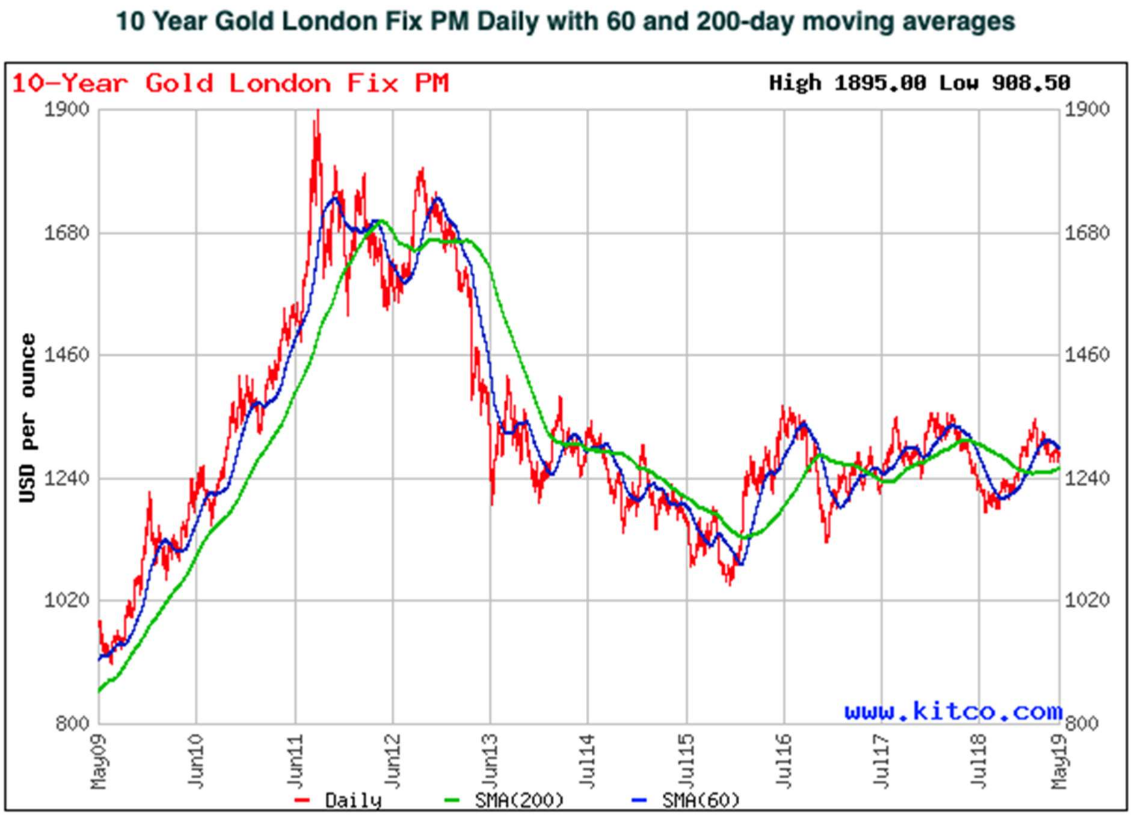


Figure 22-1 Gold Price Trend from May 2009 to May 2019 (Source: www.kitco.com)

However, recent trading prices for gold went below USD 1300/oz. With this, TVIRD Management opted to use conservative metal prices of USD 1250/oz gold and USD 15/oz silver as the base case scenario in running cash flow projections for the Balabag Project.

Using these base case metal prices, the calculated NPV at 8% discount is at USD 12.0 million with IRR of 30%. Table 22-1 summarizes the key indicators of the project.

Table 22-1 Summary of the Project Economics

Initial CAPEX **Total CAPEX** **Loan** **Equity**	**28.4M US$** **39.3M US$** **28.5M US$** **5.2M US$**
Operating Cost	Direct Cost: 1,008 US$/oz AuEq Indirect Cost: 85 US$/oz AuEq Total Cost: 1,093 US$/oz AuEq Total Cash Cost: 784 US$/oz AuEq
Tax	Excise Tax: 4% Income Tax: 30%
Metal Prices	Gold: 1,250 US$/oz Silver: 15 US$/oz
Milling Rate	2000 t/d
Project IRR	30%
Project $NPV_{8\%}$	12.0 M USD

Table 22-2 and Table 22-3 show the projected income statement and the project cashflow throughout the life of mine in accordance with the declared mineable reserve.

