UNITED STATES
SECURITIES AND EXCHANGE COMMISSION
Washington, D.C. 20549

FORM 6-K

REPORT OF FOREIGN PRIVATE ISSUER PURSUANT TO RULE 13A-16 OR 15D-16 OF THE SECURITIES EXCHANGE ACT OF 1934
For the month of April 2018

Commission File Number: 001-31819

Gold Reserve Inc.

(Exact name of registrant as specified in its charter)
999 W. Riverside Avenue, Suite 401
Spokane, Washington 99201
(Address of principal executive office)

Indicate by check mark whether the registrant files or will file annual reports under cover Form 20-F or Form 40-F.

Form 20-F ☐ Form 40-F x

Indicate by check mark if the registrant is submitting the Form 6-K in paper as permitted by Regulation S-T Rule 101(b)(1): ☐

Indicate by check mark if the registrant is submitting the Form 6-K in paper as permitted by Regulation S-T Rule 101(b)(7): ☐

Indicate by check mark whether the registrant by furnishing the information contained in this Form is also thereby furnishing the information to the Commission pursuant to Rule 12g3-2(b) under the Securities Exchange Act of 1934.
Yes ☐ No x

If "Yes" is marked, indicate below the file number assigned to the registrant in connection with Rule 12g3-2(b):

This Report on Form 6-K and the exhibit attached hereto are hereby incorporated by reference into Gold Reserve Inc.'s (the "Company") current Registration Statements on Form F-3 on file with the U.S. Securities and Exchange Commission (the "SEC").

The following exhibits are furnished with this Form 6-K:

99.1	**NI 43-101 Technical Report**
99.2	**Certificate of Qualified Person - Lambert**
99.3	**Certificate of Qualified Person - Texidor**
99.4	**Certificate of Qualified Person - Miranda**
99.5	**Certificate of Qualified Person - Altman**
99.6	**Certificate of Qualified Person - Malensek**
99.7	**Consent of Qualified Person - Lambert**
99.8	**Consent of Qualified Person - Texidor**
99.9	**Consent of Qualified Person - Miranda**
99.10	**Consent of Qualified Person - Altman**
99.11	**Consent of Qualified Person - Malensek**

CAUTIONARY STATEMENT REGARDING FORWARD-LOOKING STATEMENTS AND INFORMATION

The information presented or incorporated by reference in this report contains both historical information and "forward-looking statements" (within the meaning of Section 27A of the Securities Act and Section 21E of the Exchange Act) or "forward-looking information" (within the meaning of applicable Canadian securities laws) (collectively referred to herein as "forward-looking statements") that may state our intentions, hopes, beliefs, expectations or predictions for the future.

Forward-looking statements are necessarily based upon a number of estimates and assumptions that, while considered reasonable by us at this time, are inherently subject to significant business, economic and competitive uncertainties and contingencies that may cause our actual financial results, performance or achievements to be materially different from those expressed or implied herein and many of which are outside our control.

Forward-looking statements involve risks and uncertainties, as well as assumptions, including those set out herein, that may never materialize, prove incorrect or materialize other than as currently contemplated which could cause our results to differ materially from those expressed or implied by such forward-looking statements. The words "believe," "anticipate," "expect," "intend," "estimate," "plan," "may," "could" and other similar expressions that are predictions of or indicate future events and future trends, which do not relate to historical matters, identify forward-looking statements. Any such forward-looking statements are not intended to provide any assurances as to future results.

Numerous factors could cause actual results to differ materially from those described in the forward-looking statements, including, without limitation:

- The risk that the conclusions of management and its qualified consultants contained in the most recent Preliminary Economic Assessment of the Siembra Minera Gold Copper Project (the "Project") in accordance with National Instrument 43-101 Standards of Disclosure for Mineral Projects may not be realized in the future.
- delay or failure by Venezuela to make payments or otherwise honor its commitments under the Settlement Agreement, including with respect to the sale of the Mining Data or the payment of the Award;
- the risk that Venezuela may not transfer the funds deposited to the trust account for the benefit of the Company at Banco de Desarrollo Económico y Social de Venezuela ("Bandes Bank") (the "Trust Account"), a Venezuelan state-owned development bank, to our U.S. or Canadian bank accounts;
- the risk of the imposition of further sanctions by the U.S., Canada or other jurisdictions that may negatively impact our ability to freely transfer funds held in the Trust Account or our ability to do business in Venezuela;
- the ability of the Company and Venezuela to (i) successfully overcome any legal, regulatory or technical obstacles to operate Siembra Minera and develop and later operate the Siembra Minera Project, (ii) obtain any remaining governmental approvals and (iii) obtain financing to fund the capital and initial operating costs of the Siembra Minera Project;

- risks associated with exploration, delineation of adequate resources and reserves, regulatory and permitting obstacles and other risks incident to the exploration, development and operation of mining properties in Venezuela and generally for mining projects including our ability to achieve revenue producing operations in the future;
- local risks associated with the concentration of our future operations and assets in Venezuela, including operational, security, legal, regulatory, political and economic risks;
- our ability to resume our efforts to enforce and collect the Award, including the associated costs of such enforcement and collection effort and the timing and success of that effort, if Venezuela fails to make payments to the Trust Account under the Settlement Agreement, it is terminated and further efforts to meet the commitments in the Settlement Agreement are abandoned;
- pending the receipt of payments to the Trust Account and transfer of such payments under the Settlement Agreement to our U.S. or Canadian bank accounts, our continued ability to service our obligations as they come due and access future additional funding, when required, for ongoing liquidity and capital resources, including as a result of payments of certain of those funds that must be made to our shareholders and holders of CVRs;
- potential shareholder dilution resulting from future financings;
- our prospects in general for the identification, exploration and development of additional mining projects;
- risks associated with the abilities and continued participation of key employees; and
- changes in U.S., Canadian and/or other tax laws to which we are subject.

See “Risk Factors” contained in our Annual Information Form and Annual Report on Form 40-F filed on www.sedar.com and www.sec.gov, respectively for additional risk factors that could cause results to differ materially from forward-looking statements.

Investors are cautioned not to put undue reliance on forward-looking statements, and investors should not infer that there has been no change in our affairs since the date of this report that would warrant any modification of any forward-looking statement made in this document, other documents periodically filed with the SEC or other securities regulators or presented on the Company’s website. Forward-looking statements speak only as of the date made. All subsequent written and oral forward-looking statements attributable to us or persons acting on our behalf are expressly qualified in their entirety by this notice. We disclaim any intent or obligation to update publicly or otherwise revise any forward-looking statements or the foregoing list of assumptions or factors, whether as a result of new information, future events or otherwise, subject to our disclosure obligations under applicable U.S. and Canadian securities regulations. Investors are urged to read the Company’s filings with U.S. and Canadian securities regulatory agencies, which can be viewed online at www.sec.gov and www.sedar.com, respectively.

SIGNATURE

Pursuant to the requirements of the Securities Exchange Act of 1934, the registrant has duly caused this report to be signed on its behalf by the undersigned, thereunto duly authorized.

Dated: April 6, 2018

GOLD RESERVE INC. (Registrant)

By: /s/ Robert A. McGuinness
Robert A. McGuinness, its Vice President of Finance,
Chief Financial Officer and its Principal Financial and Accounting Officer

Exhibit 99.1

GOLD RESERVE INC.

TECHNICAL REPORT ON THE SIEMBRA MINERA PROJECT, BOLIVAR STATE, VENEZUELA

NI 43-101 Report

Qualified Persons:

Richard J. Lambert, P.E., P.Eng. Hugo Miranda, C.P.

José Texidor Carlsson, P.Geo. Kathleen A. Altman, Ph.D., P.E. Grant A. Malensek, P.Eng.

March 16, 2018

Report Control Form

Document Title	Technical Report on the Siembra Minera Project, Bolivar State, Venezuela		
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Project Director Approval	Richard J. Lambert	(Signed)	

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1 SUMMARY

EXECUTIVE SUMMARY

Roscoe Postle Associates Inc. (RPA) was retained by Gold Reserve Inc.(GRI), and its wholly owned subsidiary GR Engineering Barbados, Inc. (GRE) to prepare an independent Technical Report on the Siembra Minera Project (the Project), located in Bolivar State, Venezuela. The operating company Empresa Mixta Ecosocialista Siembra Minera, S.A. (Siembra Minera), which holds the rights to the Siembra Minera Project, is a mixed capital company with 55% being owned by a Venezuelan state entity [owned by the Bolivarian Republic of Venezuela through the Corporación Venezolana de Minería (CVM)] and 45% by GR Mining Barbados, Inc. (GRM), a wholly-owned subsidiary of GRI. GRE has been set up to perform engineering, procurement, construction, and operation of the Project.

The Project is a combination of the Brisas and Cristinas properties into a single project now called the Siembra Minera Project. The purpose of this report is to provide GRI and GRE with an initial assessment of the Siembra Minera Project including a resource estimate, conceptual mine plan, and a preliminary economic review. This Technical Report conforms to NI 43-101 Standards of Disclosure for Mineral Projects. RPA visited the Project on September 19, 2017.

The Siembra Minera Project is a gold-copper deposit located in the Kilometre 88 mining district of Bolivar State in southeast Venezuela. Local owners and illegal miners have worked the property for many years. Shallow pitting and hydraulic methods were used to mine the upper saprolite zone, and coarse gold was recovered by gravity concentration and amalgamation with mercury. Most of the large-scale exploration work at Cristinas was performed by Placer Dome Inc. (Placer), which worked on the property from 1991 to 2001. At Brisas, GRI carried out the exploration program on the concession from 1992 to 2005. The most recent Technical Report for Cristinas is dated November 7, 2007, which is based on a feasibility study and includes historic mineral reserves. The most recent Technical Report for Brisas is dated March 31, 2008, which is also based on a feasibility study and includes historic mineral reserves.

RPA has relied on data derived from work completed by previous owners on the Cristinas concessions and by GRI on the Brisas concessions. The current resources for Cristinas were estimated by RPA based on the drill hole data supplied by Corporación Venezolana de

Guayana (CVG) to GRI in 2002. The database had 1,174 drill holes and 108 trenches which were included in the Cristinas database. Hard copies of the assay data sheets were not available, however, GEOLOG data files from Placer were provided including assay data, geological descriptions, structural data, geotechnical data, and check sample data. The current resources for Brisas were estimated by RPA based on drill hole data supplied by GRI in Geovia GEMS format which formed the basis of the last Technical Report by Pincock Allen & Holt (PAH) in 2008.

This report is considered by RPA to meet the requirements of a Preliminary Economic Assessment (PEA) as defined in Canadian NI 43-101 regulations. The mine plan and economic analysis contained in this Technical Report are based, in part, on Inferred Mineral Resources, and are preliminary in nature. Inferred Mineral Resources are considered too geologically speculative to have mining and economic considerations applied to them that would enable them to be categorized as Mineral Reserves. There is no certainty that economic forecasts on which this PEA is based will be realized.

CONCLUSIONS

RPA offers the following conclusions by area.

GEOLOGY AND MINERAL RESOURCES

- A number of exploration programs completed by Placer and GRI were successful in locating and defining the extents of the various mineralized zones on each of their respective property holdings. The recently established Siembra Minera Economic Zone has unified the land tenure.
- The geology of the deposit is well understood in general. RPA is of the opinion that the distribution of high grade areas in the Main Zone should be studied in more detail.
  In the southern two-thirds of the Cristinas concessions and the entirety of the Brisas concessions, the mineralization occurs in a large tabular body, which strikes approximately north-south and dips moderately to the west. In the northern third of the Cristinas concessions, the mineralization can occur as pipe-shaped forms, and as thinner tabular forms with sub-vertical dips and strikes to the southeast.
- The large tabular, strataform mineralized zone (referred to herein as the Main Zone) forms most of the Mineral Resource. The Main Zone has a minimum thickness of 10 m at the south end and reaches a maximum thickness of 350 m. The average thickness is approximately 200 m. While the southern limits of the Main Zone have been outlined by the existing drilling pattern with a reasonable degree of confidence, the down-dip limits have not been defined by drilling. The northern limits of the Main Zone are also reasonably well defined by the existing drilling pattern.

- The drill hole information collected by Placer and GRI was merged into one master database that was then used to prepare the Mineral Resource estimate. Additional drill hole information collected by Crystallex International Corporation (Crystallex) on the Cristinas concessions could not be used to prepare the current estimate of the Mineral Resources, as the detailed information required was not available. The drill hole data from Placer contained drilling information and analytical results up to 1997 while the drill hole data from GRI included information up to 2006.
- In RPA's opinion, the drill hole data is adequate for use in the preparation of Mineral Resource estimates.
- The outline of the gold mineralization was created by drawing wireframes using approximately a 0.20 g/t Au cut-off grade and the copper mineralization was outlined using broad wireframes based on approximately a 0.04% Cu cut-off grade. A total of 24 wireframes were constructed to represent the gold mineralization zones and six wireframes to represent the copper mineralization zones. RPA also prepared wireframe surfaces to represent the three main weathering profiles for the mineralized zones: oxide saprolite, sulphide saprolite, and hard rock.
- RPA applied variable capping values for gold and copper grades for each of the mineralized wireframe domains. The capped assay values were composited into three metre lengths. The composites were then used to estimate the gold and copper grades into a grade-block model that used block sizes of 10 m by 10 m by 6 m. Gold and copper grades were estimated into blocks using inverse distance squared and dynamic anisotropy with the Surpac v.6.8 software package. The estimated gold and copper grades were used to calculate Net Smelter Return (NSR) values for each mineralized block.
- Mineral Resources were prepared using an NSR cut-off value of US$7.20/t for the oxide saprolite and US$5.00/t for the sulphide saprolite and fresh rock. An open pit shell was created using the Whittle software package to constrain reporting of the Mineral Resources.
- The Mineral Resource estimate conforms to Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards for Mineral Resources and Mineral Reserves dated May 10, 2014 (CIM, 2014).
- The Mineral Resources are estimated at 10 million tonnes at an average grade of 1.02 g/t Au and 0.18% Cu containing 318,000 ounces of gold and 17,000 tonnes of copper in the Measured category, 1.17 billion tonnes at an average grade of 0.70 g/t Au and 0.10% Cu containing 26.5 million ounces of gold and 1.2 million tonnes of copper in the Indicated category. Mineral Resources in the Inferred category are estimated at 1.30 billion tonnes at an average grade of 0.61 g/t Au and 0.08% Cu containing 25.4 million ounces of gold and 1.0 million tonnes of copper.

MINING

- Mine production is scheduled to be carried out at a maximum mining rate ranging from 330 ktpd to 380 ktpd of total material.
- Stripping ratios are expected to average 1.16 over the Life of Mine (LoM) plan

- A separate equipment fleet of smaller excavators and articulated dump trucks is included in the mining capital for saprolite mining in the first 10 years. Typically, undisturbed saprolite material can be difficult to mine as the moisture creates operation problems. As the Project area has essentially been disturbed, RPA has assumed most saprolite is handled by the larger equipment fleet. The larger mine fleet is more productive and prior experience at Cristinas shows that rigid frame trucks can operate in the saprolite.
- Stockpiles are required for blending the process feed to achieve sufficient copper grades in flotation to produce a copper concentrate above 20%. Stockpiles fluctuate year to year, but achieve maximum capacity of just over 70 million tonnes.

MINERAL PROCESSING

- Both Brisas and Cristinas were developed to the feasibility-level stage and beyond in 2006 to 2007 so the quantity of information available is greater than would typically be available at the PEA stage of a project.
- The material to be mined from Siembra Minera is demonstrated to be amenable to both cyanide leaching and to sulphide flotation. For materials that contain lower concentrations of copper, cyanide leaching is more cost effective and for material that contains higher concentrations of copper, sulphide flotation is more cost effective.
- The prior metallurgical test work met industry standards at the time the studies were completed, however, technology has progressed in the subsequent ten plus years and industry standards have evolved. Current standards include testing of a large number of variability samples and development of geometallurgical models, as opposed to testing composite samples to represent “average” material to be processed, which was the emphasis for the Brisas test program.

ENVIRONMENT

- GRE is in the process of preparing environmental reports and programs to meet municipal, provincial, and national regulatory requirements, as well as generally accepted international standards.
- Two separate but parallel Environmental and Social Impact Assessments (ESIA) are being prepared for the Project, one that meets Venezuelan regulatory requirements and one that meets international standards and guidelines.
- A conceptual plan for small-scale mining management is in place. The conceptual plan includes relocation of the artisanal miners away from the active, large scale mining operations and establishment of an oxide saprolite processing and stockpile area with concrete tailings ponds that collect and transport tailings from the artisanal mining operations to the Project tailings management facility (TMF).

RECOMMENDATIONS

Given the positive economic results presented in this report, RPA recommends that the Project be advanced to the next stage of engineering study and permitting.

RPA offers the following recommendations.

GEOLOGY AND MINERAL RESOURCES

- Acquire new topographic data.
- Drill approximately 150 to 200 drill holes totalling approximately 75 km to 100 km. This drilling would have a number of objectives including:
  - Conversion of Inferred Mineral Resources to Indicated with priority set on Inferred Mineral Resources situated in the 5 and 10-year pit shells.
  - Drilling to determine the extent of mineralization at depth in the Main Zone as this will determine the limits of the largest possible pit and help with the location of features such as dumps and roads.
  - Better definition of the copper mineralization in the Main Zone footwall.
  - Improving preliminary artisanal mining sterilization assumptions.
  - Condemnation drilling of proposed waste rock storage sites.
  - Closer spaced drilling in the El Potaso area between Brisas and Cristinas concession areas.
  - Drilling on the northwest extensions of the mineralization in the Morrocoy and Cordova areas.
  - Drilling on the Cristinas Main Zone for density measurements.
- Improve understanding of the geological and structural controls on the shapes and local trends of high grade lenses in the Main Zone. Northwest striking cross-faults need to be identified and modelled and structural sub-domains built to improve future variography studies and dynamic anisotropy trend surfaces. This will improve the local accuracy of future gold and copper grade models.
- Carry out additional 3D mineralization trend analysis studies, domain modelling, and variography work for the gold and copper mineralization. This will also assist in evaluating if additional 5-spot drill holes are needed to support the Indicated classification in some areas with more complex geology.
- Depending on the outcome of new variography work, build gold and copper models using ordinary kriging.
- Develop a new lithology model once new drill holes have been drilled so that an improved material densities model can be created.
- Build a structural model.
- For the proposed drilling, implement field and coarse duplicate sampling programs at Siembra Minera at a rate of approximately 1 in 50.
- Acquire three or four matrix matched certified reference materials that approximate the cut-off grade, average grade, and high grades and insert them in all future drill programs at the Project at a rate of approximately 1 in 25.
- Implement external laboratory check assays at a rate of approximately 1 in 20.

MINING

- RPA is of the opinion that one of the most important factors influencing mining will be the amount of water entering the pit. RPA recommends contracting a groundwater hydrologist to evaluate the combined Project based on past work.
- A LoM schedule should be generated for the mining and processing of the Siembra Minera mineralized material. This study should include optimization and blending of the materials to achieve a sufficiently high copper grade to produce a copper concentrate grade above 20%.
- A trade-off study should be completed for the backfilling of the open pit with waste rock and/or neutralized tailings.
- A geotechnical investigation program should be carried out to confirm the subsurface conditions under the proposed new open pit, waste dump locations, and stability analysis undertaken to verify design recommendations.

MINERAL PROCESSING

- Every effort should be made to acquire access to the detailed metallurgical and plant data for Cristinas. In the absence of that data, detailed metallurgical sampling and testing are required to provide the information required to design the oxide leaching plant.
- Additional test work should be conducted for the flotation plant using variability samples taken from throughout the deposits with particular emphasis on Cristinas where limited variability testing was done using the flotation flowsheet. Currently, industry standard emphasizes the use of variability samples as opposed to the composite samples that were predominantly used in previous flotation testing.
- RPA is of the opinion that there is considerable potential for optimization of the flowsheet of the Siembra Minera Project. Examples include:
  - Increased efficiency if larger equipment sizes are utilized in the design. Due to cost savings and enhanced performance, the sizes for grinding mills and flotation cells have increased substantially. As examples, semi-autogenous grinding (SAG) mills that are now available are as large as 12.2 m diameter by 8.8 m long as opposed to the 11.6 m by 6.7 m that are in the current design and flotation cells now have capacities of 600 m^3 instead of the 160 m^3 that are in the current design. The larger pieces of equipment result in a reduced footprint and fewer pieces of equipment and, therefore, lower installed costs.
  - The use of an adsorption desorption recovery (ADR) that is designed for the combined Project will probably result in less cost than merely doubling the size of the current design. In addition to this, consolidating the ADR from the oxide leach plant into a plant that can later be expanded to process the doré from the flotation plant has the potential to not only cut costs but also reduce security concerns and efforts.
- RPA is of the opinion that the current conceptual design for the oxide leach plant does not include the best options for Siembra Minera. Areas that require detailed evaluations include:

- Use of carbon-in-leach (CIL) instead of carbon-in-pulp (CIP) particularly since the plant designs for both Cristinas and Brisas were changed to CIL from CIP during previous studies.
- Investigate elimination of the copper circuits. Data from the Cristinas feasibility study shows that copper is only soluble in the sulphide saprolite and that it is not soluble in material that has lower copper concentrations. Therefore, the copper circuit should not be needed as the sulphide saprolite that contains higher concentrations of copper will be processed in the flotation plant and not in the oxide leach plant.
- Changes to the gravity separation circuit. The use of continuous centrifugal concentrators instead of batch units to eliminate manual labour and reduce potential for theft. Use intensive cyanide leaching to process the gravity gold concentrate instead of shaking tables. Prior studies showed that intensive cyanide leaching was preferable for treatment of the gravity concentrate for both Brisas and Cristinas.
- Selection of designs that are appropriate for processing clay-like saprolitic material, including:
  - Appropriate tank sizing using slurry densities that are consistent with the material that has a low specific gravity and is viscous in nature
  - Proper agitator selection
  - Selection of pumps and design of piping

- Design of the TMF for the combined Project is preliminary. Further detailed geotechnical work is required to complete a design for the final tailings. Preliminary plans are to use the feasibility level design from the SNC-Lavalin 2007 study as Stage 1 of construction with the final tailings inundating the Stage 1 structure.

ENVIRONMENT

- GRI has held discussions with the small miners, indigenous groups, and local people. RPA recommends continuing discussions with these groups.
- Due to the increase in mineral resources, additional work is required for the increased waste rock dump (WRD) and TMF, and redesign/update of the acid rock drainage (ARD) mitigation measures.
- A new ESIA will be required for the combined project with an updated project plan and in conjunction with detail design and feasibility study.

COSTS AND ECONOMICS

- After the designs are complete for the Siembra Minera Project, a new capital and operating cost estimate should be completed.
- An updated copper concentrate marketing study should be completed. Recent changes in the world copper concentrate supply have reduced treatment and refining charges for copper and reduced participation charges.

PROPOSED PROGRAM AND BUDGET

RPA's proposed program for the next stage of study is summarized in Table 1-1.

TABLE 1-1 PROPOSED PROGRAM

GR Engineering (Barbados), Inc. – Siembra Minera Project

Description	Cost (US$ M)
Drilling to upgrade Inferred Mineral Resources – 150 to 200 holes	20
Geotechnical Studies	2
Hydrogeology Study	1
Metallurgical Studies	2
Pre-feasibility/Feasibility Study	5
ESIA and Permitting	2
Total	**32**

ECONOMIC ANALYSIS

The economic analysis contained in this report is based, in part, on Inferred Mineral Resources, and is preliminary in nature. Inferred Mineral Resources are considered too geologically speculative to have mining and economic considerations applied to them that would enable them to be categorized as Mineral Reserves. There is no certainty that economic forecasts on which this PEA is based will be realized.

A Cash Flow Projection has been generated from the LoM production schedule and capital and operating cost estimates, and is summarized in Table 1-4. All currency is in US dollars (US$ or $). A summary of the key criteria is provided below.

ECONOMIC CRITERIA

PRODUCTION

- The LoM production plan assumes that leach plant detailed engineering/early earthworks will commence in Q1 of Year -2.
- The LoM production plan assumes concentrator plant detailed engineering will commence in Q1 of Year -2.
- A 2-year pre-production period for the leach plant, 2 additional years for completion of the flotation concentrator, and a 45 year overall mine life.
- The leach plant has nameplate capacity of 15,000 tpd from year 1 through year 10, which increases in year 11 to 35,000 tpd through year 45 (End of Mine, or EoM) (5.8 Mtpa to 12.25 Mtpa, respectively).
- The concentrator plant has nameplate capacity of 140,000 tpd from year 3 through year 10, which decreases in year 11 to 105,000 tpd through year 45 EoM (58 Mtpa to 36.75 Mtpa, respectively).

- Total combined leach and concentrator production is 2.0 billion tonnes, at a grade of 0.70 g/t Au, 0.50 g/t Ag, and 0.090% Cu.
- The copper head grades in the mine plan are 302 Mt at 0.017% Cu and 1,703 Mt at 0.106% Cu for the leach and concentrator plants, respectively. However, the leach plant does not recover copper, thus the overall average copper head grade in the total mill feed is 2,005 Mt at 0.090% Cu.
- Average overall metal recovery of 84% Au, 53% Ag, and 84% Cu.
- Total recovered metal of 38.1 Moz Au, 17.1 Moz Ag, and 3.3 billion lb Cu.
- Average LoM annual recovered metal production of 847 koz Au, 380 koz Ag, and 78 million lb Cu.
- Average annual recovered metal production in Years 3 through 18 of 1,229 koz Au, 469 koz Ag, and 77 million lb Cu.
- Average annual recovered metal production in Years 19 through 45 EoM of 674 koz Au, 353 koz Ag, and 78 million lb Cu.

REVENUE

- Doré payable factors at refinery are 99.9% Au and 98% Ag.
- Copper concentrate average payable factors at smelter are 98% Au, 97% Ag, and 95.8% Cu.
- Payable metal sales for the Project total 37.6 Moz Au, 16.6 Moz Ag, and 3.2 billion lb Cu split as follows:
  - From Doré: 14.4 Moz Au and 4.1 Moz Ag.
  - From Concentrate: 23.2 Moz Au, 12.5 Moz Ag, and 3.2 billion lb Cu.
- Metal prices: US$1,300 per troy ounce Au; US$17 per troy ounce Ag and US$3.00 per pound Cu.
- NSR for doré includes transport and refining costs of $0.50 per ounce doré and $6 per ounce gold/$0.40 per ounce silver, respectively.
- NSR for copper concentrate includes:
  - Cost Insurance and Freight (CIF) charge of $103 per wet tonne concentrate (8% moisture content) consisting of:
    - Road Transport (350 km one way): $11/t
    - Port Charges (Puerto Ordaz) : $17/t
    - Ocean Transport (Europe): $75/t.
  - Smelter treatment charge of $95 per dry tonne concentrate.
  - Smelter refining charges of $0.095/lb Cu, $6/oz Au, and $0.40/oz Ag.
  - Copper price participation is not included.

COSTS

- Pre-production period to CIP plant First Production: 24 months (January Year -2 to December Year -1).
- Pre-production period to concentrator First Production: 48 months (January Year -2 to December Year 2).
- Project development capital totals $2.57 billion, including $459 million in contingency (22% of direct and indirect capital).
- Sustaining capital of $1.42 billion.
- Average unit operating costs in $/t milled over the mine life:

o Mine ($1.36/t mined):		2.89
o Process:		4.93
o G&A:		1.32
o Other Infrastructure:		0.14
o **Direct Operating Costs**		**9.29**
o Concentrate Freight		0.36
o Off-site Costs		0.54
o **Total**	**$**	**10.19**

ROYALTIES AND GOVERNMENT PAYMENTS

Royalties and other government payments total $5.6 billion, or $2.77/t milled, over the LoM as shown in Table 1-2.

TABLE 1-2 ROYALTIES AND GOVERNMENT PAYMENTS

GR Engineering (Barbados), Inc. – Siembra Minera Project

Item	US$ M	US$/t milled
NSR Royalty	3,262.8	1.63
Special Advantages Tax	1,710.0	0.85
Science, Technology and Innovation Contributions	588.1	0.29
Total	**5,560.9**	**2.77**

The Project will pay an annual NSR royalty to Venezuela on the sale of gold, copper, and silver and any other strategic minerals of 5% for the first ten years of commercial production and 6% thereafter.

The Project is subject to an additional 3% NSR annual royalty called Special Advantages Tax which is a national social welfare fund.

The Project is subject to a 1% gross revenue levy as part of the Science, Technology and Innovation Contributions fund (LOCTI).

Customs duties and Value Added Taxes (VAT) are assumed to be waived for the Project.

INCOME TAXES, WORKING CAPITAL, AND OTHER

Income taxes/contributions, upfront working capital, and reclamation/closure costs total $8.3 billion as shown in Table 1-3. Withholding taxes on corporate dividends and interest payments are not incorporated into the Project economic analysis.

TABLE 1-3 INCOME TAXES, WORKING CAPITAL, AND OTHER

GR Engineering (Barbados), Inc. – Siembra Minera Project

Item	US$ M
Anti-Drug Contributions	283.9
Sports Contributions	283.9
Corp. Income Taxes Paid	7,373.8
Upfront Working Capital (Yrs 1-4)	195.4
Reclamation and Closure	150.0
Salvage Value	0
Total	**8,286.9**

Anti-drug and Sport Contributions

These profit-based taxes are assessed at 1% of current year and previous year operating income, respectively. The annual operating margin is calculated by taking annual gross revenues and deducting all operating costs and depreciation/amortization allowances.

Corporate Income Tax

The Project economic analysis incorporates a sliding scale of tax rates applicable on income based on Project phases starting in Year 1 of commercial production as follows:

- Years 1 through 5: 14%
- Years 6 through 10: 19%
- Years 11 through 15: 24%
- Years 16 through 20: 29%
- Years 21+: 34%

Year 1 is the first year of gold production, after commissioning of the 15,000 tpd oxide plant.

Deductions from income for the purpose of estimating income subject to tax include the following items:

- Operating Expense

  Expensed operating costs are deducted 100% in year incurred.
- Stockpile adjustments

  As a result of large stockpiles of mill feed being generated during the life of the mine, the Project economic analysis includes annual adjustments to EBITDA to match mining costs with recognized revenue. The net effect of these adjustments over the life of the mine is zero but the adjustments increase EBITDA in years where stockpiling exceeds processing and inversely decrease EBITDA when processing stockpile material exceeds stockpile placement amounts.
- Depreciation/Amortization
  - All prior expenditures before January 2018 are considered sunk with respect to this analysis.
  - Depreciation commences once the facilities are placed into service and the mine and mill are operating.
  - Heavy mine fleet equipment capital is depreciated using 8-year straight line (SL) method. Light vehicle capital is depreciated using 5-year SL method.
  - All process and infrastructure capital are depreciated using the Units of Production (UoP) method.
  - Capitalized pre-production activities such as pre-stripping and water management are amortized the UoP method.
  - The Project economic analysis incorporates an accelerated depreciation methodology which combines the first 12 years of annual SL depreciation allowances with the standard UoP cost basis. The resulting combined UoP/SL basis is then re-calculated using the UoP method. After 12 years, the depreciation allowances come directly from each UoP or SL category.
  - Reclamation costs are amortized during the LoM by an annual accrual of $0.035/t mined ($150 million cost divided by 4.33 billion tonnes mined). This allowance is adjusted annually by periodic reclamation capital expenditures during the LoM.
- Other Deductions

  Other deductions from income for the purposes of estimating taxable income include management fees which amount to 5% of annual operating and capital costs. The annual management fees derived from operating costs are within the G&A opex category and thus expensed 100% in the year incurred while the annual fees derived from capital costs are amortized using the UoP method starting in the year they are incurred.
- Loss Carryforwards

  Income tax losses may be carried forward indefinitely but may not be used for prior tax years.

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Upfront Working Capital

A total of $195 million has been allocated for upfront working capital in Years 1 to 4. This amount covers year over year changes in accounts receivable and payable plus consumable inventory.

Reclamation/Closure Costs

The Project economic analysis has a $150 million LoM closure cost estimate.

Salvage

No salvage value was estimated as part of the Project economic analysis.

CASH FLOW ANALYSIS

The Project as currently designed has significant variations in the mining schedule, processing methods, and head grades over its planned 45-year life. These variations are shown in Figures 1-1 and 1-2 and the resulting impact on the pre-tax free cash flow profile is shown in Figure 1-3.

FIGURE 1-1

MINE VS. MILL PRODUCTION

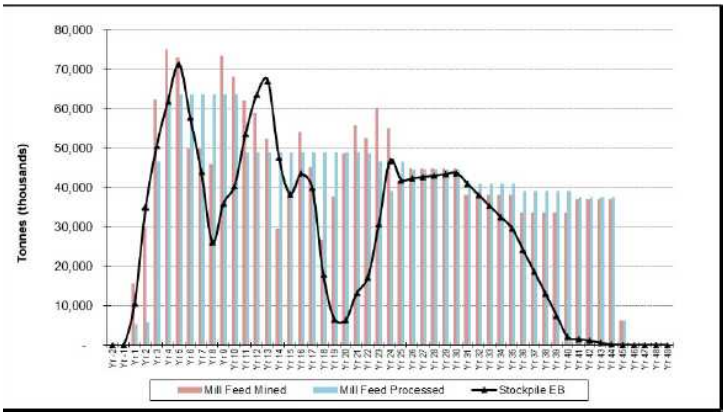


FIGURE 1-2 MILL PRODUCTION PROFILE BY PLANT

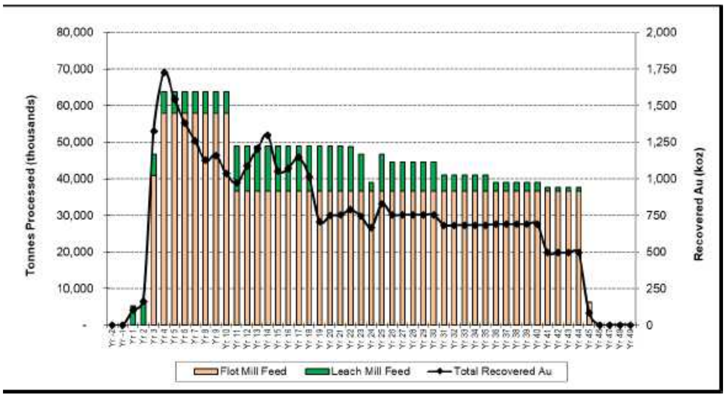


FIGURE 1-3 PROJECT PRE-TAX METRICS SUMMARY

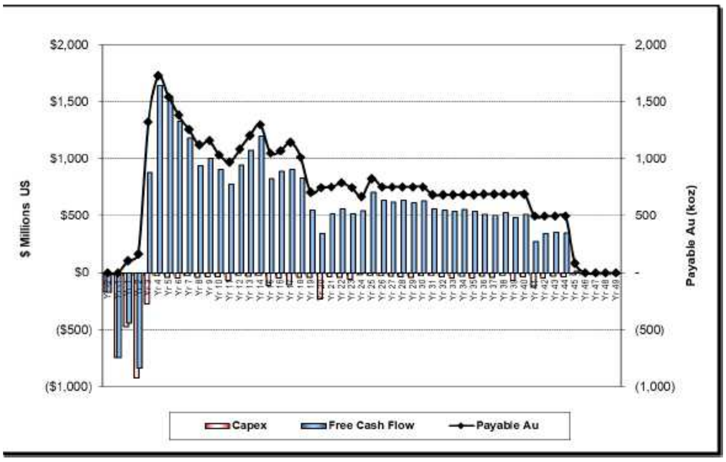


Table 1-4 shows the LoM total metrics for the Project as currently designed. Due to the length of the 45-year mine life, the full annual cash flow model is presented in Appendix 1.

