Rosita Mining Corporation

NI 43-101 PRELIMINARY ECONOMIC ASSESSMENT STUDY FOR THE SANTA RITA PROJECT, ROSITA, NICARAGUA

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Effective Date (PEA): March 6, 2017

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1 SUMMARY

Rosita Mining Corporation (hereafter called “Rosita Mining”), a TSX Venture Exchange listed company (Symbol: RST), commissioned D.E.N.M. Engineering Ltd. (D.E.N.M.) to prepare an independent preliminary economic assessment for the Santa Rita Gold-Copper-Silver project in eastern Nicaragua. The work entailed supervision of updated metallurgical testwork, preliminary flowsheet development, operating and capital costs, and the present project economics. A simple proposed site plan is shown in Section 18, Drawings #’s 00-G-001, 002. This Preliminary Economic Assessment (PEA) report is prepared to the standards of NI 43-101

Previous studies on the property were carried out by Wu's Mining Geological Consulting Inc. (WMGC) who undertook an independent study for a resource estimate in compliance with National Instrument NI 43-101 on six stockpiles and tailings for Rosita Mining. The results from this study were reported and published on February 8, 2016 (Effective date) and March 23, 2016 (Signing Date). Sections of that report are incorporated and included in this PEA report in specific sections noted and form the basis for resource tonnages and grades for the project.

Mr. D. Salari, P.Eng., principal of D.E.N.M. Engineering Ltd. (who visited the site and area on Sept. 18-20, 2016) is an Independent Qualified Person for matters relating to metallurgy and mineral processing.

Mr. Yungang Wu, P.Geo., principal geologist of WMGC (who visited the site on November 6 to 7, 2015) is an Independent Qualified Person for matters relating to geology and resource estimates.

1.1 Project Overview

Rosita Gold-Copper-Silver project (Rosita) located in the municipality of Rosita in the Región Autónoma de la Costa Caribe Norte (RACCN), Nicaragua. The Rosita project is situated an approximate distance of 390 kilometres northeast of the capital city of Managua.

The Rosita project is registered with the Ministerio de Energía y Minas ("MEM") as exploitation concession number 821, Accord number 55-DM-38-2007 comprising 3,356.9 hectares with an expiration date of June 9, 2044.

Rosita D concession is owned by CXB Nicaragua S.A, a subsidiary of Vancouver-based, Calibre Mining Corp. Alder Resources Ltd. entered into an option agreement in August 2011 to acquire a 65% interest in the Rosita D concession from Calibre Mining. On July 24, 2015, Midlands Minerals Corporation acquired all the outstanding common shares of Alder Resources and changed its name to "Rosita Mining Corporation". By November 30, 2015, RST has earned a 65% interest in the Rosita-D Concession from Calibre. RST and Calibre Mining have entered into a Joint Venture

Agreement. Taxes and rent to maintain the concession are paid by Calibre to the government of Nicaragua and reimbursed by RST.

The Rosita D concession is the site of the Santa Rita Copper Mine which closed in 1976 because of low copper prices and civil unrest. This was an open pit operation that produced copper and gold. Various stockpiles, which were uneconomic, were left as were the tailings as deposited.

The gold, silver, and copper can be extracted from these stockpiles and tailings by conventional cyanide leaching. The feasibility for production of gold-silver-copper from the stockpile and tailings resources is greatly enhanced by the addition of the "SART" (sulphidization-acidification-recycling-thickening) copper recovery process to control the effects of soluble copper on the gold-silver recovery and to create a marketable Cu_2S copper product.

The towns of Rosita, Siuna and Bonanza, collectively form the "mining triangle" of northeast Nicaragua. The main access road to the area from Managua is via paved highway and unpaved road. Northeast Nicaragua is typical lowland humid tropical climate with warm temperatures averaging 25-32°C. Rosita is located along the break between the hilly interior highlands and the flat Atlantic Coastal Plain. The area is drained by the Bambana and Banacruz Rivers. The town of Rosita is serviced by a municipal water system via a local reservoir as well as individual owner wells. Aside from mining, the principal economic activities in the Rosita area are logging, small scale farming, livestock and service industries.

1.2 Geology

The Santa Rita pit within the Rosita D Concession is a Cu-Au-Ag skarn deposit that has been previously mined in the period 1959 to 1975. Some current artisanal mining activity continues today.

Northeast Nicaragua lies within the eastern extension of the North Interior Highlands geomorphic province. Limited exposures of ultramafic rocks indicate that portions of the region are underpinned by oceanic crust of postulated Mesozoic age. The eastern third of the Rosita concession is underlain mainly by folded and faulted carbonate sedimentary rocks of the Todos Santos Formation. To the west are andesitic to basaltic volcanic rocks that have been intruded by a series of stocks and plugs that include diorite, quartz diorite, granodiorite, quartz monzonite, and granite. Hydrothermal alteration associated with emplacement of the intrusives has led to the development of large areas of skarn and hydrothermally altered rock.

1.3 Exploration and Resource Status (Previous Technical Report)

Since 2011, RST has completed channel sampling, topographic survey, density measurements, 110 RC drill holes totalling 2,615m on all stockpiles and auger sampling on the tailings. The sampling programs generally met the industry standard and results are acceptable to support the resource estimate of the stockpiles and tailings.

The Rosita project was visited by Mr. Yungang Wu, P.Geo., on November 6-7, 2015 for the purposes of completing site visits and due diligence sampling. General data acquisition procedures, hole logging procedures and quality assurance/quality control (QA/QC) were reviewed.

Prior to the resource estimate, preliminary metallurgical tests were carried out by SGS Lakefield for Rosita stockpiles and tailings in 2014-2015. One stockpile sample with grade of 0.98 g/t Au, 0.64% Cu, 0.17% CuCNsol and 1.89% S, was tested to determine its amenability to acid heap leaching for the recovery of copper. A sample of minus 13 mm ore was leached over 30 days by intermittent bottle rolling, and the extraction of copper was 47.7% with the acid consumption of 46.1 kg/t H_2SO_4. A size fraction analysis of the leach residue showed that the extraction of copper was similar throughout indicating that finer crushing would have little impact on copper recovery.

Two cyanidation tests were conducted on the Stockpile sample. A heap leach amenability test was conducted on minus 13 mm material and the extraction of gold was 83.1% leaving a residue which assayed 0.13 g/t Au. The second test was conducted on a sample ground to a P80 of 58 µm. The gold extraction from the ground sample was 94.0% and the residue assayed 0.05 g/t Au. The consumption of cyanide was high for both tests due to the cyanide-soluble copper present in the sample.

This testwork conducted on the Stockpile sample and Tailing sample were using simple and low cost methods to recover gold and/or copper at that time, and it was recommended that Rosita Mining should undertake further detailed metallurgical testing on the stockpile and tailing materials to advance the project toward possible production.

The Mineral Resource Estimate has been prepared in compliance with NI 43-101 and Form 43-101F1 which require that all estimates be prepared in accordance with the "CIM Definition Standards on Mineral Resources and Mineral Reserves as prepared by the CIM Standing Committee on Reserve Definitions" and in effect as of the effective date of this report. Mineral Resources are tabulated in Table 1.1.

Table 1.1

Mineral Resource Estimate for Rosita Stockpiles and Tailings (1) (2) (3) (4)(5)(6)(7)(8)

Stockpiles	Class	Tonne (1,000t)	Au (g/t)	Contained Au (1,000oz)	Cu %	Contained Cu (1,000t)	Ag (g/t)	Contained Ag (1,000oz)
North	Indicated	2,007	0.66	42.4	0.89	17.8	10.94	706.0
	Inferred	907	0.65	19.0	0.95	8.6	12.28	358.0
East	Indicated	1,049	0.30	10.1	0.43	4.5	8.77	295.8
	Inferred	520	0.31	5.1	0.81	4.2	12.84	214.5
South	Indicated	800	0.52	13.5	0.46	3.7	5.88	151.1
	Inferred	634	0.43	8.9	0.29	1.9	3.90	79.5
Southwest	Indicated	2,603	0.37	30.7	0.24	6.2	4.39	367.6
	Inferred	796	0.41	10.5	0.27	2.2	4.21	107.7
Northeast	Inferred	431	0.26	3.5	0.71	3.1	12.39	171.7
North2	Inferred	150	0.68	3.3	0.71	1.1	5.42	26.1
Stockpile Total	Indicated	6,460	0.47	96.7	0.50	32.2	7.32	1,520.5
	Inferred	3,437	0.46	50.3	0.61	21.0	8.66	957.5
Tailings	Inferred	1,956	0.56	35.2	0.21	4.0	9.65	607.0

Source: Wu 2016

1. *Mineral resources which are not mineral reserves do not have demonstrated economic viability. The estimate of mineral resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues.*
2. *The quantity and grade of Reported Inferred resources in this estimation are uncertain in nature and there has been insufficient exploration to define these Inferred resources as an Indicated or Measured mineral resource and it is uncertain if further exploration will result in upgrading them to an Indicated or Measured mineral resource category.*
3. *The mineral resources were estimated using the Canadian Institute of Mining, Metallurgy and Petroleum (CIM), CIM Standards on Mineral Resources and Reserves, Definitions and Guidelines prepared by the CIM Standing Committee on Reserve Definitions and adopted by the CIM Council.*
4. *Mineral Resources were estimated utilizing Gemcom software and conventional block modelling within 3D wireframes defined on a cut-off grade of $10NSR for stockpiles and 0.3g/t Au for tailings, capped composites and Inverse Distance Squared grade interpolation.*
5. *A gold price of US$1,200/oz., copper price of US$2.5/lb and silver price of US$16/oz were utilized in the cut-off calculations of block values with process recoveries of 80% for gold, 35% for Cu (10% deducted for smelting) and 65% for silver. These values were equated against a cut-off grade of US$10 for stockpiles and 0.3 g/t Au for tailing mineral resources.*

6. *For the cut-off grade, mining costs were assumed at US$1.00/t, process costs at US$7.50/t and G&A costs at US$1.50/t.*
7. *Artisanal mined tonnages since 2012 are considered minor and not depleted from the resources of the stockpile.*
8. *Totals in the table may not sum due to rounding.*

1.4 Royalties

A royalty of 3 % net smelter return royalty (NSR) is due to country of Nicaragua once commercial production commences at the mine. This royalty is provided for in the base cash flow model (Section 22) on all saleable products.

1.5 Environmental Issues

At present, there are no reported environmental issues at the Santa Rita Project site – stockpiles, tailings, impounded water within the two (2) existing pits, and old mill infrastructure. Presently, RST has commenced base line sampling and testing in all areas as a precursor to their environment permit applications.

1.6 Permits for Operation of a Mining Site

The major permit to allow operation at Santa Rita is the Environmental Permit from the Ministry of Environment and Natural Resources. As the Santa Rita Project, located in the North Autonomous Region of Nicaragua (RACCN), the permit is done via the Secretary of Ministry of Natural Resources (SERENA). There are well documented steps and requirements for this permit application and similar in nature to permitting in Canada. Typical permitting times for this take 8-10 months.

Other applicable permits required include Forestry (INAFOR), Water Use (ANA), Cyanide importation, Municipal (Rosita) government permit for land use – project site, and Importation of process equipment to void importation taxes. At the time of this report, none of the permits have been applied for/or obtained.

As these permit applications and approvals are standard practise and have been carried out by previous mining companies in the region, the right and / or ability to perform work and develop the Santa Rita project is not insurmountable.

1.7 Local Resources and Project Infrastructure

As noted, the location of the Santa Rita project is in the “mining triangle” of northeast Nicaragua and there is local experienced labour in the surrounding mining communities. Materials and components can be acquired in Managua (south-west) area with specific process chemicals and process machinery components to be imported into the country for the project. There are two (2) major ports located on the two coasts that service the country – Corinto (west) and Puerto Cabezas (east).

Electrical power to the property is via an existing 138 kV power line adjacent to the site with available load on the grid for the process facility. Process water is available from impounded and collected surface water in both open pits (Santa Rita and R-13). The approximate water volumes in these pits are Santa Rita – 850,960 cu.mt and R-13 – 281,727 cu.mt. for a total approximate of 1.13 M cu.mt. of usable water.

1.8 Mineral Processing and Metallurgical Testing (2016-2017)

Updated and optimization testwork was conducted by SGS – Lakefield with previously collected samples from Rosita stockpiles and tailings as well as additional fresh samples from the North Stockpile. The scope of this work included but was not limited to:

- Sample and size analysis, characterization, and mineralogy of the material.
- Bond Ball Mill Grindability Testing.
- Cyanidation Testing – cyanide and leach kinetics for gold and copper.
- SART testing including SART process recycle cyanidation.
- Agglomeration and Heap Leach Column Testing.
- Preliminary Carbon Loading Testing of SART discharge solution.
- Thickening and Rheology Testing with Counter Current Decantation (CCD) Modelling.
- Copper Acid Leach.

The results were very positive and confirmed previous testing carried out by SGS – Lakefield and other testing facilities. The updated SGS – Lakefield report is "An Investigation into The Recovery of Gold and Copper from Rosita Stockpile and Tailings Samples prepared for Rosita Mining Corporation – Project # 15278-002 – March 31, 2017". The results of this report were the basis for the preliminary design, costing, and economic analysis for this Preliminary Economic Assessment (PEA) report. A summary of the test results presented from this report with comments are as follows:

- Extraction of the Stockpile minus ¼-in. material and Tailings was high – 90 % as long as sufficient cyanide was added to ensure leaching and complexing of the cyanide-soluble copper and maintain sufficient free cyanide.

- Initial high cyanide was required – 12 kg/T NaCN for the Stockpile and 8 kg/t NaCN for tailings – noting that almost all of the cyanide added can be attributed to the copper cyanide and free cyanide. Both are recoverable using the efficient SART process and recycled back to cyanidation with no detrimental effects on gold extraction.

- Heap Leach Testing on the Stockpile plus -¼- in material residue had extractions of 72.4 % gold and 25.1 % copper. Cyanide addition was high at 9.6 kg/T because of soluble copper but again, most of the cyanide are recovered by SART

- Bond Work Index – Stockpile (deslimed) was determined to be 13.9 kWh/t (medium hardness) with overall stockpile (with slimes) was 8.7 kwH/T (very soft).

- Acid Leach performed on the cyanide leach tailings resulted in copper extraction of 34.1 % representing an additional 20.3 % of copper in the original feed. Copper extraction in the cyanide leach was 40.5 %.

- Solid liquid separation indicated an underflow density of 59.5 % with the use of anionic flocculant during dynamic testing.

1.9 Mining and Reclamation

The resource is in several surface stockpiles and impounded dry tailings that are located close to the proposed processing facilities and numerous access roads. The reclamation will be via conventional earthmoving equipment with the use of local contractors and equipment and transported to the mobile crushing facility. The staged mining of the stockpiles will be based on grade to ensure higher grade feeding the process in the initial project years which can be further sampled and analysed during the mining phase. During the mining, indicated resources and inferred resources will be both excavated. The grades from the estimated resources have been used in the cash flow statement.

Some challenges will be working during the local rainy season that will affect loading, hauling, crushing, and subsequent stockpiling mill and heap leach feed material. Water diversion, collection, and control will be paramount in the project design.

1.10 Process Overview

The Santa Rita process facility will consist of the following circuits and include all associated pumping, piping, and electrical components.

- Mobile Crushing Plant

- Gold and Copper Heap Construction – pads and ponds – agglomeration - conveyance

- Milling Circuit including ball mill, leach tanks, and thickeners.
- SART Plant (Sulphidization – Acidification – Recycling – Thickening)
- Carbon in Columns Gold Recovery Circuit

The proposed circuits are included in the Process Flow Diagrams shown in Section 17

1.11 Capital and Operating Costs

1.11.1 Capital Costs

Pre-production capital costs for the Santa Rita Project including a 30 % contingency is $US 11.44M. All costs associated with loading and transporting of the stockpile and tailings material to the crushing facility are covered in the operating costs.

The Total capital over the life of mine (10 years) including a 30 % contingency is $ US 26.1 M to allow for expansions to the plant and heap pads.

The project indirect capital costs including EPCM have been allowed for in the contingency estimates.

1.11.2 Operating Costs

Operating costs for transport of the stockpiled material and the impounded tailings assumes a contract rate for loading and hauling to the crusher of $US 2.00 / tonne.

Total blended operating costs over the life of mine (10 years) is $US18.50 / tonne which allows for ramp up in production and introduction of the copper acid leach in Year 4 of the project plan. The approximate local grid power cost is $US 0.14/kwh.

Estimated cash operating costs of the 10-year mine life of the project are shown in the Table 1.2

Table 1.2 Summary of Life-of Mine Costs

Summary of Life-of-Mine Operating Costs		
Area	Life-of-mine Cost – ($US 000)	Unit Cost - $US/tonne ore treated
Loading and Material Handling	$11,169	$2.00
Total Plant Labour (Years 1-3)	$3,559	$3.25
Total Plant Labour (Years 4-10)	$9,979	$1.98
Sub-Total Labour	$13,538	
Mill Process Reagents	$5,585	$6.00
Overall Plant Power	$12,410	$2.00
Heap Leach	$31,646	$6.00
SART Process	$39,153	$6.31
Copper Leach (Acid)	$18,396	$4.00
Total Operating Costs	$131,897	$18.50

Source: DENM 2017

1.12 Economic Analysis and Base Case Cash Flow

The metal prices assumed for the economic analysis are:

- Gold $ US 1,250/oz.
- Silver $ US $18.00/oz.
- Copper $ US $2.50

The LOM ten (10) year summary projected cash flow is presented in Table 1.3

Table 1.3 Summary of Life-of-Mine Project Cash Flow

	$ US
Gross Revenue (Gold, Silver, Copper) – Milling and Heap Leach	$236.5 million US
Royalties (1)	$7.1 million US
Net Revenues	$229.4 million US
Total Operating Costs	$131.9 million US
Cash Flow before Capital	$97.5 million US
Capital Expenditure	$26.1 million US
Pre-tax Cash Flow	$71.4 million US
Net Cash Flow After Tax (2)	$51.2 million US

Source: DENM 2017

1.) Nicaraguan royalty rate of 3 % NSR applied to all saleable products.

2.) The Nicaraguan income tax rate of 30 % after depreciation of fixed assets at 10 %

The base case evaluates to an IRR of 51 % before taxes and 41 % after taxes. Applying a discounted rate of 7 % for the project, the net present value (NPV) is $ US 39.9 M pre-tax and $ US 33.9 M post-tax.

The payback based on after tax revenue and pre-production revenue is 2.8 years.

The complete Base Cash Flow is included in Section 22 of this report.

1.13 Recommendations

Higher gold prices, additional metallurgical work, and detailed capital costing will demonstrate the increased potential viability of the Santa Rita Project. The following continued development is recommended.

A proposed budget ($US) for the Santa Rita project is presented as follows:

Phase 3 Process Optimization Testwork …………	$150,000	Q3/4-2017
Exploration and Resource Work ………………….	$250,000	Q3/4-2017
Prefeasibility Study …………………………………	$500,000	Q1-2018
Permitting Application Process …………………….	$50,000	Q3-2017
Geotechnical Site Report …………….……………	$30,000	Q3-2017
Closure Plan (included in Permit application) …….	$0	Q3-2017
Front End Engineering (FEED) …………………….	$200,000	Q4-2017 – Q1/Q2-2018

Apart from the Resources of the Stockpiles and Tailings, the Rosita D Concession has many exploration targets, where there is good potential to find metal bearing material that would be treatable in the Treatment Plant that is planned.

2 INTRODUCTION

2.1 Scope of work

Rosita Mining Corporation, a TSX Venture Exchange listed company (Symbol: RST), commissioned D.E.N.M. Engineering Ltd. (D.E.N.M.) to prepare an independent preliminary economic assessment for the Santa Rita Gold-Copper-Silver project in eastern Nicaragua. This Preliminary Economic Assessment (PEA) report is prepared in accordance with the reporting standards and definitions required under Canadian National Instrument (NI) 43-101.

Rosita Mining Corporation. is an Ontario registered company, trading under the symbol of “RST” on the TSX-V Exchange with its corporate head office as follows:

120 Adelaide Street West, Suite 2400
Toronto, Ontario
Canada M5H 1T1

This Report summarizes the results of this assessment and is based on the estimate of mineral resources at the Santa Rita project of mine stockpiles and tailings prepared by Wu in February 2016. This independent Technical Report for the Santa Rita site was dated March 23, 2016 and was filed on SEDAR on March 23, 2016 (Wu 2016). Since this report has been filed and the subsequent mineral estimate stated, no further exploration work has been done. The scope of this study includes the reprocessing of the stockpiles and tailings to extract gold-silver-copper from this resource via a combination of conventional milling, cyanide leaching, heap leaching, and the SART copper recovery process. An initial short term mine life of only ten (10) years has been addressed in this report with the ability to extend with the existing resources defined. The proposed sequencing of processing will be at a rate of 1000 mtpd for three years with a ramp up to 2000 mtpd for years 4-10. The processing feed to the plant will be a combination of stockpile and tailings in different splits – these splits are detailed in the cash flow in Section 22.

Mr. D. Salari, P.Eng., principal of D.E.N.M. Engineering Ltd. (who visited the site and area on Sept. 18-20, 2016) is an Independent Qualified Person for matters relating to metallurgy and mineral processing.

Mr. Yungang Wu, P.Geo., principal geologist of WMGC (who visited the site on November 6 to 7, 2015) is an Independent Qualified Person for matters relating to geology and resource estimates.

The PEA has an effective date of March 6, 2017

2.1 Source of Information

The principal sources of information used to compile this report by DENM were supplied by RST (Toronto and Nicaragua) which are detailed in the References section of this document. This report is based, in part, on internal company technical reports and maps, published government reports, company letters, memoranda, public disclosure and public information as listed in the References section of this document. Sections from Reports Authored by other consultants have been directly quoted or summarized in this document, and are so indicated where appropriate.

2.2 Independence

Mr. Salari and Mr. Wu have no relationship with RST, and hold solely a professional association between client and independent consultants. This report is prepared in return for fees based upon agreed commercial rates and the payment of these fees is in no way contingent on the results of this report.

2.3 Units of Measurements and Currency

Metric units are used throughout this report unless noted otherwise. Currency is U.S. dollars ("US$"). At the time of writing this report the currency exchange rate was 29.0 NIO per US$1. RST uses US$ for most of its official cost and budget numbers and as such this report did not convert any currency figures during this study. A conversion factor of 31.1035 grams per Troy ounce gold and silver and 2,205 lb per metric tonne were used for the previously reported resource estimate.

2.4 Abbreviations

Abbreviations applied in this report are listed in Table 2.1 below.

TABLE 2.1

LIST OF ABBREVIATIONS

Description		Description	
3D	Three Dimensional	mm	Millimetre
AAS	Atomic Absorption Spectrometry	NIO	Nicaragua Currency
Ag	Silver	NN	Nearest Neighbour
Au	Gold	NQ	Size of Diamond Drill Rod/Bit/Core
CIM	Canadian Institute of Mining, Metallurgy and Petroleum	NSR	Net Smelter Return
cm	Centimetre	oz	Ounce
Comp	Composite	ppb	Parts Per Billion
CRM	Certified Reference Material or Certified Standard	ppm	Parts Per Million
Cu	Copper	QA	Quality Assurance
CV	Coefficient of Variation	QC	Quality Control
DDH	Diamond Drill Hole	QP	Qualified Person
g	Gram	RC	Reversed Circulation Drillhole
		RACCN	Región Autónoma de la Costa Caribe Norte
g/m^3	Grams Per Cubic Metre	ROM	Run Of Mine
g/t	Grams Per Tonne	RQD	Rock Quality Designation
ICP	Inductively Coupled Plasma	RST	Trading Symbol of Rosita Mining Corp.
ICP-AES	Inductivity Coupled Plasma Atomic Emission Spectroscopy	SD	Standard Deviation
ID2	Inversed Distance Squared	SG	Specific Gravity
IP	Induced Polarization	SMU	Selective Mining Unit
ISO	International Standards Organisation	T	Tonnes
kg	Kilogram	t/m^3	Tonnes Per Cubic Metre
km	Kilometres	Tpa	Tonnes Per Annum
km^2	Square Kilometres		
koz	Thousand Ounces	US$	United States of America Dollars
kt	Thousand Tonnes		
lb	Pound	UTM	Universal Transverse Mercator
m	Metres	WMGC	Wu's Mining Geological Consulting Inc.
Ma	Million Years	X	Easting
Mag	Magnetometer Survey	Y	Northing
		Z	Elevation

Source: Wu 2016

3 RELIANCE ON OTHER EXPERTS

The Main author of this PEA Report (Mr. Salari) has assumed, and relied on the fact, that all the information and existing previous technical documents listed in the References section of this report are accurate and complete in all material aspects. Specific sections of the report that relate to location, property description, history, deposit, exploration, drilling and assaying (Sections 4 to 12, 14) are taken from the previous Technical Report prepared by Mr. Wu as well as any updated information provided by RST. While all the available information has been carefully reviewed, the Author cannot guarantee its accuracy and completeness. The Author reserves the right, but will not be obligated to revise the report and conclusions if additional information becomes known to the Author subsequent to the date of this report.

Although copies of the tenure documents, operating licenses, permits, and work contracts were reviewed, an independent verification of land title and tenure was not performed. The Authors have not verified the legality of any underlying agreement(s) that may exist concerning the licenses or other agreement(s) between third parties but has relied on the client's solicitor to have conducted the proper legal due diligence.

A draft copy of this report has been reviewed for factual errors by RST and the Authors have relied on RST's historical and current knowledge regarding ownership, joint venture agreements, permits (or lack of), and local labour costs for the project. Any statements and opinions expressed in this document are given in good faith and in the belief, that such statements and opinions are not false and misleading at the date of this report.

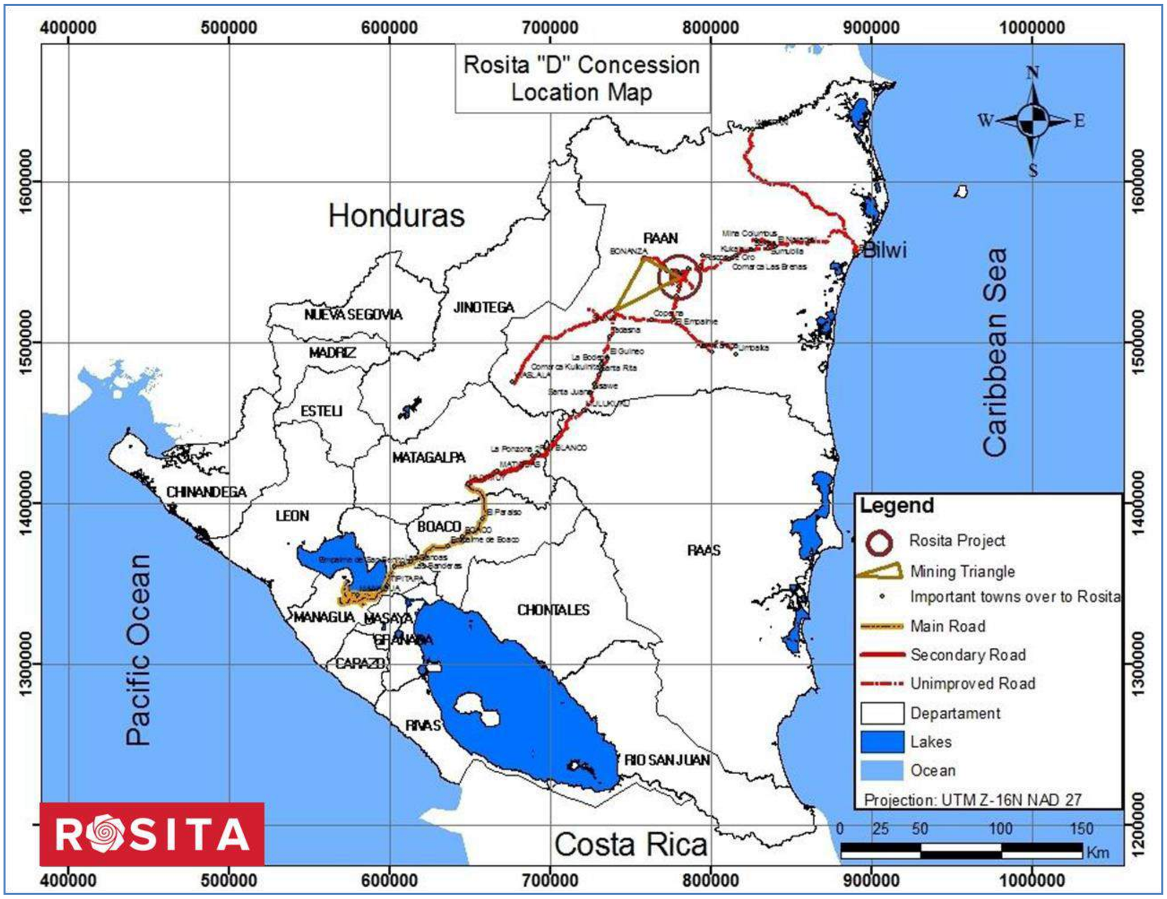
4 PROPERTY DESCRIPTION AND LOCATION

4.1 Project Location

The Rosita project, centrally situated in the municipality of Rosita in the Región Autónoma de la Costa Caribe Norte (RACCN), Nicaragua, is located an approximate distance of 390 kilometres northeast of the capital city of Managua and 120 kilometres west of the port town of Puerto Cabezas (Bilwi) (Figure 4.1). The facility at Puerto Cabezas is a shallow water port with capacity for large ships (500ft) and serviced by three shipping lines.

FIGURE 4.1

LOCATION OF ROSITA PROJECT



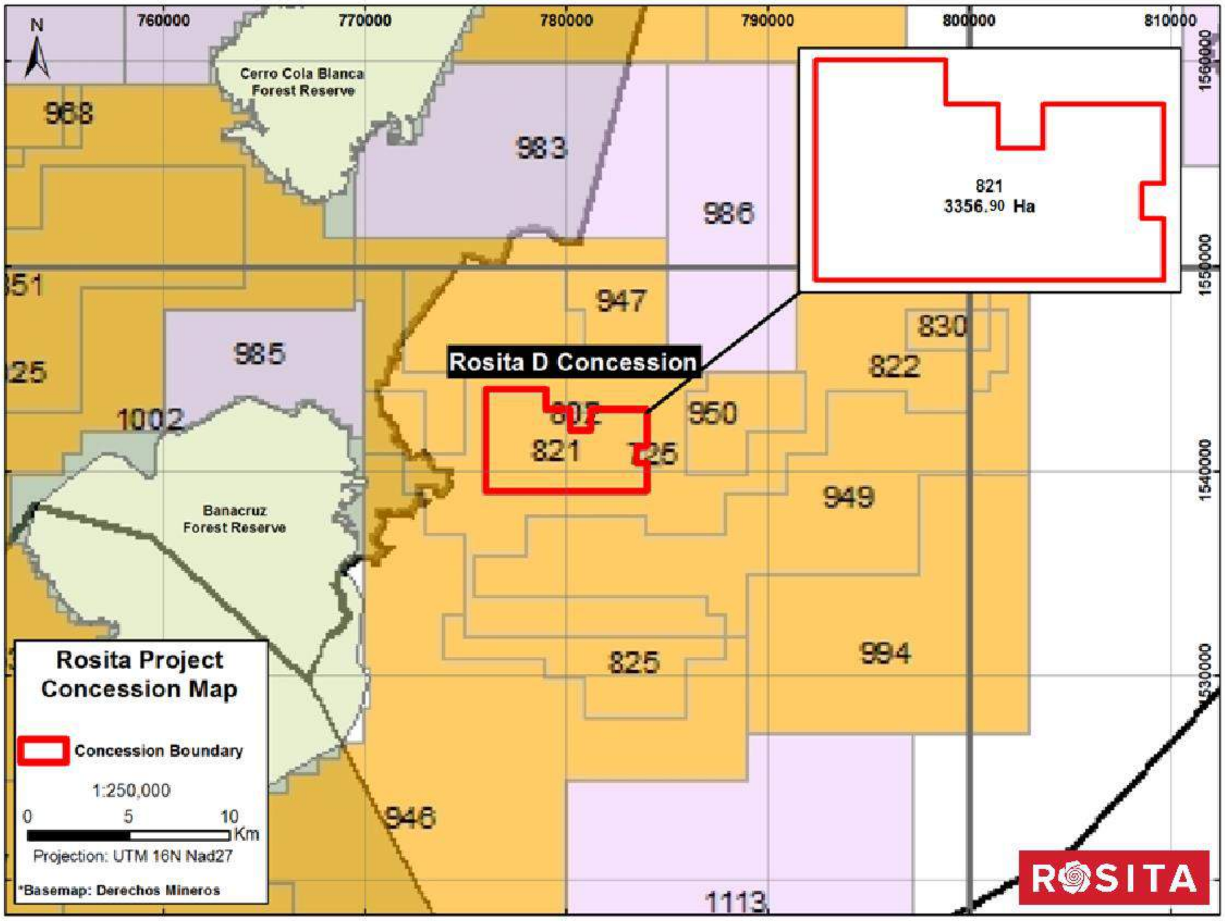
Source: Wu 2016

4.2 Project Ownership

The Rosita project is registered with the Ministerio de Energía y Minas ("MEM") as exploitation concession number 821, Accord number 55-DM-38-2007 comprising 3,356.9 hectares with an Expiration Date of June 9, 2044 (Figure 4.2).