Table 22-2 Income Statement Projection per Year

YEAR	0	1	2	3	TOTAL
	2019	2020	2021	2022	
Revenues	-	51,545,327	78,836,764	33,380,834	163,762,925
Less: Operating Costs					
Mining Cost	-	8,299,636	16,260,821	5,498,061	30,058,518
Milling Cost	-	8,955,354	13,647,686	6,208,899	28,811,940
Overhead	-	1,825,000	1,825,000	760,417	4,410,417
Other costs					
a) Selling, Gen. and Admin. Cost	-	2,289,052	2,472,767	927,529	5,689,348
b) Treatment and Refining Cost	-	623,041	707,414	243,938	1,574,394
c) Royalties (IP & Claim Owner)	-	1,804,086	2,759,287	1,168,329	5,731,702
Interest expenses	1,113,875	1,770,266	914,969	79,563	3,878,672
EPEP Cost	-	2,020,416	3,449,491	1,429,075	6,898,981
SDMP Cost	-	612,891	1,001,312	401,346	2,015,548
Taxes and fees	-	2,642,086	3,668,171	1,720,466	8,030,723
Total Operating Costs	**1,113,875**	**30,841,828**	**46,706,918**	**18,437,622**	**97,100,242**
Net Operating Income	**(1,113,875)**	**20,703,499**	**32,129,846**	**14,943,212**	**66,662,682**
Less: Non-cash Charges					
Depreciation and amortization	-	9,159,107	18,536,490	7,679,403	35,375,000
FMRDP Amortization	-	1,471,336	2,512,037	1,040,701	5,024,074
Total Non-cash Charges	-	10,630,443	21,048,527	8,720,104	40,399,074
Net Income Before Tax	**(1,113,875)**	**10,073,056**	**11,081,319**	**6,223,108**	**26,263,608**
Less: Income Tax	-	2,687,754	3,324,396	1,866,932	7,879,082
Net Income After Tax	**(1,113,875)**	**7,385,302**	**7,756,923**	**4,356,176**	**18,384,526**

Table 22-3 Project Cashflow

YEAR	0	1	2	3	TOTAL
	2019	2020	2021	2022	
Income (Loss) before tax	**(1,113,875)**	**10,073,056**	**11,081,319**	**6,223,108**	**26,263,608**
Cash Flow From Operating Activities					
Adjustments for:					
Interest Expense	1,113,875	1,770,266	914,969	79,563	3,878,672
FMRDP Amortization	-	1,471,336	2,512,037	1,040,701	5,024,074
Depreciation and Amortization	-	9,159,107	18,536,490	7,679,403	35,375,000
Net Cash Used in Operation	**-**	**22,473,765**	**33,044,815**	**15,022,774**	**70,541,354**
Interest Received/Paid	(1,113,875)	(1,770,266)	(914,969)	(79,563)	(3,878,672)
Income Tax paid	-	(2,687,754)	(3,324,396)	(1,866,932)	(7,879,082)
Net cash used in operating activities	**(1,113,875)**	**18,015,745**	**28,805,450**	**13,076,280**	**58,783,600**
Cash Flow From Investing Activities:	**(28,400,000)**	**(11,824,074)**	**(4,100,000)**	**3,925,000**	
Acquisition of Property and Equipment	(28,400,000)	(6,800,000)	(4,100,000)	3,925,000	**(35,375,000)**
FMRD Fund	-	(5,024,074)	-	-	**(5,024,074)**
Cash Flow from Financing Activities:					
Capital Infusion	5,200,000	-	-	-	5,200,000
Loan Proceeds	28,500,000	-	-	-	28,500,000
Loan Payments	-	(7,125,000)	(14,250,000)	(7,125,000)	(28,500,000)
Net cash from (used in) financing activities	**33,700,000**	**(7,125,000)**	**(14,250,000)**	**(7,125,000)**	**5,200,000**
Net Increase (Decrease) in Cash & Cash Equivalents	**4,186,125**	**(933,329)**	**10,455,450**	**9,876,280**	**23,584,526**
Cash Balance, Beg.	-	4,186,125	3,252,796	13,708,246	
Cash & Cash Equivalent, End.	**4,186,125**	**3,252,796**	**13,708,246**	**23,584,526**	