TABLE 1-4 INDICATIVE PROJECT ECONOMICS

GR Engineering (Barbados), Inc. – Siembra Minera Project

Item	Unit	Value
Realized Market Prices		
Au	US$/oz	1,300
Ag	US$/oz	17.00
Cu	US$/lb	3.00
Payable Metal		
Au	Moz	37.6
Ag	Moz	16.6
Cu	Mlb	3,197.6
Total Gross Revenue	**US$ M**	**58,806.2**
Mining Cost	US$ M	(5,790.9)
Process Cost	US$ M	(9,881.0)
G & A Cost	US$ M	(2,653.6)
Other Infrastructure Cost	US$ M	(288.9)
Concentrate Freight Cost	US$ M	(728.0)
Off-site Costs	US$ M	(1,076.5)
NSR Royalty Cost	US$ M	(3,262.8)
Special Advantages Tax Cost	US$ M	(1,710.0)
Science (LOCTI) Contributions	US$ M	(588.1)
Total Operating Costs	**US$ M**	**(25,979.7)**
Operating Margin (EBITDA)	**US$ M**	**32,826.5**
Anti-Drug Contributions	US$ M	(283.9)
Sport Contributions	US$ M	(283.9)
Effective Tax Rate	%	22.5%
Income Tax	US$ M	(7,373.8)
Total Taxes	**US$ M**	**(7,941.5)**
Working Capital ($195 M in Years 1 to 4)	US$ M	0
Operating Cash Flow	**US$ M**	**24,885.0**
Development Capital	US$ M	(2,570.6)
Sustaining Capital	US$ M	(1,941.7)
Closure/Reclamation Capital	US$ M	(150.0)
Total Capital	US$ M	**(4,662.3)**
Pre-tax Free Cash Flow	US$ M	28,164.2
Pre-tax NPV @ 5%	US$ M	**11,209.4**
Pre-tax NPV @ 10%	**US$ M**	**5,534.5**
Pre-tax IRR	**%**	**36.8%**
After-tax Simple Payback	Years	3.8
After-tax Free Cash Flow	US$ M	20,222.7
After-tax NPV @ 5%	US$ M	**8,101.2**
After-tax NPV @ 10%	**US$ M**	**3,930.1**
After-tax IRR	**%**	**31.1%**
After-tax Simple Payback	Years	4.1

On a pre-tax basis, the undiscounted cash flow totals $28,164 million over the mine life. The pre-tax Internal Rate of Return (IRR) is 36.8%, and simple payback from start of commercial production occurs in 3.8 years. The pre-tax Net Present Values (NPV) are:

- $11,209 million at a 5% discount rate.
- $5,534 million at a 10% discount rate.

On an after-tax basis, the undiscounted cash flow totals $20,223 million over the mine life, the IRR is 31.1%, and simple payback from start of commercial production occurs in 4.1 years. The after-tax NPVs are:

- $8,101 million at a 5% discount rate.
- $3,930 million at a 10% discount rate.

The average annual gold sales during the forty-five years of operation is 836 koz per year (37.6 Moz over the LoM) at an average all in sustaining cost (AISC) of US$483 per ounce. Table 1-5 shows the AISC build up which is net of a US$262/oz copper and silver by-product credit (nbp).

TABLE 1-5 ALL-IN SUSTAINING COSTS COMPOSITION

GR Engineering (Barbados), Inc. – Siembra Minera Project

Item	US$M	US$/oz Au
Mining	5,790.9	154
Process	9,881.0	263
G & A	2,653.6	71
Other Infrastructure	288.9	8
Subtotal Site Costs	**18,614.3**	**495**
Transportation	728.0	19
Off-site Treatment	1,076.5	29
Subtotal Off-site Costs	**1,804.5**	**48**
Direct Cash Costs	**20,418.8**	**542**
Ag and Cu By-Product Credit	(9,875.4)	(262)
Total Direct Cash Costs (nbp)	**10,543.4**	**280**
NSR Royalty	3,262.8	87
Special Advantages Tax	1,710.0	45
STI Contributions	588.1	16
Total Indirect Cash Costs	**5,560.9**	**148**
Total Production Costs	**16,104.3**	**428**
Sustaining Capital Cost	1,941.7	52

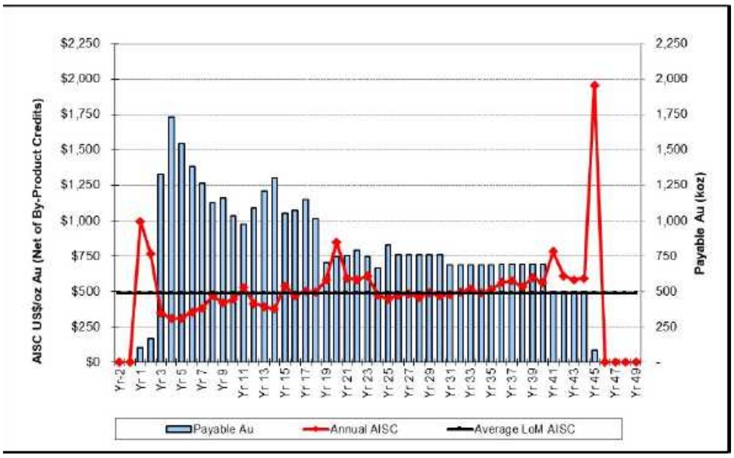
Item	US$M	US$/oz Au
Closure/Reclamation Capital	150.0	4
Corporate G&A	0.0	0
Off-mine Exploration	0.0	0
Total Sustaining Costs	**2,091.7**	**56**
Total All-in Sustaining Costs	**18,196.0**	**483**

Figure 1-4 shows the annual AISC trend during the mine operations against an overall average AISC of US$483/payable oz over the 45-year LoM at an annual production rate of 836 koz Au per year. The AISC variations are mainly driven changes in grades, mine schedule, and processing methods. The AISC metric can range from US$309/oz to US$992/oz Au in a given year (excluding final year spike in Year 45 of $1,956/oz) but can be subdivided into three distinct phases:

- Phase 1: Years 1 and 2 (CIP only) - 133 koz/yr Au at $853/oz.
- Phase 2: Years 3 through 18 (mining highest grades) - 1,191 koz/yr Au at $411/oz.
- Phase 3: Years 19 through 45 EoM (mining lower grades) - 665 koz/yr Au at $554/oz.

FIGURE 1-4

ANNUAL AISC CURVE PROFILE



SENSITIVITY ANALYSIS

Project risks can be identified in both economic and non-economic terms. Key economic risks were examined by running cash flow sensitivities:

- Head grade
- Gold recovery
- Gold price
- Operating costs
- Capital costs
- Discount rates

Pre-tax NPV and IRR sensitivities over the base case has been calculated for -20% to +20% variations metal-related categories. For operating costs and capital costs, the sensitivities over the base case has been calculated at -15% to +35% variation. The sensitivities are shown in Table 1-6 and in Figures 1-5 and 1-6, respectively.

TABLE 1-6 PRE-TAX SENSITIVITY ANALYSIS

GR Engineering (Barbados), Inc. – Siembra Minera Project

Factor Change	Head Grade (g/t Au)	NPV at 10% (US$ M)	IRR (%)
0.8	0.56	3,477.3	28.3%
0.9	0.63	4,505.8	32.7%
1	**0.70**	**5,534.5**	**36.8%**
1.1	0.78	6,563.2	40.6%
1.2	0.85	7,591.9	44.3%

Factor Change	Recovery (% Au)	NPV at 10% (US$ M)	IRR (%)
0.8	67	3,477.3	28.3%
0.9	76	4,505.8	32.7%
1	**84**	**5,534.5**	**36.8%**
1.1	92	6,563.2	40.6%
1.2	100	7,489.0	44.0%

Factor Change	Metal Price (US$/oz Au)	NPV at 10% (US$ M)	IRR (%)
0.8	1,040	3,166.4	27.2%
0.9	1,170	4,350.4	32.2%
1	**1,300**	5,534.5	**36.8%**
1.1	1,430	6,718.5	41.1%
1.2	1,560	7,902.5	45.1%

Factor Change		Operating Costs (US$/t milled)	NPV at 10% (US$ M)	IRR (%)
0.85	$	11.57	6,068.2	38.6%
0.93	$	12.27	5,801.3	37.7%
1.00	**$**	**12.96**	5,534.5	**36.8%**
1.18	$	14.59	4,911.7	34.6%
1.35	$	16.21	4,289.0	32.3%

Factor Change		Capital Costs (US$ M)	NPV at 10% (US$ M)	IRR (%)
0.85	$	4,222	5,812.0	41.1%
0.93	$	4,385	5,673.2	38.8%
1.00	**$**	**4,547**	5,534.5	**36.8%**
1.18	$	4,927	5,210.7	32.7%
1.35	$	5,306	4,886.9	29.3%

FIGURE 1-5 PRE-TAX NPV 10% SENSITIVITY ANALYSIS

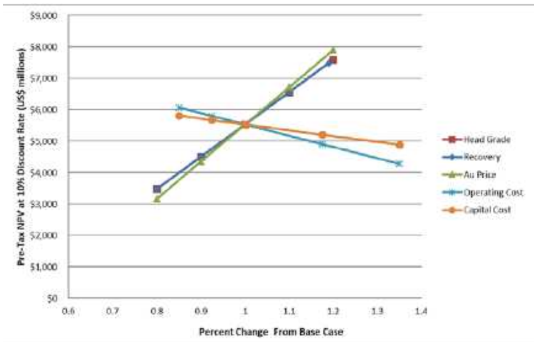
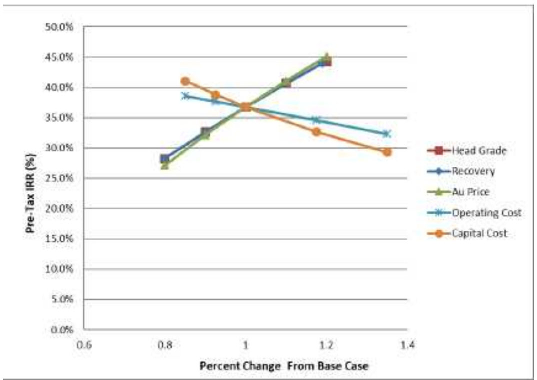


FIGURE 1-6 PRE-TAX IRR SENSITIVITY ANALYSIS



A sensitivity analysis of discount rates is presented in Figure 1-7 and 1-8 and shows that the Project as currently designed would be NPV positive through a 20% discount rate.

FIGURE 1-7 PRE-TAX DISCOUNT RATE SENSITIVITY ANALYSIS

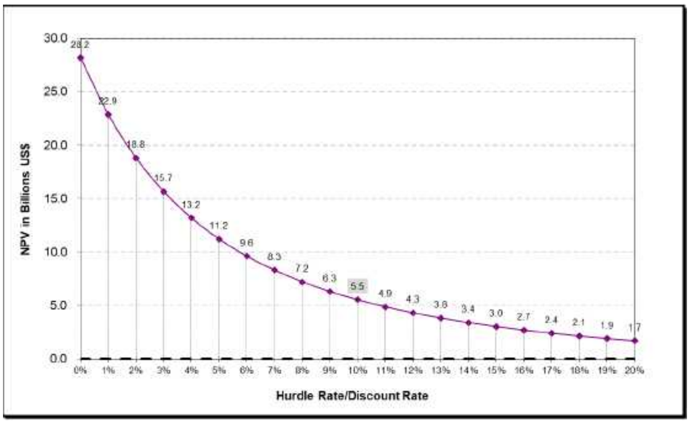
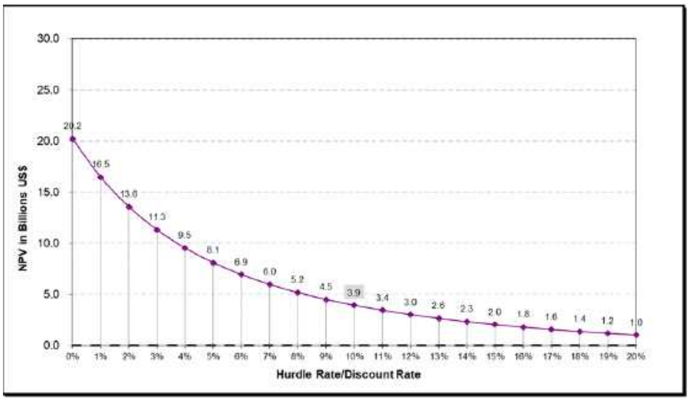


FIGURE 1-8 AFTER-TAX DISCOUNT RATE SENSITIVITY ANALYSIS



TECHNICAL SUMMARY

PROPERTY DESCRIPTION AND LOCATION

The Siembra Minera Project is located in the Kilometre 88 mining district of Bolivar State, in southeast Venezuela at Latitude 6° 11’ North and Longitude 61° 28’ West. The property is approximately 3.5 km west of Highway 10. Las Claritas is the closest town to the property.

The Project site is located in the Guyana region, which covers approximately one-third of Venezuela’s national territory. The closest nearby large city is Ciudad Guayana, with approximately 1,050,000 inhabitants (2001), situated on the Orinoco River near its confluence with the Caroní River. Ciudad Guayana consists of the old town of San Félix to the east and the new town of Puerto Ordaz to the west. Puerto Ordaz is home to most of the major industrial facilities such as aluminum smelters and port facilities. Puerto Ordaz has major port facilities accessible to ocean-going vessels from the Atlantic Ocean via the Orinoco River, a distance of approximately 200 km. There is regularly scheduled commercial airline service to Puerto Ordaz from Caracas.

Highway 10 provides paved access from Ciudad Guayana, which is 373 km northwest of the property, to within 3.5 km of the Project site. Unpaved roads provide the remaining 3.5 km of access.

The Project area encompasses approximately 18,951 ha and has been designated as an Economic Zone by the Venezuelan Government.

HISTORY

Gold in the Siembra Minera region was first discovered in 1920. Gold mining in the Project area was initiated in the 1930s and continued sporadically on a minor scale until the early 1980s when a gold rush occurred. Approximately 5,000 to 7,000 small miners worked alluvial and saprolite-hosted gold deposits using hydraulic mining techniques. The amount of gold recovered is unknown and much of the area of the concessions is now covered with tailings.

Placer conducted essentially all of the modern exploration on Cristinas during its tenure on the property from 1991 to 2001. Placer completed line cutting, mapping, rock and soil sampling, geophysics, and drilling of 1,174 drill holes for a total of 158,738 m of drilling. In 2003, Crystallex undertook drilling of 12 holes totalling 2,199 m to confirm the tenor of mineralization presented in the pre-existing database and also assayed check samples. Between 2003 and 2007, Crystallex released at least two feasibility studies and several resource and reserve estimates for Cristinas, all of which are historic in nature and should not be relied upon.

The Brisas concession was acquired by GRI in August 1992 with the acquisition of Compañia Aurifera Brisas del Cuyuni C.A. A large stratabound gold-copper mineralization was discovered in both alluvial and hard rock material by a drilling program in 1993. A majority of the exploration and development drilling took place in 1996 and 1997, with additional drilling completed in 1999, 2003, 2004, and 2005. As of 2005, 802 exploration holes had been drilled including 186,094 m of core drilling and 189,985 m of exploration core and auger drilling. In 2005-2006, an additional 76 holes were drilled on the Brisas concessions for geotechnical and other studies. A number of resource estimates have been completed for the Brisas deposit, all of which are superseded by the current Mineral Resource estimate in this report. A pre-feasibility study was carried out in 1998 and a feasibility study in 2005, with a feasibility update in 2008, all including historic reserve estimates.

GEOLOGY

The Siembra Minera Project is within the Guyana Shield in northern South America. The shield covers easternmost Colombia, southeastern Venezuela, Guyana, Suriname, French Guiana, and northeastern Brazil. The Venezuelan portion of the shield is subdivided into five geological provinces with different petrological, structural and metallogenic characteristics. The provinces are, from oldest to youngest, Imataca, Pastora, Cuchivero, Roraima, and Parguaza. Only the Imataca, Pastora and Roraima provinces are found in the vicinity of the Siembra Minera deposit.

The Siembra Minera deposit lies within a portion of the lower Caballape Formation volcanic and volcanic-related sedimentary rocks. The units present are (1) andesitic to rhyolitic tuffaceous volcanic beds, (2) related sedimentary beds, and (3) a tonalitic intrusive body. All rocks have been tilted and subjected to lower greenschist facies metamorphism. It is thought, based on information from nearby properties, that the Siembra Minera Project occupies one limb of a large regional fold. Limited direction-indicating structures show the strata to be top-up. In the main mineralized trend, moderate to strong foliation is oriented N10°E and dipping 30° to 55° northwest. This foliation appears to be parallel to the original bedding and tends to be strongest in the finer-grained rocks. A much weaker foliation orientation appears in outcrop exposures, striking north-northwest and dipping to the southwest.

There are four distinct types of gold and copper mineralization present at Brisas, defined by geometry, associated minerals, and the gold-copper ratio. These zones are the Blue Whale body, disseminated gold+pyrite (± copper), disseminated high copper, and shear-hosted gold. Only the first three types are encountered within the proposed pit geometry.

Two distinct styles of mineralization are present at Cristinas: hydrothermal breccia-hosted mineralization at Mesones-Sofia and stratiform mineralization at Conductora, Morrocoy, and Cordova. The vast majority (approximately 95%) of the gold at Cristinas is contained in the stratiform mineralized zone.

EXPLORATION STATUS

Drilling at Brisas was carried out by GRI from late 1992 to 2006 and consisted of 975 drill holes totalling approximately 207,000 m. In addition, four trenches were dug for a total of 60 m. At Cristinas, drilling was carried out by Placer from 1992 to 1997, consisting of 1,182 drill holes totalling approximately 155,000 m, and by Crystallex from 2003 to 2007, consisting of 90 holes totalling approximately 28,000 m. The Crystallex drill hole data was not available for RPA's resource modelling work.

The Siembra Minera mineralization is open down dip in all zones and along strike to the northwest in Morrocoy and Cordova because of insufficient drilling. Current plans for exploration are based on brownfield expansion of the existing deposit. As the Project advances, GRE proposes to carry out approximately 75,000 m to 100,000 m of new drilling.

MINERAL RESOURCE ESTIMATES

A Mineral Resource estimate, dated December 31, 2017, was completed by RPA using the Surpac and Leapfrog Geo software packages. Wireframes for geology and mineralization were constructed in Leapfrog Geo based on geology sections, assay results, lithological information, and structural data. Assays were capped to various levels based on exploratory data analysis and then composited to three metre lengths. Wireframes were filled with blocks measuring 10 m by 10 m by 6 m (length, width, height). Block grades were estimated using dynamic anisotropy and inverse distance squared algorithms. Block estimates were validated using industry standard validation techniques. Classification of blocks was based on drill hole spacing distances and other criteria.

A summary of the Mineral Resources at the Project is provided in Table 1-7.

TABLE 1-7 SUMMARY OF MINERAL RESOURCES – DECEMBER 31, 2017
GR Engineering (Barbados), Inc. – Siembra Minera Project

Category	Tonnes (Mt)	Grade (g/t Au)	Grade (% Cu)	Contained Gold (koz Au)	Contained Copper (kt Cu)	Contained Copper (Mlb Cu)
Measured	10	1.02	0.18	318	17	38
Indicated	1,174	0.70	0.10	26,504	1,202	2,649
Total Measured + Indicated	**1,184**	**0.70**	**0.10**	**26,823**	**1,219**	**2,687**
Inferred	1,291	0.61	0.08	25,389	1,044	2,300

Notes:

1. CIM (2014) definitions were followed for Mineral Resources.
2. Mineral Resources are estimated at an NSR cut-off value of US$7.20 per tonne for oxide-saprolite material and US$5.00 per tonne for sulphide-saprolite and fresh rock material.
3. Mineral Resources are constrained by a preliminary pit shell created using the Whittle software package.
4. Mineral Resources are estimated using a long-term gold price of US$1,300 per ounce, and a copper price of US$3.00 per pound.
5. Bulk density varies by material type.
6. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
7. Numbers may not add due to rounding.

RPA is not aware of any environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant factors that could materially affect the Mineral Resource estimate.

MINING

The Siembra Minera Project is an open pit gold-copper mining project that will utilize 30 m^3 hydraulic shovels and 236-tonne trucks as the primary mining equipment.

The resource pit optimization was developed by RPA based on the RPA Mineral Resource estimate (Table 1-7). Blocks classified as Measured, Indicated, and Inferred Mineral Resources were included in the resource pit optimization process for the Siembra Minera deposit. The resource pit is approximately 6,000 m long and 1,900 m wide with a maximum depth of approximately 700 m. The pit slope on the east wall follows the mineralization with slopes from 36° to 38°, while the west wall final pit has overall pit slopes ranging from 48° to 50°.

Mine production is scheduled to be carried out at a maximum mining rate ranging from 330 ktpd to 380 ktpd of total material. Stripping ratios are expected to average 1.16 over the LoM plan. The production schedule was produced using Whittle software to guide the mining sequence, Vulcan to design phases, waste dumps and the final pit, and XPAC to schedule the phases following the processing requirements.

During the first ten years of the Project, 5.8 million tonnes per annum (Mtpa) of oxide saprolite that does not require grinding will be processed in the oxide saprolite plant. The flotation plant starts two years after the oxide plant. Feed to the flotation mill is scheduled for 58.0 Mtpa for years 3 to10, while softer high copper sulphide saprolite material is available. In year 11, one quarter of the flotation grinding mill (12.25 Mtpa) is converted to oxide to accommodate the harder low-copper sulphide saprolite and low-copper hard rock materials. The other 36.75 Mtpa of capacity in the grinding mill are used for the harder higher-copper material in the flotation. The oxide plant will start processing with a combination of saprolite and low copper hard rock using the leach tanks from the oxide saprolite plant and additional leach tanks required for processing. The hard rock and sulphide saprolite was divided into high copper and low copper using a 0.02% Cu threshold.

In order to supply the processing input required in the first 10 years of production, the total material mined must achieve up to 120 Mtpa from a combination of the mining phases. The mining rate will change depending on stockpile size, increasing total mining rate to 140 Mtpa in year 20.

Total resources potentially mineable by open pit are estimated at approximately 2.0 billion tonnes of mineralized material at a gold grade of 0.705 g/t and a copper grade of 0.1% with 2.3 billion tonnes of waste for a stripping ratio of 1.16 tonnes of waste per tonne of mineralized material.

All of the waste rock, except that used for TMF construction, will be disposed of in the WRD facilities located to the north, west, and south of the pit. It appears that a portion of the Siembra Minera pit could be backfilled with waste rock, however, further investigation into tailings disposal and pit backfill opportunities are recommended.

MINERAL PROCESSING AND METALLURGICAL TESTING

The Siembra Minera Project consists of three rock types. Hard rock ore comprises approximately 87% of the material that will be processed. The remaining 13% of the mineralized material is saprolite with a split composed of approximately 43% oxide saprolite and 57% sulphide saprolite. Metallurgical test work was conducted on hard rock that contains higher and lower copper concentrations, and on blends that simulate the blends projected for the plant operation.

Based on the results of metallurgical testing, the conceptual processes selected for the combined project include a cyanide leach plant to process oxide saprolite and sulphide saprolite that contains low concentrations of copper to recover gold as doré from gravity concentration and cyanide leaching plus a flotation concentrator to process sulphide saprolite and hard rock that contain higher concentrations of copper. The flotation concentrator will recover copper and gold into a copper flotation concentrate and gold as doré utilizing gravity concentration and cyanide leaching of cleaner scavenger tailings.

The production schedule for this PEA is based on initially processing oxide saprolite through a 15,000 tpd cyanide leach plant. The crushing and screening plant feed is designed to process approximately 10% higher assuming that some of the material will be rejected due to oversize and/or rock material. Starting in year 7, the majority of the oxide saprolite is depleted and sulphide saprolite that contains low concentrations of copper will also be fed to the plant. In years 9 and 10, only low copper sulphide saprolite will be fed to the oxide plant.

In year 4, the flotation concentrator will be commissioned. The feed to the plant includes sulphide saprolite that contains higher concentration of copper and a combination of high and low copper hard rock material at a nominal rate of 140,000 tpd although the actual feed rate is higher in the early years due to the presence of sulphide saprolite which is easier to grind.

In year 11, the quantity of hard rock with suitable copper grades to produce acceptable concentrate grades in the flotation plant diminishes so the plant will be re-configured to process less material through the flotation plant and additional material through the oxide leach plant. The conceptual plan, at this early stage of the Project development, is to reduce the feed to the flotation concentrator to approximately 105,000 tpd and increase the tonnage to the oxide leach plant to 35,000 tpd. The low copper hard rock material will be ground in the existing milling circuit in the flotation plant and the leach plant will be expanded to accommodate the higher tonnage of material. The ball mill in the oxide leach plant, which is only sized to process saprolite, can be decommissioned or used to grind saprolite that is pumped from the open pit mine to the oxide leach plant.

ENVIRONMENT

Two separate, but parallel ESIA are being prepared for the Project. One ESIA is intended to meet Venezuelan regulatory requirements and the second one, international standards and guidelines. The Venezuelan ESIA is expected to be completed and submitted to the Ministry of People's Power for Ecosocialism and Water (MINEA) in 2018; and the International ESIA will be completed soon thereafter.

Prior to submission of the ESIA, an Authorization to Occupy the Territory (AOT) must be obtained and a Term of Reference (TDR) approved. The AOT certifies that the proposed use of the land by the Project is compatible with the land use designation of the area and the TDR defines the scope and contents of the ESIA. Both AOT and TDR must be submitted to MINEA. GRE has submitted the application for an AOT, and the TDR for the Project will be submitted as soon as the AOT is approved. Upon the approval of the TDR, GRE will prepare and submit the ESIA to MINEA. An application for the Authorization to Affect Natural Resources (AANR), a permit for exploitation, will be submitted as soon as the Project ESIA is approved, which is expected to be in 2018.

In addition to the ESIAs, GRE is in the process of developing a series of environmental and social management plans and programs. Thousands of small-scale miners are actively working in the Project area and adequate management of small-scale mining is critical to the success of the Project. A conceptual plan for small-scale mining management has been developed by GRE to relocate these miners to the Oro concession area.

Based on the current Project design, reclamation activities will commence soon after construction begins, and will continue throughout the life of the Project. Closure activities will continue for three years after the end of the mine life in year 27. Some intermittent reclamation would also take place before year 23, when areas are no longer needed for mine operation activities. Total expenditures for reclamation and closure are currently estimated to be approximately US$150 million.

CAPITAL COST ESTIMATE

A summary of capital costs is shown in Table 1-8.

TABLE 1-8 CAPITAL COST SUMMARY

GR Engineering (Barbados), Inc. – Siembra Minera Project

Description	Development	Sustaining	LoM Total
Mineral Reserve Definition	0.0	100.0	100.0
Mining	436.6	1,212.6	1,649.2
Processing - CIP	97.0	0.0	97.0
Processing - Concentrator	696.8	11.0	707.8
Processing - Tailings Dam	54.9	322.5	377.4
Processing - Port/Diversion/Vehicles	74.8	34.2	109.0
Processing - CIP Plant Conversion to 35 ktpd	0.0	35.0	35.0
Engineering & Geology	15.9	30.1	46.0
ARD Plant	2.3	0.0	2.3
Site Infrastructure	111.8	9.5	121.3
Subtotal Direct Cost	**1,490.1**	**1,754.9**	**3,245.0**
Indirects - CIP	34.3	0.0	34.3
Indirects - Concentrator	278.1	0.0	278.1
Indirects - Owner's Cost	310.4	150.6	461.0
Total Cost Before Contingency	**2,112.8**	**1,905.5**	**4,018.3**
Contingency - Mining	30.0	0.0	30.0
Contingency - CIP	26.3	0.0	26.3
Contingency - Concentrator	238.6	0.0	238.6
Contingency - TMF	16.5	0.0	16.5
Contingency - Port/Diversion/Vehicles	18.2	0.0	18.2
Contingency - Infrastructure	35.2	0.0	35.2
Contingency - Owner's Cost	93.1	36.2	129.3
Total Contingency	**457.8**	**36.2**	**494.0**
% Contingency	**21.7%**	**1.9%**	**12.3%**
Total Capital Cost	**2,570.6**	**1,941.7**	**4,512.3**
Reclamation/Closure Cost	0.0	150.0	150.0
Total Capital Cost excl. Working Capital	**2,570.6**	**2,091.7**	**4,662.3**
Working Capital[1]	195.4	0.0	195.4
Total LoM Capital Cost	**2,766.0**	**2,091.7**	**4,857.7**

Notes:

1. Upfront working capital of $195 million during Yrs 1 to 4. Recaptured at end of mine life.

OPERATING COST ESTIMATE

The Siembra Minera Project will process approximately 2,005 million tonnes of mineralized material over its planned 45-year mine life. The estimated average operating costs for the Project life are shown in Table 1-9.

TABLE 1-9 ESTIMATED LOM OPERATING COSTS

GR Engineering (Barbados), Inc. – Siembra Minera Project

Area	US$/t Milled
Mining (US$1.36/t mined)	2.89
Process	4.93
G&A	1.32
Other Infrastructure	0.14
Transportation	0.36
Off-site Treatment	0.54
Subtotal Operating Costs Before Royalties	**10.19**
Royalties/Production Taxes	2.77
Total	**12.96**

Operating costs for this Project appear to be low, however, the diesel fuel price of $0.02/L, the electricity cost of $0.038/kWh ($38/MWh), and the low labour costs have a significant impact on the unit operating costs.

2 INTRODUCTION

Roscoe Postle Associates Inc. (RPA) was retained by Gold Reserve Inc.(GRI), and its wholly owned subsidiary GR Engineering Barbados, Inc. (GRE) to prepare an independent Technical Report on the Siembra Minera Project (the Project), located in Bolivar State, Venezuela. The operating company Empresa Mixta Ecosocialista Siembra Minera, S.A. (Siembra Minera), which holds the rights to the Siembra Minera Project, is a mixed capital company with 55% being owned by a Venezuelan state entity [owned by the Bolivarian Republic of Venezuela through the Corporación Venezolana de Minería (CVM)] and 45% by GR Mining Barbados, Inc. (GRM), a wholly-owned subsidiary of GRI. GRE has been set up to perform engineering, procurement, construction, and operation of the Project.

The Project is a combination of the Brisas and Cristinas properties into a single project. The purpose of this report is to provide GRI and GRE with an initial assessment of the Siembra Minera Project including a resource estimate, conceptual mine plan, and a preliminary economic review. This Technical Report conforms to NI 43-101 Standards of Disclosure for Mineral Projects.

The Siembra Minera Project is a gold-copper deposit located in the Kilometer 88 mining district of Bolivar State in southeast Venezuela. Local owners and illegal miners have worked the property for many years. Shallow pitting and hydraulic methods were used to mine the upper saprolite zone, and coarse gold was recovered by gravity concentration and amalgamation with mercury. Most of the large-scale exploration work at Cristinas was performed by Placer Dome Inc. (Placer), which worked on the property from 1991 to 2001. At Brisas, GRI carried out the exploration program on the concession from 1992 to 2005. The most recent Technical Report for Cristinas is dated November 7, 2007, which is based on a feasibility study and includes historic mineral reserves. The most recent Technical Report for Brisas is dated March 31, 2008, which is also based on a feasibility study and includes historic mineral reserves.

RPA has relied on data derived from work completed by previous owners on the Cristinas concessions and by GRI on the Brisas concessions. The current resources for Cristinas were estimated by RPA based on the drill hole data supplied by Corporación Venezolana de Guayana (CVG) to GRI in 2002. The database had 1,174 drill holes and 108 trenches which were included in the Cristinas database. Hard copies of the assay data sheets were not available but GEOLOG data files from Placer were provided including assay data, geological descriptions, structural data, geotechnical data, and check sample data. The current resources for Brisas were estimated by RPA based on drill hole data supplied by GRI in Geovia GEMS format which formed the basis of the last Technical Report by Pincock Allen & Holt (PAH) in 2008.

This report is considered by RPA to meet the requirements of a Preliminary Economic Assessment (PEA) as defined in Canadian NI 43-101 regulations. The mine plan and economic analysis contained in this Technical Report are based, in part, on Inferred Mineral Resources, and are preliminary in nature. Inferred Mineral Resources are considered too geologically speculative to have mining and economic considerations applied to them that would enable them to be categorized as Mineral Reserves. There is no certainty that economic forecasts on which this PEA is based will be realized.

SOURCES OF INFORMATION

GRI provided access to their dataroom which included all the previous studies and mineral resource models. Discussions were held with personnel from GRI including:

- Mr. Doug Stewart, Project Manager
- Mr. Brad Yonaka, Chief Geologist

For this Technical Report, overall management was carried out by Richard J. Lambert, P.Eng., RPA Principal Mining Engineer. José Texidor Carlsson, P.Geo., RPA Senior Geologist, developed the mineral resource model under the supervision of Luke Evans, M.Sc., P.Eng., Principal Geologist. Hugo M. Miranda, P.C., RPA Principal Mine Engineer developed the pit optimization and production schedule. Kathleen A. Altman, Ph.D., P.E., RPA Principal Metallurgist, reviewed the metallurgical test work and process design. Grant A. Malensek, P. Geo., P. Eng., RPA Principal Valuation Engineer, was responsible for the Project economics. The site was visited by Mr. Miranda on September 19, 2017 and was previously visited by Mr. Lambert in February 2008.

Mr. Lambert is responsible for Sections 15, 16, 19 and 20 of this report and shares responsibility for Sections 1, 2, 3, 18, 21, 24, 25, and 26. Mr. Texidor is responsible for Sections 4 to 12, and 14 and shares responsibility for Sections 1, 2, and 23 to 26. Dr. Altman is responsible for Sections 13 and 17 and shares responsibility for Sections 1, 18, 20, 21, 24, 25, and 26. Mr. Miranda is co-author for Section 16 and shares responsibility for Sections 1, 2, 3, 24, 25, and 26. Mr. Malensek is responsible for Sections 21 and 22 and shares responsibility for Sections 1, 2, 3, 24, 25, and 26. The documentation reviewed, as well as any other sources of information, are listed at the end of this report in Section 27 (References).

LIST OF ABBREVIATIONS

Units of measurement used in this report conform to the metric system. All currency in this report is US dollars (US$) unless otherwise noted.

a	annum	kWh	kilowatt-hour
A	ampere	L	litre
bbl	barrels	lb	pound
btu	British thermal units	L/s	litres per second
°C	degree Celsius	m	metre
C$	Canadian dollars	M	mega (million); molar
cal	calorie	m^2	square metre
cfm	cubic feet per minute	m^3	cubic metre
cm	centimetre	µ	micron
cm^2	square centimetre	MASL	metres above sea level
d	day	µg	microgram
dia	diameter	m^3/h	cubic metres per hour
dmt	dry metric tonne	mi	mile
dwt	dead-weight ton	min	minute
°F	degree Fahrenheit	µm	micrometre
ft	foot	mm	millimetre
ft^2	square foot	mph	miles per hour
ft^3	cubic foot	MVA	megavolt-amperes
ft/s	foot per second	MW	megawatt
g	gram	MWh	megawatt-hour
G	giga (billion)	oz	Troy ounce (31.1035g)
Gal	Imperial gallon	oz/st, opt	ounce per short ton
g/L	gram per litre	ppb	part per billion
Gpm	Imperial gallons per minute	ppm	part per million
g/t	gram per tonne	psia	pound per square inch absolute
gr/ft^3	grain per cubic foot	psig	pound per square inch gauge
gr/m^3	grain per cubic metre	RL	relative elevation
ha	hectare	s	second
hp	horsepower	st	short ton
hr	hour	stpa	short ton per year
Hz	hertz	stpd	short ton per day
in.	inch	t	metric tonne
in^2	square inch	tpa	metric tonne per year
J	joule	tpd	metric tonne per day
k	kilo (thousand)	US$	United States dollar
kcal	kilocalorie	USg	United States gallon
kg	kilogram	USgpm	US gallon per minute
km	kilometre	V	volt
km^2	square kilometre	W	watt
km/h	kilometre per hour	wmt	wet metric tonne
kPa	kilopascal	wt%	weight percent
kVA	kilovolt-amperes	yd^3	cubic yard
kW	kilowatt	yr	year

kWh

3 RELIANCE ON OTHER EXPERTS

This report has been prepared by RPA at the request of GR Engineering (Barbados).