FIGURE 4.2

ROSITA PROJECT CONCESSION MAP



Source: Wu 2016

Rosita D concession was granted to Hemco De Nicaragua, Sociedad Anónima in 1994, and subsequently transferred to Desarrollo Minero De Nicaragua, Sociedad Anónima (DESMINIC) in 2006, and then to Yamana Nicaragua, Sociedad Anonima (Yamana Gold Inc.) in 2007. Calibre Mining began operation in Nicaragua in 2009 in all of Yamana concessions in the mining triangle. In 2012, Yamana Nicaragua S.A changed its name to CXB Nicaragua S.A according to the agreement No.051-DM-357-2012, all concessions of Yamana, including Rosita D are considered that continue to belong to CXB Nicaragua, Sociedad Anónima, a wholly owned subsidiary of Vancouver-based, Calibre Mining Corp (CXB: TSX-V) ("Calibre"). Alder Resources Ltd. (ALR:TSV-V) ("Alder") entered into an option agreement in August 2011 to acquire a 65% interest in the Rosita D concession from Calibre by issuing 1,000,000 shares and incurring expenditures of $4,000,000 over four years.

On July 24, 2015, as filed on Sedar.com, Midlands Minerals Corporation (MEX: TSX-V) ("Midlands") acquired all of the outstanding common shares of Alder by way of a plan of arrangement (the "Arrangement"). Under the Arrangement, shareholders of Alder received consideration of 1.81 of a common share of Midlands per Alder Share, calculated on a pre-consolidation basis. Upon completion of, and in connection with, the Arrangement, Midlands consolidated the outstanding Midlands Shares (including the Midlands Shares to be issued to former holders of Alder Shares under the Arrangement) on the basis of one new common share for every 10 existing common shares and changed its name to "Rosita Mining Corporation" (RST: TSX-V)

As filed on Sedar.com on November 30, 2015, RST has received confirmation from Calibre that RST has completed the expenditure requirements to earn a 65% interest in the Rosita-D Concession in Nicaragua. This was achieved following the completion of a 1,939-metre drill program which tested and infilled the extensive surface stockpiles and two priority exploration targets on the property in 2015.

Calibre owns surface rights to several parcels of land in the vicinity of the old open pits at Rosita (Figure 4.3), such as Escombrera No.1 (R-13 and surroundings), Escombrera No.2 (tailings area) and Industrial Area dump and El Tajo (Santa Rita and surroundings). The Escombrera No. 1 and 2 properties are partially occupied by local people who claim to have legal titles. In the Industrial and El Tajo area there are some private houses, two artisan mills and some guiriceros extracting gold from the stockpiles. Nicaraguan mining law under MEM allows artisanal mining on 1% of a concession.

A copy of NBIT receipt of taxes for surface right was reviewed which indicated that the taxes of the first half of the year 2016 were paid by Calibre for all its concessions including Rosita D concession. Audited statements for Rosita have been completed until the end of 2016. Property ownership and taxation is thus confirmed. In addition, Concession taxes have been paid by Calibre for the first six (6) months of 2017 and they have been reimbursed by Rosita.

Exploitation concessions in Nicaragua are subject to annual payments of US$2.00/ha in years 1 and 2, US$4.00/ha in years 3 and 4 and US$8.00/ha thereafter. The Rosita D Concession currently carries an annual payment of US$26,855 which was paid for year 2015 according to NBIT receipt copies of the payments provided by RST. As stated above, all taxes for the Concession have been paid up to including the first six (6) month of 2017 at the date of this Report.

FIGURE 4.3

SURFACE RIGHTS OF CALIBRE (Source: Wu:2016)

4.3 Environment Liabilities

4.3.1 Environment

Owing to previous mining operations on the Property there has been considerable environmental disturbance in the Santa Rita pit area. It has been reported (Equity Exploration Consultants Ltd., 2009) that the Nicaraguan government is responsible for any environmental impact from mining and exploration activities prior to privatization in 1994. This information has not been confirmed by Author of this Report.

An environmental permit is required from the national and regional (RACCN) authorities for all activities of mineral exploration. The permit requires a report that includes an environmental baseline study together with exploration plan, time-line and cost estimate among others. The report must be submitted to the Secretaría de Recursos Naturales ("SERENA") in Puerto Cabezas. RST's exploration activities fall under a report submitted by Yamana in 2009. An amendment to this permit was submitted to SERENA and subsequently approved. The local municipality receives a copy of all documentation supplied to the regional authority and exercises control and supervision of all activities developed on their territory.

All exploration work carried out by RST (previously Alder Resources) on the Rosita D Concession have implemented a number of industry standard environmental practices. All trenches have been refurbished by planting of grass to accelerate reforestation and minimize soil erosion. Complying with the environmental impact study, RST carried out sampling of surface waters in the areas where exploration activities occurred.

In conjunction with staff of Calibre Mining's local subsidiary, RST has met the requirements of Mines Direction, SERENA and the municipality. All exploration activities have been approved by the various Authorities.

4.3.2 Artisanal Mining

The artisanal mining is developed in the Rosita D Concession with little control of local Mines Direction and Environmental authorities. The guiriceros at Rosita area extract the gold by archaic methods that pollute the water through the use of mercury and cause environmental damage during their extraction activities. The artisanal mining activities are mainly concentrated in the North and Southwest stockpiles and in the western margin of the old Santa Rita pit, where the access road is damaged.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Accessibility

The towns of Rosita, Siuna and Bonanza, collectively form the “mining triangle” of northeast Nicaragua. The main access road to the area from Managua is via paved highway for about 200km to Rio Blanco, and some stretches of road between Rio Blanco and Mulukuku are hydraulic concrete, then an unpaved road to Rosita totalling 190km. There are unpaved roads among Siuna, Rosita and Bonanza. Access from the port of Puerto Cabezas on the Atlantic coast is via a well-maintained gravel road west for a distance of 120km.

Aside from the principal unpaved roads, the Rosita area is traversed by a series of dirt tracks accessible by 4-wheel drive vehicle and footpaths that connect outlying villages and farms. The stockpiles and tailings are closed to Town of Rosita and accessible via gravel roads.

The near-by city of Bonanza is serviced by commercial airline La Costeña with daily flights from and to Managua.

5.2 Climate

Northeast Nicaragua is typical lowland humid tropical climate with warm temperatures averaging 25-32˚C. Annual rainfall is around 2,120mm, with a dry season from December to May and a rainy season from June to November. The transition between the two seasons varies slightly from year to year and across the Property. The rainy season is marked by generally clear mornings and daily cloudbursts in the afternoon, which are often quite heavy. Field work can be performed year-round.

5.3 Physiography

Rosita is located along the break between the hilly interior highlands and the flat Atlantic Coastal Plain. The topography in the highlands is gentle to steep hills that range in elevation from 100 to 1,000 metres above sea level. The Atlantic plain is found in the Rosita area and is flat to gently undulating and poorly drained with an elevation range of 50 to 250 metres above sea level. The area is drained by the Bambana and Banacruz Rivers.

5.4 Local Resources and Infrastructure

The town of Rosita is serviced by a municipal water system via a local reservoir. Service is unreliable, and consequently, shallow wells provide much of the local domestic water supply. Water for industrial use and drilling is readily available and plentiful in Rosita but is less reliable in the dry season. Water for future mining and milling operations will also be available from the old water-filled Santa Rita and R-13 pits.

Telephone service is provided by landlines through the national telephone company, Enitel. As well, cell phone and internet coverage is good in Rosita and along the major transportation routes. Satellite communication services are provided by a number of smaller companies.

Aside from mining, the principal economic activities in the Rosita area are logging, small scale farming, livestock and service industries. Unskilled labour is plentiful and most jobs can be filled using local workers. Some skilled workers are available having developed their skill sets by working at the various mines in Nicaragua.

6 HISTORY

6.1 Exploration History

The historical exploration activities over the Rosita D Concession are summarized in Table 6.1.

TABLE 6.1

EXPLORATION HISTORY SUMMARY

Year	Exploration Activities	Company
1906-1912	Exploration and Mining production	Eden Mining Company
1916-1918	Tunnel and drilling	Tonopah Nicaragua
1950	Tunnel sampling and diamond drilling	La Luz Mines Ltd.
1955	Diamond drilling	La Luz Mines Ltd.
1963-1965	Magnetic and radiometrics survey	Hunting Survey Corp
1969	Electromagnetic and magnetic survey	Geoterrex Ltd.
1974-1979	Exploration drilling	Rosario Resources Corp
1981-1983	Geophysical survey, soil sampling and diamond drilling	E.K. Lehman and Associates
1996-1998	RC drilling, Geophysical survey, soil sampling	Greenstone Resources Ltd.
2008	Mapping and rock sampling	Yamana Nicaragua S.A
2010-2011	Trenching, mapping, soil sampling, rock sampling and diamond drilling	Calibre Mining Corp
2011-2012	Channel sampling and RC sampling on stockpiles, geophysics survey , diamond drilling and trenching on exploration targets	Alder Resources Ltd
2012	Technical Report on the Copper-Gold-Silver Porphyry/Skarn Project at the Rosita D Concession	Carter.G.S
2012	NI43-101 Technical Report on mineral resource estimate of Rosita stockpiles	Wu,Y.

Source: Wu 2016

6.2 Production History

Mining and milling at Rosita were reportedly commenced in 1906 and continued for 6 years. Originally gold was only recovered from oxidized material near surface. No production figures are available.

In 1954, La Luz Mines Ltd. acquired ownership from Tonopah Nicaragua Company and a 600 ton mill was constructed in 1959, designed to use the leach-precipitation-flotation process.

According to P.A. Bevan (1973), from March 1959 to September 1971, the mill had treated 3.8 million tons[1] of ore with a grade of 3% copper and yield of 175 million lbs of copper, 123,000 ozs of gold and 1.8 million ozs of silver. From 1959 to 1964, more than 650,000 tons of carbonate ore were treated by the mill. The ore minerals were mainly malachite with some azurite, chrysocolla,

[1] The reference Bevan (1973) reported imperial tons and all tons in this section on Production History are also Imperial tons.

chalcanthite, tenorite, cuprite and native copper. The grade of was over 5% copper; material under 2% copper was stockpiled. Seventy per cent of the total copper in the heads was recovered.

In 1964, the mill circuit was changed to deal with the treatment of secondary sulphides, chiefly chalcocite, at an average of 900 tons per day. In 1967, primary sulphides started to appear in abundance and chalcopyrite was the chief mineral. Recoveries from ore produced in the east and west ends of the pit were roughly 80 per cent; recovery from the central zone was 50-60 per cent. In 1970, the production expanded to 2000 tons daily. The mine was closed in 1975 due to low copper price and civil unrest.

According to the previous NI 43-101 technical Report (Carter, 2012), the total historical production from 1959 to 1975 was 111,000 tonnes of copper, 160,000 ounces gold and 2,610,000 ounces silver from 5,373,587 tonnes of ore with average grades of 2.06% copper, 0.93 g/t gold and 15.08 g/t silver. The Author of this Report has not verified these records.

A few local artisanal miners are currently working on the North and South stockpiles. The work primarily consists of sieving and sluicing the stockpiles for gravity-recoverable gold. The material collected is either processed on-site using small scale mercury extraction, or shipped off-site to other known mills in the region. The Nicaraguan mining law states that 1% of mining concessions must be made available to local artisanal miners using traditional methods. The concession holder reserves the right to choose which 1% is made available and active miners must relocate at the company's request.

6.3 Previous Resource Estimate

Coffey Mining retained by Alder completed an initial NI 43-101 compliant Mineral Resources Estimate on the Rosita stockpiles in May 2012 (Table 6.2). The resources of the stockpiles were estimated using 55 RC drill holes and 17 channels.

TABLE 6.2

MINERAL RESOURCE STATEMENT FOR ROSITA STOCKPILES AT 0.15% CUEQ CUT-OFF GRADE

STOCKPILE	Resource Category	Tonnage (Mt)	Cu (%)	Au (g/t)	Ag (g/t)	CuEq (%)	Copper (Mlb)	Gold (oz)	Silver (oz)
NORTH	Inferred	3.33	0.78	0.58	10.3	1.25	56.99	62,100	1,100,900
SOUTH	Inferred	2.20	0.33	0.49	5.1	0.69	16.16	34,700	360,000
NORTHEAST	Inferred	0.55	0.50	0.22	9.6	0.75	6.06	3,800	168,300
EAST	Inferred	1.88	0.71	0.30	12.0	1.03	29.33	17,900	725,100
TOTAL	Inferred	7.95	0.62	0.46	9.2	1.01	108.54	118,500	2,354,300

Source: "NI 43-101 Technical Report on Mineral Resource Estimate of Rosita Stockpiles" (Wu, 2012). The CuEq cut-off was calculated using copper price of US$2.90/lb, a gold price of US$1,200/oz and a silver price of US$24/oz.

7 GEOLOGICAL SETTING AND MINERALIZATION

7.1 Regional Geology

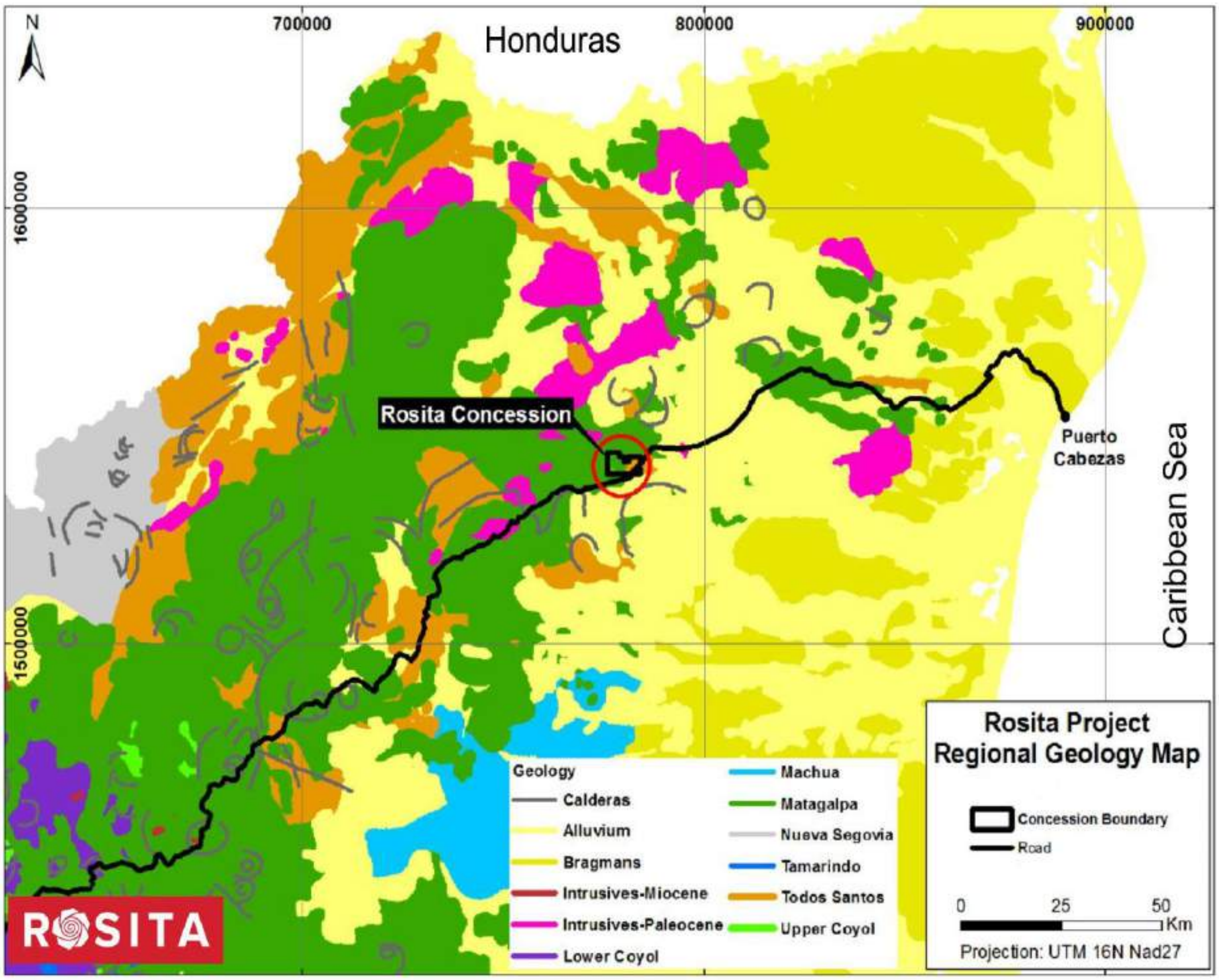
The geology of northeast Nicaragua is illustrated in Figure 7.1. Northeast Nicaragua lies within the eastern extension of the North Interior Highlands geomorphic province. Limited exposures of ultramafic rocks indicate that portions of the region are underpinned by oceanic crust of postulated Mesozoic age. These rocks are overlain and in fault contact with an interbedded sequence of limestone, mudstone, tuffaceous shale, greywacke, and marl of the early Cretaceous Todos Santos Formation. The sedimentary rocks are locally interbedded with andesitic tuffs and flows, and in places intruded by subvolcanic andesite dikes and sills, also of Cretaceous or perhaps lower Tertiary age and later stocks and plugs that include diorite, quartz diorite, granodiorite, quartz monzonite, and granite. Extensive accumulations of largely andesitic flows, breccias, and tuffs, commonly mapped as Tertiary Matagalpa Formation, cover much of eastern Nicaragua, commonly concealing these older lithologies.

In northeast Nicaragua the Todos Santos Formation occurs in three main areas. To the west of the Property they form a nearly continuous trend within the Iyas-Bocay Graben structure. To the east of the Property this sequence is exposed as a series of northeast-trending, isolated erosional windows within pre-Tertiary and Tertiary volcanics and intrusives; the Rosita D concession occurs within this area. The third area is about midway between the Property and the Caribbean coast, where Cretaceous limestone occurs in an east-west trending window within the volcanics and younger sedimentary rocks.

The complex interplay between plate tectonic structural elements has resulted in several compression and extensional events. One of the earliest structural elements in the region is a north trending anticline-syncline couplet formed in the Cretaceous age sedimentary rocks. Age dates in the Siuna area indicate that this folding, as well as emplacement of mineralization, occurred in the upper Cretaceous. Several episode of Tertiary age extensional tectonics are manifest in the Iyas-Bocay graben, and numerous prominent northeast-trending magnetic and topographic lineaments are also present.

The northeast-striking lineaments appear to be older and offset by other major northwest-trending faults and lineaments derived from satellite imagery and aeromagnetic data. Collectively the northeast and northwest fault and fracture patterns define a system of conjugate structures. In addition to these lineaments, there are a series of circular and semi-circular features in the region which vary from 1 to 25 km in diameter. These features are interpreted to be calderas, volcanic-intrusive related domal structures, stocks, and plugs. In the Rosita area, the intrusives collectively define a regional northwest trend.

FIGURE 7.1: ROSITA CONCESSION REGIONAL GEOLOGY MAP - Source: Wu (2016)



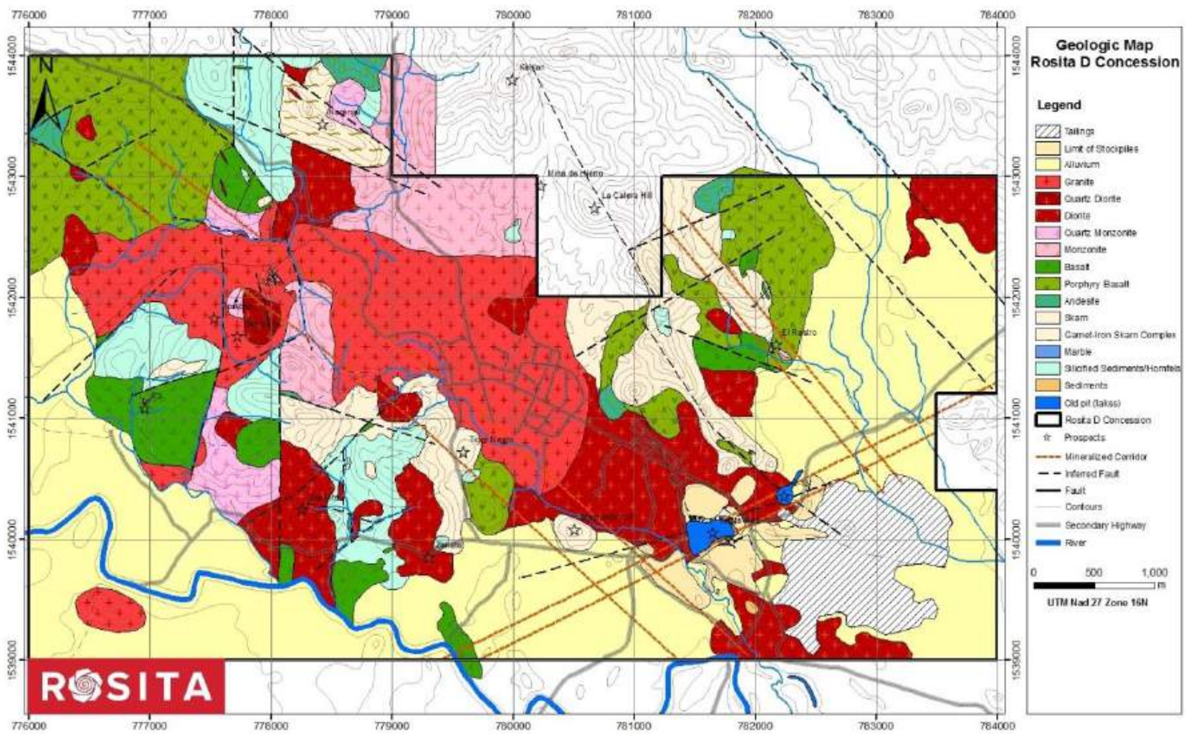
7.2 Local Geology

Rosita concession geology is presented in Figure 7.2. The eastern third of the property is underlain mainly by folded and faulted carbonate sedimentary rocks of the Todos Santos Formation. To the west are andesitic to basaltic volcanic rocks that have been intruded by a series of stocks and plugs including diorite, quartz diorite, granodiorite, quartz monzonite, and granite. Hydrothermal alteration associated with emplacement of the intrusives has led to the development of large areas of skarn and hydrothermally altered rock. Locally, tectonically emplaced bodies of Mesozoic ultramafic rock/ophiolite crop out in the area, and suggest that the region is at least partly underpinned by oceanic crust. The principal tectonic features in the Rosita area are a series of subparallel, east-northeast and northwest striking lineaments and faults. The most obvious of the northeast features is the Rosita Fault, a broad shear zone that can be traced for at least 3 km through the R-13 and Santa Rita pits toward the southwest. The east-northeast trending structures are locally displaced by northwest striking lineaments manifest as faults and trends of intrusive bodies. On a regional scale, the Rosita Fault forms a segment of a 45 km long lineament, defined by a series of magnetic lows. This feature is interpreted to be a deep crustal discontinuity that may represent the northeast edge of a crustal block (Leyton, 1994). The Rosita skarn and several other prospects occur along or proximal to this feature.

The geology of the Rosita mine, as described by Plecash and others (1963) and Bevan (1971), consisted of a plug of granite that intrudes the sedimentary and overlying volcanic rocks giving rise to garnet-epidote skarn, marble, and hornfels. A northeast-trending shear zone, The Rosita fault, which contains extensive brecciation and associated hydrothermal alteration, is believed to have been a major control for skarn formation at Rosita. Secondary shears and fracture zones striking northwest, in places cut this structure and appear to have guided the emplacement of feldspar porphyry and andesite dikes. The northwest-striking structures also appear to have promoted late-stage mineralizing events, some of which appear to post-date skarn development.

FIGURE 7.2

ROSITA CONCESSION GEOLOGY MAP Source (Wu 2016)



7.3 Mineralization

7.3.1 Santa Rita pit

The Santa Rita pit is a skarn type Cu-Au-Ag deposit (Bevan, 1973). The marble, garnet and epidote skarn rocks have been formed by the metamorphism of interbedded Cretaceous sediments of calcareous and siliceous nature and andesitic volcanics. The metasomatism was brought about by Tertiary intrusions, mainly diorite and monzonite. The regional strike is approximately northeast.

The main mineralization lies on the southern flank of a small dioritic intrusion. In the mine the favourable garnet skarn horizon is about 152m thick, strikes easterly and dips 50 degrees to the southeast. It is underlain by altered diorite and overlain by chloritized andesites and calcareous tuffs. Intense lime, potash and siliceous metasomatism have altered the calcareous sediments to marble or to garnet-quartz-calcite-epidote-orthoclase-pyrite skarn. The interbedded volcanic and andesitic and dioritic dykes have been altered in many cases to epidote skarn and in others to siliceous skarn.

Garnet skarn is the host rock for the mineralization. Red, brown, yellow and green varieties of garnet are present. The mineralization zone occurs as lenses, pods and stringers of massive sulphides in well-fractured or brecciated skarn. There is commonly more chalcopyrite than pyrite. Massive pyrrhotite occurs in one zone on the north side of the pit near the footwall. Gold values are localized by a north-northwest-trending fault.

In the central part of the pit there is a quartz-garnet skarn breccia zone with finely disseminated pyrite and chalcopyrite. The garnet is chiefly red or red-brown. The zone itself might be a breccia pipe of the Cananea type. In the east end of the mine the garnet skarn is mainly composed of the yellow variety, particularly adjacent to bands or masses of marble. The mineralization may be disseminated or massive chalcopyrite, often associated with chlorite, magnetite, pyrrhotite and pyrite. It may also occur as lenses or veins of quartz-chalcopyrite-pyrite.

The mineralization zones appear to have been localized in part by two major fault systems: (a) north-northwest-trending shears and quartz stringers and replacement zones with steep dips; and (b) northeast-trending shear zones which offset the north-northwest faults. Stubby east-west breccia zones feather out from the northeast trending shears.

Capping the three primary sulphide zones were secondary enriched zones of chalcocite, dipping southwest, and oxidized zones composed principally of malachite. Other copper minerals noted include native copper, cuprite, azurite, chrysocolla, chalcanthite, covellite, tenorite and "grey coppers".

7.3.2 R-13 Pit

The R-13 Zone is a northeastern extension of the Santa Rita mineralized zone. The deposit contains copper, silver and gold concentrations in a northwest trending shear zone hosted exclusively within an intensely fractured and propylitized quartz diorite. The main hypogene minerals found in the drill cuttings, in order of decreasing abundance, are reported as: quartz, pyrite, chalcopyrite and bornite. Pyrite in the R-13 deposit occurs as discrete grains in quartz-pyrite veinlets and in fracture zones containing massive chalcopyrite and quartz. Chalcopyrite is not as widespread as pyrite and is concentrated along the main northwest shear zone. Argentite is identified as the main silver mineral in the R-13 deposit. Gold in the fracture zones is closely associated with copper and silver. Drilling has shown that this relationship is confined to intervals of silicic alteration within a propylitically altered quartz diorite. This spatial association suggests that the gold was deposited during a late stage or completely separate hydrothermal event in the Rosita Fault.

7.3.3 Other Mineralization on Some Exploration Targets

A zone of a superficial supergene enrichment present above a porphyry-type Cu-Au-Ag mineralized monzonite intrusion at Tipispan area, which was encountered in trenches and drill holes.

T3 is a secondary copper mineralization zone on a south facing slope in the western part of the Rosita D Property. Historically this area had been subject to selective mining. Trenching, soil sampling, IP geophysical survey and drilling indicated the presence of an exotic copper deposit on the side of a hill.

8 DEPOSIT TYPES

The main types of deposit on the Rosita property are Cu-Au-Ag skarn at Santa Rita, R-13 and Tigre Negro, Fe-Cu-Au skarn at Magnetite Hill and Cu-Au-Ag porphyry at Bambana (Tipispan and T3 area). The skarn deposits are characterized by calc-silicate metasomatism, retrograde alteration and silicification. The porphyry copper mineralization at Bambana is characterized by propylitic, silicic and potassic alteration.

Skarn deposits form through the physical and chemical reaction between igneous rocks intruded into calcareous sedimentary rocks. They occur adjacent to (exo-skarn) or within (endo-skarn) an intrusive body. Emplacement of the intrusive is controlled largely by transfer structures in the back-arc basin as well as splays along arc parallel structures in the magmatic arc environment. Alteration zone is controlled by the temperature gradient and is overprinted by metasomatic and retrograde alteration. Mineralization is commonly vertically zoned from chalcopyrite-magnetite to chalcopyrite-bornite-gold-pyrite to pyrite-chalcopyrite. The copper-gold-silver deposit at the Santa Rita pit is examples of skarn mineralization.

The targets of this resource study are tailings and six historical low grade stockpiles around the Santa Rita pit. Each stockpile (North, Northeast, East, South and Southwest) was named based on the direction to the Santa Rita Pit. RST believes the stockpiles were originally derived from Santa Rita pit.

Based on P.A. Bevan reporting, during the production from Santa Rita mine, material containing less than 2% copper was stockpiled. All the stockpiles are mixtures of oxide and sulphide materials and from clay to boulder size. The ore minerals are mainly malachite, chalcocite and chalcopyrite with some azurite, chrysocolla, chalcanthite, tenorite, cuprite, native copper and native gold.

9 EXPLORATION

9.1 Channel Sampling

RST (previously known as Alder Resources Ltd) completed a program of vertical channel sampling around the fringes of four stockpiles in October and November 2011. A total of 236 samples from 17 channels and were collected; channel locations are provided in Figure 9.1.

FIGURE 9.1

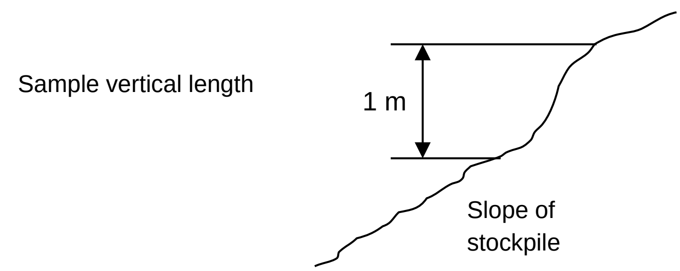
LOCATION of CHANNELS

Source: Wu (2016)

Prior to taking the channel sample, the surface was cleaned to remove the transported material on the stockpiles. The interval of each sample was marked on the ground with paint, based on a one meter vertical length. A channel of approximate 10cm depth and 10cm width was excavated for sampling. The sample length on the ground varied with slope angle but all samples had equal vertical length of 1m as indicated in Figure 9.2. Samples were continuously collected along the stockpile slope from top to bottom. Each sample of approximately 5 kg was weighed, bagged, labelled, sealed and sent for analysis. Sampling was briefly logged to record the material type.

FIGURE 9.2

CHANNEL SAMPLE METHOD



Source: Wu (2016)

9.2 Survey

The survey coordinates system using on the project is UTM (NAD27, zone 16N, Central America).

Jairo Camilo Perez Pastrana, a qualified surveyor of Nicaragua with identification of 321-020871-0001E was commissioned to perform the stockpile topographic survey in 2011. The survey was carried out with total station Sokkia Model 650 RX.

Channel sample locations were surveyed by the RST field crew with a handhold GPS, therefore, the channels could not be properly projected on the topography surface during the course of this resource modeling. The Author of this Report adjusted the coordinates of the channel samples to match the topography surface which was created based on the survey data.

In the opinion of the QP, the method of channel sampling met the project purpose, however, the survey by handhold GPS was not industry standard practice. The main difference between the handhold GPS and total station survey was in elevation reading (Z), the differences of X and Y reading were in an acceptable range. The QP believes that the adjusted coordinates of channel samples are relatively reliable to perform resource estimation; however, it is suggested that all sample locations should be surveyed by qualified surveyor(s)

9.3 Density Measurement of the Stockpiles

9.3.1 Mini Bulk Density Sampling of the Stockpiles

A total of 64 wet density samples have been tested in 2012 at 32 localities on five stockpiles. Near-vertical channel samples were collected over the stockpiles into a 20-litre plastic bucket, using a geologist's rock hammer and shovel. Care was taken to ensure that possible voids in the bucket were filled with stockpile material. All samples were compressed into the sampling bucket, to try and replicate the compacted nature of the stockpile material. Excess material at the top of the bucket was scraped off to form a level upper surface, representative of the known sample volume.

The bucket was weighed on-site using a hanging “watch type” spring balance. Its weight in kilograms (minus the tare weight of the bucket), sample location and characteristics were recorded into a field notebook. Two samples were collected at each locality within approximately 5 meters of one another, to test for local density variability.

This sampling technique is fast, allowing many measurements to be obtained over the stockpiles. Shortcomings of this method are that large boulders found occasionally in the stockpiles could not be included in the sample, and it is also likely that the sample material in the bucket is slightly less compacted than the “in-situ” stockpile material. Both factors will tend to produce a bulk density measurement slightly lower than the “in-situ” density for the stockpiles. Table 9.1 summarizes the results.

TABLE 9.1

MINI BULK DENSITY OF STOCKPILES

Stockpile ID	# of Mini Bulk Samples	Average Wet Density (t/m^3)
North Stockpile	30	1.97
South Stockpile	14	2.04
Northeast Stockpile	10	1.96
R-13 Stockpile	8	1.79
Southwest Stockpile	2	2.13
Overall Average	64	1.97

Source: Wu (2016)

9.3.2 Bulk Density Sampling

As recommended by the Author of this Report during the site visit on March 22, 2012, RST has completed a total of 8-dimensional excavation bulk samples over three stockpiles in 2012. The samples were excavated in dimension of 1m x 1m x 0.25 - 0.30m. The weights for the material excavated ranged from 1,123 lbs (509.5 kg) to 1,491 lbs (676.5 kg). Bulk density results are listed in Table 9.2. Four samples measured in North stockpile are showing consistent value of 2.03 - 2.32g/cm^3 with averaged wet density of 2.15g/cm^3. There is considerable variability in the Southwest stockpile, with range from 1.80 to 2.97 g/cm^3, for an average of 2.53 g/cm^3. The Mini bulk density above also illustrated the Southwest stockpile has the highest density. The field observation noticed that there are more, large-sized fresh rock boulders in Southwest stockpile than the other ones, which may explain the higher density on Southwest stockpile.