To save on interests, it is planned that the USD 28.5M loan shall be drawn quarterly in one year from the first drawdown. Construction is set to begin in 2019 and is expected to finish in the first quarter of 2020, when operations start. Revenue starts to flow in by the end of third quarter of 2020. Year 2, 2021, will be a full production year. As per bank loan agreement, the loan will be paid quarterly starting on the sixth quarter from the date of the first loan drawdown. The loan will be fully paid by the start of the third year. The total ending cash balance by the end of the 153,000 AuEq. mineable reserve is at USD 23.6M.

22.2 Sensitivity Analysis

Sensitivity analysis was conducted to test the impact of varying economic and operating criteria to the project. Variations to gold price, stripping ratio, capital cost, gold grade and gold recovery were applied to assess the viability of the Balabag Project.

The matrix table below summarizes the variations applied to the cash flow and provides the resulting discounted values for the 4 -year project. The impact on NPV and IRR of increasing or reducing the key parameters by 10% increment is presented in Table 22-4 and Table 22-5, respectively.

Table 22-4 Sensitivity Analysis Summary (NPV)

% Change	Gold Price USD 1200/oz	Strip Ratio 9.5	CAPEX USD 23.6 M	Au Grade 2.5 g/t	Au Recovery 89%	% Change
50%	45,787,325	10,327,888	7,593,372	45,769,533	23,378,028	10%
40%	40,009,204	11,664,411	9,454,042	39,994,971	22,081,767	8%
30%	34,231,083	12,974,633	11,314,712	34,220,408	20,785,505	6%
20%	28,452,962	14,266,076	13,175,381	28,445,846	19,489,243	4%
10%	22,674,841	15,586,498	15,036,051	22,671,283	18,192,982	2%
Base Case	**16,896,720**	**16,896,720**	**16,896,720**	**16,896,720**	**16,896,720**	**Base Case**
-10%	11,118,600	18,206,943	18,757,390	11,122,158	15,600,459	-2%
-20%	5,330,362	19,527,365	20,618,060	5,337,873	14,304,197	-4%
-30%	(1,019,466)	20,818,808	22,478,729	(1,007,712)	13,007,936	-6%
-40%	(8,033,927)	22,129,030	24,339,399	(8,014,715)	11,711,674	-8%
-50%	(15,833,091)	23,386,395	26,200,068	(15,809,076)	10,415,413	-10%

Table 22-5 Sensitivity Analysis Summary (IRR)

% Change	Gold Price USD 1200/oz	Strip Ratio 9.5	CAPEX USD 23.6 M	Au Grade 2.5 g/t	Au Recovery 89%	% Change
50%	67.87%	24.31%	15.58%	67.85%	41.03%	10%
40%	61.16%	26.09%	18.06%	61.14%	39.39%	8%
30%	54.32%	27.80%	20.90%	54.31%	37.73%	6%
20%	47.33%	29.45%	24.19%	47.32%	36.06%	4%
10%	40.14%	31.08%	28.06%	40.13%	34.38%	2%
Base Case	**32.67%**	**32.67%**	**32.67%**	**32.67%**	**32.67%**	**Base Case**
-10%	24.84%	34.23%	38.29%	24.85%	30.95%	-2%
-20%	16.46%	35.77%	45.31%	16.47%	29.21%	-4%
-30%	6.25%	37.24%	54.34%	6.27%	27.45%	-6%
-40%	N.C.	38.72%	66.43%	N.C.	25.66%	-8%
-50%	N.C.	40.07%	83.48%	N.C.	23.85%	-10%

N.C. – Not calculated

The results of the sensitivity analysis show that the project is highly sensitive to gold price and grade. The mine is still expected to generate cash and yield positive NPV even at 20% drop in any of the mentioned key economic parameters. Detailed results of the analysis are presented in **Appendix I-A.**

23 ADJACENT PROPERTIES

The Balabag resource is open to the east, west and south. Priority gold targets within the Balabag MPSA can be seen at Balabag Dos, Miswi South, Paguringon and Ditay. Lower priority base-metal targets occur in several areas, showing clues for porphyry copper, skarn and polymetallic mineralization styles. The exploration strategy requires diligent surface geological mapping coupled with lithogeochemistry, and if required in conjunction with soil geochemistry and trenching.