The information, conclusions, opinions, and estimates contained herein are based on:

1. information available to RPA at the time of preparation of this report,
2. assumptions, conditions, and qualifications as set forth in this report, and
3. data, reports, and opinions supplied by GRI and GRE and other third party sources.

For the purpose of this report, RPA has relied on ownership information provided by GRI. RPA has not researched property title or mineral rights for the Project and expresses no opinion as to the ownership status of the property.

RPA has relied on GRI for guidance on applicable taxes, royalties, and other government levies or interests, applicable to revenue or income.

Except for the purposes legislated under provincial securities laws, any use of this report by any third party is at that party's sole risk.

4 PROPERTY DESCRIPTION AND LOCATION

The Siembra Minera property is located in the Kilometre 88 mining district, Bolivar State, in southeast Venezuela at a Latitude of 6°10' North and Longitude 61°28' West. Highway 10, which is a major highway in Venezuela, is only 3.5 km from the property (Figure 4-1).

The Project site is located in the Guyana region, which covers approximately one-third of Venezuela's national territory. The closest nearby large city is Ciudad Guayana, with approximately 1,050,000 inhabitants (2001), situated on the Orinoco River near its confluence with the Caroní River. Ciudad Guayana consists of the old town of San Félix to the east and the new town of Puerto Ordaz to the west. Puerto Ordaz is home to most of the major industrial facilities such as aluminum smelters and port facilities. Puerto Ordaz has major port facilities accessible to ocean-going vessels from the Atlantic Ocean via the Orinoco River, a distance of approximately 200 km. There is regularly scheduled commercial airline service to Puerto Ordaz from Caracas.

Ciudad Guayana is the centre of major industrial developments in the area, including iron and steel mills, aluminum smelters, iron and bauxite mining, and forestry. These industries are supported by major dams and hydroelectric generating plants on the Caroní River, providing 12,900 MW of electricity.

LAND TENURE

The Project survey control is based on the Universal Transverse Mercator (UTM) coordinate system. It is based on the Zone 20 North projection, using the World Geodetic System 1984 (WGS'84) datum. Surco, S.A. (Surco), a local survey firm based in El Callao, Venezuela, established permanent survey reference points within the Project area. The base for all surveys was Global Positioning System (GPS) survey, defined and checked by the survey company with a traverse from a nearby GPS station (Cristinas) with satisfactory accuracy. Surco surveyed the drill holes for both Placer and GRI.

The Siembra Minera Economic Zone (Project boundary) occupies a rectangular area of approximately 18,951 ha. The dimensions of the property are 20.5 km (north-south) by 11.2 km (east-west). The Economic Zone will contain the open pit mine, all Project infrastructure, and waste rock dumps (WRD) and tailings management facilities (TMF). The land includes the prior concessions of Cristinas 4, Cristinas 5, Cristinas 6, Cristinas 7, Oro 1, Albino 1, Brisas del Cuyuni, Carabobo, Morauana, Barbara, Zuleima, El Pauji, Esperanza, and portions of Guarimba, Mireya, Virgen de Lourdes, Lucia, and a few smaller concessions. The Economic Zone is shown in Figure 4-2.

The Economic Zone is designated by Presidential Decree No. 30 dated October 31, 2016 and authorized by Nicolás Maduro Moros. The decree delimits the geographic area in which Siembra Minera shall perform activities of exploration and exploitation of gold mines and deposits, including their production. The coordinates of the Economic Zone are presented In Table 4-1.

TABLE 4-1 UTM COORDINATES OF ECONOMIC ZONE

GR Engineering (Barbados), Inc. – Siembra Minera Project

Point	East	North
1	663.273,418	689.184,268
2	668.271,235	689.198,703
3	669.280,000	689.540,000
4	673.340,000	689.540,000
5	673.284,000	685.280,000
6	673.284,000	680.000,000
7	674.500,000	678.000,000
8	674.500,000	668.972,500
9	664.925,000	668.972,500
10	664.925,000	685.186,866
11	663.273,418	685.186,866

RPA is not aware of any environmental liabilities on the property other than existing mercury levels from artisanal miners as discussed in Section 20. RPA is not aware of any other significant factors and risks that may affect access, title, or the right or ability to perform the proposed work program on the property.

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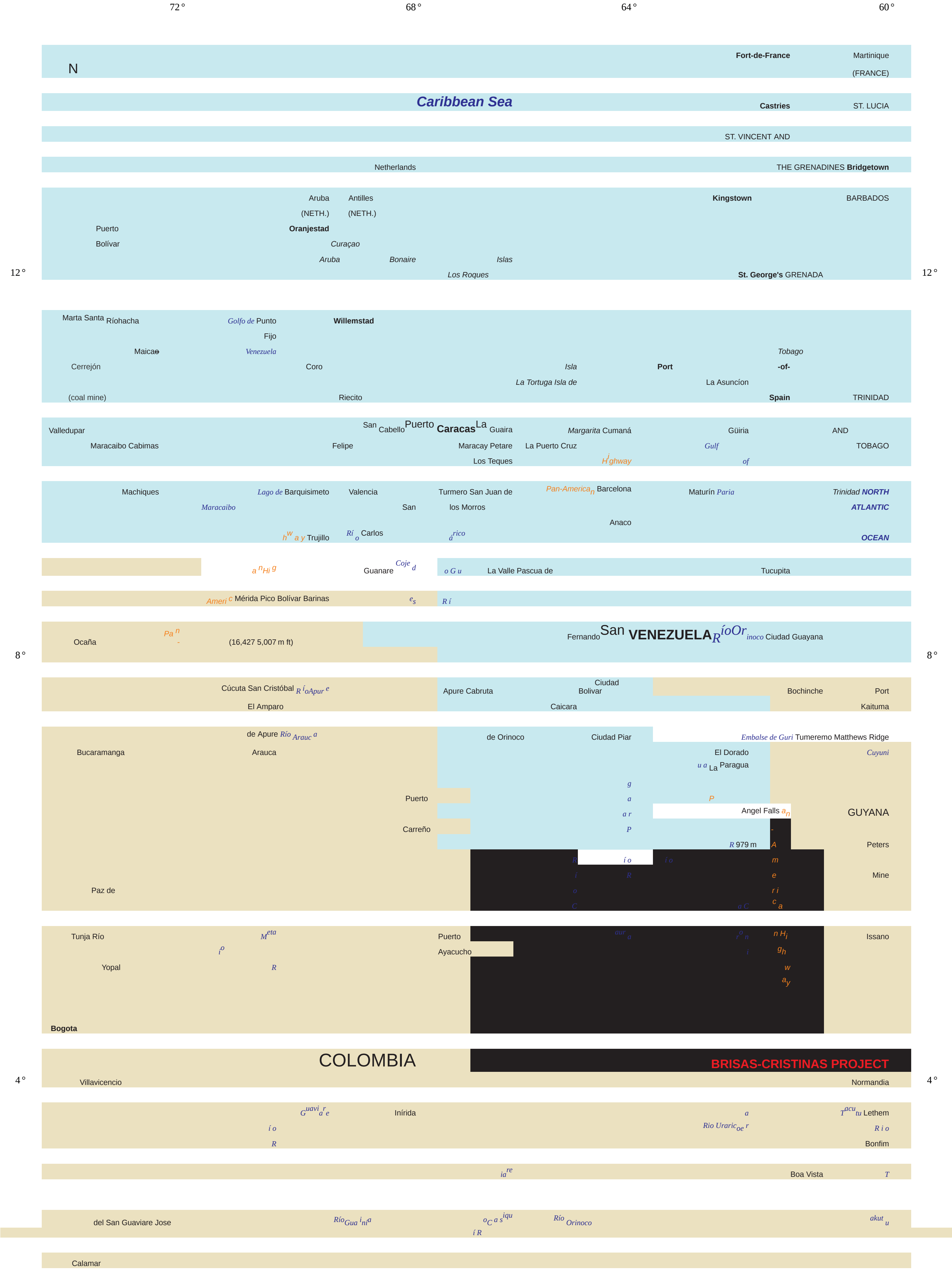

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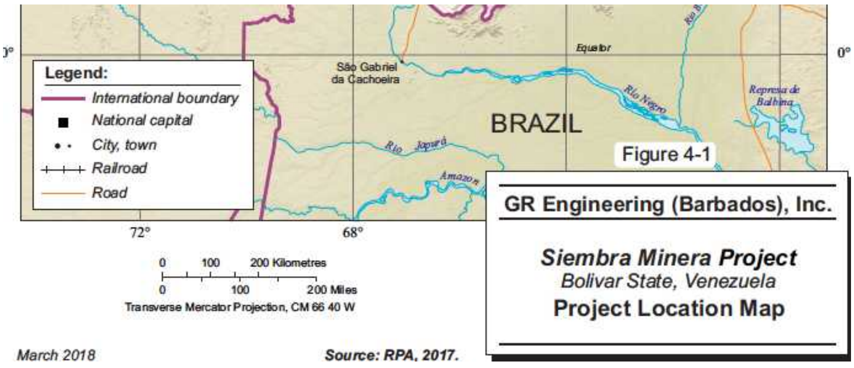


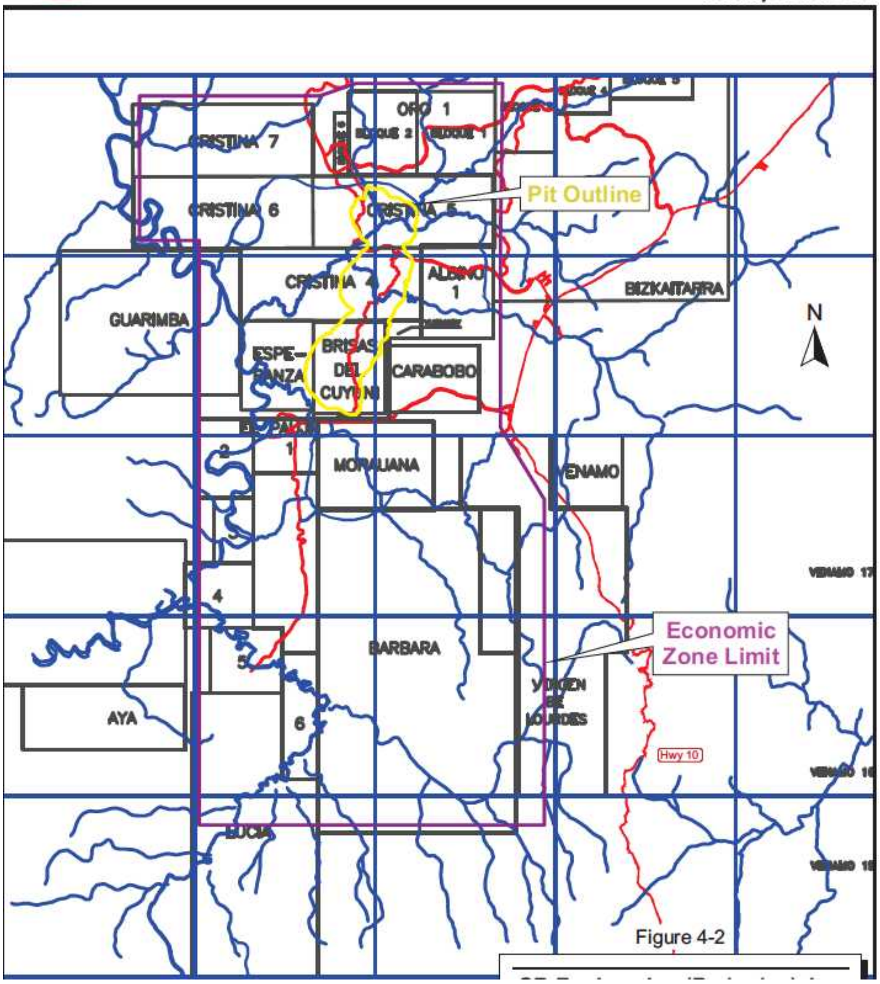


Figure 4-2

GR Engineering (Barbados), Inc.

Siembra Minera Project

Bolivar State, Venezuela

Economic Zone and Land Position

0 1 2 3 4

Kilometres

March 2018 ***Source: Gold Reserve Inc., 2015.***

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

ACCESSIBILITY

El Dorado (population est. 5,000, 2011), which is 88 km north of the Project site on Highway 10, is historically the nearest population centre of any size. Over the past ten years, the town of Las Claritas (population est. 6,000, in 2011, previous est. 2,500, in 2001) adjacent to the Project has surpassed El Dorado with the influx of artisanal miners. Las Claritas is a small town located on Highway 10.

Highway 10 provides paved access from Ciudad Guayana and Puerto Ordaz, which is 373 km northwest of the property, to within 3.5 km of the Project site. Unpaved roads provide the remaining 3.5 km of access. Upgrading the unpaved roads is part of the infrastructure improvement plans for the Project area which will include three main Project access routes, one from the north, one from the east near Las Claritas, and a third to the south providing direct access to the process plant from Highway 10.

CLIMATE

The climate is tropical with January through March being drier months and June through July being wetter months. Humidity is high (monthly average 80-87%) and annual precipitation is over 3,000 mm. Temperatures are fairly uniform with average highs around 35°C and average lows around 23°C. Daily temperatures range from 21°C to 38°C. Prevailing winds are from the west – southwest, with a speed mostly from 0.5-2.1 m/sec. There are extensive plans for surface and ground water control so that mining can be conducted year-round.

LOCAL RESOURCES

Plans are to construct a dual use camp facility at a location adjacent to the process area. This camp will initially house construction workers and at the conclusion of construction will be converted to a permanent facility for a portion of the operating personnel. The construction camp configuration will have a nominal capacity of 2,400 men based on an occupancy rate of two men per room. This may be increased during peak periods if required.

When the construction has been completed the camp will be re-configured to provide a single room for each man and permanent recreation facilities for the operating staff. The permanent operations camp facility will have a capacity of 1,200 staff. It is assumed that the rest of the operating staff (~300 personnel) will come from the local communities.

Busing will be considered if required from El Dorado and other communities.

INFRASTRUCTURE

To support the mining and milling operations at the Siembra Minera Project, a number of ancillary facilities will be required. These include a mine equipment maintenance shop, warehouse, reagent storage building, laboratory, and administration offices. A construction camp will be prepared and will be converted to an operation camp. The operational man camp size is based on the assumption that approximately half the work force is away on scheduled time off due to crew rotations. Previously, there was a small camp with several cinder block buildings that could house up to 100 people at Brisas and a larger camp at Cristinas that was constructed by Placer. The camp at Brisas was used to support the exploration programs and the camp at Cristinas was used to support the initial construction efforts. Both camps have been destroyed and a new larger camp will be constructed near the plant site.

Three unpaved roads are used to access the Project from Highway 10. Plans are to improve these to provide access to the mining area and the process area. A network of service roads will be constructed to allow access to the camp facility, tailings dam, sedimentation ponds, explosives magazine, and other remote installations. Major deliveries will use either the north access road or the south access road and will avoid the Las Claritas village.

A water supply and distribution system will be constructed, using the pit dewatering wells as a source of fresh water. The mill area, mining area, and the campsite will each be provided with a sewage collection and treatment system.

The power authority, Corporación Eléctrica Nacional S.A. (Corpoelec) is the fully integrated state power corporation of Venezuela. It was created in 2007 by merging ten state-owned and six private-owned power companies. Corpoelec constructed a power line south from Ciudad Guayana into Brazil. The authority has also constructed a substation at Las Claritas located approximately 3.5 km from the Project, which has sufficient power to supply the Siembra Minera Project.

PHYSIOGRAPHY

The Project area is located at the foot of the Sierra de Lema high plateau; and, the topography is moderately homogenous, dominated by plains with some rolling hills. Terrain in the mine area is relatively flat with elevations ranging from 120 MASL to 220 MASL with higher elevations near the east and southeast margins of the Project area. Near the plant site and tailings disposal areas, the terrain goes from being relatively flat to fairly steep. The tailings disposal site design has used this as an advantage by constructing the dam in the flat area and using the hillside as the back of the tailings disposal facility. Elevations in the plant and tailings disposal area range from 130 MASL to 200 MASL. The plant site will be at 190 m above sea level.

Most of the area is covered by moderately dense sub-Amazon rainforest. Trees range from 25 m to 35 m in height. Low-lying areas tend to be wet and swampy. There are many pits left by artisanal miners that are filled with water. They will require pumping prior to mining.

6 HISTORY

EXPLORATION AND DEVELOPMENT HISTORY

General Fernandez Amparan first discovered gold in the Siembra Minera region in 1920. Gold mining at the site was initiated in the 1930s and continued sporadically on a minor scale until the early 1980s when a gold rush occurred. Approximately 5,000 to 7,000 small miners worked alluvial and saprolite-hosted gold deposits using hydraulic mining techniques. Many square kilometres of jungle have been stripped of soil and saprolite. This material was processed in sluices and small hammer mills. The amount of gold recovered is unknown and much of the area of the concessions is now covered with tailings. The name Kilometre 88 for the district came from the area being located near kilometre 88 marker of the road linking El Dorado with the Brazilian border (Pan American Highway or Highway 10).

Placer conducted essentially all of the modern exploration on Cristinas during their tenure on the property from 1991 to 2001. Placer completed line cutting, mapping, rock and soil sampling, geophysics, and drilling of 1,174 drill holes for a total of 158,738 m of drilling. After extensive exploration, Placer announced commencement of construction of the Las Cristinas mine on August 2, 1997. The inauguration took place at the site with officials of Placer, CVG, and representatives of the Venezuelan Government present. On January 20, 1998, Placer announced that its operating company in Venezuela, Minera Las Cristinas C.A., had decided to suspend construction. Construction resumed in May 1999 but was again suspended on July 15, 1999 due to uncertainty with respect to gold prices and title. Up until that time, Placer had reportedly spent US$168 million on the Project. CVG took possession of the Cristinas property in 2001 and in 2002 signed a mine operating agreement (MOA) with Crystallex International Corporation (Crystallex) whereby Crystallex was required to explore, mine, and produce gold at Las Cristinas.

Crystallex undertook drilling to confirm results of the previous operator prior to their first resource estimate. Crystallex drilled 12 holes totalling 2,199 m in 2003 to confirm the tenor of mineralization presented in the pre-existing database and also assayed check samples. For additional confirmation, Crystallex re-assayed 262 pre-existing pulps, 200 pre-existing coarse rejects, and 342 pre-existing quarter-core samples. During the course of the drill data verification and the resource expansion drilling in 2005, it was noted (MDA 2005) that some biases existed between Crystallex and Placer data, the latter of which represent by far the bulk of the exploration data. A heterogeneity study was undertaken to better understand the grade biases noted, to define more appropriate sub-sampling procedures and protocol, and to maximize the efficiency of the upcoming grade-control program during mining operations. A report by Francis Pitard (2005) suggested that the grade bias of Crystallex grades being lower than Placer grades was likely due to the difference in size of the core samples. Pitard further pointed out that the samples taken by Placer also could be understating the global grade of the Cristinas deposit. Crystallex completed a 46-hole drill program in February 2007. The audits concluded that Placer and Crystallex procedures met or exceeded industry standards at the time, and assay laboratories provided reliable and acceptable results.

In February 2011, the MOA was terminated by CVG.

The Brisas concession was acquired by GRI in August 1992 with the acquisition of Compañia Aurifera Brisas del Cuyuni C.A. Prior to 1992, no known drill holes existed on the Brisas site. Initial work by GRI included surface mapping, regional geophysical surveys, and geochemical sampling. Several anomalies were identified on the property and drilling and assaying began in 1993. A large deposit with stratabound gold-copper mineralization was discovered in both alluvial and hard rock material early in the drilling program. Additional work followed with petrology, mineral studies, density tests, metallurgical sample collection, and laboratory test work.

Initial exploration drilling by GRI commenced in 1993 utilizing both auger and core drilling methods. Most of the exploration and development drilling took place in 1996 and 1997. From 1996 on, all exploration drilling has been completed utilizing diamond drill core rigs. Additional exploration drilling was completed in 1999, 2003, 2004, and 2005. As of 2005, 802 exploration holes had been drilled of which 731 are diamond core holes. This represents 186,094 m of core drilling, and 189,985 total m of exploration drilling, core and auger. All split core was stored on site until 2008, but has since been ransacked and displaced.

Since 2005, an additional 76 holes have been drilled on the Brisas concessions for geotechnical and other studies. These holes have not been included in any resource modelling because they were not drilled for exploration purposes.

Independent verification by Behre Dolbear & Company. Inc. (Behre Dolbear) of drilling, assaying, and data collection procedures was undertaken in 1997 and verification of the computer database, mine modelling procedures, and reserve estimate was completed in 1998. The audits concluded that GRI procedures met or exceeded industry standards, and assay laboratories provided reliable and acceptable results.

In August 2016, GRI signed a mixed company agreement with Venezuela for the formation of a jointly owned company and, in October 2016, established Siembra Minera, a mixed capital company with 55% owned by a Venezuelan state entity and 45% by GRM, a wholly-owned subsidiary of GRI.

HISTORIC RESOURCE ESTIMATES AND FEASIBILITY STUDIES

CRISTINAS

Resource and reserve estimates for the Cristinas deposit were completed by Mine Development Associates (MDA) on April 30, 2003. These results were filed on SEDAR as the Technical Report titled “Resources and Reserves, Las Cristinas Gold and Copper Deposits, Bolivar State, Venezuela” prepared by MDA. The measured and indicated resource was estimated at 439 million tonnes with an average gold grade of 1.09 g/t for a total of 15 million contained ounces based on a gold cut-off grade of 0.5 g/t. Proven and probable mineral reserves were estimated at 224 million tonnes with an average gold grade of 1.33 g/t containing 9.54 million ounces (MDA, 2003).

Subsequent to the filing of the 2003 Technical Report by MDA, there have been other resource and reserve estimates released by Crystallex. A feasibility study for Las Cristinas was completed in 2004 and a Development Plan, in 2005 by SNC-Lavalin. An updated Technical Report was filed on SEDAR in August 2005 based on the new Development Plan. The 2005 Technical Report shows proven and probable reserves at Las Cristinas of 294 million tonnes grading 1.32 g/t Au for a total of 12.5 million contained ounces of gold (SNC-Lavalin, 2005).

An updated resource and reserve estimate and a Technical Report were completed for the project by MDA in conjunction with SNC-Lavalin on November 7, 2007. These results were filed on SEDAR as the Technical Report titled “Technical Report Update on the Las Cristinas Project, Bolivar State, Venezuela” prepared by MDA. The measured and indicated resource was estimated at 629 million tonnes with an average gold grade of 1.03 g/t for a total of 21 million contained ounces based on a gold cut-off grade of 0.5 g/t. Proven and probable mineral reserves were estimated at 464 million tonnes with an average gold grade of 1.13 g/t containing 16.86 million ounces of gold (MDA, 2007).

The resource and reserve estimates quoted above are considered to be historic in nature and should not be relied upon, however, these are relevant as they indicate the presence of mineralization on the Project.

BRISAS

J.E. MinCorp, a division of Jacobs Engineering Group, Inc. completed a pre-feasibility study on the Brisas Project in February 1998. In addition, a supplement to the pre-feasibility study was completed in August 1998, addressing the merits of the Cominco Engineering Services Ltd. (CESL) hydro-metallurgical process. The CESL process was a method of treating copper concentrates on site by pressure oxidation, acid leaching with solvent extraction/electrowinning recovery of copper in the form of copper cathode, and gold recovery by a cyanide leach of the solids. Work completed since 1998 and directed at project optimization includes updating the mine computer model and ultimate pit designs, mine planning and optimization of cut-off grades, and updated slope stability design criteria. In addition, work was completed on mill tailings characterization and analysis of physical properties, cyanide destruction test, and settling and thickening tests for plant design criteria.

In 2003, Behre Dolbear completed Mineral Resource and Mineral Reserve estimates for the Brisas deposit and documented it in a Technical Report filed on SEDAR. The estimates were based on two scenarios for treating the copper concentrates, a conventional smelter and refining case and the use of CESL hydro-metallurgical process (Behre Dolbear, 2003). Using a gold price of $325 per ounce and a copper price of $0.85 per pound, proven and probable reserves were estimated to be 257 million tonnes grading 0.805 g/t Au and 0.135% Cu containing 6.64 million ounces of gold and 764 million pounds of copper for the smelter case and 328 million tonnes grading 0.708 g/t Au and 0.150% Cu containing 7.48 million ounces of gold and 1.08 billion pounds of copper for the CESL case.

GRI commenced work for a bankable feasibility study in the last quarter of 2003. Several major engineering and consulting companies were selected to complete the work necessary for the feasibility study. They were Aker Kvaerner, an engineering and construction company specializing in mining and mineral processing; Vector Colorado, a tailings dam design, geotechnical and hydrology specialist; and PAH for the mineral resource and reserve estimate, pit design, mine planning, and mine cost estimation. This feasibility study was completed in January 2005. In addition, AATA International and Ingenieria Caura, S.A. was selected to complete an Environmental and Social Impact Assessment (ESIA) Study in compliance with the World Bank and International Finance Corporation (IFC) Standards for meeting Equator Principles criteria. The feasibility study operating plan assumes a large open pit mine containing proven and probable reserves of approximately 9.2 million ounces of gold and 1.2 billion pounds of copper in 414 million tonnes of ore grading 0.69 g/t Au and 0.13% Cu (PAH, 2005).

The combination of the 2005 feasibility study and subsequent studies provided the basis of the 2008 Technical Report. Several optimization studies were conducted to determine the most economic process plant option and production rate. Total proven and probable mineral reserves for the Brisas Project in 2008 were estimated at 482.7 million tonnes grading 0.66 g/t Au and 0.13% Cu. A total of 1.08 billion tonnes of waste was estimated in the pit resulting in a stripping ratio (waste:ore) of 2.24:1.0 (PAH, 2008).

All Mineral Reserve estimates quoted above are considered to be historic in nature and should not be relied upon, however, these are relevant as they indicate the presence of mineralization on the Project.

All previous Mineral Resource estimates for the Brisas Project are superseded by the current Mineral Resource estimate in Section 14 of this report.

PAST PRODUCTION

There are no records of previous gold production from artisanal miners.

7 GEOLOGICAL SETTING AND MINERALIZATION

REGIONAL GEOLOGY

The Siembra Minera Project is located within the Guyana Shield in northern South America. The shield covers easternmost Colombia, southeastern Venezuela, Guyana, Suriname, French Guiana, and northeastern Brazil. The Venezuelan portion of the shield is subdivided into five geological provinces with different petrological, structural, and metallogenic characteristics. The provinces are, from oldest to youngest, Imataca, Pastora, Cuchivero, Roraima, and Parguaza. Only the Imataca, Pastora, and Roraima provinces are found in the vicinity of the Siembra Minera deposit.

Rocks of the Imataca Province constitute the oldest terrain in the Venezuelan Guyana Shield and include quartzo-feldspathic gneiss, felsic, and mafic granulites, and iron formation. This province is located along the Orinoco River in the northern portion of the Guyana Shield. Rocks in the terrain are tightly folded, highly metamorphosed, and have ages ranging from 3,700 Ma to 2,150 Ma. The oldest age represents the protolith, whereas the younger age represents the Trans-Amazonian orogeny of Lower Proterozoic age. The Imataca Province is known for iron deposits hosted by banded iron formations.

The Pastora Province is separated from the Imataca terrain by the Guri fault on its northern edge and extends to the Kilometre 88 gold district in the south. This province is characterized by several penecontemporaneous tholeiitic and calc-alkaline volcano-sedimentary sequences. Rock types that have been described and are not necessarily present in all sequences include pillow basalt, andesite, dacite, rhyolite, tuffaceous and pyroclastic sediments, greywacke, pelite, tuff, and chemical sedimentary rocks. Rocks of the province were metamorphosed to greenschist facies and intruded at various levels by granitic rocks of the Supamo Complex (2,230 Ma to 2,050 Ma). This petrologic assemblage constitutes the granite-greenstone belts of Lower Proterozoic age, which extends into Guyana, Suriname, French Guiana, and Brazil. The Trans-Amazonian orogeny (2,150 Ma to 1,960 Ma) was a period of deformation, metamorphism, magnetism, and enrichment of previously deposited gold-bearing volcano-sedimentary rocks in the Venezuelan part of the Guyana Shield as well and in the other mentioned countries. Rocks of this province have been intruded by Lower Proterozoic (1850

Ma to 1650 Ma) and Mesozoic (210 Ma to 200 Ma) diabase dikes, sills, and gabbroic bodies related to crustal extension.

The Roraima Province of Middle Proterozoic age (1,600 Ma) is exposed to the south of the Kilometre 88 district. This province includes sedimentary rocks of continental origin that were laid unconformably on top of the granite-greenstone terrain. These rocks are not metamorphosed, have horizontal to low angle dips, and are intruded by Mesozoic diabase dikes and sills.

LOCAL AND PROPERTY GEOLOGY

The greenstone belt present in the Kilometre 88 district consists of four formations, listed below oldest to youngest:

1) Lower Carichapo Group meta-lavas, meta-tuffs, amphibolites, and ferruginous quartzites.

2) Lower Proterozoic greenstone basalts, andesites, tuffaceous rocks, pyroclastic breccias, and metagraywackes. These rocks are lithologically similar to the Caballape Formation defined in the Botanamo district to the north east, but geographically isolated. For convenience, they are referred to as Lower Caballape in this report.

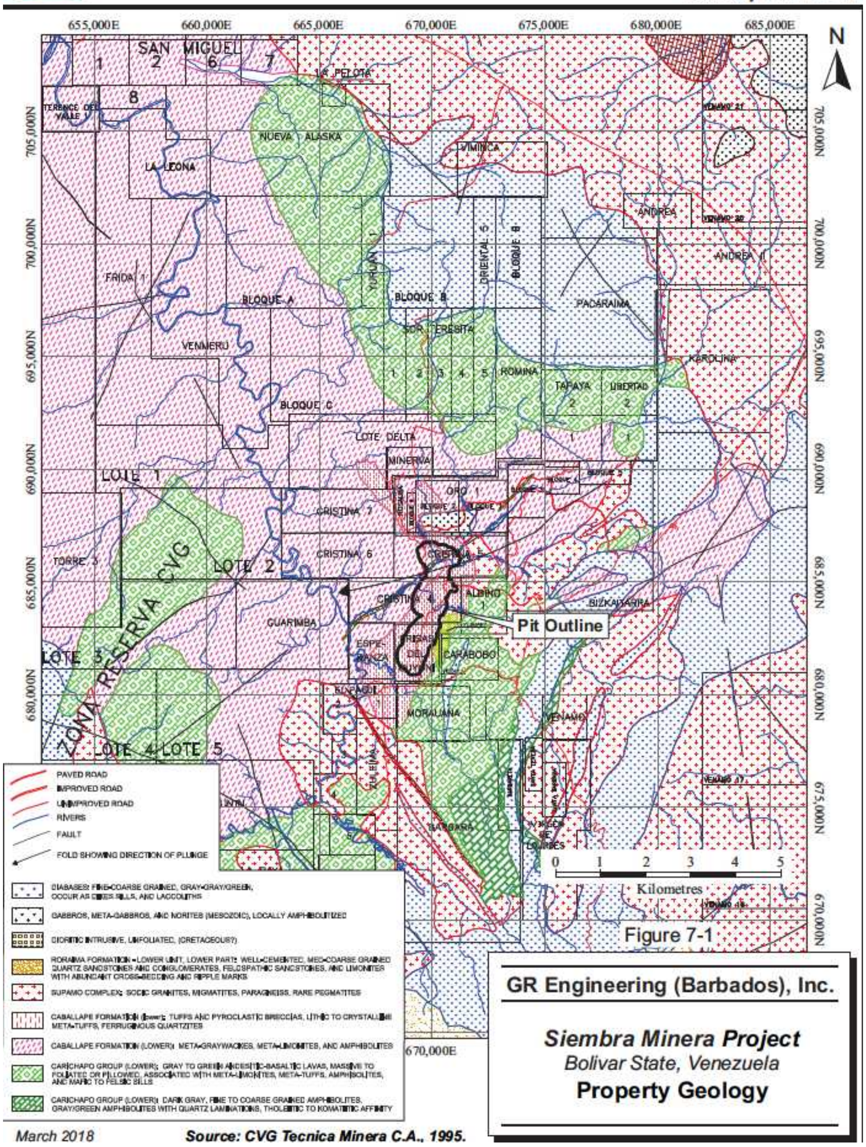
3) Granites and granodiorites of the Supamo Complex.
4) Diabasic and gabbroic dikes and sills of Lower Proterozoic and Mesozoic ages.

The position and coverage of the above units have been established, at least on a regional scale, through aerial photos. Ground reconnaissance by government missions and more recently by private entities has either confirmed or mapped modifications to the aerial interpretations. The present geologic map is a composite of the above work. Rocks of the Carichapo Formation surround the concessions to the southwest, southeast, east, and north. They generally correspond to areas of higher topographic expression and are not commonly host to significant gold deposits. Greenschist volcanic and volcano-sedimentary rocks of calc-alkaline composition (called Lower Caballape Formation in this report) constitute the major units present in the areas of gold deposits, including the Brisas and Cristinas properties. The older unit covering the concessions consists primarily of intermediate tuffaceous rocks, and the younger unit to the west consists of intermediate to felsic tuffs, lavas, and volcano-sedimentary rocks. This sequence of rock units corresponds to areas of low, flat topography, forming hills only where the rock mass is more silicified. Relatively unfoliated intrusions of Supamo Complex granites are restricted to the south, east, and northeast, where they are topographically indistinct from the greenschist volcanics. All of the above units are intruded by Lower Proterozoic and Mesozoic diabases and gabbroic bodies, both as large mappable features, and as thin dikes and sills occurring in the volcanic units.

The Siembra Minera deposit lies within a portion of the lower Caballape Formation volcanic and volcanic-related sedimentary rocks. The units present are (1) andesitic to rhyolitic tuffaceous volcanic beds, (2) related sedimentary beds, and (3) a tonalitic intrusive body (Figure 7-1). All rocks have been tilted and subjected to lower greenschist facies metamorphism. It is thought, based on information from nearby properties, that the Siembra Minera Project occupies one limb of a large regional fold. Limited direction-indicating structures show the strata to be top-up. In the main mineralized trend, moderate to strong foliation is oriented N10°E and dipping 30° to 55° northwest. This foliation appears to be parallel to the original bedding and tends to be strongest in the finer-grained rocks. A much weaker foliation orientation appears in outcrop exposures, striking north-northwest and dipping to the southwest.

Dikes and quartz veins cut the lower Caballape Formation. The strata and intrusive rocks are cut by N30°W striking mafic dikes emplaced at regular intervals (200 m to 600 m), some of which have displacement in the order of tens of metres. These dikes are thought to be related to the Mesozoic diabase intrusions present throughout the district. Quartz veins populate the concession and have been noted both in outcrop and in drill intersection. The most common are sets of thick, boudinaged, and en echelon vein structures that follow foliation/bedding orientation. They are thought to relate in part to movement of quartz during metamorphism. Other quartz veins exist in various orientations that cannot be definitively linked to the structural elements described above.

One of the largest and best-defined stock reaches surface, in the saprolite, in a northeast-trending zone in the Potaso area on the south edge of the Cristinas deposit. The diorite, located north of the Potaso area, is asymmetric in a north-south section: it has a sub-vertical northwest face while its roof is shallowly inclined, dipping south at an angle of approximately 30° beneath the northern edge of the Brisas de Cuyuni deposit. This diorite stock occupies the gap in economic mineralization between the Cristinas and Brisas de Cuyuni deposits. The second diorite stock is located in the northern part of the Cristinas concessions, where it occupies the gap in mineralization between the Mesones and Morrocoy areas.



GR Engineering (Barbados), Inc.