TABLE 9.2 BULK DENSITY OF STOCKPILES	
Location	Density (g/cm$^{3)}$
South Stockpile	1.98
North Stockpile	2.06
North Stockpile	2.03
North Stockpile	2.32
North Stockpile	2.18
Southwest Stockpile	2.82
Southwest Stockpile	1.80
Southwest Stockpile	2.97

Source: Wu (2106)

9.3.3 Moisture

Table 9.3 shows the measured moisture content of the stockpile materials. The moisture samples were collected in eight 20-litre buckets and sent to Inspectorate's laboratory for dry processing in 2012. Samples were oven dried at 60°C in Inspectorate's laboratory; and the weights were determined before and after the material dried. The average water content for the 8 samples is 9.37%.

TABLE 9.3 MOISTURE CONTENT OF THE STOCKPILES	
Location	Moisture (%)
South Stockpile	7.6
South Stockpile	4.8
Southwest Stockpile	6.8
Southwest Stockpile	18.33
North Stockpile	9.82
Northeast Stockpile	8.31
R-13 Stockpile	10.37
East Stockpile	9.47

Source: Wu (2016)

9.3.4 Comment on the density measurement

The bulk density measurements were not sufficient to cover all stockpiles; only 8 bulk density samples over 3 of 6 stockpiles have been completed. Mini bulk samples tend to undervalue the density of stockpiles due to compaction and large sized material bias. There are no density samples taken from the East stockpile at all. It is recommended that RST carry out the bulk density sampling over all stockpiles in multiple locations, along with moisture testing.

9.4 Density Measurement of the Tailings in 2013

RST measured 19 samples from 17 sites within the resource estimate area, and returned an average density of 1.818 t/m^3. The density tests were performed using a metal box of known dimensions not sealed on the bottom to allow the penetration into the tailings. The tailings of the contained within the box were removed from ground and weighed. The density was calculated using the formula: density = mass/volume. All samples were taken at depth not exceeding 1 metre.

As a check, six tailing samples were taken and submitted to Ingenieria de Materiales y Suelos S.A. in Managua for density testing, and the average density was 1.821 t/m^3. A density value of 1.82 t/m^3 was applied for this resource estimate. Density test results are presented in Table 9.4.

TABLE 9.4

DENSITY MEASUREMENT OF TAILINGS in 2013

Tested By	Number of Samples	Minimum Value (t/m^3)	Maximum Value (t/m^3)	Average (t/m^3)
RST	19	1.58	2.19	1.82
Laboratory	6	1.44	2.03	1.82

Source : Wu (2016)

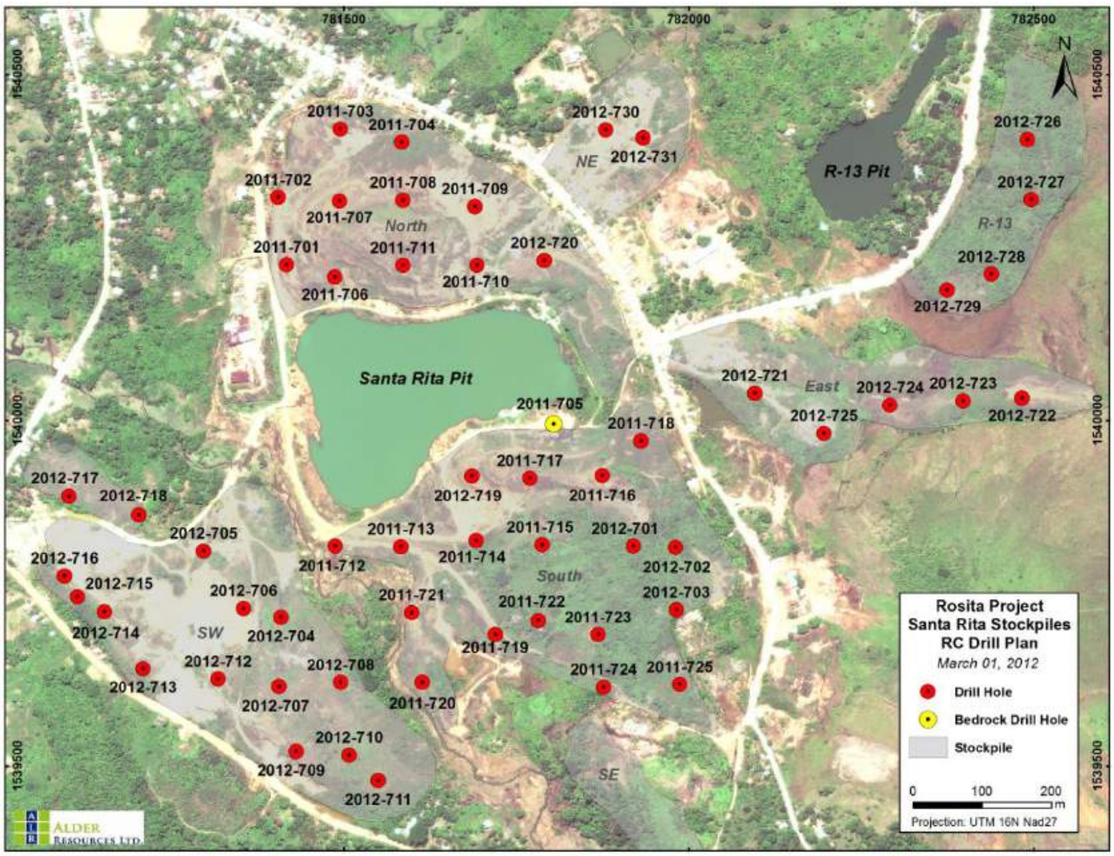
10 DRILLING

10.1 2011-2012 Drilling Program for Stockpiles

Rosita Mining, previously Alder Resources, initiated a reverse circulation drilling program in November 2011 and completed in February 2012. The purpose of the RC drilling program was to delineate the grade and size of the stockpiles. A total of 55 RC holes totalling 1,574.77m were drilled on the stockpiles, of which 24 drill holes completed in 2011 and 31drill holes in 2012. Drill hole locations are shown in Figure 10.1.

FIGURE 10.1

RC DRILL HOLE LOCATION ON THE STOCKPILES (2011-2012)



Source : Wu(2016)

The drillhole grid was planned at 100m spacing for each stockpile; the actual spacing range was 35 - 169m. To assist in mapping and interpreting in situ mineralization, all the drill holes were drilled into bedrock at 1.52 - 18.24m; 76% of drill holes penetrated 3 - 6m into bedrock. More than 99% of sample lengths were 1.52m, ranging 0.67 - 1.58m. Drill hole depth ranged from 6.1 to 54.9m and 59% of drill holes were10 - 30m deep.

A button bit, down-hole pneumatic hammer and 5 inch tricone reverse circulation (RC) drill was employed to perform the drilling. The cuttings were collected into a 50lb bucket through a cyclone. Each bucket was cleaned before filling with sample. Drill rods were cleaned between each sample using a blower. Each sample was weighed and large volume samples were split on-site with a splitter. Each sample was packed in a plastic sample bag with sample number labelled and sealed using zip tie. Samples were packed in sacks and shipped to the laboratory in Managua by truck.

Once the drill hole was finished, a concrete slab was constructed at the collar position with drill hole ID marked on it.

The QP of this Report confirmed with RST staffs that drill holes were cleaned by blowing between each sample.

Collars of 52 RC drill holes on the stockpiles were surveyed by a qualified surveyor using total station survey, along with the topography of the stockpiles. Elevations of some drill holes were slightly adjusted by the QP to match the topography during this resource estimation.

10.2 2015 Drilling Program for Stockpiles

A reverse circulation drilling program was carried out by Continental Drilling (Aquatec S.A) from August 31st to October 10th, 2015. A total of 83 drill holes, aggregating 1939.20 meters, have been completed, of which 55 vertical holes totaling 1,040 meters drilled on the North, North2, South, South West and East stockpiles (see Table 10.1 and Figure 10.2), while 899 meters were exploration drilling to test the near surface Cu enriched mineralization on the adjacent R-13/R-13 West and Tipispan copper-gold-silver zones.

The RC holes were drilled through the base of the stockpile material ranged from 12 to 34.5 meters in depth, except hole 2015-709 and 2015-710 which were terminated within the North stockpile. Protocols of drilling, logging and sampling were same as that used for 2011-2012 drilling program. Drill hole logging and sampling were performed by Rosita geologists and assistants. The drill location was surveyed by employees of RST using a hand hold GPS. The elevations of all drill holes were adjusted against the surveyed topographic surfaces of the stockpiles for this resource estimate.

TABLE 10.1 RC DRILL HOLES FOR 2015 STOCKPILES SAMPLING PROGRAM				
Stockpiles	# of Holes	Metres drilled	Range of depth (m)	No. of Samples
North	11	240.1	15-28.5	134
North2	2	33	16.5	15
South	11	175.5	15-18	84
East	7	177	16.5-34.5	106
SW	24	414.7	12-27	251
Total	55	1,040.3	12-34.5	590

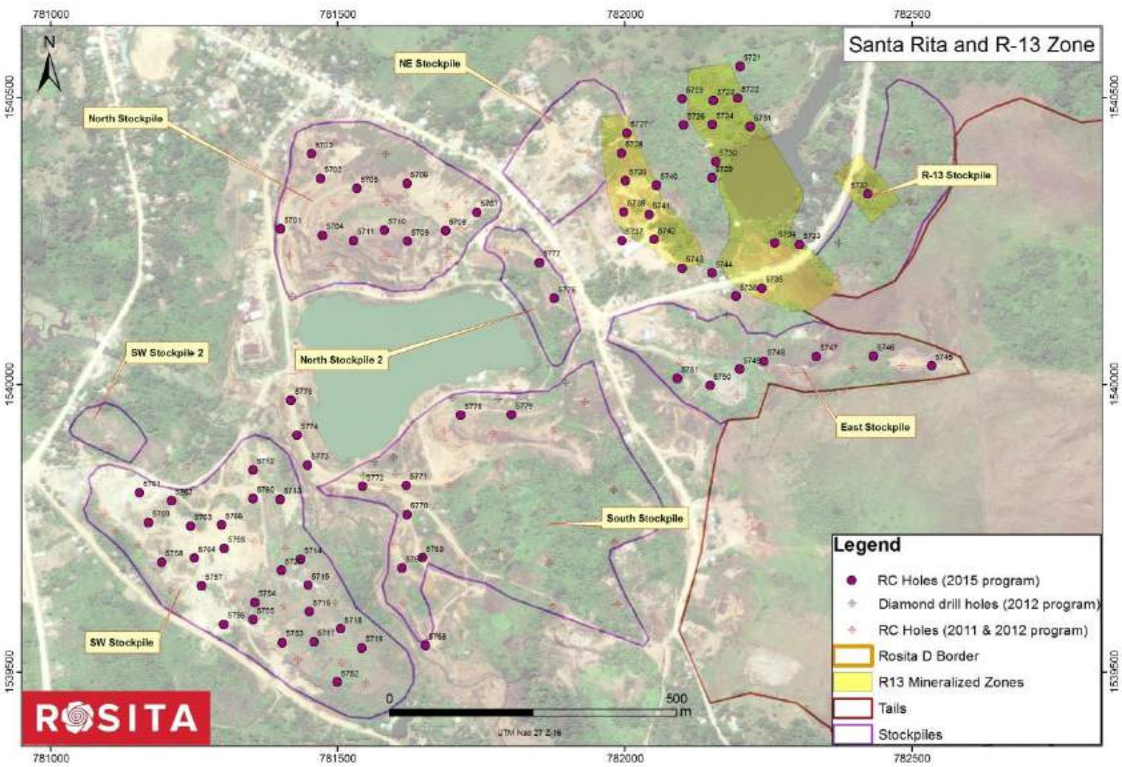
Source: Wu(2016)

Samples were taken at 1.5m interval down hole. Sample weights varied from 1lb to 555lb with average weight of 35lb and the weight ranges of samples are presented in Figure 10.3. There was one sample weighing 555lb due to caving, which was not mineralized material at Southwest stockpile. 56 out of 506 (11%) intervals within the stockpiles were not sampled due to poor recovery.

Selected mineralized intersections from 2015 RC holes are summarized in Table 10.2.

FIGURE 10.2

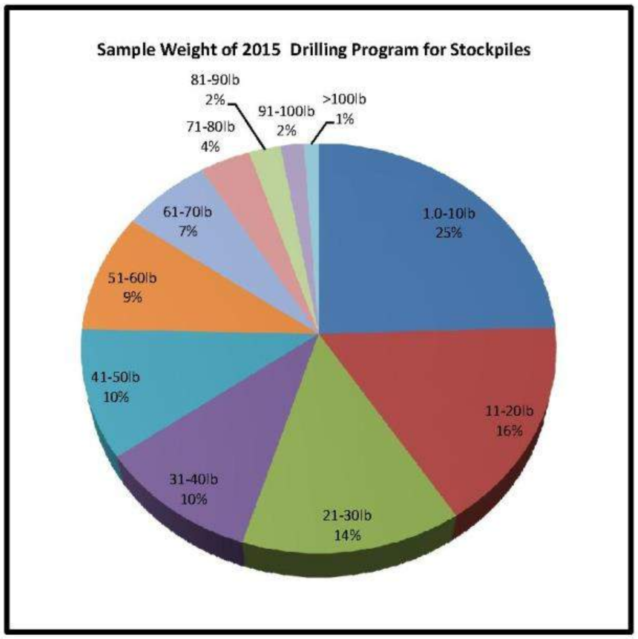
2015 RC DRILL HOLE LOCATION ON THE STOCKPILES



Source: Wu (2016)

FIGURE 10.3

SAMPLE WEIGHTS OF 2015 DRILLING PROGRAM FOR STOCKPILES



Source: Wu (2016)

TABLE 10.2

SELECTED MINERALIZED INTERSECTIONS FROM 2015 RC HOLES

HOLE-ID	FROM	TO	LENGTH	Au g/t	Cu%	Ag g/t	STOCKPILE
2015-701	0.00	6.00	6.00	0.96	0.43	33.40	NORTH
2015-702	0.00	7.50	7.50	0.67	0.86	8.60	NORTH
2015-702	9.00	13.50	4.50	0.66	0.45	3.23	NORTH
2015-703	0.00	6.00	6.00	0.61	0.39	7.35	NORTH
2015-703	7.50	19.50	12.00	0.63	0.57	4.96	NORTH
2015-704	1.50	3.00	1.50	1.01	0.77	14.40	NORTH
2015-704	4.50	6.00	1.50	1.58	0.85	24.00	NORTH
2015-704	7.50	13.50	6.00	2.50	1.17	17.65	NORTH
2015-704	16.50	24.70	8.20	0.74	0.67	12.57	NORTH
2015-705	1.50	3.00	1.50	0.51	0.19	3.20	NORTH
2015-705	6.00	7.50	1.50	1.41	0.82	32.40	NORTH
2015-705	9.00	24.00	15.00	1.13	0.47	11.14	NORTH

TABLE 10.2

SELECTED MINERALIZED INTERSECTIONS FROM 2015 RC HOLES

HOLE-ID	FROM	TO	LENGTH	Au g/t	Cu%	Ag g/t	STOCKPILE
2015-706	0.00	6.00	6.00	0.11	0.51	3.70	NORTH
2015-706	7.50	25.50	18.00	0.56	1.08	12.00	NORTH
2015-707	0.00	3.00	3.00	0.09	0.50	36.25	NORTH
2015-707	6.00	10.50	4.50	0.05	1.77	9.33	NORTH
2015-707	12.00	16.50	4.50	0.19	1.69	14.07	NORTH
2015-708	0.00	18.00	18.00	0.191	1.987	17.625	NORTH
2015-709	6.00	16.50	10.50	0.37	1.31	20.44	NORTH
2015-710	4.50	7.50	3.00	1.37	0.47	7.35	NORTH
2015-710	9.00	10.50	1.50	0.583	0.970	7.200	NORTH
2015-710	12.00	19.50	7.50	0.507	1.656	10.840	NORTH
2015-711	3.00	12.00	9.00	0.81	2.37	15.75	NORTH
2015-711	13.50	24.00	10.50	0.80	1.33	12.16	NORTH
2015-712	0.00	4.50	4.50	0.21	0.71	12.20	SW
2015-713	0.00	6.00	6.00	0.54	0.20	3.90	SW
2015-714	9.00	12.00	3.00	1.11	0.45	8.95	SW
2015-715	0.00	21.00	21.00	0.33	0.19	2.62	SW
2015-716	1.50	19.50	18.00	0.50	0.21	1.95	SW
2015-717	0.00	15.00	15.00	0.14	0.18	5.23	SW
2015-718	0.00	15.00	15.00	0.20	0.21	4.96	SW
2015-719	0.00	25.50	25.50	0.22	0.25	3.74	SW
2015-720	0.00	15.00	15.00	0.60	0.27	2.56	SW
2015-745	0.00	15.00	15.00	0.40	0.16	2.33	EAST
2015-747	0.00	1.50	1.50	0.035	0.741	3.000	EAST
2015-747	7.50	15.00	7.50	0.17	0.39	5.86	EAST
2015-748	1.50	3.00	1.50	4.778	1.070	13.000	EAST
2015-748	9.00	12.00	3.00	0.300	0.613	20.250	EAST
2015-749	0.00	10.50	10.50	0.14	0.52	6.81	EAST
2015-750	0.00	15.00	15.00	0.21	0.62	11.00	EAST
2015-750	16.50	22.50	6.00	0.32	0.20	6.17	EAST
2015-751	4.50	13.50	9.00	0.25	0.40	12.65	EAST
2015-752	0.00	21.00	21.00	0.33	0.21	3.78	SW
2015-753	0.00	15.00	15.00	0.22	0.23	3.72	SW
2015-754	0.00	15.00	15.00	0.25	0.16	2.30	SW

TABLE 10.2

SELECTED MINERALIZED INTERSECTIONS FROM 2015 RC HOLES

HOLE-ID	FROM	TO	LENGTH	Au g/t	Cu%	Ag g/t	STOCKPILE
2015-755	0.00	9.00	9.00	0.10	0.34	6.85	SW
2015-756	0.00	15.00	15.00	0.50	0.29	4.98	SW
2015-757	0.00	10.50	10.50	0.38	0.12	1.47	SW
2015-758	4.50	9.00	4.50	0.25	0.24	5.43	SW
2015-759	12.00	16.50	4.50	0.19	0.25	7.53	SW
2015-760	0.00	10.50	10.50	1.82	0.41	4.17	SW
2015-762	0.00	1.50	1.50	0.521	0.047	1.300	SW
2015-763	4.50	7.50	3.00	0.51	0.09	0.45	SW
2015-764	0.00	1.50	1.50	0.240	0.175	2.300	SW
2015-766	1.50	10.50	9.00	0.40	0.14	0.67	SW
2015-767	1.50	12.00	10.50	0.25	0.26	2.19	SOUTH
2015-768	0.00	4.50	4.50	0.33	0.15	3.13	SOUTH
2015-769	0.00	6.00	6.00	0.28	0.25	3.10	SOUTH
2015-770	0.00	6.00	6.00	0.49	0.66	4.70	SOUTH
2015-771	0.00	16.50	16.50	0.60	0.23	3.09	SOUTH
2015-772	0.00	9.00	9.00	0.49	0.20	1.77	SOUTH
2015-776	9.00	10.50	1.50	0.268	1.130	4.300	North2
2015-777	0.00	9.00	9.00	1.37	0.54	7.44	North2
2015-778	0.00	16.50	16.50	0.43	0.61	5.85	SOUTH
2015-779	0.00	15.00	15.00	0.33	0.10	3.09	SOUTH

Source: Wu (2016)

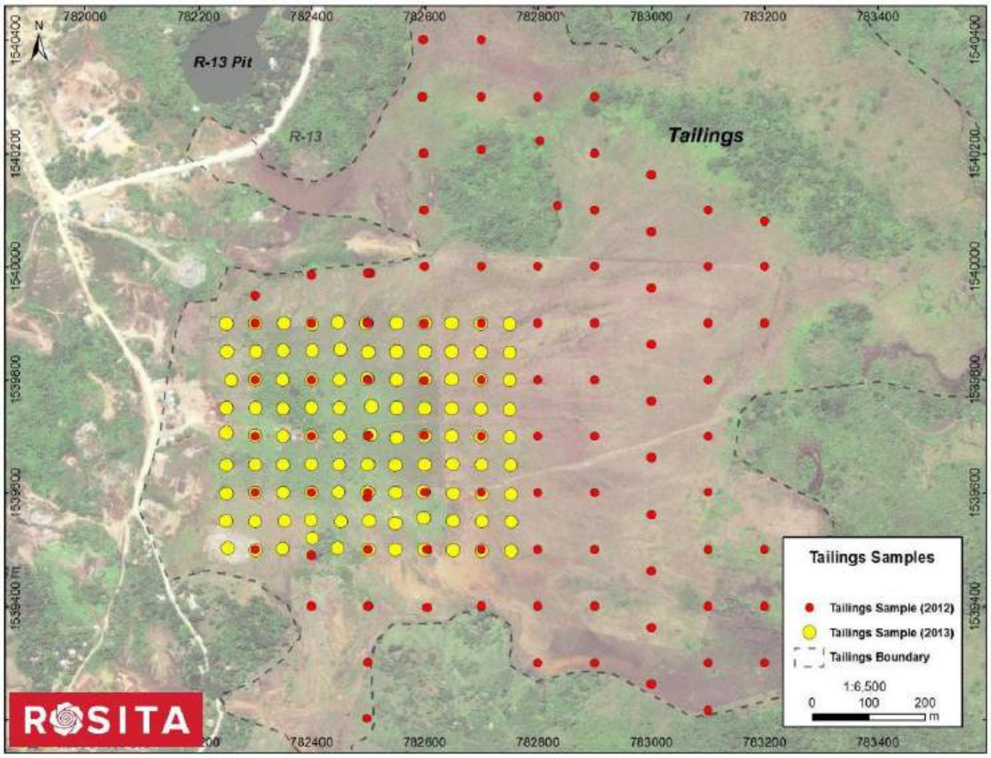
10.3 2012-2013 Sampling Program for Tailings

Sampling of the tailings was carried out in two campaigns (Figure 10.4). The initial campaign was carried out in March to May of 2012 in an area approximately 1.2 km long (North-South) by 1.0 km wide (East-West), and centered about 1 km east-southeast of the Santa Rita pit. Sampling was conducted on a 100m by 100m grid using a 3 inch auger. A total of 191 samples from 100 auger holes were collected. The depth of holes varied from 0.7 to 3.7m and aggregated 284.54m. All holes were terminated in the tailings.

The second campaign was carried out from May to July 2013 within a 400m by 400m area of the first campaign (Figure 10.4). Sampling was executed on a 50m by 50m grid using a 3 inch auger which was reduced to 2.2 inch within 2.5 inch PVC casing. 53 out of 81 holes were drilled to depth of 6m, and maximum depth was 7.2m. Samples were collected at 2m intervals and a total of 208 samples were collected at 81 sites, totaling 440.35m. Most of the auger holes were unable to reach the bottom of the tailings with the 6.0m penetration depth; however, bedrock was intersected at shallower depths near the southwest edge of the tailing field.

FIGURE 10.4

2012-2013 AUGER SAMPLING LOCATION OF ROSITA TAILINGS



Source: Wu (2016)

Sampling protocol was implemented as following:

- Auger hole location was spotted using a hand-held GPS as planned.
- An area of 3 metre radius around the hole location was cleaned.
- A 3-inch diameter auger was used to drill to a depth of 2.0 m and sampled from 0 to 2 metres as sample one.
- A 6.0-meter length of 2.5-inch diameter PVC tube was installed in the auger hole by percussion from a 6.0 m high scaffold. The tube was driven down until a 10-cm lip remaining above the

ground level. Given the use of PVC tubes, the volume of water was less and the collapse of the hole did not occur.

- While the hole advanced, the auger was reduced to 2.2-inch diameter to fit in the PVC casing.
- All materials within the PVC tube was extracted and sampled at 2 m interval, sample two from 2 to 4 m depth followed by sample three from 4 to 6 m.
- The sample was collected in a plastic bag, labeled and sealed under the supervision of a geologist of RST.
- The auger and all tubes were cleaned whenever the sample was extracted to avoid the contamination.
- In general, the last sample interval was ended at 6.0 m or to the point where organic material encountered which marks the base of the tailings.
- Each sample site was backfilled and marked with a small labeled post.
- All the extracted materials were described by the geologist and photo archived.
- The samples were trucked to the RST office in Rosita and then shipped to the laboratory in Managua by RST staff.

The tailings contain a significant sand-size fraction (medium-grained and angular), dominated by quartz, garnet, calcite, epidote, pyrite, feldspar and magnetite along with clay components. Although only a thin (10 to 20 cm thick) soil horizon is developed over the years, the upper portion of the tailings is variably oxidized, and in places weakly cemented by limonite and hematite. Such oxidation is generally limited to the upper 0.7 to 1.0 metres, and further down, the tailings are pyritic, friable and water-saturated, behaving a lot like beach sand.

The elevation of the 2012 sampling location for the tailings was not recorded in the database and the topography of the tailings was not surveyed, therefore the 2012 samples were not used for this resource estimate, except six holes adjacent to the 2013 samples at east edge of the resource estimate area. As the QP suggested during the site visit, some of 2012 sample locations were surveyed by a RST geologist using a hand hold GPS, including the six holes used for the resource estimate.

The elevation of the 2013 sampling location was varied from 43m to 77m in the database, which is far off from the actual topographic undulation of the tailings. Therefore, the elevations of the sample points were adjusted to smooth the surface of tailing model for the resource estimate.

A recommendation to RST is that the topography and sample location of the tailings should be surveyed by a qualified surveyor in future.

In the opinion of the QP, the sampling program generally meets the industry standard and results are acceptable to support the resource estimate of the stockpiles. 2013 Sampling for the tailings can be used for Inferred resource estimate.

11 SAMPLE PREPARATION, ANALYSES AND SECURITY

11.1 Sample Preparation

All samples of stockpiles and tailings were submitted to Bureau Veritas Mineral Laboratories (previously known as Inspectorate America Corporation) for preparation in Managua, Nicaragua and analysed in Vancouver, Canada. The QP of this Report visited the preparation laboratory of Bureau Veritas in Managua, which is an ISO9001 certified lab.

The sample is prepared by the following steps:

- Once sample is received from the client, the laboratory sets up a project for the sample through the laboratory information system.
- Weigh the sample wet with sample bag and record the mass in the system.
- The sample is placed in clean metal trays with sample ID tracked by recording the tray numbers. Then the sample is dried in an oven for 12 hour at 60°C.
- The sample is crushed to +80% passing through 1.7mm square mesh sieve.
- The crushed sample is repeatedly split several times (depending on the sample size) until sample mass reaches 250 - 270g. The sample and residue are bagged separately and labelled with the sample ID. The residue is stored in the laboratory for 90 days and dispatched depending on the client's instruction.
- The 250g sample is pulverized to +85% passing -200 mesh.
- The sample is split into two 125g pulps and bagged separately with the sample ID labelled. One bag of pulp is sent to Bureau Veritas (Inspectorate) Vancouver laboratory for assay and another pulp is stored in the preparation laboratory for 90 days.

The crushers, splitters, pulverisers, sieves and workstation are cleaned by blowing air and with a silica wash after each sample. The laboratory has standard operating procedures displayed at each workstation. Quality control is undertaken in the laboratory by checking the size distribution regularly.

The QP is satisfied the sample preparation followed industry standard practice; the quality control and sample assurance are reasonably well performed.

11.2 Sample Analysis

The samples prepared in the Managua laboratory were shipped to analytical laboratory of Bureau Veritas (Inspectorate) in Vancouver for analysis. In its Vancouver laboratory, each sample was analysed for copper and silver using aqua regia digestion and a 30 element ICP-ES (inductively coupled plasma-atomic emission spectrometry) method, soluble copper using dilute sulfuric acid digestion with AA (atomic absorption) finish and gold using fire assay with AA finish.

11.3 Security

No special security measures were taken other than routine careful marking, handling, transportation and storage of samples. Samples were delivered to the Bureau Veritas Laboratories by RST employees.

11.4 Comments on Sample Preparation, Analyses and Security

Sample preparation, analyses, and security were generally performed in accordance with exploration best practices and industry standards.

12 DATA VERIFICATION

The Rosita project was visited by Mr. Yungang Wu, P.Geo., an independent Qualified Person in terms of NI 43-101, on two separate occasions, March 21-22, 2012 and November 6-7, 2015 for the purposes of completing site visits and due diligence sampling. General data acquisition procedures, hole logging procedures and quality assurance/quality control (QA/QC) were discussed with the RST staff during the site visits.

12.1 Independent Sampling in 2012

A total of 7 samples were collected by Mr. Wu during his site visit on March 21-22, 2012, of which 5 samples were from 5 different stockpiles and 2 samples from tailings. The samples presented in Table 12.1 were included in the 2012 stockpile resource estimate.

TABLE 12.1

DUE DILIGENCE SAMPLE RESULTS (MARCH 2012)

Sample ID	Location	Cu (%)	Au (g/t)	Ag (g/t)
1	North Stockpile	0.74	1.86	12.2
2	Southwest Stockpile	0.17	0.11	4.7
3	Northeast Stockpile	0.22	0.12	10.6
4	South Stockpile	0.42	0.27	5.0
5	East Stockpile	0.50	0.40	4.5
6	Tailing	0.05	0.19	14.0
7	Tailing	0.02	0.24	15.4

Source: Wu (2016)

Once the independent samples were collected and sealed, they were trucked to the Inspectorate laboratory in Managua. Chain of custody was maintained during shipment to the laboratories. All samples were registered and weighed while Mr. Wu was watching in the laboratory.

The independent samples gave similar results to the channel and RC samples and confirmed the mineralization of the stockpiles.

12.2 Independent Sampling in 2015

A total of seventeen (17) samples were taken by Mr. Wu during his site visit on November 6-7, 2015, which consisted of four tailing samples and 13 stockpile samples (Table 12.2). The tailing samples were collected at depth of 0.4-1.0m from surface using an auger. The stockpile samples were selected from the rejects of 2015 RC cutting samples. Each sample was placed in a plastic bag with a unique sample tag. All samples were delivered by Mr. Wu to the preparation laboratory of Bureau Veritas in Managua.

The samples prepared in the Managua laboratory were shipped to the analytical laboratory of Bureau Veritas in Vancouver for analysis. In its Vancouver laboratory, each sample was tested for 33 elements using 1:1:1 aqua regia digestion ICP-ES analysis, and gold using fire assay with AA finish.

Table 12.2

2015 DUE DILIGENCE SAMPLES AND RESULTS (NOVEMBER 2015)

Hole ID	Sample ID	Location	From	To	Length	Au g/t	Ag g/t	Cu%
	W1398853	Tailings	0.4	0.6	0.2	1.154	5.5	0.043
	W1398854	Tailings	0.4	0.6	0.2	1.011	4.6	0.060
	W1398855	Tailings	0.8	1.0	0.2	0.297	31.1	0.009
	W1398856	Tailings	0.8	1.0	0.2	0.380	46.0	0.015
2015-703	W1398857	North Stockpile	13.50	15.00	1.50	1.221	6.9	0.60
2015-705	W1398858	North Stockpile	9.00	10.50	1.50	1.829	11.4	0.53
2015-706	W1398859	North Stockpile	15.00	16.50	1.50	1.242	16.2	1.03
2015-707	W1398860	North Stockpile	7.50	9.00	1.50	0.051	6.7	1.81
2015-708	W1398861	North Stockpile	15.00	16.50	1.50	0.019	15.1	1.77
2015-711	W1398862	North Stockpile	13.50	15.00	1.50	0.889	20.4	1.99
2015-714	W1398863	SW Stockpile	12.00	13.50	1.50	0.408	1.4	0.18
2015-716	W1398864	SW Stockpile	6.00	7.50	1.50	0.326	1.1	0.55
2015-718	W1398865	SW Stockpile	12	13.5	1.50	0.224	16.1	0.37
2015-719	W1398866	SW Stockpile	13.50	15.00	1.50	0.138	11.2	0.86
2015-720	W1398867	SW Stockpile	9.00	10.50	1.50	0.722	5.6	0.80
2015-750	W1398868	East Stockpile	10.5	12	1.50	0.188	8.2	1.22
2015-779	W1398869	South Stockpile	6	7.5	1.5	0.146	1.5	0.06

Source: Wu (2016)

The results of the due diligence samples were compared with assays of RST samples and presented in Figures 12.1 through 12.3. The results of due diligence samples matched well to that of RST samples for stockpiles.