Rio Tinto Exploration Philippines Corporation (RTEPC) conducted regional geological mapping and rock and stream sediment geochemical sampling within the MPSA area. Based on these activities, RTEPC reported the presence of drainage geochemical anomalies of 100 to 360 ppm copper coupled with 10 ppm molybdenum in Legumbong, southwest part of the area which is suggestive of possible porphyry copper mineralization. K-silicate alteration (secondary biotite and magnetite) was also observed in the diorite intrusive in the same area.

Rock geochemical anomalies in San Pedro, at the north-central section of the MPSA, reflect copper skarn type of mineralization believed to be related to andesite porphyry. High zinc and lead values were also obtained from outcrop samples in the headwaters of Dipili River. Rock chips and grab samples from the Upper Dipili copper skarn gave assay results ranging from 1.72% to 9.49 % Cu, 148 to 242 g/t Ag, 0.1 to 1.59% Pb and 1.1 to 8.84% Zn.

Stream sediment samples from the San Pedro copper skarn area returned results that range from 0.14 to 0.68 ppm Au, 40 to 85 ppm Pb and 100 to 180 ppm Zn, while grab samples taken from the Upper Dipili skarn base metal gave 0.86 g/t Au, 1.24% Zn and 0.27% Pb.

Shown below are the locations of potential Au and Cu prospects identified by RTEPC within the MPSA. Also shown are the prospective areas of Makinaryas, Ditay, Malagak and Depore which host small –scale gold mining.

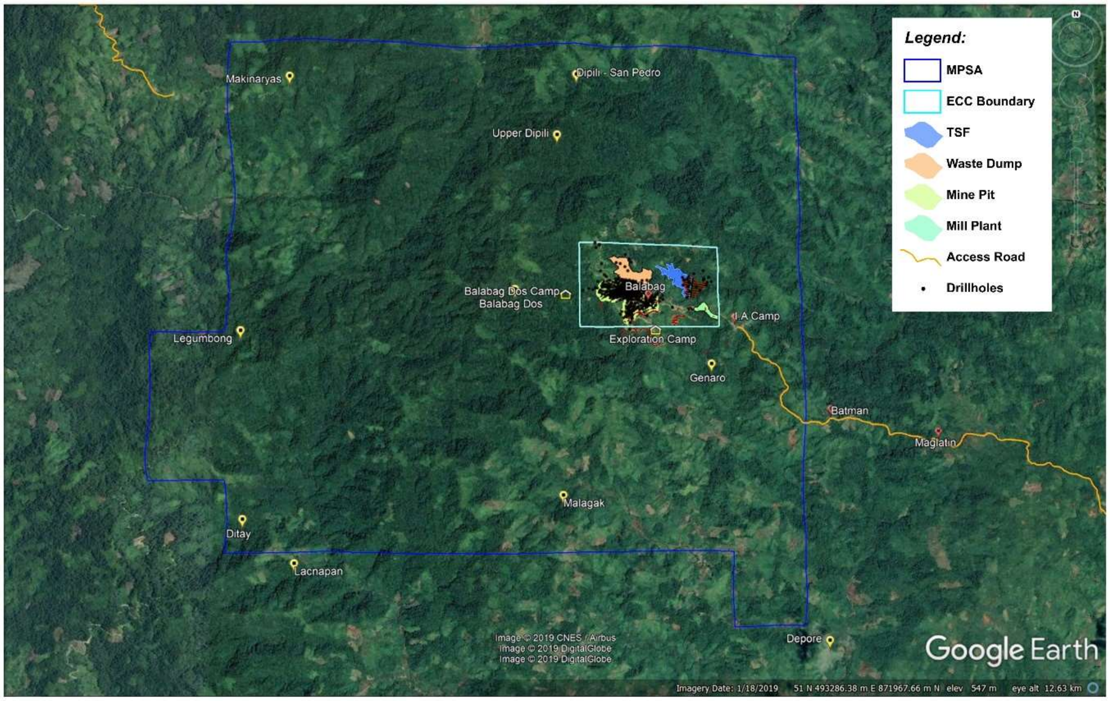


Figure 23-1 Locations of Potential Gold and Copper Prospects within MPSA

Shown above are the locations of potential gold and copper prospects within the MPSA, indicating the locations of potential gold and copper prospects outside the ECC. Priority areas to be investigated are those areas with possible similarity to Balabag Au -Ag deposit. These areas are Makinaryas, Ditay, Malagak and Depore.