Siembra Minera Project
Bolivar State, Venezuela
Property Geology

March 2018 **Source: CVG Tecnica Minera C.A., 1995.**

ROCK UNITS

There are two general categories of rock units: weathered and unweathered rock. Weathered rock is further defined by degree of oxidation and mineral replacement due to weathering processes. Unweathered rock is further defined by lithology into various subdivisions of volcanic extrusive or intrusive units.

WEATHERED ROCK AND SAPROLITE

Oxidized Saprolite. A red-brown to yellow saprolite occurs in almost all parts of the concession from the surface to an average depth of 24 m. It is absent in the few areas where hard rock material outcrops. It is composed of clays, quartz, and hard ferruginous material in which all sulphide minerals have been oxidized and most other rock-forming minerals have been broken down to clay minerals and quartz.

Sulphide Saprolite. Sulphide saprolite, varying in thickness from less than one metre to 80 m, occurs immediately below oxidized saprolite. The water table constitutes the contact between the two and is generally sharp. It is noted on the geologic logs as "BOS" (base of oxidized saprolite). Sulphide saprolite is predominantly clay with both primary and secondary sulphides, the original rock having been broken down beyond recognition. Fragments of hard tuffaceous rock can occur. The initial occurrence of hard rock fragments in this unit (or in oxidized saprolite) is denoted on the geologic logs by the acronym "BAS" (base of 100% clay material). This boundary can exist in either sulphide or oxidized saprolite. The sulphide saprolite is well developed in the mineralized zone of the concession, but can be quite thin or absent in areas distal to mineralization.

Weathered Rock. Weathered rock is a label for any hard rock existing in a state of intense weathering, but not sufficiently broken down into clay to qualify as a saprolite. In general it falls between two contacts noted on geologic logs as "BZM" (base of mixed clay/hard rock material) and 'BDM' (base of weathering). In practice it is logged as the original rock type or as schist in the event that the original texture cannot be distinguished. Below the BDM, rock exists in a state of weathering in which the only chemical change is the leaching of calcite. The base of this layer is denoted as "BDL" (base of leaching), and below the rock is considered completely fresh.

UNWEATHERED ROCK

Schist Units. The classification of schist is used when the original tuffaceous texture of the rock units has been erased by metamorphic processes. Schistosity is developed parallel to bedding, so schist units generally, but not always, follow dip of the tuffaceous units. Two types of schist have been defined: chlorite-sericite-biotite schist and quartz-sericite-pyrite schist.

Volcanic Units. The original unweathered rock types are calc-alkaline volcanic tuffs, generally of andesitic to dacitic composition. Occurrences of tuffaceous units reworked by sedimentary processes have been noted, but not to any great extent. Nomenclature of tuffaceous units has been established through observation of core, petrographic analysis, and geochemical data. Bedding and, to a lesser extent, graded bedding are commonly recognized. In general, feldspar crystal abundances are counted only with crystals exceeding one millimeter in diameter, and the field term of a “lapilli” is a pyroclast exceeding two millimeters in diameter. The ternary diagram provided in Figure 7-2 illustrates the composition of the various volcanic rock types recognized on the concessions.

a) **Vitric Tuff.** Vitric tuff (TV) is a fine-grained, crystal-poor tuffaceous volcanic rock usually black in colour where not highly sericitized. It consists predominantly of glassy material, now devitrified, from the fallout of ash-sized particles. By definition it contains less than 10% feldspar crystals and less than 10% lithic fragments. It varies from a finely-banded volcanic sediment, to more massive mud-flow type deposit, which may contain lapilli pyroclasts, to a fine-grained massive texture. It is fully gradational into TVC-M and TL units (defined later).

b) **Crystal-Vitric Tuff.** Crystal-vitric tuff (TVC-M) is defined as a tuffaceous unit having 10% to 40% feldspar crystals, and less than 10% lithic fragments. Locally the crystal content can drop as low as 10% but averaged over an entire depositional unit must exceed 10%. The upper boundary of 40% crystals is arbitrarily set, local fluctuations being ignored. When lapilli are observed and amount to more than a few percent of the rock mass, the unit is described as a lapilli-bearing TVC-M or TV.

C) **Crowded-Crystal-Vitric Tuff.** Crowded-crystal-vitric tuff (TVC-C) is defined as a tuffaceous unit having greater than 40% feldspar crystals and less than 10% lithic fragments. It commonly contains significant mafic minerals (e.g., amphibole altered to biotite). If more than a few percent lithic fragments are observed, the unit is described as a lapilli-bearing TVC-C. Crowded-crystal-vitric tuff commonly resembles andesite porphyry, but numerous small lithic lapilli and grain size variations refute this possibility.

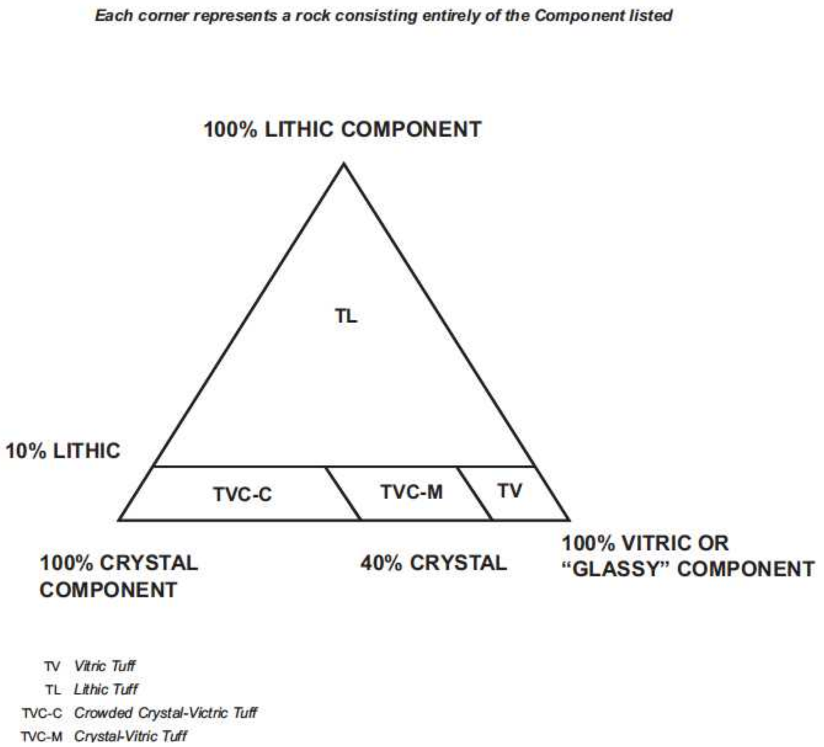


Figure 7-2

GR Engineering (Barbados), Inc.

Siembra Minera Project

Bolivar State, Venezuela

Ternary Diagram for Classification of Tuffaceous Units

March 2018

d) **Lithic Tuff.** Lithic tuff (TL) is defined as a tuffaceous unit containing greater than 10% lapilli-sized fragments. This definition is used without regard to presence or absence of feldspar crystals in the matrix, as field rock descriptions do not allow for further textural distinction. The fragments in some cases appear to be pieces of tuffaceous rock, presumably torn from its location by later volcanic activity. Pumice fragments have also been noted. It has been found to be important as a marker horizon, as it has an unmistakable texture and for the most part is observed in thin but easily definable units.

Intrusive Units. There are three mineralogically and texturally distinct occurrences of intrusive units, which vary from basaltic to dioritic in composition, all of which, are younger than the tuffaceous units described above.

a) **Mafic Dikes.** This fine-grained, probably hypabyssal rock has a prominent 'spinifex' texture defined not by olivine, but rather by feldspar grains. They are unaltered, unfoliated, and magnetic. There are six such dikes on the concession, striking generally N40W, spaced 200- to 600-meters apart. They range from less than one metre to over five metres in width. Cross-cutting relationships indicate that they are the youngest rocks on the Concessions.

b) **Intermediate Coarse-Grained Intrusive.** A coarse-grained tonalitic intrusive has been identified in only one area in the eastern part of the Concessions. It appears to be amorphous in shape and drilling has not encountered a lower contact. It is a coarse- grained, equigranular rock in large part unfoliated, but cut by discrete zones of strong deformation, both with and without sulphides and alteration. Zones of fracture-controlled chalcopyrite are also present, though the body does not exhibit economic Au or Cu mineralization. The only contacts observed to date are with TVC-C and are difficult to pinpoint as the two units can appear similar in hand sample. In one drill hole, it is cut by a mafic dike. The equigranular texture, high quartz content, and grain size are diagnostic. TVC-C, with which, it is sometimes confused, tends to have much greater variation in crystal content.

c) **Intermediate Aphanitic Sill/Dike.** Intermediate hypabyssal intrusives occur as sill- like bodies less than one metre thick. These intrusives are usually aphanitic and are weakly foliated. They are useful as marker horizons within the volcaniclastic pile.

STRATIGRAPHY

A stratigraphic column for the concession has been defined from the lithologic interpretation of the drill holes and is presented below in outline form (oldest to youngest):

a) The lowest grouping is a sequence of crystal and crowded-crystal tuffaceous units that have a uniform appearance with very gradual changes in crystal percentages. The base of this sequence has not been reached by drilling.

b) A thick crystal-vitric tuff and underlying vitric tuff that appears in the Cristinas concessions and the northern part of the Brisas concession (north of 682,500 N).

South of this line the unit either pinches out, or drilling has been insufficient at depth to properly define it.

c) A 150 m to 200 m thick sequence consists of rapidly alternating TL, TV, and TVC-M units. A prominent band of TL defines the base. Within this group only the TL bands and one TV bed are found to be laterally continuous, though even they are highly variable in thickness and extent. The bulk of economic gold mineralization occurs within this sequence. A sill of intermediate composition exists near the base of this sequence and is traceable throughout the concessions. The entire sequence thins toward the south, narrowing to less than 100 m at 681,500 N.

d) A TV unit greater than 200 m thick appears throughout the concessions and contains minor TVC-M and TL bands. Much of this unit has a very even texture, and the contact with the underlying unit is readily apparent in most drill holes.

e) A poorly defined sequence of TL, TV, TVC-M, and TVC-C units overlies (D), but is well outside, the mineralized zone and only encountered in a few condemnation drill holes to the west. This area has a strongly developed foliation, to the point where many units have been lumped together as “schist.”

f) A diorite/tonalite intrusive feature exists on the eastern edge of the concessions that appears to postdate emplacement of the tuffaceous units, as it cross-cuts the stratigraphy, however, information about the contact between this body and the tuffaceous units is limited. No strong mineralization has been discovered in or at the margins of this body.

Regional mapping by the Venezuelan Geological Survey shows the Cristinas Project lying within the Caballape Formation of the Botanamo Group. The Caballape Formation is described as consisting largely of graded wackes and other sedimentary facies with minor andesitic to rhyodacitic volcanic intercalations. This description contrasts with the dominantly mafic to intermediate composition volcanic nature of the sequence that hosts the mineralization at Cristinas. The host sequence at Cristinas is now considered to constitute part of the Carichapo Group of the Pastora Supergroup (Table 7-1).

TABLE 7-1 REGIONAL STRATIGRAPHY AND BROAD DESCRIPTION
GR Engineering (Barbados), Inc. – Siembra Minera Project

			Unit	Lithology	Age
	Intrusive		**Supamo Complex**	Granite, tonalite, trondhjemite, granodiorite, quartz monzonite, gneiss, & migmatite.	
		Botanamo Group	**Los Caribes Formation**	Intercalation of grey and green phyllites that grade to red phyllite that are intercalated with red sandstones and polymictic conglomerates with minor felsic tuff.	
		Botanamo Group	**Caballape Formation**	Graded graywacke, siltstones, & conglomerates (80%) with minor tuffs, breccias and pyroclastic flows of andesitic to rhyodacitic composition.	
Pastora Province			**Yuruari Formation**	Epiclastic rocks (phyllite, schist. Slate and quartzite). Local tuff breccias and dacitic lavas. Regional metamorphism (greenschist facies) and local thermal metamorphism (cordierite-hornblende facies).	2131 +/-10 (Day et al. 1995) U-Pb date on zircon separates from the Yuruari Formation
Pastora Supergroup	**Carichapo Group**		**El Callao Formation**	Low-K, high Fe basaltic to andesitic lavas. Greenschist to amphibolite facies metamorphism.	
Pastora Supergroup	**Carichapo Group**		**Cicapra Formation**	Submarine tuffs, graywacke turbidites, and volcanic siltstones, lithic tuffs, tuff breccias, agglomerates, and the upper part contain green chert, and porphyroblastic schist.	
Pastora Supergroup	**Carichapo Group**		**Florinda Formation**	Pillow basalts of tholeiitic to komatiitic composition.	

(from Day et al., 1995)
Note 1:Lithology for Greenstone Rocks of the Guyana Shield in Venezuela.

ALTERATION

Alteration of the original rock-forming minerals, such as amphibole and feldspar, and addition of elements such as boron and sulphur, is a result of hydrothermal, metamorphic, and weathering processes. The overprinting of these three processes has created a number of gradational alteration assemblages, which include varying amounts of quartz, secondary biotite, chlorite, sericite, calcite, epidote, metallic sulphides, tourmaline, magnetite, and minor fuchsite and anhydrite.

Hydrothermal alteration is most intense within the Blue Whale body, and in other isolated pockets of similar appearance scattered throughout the main mineralized trend. The alteration type of the breccia approaches a greisen, with components of phyllic alteration in the schist. In many cases within the breccia pipe, fragments have been completely replaced by tourmaline, and associated zones of quartz may be a result of tourmalinitization of feldspars. Petrographic analysis shows two separate phases of growth in some tourmaline crystals. Massive occurrences of sulphides typically show an earlier phase of pyrite formation with subsequent fracturing and infilling of fractures by chalcopyrite.

Weaker propylitic alteration is present in tuffaceous units surrounding the Blue Whale body as strong calcite+epidote+pyrite and calcite+chlorite+pyrite+epidote+chalcopyrite assemblages. Typically, in lenses of high Cu/low Au mineralization, the alteration package is more potassic (high secondary biotite+chlorite±sericite). Many veins with these alteration assemblages are highly deformed, indicating emplacement prior to metamorphism.

Metamorphic alteration occurs throughout the concession and is thought to be the result of regional burial. Petrographic analysis identifies both biotite grade and chlorite grade metamorphic facies, occurring in the lower mesozone and upper epizone, respectively. This corresponds to a temperature range of 300°C to 500°C, and hydrostatic pressures. The gold+pyrite±Cu disseminated lenses appear to be associated with fluids present during this metamorphic event. The primary orientation of schistosity is thought to be parallel to bedding, with a weakly developed secondary schistosity at about 10° to bedding. Some chlorite and epidote formation may be attributed to subsequent retrograde metamorphism. Overprinting this initial metamorphism is an alteration assemblage possibly related to a tensional event that resulted in the development of barren calcite±quartz veins.

Weathering has resulted in the breakdown of the above mineral assemblages according to their compositions, ultimately resulting in the formation of smectite, illite, and kaolinite. Pyrite is retained in the unoxidized material, though is typically very fine grained and sub- to euhedral, suggesting secondary formation. Chalcocite is present in areas of high copper. Above the water table iron oxides have formed after sulphide minerals, releasing free gold. The assemblage most resistant to this process is the Blue Whale breccia, due to the high silica and tourmaline content.

MINERALIZATION AT BRISAS

There are four distinct types of gold and copper mineralization present at Brisas, defined by geometry, associated minerals, and the gold-copper ratio. These zones are the Blue Whale body, disseminated gold + pyrite ± copper, disseminated high copper, and shear-hosted gold. Only the first three types are encountered within the proposed pit geometry. A more detailed description of the mineralization follows.

THE BLUE WHALE BODY

The Blue Whale mineralized body is a discrete, sharply bounded, flattened, cigar-shaped feature that trends more or less parallel to the local schistosity and plunges approximately 35° southwest. It outcrops in the Pozo Azul pit in the northeast portion of the Brisas concessions, and is intersected by 45 drill holes. It is 20 m in diameter at its widest point, and tapers off at depth. It is volumetrically a small fraction of the economically mineralized ground at the Siembra Minera Project, but it possesses the highest gold and copper grades.

Mineralogically, the Blue Whale is a sericite-tourmaline-pyrite-chalcopyrite-quartz schist, with a smaller volume of quartz-tourmaline-sulphide breccia. The schist is fine-grained and exhibits an almost complete alteration of the original rock. What appears to have been feldspar crystals and lapilli fragments are now replaced by tourmaline, and in some cases tourmaline bands occur in multiple deformed sheath fold structures. It is unclear whether the tourmaline itself has undergone this deformation, or if it has replaced minerals in a pre-existing structure. Thin quartz veins that cut the schist also show varying degrees of deformation, both brittle and ductile. Gold and copper grades are highly variable in the schist, normally increasing toward the contacts between the schist and the breccia. Pyrite/chalcopyrite is up to 25% of the rock mass, with abundant chalcopyrite and molybdenite.

The quartz-tourmaline breccia portion of the Blue Whale exhibits the highest gold and copper grades of the Siembra Minera Project. Tourmaline has completely replaced blocks of the breccia, while quartz has flooded the matrix. This rock does not show the strong ductile deformation of the sericite-pyrite-quartz schist. Chalcopyrite is the dominant sulphide, with lesser pyrite, bornite, covellite, and molybdenite. Other alteration minerals present are sericite, rutile, calcite, albite, siderite, and minor anhydrite (the latter occurring in undeformed, crosscutting veinlets).

DISSEMINATED GOLD+PYRITE± COPPER

The bulk of mineralization occurs in disseminated, coalescing, lensoid bodies, and high in gold and in most cases low in copper. These bodies lie almost exclusively in the lapilli-rich, rapidly alternating sequence of tuffaceous units and are clearly aligned along foliation. Together, these lenses form a generally well defined mineralized band, which mimics the dip of the foliation/bedding and remains open at depth. It remains at a similar thickness from the northern concession boundary for a distance of 1.4 km south, after which, it tapers rapidly. Alteration minerals characteristic of these lenses are epidote, chlorite, secondary biotite, and sericite.

The gold in the stratiform lenses is highly disseminated but only roughly associated with high occurrences of pyrite. Fine-scale sub-sampling of three metre assay intervals indicates good correlation between gold and small (<1 cm) calcite/quartz veins. Correlation also exists with zones of high occurrence of epidote, and in lapilli-sized lithic fragments that have been partially to completely replaced by epidote and sulphides. Sub-sampling evidence also suggests that gold is more evenly distributed through the rock near the center of the large mineralized lenses than it is near the margins. In section, east-west contours of gold grades at 0.75 g/t or 1.0 g/t show a geometry that essentially mimics contours drawn at 0.40 g/t.

DISSEMINATED HIGH COPPER/LOW GOLD

Stratiform lenses of high copper with or without high gold underlie the gold+pyrite lenses described above. These lenses outcrop in the northern part of the deposit, and plunge to the south in a manner similar to the Blue Whale and high gold/low copper lenses but with variable dips. Deep drilling has intersected these lenses as far south as 681,900 N. Within the stratigraphic column, these lenses generally occupy the TV and TVC-M units. Rock in the mineralized zones is characterized by a high degree of lapilli and crystal replacement by chalcopyrite, and in some cases, by bornite and covellite. High chalcopyrite in the rock matrix is often accompanied by high chlorite, secondary biotite, and in some cases molybdenite.

GOLD-BEARING SHEAR ZONES

Shear-hosted gold occurrences exist in the southern part of the concession, running parallel to the foliation as with mineralization further north. Stratigraphically, they occur above the large disseminated lenses previously described. The gold and copper grades are erratic and discontinuous.

MINERALIZATION AT CRISTINAS

The main two styles of mineralization present at Cristinas are:

1. Stratiform mineralization at Conductora, Morrocoy, and Cordova.
2. Hydrothermal breccia-hosted mineralization at Mesones-Sofia.

STRATIFORM MINERALIZATION

The Conductora (including Cuatro Muertos and Potaso), Morrocoy, and Cordova areas contain over 95% of the gold resource at Cristinas. Mineralization in these zones (here called Conductora-style mineralization) is stratiform in nature and is concentrated in volcaniclastic units within the mafic-to intermediate-composition volcaniclastic host sequence. The distribution of mineralization is controlled by the permeability of the host rocks; gold grade and alteration intensity typically decrease abruptly at the contact between permeable volcaniclastic units and impermeable lava layers, for example. Pre-mineralization, altered dioritic intrusive stocks are largely devoid of significant gold mineralization due to their low permeability.

Mineralization occurs in a greater than three-kilometre long, north-trending zone that dips moderately (30° to 40°) to the west, sub-parallel to the volcanic stratigraphy and to the pervasive (S1) cleavage. Gold mineralization is associated with a sulphide assemblage that consists essentially of pyrite and chalcopyrite.

Alteration mineral assemblages in Conductora are secondary biotite, minor potassium feldspar, calcite, chlorite and minor epidote and sericite. Calcite is ubiquitous, occurring mainly as disseminations, in addition; in carbonate-sulphide veinlets, carbonate-only veinlets, and quartz-carbonate veinlets. Silicification is minimal in Conductora-type mineralization. Minor tourmaline disseminations occur in some parts of Conductora, but in much lower concentrations than in the Mesones-Sofia area. The most consistent gold mineralization occurs in zones in which secondary biotite is most intensely developed. Many sulphide clots within these biotite-dominated alteration zones are rimmed by a green chlorite alteration that has overprinted the secondary biotite.

Pyrite and chalcopyrite constitute the only sulphide species of significance in primary ore. The average pyrite/chalcopyrite ratio is greater than five. Sulphides occur principally as disseminations, but also in narrow veinlets 1 mm to 2 mm wide. These veinlets are variable in composition ranging from sulphide-only to sulphide-calcite and sulphide-calcite-quartz.

These veins have selvages of secondary biotite, chlorite, or chlorite-epidote.

Quartz-sulphide veins are rare, but where they do occur, they are in zones of intense secondary biotite development against which they have indistinct margins and are associated with multi-ounce gold values. Higher than average gold grades (>2 g/t) are associated with areas in which pyrite occurs as coarse clots up to 2 cm in diameter in zones of intense secondary biotite alteration. Generally, however, the sulphides are fine-grained, and much more so than in Mesones-Sofia.

Molybdenite is locally quite abundant, occurring in quartz-calcite-sulphide veinlets, and disseminated with pyrite and chalcopyrite. The Potaso area contains disseminated molybdenite that appears to have no spatial relationship with pyrite and chalcopyrite on a hand-specimen scale.

QUARTZ-TOURMALINE-SULPHIDE-CALCITE VEIN BRECCIAS

Mineralization in Mesones-Sofia is concentrated in the quartz-tourmaline-sulphide-calcite vein breccias and extends laterally into the adjacent country rocks. The breccias are sufficiently closely spaced that the country rock between them also constitutes ore in the central part of Mesones-Sofia. Grades in the country rock on the periphery of the system decrease as the distance between the breccias increase.

Breccias consist of quartz, tourmaline, calcite, and sulphides, and the country rock alteration assemblage consists of fine-grained quartz, muscovite (sericite), calcite, tourmaline, and disseminated clots of sulphides. Silicification is variably developed, with pervasive silicification largely confined to the breccias where it encapsulates the sulphides. Muscovite gives way to secondary biotite in the deepest intercepts in Mesones-Sofia. The occurrence of relict laths of biotite within intensely sericitized zones, as well as relict biotite in the central parts of larger muscovite laths, provides evidence that muscovite replaced pre-existing secondary biotite in the upper parts of the Mesones-Sofia hydrothermal breccia system. Patchy potassium-feldspar alteration is evident in the central part of Mesones-Sofia.

Sulphides commonly occur in aggregates up to 5 cm in diameter at Mesones-Sofia. Sulphides also occur as semi-massive replacements in the matrix of the quartz-tourmaline breccias and as disseminations both in the breccias (in the matrix and in breccia clasts) and in the enclosing country rocks. Sulphide content in primary, hard-rock ore is 5% to10% with a pyrite/chalcopyrite ratio of less than 5. Pyrite and chalcopyrite are the only common sulphides in Mesones-Sofia; molybdenite is scarce, but where it does occur, it is associated with pyrite and chalcopyrite. There is evidence that chalcopyrite gives way to pyrite upwards in the breccia bodies. For example, breccia bodies at Morrocoy, located structurally 200 m above Mesones- Sofia, have similar overall sulphide contents but contain only a minor proportion of chalcopyrite. There is no appreciable difference in the nature and distribution of sulphides, sulphide species, or grade, between the muscovite- and biotite-dominated alteration assemblages. This implies that the majority of the mineralization was in place by the time that secondary biotite was overprinted by muscovite.

OTHER MINERALIZATION

Discrete auriferous quartz veins are located adjacent to the Cristinas deposit. Such veins include the Los Rojas and Albino veins that lie approximately one kilometre to the east of the Conductora area, and the Hofman vein, which lies about one kilometre to the west of the Cordova area. These veins consist of quartz with gold mineralization associated with pyrite (there is no appreciable chalcopyrite). The veins have chlorite selvages about 50 cm wide. Although gold mineralization in these veins does not constitute part of the Cristinas resource, they are considered to be genetically related, and peripheral, to the Cristinas deposit. This type of mineralization is not discussed further in this report.

8 DEPOSIT TYPES

BRISAS

The Blue Whale body has been interpreted to be structural feature, a dilational zone of weakness that has acted, at some point after deposition of the tuffaceous rocks, as a conduit for mineralizing fluids. Based on structures seen within the Blue Whale, this occurred before or during regional metamorphism. The initial pulse of mineralization probably occurred when the system was relatively young. Brecciation, on a limited scale, took place along a pre-existing fracture with fluids rich in B, Cu, Au, and lesser Mo. Alteration in and directly around this feature was intense, causing complete replacement of breccia fragments by tourmaline, massive quartz, and copper.

A possible deposit analogy is of a copper porphyry forming over a magmatic source (yet to be discovered) that was very rich in boron. A peraluminous granite might fit the boron requirements and a sufficient volume of basaltic/andesitic rock could provide the copper. Thin lenses of high Cu and Mo extending away along bedding/foliation planes could be the result of periodic high confining pressures within the Blue Whale that forced mineralizing fluids outward along these planes. The fluids replaced crystals and lithic fragments, evidence of which can be viewed in drill core.

The bulk of gold mineralization at the Brisas deposit appears to have been emplaced after formation of the Blue Whale mineralization. It occurs over a wide area and the highest gold grades do not occur in proximity to the Blue Whale. Although on a small scale gold appears to link with zones of higher schistosity and development of alteration minerals, on a larger scale it was deposited in favourable lithologic hosts, comprising mostly thin and variable tuffaceous rocks. Improved permeability related to bedding discontinuities and relatively porous lithic fragments may have been the overriding factor in mineralization deposition. The fluid pressures must have been high to disseminate them through an unfractured volcanic pile rather than along obvious shear planes or fractures. Mineralogically, this phase of deposition bears some similarity to the high temperature B, Cu, Au, and Mo fluid phase proposed for the Blue Whale, specifically in regards to the formation of disseminated lenses. Geometrically, this package of lenses plunges to the south, where it can still be detected by deep drilling. This pattern is similar to what is observed in the Blue Whale.

CRISTINAS

In terms of classification, Cristinas has been assigned to shear zone-hosted systems by some geologists, and to a porphyry association by others; however, several key elements of the Cristinas deposit must be satisfied in any attempt to classify the deposit. These include:

- Hydrothermal quartz-tourmaline breccias are present at the core of the mineralized system. Although the quartz-tourmaline breccias contain less than 5% of the Mineral Resource at Cristinas, the concentric arrangement of alteration facies record a decrease in hydrothermal fluid temperatures away from the breccias, showing that these constituted the core of the hydrothermal system. The quartz-tourmaline breccias are features that cross-cut stratigraphy and are clearly not shear zone related. Similar breccias are common in porphyry environments, and some show alteration and metal zoning similar to that observed at Mesones-Sofia, such as secondary biotite at depth to quartz-sericite at shallower levels, as well as a decrease in chalcopyrite upwards within the breccias.
- Alteration zoning at Cristinas is similar to that associated with porphyry systems with proximal secondary biotite giving way to distal chlorite-epidote-calcite (propylitic) facies. Quartz-sericite alteration is superimposed on other alteration facies and is likely to have resulted from the draw-down of meteoric water as the prograde hydrothermal system collapsed.
- The metal association of gold with copper and minor molybdenum is similar to porphyry systems and is not common in shear zone-related gold systems.

Despite these factors that are typical of porphyries, Cristinas clearly is not a classic porphyry system, since mineralization is not contained within, or closely associated with, any porphyritic intrusive stock. Furthermore, the abundant quartz veining associated with most porphyries is largely absent from the Cristinas deposit.

9 EXPLORATION

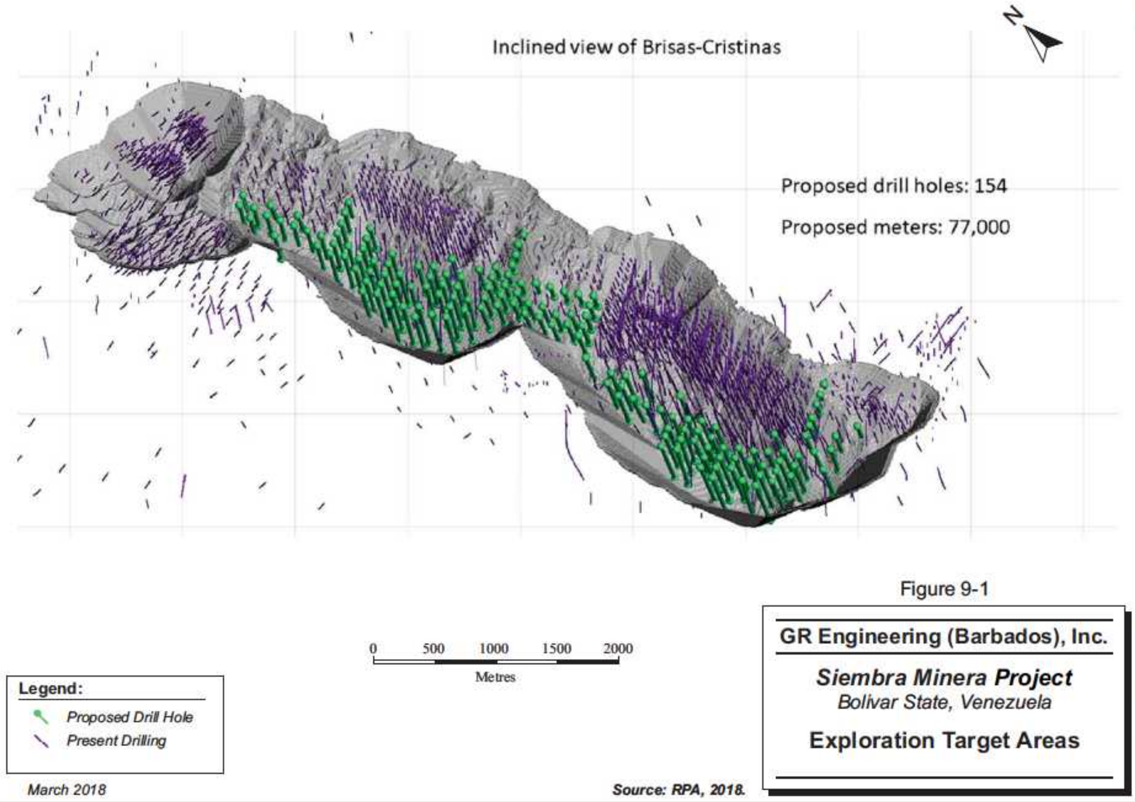
The history of exploration work completed on the Project is described in Section 6 History.

EXPLORATION POTENTIAL

The Siembra Minera mineralization is open down dip in all zones and along strike to the northwest in Morrocoy and Cordova because of insufficient drilling. Current resource pit shells are limited by drilling instead of economics. Current plans for exploration are based on brownfield expansion of the existing deposit. RPA is of the opinion that there is excellent potential to increase the resources and to convert a significant portion of the Inferred Mineral Resources to Indicated with more drilling. RPA recommends drilling approximately 150 to 200 drill holes totalling approximately 75 km to 100 km. This drilling would have a number of objectives including:

- Conversion of Inferred Mineral Resources to Indicated with priority set on Inferred Mineral Resources situated in the 5 and 10 year pit shells.
- Drilling to determine the extent of mineralization at depth in the Main Zone as this will determine the limits of the largest possible pit and help with the location of features such as dumps and roads.
- Better definition of the copper mineralization in the Main Zone footwall.
- Improving preliminary artisanal mining sterilization assumptions.
- Condemnation drilling of proposed waste rock storage sites.
- Closer spaced drilling in the El Potaso area between Brisas and Cristinas.
- Drilling on the northwest extensions of the mineralization in the Morrocoy and Cordova areas.
- Drilling on the Cristinas Main Zone for density measurements.

Figure 9-1 illustrates some of the exploration targets. Most of the drill holes target down dip extensions of the Main Zone and the boundary area between the Brisas and Cristinas concessions known as Potaso. The average length of these holes has been estimated to be approximately 500 m. An approximate cost for this drilling ranges from approximately $15 million to $20 million.



March 2018

Source: RPA, 2018.

Figure 9-1

GR Engineering (Barbados), Inc.

Siembra Minera Project

Bolivar State, Venezuela

Exploration Target Areas

RPA

www.rpacan.com

10 DRILLING

GENERAL

GRI began exploration activity on the Brisas concessions in late 1992, and continued with various drilling programs through 2006. A total of 975 drill holes with a total drilled length of 207,442 m have been completed by GRI on the Brisas concessions as of September 2006 (Table 10-1). Of these holes, 802 representing 189,985 m of drilling were completed specifically for exploration. The remaining holes were drilled for hydrologic, geotechnical, and metallurgical testing, and in some cases, were assayed and used in modelling.

Drill holes within and around the planned pit area are mostly spaced at approximately 50 m or 100 m apart. Drill hole spacing in the Blue Whale area is approximately 25 m. The majority of the exploration drilling was performed using standard diamond core-barrel recovery techniques although a small amount of auger drilling was carried out at the beginning of the exploration campaign. Auger holes (“A” holes) are generally very shallow, located throughout the Project area and in particular between later-drilled core holes; many auger holes are outside the pit area.

Placer drilled 1,182 holes on the Cristinas concessions between 1994 and 1997 (Table 10-2). Once early exploration drilling determined the approximate location and strike direction of mineralization, further drilling was undertaken on section lines orientated perpendicular to that trend. The drilling completed in the southern two-thirds of the Cristinas concessions has shown that the mineralization occurs in a large tabular body that strikes approximately north-south, and dips moderately to the west. The drilling completed in the northern third of the Cristinas concessions has shown that the strike of the mineralization has changed in this area. In this northern portion of the Cristinas concessions, the mineralization can occur as pipe-shaped forms, and as thinner tabular forms with sub-vertical dips.

TABLE 10-1 SUMMARY OF GRI DRILLING-BRISAS CONCESSIONS

GR Engineering (Barbados), Inc. – Siembra Minera Project

Year	No. Holes	Total Metres	No. Assays
1992	-	-	-
1993	49	5,828	1,921
1994	130	16,091	5,479
1995	98	18,859	6,308
1996	259	52,159	17,359
1997	214	66,353	21,803
1999	13	5,726	1,833
2003	9	1,822	1,103
2004	101	24,448	5,820
2005	37	10,866	3,262
2006	65	5,291	
Drill Hole Total	975	207,442	64,888
Trench	4	60	36
Trench Total	4	60	36
Grand Total	**979**	**207,502**	**64,924**

TABLE 10-2 SUMMARY OF PLACER AND CRYSTALLEX DRILLING-CRISTINAS CONCESSIONS

GR Engineering (Barbados), Inc. – Siembra Minera Project

Year	No. Holes	Total Metres	No. Assays
Placer			
1992	165	8,474	8,461
1993	201	29,998	30,146
1994	383	53,754	56,559
1995	269	34,166	32,669
1996	148	24,160	26,610
1997	16	4,901	5,104
Placer Total	**1,182**	**155,454**	**159,549**
Crystallex (Data Unavailable)			
2003	12	2,199	1,079
2004	18	7,131	5,993
2005	14	5,419	5,419
2006-2007	46	13,574	12,178
Crystallex Total	**90**	**28,323**	**19,769**

Crystallex drilled 90 holes on the Cristinas concessions from 2003 to 2007. The information from these drill holes were used by MDA in 2007 to prepare the update of the Mineral Resource estimate and were discussed in summary form in MDA's accompanying Technical Report (MDA, 2007). While the significant intersections from these drill holes were reported in news releases, insufficient details regarding the exact location and inclination of the Crystallex drill holes or the individual assay results were presented in these news releases to be useful. As the results from this drilling campaign were not available to RPA and the information could not be reconstituted from the news releases, none of the drill holes completed by Crystallex were used in preparing the current Mineral Resource estimate.