FIGURE 12.1

RESULTS OF DUE DILIGENCE SAMPLES FOR CU ON ROSITA STOCKPILES

Source: Wu (2016)

FIGURE 12.2

RESULTS OF DUE DILIGENCE SAMPLES FOR AU ON ROSITA STOCKPILES

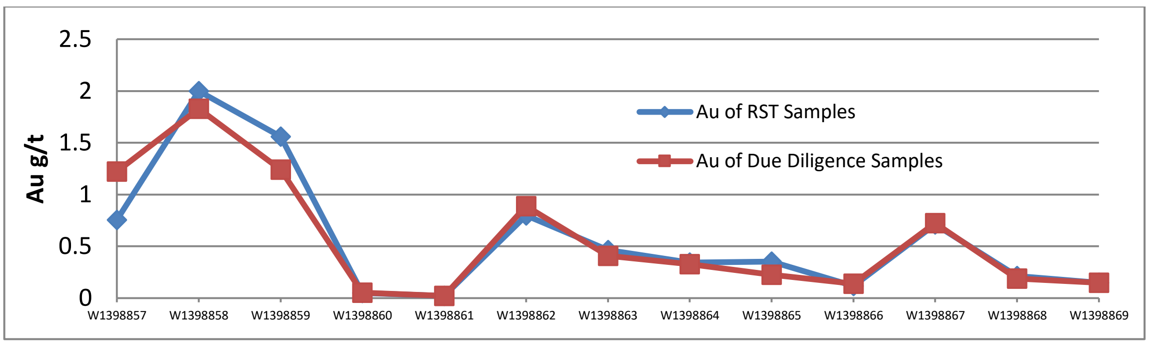
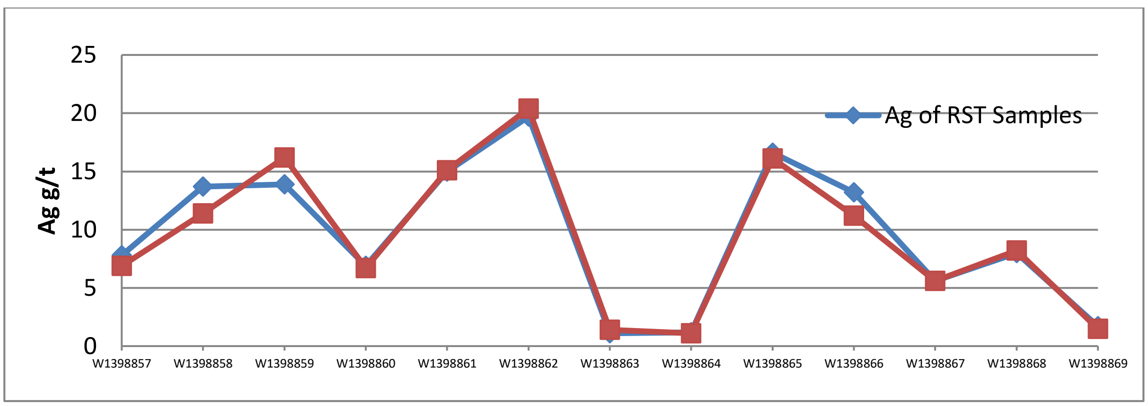


FIGURE 12.3

RESULTS OF DUE DILIGENCE SAMPLES FOR AG ON ROSITA STOCKPILES



Source: Wu (2016)

Au and Ag results of due diligence samples of the tailings were similar as that of RST samples; however, the Cu results of due diligence samples were significantly lower than that of RST samples from the tailings. The due diligence samples were taken at depth of 0.4-1.0m from surface of tailings, while RST sampled down to at least 2m deep. It is possible that Cu was leached near surface of the tailings over years. It is recommended that RST evaluates possible reasons for the low bias and that further verification work should be carried out.

12.3 Quality Assurance and Quality Control

RST implemented and monitored a quality assurance/quality control program (“QA/QC”) for the sampling programs at the Rosita Project over the periods of 2011-2015. QC protocol included the insertion one certified standard, one blank and one field duplicate into every batch of approximately 30 samples.

12.3.1 QA/QC of 2011-2012 Samples for Stockpiles

The QP of this Report had reviewed the QA/QC program of 2011-2012 stockpile sampling during the initial resource estimate for the Rosita stockpiles which was filed on Sedar titled as "NI 43-101 Technical Report on Mineral Resource Estimate of Rosita Stockpiles, Rosita Cu-Au-Ag Project, RAAN, Nicaragua" with an effective of May 8, 2012. The QA/QC procedures adopted for the project were reasonable and the protocols meets industry standards and the resulting analyses are appropriate for the resource estimate studies.

12.3.2 QA/QC of 2015 RC Samples for Stockpiles

Table 12.3 presents the QC samples implemented for 2015 drilling program on stockpiles.

TABLE 12.3

QC SAMPLES FOR 2015 STOCKPILE SAMPLING

Sample Type	No. of Samples	Percentage (%)
RC Cuttings	1,180	100
Standards	42	3.6
Duplicates	41	3.5
Blanks	44	3.7

Source: Wu (2016)

<u>Standards</u>

Certified standards were inserted by RST sequentially every 28 samples from stockpiles. Four standards used for 2015 sampling were supplied by WCM Mineral, British Columbia, Canada. The standard contents are listed in Table 12.4.

TABLE 12.4

STANDARDS USED FOR 2015 SAMPLING OF STOCKPILES

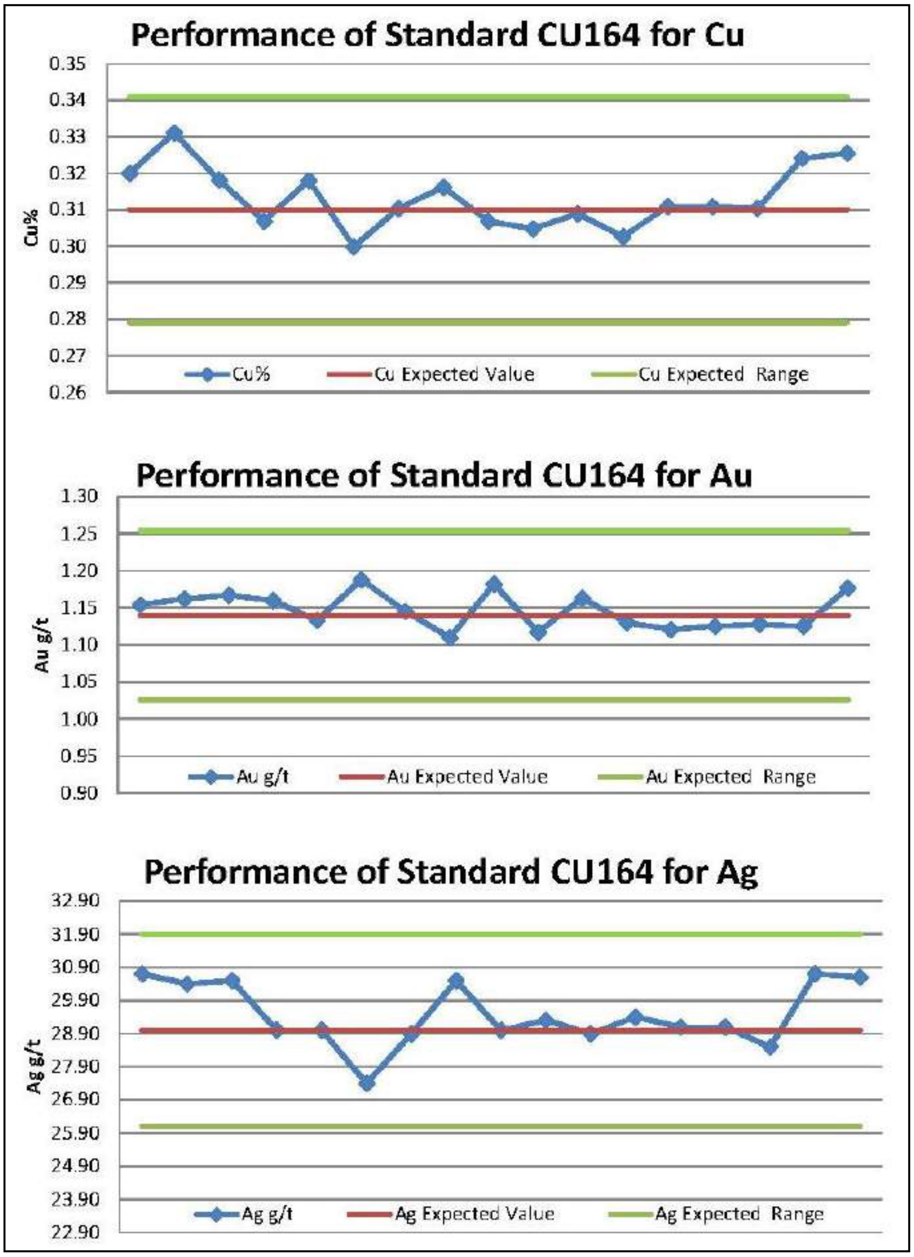
Standard	Au (g/t)	Cu (%)	Ag(g/t)	No. of Inserted
Cu164	1.14	0.31	29	17
CU165	1.42	0.31	31	5
Cu186	1.63	0.60	14	19
CU187	0.51	0.38	12	1

Source: Wu (2016)

As shown in Figures 12.4, 12.5 and 12.6, standards CU164, CU165 and CU186 for samples of the stockpiles exhibit an acceptable performance.

FIGURE 12.4

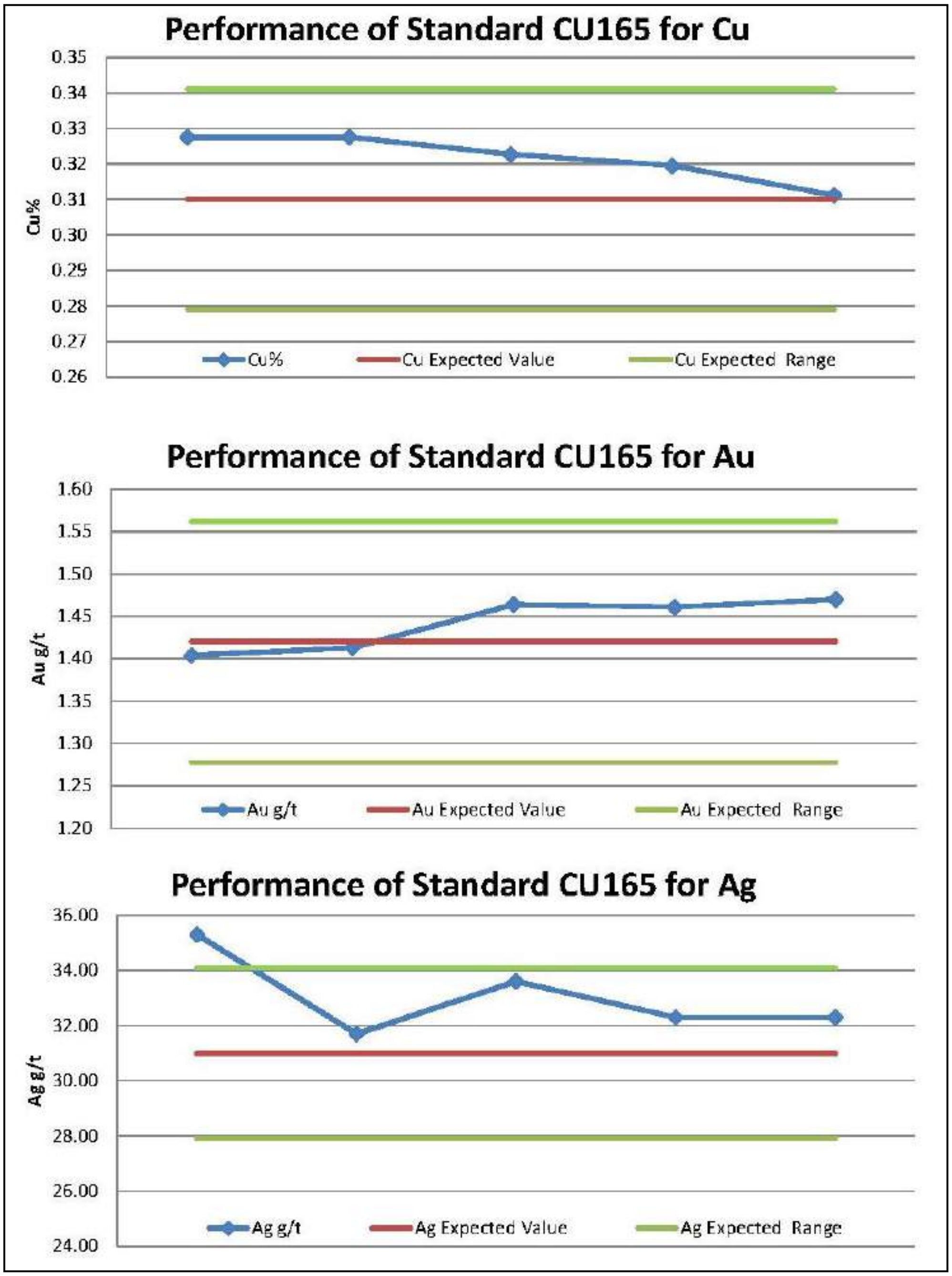
PERFORMANCE OF CERTIFIED STANDARD CU164



Source: Wu (2016)

FIGURE 12.5

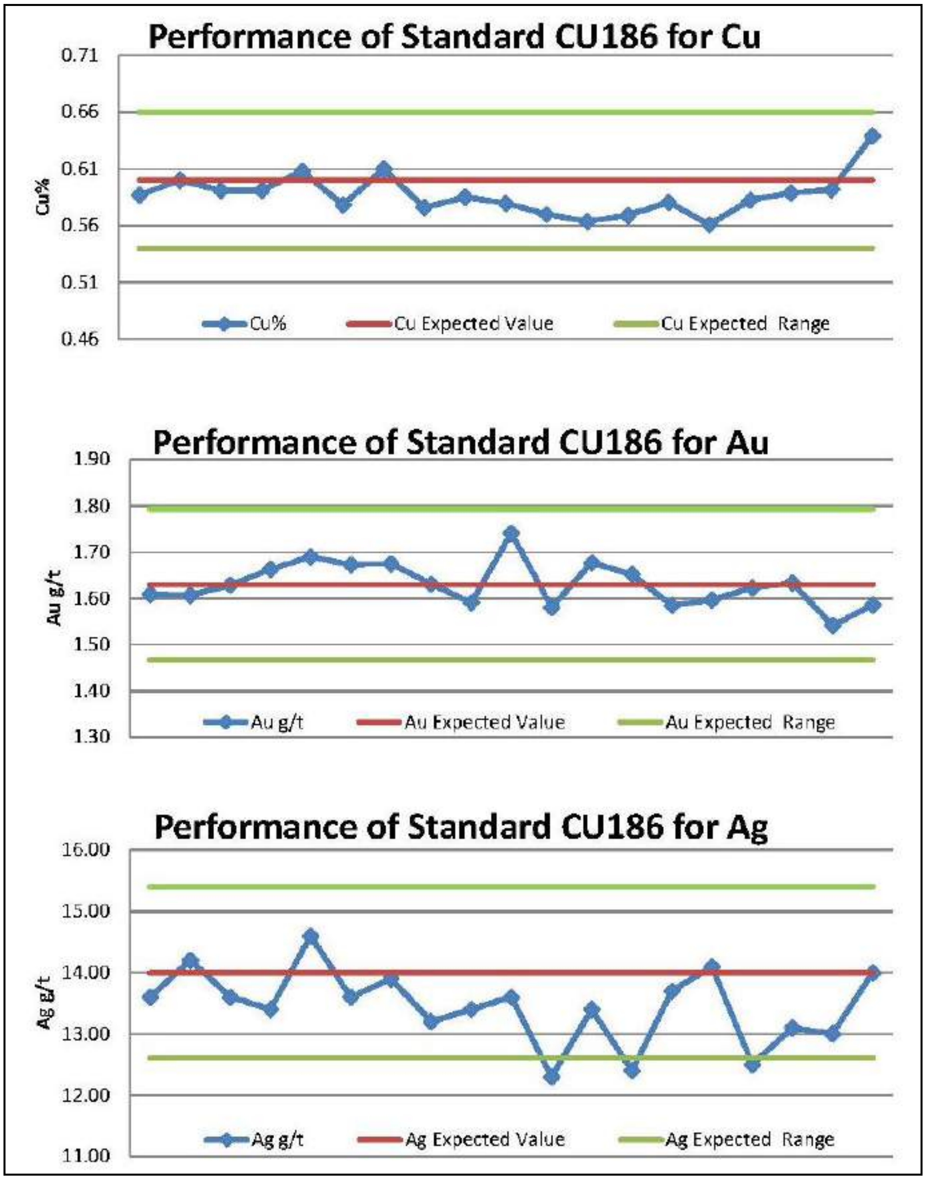
PERFORMANCE OF CERTIFIED STANDARD CU165



Source: Wu (2016)

FIGURE 12.6

PERFORMANCE OF CERTIFIED STANDARD CU186



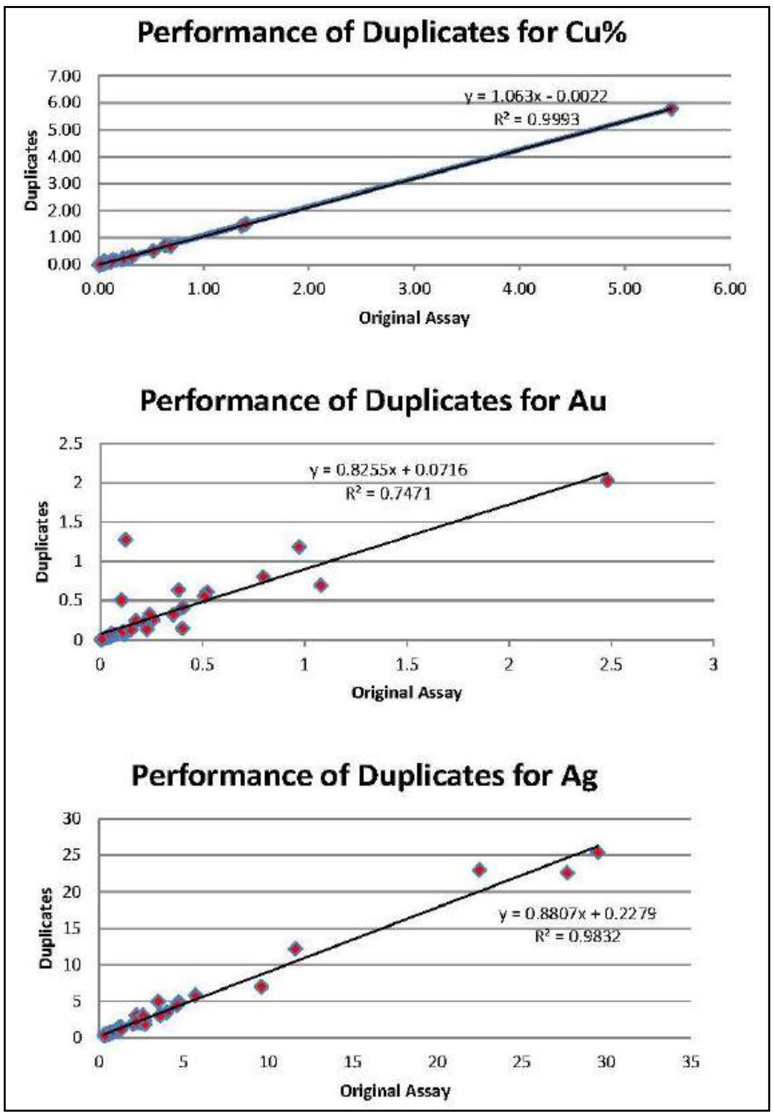
Source: Wu (2016)

Duplicates

Field duplicate samples were prepared by RST personnel and used to monitor the potential mixing up of samples and data precision. The original and duplicate samples were tagged with consecutive sample numbers and sent to the laboratory as separate samples. Duplicate samples were collected at a rate of 1 in 29 samples. A total of 41 duplicate samples were taken, representing 3.5% of the total samples. The results of the duplicate sampling are shown graphically in Figures 12.7.

FIGURE 12.7

PERFORMANCE OF DUPLICATES FOR 2015 SAMPLING



Source: Wu (2016)

Blanks

Blank samples were inserted to monitor possible contamination during both preparation and analysis of the samples in the laboratory. Blanks used by RST for 2015 sampling were volcanic tuff. A total of 44 blanks were inserted into the sample stream at rate of one blank every 27 samples and results are tabulated in Table 12.5.

TABLE 12.5

PERFORMANCE OF BLANK FOR 2015 STOCKPILE SAMPLING

Element	# of Samples	Minimum	Maximum
Cu%	44	0.0087	0.0122
Au g/t	44	<0.005	0.009
Ag g/t	44	<0.3	1.7

Source: Wu (2016)

12.3.3 QA/QC of 2012-2013 Sampling for Tailings

QC samples used for 2012-2013 sampling program of tailings are presented in Table 12.6. All 2013 samples were employed for this resource estimate, while only 12 samples of 2012 were used.

TABLE 12.6

QC SAMPLES FOR SAMPLING OF TAILINGS

Year of sampling	Sample Type	No. of Samples	Percentage (%)
2012	Tailings	194	100
	Standards	3	1.5
	Duplicates	0	0
	Blanks	3	1.5
2013	Tailings	221	100
	Standards	11	5.0
	Duplicates	12	5.4
	Blanks	12	5.4

Source: Wu (2016)

Standards

Certified standards were implemented by RST in 2013 sequentially every 20 tailing samples. Four standards used for the sampling were supplied by WCM Mineral, British Columbia, Canada. The standard contents are tabled in Table 12.7.

TABLE 12.7

STANDARDS USED FOR 2012-2013 SAMPLING OF TAILINGS

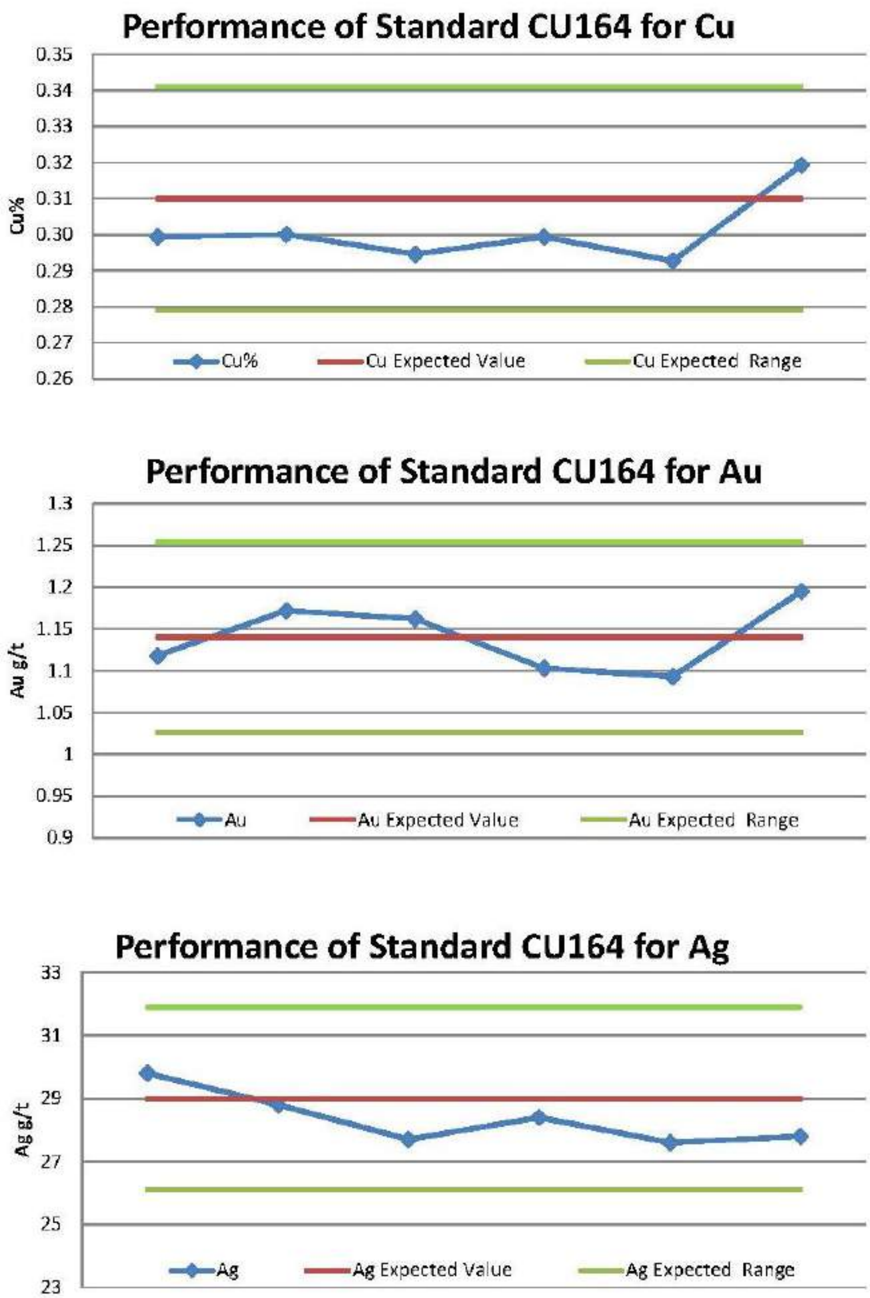
Standard	Au (g/t)	Cu (%)	Ag(g/t)	# of Inserted	Year of Sampling
CU157	0.84	0.48	15	1	2012
CU159	2.14	0.51	49	2	2012
Cu164	1.14	0.31	29	6	2013
Cu186	1.63	0.60	14	5	2013

Source: Wu (2016)

As shown in Figures 12.8 and 12.9, performance of standards CU164 and CU186 for 2013 samples of tailings was acceptable with 100% of expected values within tolerance range.

FIGURE 12.8

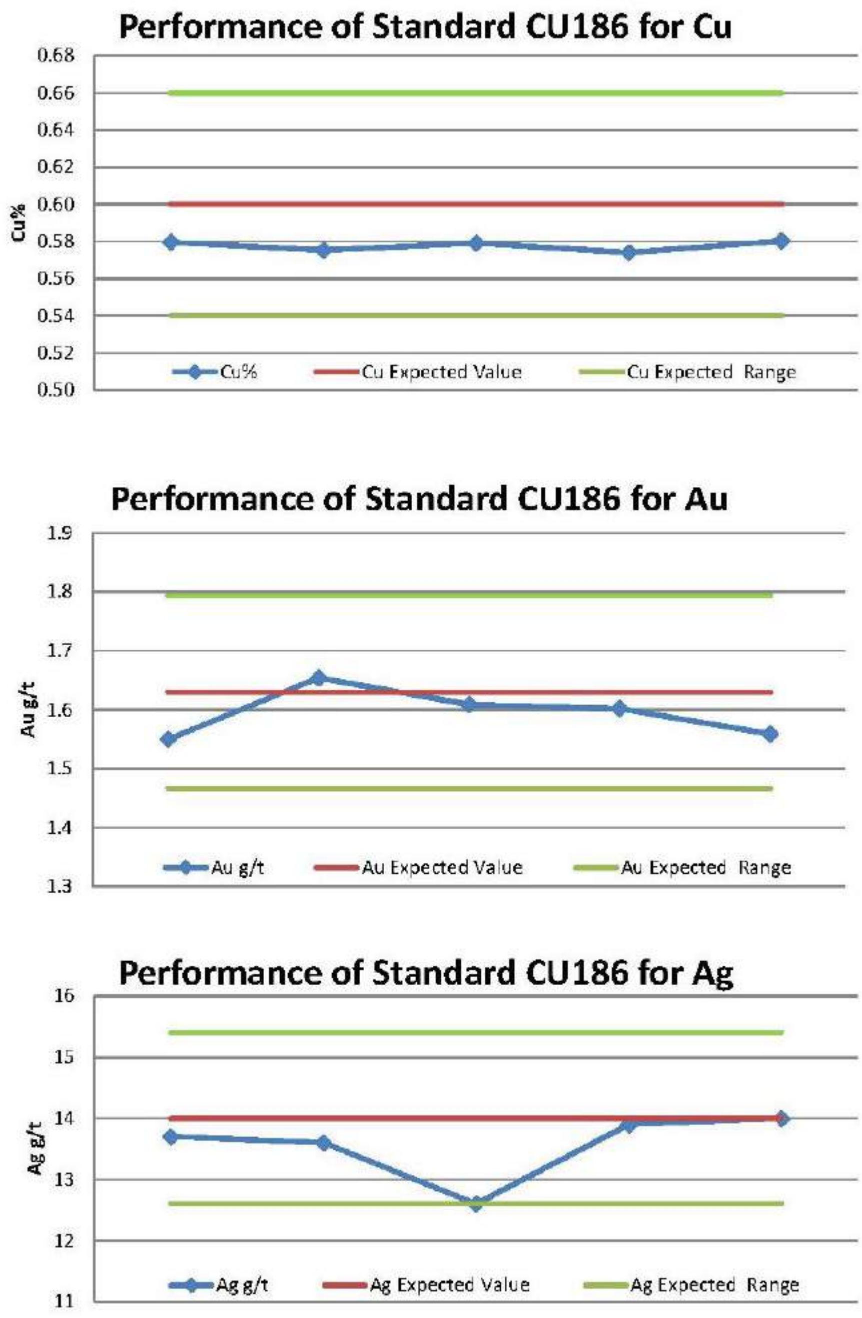
PERFORMANCE OF CERTIFIED STANDARD CU164



Source: Wu (2016)

FIGURE 12.9

PERFORMANCE OF CERTIFIED STANDARD CU186



Source: Wu (2016)

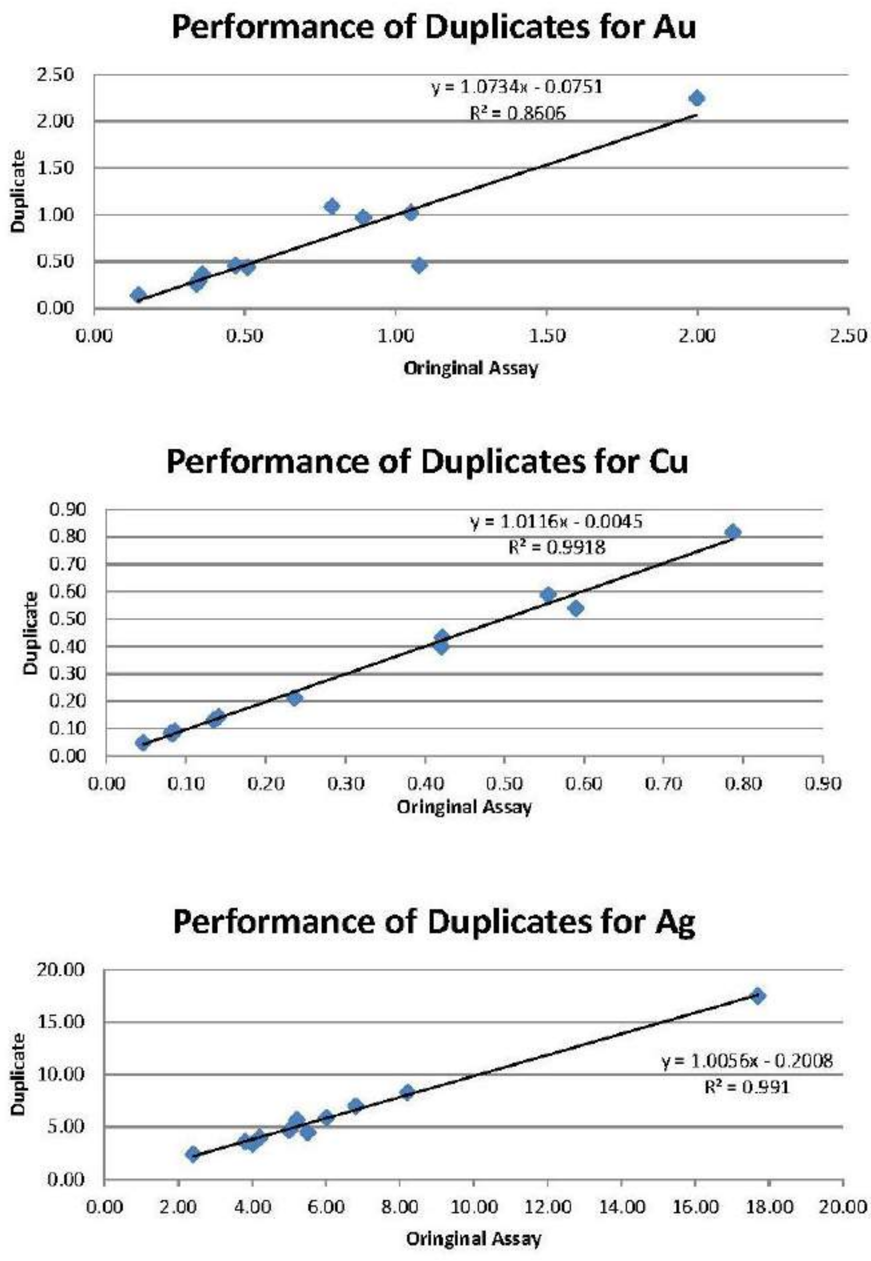
Duplicates of 2013 Tailing Samples

A total of 12 duplicate samples were taken, representing 5.4% of the total samples analysed in 2013. The field duplicate samples were selected by RST personnel. The duplicate samples were labelled with consecutive sample numbers as the normal tailing samples and sent to the laboratory as separate samples.

The results of the duplicate sampling are shown graphically in Figures 12.10.

FIGURE 12.10

PERFORMANCE OF DUPLICATES FOR TAILING SAMPLES



Source: Wu (2016)

Blanks

Blanks used by RST for 2012-2013 sampling of the tailings were volcanic tuff. 3 and 12 blanks were inserted into the sample streams for 2012 and 2013 program respectively. The testing results of Au and Ag were all below or near the detection limits (0.005g/t for Au and 0.1g/t for Ag), while Cu all were around 0.01% or lower.

12.4 Comments on QA/QC

The QC sample inserted for the stockpile sampling program was less than 5%. 2012 tailing sampling program didn't select duplicates and standards and blanks only accounted for 1.5% of total samples; however only 12 samples from 2012 tailing samples were used for the resource estimate. It is recommended that a minimum of 5% of QC samples should be inserted for future sampling programs.

The QA/QC procedures adopted for the project are reasonable and it is the opinion of the QP that the resulting analyses are appropriate for using in the resource estimate studies.