There is mining activity by small scale miners in Sitio Makinaryas, Barangay Bunawan, Godod, Zamboanga del Norte. Exploration work is ongoing to sample the veins being mined by small scale miners and assess their potential.

23.1 Base Metal Occurrence

23.1.1 Legumbong

The Legumbong anomaly, some 4.5 km west of Balabag was reportedly a porphyry copper occurrence. The drainage area is anomalous in both copper and gold in stream sediments.

23.1.2 San Pedro and Upper Dipili

There are reportedly skarn copper occurrences at San Pedro and Upper Dipili, and a Zn-Pb-Au skarn mineralization style at Upper Dipili, situated all to the north and northeast of Balabag.

These are areas with anomalous gold in stream sediments and with a single arsenic anomaly at Dipili.

23.1.3 NW Balabag (Moly)

Molybdenum and copper anomalies in stream sediments were located in an area 2 km northwest of Balabag.

23.1.4 Magnetic Low Zone

Previous regional geophysical interpretation has delineated a NE-trending zone of ‘magnetic low’ located on the northwest corner of the Balabag MPSA, which lies along strike of the regional Kabayugan Fault zone. This area also marks a prominent circular feature interpreted from relief models. Weak copper anomaly in stream sediments was detected earlier by Rio Tinto in the same drainage of this circular feature.

24 OTHER RELEVANT DATA AND INFORMATION

24.1 Schedule of Project

Balabag Gold-Silver Project is projected to have its plant commissioning and first gold pour by the third quarter of 2020. At present, the company has already engaged the services of various departments to lay the groundwork on engineering, environmental compliance and social aspects of the project.

With the removal of all illegal miners and structures in the last quarter of 2012, the development is planned to commence shortly after the company has secured the approval of the Declaration of Project Mining Feasibility – a requirement from MGB. In essence, this stage defines the period during which site preparation, initial earthmoving activities and construction of mine infrastructure are carried out. This encompasses the phase in which all pertinent mine facilities that are critical to production are put up. It will be composed of the following activities or milestones:

- Construction of access roads:
  The roads will be established initially to provide access to heavy equipment and supplies that will be needed for the construction of mine facilities. During production stage, access roads will be used to provide supplies to the mine site, and to ship out products.

- Site preparation and clearing:
  Construction sites, as well as the mine and waste dump limits will be cleared of trees and vegetation. The recovered lumber will be used for EMB-approved community projects.

- Construction of mine infrastructure including, but not limited to the following:
  1) Mill plant and assay laboratory
  2) Tailing storage facility
  3) Personnel accommodation
  4) Administration office
  5) Power and water supply facilities

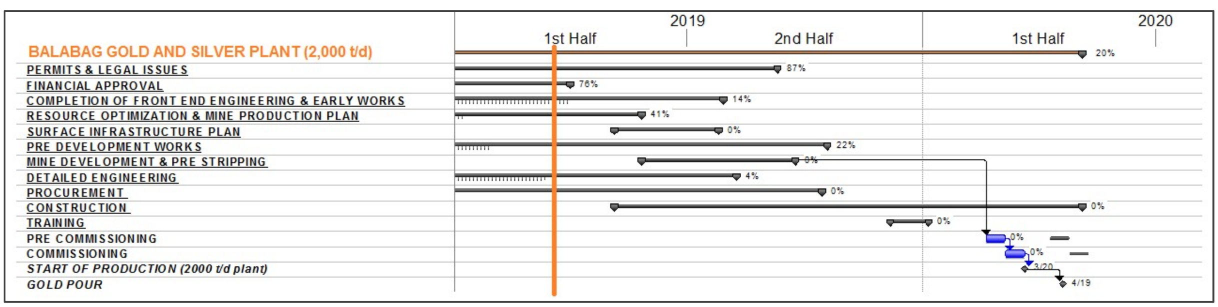
- Pre-stripping:
  Depending on the time frame, pre-stripping may be initiated to ensure that there will be sufficient, readily mineable ore when plant commissioning and production commences.