The drill hole locations are shown in Figure 10-1.

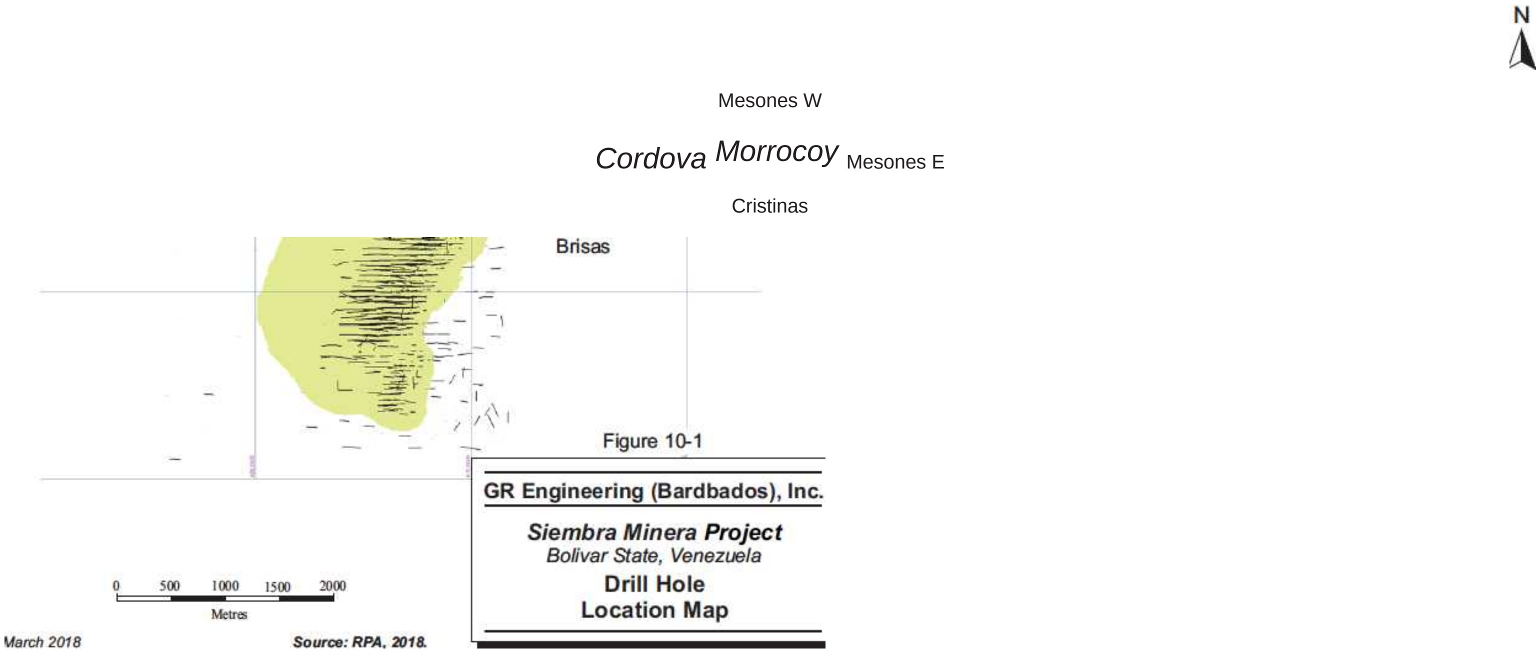


Figure 10-1

GR Engineering (Bardbados), Inc.

Siembra Minera Project
Bolivar State, Venezuela

Drill Hole Location Map

BRISAS CONCESSIONS

The following is taken from PAH (2005 and 2008).

COLLAR AND DOWNHOLE SURVEYING

For all auger and core holes at the Brisas Project, the field location of the drill hole collars before drilling and collar surveying after drilling have been performed by Surco, a local survey firm based in El Callao, Venezuela. This company was also responsible for establishing the concession boundaries and setting up permanent survey reference points within the concession. The base for all surveys was GPS defined and checked by the survey company with a traverse from a nearby GPS station (Las Cristinas) with satisfactory accuracy.

The setting up of the bearing and inclination of the drill rig was made with compass and inclinometer. All core holes were surveyed with a Sperry Sun photographic instrument mounted inside a rod that can be inserted into the drill hole using the drill equipment, recording azimuth and dip at varying depths by technicians employed by GRI. The first photo was normally taken at a depth of approximately 20 m (without casing), a second photo at 6 m below the cased intervals (below the saprolitic zone), and subsequent photos every 100 m to 150 m thereafter. The reading on the developed film was checked by a geologist and the information entered into a field book.

CORE LOGGING

The logging format for the Brisas Project had several changes through the different drilling stages as adjustments to Rock Quality Designation (RQD) measurements and standardization of lithologic and alteration codes were made. The code standardization was implemented after drill hole D95, and many of the previous holes were re-logged to avoid differences in log codings. Two different log forms, geotechnical and geological, were completed when logging.

The geotechnical log completed for each hole included depth, bit diameter, core recovery, rock hardness, sampling intervals, and RQD. Core recoveries were generally good averaging approximately 96%. An average recovery of 87% was obtained in saprolite, and 98% in hard rock. The core recovery for the Blue Whale was 91%. RQD, as measured by GRI, is the ratio between the cumulative length of naturally un-fragmented/un-fractured core longer than 0.1 m and the total core length within a 3.0 m standard measurement interval. RQD readings were obtained before sampling and/or destruction of the core and recorded in the logs. Drilling was normally performed with HQ diameter (2.5 in. or 6.35 cm) to the saprolite-hard rock contact where bits were changed to NQ diameter (1-7/8 in. or 4.76 cm). Due to the characteristics of saprolite and other intensely weathered rock, RQD readings were not made above the hard rock-saprolite contact.

Detailed geological logs were recorded on a form with the following information:

- hole number, summary of location (coordinates, elevation), drilling company, geologist, and date;
- log description, including rock type, degree of oxidation and weathering, intensity and type of alteration, sulphide mineralization, veining or other structure(s) (jointing, fracturing and brecciation), and rock color. Alteration and mineralization minerals were quantitatively estimated and recorded, as well as all other parameters, for computer input.

A summary log was then completed from the detailed information, along with a graphical interpretation of the log, as well as gold and copper assay results. Logging procedures followed by GRI were well established and have been followed by all geologists, with minor changes, through the different exploration stages. Quality was assured through the use of an internal manual: “Procedures for geological logging at Brisas del Cuyuni”, which provides guidance in the use of geological terms, defines different lithological units, structure and visual evaluation of alteration and mineralization contents.

TWIN DRILLING VERIFICATION

Twin hole tests were run occasionally throughout the Brisas Project drilling program. A total of seven twin holes were drilled at different times and locations within the property. Both the initial and the twin were core holes. A more detailed discussion of twin hole data results is presented in Section 12 of this Technical Report.

CONDEMNATION DRILLING

Condemnation drilling has been performed extensively on the Brisas concessions. Both condemnation and geotechnical drilling has been performed on the proposed waste dump areas and plant site. Geotechnical drilling was conducted on the proposed tailings dam area for which some assay information was obtained. None of the drilling of these areas has yielded geological or geochemical information suggestive of the presence of significant mineralization, and therefore no additional condemnation drilling was recommended for Brisas by PAH (2008).

RPA recommends investigating if additional condemnation drilling will be required for the Siembra Minera Project.

SAMPLING

In auger drilling, each three-metre auger flight was lifted onto a table and the soft saprolite was peeled off, dried, and prepared for assaying. In core drilling the soft saprolite was cut longitudinally by machete and the hard rock core cut by a standard Clipper 12-inch diamond saw. Half of the core was placed back in the core box for storage while the other half was placed in metal trays for drying in a fuel oil boiler for sample preparation. Most core drilling was done with HQ core (63.5 mm diameter) but deeper holes were sometimes reduced to NQ (47.6 mm diameter) to accommodate the depth capacity of the drill rig.

GRI maintained a full record of split core for the entire drill program. The sampling interval was generally three metres, with the exception of samples adjacent to the saprolite-hard rock contact, where in some cases adjustments were made to differentiate sample types, or in a few holes located in exploration areas outside the main mineralized zone (e.g., D722-D727), where a one-metre interval was used. The sample size was nominally eight kilograms in weight for the three-metre sample.

The gold and copper mineralization at the Siembra Minera Project is broadly disseminated and amenable to bulk mining. The deposit is proposed to be mined on six-metre benches in ore zones and 12 m benches in waste zones. In RPA's opinion, the three-metre sample length is adequate and generally provides sufficient resolution in defining the ore and waste zone boundaries for the mineralization except perhaps for the Blue Whale zone, which tends to be narrower and of higher grade than the rest of the deposit and hence, a shorter interval may have allowed for better boundary definition. On the other hand, the longer interval will tend to incorporate some dilution to the model.

RPA is of the opinion that the drilling, sampling and logging procedures carried out on the Brisas concessions meet industry standards, and are suitable for use in the preparation of Mineral Resource estimates.

CRISTINAS CONCESSIONS

The following is taken from MDA (2007).

COLLAR AND DOWNHOLE SURVEYING

According to historic Placer documentation of the drilling procedures, drill hole locations were established using a prismatic or Brunton compass, and adjusted into position with a Brunton compass. After completion, each hole was fitted with a collar pipe, and a cement collar block was inscribed with the drill hole number. Final drill hole collar locations were then surveyed in UTM coordinates by Surco, translated into local grid coordinates, and entered into a GEOLOG database (a proprietary drill hole database format). Examination of the drill hole deviation measurements shows that 907 of the 1,174 holes (77%) have at least one downhole survey. Downhole survey readings, generally taken approximately every 50 m, were completed using a Sperry Sun single-shot survey camera or a Pajari compass. The GEOLOG database contains detailed geological descriptions, geological codes, check assay data, specific gravity data, core recovery, RQD data, and some trace element geochemical data.

CORE LOGGING

Drill core was logged for rock type, alteration, mineralization, structure, and magnetic susceptibility. In addition, RQD, core recovery, rock strength, and joint roughness and coating were logged. If core recovery in the saprolite averaged less than 80%, the hole was re-drilled at the contractor's expense; global average core recovery in saprolite was between 85% and 90%. Hard-rock core recovery was above 95%. Oriented core was drilled in selected areas using a downhole "crayon test" for determining the true orientation of foliation, bedding and lineation, as well as the orientation of veins and veinlets.

SAMPLING

Drilling in an intensely weathered tropical environment presented challenges and, consequently, several different drilling techniques were attempted by Placer before choosing triple tube diamond drilling. Other methods tested include Vibracore, auger, and reverse circulation rotary drilling, none of which produced acceptable results. Up to seven hydraulic diamond drill rigs were used simultaneously to complete the drilling. The best recovery was achieved with PQ tools (85 mm diameter) in saprolite, and with HQ tools (61 mm diameter) in bedrock. HQ was also used to drill some of the saprolite, as not all rigs were equipped to handle PQ (85 mm diameter) core. NQ (47.6 mm diameter) was used systematically in bedrock during the infill drilling phase within the Stage I pit area and occasionally in difficult drilling situations. The saprolite interval was drilled uncased until casing could be set in bedrock. Sample lengths ranged from 0.1 m to 8.0 m, with most being approximately one metre.

RPA is of the opinion that the drilling, sampling, and logging procedures carried out on the Cristinas concessions meet industry standards and are suitable for use in the preparation of Mineral Resource estimates.

11 SAMPLE PREPARATION, ANALYSES AND SECURITY

BRISAS CONCESSIONS

The following is taken from PAH (2005).

SAMPLE PREPARATION

Sample preparation, including drying, crushing and pulverizing, was performed on site at GRI's own sample preparation facility using the sample preparation routine summarized in Figure 11-1. The sample pulps were shipped to assay laboratories in Puerto Ordaz, Monitor Geochemical Laboratory de Venezuela, C.A. (Monitor) and Triad Laboratory (Triad), during the earlier campaigns before 1999. The Triad laboratory located at Minera Hecla's La Camorra mine site was used for the later round of drilling (2003-2004). After drying, all samples were crushed to 90% –8 mesh (2.36 mm). Half of the crushed sample was bagged and sorted for reference; and a split of approximately 500 g from each sample was pulverized to 90% –150 mesh (0.106 mm). Crushing was carried out with 6x4-inch Morse and 4x8-inch Marcy jaw crushers and a roller crusher. Pulverizing was accomplished with Bico puck and ring pulverizers, although Bico disk pulverizers were also available. Pulverizer cleaning with barren sand was performed after every ten samples. Quality assurance/quality control (QA/QC) procedures included sending pulps to Acme Labs in Vancouver for checking one of every 20 samples and inserting standards prepared with the Brisas mineralized material by Hazen Labs at a rate of 1 in every 30 samples.

Assay laboratories used during the early stages of the Brisas Project drilling were Barringer Research Labs (Barringer) and Bondar Clegg Labs (Bondar Clegg). Monitor was used as the primary assay laboratory and Triad was used as the check assay laboratory from 1994 to 1999 when checks established confidence for these two local laboratories based then in Puerto Ordaz, Venezuela. For the 2003-2004 drilling program, Triad located at Minera Hecla's La Camorra mine site was used. During that time period, Triad worked with Acme Labs in Vancouver, Canada, for check assaying purposes and also participated in international round robin assaying programs.

For the 2003-2004 drilling program, samples were prepared on site and pulps sent to Triad. The laboratory routinely ran samples for Hecla's La Camorra mine where it was located, as well as for other companies operating in Venezuela. Assaying control procedures included log record and tag identification of samples, a control list, blank and rejects run on approximately 10% of samples, assay check runs on approximately one of every 15 samples. Both gold and copper assaying were performed using standard fire assay (FA) and atomic absorption (AA) techniques for the Brisas Project. RPA toured the Triad laboratory during a visit to La Camorra in May 2005 and found it to be reasonably well operated. Triad had begun the accreditation process and participated in bi-annual round robin assaying of Geostats reference standards. In 2005, Triad was registered in Arizona as a fire assay laboratory and had operated in Venezuela for twelve years, including the past five years at the La Camorra mine site.

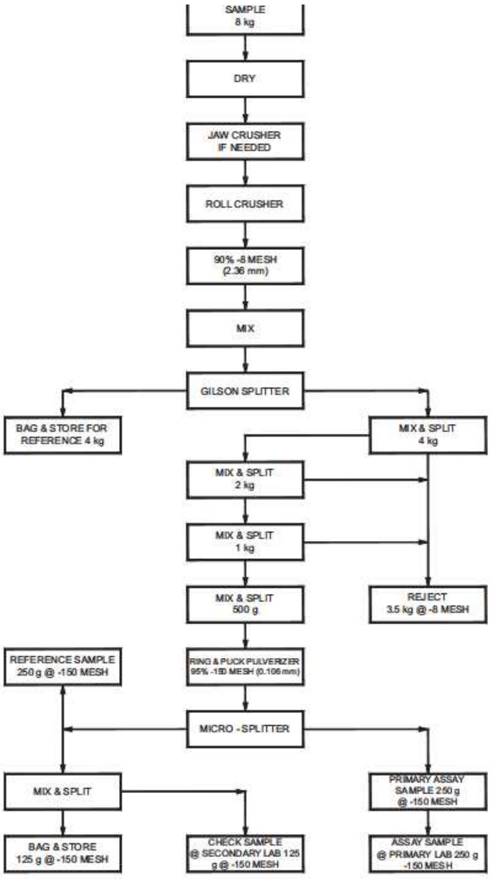


Figure 11-1

GR Engineering (Barbados), Inc.

Siembra Minera Project
Bolivar State, Venezuela

Sample Preparation Flow Sheet, Brisas Concessions

March 2018

Source: Pincock, Allen & Holt, 2006.

ANALYTICAL METHODS

Analytical methods used for the early stage of the drilling programs were metallic screen analysis for gold and geochemical analysis for copper. During 1994 to 1999, all pulp samples were analyzed for gold by FA with an AA finish. Samples with gold values over 1.5 g/t Au were re-assayed with 1.0 assay ton FA with a gravimetric finish. Copper assay was performed using standard AA with long iodide titration verification when values were obtained above 0.3% Cu.

QUALITY ASSURANCE AND QUALITY CONTROL

The Siembra Minera Project generated a large amount of assay information consisting of original assays, checks, and standards that were routinely received. These data were kept both in original hardcopy and in digital format. Assays were checked for correct sample number, intervals, actual values from the laboratories, and finally for conflicts within the primary laboratory, and between the primary laboratory and the check laboratory. If samples had conflicts (i.e., [A-B]/[A+B] > 33% variance), they were reviewed and if necessary the laboratories were requested to re-assay. In some cases, there were up to five check assays for a given sample interval for several high-grade gold assays. For standards, the tolerance was a variance of 12% for both copper and gold. For drill holes with serious standards conflicts, the entire drill hole could be requested to be re-assayed. Once the conflicts were resolved, all assay data were kept in an "Accepted Assays" spreadsheet under the control of the project manager. The analysis of assays through the use of the spreadsheet as a control provided a reliable method of determining conflicts between primary and check laboratories. This method was designed by GRI in 1995 with subsequent audits and modifications by independent parties (Mark Springett 1995, Behre Dolbear 1997). The actual assay value included in the drill hole database and utilized in modelling is the average of all accepted assays for a given sample interval.

DENSITY MEASUREMENTS

From 1994 to 1997, there was an ongoing program totalling hundreds of field measurements of bulk densities and moisture contents. The following methods were used for bulk density measurements:

- Method 1: known volumes were excavated from road or trench cuts and weighed;
- Method 2: boxes of drill core were weighed and adjusted for the box weight and estimated volume of core loss; and

- Method 3: individual core pieces were weighed and volumes estimated by volume displacement.

Moisture content was measured by weighing new core, drying it overnight, and re-weighing it. In-place, dry bulk densities and moisture content for different rock/alteration types were compiled by GRI for resource/reserve studies based on all valid information using a weighted average method (Table 11-1). The densities were used in the January 2005 feasibility study and subsequent studies and are grouped by rock type and degree of weathering. The main groups are oxide saprolite, sulphide saprolite, and un-weathered rock. An additional category was created for schist, because it consistently had a lower density for un-weathered rock than other rock types that are generally considered as "tuff."

TABLE 11-1 MATERIAL DENSITIES AND MOISTURE

GR Engineering (Barbados), Inc. – Siembra Minera Project

Description	Oxide Saprolite	Sulphide Saprolite	Hard Rock
Bank Wet Density (t/m^3)	1.88	2.07	2.82
Moisture Percentage (%)	23.0	16.0	1.0
Bank Dry Density (t/m^3)	1.45	1.74	2.79

CRISTINAS CONCESSIONS

The following is excerpted from MDA (2007).

SAMPLE PREPARATION

Although sample preparation and analytical procedures are well described in Placer's reports, it is not clear what special security procedures were in place at that time. The Triad laboratory of Tumeremo, Venezuela, and Bondar Clegg, Vancouver, Canada assayed all samples taken at Las Cristinas in 1992. Beginning in January 1993, Placer Research Centre in Vancouver, Canada, assayed all core samples, while Monitor analyzed trench samples.

All samples were prepared on-site. In 1993, staff from Placer Research Centre reviewed and amended laboratory procedures to conform to Placer standards. Figure 11-2 shows Placer's sample preparation procedures.

ANALYTICAL METHODS

All samples were fire assayed for gold and "geochemically" analyzed for silver, molybdenum, copper, and cyanide-soluble copper. Table 11-2 shows the assay techniques used on Las Cristinas samples by the Placer Research Centre. Note that the term "geochem" was not explained.

TABLE 11-2 SUMMARY OF PLACER'S ASSAYING PROCEDURES, CRISTINAS CONCESSIONS
GR Engineering (Barbados), Inc. – Siembra Minera Project

Laboratory	Element	Method
Placer Research Centre	Au	Fire Assay, AA finish[1], 25 g sample
Placer Research Centre/ Bondar Clegg/Triad	Ag	Geochem, AA finish[2]
Placer Research Centre/ Bondar Clegg/Triad	Cu	Geochem, AA finish[3]
MINEN	CNSCu[4]	Cyanide Leach
Placer Research Centre/ Bondar Clegg/Triad	Mo	Geochem, AA finish[5]

Notes:

1. Au > 3 g/t were re-analyzed with a gravimetric finish.
2. Ag > 10 g/t were re-analyzed using same analytical procedures.
3. Cu > 4,000 ppm were re-analyzed using same analytical procedures.
4. CNSCu is cyanide soluble copper.
5. Mo > 1,000 ppm were re-analyzed using same analytical procedures.

In addition to the above elements, core samples collected early in the program were analyzed for mercury, antimony, arsenic, zinc, and lead. Multi-element analysis was also performed on 3,700 surface samples. Additional multi-element analyses were completed on five metre downhole composites from ten holes drilled on cross section 9,600N in the Conductora deposit.

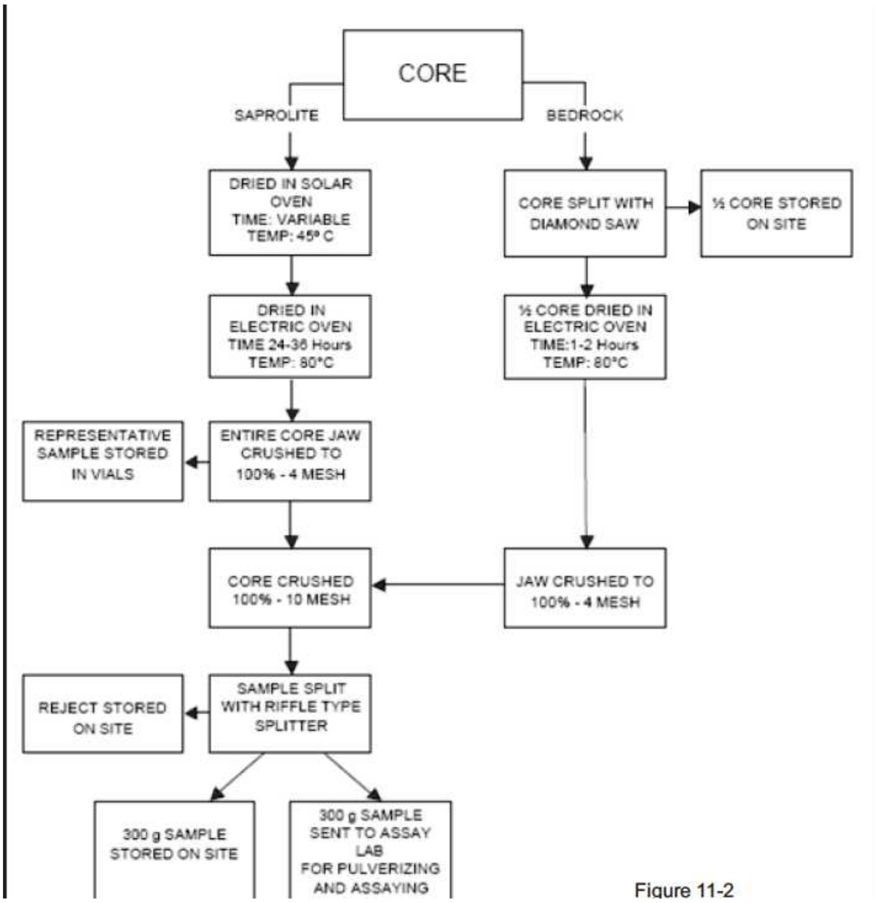


Figure 11-2

GR Engineering (Barbados), Inc.

Siembra Minera Project
Bolivar State, Venezuela

Sample Preparation Flow Sheet,
Cristinas Concessions

March 2018

Source:Mine Development Associates, 2007.

QUALITY ASSURANCE AND QUALITY CONTROL

In July 1993, R. Mohan Srivastava conducted an inter-laboratory bias analysis to compare Triad, Bondar Clegg, and Placer Research Centre assay results. The study concluded that the Triad results tended to be biased on the low side, while some of the Bondar Clegg results tended to be biased on the high side. Consequently, it was decided to re-assay all Triad and Bondar Clegg samples for gold only at the Placer Research Centre and to use only Placer's gold assays on drill core for the 1996 resource study.

Standard, duplicates, and blanks were used for quality control of the on-site sample preparation laboratory. For every suite of 20 samples, there was one each of a duplicate, standard, and blank, which were submitted as blind samples to the assay laboratory.

Thirteen standards were prepared by the Placer Research Centre representing a broad range of gold grades from Las Cristinas surface and core material. These were used to monitor accuracy of the assay laboratory as well as to detect potential contamination in sample preparation. Duplicates were taken from a split of the preceding sample and were used to test the precision of the assays and the homogeneity of nugget effect of the samples. Blanks were obtained from a nearby diorite quarry and were used to detect possible contamination during sample preparation as well as to verify sample order.

Standards, replicate samples on the same sample pulp, and blanks were also used for the quality control program for gold assays at the Placer Research Centre. In each suite of 24 samples, one each was a replicate, standard, and a blank. According to Placer, quarterly statistical evaluations of the QA/QC data indicated that Placer's laboratory produced accurate and precise gold assay results. Results from a geochemical quality control program also indicated that the Placer Research Centre's geochemical analyses for copper, silver, and molybdenum were highly accurate and precise.

In addition, 10% of the samples were sent to an outside laboratory for an independent check; the laboratory was the IPL laboratory (IPL) of Vancouver, Canada. Of the 5,866 samples analyzed from 1993 to 1995, the two data sets were quite similar with minor differences between the two laboratories especially for gold grades less than 1.0 g/t Au, according to Placer's 1996 feasibility report. The average inter-laboratory bias appeared to be approximately 5% to 10%, with Placer's laboratory results being higher than IPL's. The Placer 1996 feasibility report noted that this grade range was important because the economic cut-off for the project is between 0.6 g/t Au and 0.7 g/t Au. That report stated that *"It appears that the PDI laboratory is providing more reliable assays of the less than 1.0 g/t Au gold grades than is the IPL laboratory. IPL appears to be understating the gold grade of the less than 1.0 g/t Au grades by about 5% to 10% …. From this analysis the PDI assay results can be considered appropriate for the resource estimation."* MDA (2007) was unable to definitively analyze and compare the samples and check samples to verify Placer's above conclusion.

QA/QC information was also gathered on assay samples from the trenching program; these samples were assayed by Monitor. The Placer Research Centre helped Monitor implement in-house standards and also completed a check assay program on samples sent to Monitor. A 1995 evaluation indicated that it appeared Monitor's assays were on average 5% to 10% higher than the expected means of the standards' values and that Monitor's mean gold grades were approximately 7% higher than Placer's mean gold grades on trench samples assayed by both laboratories. Placer's 1996 feasibility study concluded that *"The systematic bias in the Monitor assay results presented above is not thought to have a significant impact on the 1996 Conductora/Cuarto Muertos resource estimate because the trench data are only a small part of the data base used for resource estimation."* A similar check on Monitor's results from the 1998 trenching program showed that Placer results were approximately 3% lower than the Monitor results.

Monitor also assayed all the Mesones-Sofia drill core from the 1996 drilling, which represents approximately 55% of all the assays in the Mesones-Sofia area. Placer's 1998 feasibility study reported that, as with the trench samples, Monitor's drill core assays appeared to be approximately 5% to 10% higher than check assays by Placer Research Centre. This problem was to be studied further, but MDA (2007) was not aware of any further reported conclusions.

Diamond drilling in the intensely weathered environment, i.e., saprolite, presented potential sample bias (Placer used the term "contamination" and considered it similar to that encountered in wet reverse circulation drilling; to be consistent with Placer's terminology, the same wording will be used here). Crystallex and MDA noted that this was particularly apparent at Mesones-Sofia, where chunks of siliceous or tourmalinized hard rock were floating in the saprolitic clays. During drilling, water flowing around the core could wash out the clays, relatively increasing the amount of hard, possibly better-grade material.

Placer's care for this aspect of sampling is reportedly excellent. While great effort was made to eliminate "contamination", occasional contaminated intervals were unavoidable, according to Placer. Placer stated that *"Suspected contaminated intervals greater than 20 cm were sampled and logged as discrete intervals. If the contaminated interval was less than 20 cm the interval was marked and photographed in place and then removed prior to sampling. All sampled intervals were assayed for gold, copper, and molybdenum in order to assess the potential for additional unrecorded downhole contamination on a case by case basis. A total of 831 samples deemed to potentially be contaminated were eliminated from use by coding. The mean grade of these "contaminated" samples is 3.13 g/t Au with a maximum of 29.73 g/t Au. In addition, 32 trench samples deemed to potentially be contaminated were also eliminated from use in estimation"*.

MDA evaluated the "contaminated" samples by selecting all samples lying within the area where "contaminated" samples exist. Descriptive statistics were calculated on all "contaminated" and "not-contaminated' samples. The results showed that there is a large discrepancy in mean grades between the two sets of data for gold, silver, and copper. MDA capped the outlier samples to evaluate if the differences were caused by these few high-grade samples, but the results remained the same. Placer's elimination of these "contaminated" samples was justified, and MDA (2007) continued with the practice of not using these samples.

REVIEW OF THE QA/QC RESULTS

QA/QC PROCEDURES

QA/QC data for the Cristinas and Brisas deposits have been collected from various sources, however, full documentation for the portion of the deposit located on the Cristinas concessions generally is not available. The following sections outline the available information on the procedures and policies in place during the data collection at both sites, as well as summarizes and analyzes available results.

CRISTINAS CONCESSIONS

Placer geologists included a duplicate, standard and blank sample within every suite of 24 samples submitted for assay. In addition, 10% of the samples were sent to an outside laboratory for an independent check (IPL). MDA (2002) reports that comprehensive reports reviewing the results of the QA/QC data were prepared quarterly, however, these were not available to RPA.

RPA received a series of text files containing results of QA/QC data integrated during the drill hole programs from 1992 to 1997. Supporting documentation for the QA/QC data is limited, however, Table 11-3 outlines that from 1993 to 1997, Placer was diligent in including blanks, standards, and duplicates in its sample stream.

TABLE 11-3 SUMMARY OF AVAILABLE QA/QC DATA, CRISTINAS CONCESSIONS

GR Engineering (Barbados), Inc. – Siembra Minera Project

Year	1992	1993	1994	1995	1996	1997	Total
No. Holes	165	201	383	269	148	16	**1,182**
Total Metres	8,474	29,998	53,754	34,166	24,160	4,901	**155,454**
No. Assays	8,461	30,146	56,559	32,669	26,610	5,104	**159,549**
			QA/QC Samples				
Blanks							
Count		1,710	3,037	1,571	837	360	**7,515**
% of Assays		6%	5%	5%	3%	7%	**5%**
Standards[1]							
Count		1,357	3,028	1,250	825	275	**6,735**
% of Assays		5%	5%	4%	3%	5%	**4%**
Duplicates[2]							
Count		1,909	3,079	1,650	878	276	**7,792**
% of Assays		6%	5%	5%	3%	5%	**5%**

Notes:

1. Expected values and ranges of standards are not available.
2. The type of duplicate is not specified in the documentation (pulp, reject, coarse, field, check) but is thought to be a pulp duplicate.

As drill hole information from the Crystallex campaigns (2001 to 2006) is not available to RPA and not included in the Mineral Resource database. Summary information from previous reports on the Cristinas Project (MDA, 2002, 2003, and 2007) which outline procedures and results of QA/QC data used to support the Crystallex drilling campaigns are not discussed in this report.

BRISAS CONCESSIONS

Beginning in 1994, one standard sample was inserted for every 20 samples and one check sample was sent to a secondary laboratory for every 10 samples throughout the drilling campaigns, except for 2003-2004 when one in 20 samples were checked and one in 30 samples was a standard. In addition, blank samples were inserted at random to check residual contamination (normally one per drill hole and at the end of each sample run). Standards were included in some of the check assay runs.

In 1995, GRI developed a method of identifying potential integrity issues with assay results from outside laboratories, which was audited and modified by independent parties in 1995 and 1997. The method involved routine monitoring of results at Brisas including checking results for:

- Transcription errors;
- Conflict between the primary and secondary laboratories (tolerance for these sample pairs was 33%);
- Standards reporting assay values outside of accepted tolerance levels (12% for both Cu and Au).

In cases where conflicts were identified, assay re-runs were requested and reviewed. Re-runs were extended to the surrounding samples under the discretion of the reviewer. Up to five check assays have been performed using this monitoring system, with the final accepted value represented as an average value of all of the results, following removal of outliers. Potential issues identified in waste material were not always resolved as they did not affect the integrity of the resource database. RPA did not uncover details or results of any coarse or field duplicate programs.

According to PAH (2008), the reliability of assay results was tested throughout the drilling programs including several specific detailed studies by independent parties, including Mr. Mark Springett (1995 and 1996) and Behre Dolbear (1997), all of which indicated a satisfactory level of precision and accuracy.

A summary of available QA/QC data from the Brisas concessions is presented in Table 11-4.

TABLE 11-4 SUMMARY OF AVAILABLE QA/QC DATA, BRISAS CONCESSIONS
GR Engineering (Barbados), Inc. – Siembra Minera Project

Year[3]	1993	1994	1995	1996	1997	1999	2003	2004	2005	Total
No. Holes	49	130	98	259	214	13	9	101	37	**975**
Total Metres	5,828	16,091	18,859	52,159	66,353	5,726	1,822	24,448	10,866	**207,442**
No. Assays	1,921	5,479	6,308	17,359	21,803	1,833	1,103	5,820	3,262	**64,888**
					QA/QC Samples					
Blanks										
Count			27	226	206	7				**466**
% of Assays			<1%	1%	1%	<1%				**1%**
Standards[1]										
Count		4	34	532	1,209	101	45	199	69	**2,193**
% of Assays		<1%	1%	3%	6%	6%	4%	3%	2%	**3%**
Pulp Replicates[2]										
Count	1,341	878	1,376	5,854	7,369	632	115	816	308	**18,689**
% of Assays	70%	16%	22%	34%	34%	34%	10%	14%	9%	**29%**
Check Assay										
Count	1,054	651	803	2,538	2,237	189	57	275	114	**7,918**
% of Assays	55%	12%	13%	15%	10%	10%	5%	5%	3%	**12%**

Notes:

1. Expected values and ranges of standards are not available.
2. The type of duplicate is not specified in the documentation (pulp, coarse, field) but it is thought by RPA to be a pulp replicate sample.
3. Despite drilling campaigns, RPA has no record of QA/QC samples taken in 1992 or 2006.

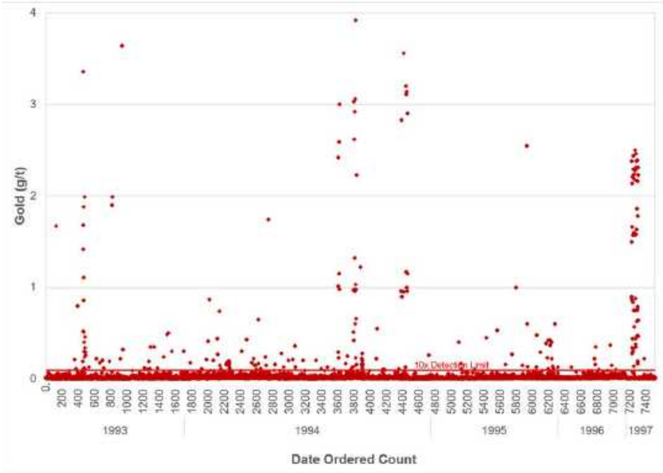
BLANKS

The regular submission of blank material is used to assess contamination during sample preparation and to identify sample numbering errors.