13 MINERAL PROCESSING AND METALLURGICAL TESTING – 2016-2017

SGS Canada Inc., Lakefield, Ontario, Canada, accredited to the requirements of ISO/IEC 17025 for geochemical, mineralogical, and trade mineral tests, carried out additional metallurgical testing for samples from Rosita stockpiles and tailings in 2016-2017. The results of this testing are detailed in a Report titled "An Investigation into the Recovery of Gold and Copper from Rosita Stockpile and Tailings Samples" and was prepared for Rosita Mining Corporation on March 31, 2017. The Report is summarized as following:

13.1 Metallurgical Testing of Tailings Composite and Stockpile Samples – Cyanidation

The primary purpose of this advanced testwork was to further investigate the recovery of gold and copper in the received samples. This included a series of conventional cyanidation tests followed by SART testing recovery on the leach liquor to precipitate copper sulphide (Cu_2S) from the copper cyanide complex and recover and regenerate the associated cyanide.

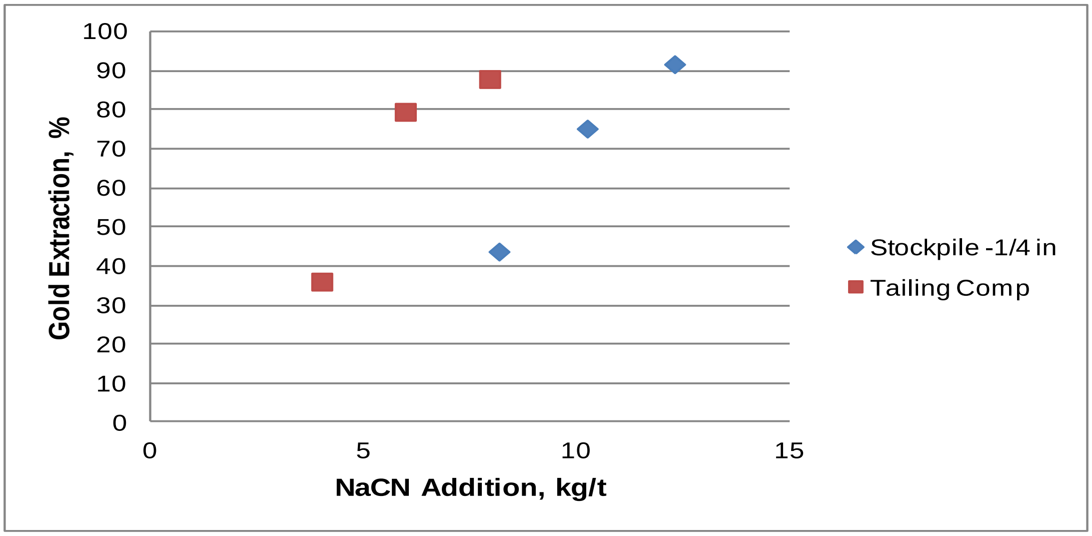
The tailings composite was 1.28 g/t Au, 0.28% Cu, 0.16% CuCNsol and 7.6 % S. Pyrite primarily present as chalcopyrite. The feed size of the tailings composite had a P80 of 259 µm. As indicated in the previous work, clay minerals were present with kaolinite as the major clay mineral.

The stockpile sample received was screened at ¼-in and split to mimic the proposed processing scenario. The oversize (+¼-in.) following to heap leach column testing with the undersize (-¼-in.) for subsequent milling and cyanidation. The undersize was 0.47 g/t Au, 1.01 % Cu, 0.36 CuCNsol. A second sample from the North stockpile SP2 was 2.34 g/t Au and 1.39 % Cu. In the case of the tailings where the major copper mineral was chalcopyrite, the stockpile -¼-in. material had copper present as copper oxides, hydroxides, carbonates, and silicates. There were more clay minerals present in the stockpile which can be attributed to the non-processing vs the previous processed tailings material.

As a part of the processing scenario, Bond Index work was carried out on the stockpile -¼-in. material and was determined to be 13.9 kWh/t (deslimed) and 8.7 kWh/t (with slimes) indicating medium to very soft hardness.

Cyanidation tests were carried out on tailing composite and the stockpile material that had been screened to -¼-in. In all tests the fineness of grind was a P80 of 75 µm. The resultant cyanidation tests show the effect of cyanide addition on gold extraction for a 48-hour leach time shown Figure 13.1 below

Figure 13.1: Cyanidation Addition vs Gold Extraction



Source: SGS (2017)

The cyanide addition is shown to have a dramatic effect on extraction largely due to the high content of the cyanide soluble copper in the tested samples.

As shown above, the extraction of gold was 91.8 % with an addition of 12.3 kg/t NaCN for the ground -1/4-in. stockpile material. The corresponding copper extraction was 35.4 %. In leach tests for the tailing composite, residue assays in the range of 0.11 g/t Au were achieved with 8 kg/t NaCN. This resulted in gold extraction in the 88.0 % to 94.9 % (varying calculated head assays) and copper extraction ranging from 55.9 % to 62.6 %.

Blending of the two (2) materials was also tested at a 50:50 ratio and resulted in an average gold extraction of 88.2 % (residue assay average 0.09 g/t Au) and copper extraction average of 42.2 %.

As in previous testing and confirmed by this advanced work, the concentrations of copper cyanide and free cyanide in the leach liquor accounted for almost all of the high cyanide addition rates. This is typically seen in most cases where soluble copper is present – maximizing copper dissolution is a requirement to maximize gold recovery albeit with high cyanide addition rates. It was previously reported in past testing, 62-71% (stockpile) and 55 % (Tailing) of the cyanide consumption was due to the cyanide-soluble copper present in the samples. In this recent advanced testing, in excess of 90 % was attributed to cyanide-soluble copper.

13.2 SART (Sulphidization – Acidification- Recycling – Thickening) Testwork

The leach liquors from the cyanide testing work in Section 12.1 were tested via the SART process to recover the copper cyanide and free cyanide in these solutions. SART testing was carried out at pH of 4 with the addition of 100% stoichiometric NaHS requirement. Copper precipitation ranged from 96.7 % - 99.1 % in the tests carried out. The results of the SART testing also showed that the SART process effectively recovered the copper cyanide and free cyanide from the leach solutions.

Subsequently, recycle testing of the SART solution was carried out to determine the gold extraction rates on the stockpile -¼-in. material and if any detrimental effect of the recycle stream was evident. Fresh make-up cyanide (approx. 2.5 kg/t) was added to the recycle stream for the leach testing. It was determined that 82 % of the cyanide addition was provided by SART recycle stream. The gold extraction was 92.3% - 95.1 % in these tests with a leach residue assay consistently 0.04-0.05 g/t Au demonstrating that SART recycle solution can be used to leach the gold.

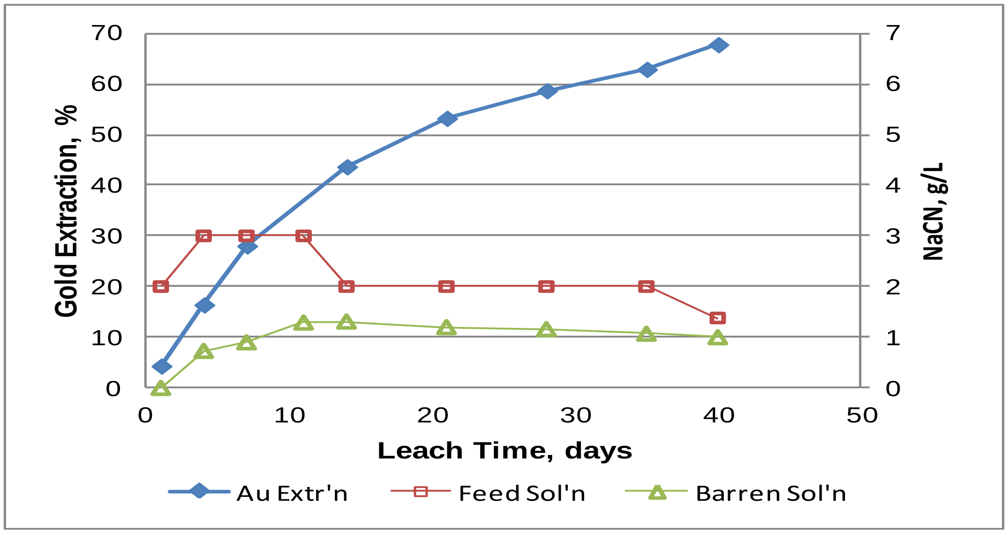
13.3 Heap Leach Column Testing

A preliminary column testing was to be carried out on sample of a 50:50 blend of Stockpile +¼-in 1/4-in. material and 75 µm leach residue from the conventional cyanidation process. The material for the Stockpile +¼-in. was a mixture of Stockpile plus North SP2 material. The leach residue was from the milled Stockpile -¼-in. material plus Tailing.

The mixture was altered as the leach residue could only be thickened to 51 % in the testing so the amount was reduced. This was to mimic the proposed flowsheet of mixing and agglomerating these two streams for conveyance to the heap leach at a desired moisture. Lime and Portland cement were added to assist in the agglomeration and stability of the column material and loaded into the column. The resultant blend was 80:20 of Stockpile and leach residue.

The result of this leach column test is shown in Figure 13.2. The extraction of gold was 67.8 % after the 40-day cycle time. Column residue assayed 0.46 g/t Au with a calculated head assay of 1.42 g/t Au. Cyanide addition was 5.7 kg/t over the leach cycle time. Copper extraction was 16.8 %.

Figure 13.2: Leach Time vs. Gold Extraction and Cyanide Addition



Source: SGS (2017)

13.4 Acid Leaching

As further project development to the Santa Rita resource, an acid leach test was carried out on the cyanide leach tailing from the Stockpile-Tailing comp to investigate the recovery of additional copper under it acidic conditions. This was to investigate the conversion of initial gold heap leaching to acidic copper heap leaching during the life of mine.

The residue was leached with sulphuric acid at a pH of 1.5 for a period of 24 hours. Ferric sulphate and hydrogen peroxide were also added to maintain test conditions. The extraction of copper was 34.1 % representing the recovery of an additional 20.3 % over the copper in the initial feed. Acid consumption for this test was 57.2 kg/t H_2SO_4.

13.5 Recommendations

Further work recommended following this recent advanced testing is:

- Stage 4 Variability Test Program for further property evaluation and economics
- Detailed Heap testing to review lower than anticipated copper recovery (vs heap amenability)
- Further column testing of “only” the coarse fraction stockpile leaching vs the blending of wet leached fines.

14 MINERAL RESOURCE ESTIMATES

14.1 INTRODUCTION

This Report section is to illustrate the Mineral Resource Estimate on the Rosita stockpiles and tailings of Rosita Mining Corp. The Mineral Resource Estimate presented herein is Reported in accordance with the Canadian Securities Administrators' National Instrument 43-101 and has been estimated in conformity with generally accepted CIM "Estimation of Mineral Resource and Mineral Reserves Best Practices" guidelines. Mineral resources are not mineral reserves and do not have demonstrated economic viability. There is no guarantee that all or any part of the mineral resource will be converted into mineral reserve. Confidence in the estimate of Inferred mineral resources is insufficient to allow the meaningful application of technical and economic parameters or to enable an evaluation of economic viability worthy of public disclosure. Mineral resources may be affected by further infill and exploration drilling that may result in increases or decreases in subsequent mineral resource estimates.

This resource estimate was undertaken by Yungang Wu, P.Geo., an independent Qualified Persons in terms of NI43-101, from information and data supplied by Rosita Mining. The effective date of this resource estimate is Feb. 8, 2016.

14.2 Resource database

All drilling and assay data were provided in the form of Excel data files by Rosita Mining. The stockpile database comprises 106 RC drill holes totalling 2,351m and 17 channels from six historical mine stockpiles, of which 55 holes aggregating 1,040m were completed in 2015. A total of 1,271 assays of Cu, Au and Ag were employed for the stockpile resource estimates.

The tailing resource estimate was based on 87 auger holes totalling 460m, of which 81 holes were drilled in 2013 and 6 holes in 2012. The database consists of 232 assays of Au, Ag, Cu and other contents.

The database of Geovia Gems 6.7.1 was constructed and validated by checking for inconsistencies in naming conventions or analytical units, duplicate entries, interval, length or distance values less than or equal to zero, out-of-sequence intervals, intervals or distances greater than the reported drill hole length, inappropriate collar locations and survey, and missing interval and coordinate fields.

Elevation of the channels and drill holes of the stockpiles were adjusted against surveyed topography surface, while elevation of tailing samples adjusted to smooth the surface, since the locations were surveyed using hand hole GPS.

14.3 Data Verification

Assay database was verified against original laboratory electronically issued certificates from Bureau Veritas Mineral Laboratories Canada. 100% of the constrained assays were checked; and no errors were discovered in the assay database. The Author of this Report believes that the supplied database is suitable for mineral resource estimation, however, it is suggested that Rosita Mining should perform topography survey for tailings.

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14.4 Geological Model

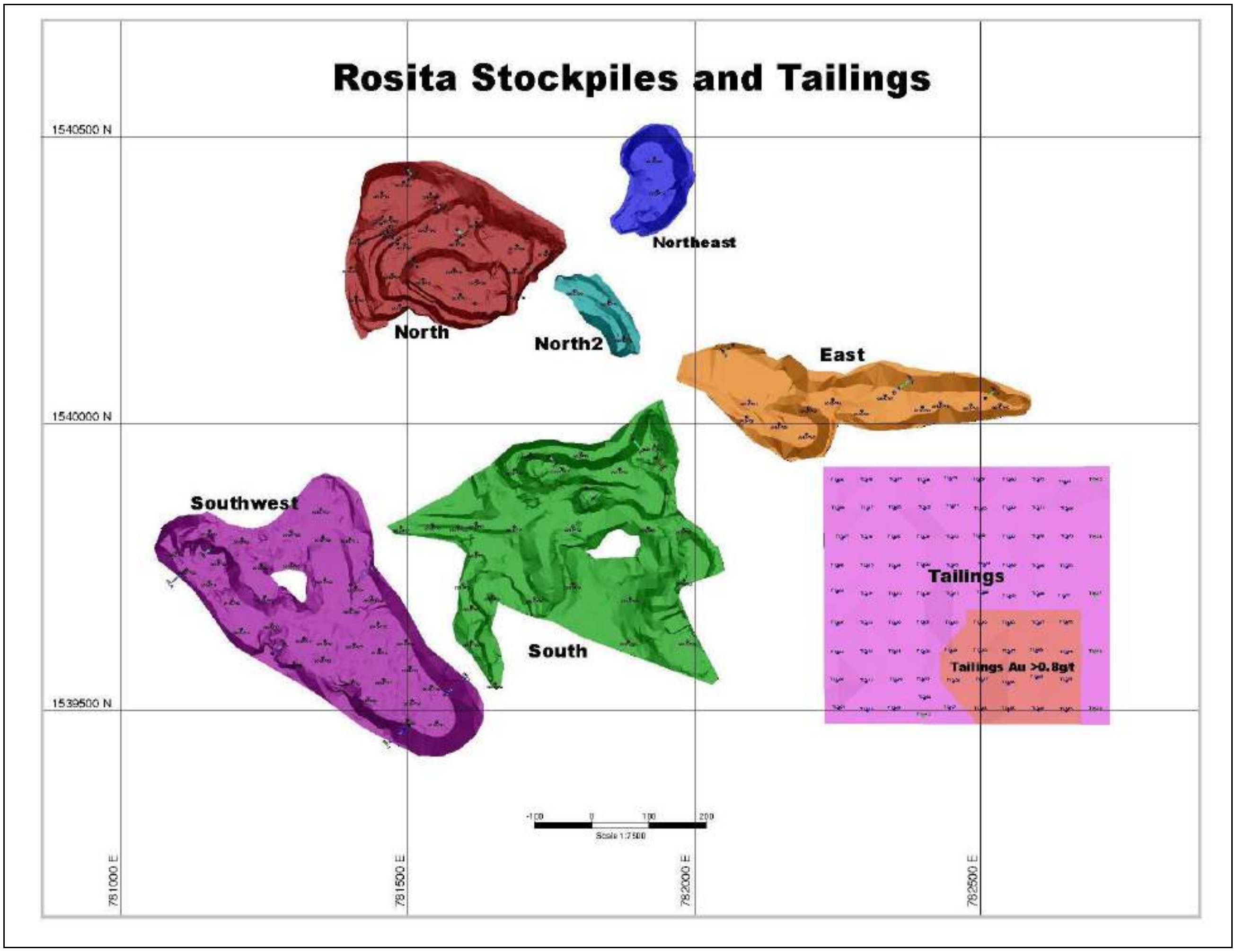
Topographic surfaces of all stockpiles were created using survey data collected in 2012. The stockpile bases were defined by drill holes completed in 2011, 2012 and 2015, most of which intersected bedrock according to the geological logging. The stockpile wireframe was generated using the topography and base surfaces for each stockpile. Artisanal mined area since 2012 at North stockpile are considered minor and not depleted.

The wireframes of tailings were modeled dominantly using 2013 auger holes which carried out at 50m spacing. Six 2012 holes were also used at east edge with 100m spacing. In southeast area of the tailings, a higher Au grade zone was recognized and wireframed separately using cut-off of Au 0.8g/t. The topographic surface of tailings were created using collars of the auger holes, while the base of tailings using toes of the auger holes. The auger holes were surveyed by the employees of Rosita Mining using a hand hold GPS, therefore the elevations of the holes provided by Rosita Mining appeared approximately 34m differences among the holes which was much greater than that of actual topography of the tailings. The elevations of auger holes were adjusted in order to smooth the surface. It is recommended that the topography of tailings should be surveyed by licensed surveyor(s) in near future. The modeled area of the tailings is approximately 450m (N-S) by 500m (E-W), the tailing is open to all directions according to the sampling programs.

The wireframes of Rosita stockpiles and tailings are presented in Figure 14.1.

FIGURE 14.1

WIREFRAMES OF STOCKPILES AND TAILINGS FOR ROSITA PROJECT



Source: Wu (2016)

14.5 Composites

The basic statistics of all constrained assays and sample lengths of stockpiles and tailings are presented in Table 14.1 and 14.2.

Over 81% of stockpile sample length was 1.50m and 87% of tailing sample length was 2.00m. In order to regularize the assay sampling intervals for grade interpolation, a 1.5m and 2.0 m compositing length was selected for stockpiles and tailings respectively. The composites were calculated for Cu, Au and Ag over the compositing lengths within the wireframe boundaries. Due to poor recovery of RC holes, the un-sampled intervals were treated as nil. Any composites that were less than 0.50 metres in length were discarded so as not to introduce any short sample bias in the interpolation process. The constrained composite data were extracted to point files for a capping study. The composite and capping statistics are summarized in table 14.3 and 14.4 for Stockpiles and tailings respectively.

TABLE 14.1

BASIC STATISTICS OF ALL ASSAYS AND LENGTHS FOR STOCKPILE SAMPLES

Variable	Length	Au	Cu	Ag
Number of samples	1271	1271	1271	1271
Minimum value	0.43	0.01	0.00	0.00
Maximum value	3.57	16.12	10.12	99.00
Mean	1.59	0.42	0.42	6.31
Median	1.52	0.23	0.24	3.80
Variance	0.11	0.60	0.33	61.41
Standard Deviation	0.34	0.78	0.58	7.84
Coefficient of variation	0.21	1.86	1.37	1.24

Source: Wu (2016)

TABLE 14.2

BASIC STATISTICS OF ALL ASSAYS AND LENGTHS FOR TAILING SAMPLES

Variable	Length	Au	Cu	Ag
Number of samples	232	232	232	232
Minimum value	0.50	0.02	0.00	0.10
Maximum value	4.25	2.00	0.95	35.70
Mean	1.98	0.53	0.21	9.28
Median	2.00	0.45	0.13	6.95
Variance	0.06	0.09	0.03	36.62
Standard Deviation	0.25	0.30	0.18	6.05
Coefficient of variation	0.12	0.56	0.86	0.65

Source: Wu (2016)

TABLE 14.3

COMPOSITING AND CAPPING SUMMARY STATISTICS OF THE STOCKPILES

Variable	Au_Comp	Cu_Comp	Ag_Comp	Au_Cap	Cu_Cap	Ag_Cap
Number of samples	1299	1299	1299	1299	1299	1299
Minimum value	0.01	0.00	0.00	0.01	0.00	0.00
Maximum value	13.35	10.12	99.00	4.03	4.00	50.00
Mean	0.41	0.43	6.51	0.40	0.43	6.43
Median	0.24	0.25	3.93	0.24	0.25	3.93
Variance	0.37	0.33	60.27	0.24	0.26	51.31
Standard Deviation	0.61	0.58	7.76	0.49	0.51	7.16
Coefficient of variation	1.50	1.34	1.19	1.24	1.19	1.11

Source: Wu (2016)

TABLE 14.4

COMPOSITING SUMMARY STATISTICS OF THE TAILINGS

Variable	Au_Comp	Cu_Comp	Ag_Comp
Number of samples	234	234	234
Minimum value	0.05	0.01	1.06
Maximum value	2.00	0.95	35.70
Mean	0.53	0.21	9.25
Median	0.44	0.13	6.90
Variance	0.09	0.03	35.58
Standard Deviation	0.30	0.18	5.97
Coefficient of variation	0.56	0.85	0.64

Source: Wu (2016)

14.6 Grade Capping

A statistical analysis was carried out on the composites for each stockpile and tailings to determine appropriate grade capping for resource estimation. The approach taken included:

- Review of the 3D grade distribution;

- Review of the composite Log-normal histograms and probability plots with significant breaks in populations used to identify possible outliers;
- Ranking of the individual composites and investigating the effect of the higher grades upon the standard deviation, coefficient of variation and the mean of the data population.

The composite histograms indicated that no outliers were present, and thus it was deemed unnecessary to cap any composites for this resource estimate of tailings.

As shown in Table 14.3, the capping resulted in a slight decrease of the naïve mean of Au and Ag for stockpiles. The Au, Cu and Ag grade capping values for the stockpile resource estimate are detailed in Table 14.5, 14.6, and 14.7 respectively. The capped composites were utilized to develop variograms and for block model grade interpolation.

Selected Log-normal histograms graphs are exhibited in Figure 14.2.

TABLE 14.5 - Source: Wu (2016)

AU GRADE CAPPING VALUES FOR THE STOCKPILES

Stockpiles	Total # of Composites	Capping Value Au (g/t)	# of Capped Composites	Mean of Composites	Mean of Capped Composites	CoV of Composites	CoV of Capped Composites	Capping Percentile
North	308	4.00	1	0.625	0.595	1.531	1.098	99.7%
South	244	No Capping	0	0.411	0.411	1.353	1.353	100.0%
East	167	2.00	1	0.338	0.321	1.213	0.801	99.4%
SW	552	No Capping	0	0.310	0.310	1.157	1.157	100.0%
NE	20	No Capping	0	0.210	0.210	0.672	0.672	100.0%

TABLE 14.6 - Source: Wu (2016)

CU GRADE CAPPING VALUES FOR THE STOCKPILES

Stockpiles	Total # of Composites	Capping Value Cu %	# of Capped Composites	Mean of Composites	Mean of Capped Composites	CoV of Composites	CoV of Capped Composites	Capping Percentile
North	308	4.00	2	0.864	0.840	0.993	0.803	99.4%
South	244	No Capping	0	0.342	0.342	0.994	0.994	100.0%
East	167	No Capping	0	0.509	0.509	1.112	1.112	100.0%
SW	552	No Capping	0	0.200	0.200	0.946	0.946	100.0%
NE	20	2.00	1	0.664	0.582	1.133	0.778	95.0%

TABLE 14.7 - Source: Wu (2016)

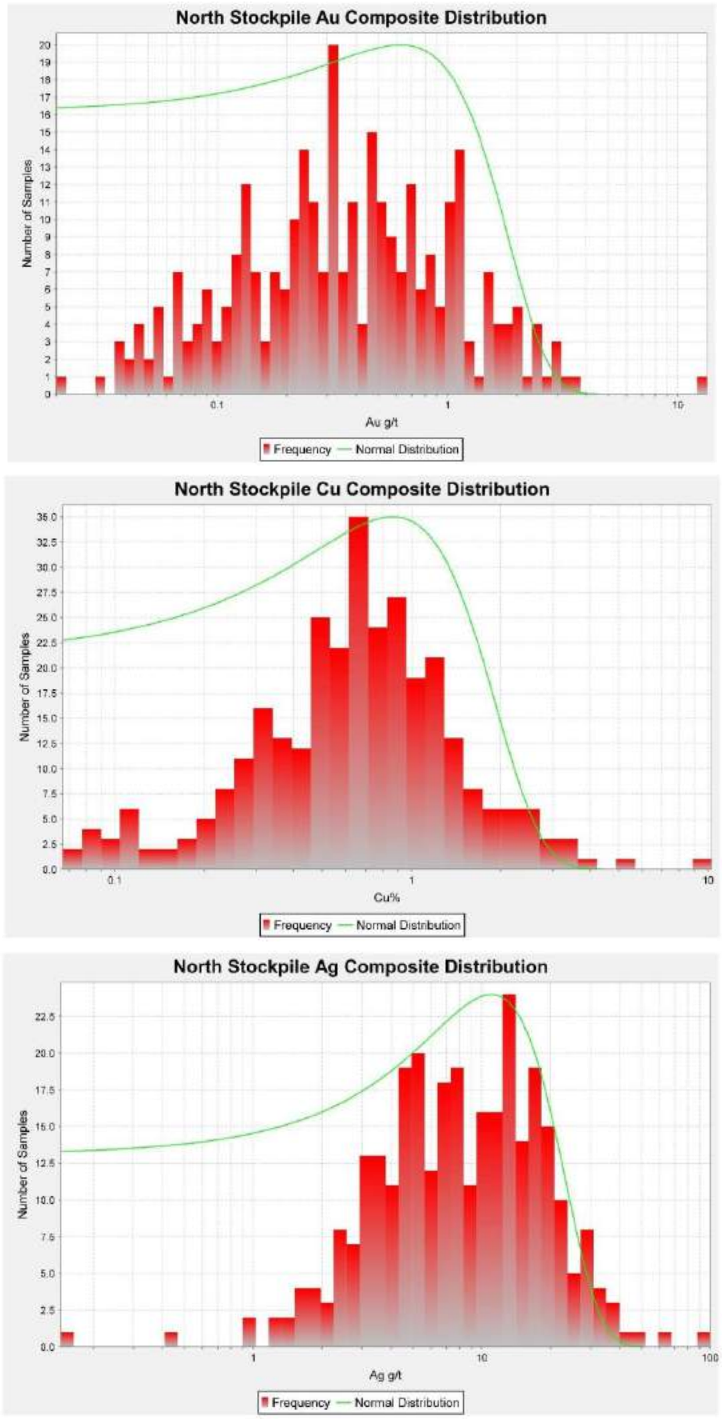
AG GRADE CAPPING VALUES FOR THE STOCKPILES

Stockpiles	Total # of Composites	Capping Value Ag (g/t)	# of Capped Composites	Mean of Composites	Mean of Capped Composites	CoV of Composites	CoV of Capped Composites	Capping Percentile
North	308	50.0	2	11.030	10.835	0.909	0.809	99.4%
South	244	No Capping	0	4.924	4.924	1.184	1.184	100.0%
East	167	40.0	1	8.751	8.591	1.040	0.963	99.4%
SW	552	No Capping	0	3.987	3.987	1.123	1.123	100.0%
NE	20	25.0	2	11.171	9.951	0.894	0.665	90.0%

FIGURE 14.2

LOG-NORMAL HISTOGRAMS OF AU, CU AND AG COMPOSITES FOR NORTH STOCKPILE

Source: Wu(2016)



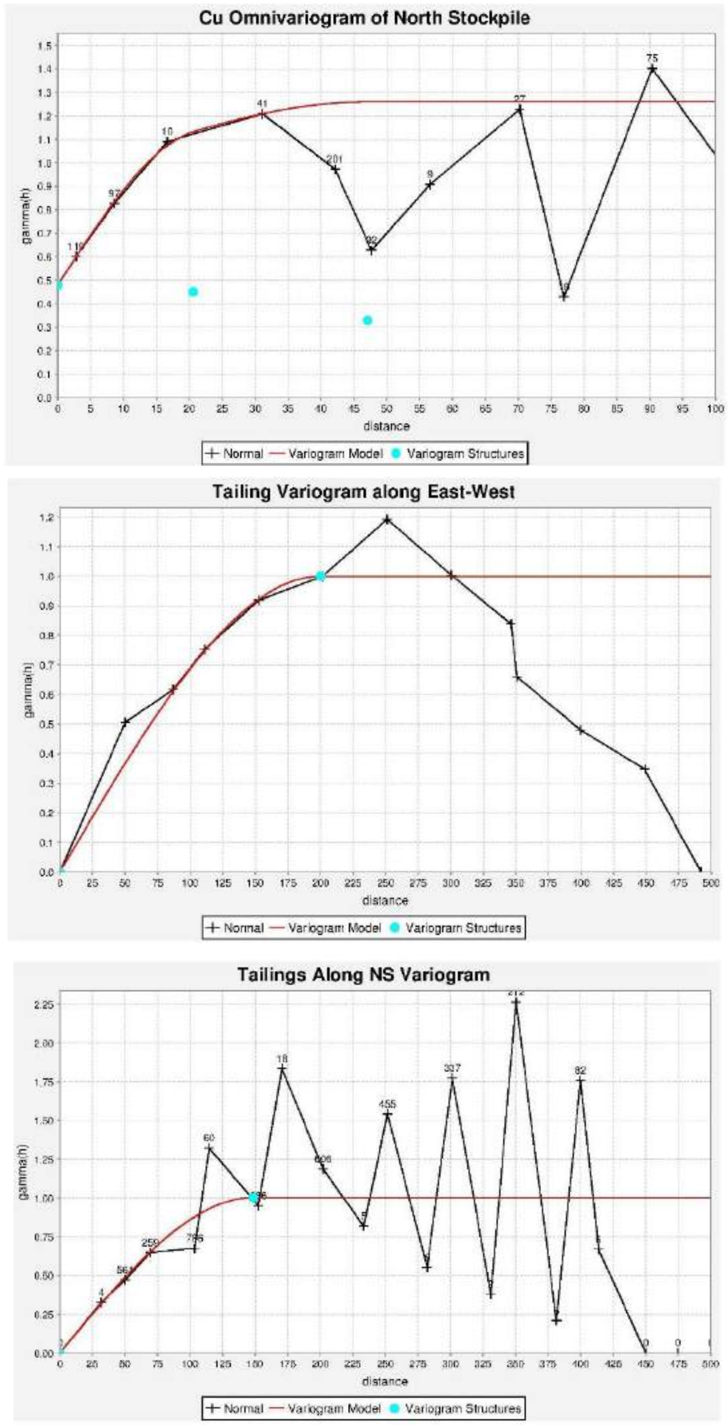
14.7 Semi-variography

A semi-variography study was performed as a guide to determining a grade interpolation search strategy. Omni, along strike, down dip and across dip semi-variograms were attempted for each stockpile using capped composites. Selected variograms are presented in Figure 14.3.

Continuity ellipses based on the observed ranges were subsequently generated and used as the basis for estimation search ranges, distance weighting calculations and mineral resource classification criteria.

FIGURE 14.3 - Source: Wu (2016)

VARIOGRAMS DEVELOPED FOR NORTH STOCKPILE AND TAILINGS



14.8 Density

Table 14.8 shows the density used for the resource estimation.

TABLE 14.8 BULK DENSITY APPLIED FOR RESOURCE ESTIMATE	
Stockpile	Dry Bulk Density
North Stockpile	1.94
South Stockpile	1.86
Northeast Stockpile	1.94
North2 Stockpile	1.94
Southwest Stockpile	2.21
East Stockpile	2.06
Tailings	1.82

Source: Wu (2016)

Dry densities of North, South and Southwest stockpile were calculated using the average wet bulk density and moisture content. Dry densities of Northeast stockpile were defined using wet mini bulk density and moisture content. Considering that the mini bulk sample results likely undervalued the densities, a factor of overall average bulk density/mini bulk density was applied to the estimation of the density for where there are no bulk density measurements. Density of the East stockpile was estimated using the average of all density values as there was no density sample measurement done on this stockpile.

14.9 Block Model Construction

Block models of stockpiles and tailings were created using Geovia Gems 6.7.1 mining software and the block model origin and block size are tabulated in table 14.9. The block model consists of separate models for estimated grade, rock type, percent, bulk density, classification and NRS attributes.

TABLE 14.9 **BLOCK MODEL DEFINITION**				
	Direction	Origin	# of Blocks	Block Size (m)
Stockpiles	X	780,920	176	10
	Y	1,539,360	122	10
	Z	106	38	2
Tailings	X	782,200	56	10
	Y	1,539,440	52	10
	Z	71	12	2
Rotation		No rotation		

Source: Wu (2016)

All wireframes of the stockpiles and tailings were utilized to code all blocks within the rock type block model that contain 1 % or greater volume within the wireframes. A percent block model was set up to accurately represent the volume and subsequent tonnage that was occupied by each block inside the constraining domains. As a result, the wireframe boundary was properly represented by the percent model ability to measure individual infinitely variable block inclusion percentages within that wireframe. The minimum percentage of the block was set to 1%.

Density model was populated with the average bulk density for each stockpile and tailings individually.

Au, Cu and Ag grades of the block models were interpolated with Inverse Distance Squared (ID2) using capped composites. Two passes were executed for the grade interpolation to progressively capture the sample points in order to avoid over smoothing and preserve local grade variability. Grade blocks were interpolated using the following parameters in Table 14.10:

TABLE 14.10 **BLOCK MODEL INTERPOLATION PARAMETERS**						
Pass	X (m)	Y (m)	Z (m)	Max # of Sample per Hole	Min # Sample	Max # Sample
I	50	50	5	2	3	12
II	150	150	10	2	1	12

Source: Wu (2016)

The NSR (Net Smelter Return) for stockpiles were manipulated using formula below:

NRS=(AuXRecovery80%X1200/31.1035) +(AgXRecovery65%X16/31.1035)+(CuXRecovery35%X 55.11558).