The operations stage will begin shortly after the completion of TSF and mill plant. This will include all production-related activities such as the following:

- Mining operations
- Milling operations
- Product shipment
- Site maintenance
- Environmental monitoring
- Community development programs
- Camp administration
- Safety programs
- Security

The schedule for Balabag Project development and production stages is presented in Table 24-1.

Table 24-1 Project Schedule



25 INTERPRETATION AND CONCLUSIONS

It is the conclusion of the under-signed Qualified Person (QP) and the Qualified Persons (QPs) as summarized in Section 3 of this report "Reliance on Other Experts" that this technical report contains adequate detail and information to support a potentially positive economic result. The report proposes the use of industry-standard equipment and operating practices. To date, the QPs are not aware of any fatal flaws for the Project.

The most significant potential risks associated with the Project are socio-political and environmental resistance to development of the mine, operating and capital cost escalation, grade of plant feed, plant metal recovery, unforeseen schedule delays, changes in regulatory and tax requirements and ability to raise financing and a reduction in gold price. These risks are common to most mining projects, many of which may be mitigated, at least to some degree, with additional information, adequate engineering, planning and pro-active management.

All projects benefit from increasing amounts of data and information in order to improve understanding and mitigate risks. The geology, exploration, sample preparation, assays and resource/reserve modeling completed to-date are to international standards. Exploration at the project site is ongoing to improve the quantity and quality of information to decrease resource tonnage and grade risk as much as possible. Definition drilling will further refine and delineate structures and identify any potential problem areas.

The mine design has been completed to a high standard and the use of contractors for mining significantly reduces Capex. The process plant has been designed in detail partially based on experience from the successful completed Canatuan Project nearby in Zamboanga. Plant and site services capex has been reduced through use of the Canatuan equipment. The TSF has been designed by the same consultancy firm responsible for the successful TSF at Canatuan and based on the same concepts.

Local socio-political and environment groups have been resistant to mine development in the past. Potential impact could include delays in construction of access road, mine and plant. The Project Owner and Senior Staff have successfully developed, constructed, operated and rehabilitated the Canatuan Project in Zamboanga, Mindanao and are currently involved in a successful operating nickel laterite mine in Agusan del Norte, Mindanao. They have developed close relationships with the local communities, land owners and government along with a thorough Environmental and Social Impact Assessment.

The ability to achieve the estimated Capex and Opex, plant feed grade and metal recovery are important elements of Project success. Further cost estimation accuracy with the next level of study, as well as the active investigation of potential cost-reduction measures would assist in the support of reasonable cost estimates. If Opex increases then the NSR cut-off would increase and the size of the mineable resource would reduce yielding fewer reserve tonnes. The Project development and economics could be impacted by any permitting or regulatory delays. A change in schedule would alter the Project economics.

The gold price is the most sensitive variable and a relatively short mine life of 3 years adds risk of short-term negative price variation that can be partially offset by hedging gold sales. On the positive side, there is good potential to increase the resource within the MPSA and in adjacent areas which can extend mine life or allow for higher throughput. The successful track record of the Project Owner and use of experienced Senior Staff from the Canatuan Project will be key factors in the success of the Balabag Gold and Silver Project.

26 RECOMMENDATIONS

This Technical Report qualifies as a Feasibility Study under the standards established by the Department of Natural Resources and Mines and Geosciences Bureau of the Philippines for Declaration of Mining Project Feasibility. It has been reviewed and supported by the under-signed QP and additional QPs as outlined in Section 3 of this report “Reliance on Other Experts”.

It is recommended that the Balabag Gold and Silver Project proceed to obtain approval from the DENR and MGB of the Declaration of Mining Project Feasibility for the revised project timing and revised Social Development and Management Program for the period from 2019 to 2023.

Development of the project can commence if the revised project and SMDP schedules are approved by the DENR and MGB. Resource/reserve estimate, design of access road, site facilities, mine, waste dump, plant and TSF have been completed to a high standard and can proceed following approval by the relevant authorities.

It is recommended that the Mining Project Feasibility Study should further detail the schedule, engineering design, costs, plant performance and revenue as shown in the project schedule. Increased geological confidence will result from the on-going drilling at the project site. Improved accuracy of Project economics to an international standard is essential to minimize project risks, reduce the contingency factor and improve upside potential.