LAS CRISTINAS

A total of 7,515 blanks were included with samples assayed from 1993 to 1997. RPA analyzed the results using an assumed detection limit of 0.01 g/t Au and defined a sample to have failed if it returned a gold value more than ten times the detection limit (>0.1 g/t Au). RPA determined that a total of 249 samples (3%) failed the defined criteria, and observed that these failures were often clustered together temporally (Figure 11-3). RPA is of the opinion that the number of blank failures at Las Cristinas indicates a low degree of sample contamination or sample mix-ups.

FIGURE 11-3 CONTROL CHART OF BLANK SAMPLES (GOLD), CRISTINAS CONCESSIONS



BRISAS CONCESSIONS

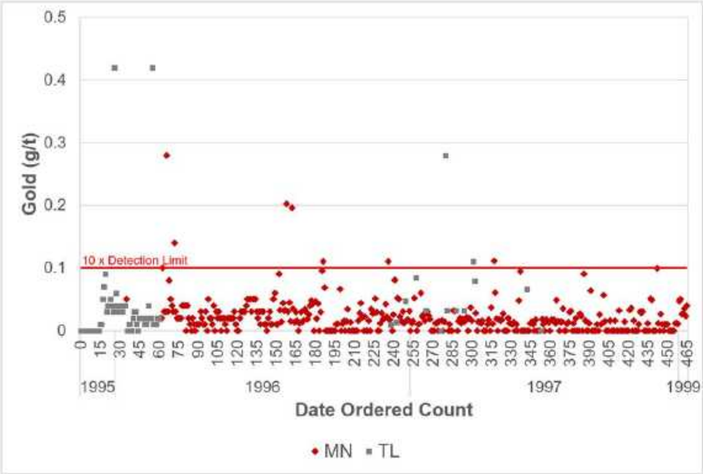
A total of 466 blanks were included in the sample stream at Brisas from 1995 to 1999. RPA analyzed the results using an assumed detection limit of 0.01 g/t Au and defined a sample to have failed if it returned a gold value more than ten times the detection limit (>0.1 g/t Au). Individual laboratory performance is listed in Table 11-5 and shown graphically in Figure 11-4.

TABLE 11-5 SUMMARY OF BLANK SAMPLE ANALYSIS, BRISAS CONCESSIONS

GR Engineering (Barbados), Inc. – Siembra Minera Project

Laboratory	Monitor			Triad		
	Failures		Total Submitted	Failures		Total Submitted
Year Submitted	Count	%		Count	%	
1995				1	4%	27
1996	7	4%	190	1	3%	36
1997	1	1%	195	2	18%	11
1999		0%	7			
Total	**8**	**2%**	**392**	**4**	**5%**	**74**

FIGURE 11-4 CONTROL CHART OF BLANK SAMPLES (GOLD), BRISAS CONCESSIONS



MN - Monitor Laboratory; TL - Triad Laboratory

CERTIFIED REFERENCE MATERIAL

Results of the regular submission of certified reference material (CRM) or reference material (standards) are used to identify problems with specific sample batches, and biases associated with the primary assay laboratory.

LAS CRISTINAS

RPA has compiled an incomplete set of QA/QC samples for the Cristinas deposit, however, the data is missing for several important components. With reference to the standard information, the expected values and ranges for the 17 individual standards reviewed by RPA were not available, limiting the ability to assess bias with the primary laboratory and identify issues with sample batches. It is also unclear whether the standard names (STD-1, etc.) could be referenced to the nomenclature employed in previous reports on the deposit. Table 11-6 summarizes the raw data available to RPA, including the approximate time frame of use of each standard, based on associated drill hole ID.

TABLE 11-6 AVERAGE GRADE AND STANDARD DEVIATION OF AVAILABLE STANDARDS SUBMITTED AT LAS CRISTINAS
GR Engineering (Barbados), Inc. – Siembra Minera Project

Standard	Time Frame	No. Submitted	Mean Grade (g/t Au)	Std. Dev. (g/t Au)
STD-1	1993 - 1994	404	0.39	0.28
STD-2	1993	279	1.06	0.14
STD-3	1993 - 1994	491	1.88	0.35
STD-4	1993	122	0.76	0.09
STD-5	1993	110	1.85	0.13
STD-6	1993	56	3.32	0.60
STD-7	1993 - 1994	217	1.26	0.33
STD-8	1994 - 1995	1,688	2.99	0.38
STD-9	1994 - 1995	1,290	1.05	0.18
STD-10	1995	457	0.81	0.98
STD-11	1995	416	1.73	0.31
STD-12	1995 - 1996	567	1.82	0.18
STD-13	1995	52	0.40	0.16
STD-14	1996 - 1997	157	2.34	0.13
STD-15	1996 - 1997	149	0.81	0.07
STD-16	1996 - 1997	143	1.68	0.12
STD-17	1996 - 1997	137	2.21	0.21
Total	**1993 - 1997**	**6,735**		

BRISAS

For much of the exploration drill programs, outlying standard values were considered less important than differences between primary assays and duplicate/repeat/check assays. Any standard differing by over 12% of the original standard value was flagged for further evaluation of the sample batch through re-checking, except in cases where:

1. The standard was inserted within waste material; or
2. Surrounding check assays showed agreement with the primary assays.

In total, 11% of standards fell outside acceptable limits, and one percent of these were followed up through re-assaying of the standard and shouldering samples.

RPA reviewed graphs compiled internally by GRI for six gold standards and seven copper standards (of a total of 21 standards included). Results were assessed temporally and by laboratory for bias and trends. Table 11-7 lists the expected value and acceptable range of gold and copper values for the standards inserted within the sample stream at Brisas. Gold standards were prepared by Hazen and Cone Laboratories, but their certification and matrix are unknown to RPA. The results from Monitor indicate a higher degree of precision at all grade ranges than Triad.

A large scale mislabelling of standards caused STD. 1, STD. 2 and STD. 3 to have different reference values pre and post drill hole D627. A review of results by GRI also caused the accepted value to change in each of these standards. The post 1997 accepted value and label are maintained in the table.

TABLE 11-7 EXPECTED VALUE AND ACCEPTED RANGE OF STANDARD

MATERIAL, BRISAS CONCESSIONS

GR Engineering (Barbados), Inc. – Siembra Minera Project

Element	Standard	Accepted Value	+ 12%	- 12%
Cu (%)	STD. 1-Y	0.11	0.13	0.10
Cu (%)	STD. 1-X	0.23	0.25	0.20
Cu (%)	STD. 2-X	0.06	0.07	0.05
Cu (%)	STD. 2-Y	0.03	0.04	0.03
Cu (%)	STD. 3-Y	0.07	0.07	0.06
Cu (%)	STD. 3-X	0.03	0.04	0.03
Cu (%)	STD. 6	0.06	0.06	0.05
Au (g/t)	STD. 1	0.56	0.63	0.49
Au (g/t)	STD. 2	0.96	1.06	0.84
Au (g/t)	STD. 3	1.49	1.57	1.23
Au (g/t)	STD. A	0.60	0.67	0.53
Au (g/t)	STD. B	1.00	1.12	0.88
Au (g/t)	STD. 6	1.01	1.13	0.89

RPA plotted the results of gold standard STD 1 (pre and post 1997) with time for the Monitor and Triad laboratories (Figure 11-5). In many cases, the standard was assayed twice and the data provided and the plotted results shown represent averages of the two results. Both laboratories show a low bias compared to the accepted value, with Monitor showing good precision and Triad showing only poor precision, which improved in the second campaign of samples.

RPA also plotted the results of copper standard STD 1Y with time for the Monitor and Triad laboratories (Figure 11-6). In some cases (approximately 15%), the standard was assayed twice and the plotted result represents an average of the two results. Both laboratories show a low bias compared to the accepted value, with Monitor showing good precision and Triad showing very poor precision with a potentially significant low bias.

RPA is of the opinion that results indicate good precision at Monitor, and recommends that future drill programs incorporate three or four matrix matched CRMs that approximate the gold cut-off grade, average grade, and high grades at the Project, and which include a relevant copper component.

The precision and bias observed at both laboratories is consistent with the observations of all other standards and grade ranges reviewed. Without the results of a round robin analysis, it is difficult to assess whether the observed bias is a result of laboratory procedures or whether a revision of the accepted value is warranted.

FIGURE 11-5 CONTROL CHART OF GOLD STD – 1Y

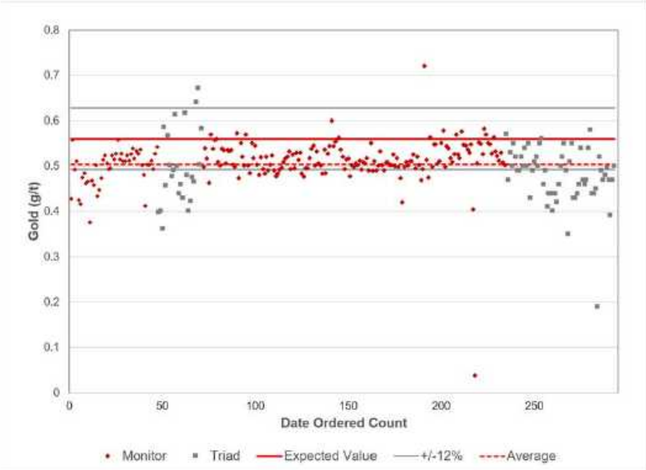
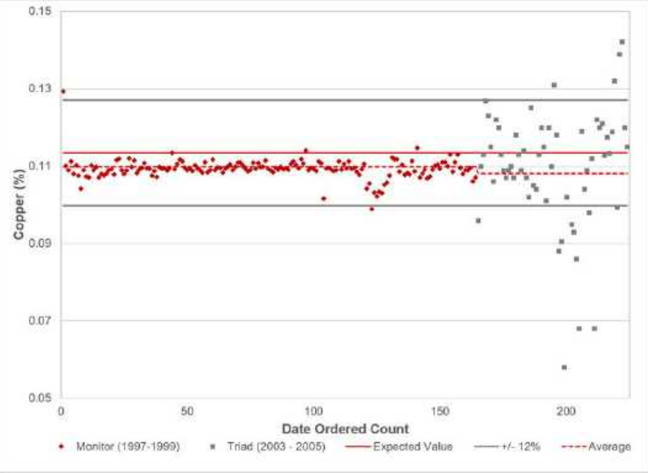


FIGURE 11-6 CONTROL CHART OF COPPER STD – 1Y

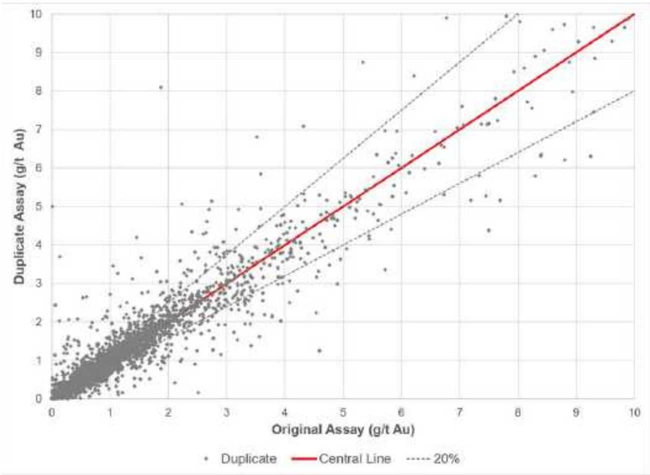


FIELD DUPLICATE SAMPLES, COARSE REJECT DUPLICATES, AND PULP DUPLICATES

LAS CRISTINAS

RPA did not uncover details or results of any coarse or field duplicate programs, and has assumed the available duplicate results are based on a program of pulp duplicates. This assumption is based on a survey of internal Placer reports, and RPA's understanding of the standard practice at the time the programs were undertaken. Duplicate data was not flagged with a laboratory identifier and therefore RPA's analysis does not comment on the precision of any specific laboratory. The correlation coefficient of the 7,792 duplicate pairs collected at the Project is 0.95, indicating good correlation for pulp duplicate samples. A total of 30% of sample pairs which had an average grade greater than 0.1 g/t Au plotted outside the expected error margin of ±20%. This is considered by RPA to be a high proportion for a pulp duplicate program but not unusual for gold mineralization. A scatter plot of duplicate sample pairs for gold is shown in Figure 11-7.

FIGURE 11-7 SCATTER PLOT OF PULP DUPLICATE SAMPLES, CRISTINAS CONCESSIONS



BRISAS

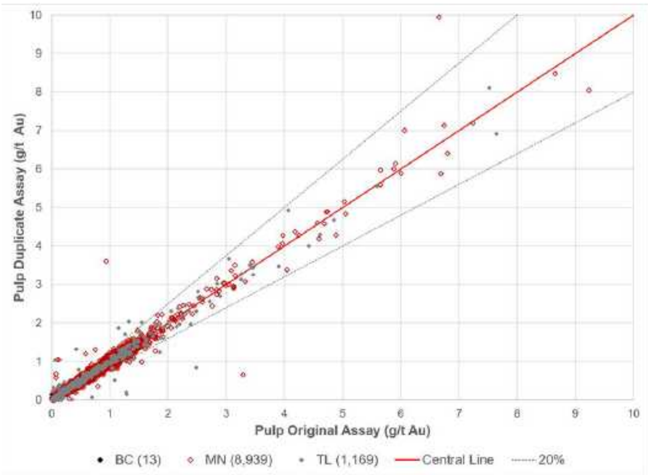
RPA did not uncover details or results of any coarse or field duplicate programs, however, GRI conducted a very high number of repeat, duplicate and check assays, up to six per sample. The final value in the Mineral Resource database represents an average grade of the repeated samples, which may represent assays with AAS and/or gravimetric finish and/or assays from different laboratories, with outliers reviewed and removed on a case by case basis by GRI geological department. In order to assess the precision of the primary laboratories employed throughout the drilling campaigns, RPA compiled and reviewed the initial and first pulp duplicate assay (AAS finish only) using basic comparative statistics (Table 11-8), scatter plots (Figure 11-8), and quantile-quantile (QQ) plots. Subsequent duplicate results on any single assay were excluded from the analysis.

TABLE 11-8 COMPARATIVE STATISTICS OF PULP DUPLICATE SAMPLES AT BRISAS

GR Engineering (Barbados), Inc. – Siembra Minera Project

	Monitor		Triad		Bondar Clegg	
	Original	Duplicate	Original	Duplicate	Original	Duplicate
Number of Samples (N):	8,939	8,915	1,169	1,169	13	13
Mean (g/t):	0.52	0.53	0.58	0.62	0.13	0.14
Maximum Value (g/t):	133.30	145.00	167.06	172.29	0.60	0.53
Minimum Value (g/t):	0.01	0.01	0.00	0.00	0.01	0.00
Median (g/t):	0.34	0.34	0.20	0.21	0.10	0.11
Variance:	4.20	4.62	24.80	28.60	0.02	0.02
Std. Dev:	2.05	2.15	4.98	5.35	0.15	0.14
Co-ef. Variation:	3.90	4.07	8.52	8.69	1.16	0.97
Correlation Coefficient	0.995		0.985		0.955	
% Diff. Between Means	-0.7%		-5.3%		-5.0%	

FIGURE 11-8 SCATTER PLOT OF PULP DUPLICATE SAMPLES AT BRISAS



BC - Bondar Clegg: MN – Monitor; TL - Triad

RPA is of the opinion that the pulp duplicate gold assay results indicate good precision at all three laboratories. There is significantly less scatter at Brisas compared to Cristinas suggesting that the gold mineralization at Cristinas is different with more coarser gold grains.

EXTERNAL LABORATORY CHECK ASSAYS

LAS CRISTINAS

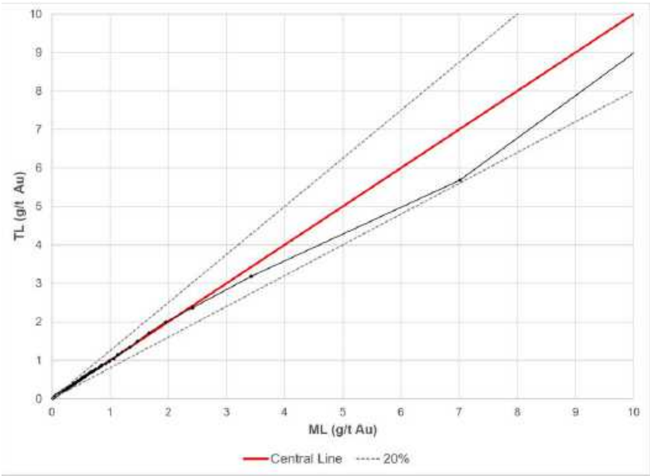
Internal documentation of procedures during the Cristinas drilling campaigns document that 10% of the samples taken were sent to an outside laboratory (IPL) for an independent check. RPA has not received the results of the check assay program and is unable to comment on any observed bias of the primary laboratory as a result.

BRISAS

As part of GRI's extensive check assay program, pulp samples from primary laboratories at various periods throughout the drilling campaigns from 1993 to 2005 (Bondar Clegg, Barringer, Monitor, and Triad) have been sent to other outside laboratories to assess bias at the primary laboratory. In addition to the laboratories mentioned above, pulp samples were sent to up to ten other external laboratories; sometimes the same pulp sample was sent to three or even four additional laboratories.

In an effort to simplify the results, RPA has limited analysis to the most prolific primary laboratories, Monitor and Triad, and one external laboratory. Figure 11-9 presents a quantile-quantile plot of the check assay pairs from Monitor and Triad (5,512 pairs). The results indicate negligible bias below 2.0 g/t Au, and a positive bias in favour of Monitor at higher grades. The correlation coefficient between the sample pairs was 93% and this bias was also visible in the scatter plot (not shown). The result may be partially explained by Monitor's more prolific use of gravimetric finish for higher grade results (2.3% of assays were repeated with a gravimetric finish at Monitor vs. 0.7% at Triad), but it also reinforces the findings of the standards analysis, where Triad results were consistently more scattered and overall lower grade than Monitor.

FIGURE 11-9 QUANTILE-QUANTILE PLOT OF CHECK ASSAY SAMPLES AT BRISAS



ML – Monitor; TL - Triad

RPA plotted 435 check assay pairs from Triad versus ActLabs in Ancaster, Ontario, Canada and found a similar correlation (94%), however, a moderately strong high grade bias of Triad compared to ActLabs was seen below 1 g/t Au. Insufficient sample pairs were available to compare ActLabs results with Monitor. Insufficient sample pairs were available to compare Monitor with any other known laboratory (almost all checks were sent to Triad from Monitor, and the checks that do exist have little supporting information to be able to draw meaningful conclusions).

QA/QC CONCLUSIONS AND RECOMMENDATIONS

RPA makes the following conclusions with regard to the QA/QC monitoring programs in place at Brisas and Cristinas from 1994 to 2005:

- QA/QC procedures in place during the time of data collection exceeded standard practices of the 1990s, however, field duplicates and coarse duplicates appear to not have been taken.

- Reference material in place at both projects does not appear to have undergone a round robin analysis, and the standards were not certified. Expected values of the standards were modified throughout the program and biases observed may be real, or may point toward the necessity of a modification of the expected value.
- Evidenced by a comparison of the Monitor and Triad check assays, and the results of the standard program at Brisas, there is a potential low grade bias at Triad, as compared to Monitor.
- Monitor shows the highest precision of the primary laboratories in place at either project, as evidenced by the high correlation coefficient of pulp duplicates. Both Triad and Bondar Clegg show only moderate repeatability for a pulp duplicate. Considering that the final accepted value in the database has undergone a thorough internal review before being accepted, and that it is often based upon the average of several results, RPA is of the opinion that the pulp duplicate program supports the integrity of the Mineral Resource database.
- Overall, RPA is of the opinion that the QA/QC program was reasonable and the assay results within the database are suitable for use in a Mineral Resource estimate.

RPA makes the following recommendations with regard to the QA/QC monitoring programs:

- Implement a field duplicate sampling program at Brisas at a rate of 1 in 50.
- Implement a coarse duplicate sampling program at Brisas at a rate of 1 in 50.
- Acquire or develop three or four matrix matched CRMs in all future drill programs that approximate the cut-off grade, average grade, and high grades at the Project. Insert at a rate of approximately 1 in 25 and accompany external laboratory check assays at a rate of approximately 1 in 20. Limit the number of laboratories in use at the Project to three and limit the number of CRMs to four.

12 DATA VERIFICATION

PAH DATA VERIFICATION - BRISAS CONCESSIONS

The following is taken from PAH (2005).

The reliability of assay results was tested throughout the drilling programs including several specific detailed studies by independent parties. Mr. Mark Springett carried out studies on the reliability of sampling and assaying in 1995 and 1996 and concluded that the results from the laboratories (250 samples) showed a satisfactory level of precision and were unbiased relative to each other. Behre Dolbear performed an independent check assay program of 36 samples from six holes in 1997 to check assays produced by Monitor or Triad against results from a third laboratory (Bondar Clegg). Samples were selected with values at different ranges of gold grades. Behre Dolbear's check assay results showed high correlation coefficients for both gold (0.92) and copper (0.99) and mean values within approximately 5% of each other for both metals.

In 1997, GRI and Behre Dolbear jointly drilled six core holes under Behre Dolbear's direct supervision and conducted assays independently at different laboratories. Behre Dolbear concluded that procedures utilized to collect assay data met or exceeded industry standards and that the assay results from all laboratories (Bondar Clegg, Monitor, and Triad), were reliable.

PAH conducted several data verifications and validations for the January 2005 feasibility study. PAH visited the Brisas Project facilities, toured the laboratory preparation and core shack areas, and inspected the core and several drill sites during the 2003-2004 drilling campaign in February 2004. PAH visited GRI's offices in Spokane, Washington to review the original drill hole logs and assay sheets in April 2004.

PAH verified the drill log data and assays against the drill hole database used for the Brisas Project feasibility study. Ten holes located in ten different vertical sections throughout the Brisas Project were checked for collar location, downhole survey, assaying and geological/geotechnical information. Minor discrepancies were found in survey and lithology information between the database and the logs; no errors or discrepancies were found on assay information. It was found that several holes in the early stages of the drilling campaigns were not surveyed for downhole deviation (e.g., most AD-series holes and some D-series holes). All AD-series holes were apparently given an average of the deviation observed in the few (approximately 20%) that did have deviation measurements.

The downhole deviation can be up to approximately 40 m on long holes (e.g., AD85 at a depth of 362 m), however, the average depth of the AD holes is 214 m and the average depth of the A holes is 27 m. The number of holes affected is less than 10% of the current database and the area covered has been drilled at closer spacing by later campaigns with deviation measurements. Therefore, the lack of downhole surveying in these holes does not appear to greatly influence the model. Also, auger holes were visually inspected in cross sections and showed generally good agreement with the much more abundant surrounding core hole data.

TWIN DRILLING VERIFICATION

Twin hole tests were run occasionally throughout the drilling program. A total of seven twin holes were drilled at different times and locations within the property. Both the initial and the twin were core holes. Visual inspection of twin drill hole intersects on cross section indicates overall a very good correspondence of mineralized areas in terms of location, length of the zones, and distribution of Au and Cu grades, although the comparison of individual samples shows some variability due to natural deposit local variations (nugget effect).

Table 12-1 shows a summary of the twin hole data. The comparison shows good reproducibility of sampling data, but also suggests consistently lower grades mainly for Au, but also for Cu in the twin or A holes, relative to the original core holes. It should be noted that while this apparent bias may be due, at least partially, to the highly variable distribution of gold within the deposit, it is, in some cases, also the result of having a single very high grade assay skewing the overall average for the hole(s) as seen in Table 12-2, for example for holes D404/D404A and D498/D498A. Without these high assays the results compare much better.

The A holes and a few other holes were drilled in 1999 by GRI under the direct supervision of Behre Dolbear as part of an independent verification of the drilling and assaying programs at the Brisas Project. In order to better understand the apparent bias on the A holes, PAH requested that GRI drill a hole (D754) as a twin hole to one of other Behre Dolbear holes drilled in 1999 (D614). As seen in Table 12-2, the PAH hole returned average grades slightly lower than the Behre Dolbear hole for Au and about the same grade for Cu, indicating that a bias more likely does not exist on the sampling and assaying data and as such the twin hole data generally confirm the original assay results.

TABLE 12-1 SUMMARY OF TWIN HOLE GOLD DATA, BRISAS CONCESSIONS
GR Engineering (Barbados), Inc. – Siembra Minera Project

	Interval	Initial Hole				Twin Hole ("A" Hole)				Ratio (Au)	Ratio (Cu)
Drill Site #	Length	Hole-ID	Length	Au (g/t)	Cu (%)	Hole-ID	Length	Au (g/t)	Cu (%)	Twin/Orig.	Twin/Orig.
1	350	D548	353	0.389	0.042	D548 A	350	0.369	0.038	0.949	0.905
2	119	D328	155	0.499	0.251	D328 A	303	0.390	0.207	0.782	0.825
3	210	D260	211	0.392	0.099	D260 A	369	0.376	0.097	0.959	0.980
4	148	D404	148	0.850	0.391	D404 A	160	0.655	0.372	0.771	0.951
5	341	D498	383	0.407	0.016	D498 A	341	0.372	0.016	0.914	1.000
6*	179	D476	179	0.183	0.015	D637	200	0.190	0.014	1.038	0.933
7	251	D754	252	0.428	0.236	D614	251	0.450	0.229	1.051	0.970
Overall Ave.	1,598	All Samples	1,681	0.427	0.119	All Samples	1,974	0.391	0.112	0.916	0.938
Overall Ave. without high grade outliers	1,598	All Samples	1,681	0.406	0.117	All Samples	1974	0.391	0.112	0.963	0.961

* DH Traces are 7 to 12 m apart

TABLE 12-2 COMPARISON OF TWIN HOLE COPPER DATA, BRISAS CONCESSIONS
GR Engineering (Barbados), Inc. – Siembra Minera Project

	Interval	Initial Hole					Twin Hole ("A" Hole)				
Drill Site #	Length	Hole-ID	Au (Max)	Au (Min)	Cu (Max)	Cu (Min)	Hole-ID	Au (Max)	Au (Min)	Cu (Max)	Cu (Min)
1	350	D548	2.924	0.011	0.680	0.003	D548 A	2.018	0.011	0.244	0.002
2	118	D328	1.92	0.044	1.368	0.011	D328 A	1.369	0.054	1.304	0.011
3	210	D260	1.615	0.040	0.847	0.004	D260 A	1.639	0.027	0.375	0.007
4	148	D404	5.188	0.030	4.345	0.002	D404 A	4.067	0.005	4.404	0.003
5	341	D498	4.376	0.018	0.234	0.001	D498 A	2.371	0.005	0.195	0.001
6*	179	D476	1.111	0.005	0.078	0.001	D637	0.995	0.005	0.056	0.001
7	251	D754	2.93	0.020	1.432	0.005	D614	4.029	0.018	1.326	0.001

* DH Traces are 7 m to 12 m apart

MDA DATA VERIFICATION - CRISTINAS CONCESSIONS

The following is taken from MDA (2007).

As most of the Las Cristinas database is derived from Placer's work, it is important to note that based on Placer's descriptions of their procedures, their data collection and exploration procedures conformed to or exceeded industry standards in effect at the time. If conducted as reported, Placer's QA/QC program was of high quality. In general, MDA found that, based on reported methodology, Placer's exploration data were collected in a technically sound manner. According to Placer documentation, quality assurance checks were in place for most of the project, and validation of data was ongoing. Nevertheless, it was clear that additional verification was necessary because one company had completed all development work, there were no independent checks or studies of the work, and most of the original hardcopy data were unavailable for detailed study or auditing.

Under the terms of the September 2002 agreement between Crystallex and CVG, Crystallex obtained an electronic database from CVG, which included Placer's drill, topographic, geological, and engineering data. At that time, data from 1,174 drill holes and 108 trenches were included in the Las Cristinas database. Although approximately 99% of the drill data were obtained, hard copies of the assay and geological data were not available, leaving a gap in the ability to validate the database.

When MDA visited the Las Cristinas site in October 2002, it found drill pads, drill collars, drill core and samples, core photographs, and other supporting data demonstrating that exploration had been done in a manner not incompatible with what was described in the documentation of Placer's work. To conduct independent verification, Crystallex drilled 2,198 m in 12 diamond drill holes, for a total of 1,079 core samples, to confirm the presence and tenor of mineralization. These 12 holes twinned previously drilled Placer holes. In addition, 275 QA/QC samples from this drill program were analyzed. The Crystallex drill results and check samples corroborate the general tenor of gold mineralization reported by the previous operator. For additional confirmation, Crystallex re-assayed 262 pre-existing pulps, 200 pre-existing coarse rejects, and 342 quarter-core samples of pre-existing core. Although mean grades are similar for both datasets, there is a large variance in grade between individual pairs of Placer's core assays and Crystallex's core check samples. The variance is lower in the pulp and coarse reject checks. As a result of some of these discrepancies, several additional studies were completed to aid in the understanding of grade variability.

Natural grade variability (heterogeneity) is an issue at Las Cristinas. Although it has become better understood through the efforts of Crystallex, it is an issue that should continue to be addressed prior to and during production, as it may result in massive misclassifications of ore and waste. The effect of material heterogeneity on the resource estimate will be dominated by local variance and may have instilled a minor low bias to the sample database. The issue is introduced by the distribution of metals originally in primary ore.

For this reason, Pitard (2005) rhetorically questioned: “Can the existing gold grade database, created with diamond drilling and conventional 30-g fire assays, lead to an accurate block model?” To which he responded: “The answer is no. But, with good geology of the various quartz and sulphide events, it can make a world of a difference.” The problem he is referring to is the ability to estimate accurately locally and with precision. MDA believes that this is difficult to do, but the consequence is not so great that it would negatively impact a mine and deposit of this scale in an open-pit scenario; essentially higher grades will be generally where higher grades are estimated to be, and the same with the mid- and low grades. While the gold occurs in the free state, it is generally not coarse grained nor visible but does appear to occur in clots of sulphides. It is not possible to compensate for the issue of a potential low bias instilled in the sample assay results.

RPA AUDIT OF DRILL HOLE DATABASE

SOFTWARE VALIDATION

RPA utilized Surpac’s validation features and Microsoft Excel to check for any errors or potential issues in the drill hole data including:

- Sample length issues;
- Maximum and minimum;
- Negative values;
- Detection limit / Zero values;
- Borehole deviations;
- Gaps;
- Overlaps;
- Drill hole collar versus topography;
- Datum;
- Laboratory certificate versus database values.

RPA did not note any significant errors.

DRILL HOLE COLLAR VERSUS TOPOGRAPHY

A total of 2,068 surveyed drill hole collars can be compared against the extents of topographic surface. None of the drill holes outside the extents of the topographic surface are part of the resource modelling. Approximately 90% of surveyed drill holes are within -4.0 m and +2.0 m of the topographic surface. A total of 40 (1.9%) of surveyed drill hole collars are more than 5.0 m below the topographic surface, with 16 (0.8%) of surveyed drill hole collars lower than 10.0 m below the topographic surface, and a total of 24 (1.2%) surveyed drill hole collars are more than 5.0 m above the surface, with just three (0.15%) being higher than 10.0 m above the topographic surface: DG792, D854 and DG796 at 21.4 m, 17.8 m and 11.3 m, respectively. None of the drill holes higher than 10.0 m above the topographic surface are located within the mineralization wireframes. Eight of the drill holes lower than 10.0 m below the topographic surface are located within the mineralization wireframes. All eight of the drill holes more than 10.0 m below the surface appear to be located within flooded areas.

A total of 108 trenches can be compared against the extents of topographic surface. Almost all the trench collars (99%) are below the topographic surface and the mean elevation below surface topography is 4.0 m. The collar of four trenches (3.7%) is between 10.0 m and 12.0 m below the topographic surface.

Widespread disturbance of the original surface at the Project has taken place and multiple collar and trench sites are now flooded and their present locations show as flat surfaces at the level of the water table. RPA is of the opinion that the vast majority of the elevation differences between surveyed drill holes and trenches with respect to the up to date topographic surface can be explained by extensive conventional small scale mining activity and disturbance of the original surface and by flooding of former works.

ASSAYS

RPA compared 4% of the sample database to the assay certificates from Triad on the Brisas portion of the Project. No major discrepancies were found.

In RPA's opinion, the drill hole data is adequate for use in the preparation of Mineral Resource estimates.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

Due to the advanced level of design achieved for both Brisas and Cristinas previously, a number of metallurgical testing programs were completed on a large number of samples. The summary of metallurgical data is taken primarily from previous Technical Reports that have been filed publicly for the Brisas and Cristinas projects and sections of a feasibility study that was completed by a previous owner for the Cristinas Project (Placer 1996, PAH 2008, and MDA 2007).

For Brisas, GRI completed a feasibility study (PAH, 2005) and basic engineering that were based on a processing flow sheet that included gravity concentration and a flotation concentrator with leaching of the cleaner scavenger tailings to produce doré and copper concentrate. For Cristinas, Crystallex completed a feasibility study that included gravity concentration and whole ore cyanide leaching in a carbon-in-leach (CIL) circuit to produce only doré (MDA, 2007).

These decisions were made because the northern side of the Brisas property contains higher concentrations of copper than the southern side of the Brisas property which has higher gold concentrations. Similarly, the northern side of the Cristinas property contains higher concentrations of copper (Mesones) and the southern side has higher gold concentrations. The data regarding rock type and grade distributions from the new RPA model are summarized in Table 13-1.

TABLE 13-1 SUMMARY OF RESOURCES AND GRADES BY AREA

GR Engineering (Barbados), Inc. – Siembra Minera Project

Zone Name	Tonnes (Mt)	Grade (g/t Au)	Grade (%Cu)	Contained Gold (oz)	Contained Copper (Mlb Cu)
Measured & Indicated					
Brisas	603	0.58	0.10	11,321	1,273
Cristinas	451	0.88	0.10	12,802	1,036
Mesones	76	0.65	0.22	1,582	361
Morrocoy	1	0.86	-	30	-
Cordova	53	0.63	0.01	1,087	17
Sub-Total, M&I	**1,184**	**0.70**	**0.10**	**26,823**	**2,687**
Inferred					
Brisas	364	0.47	0.12	5,489	971
Cristinas	761	0.70	0.07	17,065	1,140
Mesones	51	0.35	0.17	579	186
Morrocoy	92	0.60	-	1,770	-
Cordova	23	0.67	0.01	486	3
Total, Inferred	**1,291**	**0.61**	**0.08**	**25,389**	**2,300**

The relative proportions of material from the historical data is summarized in Table 13-2.

The current plan proposes to process the mined material in both a flotation concentrator and a cyanide leaching facility. The material that contains copper concentrations greater than 0.02% will be processed in the flotation plant and the material that contains lower concentrations of copper will be processed in the cyanide leach plant. For comparison purposes, a summary of the tonnes and grade by rock type from the current mine plan is presented in Table 13-2.

TABLE 13-2 CURRENT SUMMARY OF ROCK TYPES AND GRADES

GR Engineering (Barbados), Inc. – Siembra Minera Project

Material Mined	Mt	Au, g/t	Cu, %
Oxide Saprolite	43	0.638	0.050
Sulphide Saprolite Low Cu	30	0.533	0.007
Sulphide Saprolite High Cu	133	0.778	0.118
Total Sulphide Saprolite	206	0.733	0.098
Fresh Hard Rock Low Cu	266	0.565	0.012
Fresh Hard Rock High Cu	1,533	0.728	0.107
Total Hard Rock	1,800	0.704	0.093
Total	2,005	0.705	0.092

BRISAS

Four types of material were identified for Brisas. They are:

- Oxide saprolite
- Sulphide saprolite
- North hard rock (High Cu)
- South hard rock (Low Cu)

The two hard rock types (i.e., North and South) are defined based on the copper concentration. North is defined as gold-chalcopyrite-pyrite with a copper concentration greater than 0.05%. South is gold-pyrite with a copper concentration less than 0.05%.