14.10 Resource Classification

In Author's opinion, the drilling, assaying and exploration work of the stockpiles and tailings supporting this mineral resource estimate are sufficient to indicate a reasonable potential for economic extraction and thus qualify it as a Mineral Resource under the CIM definition standards. The mineral resources of stockpiles were classified as Indicated and Inferred based on the drill hole spacing. The Indicated resources were defined for the blocks interpolated by the grade interpolation Pass I, which used at least 3 composites from a minimum of two holes; and Inferred resources were categorized for all remaining grade populated blocks. The classifications have been adjusted on plan view to reasonably reflect the distribution of each category.

The resources of the tailings were classified as Inferred since the topography of the tailings was not surveyed and the elevations of auger holes in the database appeared not accurate enough to reflect the topographic variation of the tailings.

14.11 Mineral Resource Cut-off

The Mineral Resource Estimates of Stockpiles and tailings were derived from applying an NSR and Au cut-off grade respectively to the block models and reporting the resulting tonnes and grades for potentially mineable areas. The following calculation demonstrates the rationale supporting the NSR and Au cut-off.

Au Price:	US$1,200/oz
Cu Price:	US$2.5/lb
Ag Price:	US$16/oz
Au Recovery:	80%
Cu Recovery:	35% (after 10% deducted for smelting)
Ag Recovery:	65%
Mining cost:	US$1/t
Process Cost:	US$7.5/tonne milled

General & Administration: US$1.5/tonne milled

Therefore, the NSR cut-off grade for the resource estimate of stockpiles is calculated as US$10/tonne.

NRS=(AuXRecovery80%X1200/31.1035)+(AgXRecovery65%X16/31.1035)+(CuXRecovery35%X 55.11558).

The Au cut-off grade for the resource estimate of tailings is calculated as follows:

($1 + $7.5 + $1.5) / ($1,200XRecovery80%/31.1035) = 0.32, Used 0.3 g/t.

14.12 Mineral Resource Statement

Mineral Resources for the stockpiles and tailings were classified under the CIM Definition Standards for Mineral Resources and Mineral Reserves by application of a cut-off grade of $10NSR for stockpiles and 0.3g/t Au for tailings. Mineral Resources are tabulated in Table 14.11.

TABLE 14.11

MINERAL RESOURCE ESTIMATE STATEMENT (1) (2) (3) (4)(5)(6)(7)

Stockpiles	Class	Tonne (1,000t)	Au (g/t)	Contained Au (1,000oz)	Cu %	Contained Cu (1,000t)	AG (g/t)	Contained Ag (1,000oz)
North	Indicated	2,007	0.66	42.4	0.89	17.8	10.94	706.0
	Inferred	907	0.65	19.0	0.95	8.6	12.28	358.0
East	Indicated	1,049	0.30	10.1	0.43	4.5	8.77	295.8
	Inferred	520	0.31	5.1	0.81	4.2	12.84	214.5
South	Indicated	800	0.52	13.5	0.46	3.7	5.88	151.1
	Inferred	634	0.43	8.9	0.29	1.9	3.90	79.5
Southwest	Indicated	2,603	0.37	30.7	0.24	6.2	4.39	367.6
	Inferred	796	0.41	10.5	0.27	2.2	4.21	107.7
Northeast	Inferred	431	0.26	3.5	0.71	3.1	12.39	171.7
North2	Inferred	150	0.68	3.3	0.71	1.1	5.42	26.1
Stockpile Total	Indicated	6,460	0.47	96.7	0.50	32.2	7.32	1,520.5

	Inferred	3,437	0.46	50.3	0.61	21.0	8.66	957.5
Tailings	Inferred	1,956	0.56	35.2	0.21	4.0	9.65	607.0

Source: Wu (2016)

1. *Mineral resources which are not mineral reserves do not have demonstrated economic viability. The estimate of mineral resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues.*
2. *The quantity and grade of Reported Inferred resources in this estimation are uncertain in nature and there has been insufficient exploration to define these Inferred resources as an Indicated or Measured mineral resource and it is uncertain if further exploration will result in upgrading them to an Indicated or Measured mineral resource category.*
3. *The mineral resources were estimated using the Canadian Institute of Mining, Metallurgy and Petroleum (CIM), CIM Standards on Mineral Resources and Reserves, Definitions and Guidelines prepared by the CIM Standing Committee on Reserve Definitions and adopted by the CIM Council.*
4. *A gold price of US$1,200/oz, copper price of US$2.5/lb and silver price of US$16/oz were utilized in the cut-off calculations of block values with process recoveries of 80% for gold, 35% for Cu (10% deducted for smelting) and 65% for silver. These values were equated against a cut-off grade of US$10 for stockpiles and 0.3 g/t Au for tailing mineral resources.*
5. *For the cut-off grade, mining costs were assumed at US$1.00/t, process costs at US$7.50/t and G&A costs at US$1.50/t*
6. *Artisanal mined tonnages since 2012 are considered minor and not depleted from the resources of the North stockpile.*
7. *Totals in the table may not sum due to rounding.*

The sensitivities of the mineral resources to selected cut-off are shown in Table 14.12 and 14.13 for stockpiles and tailings respectively.

Table 14.12

SENSITIVITY TO RESOURCE ESTIMATE OF STOCKPILES

Stockpile	Class	Cut-off	Tonne	Au (g/t)	Contained Au (oz)	Cu %	Contained Cu (t)	AG (g/t)	Containe d Ag (oz)
North	Indicated	NSR$75	73,768	1.57	3,719	1.60	1,181	17.76	42,111
		NSR$50	548,976	1.11	19,642	1.23	6,736	14.93	263,508
		NSR$30	1,420,095	0.79	36,257	1.04	14,805	12.94	590,585
		NSR$25	1,621,394	0.74	38,822	0.99	16,122	12.37	644,999
		NSR$20	1,790,703	0.71	40,641	0.95	17,003	11.81	679,795
		NSR$15	1,946,285	0.67	42,029	0.91	17,643	11.18	699,515
		NSR$10	2,007,459	0.66	42,387	0.89	17,841	10.94	705,956
		NSR$8	2,019,408	0.65	42,433	0.88	17,869	10.89	707,025
		NSR$0	2,028,710	0.65	42,459	0.88	17,884	10.85	707,893
	Inferred	NSR$75	49,489	2.15	3,417	0.77	382	12.22	19,440

Table 14.12

SENSITIVITY TO RESOURCE ESTIMATE OF STOCKPILES

Stockpile	Class	Cut-off	Tonne	Au (g/t)	Contained Au (oz)	Cu %	Contained Cu (t)	AG (g/t)	Containe d Ag (oz)
		NSR$50	277,366	1.22	10,865	1.16	3,213	15.33	136,724
		NSR$30	644,976	0.80	16,568	1.10	7,110	14.02	290,798
		NSR$25	742,730	0.74	17,613	1.06	7,852	13.41	320,113
		NSR$20	823,297	0.70	18,398	1.01	8,318	12.79	338,464
		NSR$15	884,953	0.66	18,898	0.97	8,573	12.36	351,524
		NSR$10	906,906	0.65	19,018	0.95	8,630	12.28	358,005
		NSR$8	907,303	0.65	19,019	0.95	8,630	12.28	358,126
		NSR$0	907,636	0.65	19,019	0.95	8,631	12.28	358,223
East	Indicated	NSR$75	3,309	1.84	196	1.01	33	12.40	1,319
		NSR$50	25,175	1.32	1,071	0.87	218	14.06	11,380
		NSR$30	73,978	0.81	1,931	0.77	569	14.07	33,474
		NSR$25	215,971	0.52	3,612	0.65	1,397	13.24	91,911
		NSR$20	441,564	0.41	5,817	0.57	2,499	12.17	172,817
		NSR$15	773,515	0.34	8,416	0.49	3,773	10.14	252,058
		NSR$10	1,049,154	0.30	10,148	0.43	4,494	8.77	295,769
		NSR$8	1,118,709	0.29	10,485	0.41	4,615	8.44	303,435
		NSR$0	1,141,760	0.29	10,571	0.41	4,643	8.33	305,801
	Inferred	NSR$75	1,978	0.38	24	2.95	58	21.38	1,359
		NSR$50	57,893	0.39	725	2.15	1,248	21.15	39,374
		NSR$30	179,873	0.39	2,244	1.43	2,576	17.25	99,779
		NSR$25	260,735	0.37	3,134	1.19	3,098	15.73	131,891
		NSR$20	372,178	0.34	4,118	0.99	3,678	14.70	175,955
		NSR$15	475,601	0.32	4,848	0.86	4,091	13.47	206,020
		NSR$10	519,740	0.31	5,119	0.81	4,214	12.84	214,476
		NSR$8	524,193	0.30	5,138	0.81	4,222	12.76	215,026
		NSR$0	527,768	0.30	5,152	0.80	4,227	12.70	215,447
South	Indicated	NSR$75	15,225	1.98	967	0.96	147	19.74	9,661
		NSR$50	40,018	1.54	1,976	0.86	344	16.62	21,389
		NSR$30	258,953	0.86	7,162	0.65	1,674	8.81	73,329
		NSR$25	380,219	0.76	9,300	0.59	2,244	7.78	95,061
		NSR$20	523,166	0.66	11,019	0.55	2,893	7.13	119,918
		NSR$15	666,324	0.58	12,465	0.51	3,371	6.46	138,463
		NSR$10	799,673	0.52	13,469	0.46	3,678	5.88	151,138
		NSR$8	845,414	0.50	13,726	0.44	3,747	5.69	154,545
		NSR$0	911,429	0.48	13,951	0.42	3,807	5.38	157,535
	Inferred	NSR$75	1,304	1.78	75	0.91	12	17.46	732
		NSR$50	15,629	1.39	697	0.74	115	14.03	7,051

Table 14.12

SENSITIVITY TO RESOURCE ESTIMATE OF STOCKPILES

Stockpile	Class	Cut-off	Tonne	Au (g/t)	Contained Au (oz)	Cu %	Contained Cu (t)	AG (g/t)	Containe d Ag (oz)
		NSR$30	98,573	0.90	2,842	0.53	523	8.15	25,817
		NSR$25	158,557	0.78	3,995	0.48	761	6.92	35,273
		NSR$20	242,463	0.67	5,244	0.43	1,047	5.92	46,149
		NSR$15	372,229	0.57	6,788	0.37	1,364	4.99	59,764
		NSR$10	633,545	0.43	8,858	0.29	1,852	3.90	79,539
		NSR$8	804,265	0.39	10,064	0.25	2,015	3.42	88,305
		NSR$0	1,906,786	0.21	12,675	0.12	2,283	1.78	109,140
Southwest	Indicated	NSR$75	13,407	2.99	1,289	0.62	83	5.83	2,511
		NSR$50	27,523	2.30	2,031	0.55	151	6.61	5,850
		NSR$30	147,868	1.14	5,412	0.35	515	5.68	26,994
		NSR$25	313,299	0.86	8,682	0.34	1,062	5.80	58,433
		NSR$20	579,269	0.68	12,729	0.32	1,858	5.67	105,539
		NSR$15	1,271,521	0.50	20,249	0.29	3,687	5.15	210,597
		NSR$10	2,603,244	0.37	30,657	0.24	6,212	4.39	367,622
		NSR$8	3,199,683	0.33	34,057	0.22	7,053	4.06	417,440
		NSR$0	3,603,312	0.31	35,754	0.21	7,442	3.74	433,719
	Inferred	NSR$75	8,200	2.81	742	0.55	45	5.00	1,319
		NSR$50	21,726	2.18	1,524	0.45	97	3.97	2,772
		NSR$30	71,177	1.28	2,918	0.37	263	4.30	9,839
		NSR$25	137,346	0.92	4,046	0.40	554	5.68	25,102
		NSR$20	226,015	0.74	5,351	0.38	863	5.34	38,782
		NSR$15	456,511	0.53	7,709	0.34	1,561	4.99	73,240
		NSR$10	796,262	0.41	10,540	0.27	2,179	4.21	107,694
		NSR$8	908,521	0.38	11,208	0.26	2,328	3.96	115,668
		NSR$0	1,035,789	0.35	11,762	0.24	2,446	3.63	120,825
Northeast	Inferred	NSR$30	132,539	0.27	1,152	1.15	1,525	17.90	76,272
		NSR$25	236,485	0.29	2,178	0.95	2,239	15.72	119,515
		NSR$20	291,100	0.28	2,661	0.87	2,528	14.84	138,905
		NSR$15	381,736	0.27	3,322	0.76	2,889	13.17	161,581
		NSR$10	431,109	0.26	3,539	0.71	3,075	12.39	171,746
		NSR$8	502,114	0.23	3,701	0.65	3,263	11.22	181,124
		NSR$0	509,477	0.23	3,721	0.64	3,276	11.14	182,444
North2	Inferred	NSR$50	24,378	1.57	1,231	0.59	145	5.57	4,363
		NSR$30	107,389	0.87	2,993	0.75	803	5.90	20,361
		NSR$25	115,206	0.84	3,108	0.74	849	5.89	21,829
		NSR$20	122,705	0.80	3,174	0.73	897	5.89	23,232
		NSR$15	138,172	0.73	3,238	0.72	991	5.63	25,012

Table 14.12

SENSITIVITY TO RESOURCE ESTIMATE OF STOCKPILES

Stockpile	Class	Cut-off	Tonne	Au (g/t)	Contained Au (oz)	Cu %	Contained Cu (t)	AG (g/t)	Containe d Ag (oz)
		NSR$10	149,593	0.68	3,257	0.71	1,061	5.42	26,067
		NSR$8	149,593	0.68	3,257	0.71	1,061	5.42	26,067
		NSR$0	149,593	0.68	3,257	0.71	1,061	5.42	26,067

Source: Wu (2016)

TABLE 14.13

SENSITIVITY TO RESOURCE ESTIMATE OF TAILINGS

Cut-off Au g/t	Tonne	Au g/t	Contained Au (oz)	Cu%	Contained Cu (t)	Ag g/t	Contained Ag (oz)
1.00	171,842	1.23	6,784	0.52	887	6.40	35,357
0.90	233,912	1.15	8,660	0.49	1,148	6.44	48,397
0.80	313,037	1.08	10,832	0.46	1,438	6.89	69,326
0.70	381,563	1.02	12,471	0.43	1,633	7.54	92,470
0.60	546,043	0.91	15,889	0.36	1,956	9.17	161,064
0.50	862,291	0.77	21,395	0.30	2,549	10.06	278,868
0.40	1,347,009	0.65	28,320	0.24	3,234	10.01	433,417
0.35	1,663,030	0.60	32,136	0.22	3,647	9.87	527,915
0.3	1,956,195	0.56	35,220	0.21	4,030	9.65	607,003
0.20	2,116,830	0.54	36,639	0.20	4,205	9.42	641,013
0.10	2,119,381	0.54	36,651	0.20	4,207	9.41	641,428
0.00	2,120,453	0.54	36,654	0.20	4,207	9.41	641,492

Source: Wu(2016)

Ratios of the mineral resources by volume of each stockpile are tabulated in Table 14.14. At cut-off NSR$10, 97% of North Stockpile is potentially minable, while 40% of South stockpile minable at its north side.

TABLE 14.14

RATIO OF THE STOCKPILE RESOURCES

Stockpiles	Resource Volume (m^3)	Stockpile Volume (m^3)	Resource Ratio
North	1,466,500	1,513,580	97%
East	646,024	810,450	80%
South	607,554	1,515,170	40%
Southwest	1,029,986	2,099,141	49%
Northeast	201,754	262,617	77%
North2	66,756	77,110	87%
Tailings	1,074,832	1,165,084	92%

Source: Wu (2016)

14.13 Confirmation of Estimate

The block model was validated using a number of industry standard methods including visual and statistical methods.

- Visual examination of composite and block grades on successive plans and sections on-screen in order to confirm that the block model correctly reflects the distribution of sample grades.

- Review of estimation parameters including:

  - Number of composites used for estimation;
  - Number of holes used for estimation;
  - Mean Distance to sample used;
  - Number of passes used to estimate grade;
  - Mean value of the composites used.

- Comparison of mean grades of block model with composites, as presenting in Table 14.15.

TABLE 14.15 COMPARISON OF AVERAGE GRADE OF BLOCK MODEL WITH COMPOSITES						
Stockpiles	Block Model Grade			Capped Composites		
	Au (g/t)	Cu %	AG (g/t)	Au (g/t)	Cu %	Ag (g/t)
North	0.65	0.90	11.29	0.59	0.84	10.83
East	0.29	0.53	9.71	0.32	0.51	8.59
South	0.29	0.22	2.94	0.41	0.34	4.92
Southwest	0.32	0.21	3.72	0.31	0.20	3.99
Northeast	0.23	0.64	11.14	0.21	0.58	9.95
North2	0.68	0.71	5.42	0.90	0.63	5.84
Tailings	0.54	0.20	9.41	0.53	0.21	9.25

Source: Wu (2016)

The comparison above shows the average grades of the block models to be somewhat different to that of capped composites used for grade estimating. This is probably due to the localized clustering of some composite data which were smoothed by the block modelling grade interpolation process. The block model values will be more representative than the capped composites due to the block model's 3D spatial distribution characteristics.

- A volumetric comparison was performed with the block model volume versus the geometric calculated volume of the wireframes and the differences are detailed in Table 14.16.

TABLE 14.16 VOLUME COMPARISON OF BLOCK MODEL WITH GEOMETRIC SOLIDS			
Stockpiles	Geometric Volume of Wireframe	Block Model Volume	Difference %
North	1,513,897	1,513,580	-0.02%
East	810,716	810,450	-0.03%
South	1,515,589	1,515,170	-0.03%
Southwest	2,099,177	2,099,141	0.00%
Northeast	262,537	262,617	0.03%
North2	77,172	77,110	-0.08%
Tailings	1,167,842	1,165,084	-0.24%

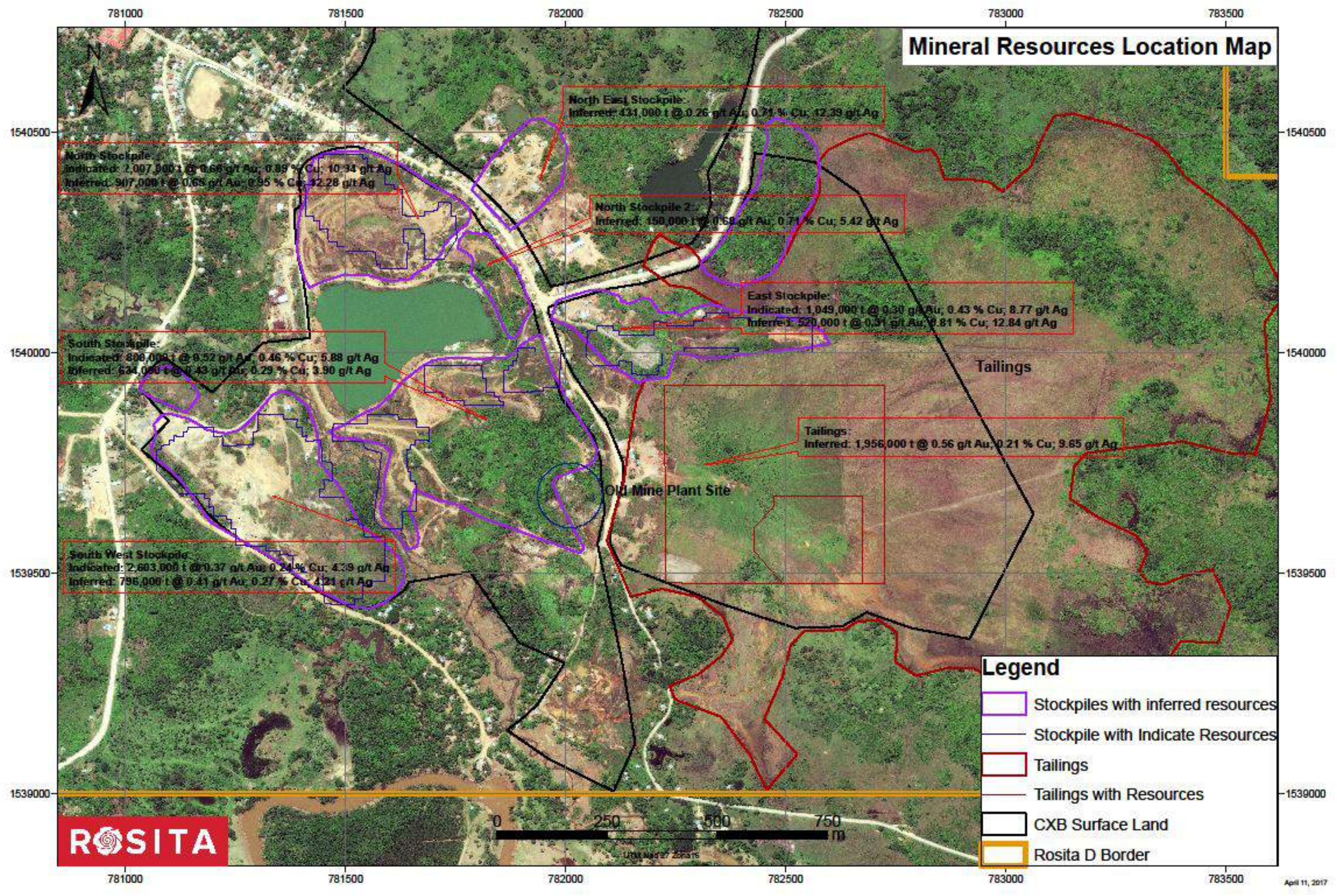
Source: Wu (2016)

14.14 Mineral Resource Location Map

Shown in the attached is the RST Mineral Resources Location Map – Figure 14.4 – Source: RST (2017)

Figure 14.4 Mineral Resource Location Map

(Source: RST 2017)



15 MINERAL RESERVE ESTIMATES

As there has been no prefeasibility or feasibility study completed on the recovery of gold-silver-copper from the Santa Rita Project, there is no mineral reserves.

16 MINING METHODS (STOCKPILE AND TAILINGS RECLAMATION).

Rosita is proposing to reclaim the Stockpile and tailings at Santa Rita with the use of local construction contractors. Conventional earth moving equipment such as hydraulic excavators or front end loaders will be utilized. The nature of the "loose" stockpile and tailings mineralized material does not require any drilling and blasting activities. The reclaimed material can be transported to the mobile crushing plant via articulated or rigid frame haul trucks.

The location of the stockpiles and tailings are in close proximity to the mobile crushing plant, and proposed process facility via existing roads in and around the area. The project does not require any selectivity or detailed grade control however, selective reclamation of specific stockpiles will take place. It is anticipated that the higher grade North Stockpile will be reclaimed initially and this has been addressed in the early year cash flow presented.

In Section 14 of this report, the resources for each stockpile and the tailings have been delineated by grade and resource classification. The bulk of the resources from each stockpile will be focussed on the indicated resources. However, where inferred resources are at the edges of stockpile they will be excavated first and thus inferred and indicated resources will be mixed in the feed to the treatment plant.

From the sequence of mining planned, the grades used for the cash flow statement have been defined as:

First 5 years: 0.65 g/t Au; 10.0 g/t Ag (for the mill) 8.0 g/t Ag (for the heap leach; 0.80% Cu

Second 5 years;0.47 g/t Au; 7.79 g/t Ag; 0.50% Cu

As the total mineralized resource will reclaimed, there is no need to address waste handling or deposition of the same. Specific lower grade stockpiles will be reclaimed as dictated on the process economics at that time.

The loading and material handling costs has been shown in the operating cost section and is based on the following:

- Production required to the crusher from the stockpile – 328,500 tonnes per year (years 1-3) and 657,000 tonnes per year (Years 4 and after)
- Tailings – 36,500 tonnes (Years 1-3) and 73,000 tonnes (Year 4 and after)

- Hours of Operation per day – 16 hours. This requirement is flexible with the major requirement being the supply of feed to crusher based the production requirements.

- Bulk Density – Stockpiles – 2.0, Tailings – 1.82

- Haul Distance to Mobile Crushing Facility – 500 m. – 1000 m.

The cost for the contractor includes equipment supply, manpower, diesel consumption, and required service lubricants.

This phase of the project can be performed by the local contractor and with minimal supervision from Rosita because of the no grade control requirement. It also allows for reclamation flexibility from the multiple stockpiles based on gold and copper grades.

Specific challenges in the reclamation of the Stockpiles and Tailings will be during the local rainy season. Excavation and hauling could be slowed due to the high clay content of the material. Increased haulage during non-rainy season could be scheduled to minimize any reduced production. This is also the case for the subsequent crushing and two (2) product stockpile for the process feed material. As mentioned previously, water diversion and control around the site will be important.

17 RECOVERY METHODS

17.1 Process Overview

The Santa Rita process facility will consist of following of the following circuits and include all associated pumping, piping, and electrical components.

- Stockpiles and Tailing Resource
- Mobile Crushing Plant to produce +1/4-in and -1/4-in material.
- Gold and Copper Heap – pads and ponds – agglomeration - conveyance
- Milling Circuit including ball mill, leach tanks, and thickeners.
- SART Circuit (Sulphurisation – Acidification – Recycling – Thickening)
- Carbon in Columns Gold Recovery Circuit to recover gold and silver from the SART and CCD mill circuits.

The Santa Rita process is shown in Process Flow Drawings (PFD's) – #'s 10-F001, 20-F-001, 30-F-001 - Source: DENM (2017).

The proposed plant plan is shown in Drawing # - 01-G-001 –Source: DENM (2017)

Figure 17.1 Process Flow Drawing 10-F001 - Source: DENM (2017)

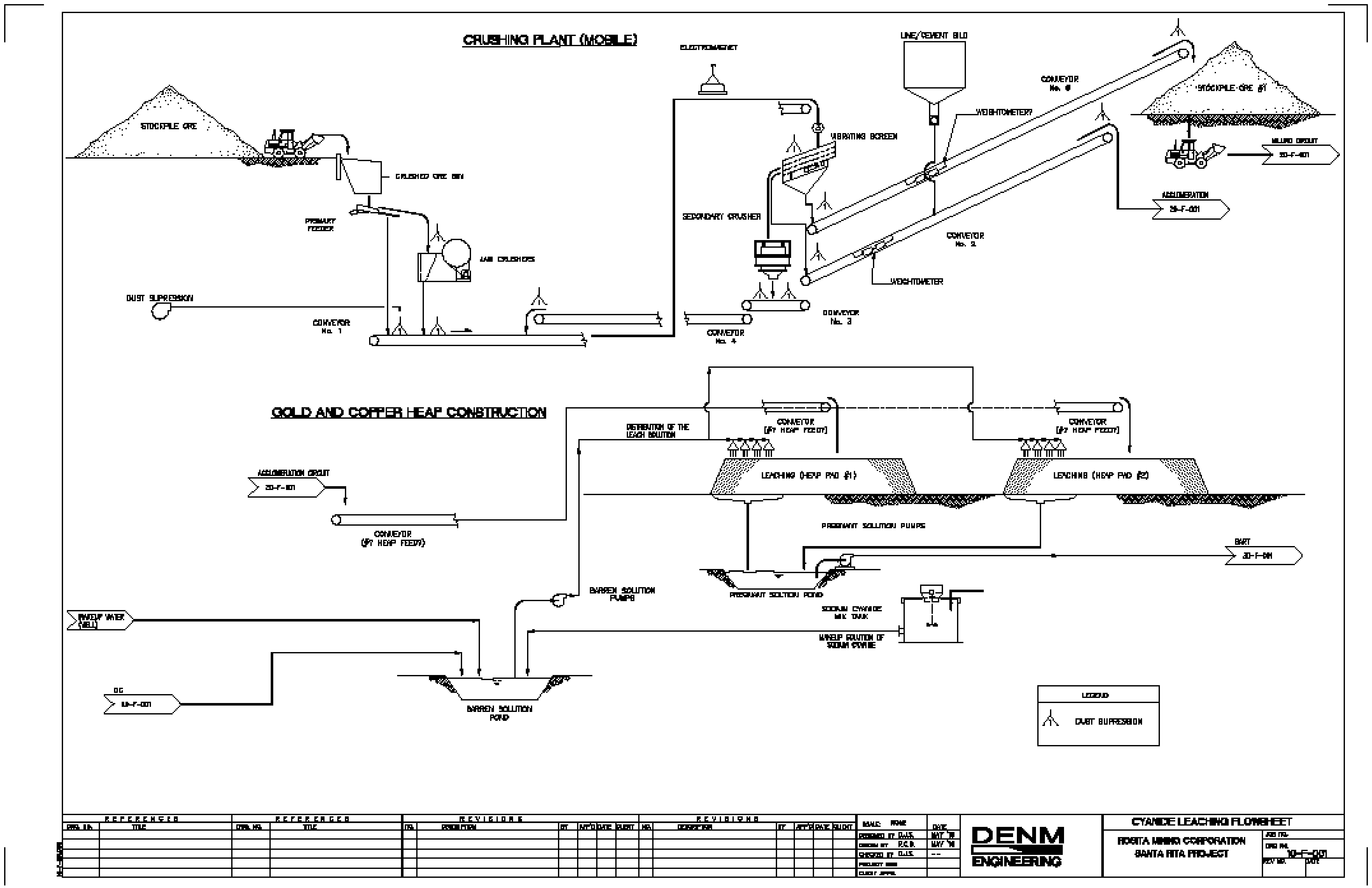


Figure 17.2 Process Flow Drawing 20-F-001- Source: DENM (2017)

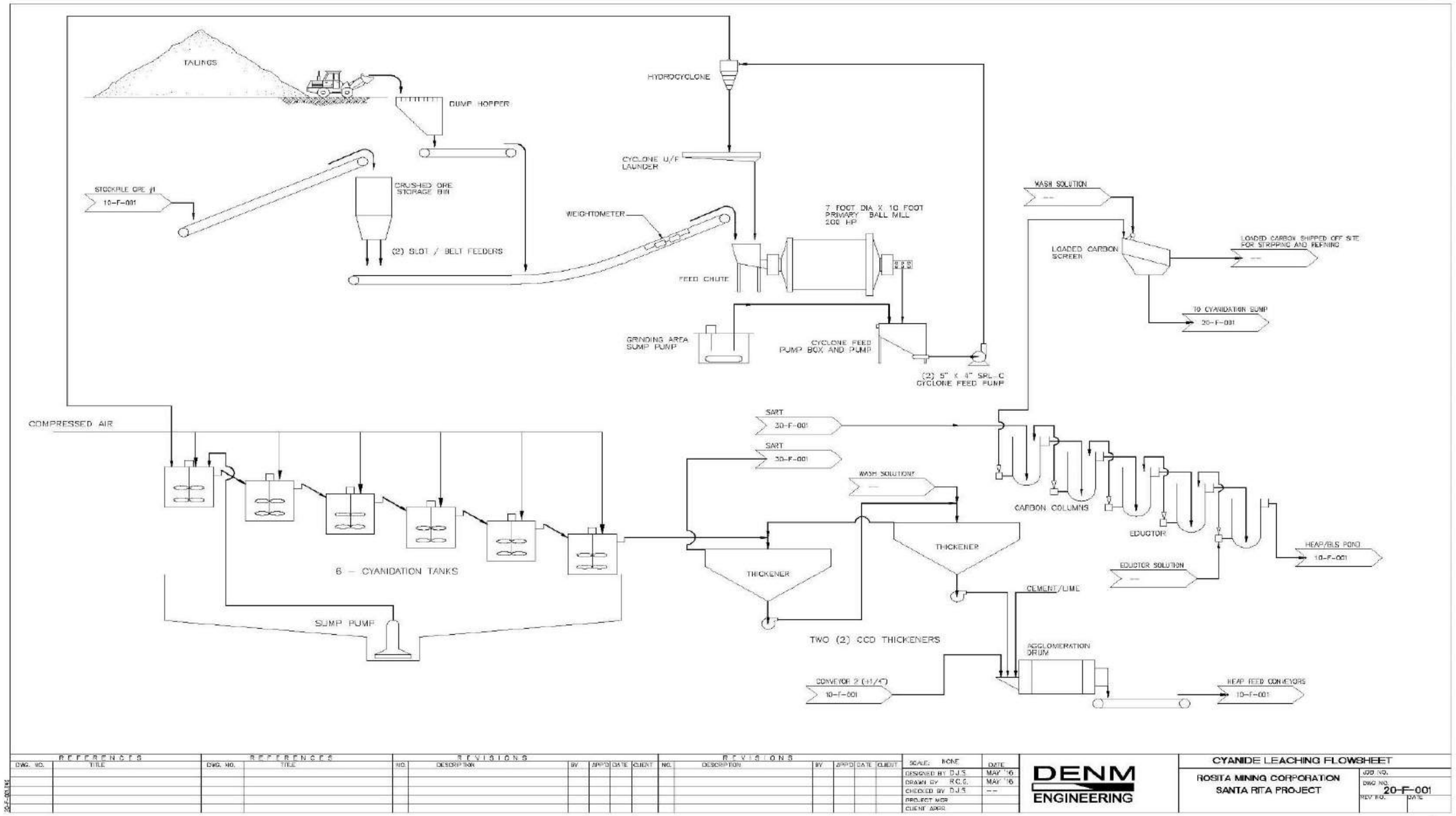


Figure 17.3 Process Flow Drawing 30-F-001 - Source: DENM (2017)