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28 UNITS OF MEASURE, ABBREVIATIONS AND ACRONYMS

Symbol/Abbreviation Description

°	degree	
°C	degrees Celsius	
3D	three-dimensions	
A	ampere	
a	annum (year)	
amsl	above mean sea level	
ARD	acid rock drainage	
Au	gold	
B	billion	
BD	bulk density	
BV/h	bed volumes per hour	
Ca	calcium	
CCD	Counter current decantation	
cfm	cubic feet per minute	
CIL	Carbon in leach	
CIM Canadian institute of mining and metallurgy	Canadian institute of mining and metallurgy	
cm	centimeter	
cm2	square centimeter	
cm3	cubic centimeter	
COG	Cut-off grade	
Cr	chromium	
CSS	Closed side setting	
Cu	copper	
d	day	
d/a	days per year (annum)	
d/wk	days per week	
dB	decibel	
DMT	dry metric ton	
DST	Dry stack tailings	
DSTF	Dry Stack Tailings Facility	
DMT/m3	density, dry metric ton per cubic meter	
EA	environmental assessment	
EIS	environmental impact statement	
FEL	Front-end loader	
FS	Feasibility study	
ft	foot	
ft2	square foot	
ft3	cubic foot	

g	gram	
G&A	general and administrative	
g/cm3	grams per cubic meter	
g/L	grams per liter	
g/t	grams per tonne	
GPa	gigapascal	
GW	gigawatt	
h	hour	
h/a	hours per year	
h/d	hours per day	
h/wk	hours per week	
ha	hectare (10,000 m2)	
hp	horsepower	
HQ	drill core diameter of 63.5 mm	
Hz	Hz hertz	
in	inch	
in2	square inch	
in3	cubic inch	
IRR	internal rate of return	
k	kilo (thousand)	
kg	kilogram	
kg/h	kilograms per hour	
kg/m2	kilograms per square meter	
kg/m3	kilograms per cubic meter	
km tre	kilometer	
km/h	kilometers per hour	
km2	square kilometer	
kPa	kilopascal	
kt	kilotonne	
kV	kilovolt	
kVA	kilovolt-ampere	
kW	kilowatt	
kWh	kilowatt hour	
kWh/a	kilowatt hours per year	
kWh/t	kilowatt hours per tonne	
L	liter	
L/min	liters per minute	
L/s	liters per second	
LOM	life of mine	
m	meter	
M	million	

m/min	meters per minute	
m/s	meters per second	
m2	square meter	
m3	cubic meter	
MAP	mean annual precipitation	
masl	meters above mean sea level	
mg	milligram	
mg/L	milligrams per liter	
min	minute (time)	
mL	milliliter	
mm millimeter	millimeter	
Mm3	million cubic meters	
mo	month	
MPa	megapascal	
Mt	million metric tonnes	
MVA	megavolt-ampere	
MW	megawatt	
NI 43-101	national instrument 43-101	
Nm3/h	normal cubic meters per hour	
NQ	drill core diameter of 47.6 mm	
oz	troy ounce	
Pa	Pascal	
PAG	potentially acid generating	
PEA	Preliminary economic assessment	
PFS	preliminary Feasibility Study	
PGE	platinum group elements	
PMF	probable maximum flood	
ppb	parts per billion	
ppm	parts per million	
psi	pounds per square inch	
QA/QC	quality assurance/quality control	
QP	qualified person	
RC	reverse circulation	
RMR	rock mass rating	
ROM	Run of mine	
rpm	revolutions per minute	
RQD	rock quality designation	
s	second (time)	
S.G.	specific gravity	
Scfm	standard cubic feet per minute	
SFD	size frequency distribution	
SG	specific gravity	
SLS	Solid-Liquid Separation	
t	tonne (1,000 kg) (metric ton)	
t/a	t/a tonnes per year	
t/d	tonnes per day	
t/h	tonnes per hour	
tph	tonnes per hour	
US	US united states	
US$	dollar (American)	
V	volt	
VMS	volcanic massive sulphide	
w/w	weight/weight	
wk	week	
wmt	wet metric ton	
WRSF	waste rock storage facility	
µm	microns	
µm	micrometer	