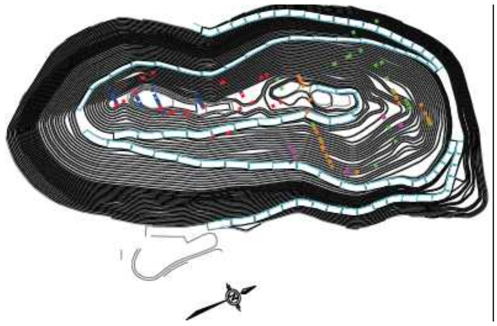
From 1992 to 2005, 20 metallurgical test programs and mineralogical investigations were completed for Brisas. Five pressure oxidation testing programs were completed using copper concentrate. Tailings analysis and characterization programs were also completed.

Grinding test work was completed by MacPherson in 1997. The gross autogenous work index was 21.3 kWh/t and the Bond ball mill work index was 15.4 kWh/t.

Test work conducted by Lakefield Research, Inc. (Lakefield) in 2005 was used as the basis of the feasibility study completed by Aker-Kvaerner and the subsequent detailed design completed by SNC-Lavalin. The testing was completed using all four rock types using the following six groups of samples:

- Oxide saprolite: composite sample obtained from hand-augered samples
- Sulphide saprolite: composite sample obtained from 27 drill hole intervals
- North hard rock: composite sample obtained from 31 drill hole intervals
- South hard rock: composite sample obtained from 29 drill hole intervals
- North hard rock: composite sample obtained from 25 drill hole intervals
- South hard rock: composite sample obtained from 25 drill hole intervals

PAH reported that they considered the samples to be representative of the Brisas deposit. The sample locations are shown in Figure 13-1.



13-4

4rs - Met. Samples
4rn - Met. Samples
5ox - Met. Samples
5ss - Met. Samples
5rn - Met. Samples
5rs - Met. Samples

March 2018

Source: Gold Reserves Inc., 2017.

Figure 13-1
GR Engineering (Barbados), Inc.
Siembra Minera Project
Bolivar State, Venezuela
Brisas Metallurgical

Sample Locations

www . rpacan . com

The test work conducted by Lakefield confirmed earlier work which showed that the North samples can produce a marketable copper/gold flotation concentrate but the South samples cannot. As a result, the plan was to blend the ores to produce marketable concentrates. Consequently, the Lakefield tests were conducted using blends of the North and South samples and included gravity separation, batch flotation tests to determine optimum conditions, and eight locked cycle flotation tests (LCTs) that were used as the basis of the process design since they are considered to be a better indication of mill performance than open circuit tests. A summary of the results from the LCTs are provided in Table 13-3.

TABLE 13-3 SUMMARY OF LOCKED CYCLE TEST DATA
GR Engineering (Barbados), Inc. – Siembra Minera Project

Material Blend Ratio			Assays	Recovery	
North:South:Sulphide Saprolite	Products	Cu, wt%	Au, g/t	Cu, %	Au, %
50:50:00	Gravity Concentrate		1,911		17.1
	Copper Concentrate	23.2	63	83.1	46.4
	1st Cleaner Scav Tails	0.30	2.38	14.4	23.1
	Rougher Tailings	0.004	0.10	2.57	13.4
	Head	0.147	0.713	100.0	100.0
60:40:00	Gravity Concentrate		3,030		13.7
	Copper Concentrate	23.3	55	87.5	52.0
	1st Cleaner Scav Tails	0.21	1.785	10.5	22.3
	Rougher Tailings	0.004	0.08	2.05	12.0
	Head	0.160	0.635	100.0	100.0
40:60:00	Gravity Concentrate		3,563		14.5
	Copper Concentrate	18.25	61.7	86.9	50.9
	1st Cleaner Scav Tails	0.16	1.81	10.9	21.6
	Rougher Tailings	0.003	0.105	2.2	13.1
	Head	0.125	0.715	100.00	100.0
52:41:7	Gravity Concentrate		478		17.9
	Copper Concentrate	28.7	70.1	85.7	41.9
	1st Cleaner Scav Tails	0.23	2.3	10.8	21.4
	Rougher Tailings	0.006	0.15	3.5	18.8
	Head	0.15	0.760	100.0	100.0

CRISTINAS

Data available for the Cristinas deposit includes a Placer feasibility study completed in 1996 and information from the Cristinas Technical Report that was completed in 2007 (MDA, 2007) which is publicly available. The locations of the metallurgical samples that were used in the two studies are shown in Figure 13-2.

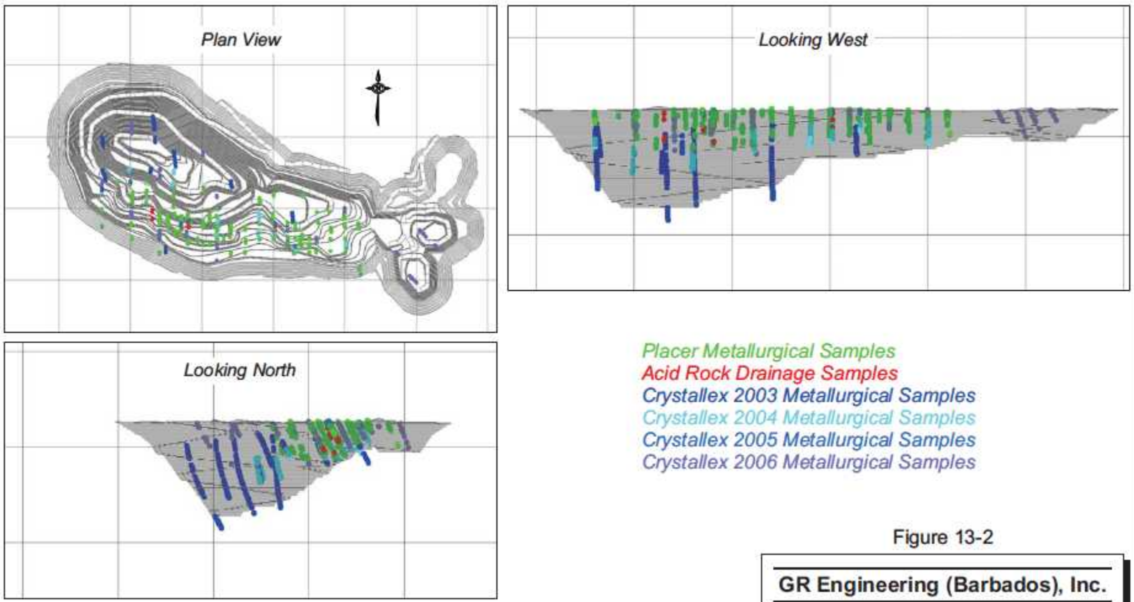


Figure 13-2

GR Engineering (Barbados), Inc.

Siembra Minera Project
Bolivar State, Venezuela

Cristinas Metallurgical Sample Locations

March 2018

Source: Crystallex International Corp., 2007.

www.rpacan.com

PLACER FEASIBILITY STUDY

In 1996, Placer completed a feasibility study that considered processing by gravity concentration and cyanide leaching for oxide ores and flotation and cyanidation for sulphide ores to produce doré and copper concentrate. The study summarized the metallurgical results from 17 metallurgical test reports.

The rock types identified for Cristinas were:

- Oxide saprolite
- Sulphide saprolite
- Carbonite leached bedrock
- Carbonite stable bedrock

OXIDE SAPROLITE

Placer reported that gold extraction from oxide saprolite samples averaged 94% with a cyanide consumption of 0.30 kg/t and that there did not appear to be a correlation between gold head grade and gold extraction for gold concentrations above the cut-off grade.

Placer also determined that the sulphide saprolite and bedrock ores had high concentrations of cyanide soluble copper and associated high cyanide consumptions. Therefore, the proposed flowsheet for those rock types included gravity concentration, flotation, and cyanide leaching of the scavenger concentrate and the cleaner flotation tailings. The test data indicated that there was no correlation between gold recovery and gold head grade although the recovery did vary by rock type. The test data also established relationships between copper head grade and recovery and copper head grade and flotation concentrate grade for sulphide saprolite and hard rock.

SULPHIDE SAPROLITE

Based on a correlation between copper recovery and the copper flotation feed head grade in g/t Cu, Placer estimated the copper recovery for sulphide saprolite as a function of the copper head grade using the equation:

Cu Recovery = $4.2017 \times (\text{Cu Head, g/t})^{0.3597}$

Or:

Cu Recovery = $4.2017 \times (\text{Cu Head, \%} \ast 10000)^{0.3597}$

The final copper concentrate grade was also estimated as a function of the copper head grade using the equation:

Concentrate, % Cu = $7.854 + 137.08 \times Cu\ Head,\% - 248.8 \times (Cu\ Head)^{2}$

Using the test data, Placer also estimated that the gold recovery from sulphide saprolite will be approximately 20% to the gravity concentrate, half of which will be recovered as a table concentrate and the remainder will be processed further. Based on the test data, it was estimated that the gold recovery in the flotation circuit is 46.7%. Placer estimated gold extraction by leaching the cleaner tailings and the second scavenger concentrate using the ratio between the sodium cyanide to copper since the cyanide addition is typically the limiting factor for gold extraction for ores that contain high concentrations of cyanide soluble copper. Placer estimated gold recovery for sulphide saprolite based on pilot plant data. The gold recovery estimates by product for sulphide saprolite are summarized in Table 13-4.

TABLE 13-4 PLACER GOLD RECOVERY ESTIMATE FOR SULPHIDE SAPROLITE

GR Engineering (Barbados), Inc. – Siembra Minera Project

Product	Au Recovery, %
Gravity Concentrate	11.4
Third Cleaner Concentrate	47.7
Leach Extraction	21.3
Total	80.4
Recovery Factor	0.99
Total Recovery	79.6
Recovery to Copper Concentrate	47.3
Recovery to Doré	32.4

HARD ROCK

Placer also estimated the copper recovery for carbonate leached and carbonate stable bedrock as a function of copper head grade using the equation:

Cu Recovery = $54.504 + 155.96 \times (Cu\ Head,\ \%) - 211.19 \times (Cu\ Head,\ \%)^{2}$

The copper concentrate grade is also estimated using the following equation:

Concentrate, % Cu = $30.33 \times (Cu\ Head,\ \%)^{0.109}$

For the bedrock samples, the gold recovery by a Knelson concentrator was found to be an average of 20% of the gold. Placer estimated that half of the gold (i.e., 10%) would be recovered as a gravity concentrate from a table concentrate and the other half from the table tailings would be leached with 95% extraction for an additional gold recovery of 9.5%. Gold distribution and recovery in the flotation circuit was estimated by correlating the flotation feed grade to the flotation scavenger tailing grade and assuming that 10% of the mass fed to the flotation circuit would be recovered in the rougher/scavenger flotation circuits. Using these correlations, the scavenger tailings would contain 16.3% of the gold fed to the plant resulting in an overall gold recovery of 83.7%. The gold recovery estimates for hard rock by product are summarized in Table 13-5.

TABLE 13-5 PLACER GOLD RECOVERY ESTIMATE FOR HARD ROCK

GR Engineering (Barbados), Inc. – Siembra Minera Project

Product	Primary Distribution, %	Leach Extraction, %	Final, %
Table Concentrate	10.0		10.0
Table Tailings	10.0	95%	9.5
Flotation Feed	80.0		
Scavenger Tailings	16.3		
Final Concentrate	47.8		47.8
Leach of Flotation Product	15.9	85%	13.5
Unaccounted Losses			-0.70
Total			80.1
Recovery to Copper Concentrate			47.8
Recovery to Doré			32.3

COMMINUTION DATA

Placer also completed SAG Mill Work Index (W_i), Ball Mill W_i, and Abrasion Index (A_i) data using samples from Cristinas. A summary of the data is provided in Table 13-6.

TABLE 13-6 PLACER COMMINUTION DATA

GR Engineering (Barbados), Inc. – Siembra Minera Project

Rock Type	SAG W_i, kWh/t	Ball W_i, kWh/t	A_i
Saprolite	N/A	7.7	- ---
Carbonate Leached	12.6	10.0	0.0909
Carbonate Stable	17.5	14.8	0.2136

CRISTINAS FEASIBILITY STUDY

According to the Technical Report (MDA, 2007), "In early 2003, Crystallex, SNC-Lavalin, and Goode reviewed available metallurgical test data and performed various trade-off studies.

These analyses indicated that the production, transportation off-shore, and smelting of a copper-gold flotation concentrate, as proposed by Placer, was a less attractive alternative and that direct leaching of most or all of the mineralized material and on-site production of bullion would give better gold recovery. The trade-off studies also showed that the direct leach process, which is the flowsheet originally selected by Placer, would simplify the process, improve plant operability, and give lower capital and operating costs."

Crystallex maintained the same rock types as those used by Placer in the earlier work.

Composite samples were composited from individual drill core intervals taken from within the limits of the planned Conductora pit and sent to SGS Lakefield for bench tests and pilot plant tests. A number of the samples were composites containing mixtures of saprolite and hard rock in order to simulate the planned operating conditions for the project.

OXIDE SAPROLITE

The as-received screen analysis for the oxide saprolite sample was 63 µm.

The average gravity recovery for the samples tested at SGS Lakefield was 5.3% at a grind size of 80% passing (P_{80}) 35 µm. Professor André LaPlante conducted his standard gravity-recoverable-gold (GRG) test at McGill University and determined that the oxide saprolite sample contained 39% GRG and concluded that approximately 25% of the gold would be recovered by gravity processing. Subsequently, Knelson used their circuit modelling system and projected a gravity gold recovery between 18% and 20% using the LaPlante data.

Intensive cyanide leaching of gravity concentrates produced from combined oxide saprolite and carbonate stable bedrock samples and mine blend samples produced gold extractions ranging from 95.7% Au to 99.3% Au.

Bottle roll tests (BRTs) were conducted on the tailings from the gravity concentration tests. Initial tests were conducted to investigate the effects of grind size. The results showed little difference between tests conducted at P_{80} 50 µm and P_{80} 75 µm so the 75 µm size was selected. The tests showed that 99% gold extraction (gravity plus leaching) was possible after 36 hours of leaching using the pure oxide saprolite sample. Other tests showed 98% gravity plus leach extraction was achieved across a range of leach times between 24 hours and 36 hours.

Preg-robbing tests showed values of less than 4%.

Copper leaching from oxide saprolite samples was generally less than 5%. Cyanide consumption was related to the concentration of cyanide soluble copper in the samples, which is highest for the sulphide saprolite samples. The cyanide consumption for samples that were not sulphide saprolite, including oxide saprolite was reported to be between 0.25 kg/t and 0.7 kg/t.

Lime consumption for the oxide saprolite samples was reported to be between 1.0 kg/t and 1.5 kg/t.

SULPHIDE SAPROLITE

The as-received screen analysis for two sulphide saprolite samples was 182 µm and 69 µm, respectively.

The gravity recovery for the sulphide saprolite that was ground to P_{80} 50 µm was 18.4% and the sample that was ground to P_{80} 63 µm was 22.9%.

Samples of pure sulphide saprolite material contained higher concentrations of cyanide soluble copper resulting in lower overall gold extraction and higher cyanide consumption. Combined gravity plus leaching gold extractions ranged between 85% and 94%. Cyanide consumption was found to be correlated with the cyanide soluble copper (CNSCu) concentration. For samples containing less than 370 ppm CNSCu, the total gravity plus leach gold extraction was between 85% and 89% with cyanide additions of 1.7 kg/t to 1.9 kg/t. Lime consumption ranged from approximately 0.4 kg/t to over 1.5 kg/t.

The samples were found to be mildly preg-robbing with 9% and 16% of a 10 ppm spike adsorbed after 24 hours.

HARD ROCK

The gravity recovery of gold ranged from 17.2% to 23.8% for samples ground to P_{80} 54 µm to P_{80} 99 µm with no apparent relationships between grind size and gravity gold recovery or head grade and recovery. LaPlante estimated that the carbonate stable bedrock sample contained 46% GRG. Knelson projected gravity gold recovery between 24% and 27%.

BRTs resulted in a combined gravity plus leaching total gold extraction between 87.8% and 90.1% with an average of 88.7%, a cyanide consumption of 0.51 kg/t and lime consumptions averaging 0.62 kg/t. Goode reported that there were no apparent relationships between gold head grade and gold extraction or reagent consumptions.

COMBINED SAMPLES

Four samples from Mesones were tested. The combined gravity plus leaching gold extraction ranged from 84% to 88% for grind sizes ranging between P_{80} 71 µm and P_{80} 103 µm. The cyanide additions were between 0.9 kg/t and 1.6 kg/t with an average of 0.77 kg/t and lime consumption average 0.44 kg/t. Goode reported that gold recovery from Mesones would likely be improved with optimization of reagent consumption strategies.

A CIL pilot plant was operated using two different blends of rock types. The first was a blend of 20% oxide saprolite and 80% carbonate stable bedrock, the second was a “mine blend” of oxide saprolite, sulphide saprolite, carbonate stable bedrock, and carbonate leached bedrock. The overall gravity plus leach extraction for the initial blend was 89.6% with a cyanide consumption of 0.7 kg/t and the mine blend sample resulted in 89.3% gold extraction with a cyanide consumption of 0.8 kg/t.

COMMINUTION

Bond W_i and A_i tests were conducted on a limited number of samples. They are reported in Table 13-7.

TABLE 13-7 CRISTINAS COMMINUTION DATA

GR Engineering (Barbados), Inc. – Siembra Minera Project

Sample	Rod W_i kWh/t	Ball W_i kWh/t	A_i kg/kWh
Carbonate Stable	17.1	15.0	0.27
80% Carbonate Stable – 20% Oxide Saprolite	-	14.2	-
Mine Blend	15.9	14.4	0.24
Carbonate Leached – Carbonate Stable	-	14.7	-

Ball mill W_i values were also estimated using the grinding data obtained when feed was being prepared for the metallurgical tests. The average W_i for tests conducted using blends of all four rock types averaged 13.1 kWh/t. The average W_i for tests conducted using only carbonate stable bedrock samples taken from various depths averaged 16.5 kWh/t. Although tests were not conducted to determine the W_i for saprolite samples, the work index was estimated to be between 6.0 kWh/t and 8.5 kWh/t using data from combined samples.

CARBON ELUTION

Two samples of carbon taken from the pilot plant test were eluted using a simulated high-pressure Zadra elution process. The results are presented in Table 13-8.

TABLE 13-8 CRISTINAS CARBON ELUTION ASSAYS
GR Engineering (Barbados), Inc. – Siembra Minera Project

	Units	Test 1			Test 2		
		Au	Ag	Cu	Au	Ag	Cu
Loaded carbon	g/t	1552	185	334	1534	287	555
Acid washed	g/t	1598	306	366	1615	319	364
Eluted carbon	g/t	32	1.2	<20	38	40	20
Recovery	%	98.0	99.6	94.6	97.6	87.8	96.4

VISCOSITY TESTS

SGS Lakefield measured viscosity using a Haake rheometer. The data showed that 100% oxide saprolite has a critical density of approximately 40% solids at a Yield Stress greater than 8 Pa.

DEWATERING TESTS

Flocculant scoping tests and thickening tests were undertaken by SGS Lakefield and Outokumpu. In general, the best results were achieved with low charge anionic flocculants (e.g., Magnafloc 919). SGS conducted static thickening tests in cylinders without rakes. They determined that the hard rock samples achieved an underflow density of approximately 45% solids by weight with a flocculant dosage of 15 g/t. The unit area required for conventional thickener designs was 0.83 $t/m^2/h$. For sulphide saprolite and a flocculant dosage of 33 g/t, the underflow density was 42% solids with a unit area of 1.04 $t/m^2/h$. Oxide saprolite samples required 23 g/t of flocculant to achieve an underflow density of 42% solids with a unit area of 0.22 $t/m^2/h$.

Outokumpu operated a continuous pilot-scale thickener to conduct 58 tests on nine blends of rock types. For the oxide saprolite samples, the results are similar to the results achieved by SGS Lakefield. The hard rock samples achieved higher underflow densities at lower solids loading rates than the SGS Lakefield tests. The results of the Outokumpu tests show that with the correct flocculant, thickener underflow densities of 50% or greater can be achieved at a loading rate of 0.47 t/m^2/h with all rock types and blends as long as the oxide saprolite content of the feed is less than 50%.

ENVIRONMENTAL TESTING

Modified US Environmental Protection Agency (EPA) acid base accounting (ABA) tests were conducted on samples from Mesones and Conductora. The tests determined that oxide saprolite samples were classified as non-acid generating, the sulphide saprolite samples may be acid generating, and the acid generating potential (AGP) of other samples was uncertain.

CYANIDE DESTRUCTION TESTS

SGS Lakefield performed natural degradation tests on tailings from the pilot plant test. The sample taken from the test that utilized the oxide saprolite-carbonate stable bedrock blend showed that the weak acid dissociable (WAD) cyanide (CN) concentration dropped from approximately 60 ppm to less than 15 ppm in 55 days. The cyanide concentration in the mine blend samples dropped from less than 110 ppm WAD cyanide to approximately 20 ppm WAD cyanide in 100 days.

Continuous cyanide destruction tests were conducted on the degraded tailing solutions using the sulphur dioxide (SO_2) – air process. Acceptable WAD cyanide levels were achieved using SO_2 to WAD cyanide ratios of six and three without copper additions.

RESULTS AND CONCLUSIONS

Based on the results of metallurgical testing using Brisas and Cristinas samples, the conceptual processes selected for the combined project include a cyanide leach plant to process oxide saprolite and sulphide saprolite that contains low concentrations of copper to recover gold as doré from gravity concentration and cyanide leaching plus a flotation concentrator to process sulphide saprolite and hard rock that contain higher concentrations of copper. The flotation concentrator will recover copper and gold into a copper flotation concentrate and gold as doré utilizing gravity concentration and cyanide leaching of cleaner scavenger tailings.

RPA compared and combined data from the Brisas and Cristinas projects in order to determine appropriate design criteria for the combined plant. This data was then used to estimate equipment sizing and the associated capital costs, estimate recoveries, and estimate operating costs. The Brisas design, equipment sizing, and cost estimates were taken directly from the SNC-Lavalin basic engineering design and cash flows. In general, the design for the oxide leach plant was taken from the Cristinas data.

A summary of the recovery estimates is provided in Table 13-9.

TABLE 13-9 RECOVERY ESTIMATES FOR PEA
GR Engineering (Barbados), Inc. – Siembra Minera Project

Material Type	Au, %	Cu, %
Oxide Leach Plant:		
Oxide Saprolite		
Gravity	21.0	
Leach	77.0	
Total	98.0	
Sulphide Saprolite Low Cu		
Gravity	21.0	
Leach	65.8	
Total	86.8	
Hard Rock Low Cu		
Gravity	20.0	
Leach	67.6	
Total	87.6	
Flotation Concentrator:		From formulas:
Sulphide Saprolite High Cu		$4.2017 \times (\text{Cu Head, \%} \times 10{,}000)^{0.3597}$
Gravity	5.0	$54.504 + 155.96 \times (\text{Cu Head, \%}) - 211.19 \times (\text{Cu Head, \%})^{2}$
Flotation	25.2	87.0
Leach	53.0	
Total	83.2	
Hard Rock Low Cu		
Gravity	9.0	
Flotation	63.0	
Leach	11.2	
Total	83.2	
Hard Rock High Cu		
Gravity	9.0	
Flotation	63.0	
Leach	11.2	
Total	83.2	

14 MINERAL RESOURCE ESTIMATE

SUMMARY

A Mineral Resource estimate, dated December 31, 2017, was completed by RPA using the Surpac and Leapfrog Geo software packages. Wireframes for geology and mineralization were constructed in Leapfrog Geo based on geology sections, assay results, lithological information, and structural data. Assays were capped to various levels based on exploratory data analysis and then composited to three metre lengths. Wireframes were filled with blocks measuring 10 m by 10 m by 6 m (length, width, height). Block grades were estimated using Inverse Distance (ID) and Nearest Neighbour (NN) interpolation algorithms. Gold and copper grades were estimated into blocks using inverse distance squared and dynamic anisotropy with the Surpac v.6.8 software package. The estimated gold and copper grades were used to calculate NSR values for each mineralized block. Block estimates were validated using industry standard validation techniques. Classification of blocks was based on distance and other criteria.

A summary of the Mineral Resources is provided in Table 14-1. Summaries of the Mineral Resources by material type and mineralized zone are provided in Tables 14-2 and 14-3.

TABLE 14-1 SUMMARY OF MINERAL RESOURCES – DECEMBER 31, 2017

GR Engineering (Barbados), Inc. – Siembra Minera Project

Category	Tonnes (Mt)	Grade (g/t Au)	Grade (% Cu)	Contained Gold (koz Au)	Contained Copper (kt Cu)	Contained Copper (Mlb Cu)
Measured	10	1.02	0.18	318	17	38
Indicated	1,174	0.70	0.10	26,504	1,202	2,649
Total Measured + Indicated	**1,184**	**0.70**	**0.10**	**26,823**	**1,219**	**2,687**
Inferred	1,291	0.61	0.08	25,389	1,044	2,300

Notes:

1. CIM (2014) definitions were followed for Mineral Resources.
2. Mineral Resources are estimated at an NSR cut-off value of US$7.20 per tonne for oxide-saprolite material and US$5.00 per tonne for sulphide-saprolite and fresh rock material.
3. Mineral Resources are constrained by a preliminary pit shell created using the Whittle software package.
4. Mineral Resources are estimated using a long-term gold price of US$1,300 per ounce, and a copper price of US$3.00 per pound.
5. Bulk density varies by material type.
6. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
7. Numbers may not add due to rounding.

TABLE 14-2 SUMMARY OF MINERAL RESOURCES BY MATERIAL TYPE – DECEMBER 31, 2017
GR Engineering (Barbados), Inc. – Siembra Minera Project

Material	Tonnes (Mt)	Grade (g/t Au)	Contained Gold (kg)	(koz)	Grade (%Cu)	Contained Copper (kt)	(Mlb)
			Measured				
Oxide Saprolite	1	0.89	575	18	-	-	-
Sulphide Saprolite	4	1.21	4,750	153	0.18	7	15
Hard Rock	5	0.90	4,579	147	0.20	10	23
Total, Measured	*10*	*1.02*	*9,904*	*318*	*0.18*	*17*	*38*
			Indicated				
Oxide Saprolite	20	0.75	14,857	478	-	-	-
Sulphide Saprolite	110	0.83	90,782	2,919	0.11	124	273
Hard Rock	1,045	0.69	718,736	23,108	0.10	1,078	2,376
Total, Indicated	*1,174*	*0.70*	*824,374*	*26,504*	*0.10*	*1,202*	*2,649*
			Measured + Indicated				
Oxide Saprolite	20	0.75	15,432	496	-	-	-
Sulphide Saprolite	114	0.84	95,531	3,071	0.12	131	289
Hard Rock	1,050	0.69	723,315	23,255	0.10	1,088	2,399
Sub-Total M&I	**1,184**	**0.70**	**834,278**	**26,823**	**0.10**	**1,219**	**2,687**
			Inferred				
Oxide Saprolite	24	0.53	12,528	403	-	-	-
Sulphide Saprolite	65	0.48	30,942	995	0.07	45	98
Hard Rock	1,201	0.62	746,201	23,991	0.08	999	2,202
Total Inferred	**1,291**	**0.61**	**789,671**	**25,389**	**0.08**	**1,044**	**2,300**

TABLE 14-3 SUMMARY OF MINERAL RESOURCES BY ZONE – DECEMBER 31, 2017

GR Engineering (Barbados), Inc. – Siembra Minera Project

Zone Name	Tonnes (Mt)	Grade (g/t Au)	Grade (%Cu)	Contained Gold (kg)	Contained Gold (koz)	Contained Copper (kt)	Contained Copper (Mlb)
			Measured				
Brisas	9	0.93	0.17	8,187	263	15	32
Cristinas	1	1.87	0.29	1,717	55	3	6
Mesones	-	-	-		-	-	-
Morrocoy	-	-	-		-	-	-
Cordova	-	-	-		-	-	-
Total, Measured	*10*	*1.02*	*0.18*	*9,904*	*318*	*17*	*38*
			Indicated				
Brisas	594	0.58	0.09	343,943	11,058	563	1,241
Cristinas	450	0.88	0.10	396,477	12,747	468	1,030
Mesones	76	0.65	0.22	49,221	1,582	164	361
Morrocoy	1	0.86	-	933	30	-	-
Cordova	53	0.63	0.01	33,800	1,087	8	17
Total, Indicated	1,174	0.70	0.10	824,374	26,504	1,202	2,649
			Measured & Indicated				
Brisas	603	0.58	0.10	352,130	11,321	578	1,273
Cristinas	451	0.88	0.10	398,194	12,802	470	1,036
Mesones	76	0.65	0.22	49,221	1,582	164	361
Morrocoy	1	0.86	-	933	30	-	-
Cordova	53	0.63	0.01	33,800	1,087	8	17
Sub-Total, M&I	**1,184**	**0.70**	**0.10**	**834,278**	**26,823**	**1,219**	**2,687**
			Inferred				
Brisas	364	0.47	0.12	170,731	5,489	441	971
Cristinas	761	0.70	0.07	530,775	17,065	517	1,140
Mesones	51	0.35	0.17	18,006	579	85	186
Morrocoy	92	0.60	-	55,046	1,770	-	-
Cordova	23	0.67	0.01	15,114	486	1	3
Total, Inferred	**1,291**	**0.61**	**0.08**	**789,671**	**25,389**	**1,044**	**2,300**

Definitions for resource categories used in this report are consistent with those defined by CIM (2014) and adopted by NI 43-101. In the CIM classification, a Mineral Resource is defined as “a concentration or occurrence of solid material of economic interest in or on the Earth’s crust in such form, grade or quality and quantity that there are reasonable prospects for eventual economic extraction”. Mineral Resources are classified into Measured, Indicated, and Inferred categories. A Mineral Reserve is defined as the “economically mineable part of a Measured and/or Indicated Mineral Resource” demonstrated by studies at Pre-Feasibility or Feasibility level as appropriate. Mineral Reserves are classified into Proven and Probable categories.

Metal prices used for reserves are based on consensus, long term forecasts from banks, financial institutions, and other sources. For resources, metal prices used are slightly higher than those used for reserves.

RPA is not aware of any environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant factors that could materially affect the Mineral Resource estimate.

DRILL HOLE DATABASE

The resource database contains drilling information and analytical results up 2006 for the Brisas concessions and up to 1997 for the Cristinas concessions. The database comprises 975 drill holes and four trenches for Brisas for a total of 207,442 m of drilling and 1,182 drill holes for Cristinas for a total of 155,454 m of drilling. These drill holes and channels were internally reviewed and were found to be acceptable to support Mineral Resource estimation.

RPA received data from GRI in comma separated values (.csv) format, as well as GEMS files and Geolog files. Data were amalgamated and parsed as required and imported by RPA into Surpac 6.8 and Aranz's Leapfrog Geo software.

Section 12, Data Verification, describes the resource database verification steps made by RPA. RPA is of the opinion that the drill hole database is valid and suitable to estimate Mineral Resources for the Project. The locations of the combined drill holes for the Brisas and Cristinas concessions was presented in Section 10.

TOPOGRAPHY

The topography used for this study is based on the Behre Dolbear topography, which is compiled from two sources.

GRI provided to Behre Dolbear digital topographic contours for the Brisas area in the UTM grid system. The contour interval is one metre for the central portion of the Brisas area and is five metres for the surrounding areas. The one metre contours are generated from ground surveys and are believed to be quite accurate. The five metre contours for the surrounding area are less accurate than the one metre contours.

Topographic data for the Cristinas area come from a Cristinas mine planning map provided by GRI to Behre Dolbear, in which topographic contours in 2.5 metre intervals are present. Behre Dolbear extracted these contours from the map and converted them from the local Cristinas mine grid system to the UTM grid system.

In preparation for more detailed engineering studies and in light of the extensive artisanal mining, RPA recommends that a new Digital Terrain Model (DTM) model be generated for the Project.

GEOLOGICAL INTERPRETATION

Wireframes of the stratiform mineralization in Brisas and Cristinas and hydrothermal quartz-tourmaline breccias in Mesones were created by RPA in Aranz's Leapfrog Geo software using approximately a 0.20 g/t Au cut-off grade for gold domains and a 0.04% Cu cut-off grade for copper domains, taking into consideration wireframes prepared by previous workers, and drill hole lithological and assay information. The earlier mineralization wireframes in Brisas were constructed by PAH considering geology and using a 0.25 g/t Au cut-off and a 0.08% Cu cutoff on vertical sections spaced 25 m apart in GEOVIA's GEMS software then transferred to plan views spaced six metres apart and digitized on a bench-by-bench basis.

The strataform mineralization in the Brisas and Cristinas (Potaso, Conductora, Cuatro Muertos) zones strikes at approximately 015° azimuth and extends along a strike length of over 5,000 m. This strataform mineralization dips approximately 35° to the west and has been modelled to the surface where appropriate. The strataform mineralization in the Cordova and Morrocoy zones strikes at approximately 310°, extends for over 800 m in strike length, and dips approximately 80° towards the southwest. The breccia mineralization in Mesones is present in two elliptical areas of approximately 600 m by 400 m and dips between 80° and 90°.

The strataform mineralization in the Brisas and Cristinas zone consists of a main zone, five hanging wall zones and one foot wall zone. The Brisas-Cristinas main zone has a minimum thickness of 10 m at the south end and reaches a maximum thickness of 350 m. The average thickness of the main zone of strataform mineralization is approximately 200 m. The strataform hanging wall and footwall zones have a minimum thickness of 10 m and a maximum thickness of 100 m with an average thickness of approximately 50 m.

The Cordova and Morrocoy strataform mineralization zones have a minimum thickness of 10 m and a maximum thickness of 200 m, with an average thickness between 40 m and 60 m.

Figure 14-1 shows a 3D perspective view of the wireframes and Figures 14-2 and 14-3 show example sections of the mineralization. A longitudinal view of the stratiform mineralization located on the Brisas and Cristinas concessions is presented in Figure 14-4.

RPA reviewed the mineralized wireframes against previous geological interpretations and drill hole information.

A geological model has been prepared over the deposit areas, delineating a series of faults, intrusives and weathering profiles. A total of 24 wireframes were constructed to represent the gold mineralization zones and six wireframes to represent the copper mineralization zones. A total of 16 wireframes were constructed to represent barren dioritic, mafic, and aplitic dykes. The existing oxidation contact surfaces between the fresh rock and the sulphide saprolite and between the oxide saprolite and the sulphide saprolite were adjusted by RPA based on drill hole data and the geological model and smoothed where required to produce a more gradual surface. The contact surface between the sulphide saprolite and the oxide saprolite was adjusted so that areas with copper (i.e., sulphide saprolite) are always below that contact.

Figure 14-1

GR Engineering (Barbados), Inc.

Siembra Minera Project

Bolivar State, Venezuela

3D Isometric View of Geological Wireframes

Legend:

- Main Zone
- Cordova
- Morrocoy
- Mesones, HG
- Mesones, LG
- Main Zone HW and FW Orebodies

0 250 500 750 1000
Metres

March 2018

Source: RPA, 2018.