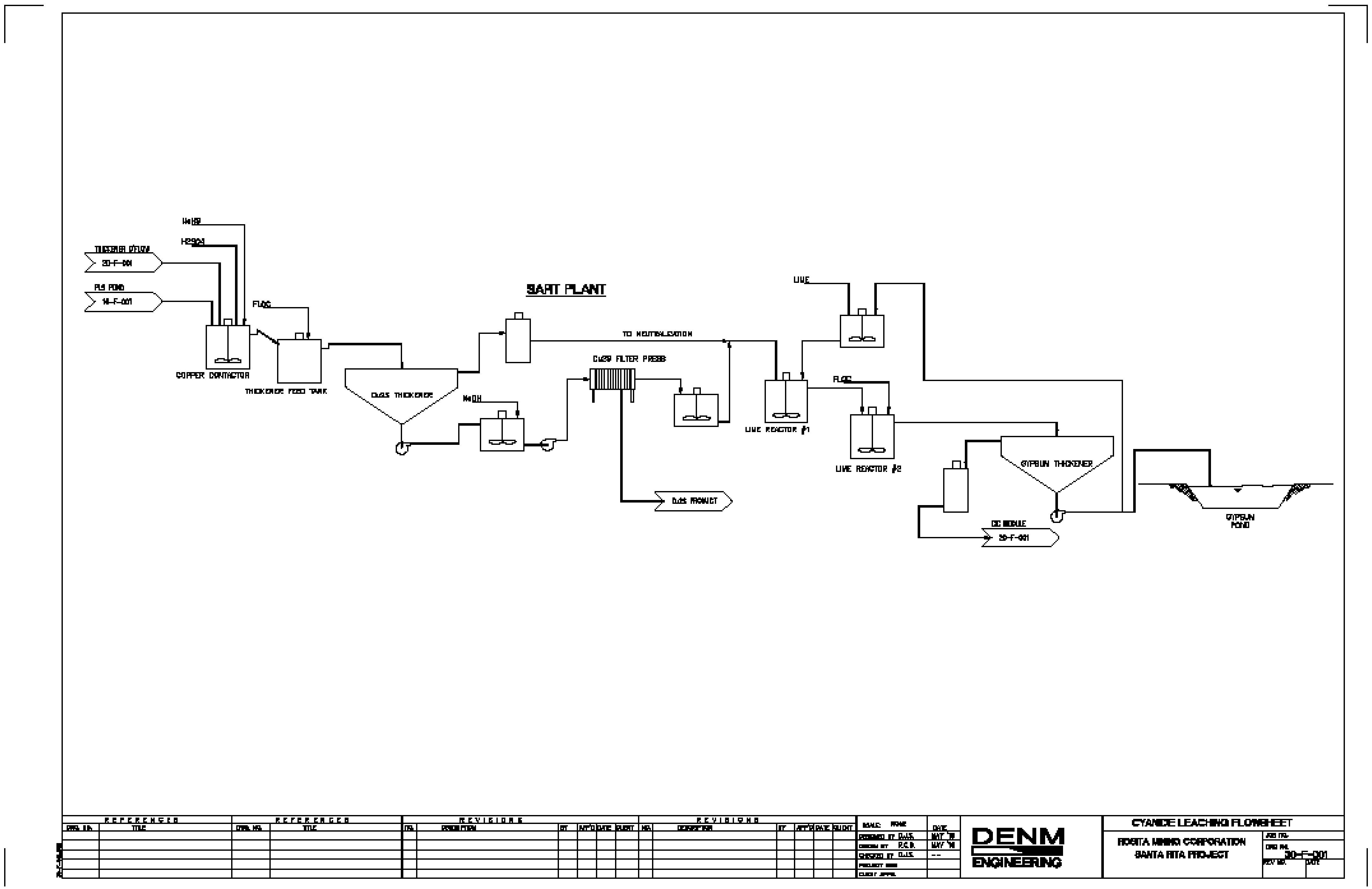
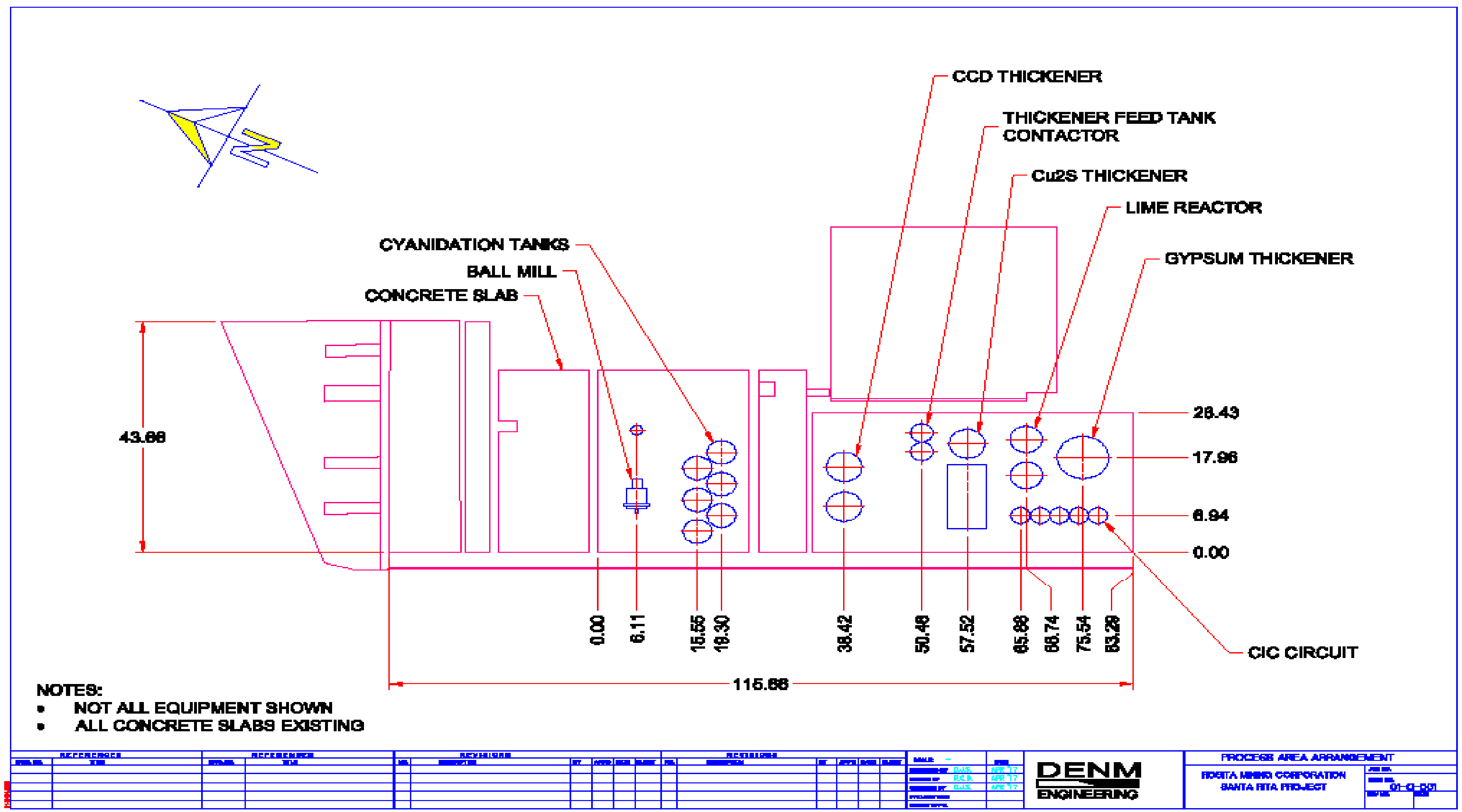


Figure 17.4 Process Area Arrangement 01-G-001- Source DENM (2017)



17.2 Stockpile and Tailing Resource

The resource material for reclamation is contained in six (6) distinct stockpiles and one(1) tailings located around the Santa Rita project area. The material has been impounded in its present location for over 20 years with little or no disturbance. The impounded material contains approximately 9 % moisture which was determined during the resource drilling work carried out in 2014 (Section 9.3.3)

Reclamation and excavation of both areas will at a minimal rate of 365,000 tonnes per year (Years 1-3) and 720,000 tonnes per year after Year 4. Transportation of the material will be via conventional dump trucks to the mobile crushing site on existing site private roadways. Haul distances range from 500 m. to 1000 m.

As mentioned previously, site water management (diversion and collection) will be implemented to collect and utilize rain water and also mitigate any risk from run-off of contamination to the surrounding populated area.

17.3 Mobile Crushing

The two (2) stage mobile crushing plant will be strategically located around the Santa Rita site to ensure close proximity to the stockpile being reclaimed and also crushed product surge capacity for the mill circuit. Closeness to the heap pad will also be beneficial to allow blending and agglomeration of the mill leach residue.

The plant will have a nominal capacity of 250-300 mtph (metric tonnes per hour) of feed material to produce two (2) final products of +¼-in. (heap leach) and -¼- in. (mill feed). Power required for the system will be via a 1000 kw (kilowatts) diesel generator.

Stage 1 Mobile Jaw Crusher will consist of a feed hopper, 1300 x 4900 vibrating grizzly feeder, PE900 x1200 primary jaw crusher and associated discharge conveyors and mounted on a tri-axle road trailer. The connected power is 165 kw and is complete with all electrical components and cabling.

Stage 2 Mobile Impact Crusher will consist of a three (3) deck vibrating screen, PF1315 impact crusher, and associated return, discharge and product stacking conveyors and mounted on tri-axle road trailer. The connected power is 250 kw and is complete with all electrical components and cabling.

The sizing of the mobile system is large enough to minimize operating hours and operating costs, and allow for Year 4 ramp up in production.

Product discharge from the system will be:

- -¼-in -in material stockpile for ball mill feed material
- +¼-in -in. material for blending of leach residue and agglomeration for heap leach pad loading

17.4 Gold and Copper Heap

The Stage 1 heap pad is designed to have a preliminary capacity of a total of three (3) years of feed material – 1.1 M tonnes. The pad will be designed to standard heap characteristics and will include excavation, compaction, geomembrane, LLDPE liner, overliner crushed material and all associated collection and drainage piping. The stage 1 pad area is 250 m. x 150 m. in dimension and will consist of four (4) x 10-m. each lifts. Event, pregnant, and barren ponds will be constructed and located in close proximity to the pad.

Mill leach residue (55 % density) and +¼-in. crushed material will be blended and agglomerated and loaded on the heap via conventional grasshopper and stacking conveyors. Agglomeration of the material is very important due the clayey nature of the material as well as to ensure proper agglomerate production for loading and leach permeability. During the agglomeration phase, lime and Portland cement are added as out lined in Section 13.

The blending of the leach residue with the dry coarse material allows for no requirement of a wet tailings area to be included in the site infrastructure which aids in permitting. Increased metal recovery is also possible.

As the pad is loaded, standard cyanide leaching and application rates will take place. Pregnant solution from the pad will be collected and pumped to the SART facility for copper recovery and cyanide recycle. Resultant solution from the Carbon-in-Column module (CIC) will partially returned to the barren pond for addition of cyanide make-up and pumping to leach distribution piping.

In Year 3, the Stage 2 pad will be constructed to be ready for Year 4 and on loading. A series of pads will continue to be built. for the life of mine. As each pad is fully loaded and rinsed, it will be available for possible copper acid leaching and extraction.

17.5 Milling Circuit

The milling process facility will be located on the existing concrete slab area previously used for the copper recovery plant (1967). The dimensions of this stepped concrete area are approximately 30 m. x 90 m. shown in the Drawing # 1-G-001 and Photos 17-1,2,3. There will be no building required for the process area but an engineered containment will encompass the area to ensure no risk of contamination to the surrounding environment.

The feed material for the mill will be a mixture from the -¼-in. stockpile plus tailings loaded into a feed hopper/ bin and discharged at a rate of 150 mtpd (6-8 mtph) (metric tonnes per day and hour) . The feed to the mill will controlled by a weightometer and variable speed belt conveyor. Recycled cyanide process water will pulp the feed material directed into a 7-ft. dia. X 10-ft. – 200 Hp modular ball mill. The ball mill discharge will be pumped to a hydrocyclone to ensure a P80 overflow sizing 75 µm (80% passing 200 mesh). The cyclone underflow is closed circuited return to the mill. The mill circuit is standard design with SRL slurry pumps, pump box, launders, and sumps. Ball loading and sizing is designed accordingly. Leach feed pH will be controlled at 11 via the addition of slurried lime.

The cyclone overflow – leach feed is fed to the leach circuit at a density of 35-40 % solids. The leach circuit consists of six (6) reactors in series to ensure a retention time of 48 hours maximum. The leach reactors will be standard fixed speed, hydrofoil type impellors, and air sparged. The 4.5 m. dia x 6 m. high tanks will be stepped to allow gravity flow between tanks. The mill leach recovery will be 85 % Au, 60 % Ag, and 35 % Cu (soluble oxide copper). Cyanide addition will be controlled to maximize gold and copper dissolution and high free cyanide to promote the required leach gradients.

After leaching, the process flow will enter a two (2) stage counter current decantation system consisting of two(2) thickeners (5.5 m. dia.) in series. Flocculant will be added to promote settling with the resultant final underflow density from Thickener # 2 at 55 % density. This will be blended into the agglomeration of the coarse material as outlined above.

The thickener overflow will be collected with the heap leach pregnant solution to feed the SART circuit.

Photo 17.1 Photograph of existing old plant area – Mill slab area (DENM 2016)

Photo 17.2 Photograph of existing old plant area – after clean-up (DENM 2016)

Photo 17.3 Photograph of existing old plant area – after clean-up (DENM 2016)

17.6 SART Circuit

The two streams – thickener overflow and heap pregnant solution – will combine to feed the SART circuit for copper (Cu_2S) precipitation and for cyanide recovery and recycle. The process is designed for the Santa Rita site specific conditions and was tested during the recent Lakefield work in 2016-2017.

A simple flowsheet schematic of the circuit is shown in Figure 17.5 – Source: DENM (2017)

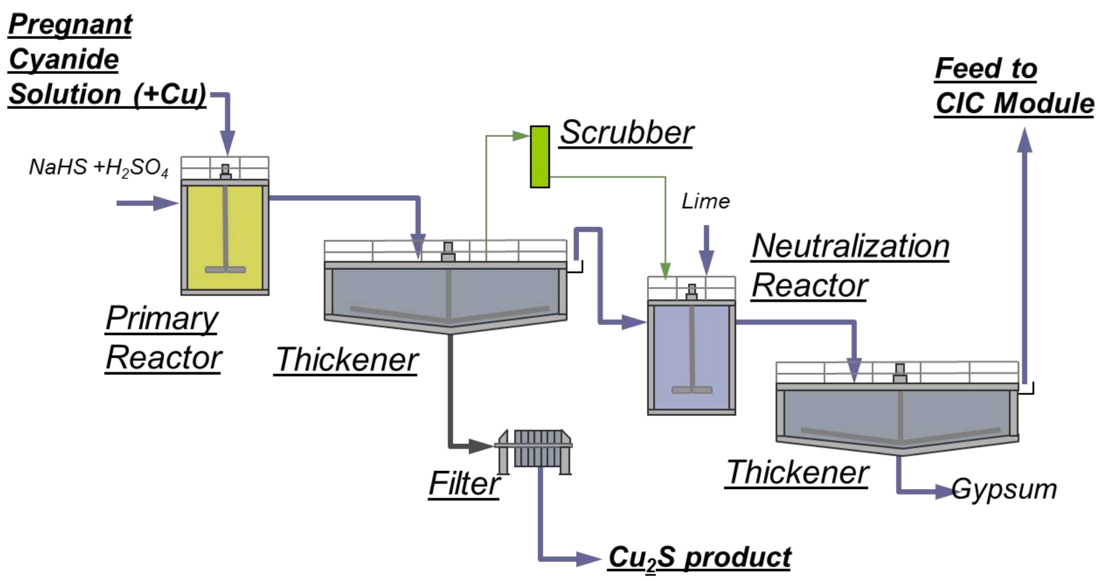


Figure 17.5 – Rosita Mining – Santa Rita SART Circuit Schematic – Source DENM (2017)

The feed to the SART enters the primary reactor (3.5 m. dia. X 4.2 m. high) at a flowrate of 200 cu.mt/hr. Sulphuric acid (H_2SO_4) is added to adjust the pH to 4 with addition of Sodium Hydrosulphide (NaHS) to precipitate the copper. The copper precipitated (in the form of Cu_2S) is that associated with the copper cyanide from the leach processes. The resultant cyanide from the copper returns as free cyanide in the process flow for recycle. The precipitated material stream flows to the thickener (5.5 m. dia.) for solid liquid separation and subsequent filtering of the copper concentrate for marketable sale. The clarified overflow (with regenerated cyanide) is neutralized to pH of 11 with line and clarified to produce a gypsum product for disposal. The overflow will feed the CIC adsorption circuit for gold and silver recovery.

The SART circuit will be adjacent and within the milling circuit footprint at Santa Rita.

17.7 Carbon in Columns (CIC) Circuit

The SART circuit stream feeds a CIC circuit designed to handle 230 m^3 /hr of pregnant solution through one (1) set pf carbon adsorption columns. The set of columns consists of five (5) – 2.5 m. dia x 3.0 m. up flow design tanks loaded with approx. 1.5 tonnes of 6 x 12 activated carbon. The tanks are stepped and in series to allow gravity flow between the tanks. Carbon advancement is done via eductors and counter-current to the process flow.

Loaded carbon is forwarded to a carbon screen for loading a shipping off site for stripping and gold and silver dore production.

The CIC barren solution drains back to the barren solution pond for cyanide make-up. A portion of this stream will also be used for pulping in the milling circuit.

17.8 Process Design Criteria

The preliminary process design criteria used for the Santa Rita PEA was developed from metallurgical testing results, calculated factors and certain process assumptions as noted in Table 17.1

Table 17.1 Process Design Criteria

	Units (metric)	Value	Comments
Feed Composition:			
Gold	g/t	0.65	Years 1-5
Silver	g/t	10.00	
Copper	%	0.80	
Gold	g/t	0.47	Years 6-10
Silver	g/t	7.79	
Copper	%	0.54	
Nominal Blended Plant Throughput	Tpa	365,000	Years 1-3 - based on split of 850 Tpd +¼-in. and 150 Tpd mill (50 Tpd -¼-in + 100 Tpd tailings)

	Tpa	720,000	Year 4-10 - based on split of 1700 Tpd +¼-in. and 300 Tpd mill (100 Tpd -¼-in + 200 Tpd tailings)
Mill Recoveries	% Au % Ag % Cu	85 % 65 % 35 %	
Heap Leach Recoveries	% Au % Ag % Cu	65 % 45 % 35 %	
Cyanide Make-Up Lime Required, CaO	Kg/t Kg/t	2.50 5.00	Addition to SART discharge solution
Mill Leaching Retention Time	Hours	48	
Grind Size , P_{80}	Microns	75	Cyclone Overflow to leach circuit
Bond Work Index	kWh/t	13.9 8.7	Deslimed With slimes
Bulk Densities	t/cu.mt.	2.0 1.82	Stockpile Tailings

Thickener Underflow to Agglomeration	%	55	Blended with +¼-in. crushed material
SART Design Flowrate	m^3/hr	200	PLS from heap leach and thickener overflow from milling
NaHS Addition in SART Lime Addition (CaO) in SART Acid (H_2SO_4) in SART	Kg/m^3 Kg/m^3 Kg/m^3	0.75 4.30 5.60	100 % stoichiometric pH 11 neutralization pH 4 acidification
Carbon in Columns (CIC)	m^3/hr	230	1.5 tonnes/column
Acid Copper Leach Recovery	%	35 %	

Source: DENM (2017)

18 PROJECT INFRASTRUCTURE

18.1 Power Supply

The preliminary plant total connected requirement has been determined to be 1000 kw. The mobile crushing plant has its own generator. Power will be supplied by connecting the local Nicaraguan high voltage grid adjacent to the proposed process facility. There is available power on the grid following discussions with the local power authorities (ENEL). There is also reported to be system upgrades and increases with the Rosita grid area.

The electrical power near the site and shown in the figure is currently supplied is a 138 kV power line. Sub-station and transformers will be required on site to reduce the incoming voltage to suit plant power of 460V/3 phase/60 cycle.

18.2 Water Supply

Supply and make-up water will be come from the large impounded volumes in the Santa Rita and R-13 pits. Water collection and management during the rainy season will supplement the process requirements. Potable water service is not available at the site from the municipality of Rosita.

18.3 Buildings

There will be no planned construction of buildings for the process plant. Rosita does have main office facilities in the town of Rosita house technical people and management. The mining contractor will have its own facilities for maintenance.

Small office and operating rooms will be done via mobile trailers with phone, internet, and power services. Temporary mobile trailers will be utilized during the construction phase of the project.

18.4 Ancillary Facilities

Potable water will be trucked in from the Rosita and small storage of the same will be at site.

Sewage services for the facility will be into a septic system and pumped as required.

Communications will via cellular phones and internet services.

18.5 Tailings Facility Storage

The preliminary process for the Santa Rita project is without a wet tailings storage facility. The leach residue will be blended and mixed with the oversize dry material to feed the heap pads. The only wet residue from the process will be the SART gypsum by-product which is minimal and will be pumped and impounded in a small lined pit.

18.6 Project Site Plan

A preliminary schematic of the site showing the location of the heap and process facility are shown in Drawing #'s – 00-G-001 and 00-G-002 – Source: DENM (2017) These locations are subject to change and adjustment based on affected landowners and municipal requirements.

Figure 18.1 Project Site Plan (Source DENM 2017)

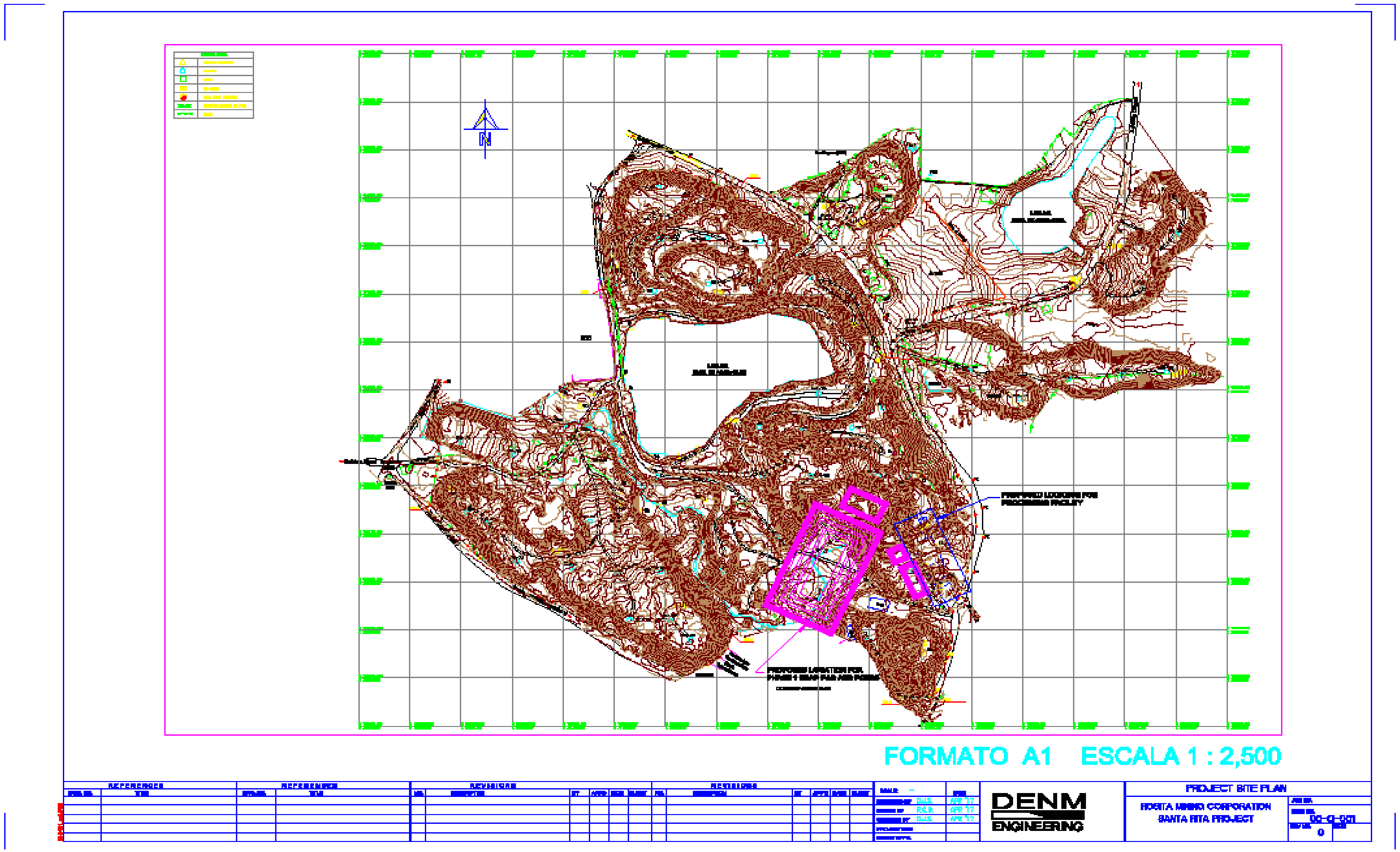
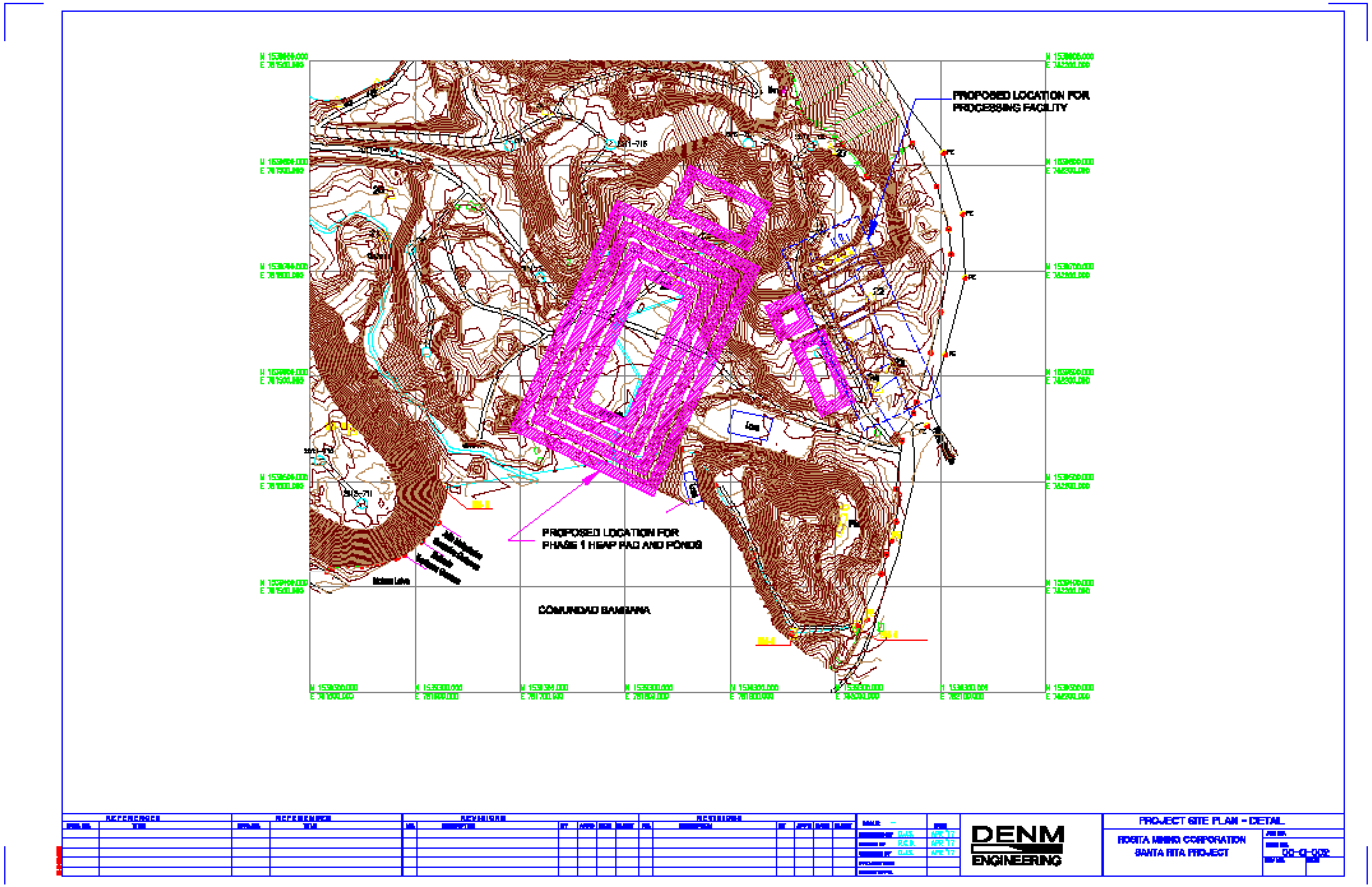


Figure 18.2 Project Site Plan – Detail (Source: DENM 2017)



19 MARKET STUDIES AND CONTRACTS

At the time of this report, DENM is not aware of any project-specific contracts or off-take agreements for the sales of the products. This would include but not be limited to a refinery contract and agreement of the sale of the dore (gold and silver) and sales of Cu_2S concentrate to a smelter. Certain assumption based on experience have been made in the values of these products.

Supply contracts required prior to production at Santa Rita will be for:

Consumables – Sodium cyanide (NaCN), Sodium hydrosulphide (NaHS), Acid (H_2SO_4) , Lime (CaO) , grinding media and other reagent requirements.

Carbon Stripping – As there is no allowance for stripping and refining of the gold and silver, this will be done off-site and will require a contract.

Supply of electrical power – the budget cost of power for this study was quoted at $US 0.14/kwh. Confirmed rates, consumption and delivery details will be required from the local power company.

20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

The Santa Rita project is located in an area of previous mining from years of 1960-1978 and a large amount of surface disturbance. This includes two(2) large open pits currently with water, a number of surface stockpiles, and a large area of impounded tailings. The project is located within the municipal limits of Rosita and its presence is of some concern to the local population. A present, there is no reported environmental concerns and urgent need for reclamation of the resource outlined.

As part of the permitting process, RST will apply for an Environmental Permit to allow operation from the Ministry of Environment and Natural Resources. As the Santa Rita Project is located in the North Autonomous Region of Nicaragua, the permit is done via the Secretary of Ministry of Natural Resources (SERENA). There are well documented steps and requirements for this permit application and similar in nature to permitting in Canada. Typical permitting times for this are 8-10 months. It is the intent of RST to file this in Q3-2017 with the permit being prepared and submitted by a local Nicaraguan consulting group. Baseline work and preparation of the permit is presently underway by RST.

Table 20.1 – List of Major Permits Required (Summary)

Organization	Permit	Timeline
Ministry of Energy and Mines (through the Directorate General of Mines)	Registration and authorization for the operation of the plant	Q3-2017
SERENA	Environmental Operating	Q3-2017
Forestry (INAFOR)	Permission to cut trees	Q4-2017
Municipality of Rosita	Land Use Permit (Construction and Operation)	Q3-2017
Water Stewardship (ANA)	Water Rights License	Q4-2017
Ministry of Energy and Mines through the Directorate	Equipment Importation	Q4-2017

General of Mines and Ministry of Treasury		
Nicaraguan Power (ENEL)	Electrical Permit and Supply Contract	Q4-2017
Direction of the Chemical Safety (attached to the Presidency of the Republic)	Cyanide Importation	Q1-2018

Source : DENM (2017)

The Santa Rita project will be a definite boast the local economy and work force. The capital costs for the initial investment year investment will be $ US 11.4 million with local equipment and construction direct costs accounting for approximately 50 % of this estimate.

The project will also require a direct labour force of 77 employees that will include general manager, administration, operators, and maintenance personnel. In year 4 after ramp up of the project, an additional 20% increase is manpower is projected. These numbers do not include any additional work force for contract loading and hauling and other local services. The local workforce labour costs have included the base salary plus the required social cost payments that results in an additional 46 %.

21 CAPITAL AND OPERATING COSTS

21.1 Capital Costs

The Santa Rita Project requires direct capital expenditure in Year 0 of $ US 11.4 M and is shown in Table 21.1. The expected accuracy of the estimates is +/- 30 %

Table 21.1 Santa Rita Direct Capital Expenditures Summary

ITEM	Initial Capital Costs ($US 000)
Portable Crushing System	$700
Milling Circuit	$1,500
CCD Circuit	$300
Stage 1 Heap Pad	$1,500
Stage 1 Heap Pumping	$200
Carbon in Column (CIC) Circuit	$1,500
SART Plant	$3,100
Capital Contingency (30 %)	$2,640
TOTAL CAPITAL EXPENDITURE	$11,440

Source: DENM (2017)

Additional sustaining capital for the project is $ US 14.7 M and allows for project ramp up, heap expansions, and an acid plant addition. Indirect costs, EPCM and working capital have been included in the individual items and contingency capital.

21.2 Operating Cost – Santa Rita Summary

Operating costs outlined in this section were derived from local supplied costs. These included power, labour, reagent consumables, and local contractor costs. In areas where quotes were unavailable, certain assumptions were made.

The following Table 21.2 is a summary of the cash flow operating costs for the ten (10) year projected life of mine scenario. These costs are reflected in the preliminary projected cash flow spreadsheet.

Table 21.2 Santa Rita Summary of Operating Cost Estimates for Life-of-Mine

TEM	Life-of mine Costs ($US 000)	$ US/ tonne
Mining (Loading and Hauling)	$11,169	$2.00
Total Plant Labour Years 1-3 Years 4-10	$13,538	 $3.25 $1.98
Mill Process Reagents	$5,585	$6.00
Overall Plant Power – $US 0.14 /kWh	$12,410	$2.00
Heap Leach	$31,646	$6.00
SART Process	$39,154	$6.31
Copper Leach (Year 4-10)	$18,396	$4.00
Total Operating Costs	$131,396	$18.50 (this is a blended # for LOM)

Source: DENM (2017)

22 ECONOMIC ANALYSIS

22.1 Summary

The preliminary projected cash flow (Figure 22.2 Cash Flow and NPV Evaluation Spreadsheet) and 7-year NPV were calculated using the following assumptions and presented in Table 22.1.

- Base Metals Prices – Gold - $US 1,250/oz., Silver - $US 18.00/ oz., Copper - $US 2.50/lb.
- Change in the Metal Prices scenarios- -10 %, base case, +10 % overall
- IRR calculations were done both “pre-tax” and “post tax”
- Net Cash Flow and Payback (on initial capital) done on “after tax basis”
- Tax rates are based on 30 % after depreciation of fixed at 10%

Table 22.1 Santa Rita Summary of Economic Analysis

	Low Value	Base Case	High Value
IRR Pre-Tax Post tax	 38% 31%	 51 % 41%	 63 % 50%
Net Cash Flow	$US 35.146 million	$US 51.205 million	$US 67.263 million
Payback	3.2 years	2.8 years	2.3 years

Source: DENM (2017)

22.2 Sensitivity Analysis

Table 22.2 and Figure 22.1 presents the sensitivity of the Rosita Mining – Santa Rita Project life-of-mine (10 years) cash flow to metals prices, operating, and capital costs. The scope of variation used for the parameters are minus 10% to plus 10%

Table 22.2 Summary of Project Life-of-Mine Cash Flow ($US millions)

Parameters	Low Value	Base Case	High Value
Overall Metal Prices Gold($US/oz.) Silver($US/oz.) Copper($US/lb.)	 $1,125 $16.20 $2.25	 $1,250 $18.00 $2.50	 $1,375 $19.80 $2.75
Net Cash Flow	$35.146	$51.205	$67.263
Sensitivity	**-10 %**	**0**	**+10%**
Overall Metal Production	$37.319	$51.205	$63.641
Capital Cost	$52.104	$51.205	$50.305
Operating Cost	$60.438	$51.205	$41.972

Source: DENM (2017)

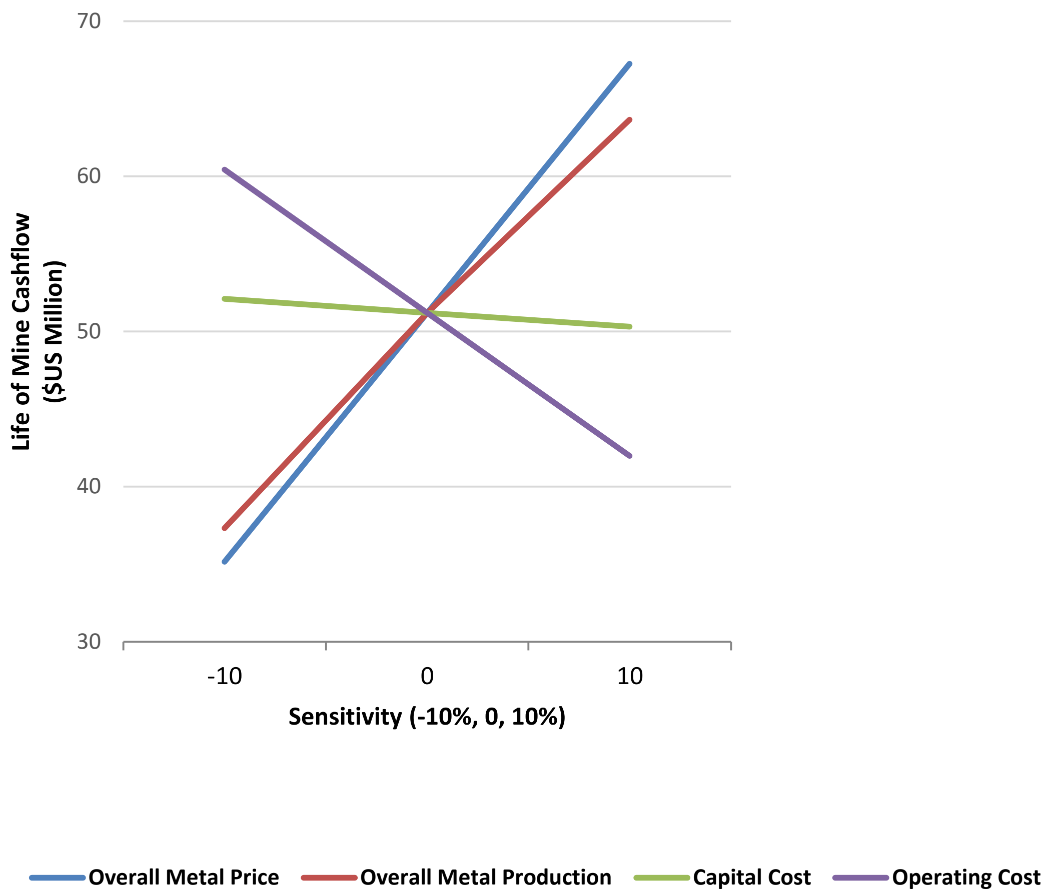


Figure 22.1Sensitivity Analysis

Figure 22.2 Cash Flow and NPV Evaluation Spreadsheet (Source DEMM 2017)

Rosita Mining															
PEA Study															
Santa Rita Project															
			Investment				Operating Years								
			Year 0		Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Project Totals
PRODUCTION:															
Stockpile to Crusher					328,500	328,500	328,500	657,000	657,000	657,000	657,000	657,000	657,000	657,000	5,584,500
SP tonnes to mill					18,250	18,250	18,250	36,500	36,500	36,500	36,500	36,500	36,500	36,500	310,250
Tailings tonnes to mill					36,500	36,500	36,500	73,000	73,000	73,000	73,000	73,000	73,000	73,000	620,500
Total Tonnes to Mill (tpy)					54,750	54,750	54,750	109,500	109,500	109,500	109,500	109,500	109,500	109,500	930,750
Mill Grade - Au (gm/Tonne)					0.65	0.65	0.65	0.65	0.65	0.47	0.47	0.47	0.47	0.47	
Mill Grade - Ag(gm/Tonne)					10.00	10.00	10.00	10.00	10.00	7.79	7.79	7.79	7.79	7.79	
Mill Grade - % Cu					0.60	0.60	0.60	0.60	0.60	0.50	0.50	0.50	0.50	0.50	
% Mill Recovery (Au)	65%				0.65	65%	65%	65%	65%	65%	65%	65%	65%	65%	
% Mill Recovery (Ag)	65%				0.65	65%	65%	65%	65%	65%	65%	65%	65%	65%	
% Mill Recovery (Cu)	35%				0.35	35%	35%	35%	35%	35%	35%	35%	35%	35%	
Recovered gold (gm)	Mill Circuit				30,249	30,249	30,249	60,499	60,499	43,422	43,422	43,422	43,422	43,422	436,687
Recovered gold (oz)					973	973	973	1,945	1,945	1,396	1,396	1,396	1,396	1,396	13,799
Recovered silver (oz)					11,440	11,440	11,440	22,881	22,881	17,814	17,814	17,814	17,814	17,814	189,180
Recovered copper (lb)					331,116	331,116	331,116	662,231	662,231	413,096	413,096	413,096	413,096	413,096	4,057,294
Mill REVENUE:															
Gold Price ($US/oz)					$1,250.00	$1,250.00	$1,250.00	$1,250.00	$1,250.00	$1,250.00	$1,250.00	$1,250.00	$1,250.00	$1,250.00	
Silver Price ($US/oz)					$18.00	$18.00	$18.00	$18.00	$18.00	$18.00	$18.00	$18.00	$18.00	$18.00	
Copper Price ($US/lb) smelter return					$2.50	$2.50	$2.50	$2.50	$2.50	$2.50	$2.50	$2.50	$2.50	$2.50	
Gold Revenue ($US)					$1,215,894	$1,215,894	$1,215,894	$2,431,387	$2,431,387	$1,745,100	$1,745,100	$1,745,100	$1,745,100	$1,745,100	$17,235,368
Silver Revenue ($US)					$205,926	$205,926	$205,926	$411,853	$411,853	$320,644	$320,644	$320,644	$320,644	$320,644	$3,044,703
Copper Revenue ($US)	Mill Only				$827,789	$827,789	$827,789	$1,655,579	$1,655,579	$1,034,737	$1,034,737	$1,034,737	$1,034,737	$1,034,737	$10,958,259
Gross Au/Ag/Cu ($US)					$2,249,409	$2,249,409	$2,249,409	$4,498,819	$4,498,819	$3,100,481	$3,100,481	$3,100,481	$3,100,481	$3,100,481	$31,248,286
Stockpile To Heap Leach (tpd)					310,250	310,250	310,250	620,500	620,500	620,500	620,500	620,500	620,500	620,500	5,274,250
Heap Grade - Au (gm/Tonne)					0.65	0.65	0.65	0.65	0.65	0.47	0.47	0.47	0.47	0.47	
Heap Grade - Ag(gm/Tonne)					8.00	8.00	8.00	8.00	8.00	7.79	7.79	7.79	7.79	7.79	
Heap Grade - % Cu					0.60	0.60	0.60	0.60	0.60	0.54	0.54	0.54	0.54	0.54	
% Recovery Heap - Au	65%				65%	65%	65%	65%	65%	65%	65%	65%	65%	65%	
% Recovery Heap - Cu	35%				35%	35%	35%	35%	35%	35%	35%	35%	35%	35%	
Recovered gold (gm)					131,081	131,081	131,081	262,161	262,161	188,163	188,163	188,163	188,163	188,163	1,888,388
Recovered gold (oz)					4,214	4,214	4,214	8,429	8,429	6,050	6,050	6,050	6,050	6,050	59,749
Recovered silver (oz)	45%				35,910	35,910	35,910	71,819	71,819	69,893	69,893	69,893	69,893	69,893	600,852
Recovered copper (lb)	(50 % SART)				1,676,323	1,676,323	1,676,323	3,752,645	3,752,645	2,533,035	2,533,035	2,533,035	2,533,035	2,533,035	35,798,434
Gold Revenue ($US)					$5,268,006	$5,268,006	$5,268,006	$10,536,011	$10,536,011	$7,562,101	$7,562,101	$7,562,101	$7,562,101	$7,562,101	74,686,845
Silver Revenue ($US)					$646,375	$646,375	$646,375	$1,292,750	$1,292,750	$1,258,072	$1,258,072	$1,258,072	$1,258,072	$1,258,072	10,814,964
Copper Revenue ($US)	Au Heap Only				$4,690,806	$4,690,806	$4,690,806	$9,381,613	$9,381,613	$6,332,588	$6,332,588	$6,332,588	$6,332,588	$6,332,588	84,496,896
Copper Revenue ($US)	Cu Heap Only rec	0.35			$0	$0	$0	$10,301,760	$10,301,760	$6,930,509	$6,930,509	$6,930,509	$6,930,509	$6,930,509	86,268,068
Gross Revenue Au/Ag/Cu ($US)					$10,605,187	$10,605,187	$10,605,187	$31,512,134	$31,512,134	$22,083,270	$22,083,270	$22,083,270	$22,083,270	$22,083,270	208,256,176
Total Gross (Mill + Heap) $US					$12,854,596	$12,854,596	$12,854,596	$36,010,952	$36,010,952	$25,183,751	$25,183,751	$25,183,751	$25,183,751	$25,183,751	$236,504,448
Royalty					$385,638	$385,638	$385,638	$1,080,329	$1,080,329	$755,513	$755,513	$755,513	$755,513	$755,513	$7,095,133
Net revenue					$12,468,958	$12,468,958	$12,468,958	$34,930,624	$34,930,624	$24,428,238	$24,428,238	$24,428,238	$24,428,238	$24,428,238	$229,409,315
OPERATING COSTS ($US):															
Loading and Material Handling	2.00	$U.S/tonne			$657,000	$657,000	$657,000	$1,314,000	$1,314,000	$1,314,000	$1,314,000	$1,314,000	$1,314,000	$1,314,000	$11,169,000
(Contract Loading and Hauling)															
Total Plant labour (77 total) + 20 % Year 4	$3.25	(Year 1-3)		$1.96	$1,186,250	$1,186,250	$1,186,250	$1,425,800	$1,425,800	$1,425,800	$1,425,800	$1,425,800	$1,425,800	$1,425,800	$13,537,950
Mill Process Reagents	$6.00	$U.S/tonne			$328,500	$328,500	$328,500	$657,000	$657,000	$657,000	$657,000	$657,000	$657,000	$657,000	$5,584,500
Overall Plant Power	$2.00				$730,000	$730,000	$730,000	$1,460,000	$1,460,000	$1,460,000	$1,460,000	$1,460,000	$1,460,000	$1,460,000	$12,410,000
Heap Leach	$1.00														
Reagents (cyanide, cement, lime)	$5.00														
Total Heap Leach	$6.00				$1,861,500	$1,861,500	$1,861,500	$3,723,000	$3,723,000	$3,723,000	$3,723,000	$3,723,000	$3,723,000	$3,723,000	$31,645,500
SART Process (180 cu.m/hr)	$6.31	$U.S./tonne			$2,303,150	$2,303,150	$2,303,150	$4,606,300	$4,606,300	$4,606,300	$4,606,300	$4,606,300	$4,606,300	$4,606,300	$39,153,550
Subtotal Operating Costs (Year 1-3)	19.56	$ U.S./tonne			$7,066,400	$7,066,400	$7,066,400	$13,185,900	$13,185,900	$13,185,900	$13,185,900	$13,185,900	$13,185,900	$13,185,900	$113,500,500
Subtotal Operating Costs (Year 4..)	18.31	$ U.S./tonne													
Copper Leach (Year 4)	$4.00	$ U.S./tonne						$2,628,000	$2,628,000	$2,628,000	$2,628,000	$2,628,000	$2,628,000	$2,628,000	$18,396,000
Total Operating Cost					$7,066,400	$7,066,400	$7,066,400	$15,813,900	$15,813,900	$15,813,900	$15,813,900	$15,813,900	$15,813,900	$15,813,900	$131,896,500
Cash Flow before Capital					$5,402,558	$5,402,558	$5,402,558	$19,116,724	$19,116,724	$8,614,338	$8,614,338	$8,614,338	$8,614,338	$8,614,338	$97,512,813
CAPITAL COSTS ($US):															
Portable Crushing System			$ 700,000												
Milling Circuit			$ 1,500,000												
CCD Circuit			$ 300,000												
Heap Pads (sq.mt.)			$ 1,500,000			$ 2,000,000	$ 3,000,000	$ 2,000,000	$ 2,000,000						
Acid Circuit							$ 200,000								
Heap Pumping			$ 200,000				$ 500,000								
CIC Circuit			$ 1,500,000				$ 1,000,000								
SART Pumping			$ 100,000												
SART Plant			$ 3,000,000				$ 2,000,000								
Cu Acid Recovery							$ 2,000,000								
Indirects and EPCM	Included in Cont.														
Working Capital	Included in Cont.														
Subtotal Capital			$ 8,800,000		$ -	$ 2,000,000	$ 8,700,000	$ 2,000,000	$ 2,000,000	$ -					$23,500,000
Capital Contingency	30%		$ 2,640,000							$ -					$2,640,000
Total capital			$ 11,440,000		$ -	$ 2,000,000	$ 8,700,000	$ 2,000,000	$ 2,000,000	$ -	$ -	$ -	$ -	$ -	$26,140,000
Cumulative capital			$ 11,440,000	$ 11,440,000	$ 11,440,000	$ 13,440,000	$ 22,140,000	$ 24,140,000	$ 26,140,000	$ 26,140,000	$ 26,140,000	$ 26,140,000	$ 26,140,000		
Net Cash Flow before Taxes			-$ 11,440,000		$ 5,402,558	$ 3,402,558	-$ 3,297,442	$ 17,116,724	$ 17,116,724	$ 8,614,338	$ 8,614,338	$ 8,614,338	$ 8,614,338	$ 8,614,338	71,372,813
Tax Credit 10% on Capital					-$ 1,144,000	-$ 1,144,000	-$ 1,229,800	-$ 1,676,640	-$ 1,678,976	-$ 1,961,070	-$ 1,782,971	-$ 1,604,674	-$ 1,444,208	-$ 1,299,786	
Cumulitive Tax Credit					-$ 11,440,000	-$ 12,296,000	-$ 19,796,400	-$ 19,769,760	-$ 19,210,784	-$ 17,829,706	-$ 16,046,735	-$ 14,442,062	-$ 12,997,856	-$ 11,698,070	
Sub-total					$ 4,258,558	$ 2,258,558	-$ 4,527,242	$ 15,140,084	$ 15,137,748	$ 6,653,268	$ 6,831,367	$ 7,009,664	$ 7,170,130	$ 7,314,552	
Less Income Taxes 30%					-$ 1,277,567	-$ 677,567	$ 1,358,113	-$ 4,642,025	-$ 4,641,324	-$ 1,989,978	-$ 2,049,410	-$ 2,102,899	-$ 2,151,040	-$ 2,194,366	
Net Cash Flow after Taxes			-$ 11,440,000		$ 4,124,991	$ 2,724,991	-$ 1,939,329	$ 12,474,699	$ 12,475,399	$ 6,624,360	$ 6,564,928	$ 6,511,439	$ 6,463,298	$ 6,419,972	
Cumulative Cash Flow			-$ 11,440,000		-$ 7,315,009	-$ 4,590,018	-$ 6,529,348	$ 6,045,351	$ 18,420,751	$ 25,045,111	$ 31,610,039	$ 38,121,477	$ 44,784,775	$ 51,204,745	
with revenues after Gov royalty (5% NSR)															
	Pre-Tax		Post tax		Rate										
	$71,372,813		$51,204,748		0%										
	$58,551,695		$41,613,614		2.5%										
	$48,246,182		$27,680,016		5%										
	$39,899,585		$33,911,342		7%										
	$33,092,166		$22,602,685		10%										
	$15,836,168		$8,750,324		20.0%										
IRR	61%		41%												

23 ADJACENT PROPERTIES

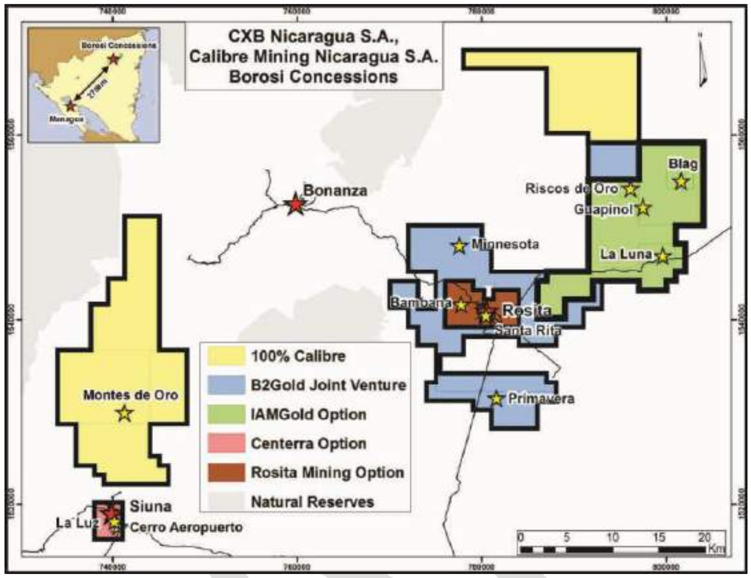
The Mining Triangle of Nicaragua, one of the most prolific mining districts of Central America, has an estimated historical production of 7.9 million ounces of gold, 4 million ounces of silver and 305 million pounds of copper (Arengi, 2002). La Luz-Siuna gold mine and Rosita copper mine were two major historic operated mines. Both deposits were of the skarn model type. A number of smaller past producing gold deposits are located on the Borosi concessions including the La Luna, Riscos de Oro and Blag historic mines. Of the three historic mining towns that make up the "Golden Triangle" only Bonanza is currently producing gold (other towns are Siuna-La Luz and Rosita).

According to www.calibremining.com, Calibre owns a 100% interest in over 413 kilometres^2 of mineral concessions in the Mining Triangle of Northeast Nicaragua including the Primavera Project, Santa Maria Project and Monte Carmelo Project. Additionally, the Company has optioned to IAMGOLD (176 km^2) and Centerra Gold (253 km2) concessions covering an aggregate area of 429 kilometres^2 and is party to a joint venture (67% owned by RST and 33% by Calibre) on the 33.6 kilometres^2 Rosita D gold-copper-silver project with Rosita Mining Corporation (Figure 23.1). The Borosi concessions have the potential to host several major deposit types including Low Sulphidation epithermal veins (gold and silver), Skarns (gold, silver, copper, zinc, and iron), Porphyry (gold and copper) and Intrusion Related deposits (gold, silver and copper).

Calibre Mining Corp has reported NI 43-101 compliant inferred resources (shown in Table 23.1) on Cerro Aeropuerto (2011), La Luna (2011) and Riscos de Oro (2012) projects within the Borosi concessions.

FIGURE 23.1

ADJACENT PROPERTIES



Source: www.calibreming.com

TABLE 23.1

INFERRED RESOURCES ON BOROSI CONCESSIONS OF CALIBRE MINING CORP.

(Using A 0.6 G/T Au eq Cutoff Grade)

Zone	Tonnage (mt)	Au g/t	Ag g/t	AuEq g/t	Contained Au (koz)	Contained Ag (koz)	Contained AuEq (koz)
La Luna	2.54	1.56	14.01	1.78	127.70	1,143.6	146
Cerro Aeropuerto	6.05	3.64	16.16	3.89	707.75	3,144.5	757
Riscos de Oro	2.16	3.20	59.67	4.14	222.30	4,142.0	287

Source: www.calibreming.com

Resource Estimates for La Luna and Cerro Aeropuerto detailed in Technical Report titled "NI 43-101 Technical Report and Resource Estimation of the Cerro Aeropuerto and La Luna Deposits, Borosi Concessions, Nicaragua", dated April 11, 2011. Gold Equivalent (AuEq) for La Luna and C. Aeropurto was calculated using $1058/oz Au for gold and $16.75/oz Ag for silver, and metallurgical recoveries and net smelter returns are assumed to be 100%.

Resource Estimates for Riscos de Oro detailed in Technical Report titled "NI 43-101 Technical Report and Resource Estimation of the Deposit, Borosi Concessions, R.A.A.N. Nicaragua", dated October 9, 2012. Gold Equivalent (AuEq) for Riscos de Oro was calculated using $1264/oz Au for gold and $19.78/oz Ag for silver, and metallurgical recoveries and net smelter returns are assumed to be 100%.

24 OTHER RELEVANT DATA AND INFORMATION

There is no additional relevant data available at that time of the report.

25 INTERPRETATION AND CONCLUSIONS

25.1 Mineral Resources

Joint Venture between RST (67%) and Calibre (33%) indicates that the mining tenure held by RST. in the Rosita D Concession is valid, and sufficient to support declaration of Mineral Resources.

The Rosita resource estimate in this Report include an update of the initial May 8, 2012 estimate for stockpiles and incorporates initial estimates on the tailings. This study is updated through 55 RC holes totalling 1,040 m of 2015 drilling for stockpiles and 87 auger holes totalling 460 m for tailings. It also incorporates a compilation and validation of 55 RC hole and 17 channel data on the stockpiles completed in 2011-2012.

The QP has evaluated drilling procedures, sample preparation, analyses and security and is of the opinion that the sampling procedures employed have provided sufficient geological information. The Author considers the data to be of good quality and satisfactory for use in the resource estimate. The independent sample verification results were compared versus the original assay results for copper, gold and silver and the results were reproducible.

The resource estimate is based on a gold price of US$1,200/oz., copper price of US$2.5/lb and silver price of US$16/oz. with process recoveries of 80% for gold, 35% for Cu (10% deducted for smelting) and 65% for silver. Mining costs were assumed at US$1.00/t, process costs at US$7.50/t and G&A costs at US$1.50/t. These values were equated against a cut-off grade of US$10 for stockpiles and 0.3 g/t Au for tailing mineral resources.

In opinion of the QP, the drilling, assaying and exploration works supporting this resource estimate are sufficient to indicate reasonable potential for economic extraction and thus qualify it as a Mineral Resource under CIM definition standards. The resulting resource estimate for the Rosita stockpiles at a NSR$10 cut-off includes: Indicated Resources of 6.46 million tonnes at a grade of 0.47 g/t Au, 0.50% Cu and 7.32 g/t Ag; and Inferred Resources of 3.44 million tonnes at a grade of 0.46 g/t Au, 0.61% Cu and 8.66 g/t Ag. Inferred resources for tailings at cut-off 0.3 g/t Au is 1.96 million tonnes at a grade of 0.56 g/t Au, 0.21% Cu and 9.65g/t Ag.

25.2 Mineral Processing

One (1) possible processing flowsheet for the Santa Rita Project is shown and detailed in this preliminary economic assessment. It is a combination of conventional crushing, milling, heap leach, carbon adsorption, and a SART recovery process. The flowsheet outlined has no wet tailings impoundment area required due to the agglomeration and blending of the dry oversize stockpile material. Overall, the material for processing from the stockpile and tailings has shown a high amenability for gold, silver, and copper extraction with the cyanidation process. The addition of the SART process is very important for the process economics and overall project viability. Overall recoveries from both the milling and heap leach streams based on the recent metallurgical testwork are expected to be high as indicated in the process design criteria.

Continued test work on both the stockpile and tailings material, will advance and confirm reagent addition rates and costs in all areas of the blended process. Possible piloting studies will determine and confirm flowsheet recoveries, mass balances, water balances, and a final processing flowsheet for project advancement to possible construction.

25.3 Infrastructure and Capital Costs

A proposed area for the plant and infrastructure has been identified that is in close proximity to the stockpile and tailing resource. Preliminary discussions with the town of Rosita in regard to this proposed location have taken place and all indications are positive. The area will utilize the existing civil works from the previous plant to save upfront construction direct costs and allow for convenient process flow. Process water and electrical power are available on and close to the Santa Rita project.

Specifics of this Santa Rita PEA are as follows:

- Preliminary stockpile reclamation has been developed to allow for initial high grade feeding to the process based on the stated mineral reserves.
- The Life-of-Mine for this study is limited to ten (10) years without the exhaustion of the stated reserves. Nominal overall throughput rate will be 365,000 Tpa (Years 1-3) and 720,000 Tpa (Years 4-10).
- The process will produce a loaded carbon for off-site extraction of gold and silver and a Cu_2S concentrate
- Pre-production capital requirements are \$ US 11.44 million with total capital over the ten (10) year life-of-mine of \$US 26.1 million
- The base case cash flow (all in \$US) has shown a Cumulative cash flow (after tax) of \$US 51.2 million, and IRR @ 7 % discounted rate of 51 % (pre-tax) and 41% (after tax).

The proposed recovery plant and treatment at the Santa Rita stockpile and tailings resource is feasible and provides a good economic return at the base metal prices.

25.4 Risks and Opportunities

Overall, the Santa Rita Project is considered to be of medium risk at the time of this report.

25.4.1 Project Risks

A list of potential risks is provided below:

- Metallurgical Performance – Metal Production – The preliminary proposed flowsheet has a number of metal production streams that effect overall metal production. The sensitivity analysis indicates the project is highly sensitive to metal production and any variations in the metal production streams will affect the project cash flow. Future planned testwork should strengthen the metallurgical design (i.e recovery and grade).

- Metal Prices – as with metal production, metal prices have a large effect in the project.

- Capital Costs – Equipment – Escalation of pricing on new equipment prior to purchase and prices

- Increased Loading and Hauling Costs – due to contract mining, costs are variable and based on local fuel costs. This will increase operating costs.

- The report for the proposed production scenario uses Indicated and Inferred Mineral Resources for the Stockpiles and Tailings. Mineral Resources do not have the same demonstrated economic viability as Mineral Reserves.

25.4.2 Project Opportunities

The two major potential project opportunities include the following:

- Metallurgical recoveries and Metal Prices. As stated previously, the project is sensitive to metal production and metal prices. At present, with the continuation toward higher metal prices (gold and copper), the project viability increases.

- Additional Acquisition Potential – The project is located in the “Nicaraguan Mining triangle”, both with on-going operations and previously operated operations. The ability to mill and process higher grade material (non Santa Rita resources) will increase the cash flow IRR and Net cash flow. It will also increase overall life-of-mine operating years.

25.5 Recommendations

RST is seriously considering advancing the stockpiles to near term production; hence, DENM recommends the following steps for the next program should aim at:

- Phase 3 Process Optimization Metallurgical Testwork – SGS Lakefield
- Additional Exploration Work – PFS standard of resources to allow increased tonnage in the indicated portion.
- Prefeasibility Study (PFS) – to confirm the proposed operating plan and capital expenditure budget. This work will be basis for recovery operations at Santa Rita and a continued path to commence operations in a practical time.
- Environmental studies (baseline) and subsequent permit application(s).
- Front End Engineering (FEED) – Mill and Heap

The costs of the recommended the further work are estimated in Table 25.1. A budget of approximately US$ 1,180,000 is required to complete the 2017-18 work on the Rosita project. This is a preliminary estimate for a firm or non-provisional program. Thorough program planning and cost estimations that will require tendered quotations from various contractors will need to be obtained before a final cost estimate can be made.

TABLE 25.1 RECOMMENDED PROGRAM AND BUDGET

Programs	Budget (US$)
Metallurgical Testwork	$150,000
Exploration and Resource Work	$250,000
Prefeasibility Study	$500,000
Environmental studies	$50,000
Geotechnical Site Report	$30,000
Front End Engineering (FEED)	$200,000
Total	$1,180,000

Source: DENM (2017)

26 REFERENCES

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SGS Canada, Inc., Lakefield, ON. An Investigation into the Recovery of Gold and Copper from Rosita Stockpile and Tailings Samples, Prepared for Rosita Mining Corporation Project 15278-002 – Final Report, March 31, 2017.

27 CERTIFICATES

CERTIFICATE OF QUALIFIED PERSONS

YUNGANG WU, P.GEO.

I, Yungang Wu, P. Geo., residing at 3246 Preserve Drive, Oakville, Ontario, L6M 0X3, do hereby certify that:

1. I am an independent consulting geologist;

2. This certificate applies to the technical Report titled "NI 43-101 Preliminary Economic Assessment Study for the Santa Rita Project, Rosita, Nicaragua" for Rosita Mining Corp. with an effective date of March 6, 2017;

3. I am a graduate of Jilin University, China, with a Master Degree in Mineral Deposits (1992). I am a geological consultant and a registered practising member of the Association of Professional Geoscientist of Ontario (Registration No. 1681).

4. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a "qualified person" for the purposes of NI 43-101;

My relevant experience for the purpose of the Technical Report is as follows:

- Geologist –Geology and Mineral Bureau, Liaoning Province, China…………………….......1992-1993
- Senior Geologist – Committee of Mineral Resources and Reserves of Liaoning, China.1993-1998
- VP – Institute of Mineral Resources and Land Planning, Liaoning, China……………......1998-2001
- Project Geologist–Exploration Division, De Beers Canada…………………………….......2003-2009
- Mine Geologist – Victor Diamond Mine, De Beers Canada…………………………….......2009-2011
- Resource Geologist– Coffey Mining Canada…………………………………………......2011-2012
- Consulting Geologist…………………………………………………………………….2012-Present

5. I have visited the property that is the subject of this Technical Report on November 6-7, 2015;

6. I am responsible for preparation of sections 5 through 12, 14 and 23, and co-authored for sections 2, 3, 4, 25, 26 and the summary of the Technical Report;

7. I am independent of the Issuer applying the test set out in Section 1.5 of National Instrument 43-101;

8. I have had prior involvement with the property that is the subject of the Technical Report. The nature of my involvement was as Author of a Technical Reports titled "NI 43-101 Technical Report on Mineral Resource Estimate of Rosita Stockpiles" prepared for Alder Resources Ltd., with an effective date of May 8, 2012; and "NI 43-101Technical Report on Mineral Resource Estimate of Rosita Stockpiles and Tailings, Rosita Cu-Au-Ag Project, Región Autónoma de la Costa Caribe Norte, Nicaragua" for Rosita Mining Corp. with an effective date of February 8, 2016;

9. I have read the National Instrument 43-101 and Form 43-101F1 and this Report has been prepared in compliance with National Instrument 43-101;

10. As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: March 6, 2017
Signing Date: this 20th day of April 2017

"Yungang Wu" (original signed and sealed)

Yungang Wu, P.Geo

David J. Salari

I, David J. Salari, P.Eng., of 59 West Street, Oakville, ON, L6L 2Y8, do hereby certify that:

1. This certificate applies to the Technical Report entitled “NI 43-101 Preliminary Economic Assessment Study for the Santa Rita Project, Rosita, Nicaragua” , with an effective date of March 9, 2017, prepared for Rosita Mining Corporation ;

2. I am a metallurgical engineer with an office at Suite 300-10, 1100 Burloak Drive, Burlington, ON, L6L 2Y8;

3. I am a graduate of the University of Toronto with a Bachelor's of Applied Science (BASc) – Metallurgy and Material Science;

4. I have been actively involved in mining and mineral processing since 1980 with extensive experience in metallurgical and mill testing and design, mill capital and operating costs, construction, commissioning, and mill operations;

5. I am a member in good standing of the Professional Engineers Ontario - #40416505 and I am the designated P.Eng. for D.E.N.M. Engineering Ltd. – Certificate of Authorization – Professional Engineers Ontario - #100102038 and Designation as a Consulting Engineer – Professional Engineers Ontario - # 4012;

6. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of NI 43-101;

7. I visited the Santa Rita project site on September 18-20, 2016;

8. I am responsible for preparation of sections 13, 15-22, 24 and co-authored for sections 2, 3, 4, 25, 26 and the summary of this report.

9. I have had no prior involvement with the property this is subject to this Technical Report;

10. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, this technical report contains all scientific and technical information that is required to be discussed to make the Technical Report not misleading;

11. I have read NI43-101, and the Technical Report has been prepared in accordance with NI 43-101 and Form 43-101F1.

Effective Date: March 6, 2017
Signing Date: this 20th day of April 2017

"David J. Salari" (original signed and sealed)

David J. Salari, P.Eng.