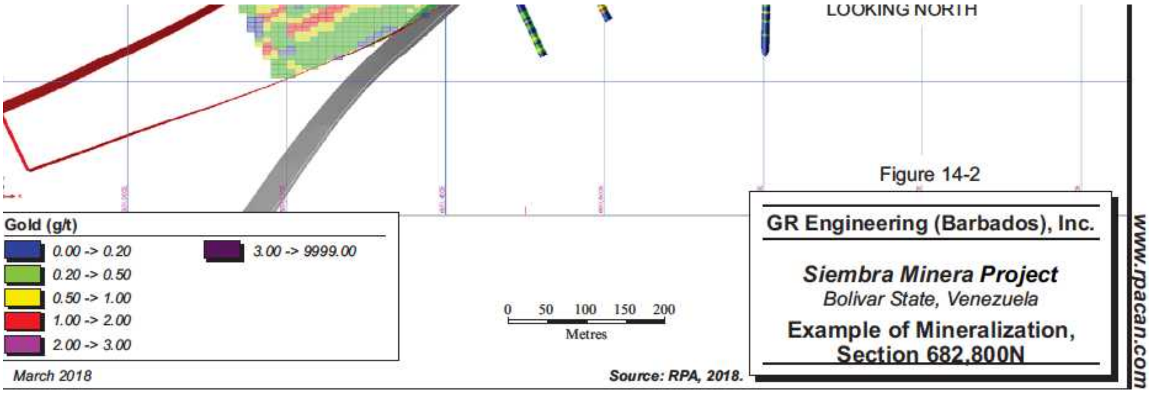


Figure 14-2

GR Engineering (Barbados), Inc.

Siembra Minera Project

Bolivar State, Venezuela

Example of Mineralization, Section 682,800N

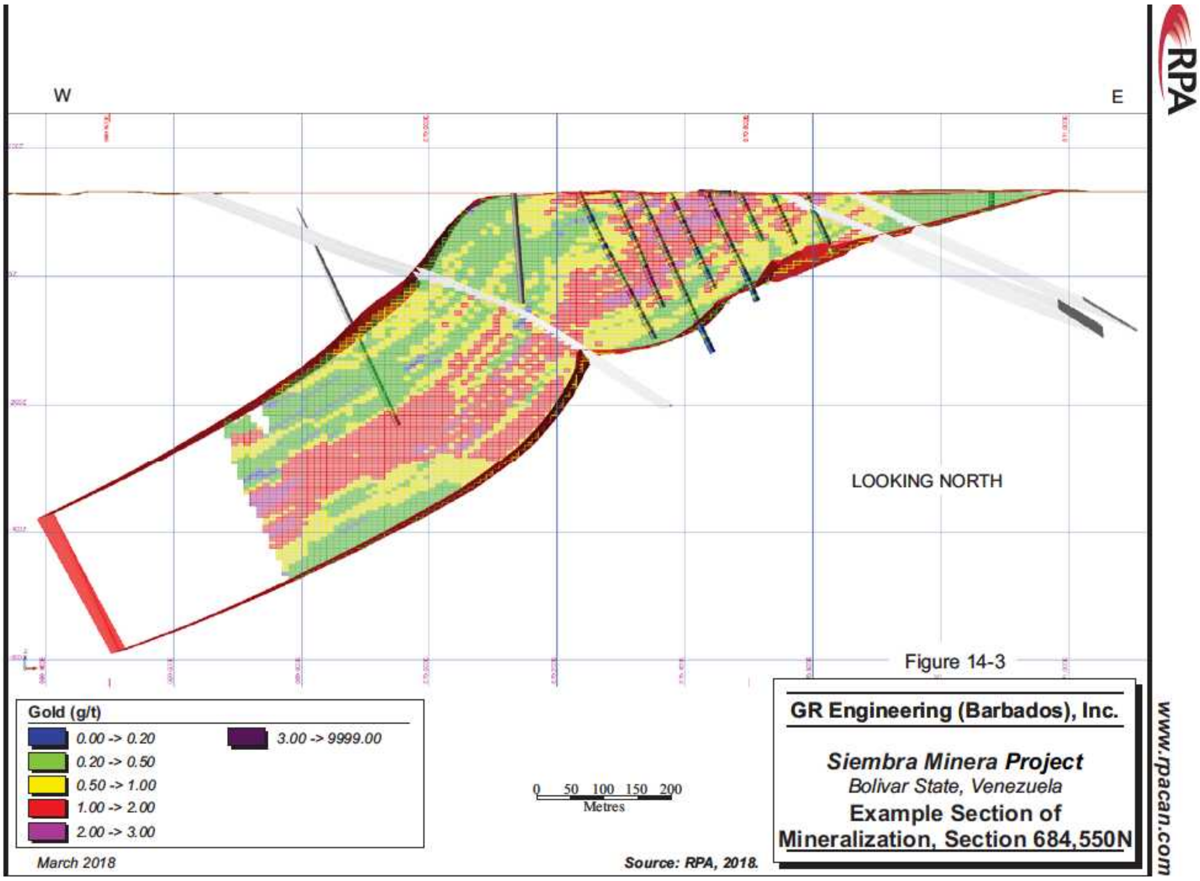


Figure 14-3

GR Engineering (Barbados), Inc.

Siembra Minera Project

Bolivar State, Venezuela

Example Section of Mineralization, Section 684,550N

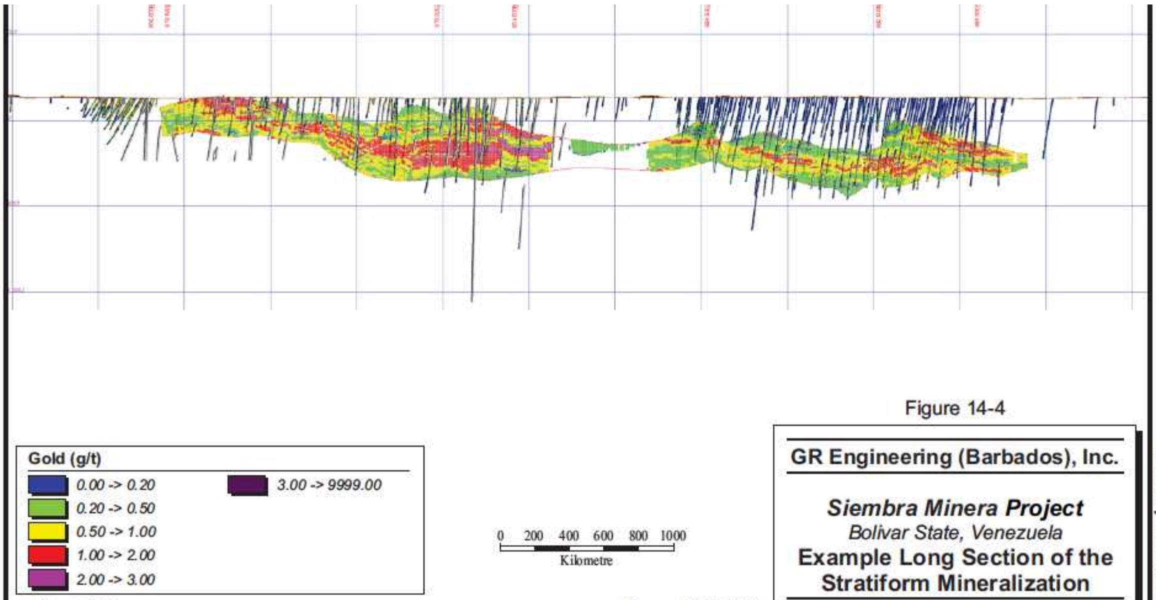


Figure 14-4

GR Engineering (Barbados), Inc.

Siembra Minera ***Project***

Bolivar State, Venezuela

Example Long Section of the Stratiform Mineralization

March 2018

Source: RPA, 2018.

STATISTICAL ANALYSIS

Assay values located inside the wireframe models were tagged with domain identifiers and exported for statistical analysis. Results were used to help verify the modelling process. Basic statistics of the uncapped assays for gold and copper are summarized in Tables 14-4 and 14-5, respectively.

TABLE 14-4 DESCRIPTIVE STATISTICS OF UNCAPPED GOLD ASSAY VALUES BY DOMAIN

GR Engineering (Barbados), Inc. – Siembra Minera Project

Statistic/Zone	**Main Zone**	**A**	**B & C**	**D**	**E**	**F**	**Blue Whale**	**Cordova & Morrocoy**	Mesones **W LG**	Mesones **W HG**	Mesones **E LG**	Mesones **E HG**
No. of cases	85,629	484	3,048	1,812	533	2,180	418	14,601	10,359	2,184	12,311	1,720
Minimum (g/t)	0.00	0.01	0.00	0.01	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Maximum (g/t)	1,296.50	148.66	19.60	51.71	15.60	16.05	35.25	617.00	128.00	96.60	135.10	80.83
Median (g/t)	0.42	0.27	0.26	0.26	0.19	0.27	1.64	0.24	0.51	1.37	0.37	1.53
Arithmetic Mean (g/t)	0.91	1.18	0.50	0.37	0.29	0.53	2.79	0.78	0.64	2.15	0.75	2.73
Weighted Mean (g/t)	0.80	1.35	0.51	0.36	0.28	0.54	2.77	0.71	0.70	2.09	0.75	2.75
Standard Deviation (g/t)	4.29	9.52	1.02	1.15	0.74	0.99	3.71	6.70	2.04	3.29	2.74	4.55
Coef. Of Var.	5.35	7.05	2.00	3.15	2.62	1.86	1.34	9.49	2.90	1.58	3.66	1.65

TABLE 14-5 DESCRIPTIVE STATISTICS OF UNCAPPED COPPER ASSAY VALUES BY DOMAIN
GR Engineering (Barbados), Inc. – Siembra Minera Project

Statistic/Zone	Main Zone	Cordova	Mesones W	Mesones E
No. of cases	127,976	1,451	13,584	12,239
Minimum (%)	0.00	0.00	0.00	0.00
Maximum (%)	9.08	3.88	17.43	8.92
Weighted Mean (%)	0.08	0.09	0.32	0.24
Standard Deviation (%)	0.19	0.24	0.53	0.37
Coef. Of Var.	2.48	2.54	1.68	1.56

CAPPING OF HIGH GRADES

RPA applied high grade capping in order to limit the influence of a small number of extremely high values located in the upper tail of the metal distributions (Figures 14-5 to 14-8 for examples at Brisas and Cristinas). Log probability plots were inspected for all of the gold and copper wireframes and some domains were combined to provide more statistically significant results. A summary of capping grades used for each of the mineralized wireframe models is provided in Table 14-6. Descriptive statistics of the capped gold and copper assays are presented in Tables 14-7 and 14-8, respectively.

FIGURE 14-5 FREQUENCY HISTOGRAM OF THE GOLD VALUES FOR BRISAS MAIN ZONE

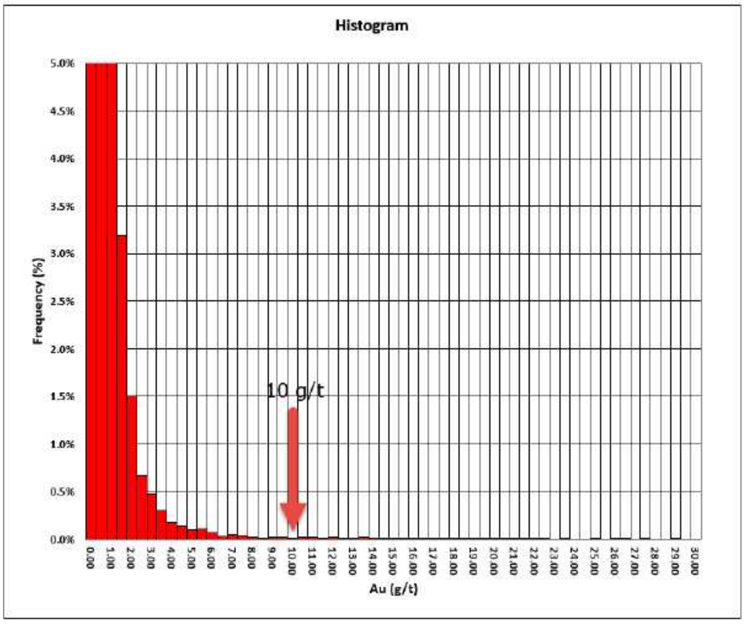


FIGURE 14-6 FREQUENCY HISTOGRAM OF THE GOLD VALUES FOR CRISTINAS MAIN ZONE

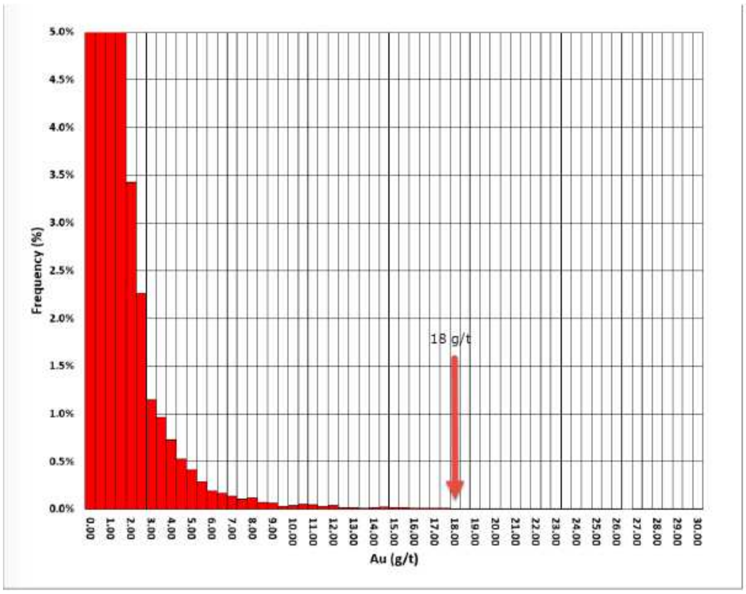


TABLE 14-6 SUMMARY OF GOLD AND COPPER CAPPING VALUES
GR Engineering (Barbados), Inc. – Siembra Minera Project

Domain	Au g/t	Cu %
Main-Brisas	10	2.40
Main-Cristinas	18	2.40
A	15	2.40
B & C	3.5	2.40
D	1.8	2.40
E	1.8	2.40
F	5.0	2.40
Blue Whale	20.0	2.40
Cordova and Morrocoy	15.0	0.30 to 1.80
Mesones W, LG	10.0	2.00
Mesones W, HG	20.0	2.00
Mesones W, LG	10.0	2.20
Mesones W, HG	20.0	2.20

FIGURE 14-7 PROBABILITY PLOTS OF THE GOLD VALUES FOR BRISAS MAIN

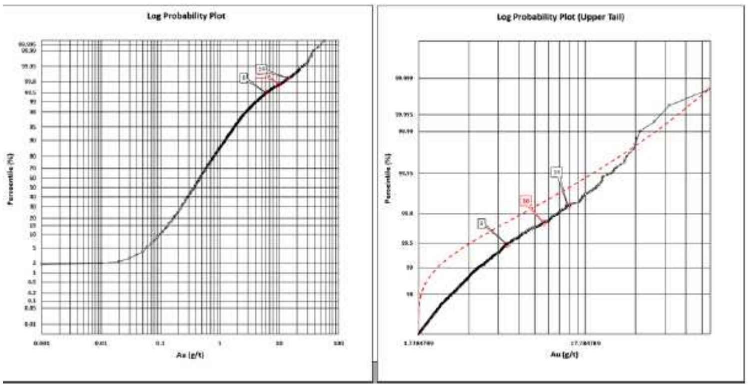


FIGURE 14-8 PROBABILITY PLOTS OF THE GOLD VALUES FOR CRISTINAS MAIN ZONE

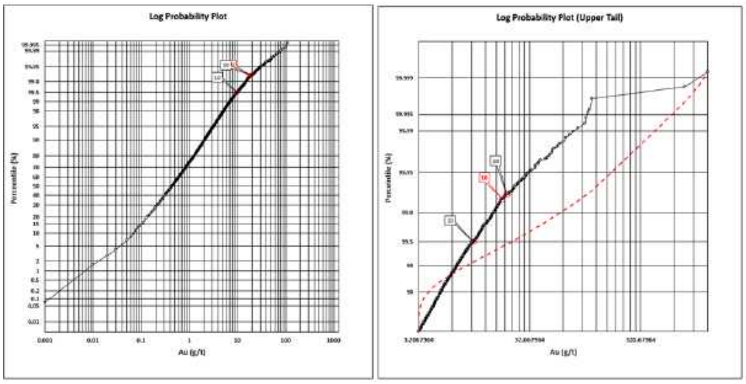


TABLE 14-7 DESCRIPTIVE STATISTICS OF GOLD CAPPED ASSAY VALUES BY DOMAIN

GR Engineering (Barbados), Inc. – Siembra Minera Project

Statistic/Zone	Main Zone Brisas	Main Zone Cristinas	A	B&C	D	E	F	Blue Whale	Cordova & Morrocoy	Mesones W LG	Mesones W HG	Mesones E LG	Mesones E HG
No. of cases	30,061	55,769	484	3,048	1,812	533	2,180	418	14,601	10,359	2,184	12,311	1,720
Minimum (g/t)	0.00	0.00	0.01	0.00	0.01	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Maximum (g/t)	10.00	18.00	15.00	3.50	1.80	1.80	5.00	20.00	15.00	10.00	20.00	10.00	20.00
Weighted Mean (g/t)	0.61	1.03	0.74	0.47	0.33	0.25	0.51	2.70	0.57	0.66	2.01	0.66	2.60
Standard Deviation (g/t)	0.87	1.55	2.00	0.60	0.26	0.28	0.75	3.24	1.32	1.02	2.30	1.11	3.28
Coef. Of Var.	1.43	1.51	2.70	1.29	0.78	1.12	1.48	1.20	2.31	1.55	1.15	1.68	1.26
Number of Caps	82	98	5	44	12	5	22	4	57	40	10	75	20

TABLE 14-8 DESCRIPTIVE STATISTICS OF COPPER CAPPED ASSAY VALUES BY DOMAIN

GR Engineering (Barbados), Inc. – Siembra Minera Project

Statistic/Zone	Main Zone	Cordova	Mesones W	Mesones E
No. of cases	127,976	1,451	13,584	12,239
Minimum (%)	0.00	0.00	0.00	0.00
Maximum (%)	2.50	1.40	2.00	2.20
Weighted Mean (%)	0.07	0.09	0.31	0.24
Standard Deviation (%)	0.17	0.17	0.47	0.35
Coef. Of Var.	2.24	1.96	1.52	1.45
Number of Caps	69	12	225	42

COMPOSITING

RPA composited assays to three metres, which corresponds to half the height of each bench (six metres). Assays were capped prior to compositing. Composites started at the top of each mineralized wireframe. The last composite in each wireframe must be at least 1.5 m to be used, otherwise it is discarded. Composites were weighted by length. Un-sampled core intervals were treated as null values when samples were isolated and clearly unrelated to a barren structure (e.g. mafic dykes) and allocated a value of zero when deemed to be part of a barren structure. Figure 14-9 shows a histogram of sample lengths for all domains combined.

The statistics for the capped, composited copper and gold grades are provided in Tables 14-9 and 14-10, respectively.

FIGURE 14-9 HISTOGRAM OF SAMPLE LENGTHS, ALL DOMAINS COMBINED

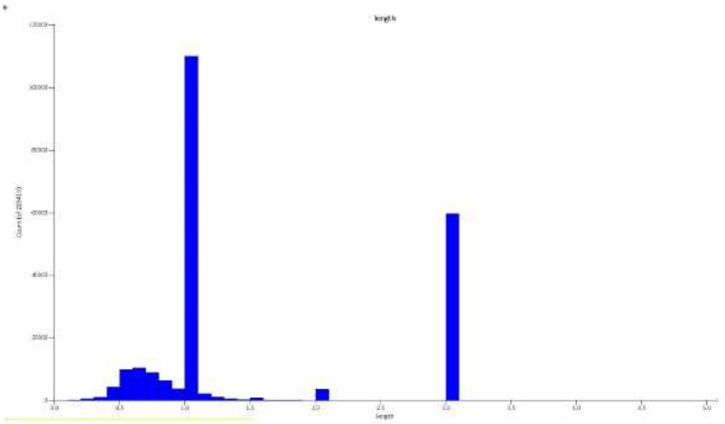


TABLE 14-9 DESCRIPTIVE STATISTICS OF CAPPED, COMPOSITED COPPER VALUES

GR Engineering (Barbados), Inc. - Siembra Minera Project

Statistic/Zone	Main Zone	Cordova	Mesones W	Mesones E
No. of cases	74,695	455	4,223	3,789
Minimum (%)	0.01	0.01	0.01	0.01
Maximum (%)	2.40	0.91	2.40	2.30
Weighted Mean (%)	0.08	0.09	0.30	0.23
Standard Deviation (%)	0.15	0.10	0.41	0.28
Coef. Of Var.	1.83	1.16	1.38	1.22

TABLE 14-10 **DESCRIPTIVE STATISTICS OF CAPPED, COMPOSITED GOLD VALUES BY DOMAIN**

GR Engineering (Barbados), Inc. – Siembra Minera Project

Statistic	Main Zone Brisas	Main Zone Cristinas	A	B&C	D	E	F	Blue Whale	Cordova & Morrocoy	Mesones W LG	Mesones W HG	Mesones E LG	Mesones E HG
No.of cases	28,916	17,172	383	539	1,715	1,001	786	418	4,534	3,149	674	3,810	541
Minimum (g/t)	0.00	0.01	0.01	0.00	0.01	0.00	0.00	0.00	0.006	0.00	0.00	0.00	0.00
Maximum (g/t)	10.00	16.50	15.00	1.80	1.80	3.50	3.97	20.00	12.00	6.74	15.94	8.23	13.19
Weighted Mean (g/t)	0.59	1.08	0.74	0.25	0.33	0.43	0.44	2.70	1.676	0.64	1.87	0.64	2.41
Standard Deviation (g/t)	0.78	1.19	1.96	0.27	0.25	0.44	0.54	3.20	0.889	0.70	1.62	0.78	2.21
Coef. Of Var.	1.31	1.10	2.64	1.07	0.76	1.02	1.21	1.19	1.57	1.10	0.87	1.21	0.92

VARIOGRAPHY

Experimental traditional variograms were generated for the main mineralization envelopes at Brisas and Cristinas and fit with two spherical models in three orthogonal directions for gold. Variograms were standardized and in general, there is a good agreement between the experimental sill and the variance of the distribution. The variograms for gold in Brisas and Cristinas are shown in Figures 14-10 and 14-11.

RPA has interpreted moderate nugget effect values ranging from 0.2 to 0.5. The variogram models exhibit steep first structures with a large proportion of the variance accounted for over short distances. The variograms were used as a guide for selecting search ellipse ranges and anisotropy ratios.

It is recommended that additional detailed variography work be carried out for gold and copper in the future.

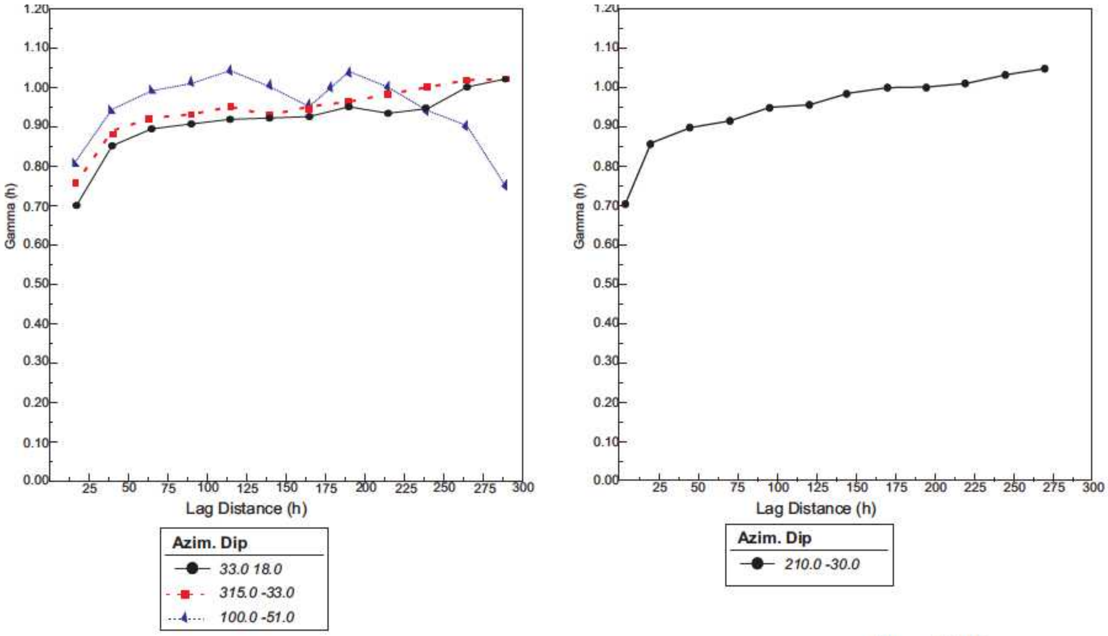


Figure 14-10

GR Engineering (Barbados), Inc.

Siembra Minera Project

Bolivar State, Venezuela

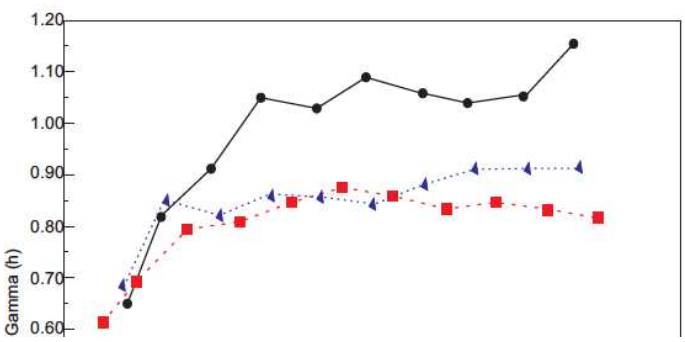
Gold Variograms for Brisas

Second Structure - - Spherical

LH Rotation about the Z axis ==> -57
RH Rotation about the Y' axis ==> 72
LH Rotation about the Z' axis ==> 35
Range along the Z' axis ==> 408.4
Range along the Y' axis ==> 250.0
Range along the X' axis ==> 57.1

Azimuth ==> 33 Dip ==> 18
Azimuth ==> 315 Dip ==> -33
Azimuth ==> 100 Dip ==> -51

Source: RPA, 2018.



0.50

14 -
24

0.40

0.30

0.20

0.10

0.00

25 50 75 100 125 150 175 200 225 250 275 300

Lag Distance (h)

Second Structure - - Spherical

Azim. Dip			
	LH Rotation about the Z axis ==> 12		
	RH Rotation about the Y' axis ==> -40	Azimuth ==> 282	Dip ==> 50
282.0 50.0	LH Rotation about the Z' axis ==> -8	Azimuth ==> 6	Dip ==> -5
91.0 40.0	Range along the Z' axis ==> 90.9	Azimuth ==> 91	Dip ==> 40
	Range along the Y' axis ==> 481.2		
6.0 -5.0			
	Range along the X' axis ==> 516.1		

Figure 14-11

GR Engineering (Barbados), Inc.

Siembra Minera Project

Bolivar State, Venezuela

Gold Variograms for Cristinas

www . rpacan . com

March 2018

Source: RPA, 2018.

DENSITIES

RPA modified where required the oxidation surfaces of fresh rock and sulphide saprolite and used these to reallocate the database density samples to the new oxide saprolite, sulphide saprolite, and un-weathered (fresh) rock domains.

A total of 2,456 dry weight measurements and 3,464 wet weight measurements from the Brisas concessions collected between 1996 and 1998 exist in the database. The Brisas samples are divided into 27 rock types, from which the main weathering domains for oxide saprolite, sulphide saprolite, and un-weathered (fresh) rock can be extracted (Figure 14-12). RPA verified the same mean density values for sulphide saprolite as were reported previously by PAH 2008 and very similar density values for oxide saprolite and fresh rock (Table 14-11).

No density data was present in the database for the historical Cristinas property (Table 14-12) outside of the Mesones area (Table 14-13), where a total of 875 dry weight measurements and 876 wet weight measurements from the Mesones area of the Cristinas Project exist in the database. It is not known when the Mesones samples were collected and no details regarding the density sample collection or weight measurement procedures are available. It was not possible to extract the historical oxidation domains for the Mesones density samples. The densities obtained in this manner for Brisas were very close to historical values but lower in Mesones, especially in the case of the fresh rock. The slight difference in sulphide saprolite density between historical and RPA values in Mesones is attributed to the Mesones density database being an incomplete dataset.

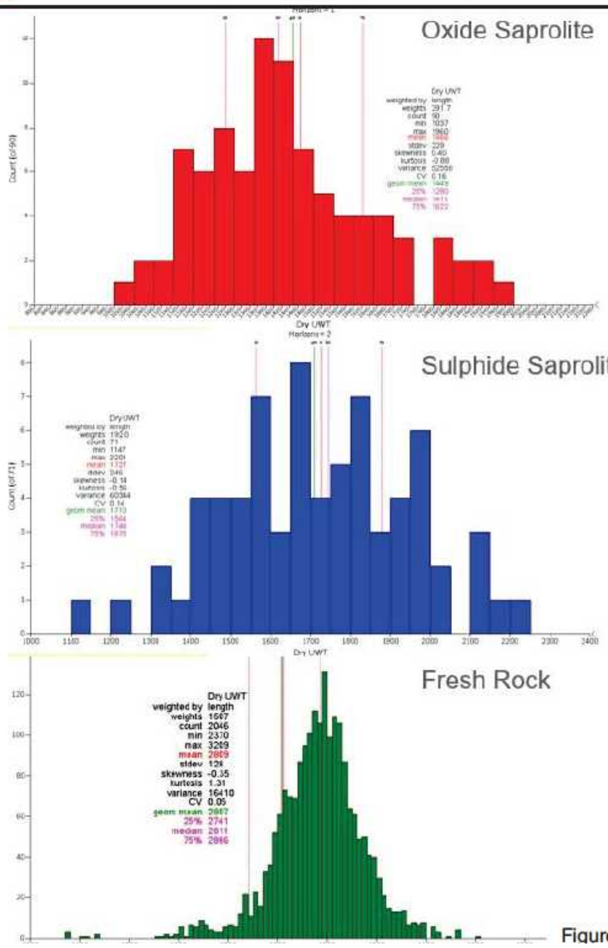


Figure 14-12

GR Engineering (Barbados), Inc.

Siembra Minera Project
Bolivar State, Venezuela
BDrisas ensity Statistics by
Historic Oxidation Domain

March 2018

Source: RPA, 2018.

TABLE 14-11 DENSITY STATISTICS FOR THE BRISAS CONCESSIONS, BY MINERALIZED DOMAIN AND STUDY

GR Engineering (Barbados), Inc. – Siembra Minera Project

Estimate	Density t/m^3 PAH 2008	Density t/m^3 RPA 2017, Historic Oxidation Domain	Density t/m^3 RPA 2017, New Oxidation Surfaces
No. of Samples	N/A	2,456	2,456
Oxide Saprolite	1.43	1.46	1.45
Sulphide Saprolite	1.72	1.72	1.72
Hard Rock (Fresh)	2.83	2.81	2.85

TABLE 14-12 DENSITY STATISTICS FOR THE CRISTINAS CONCESSIONS, BY MINERALIZED DOMAIN AND STUDY

GR Engineering (Barbados), Inc. – Siembra Minera Project

Estimate	Density t/m^3 MDA 2007	Density t/m^3 RPA 2017, Historic Oxidation Domain	Density t/m^3 RPA 2017, New Oxidation Surfaces
No. of Samples	N/A	N/A	N/A
Oxide Saprolite	1.56	N/A	N/A
Sulphide Saprolite	1.69	N/A	N/A
Hard Rock (Fresh)	2.79	N/A	N/A

TABLE 14-13 DENSITY STATISTICS FOR THE MESONES AREA, CRISTINAS CONCESSIONS, BY MINERALIZED DOMAIN AND STUDY

GR Engineering (Barbados), Inc. – Siembra Minera Project

Estimate	Density t/m^3 MDA 2007	Density t/m^3 RPA 2017, Historic Oxidation Domain	Density t/m^3 RPA 2017, New Oxidation Surfaces
No. of Samples	N/A	875	875
Oxide Saprolite	1.68	N/A	1.64
Sulphide Saprolite	1.89	N/A	1.89
Hard Rock (Fresh)	2.79	N/A	2.66

RPA chose to use for Brisas and Cristinas (except Mesones) the same density values as those reported by PAH in 2008 for the oxide saprolite (1.43 t/m^3) and sulphide saprolite (1.72 t/m^3) materials as these values were verified from the database and no verification could be performed on Cristinas density data. The Brisas and Cristinas (including Mesones) density values for fresh rock was chosen to be 2.80 t/m^3 as a compromise between the validated Brisas density of 2.83 (t/m^3) and the larger Cristinas area historical density of 2.79 t/m^3. The Mesones density values for the oxide saprolite and sulphide saprolite used are the same as those reported by MDA in 2007 as the values obtained from the density database were very close in oxide saprolite (1.68 t/m^3) or the same in sulphide saprolite (1.89 t/m^3) as the historical values.

BLOCK MODEL CONSTRUCTION

The block model cells measure 10 m by 10 m by 6 m. The block model setup is given in Table 14-14, while a description of the block model attributes is given in Table 14-15.

In RPA's view, the block model size is appropriate for the drill spacing and proposed mining method and is suitable to support the estimation of Mineral Resources and Mineral Reserves. Comparisons between wireframe and block model volumes are reasonable.

TABLE 14-14 BLOCK MODEL SETUP
GR Engineering (Barbados), Inc. – Siembra Minera Project

Parameter	X	Y	Z
Origin (m)	668,000	680,000	162
Block Size (m)	10	10	6
Number of Blocks	370	740	142
Rotation	0	0	0

TABLE 14-15 BLOCK MODEL ATTRIBUTE DESCRIPTIONS
GR Engineering (Barbados), Inc. – Siembra Minera Project

Attribute Name	Description
au_final	Au, estimated by ID^2
au_pass	Au search pass by ID^2
au_pass_id1	Au search pass by ID^1
au_rpa_id1	Au, estimated by ID^1
class3	Final classification 1 = measured, 2 = indicated, 3 = inferred
cu_final	Cu, estimated by ID^2
density	Density in t/m^3
dip	Dip for dynamic anisotropy method in Au estimation
dip_cu	Dip for dynamic anisotropy method in Cu estimation
dip_direction	Dip direction for dynamic anisotropy method in Au estimation
dip_direction_cu	Dip direction for dynamic anisotropy method in Cu estimation
disturbed	Areas deemed to be disturbed by mining activity 1 = 10 m deep, 2 = 30 m deep
domain	Au wireframe domain, see table 14-12
domain_cu	Cu wireframe domain, see table 14-13
nn_au	Au, estimated by nearest neighbour
nn_cu	Cu, estimated by nearest neighbour
nsr_au	NSR for Au only, from ID^2 estimate
nsr_copper	NSR for Cu only, from ID^2 estimate
nsr_total	Sum of NSR for Au and NSR for Cu
oxide	Oxidation 1 = fresh, 2 = sulphide saprolite, 3 = oxide saprolite
pit_december	1 = inside MII pit shell

Gold and copper grades were estimated into blocks using the ID^2, ID^1, and NN interpolation algorithms using the Surpac v.6.8 software package. The ID^1 and NN interpolation estimates were prepared for the Main Zone only for comparative purposes. Search ellipsoids were oriented based on dynamic anisotropy (DA) angles extracted for the mineralization wireframes for the main gold and copper trends and on general mineralization trends for all other wireframes. For the DA method, the orientations of the search ellipses are varied in response to changes in the azimuth and dips of the mineralization at the local scale so as to improve the accuracy of the local estimate. The search ranges were determined based on drill hole spacing, variogram ranges, and data density and continuity. Minimum and maximum number of composite parameters were adjusted where needed to minimize smoothing of the grades. The sample selection strategy and search ranges are given in Tables 14-16 and 14-17.

TABLE 14-16 GOLD SAMPLE SELECTION STRATEGY

GR Engineering (Barbados), Inc. – Siembra Minera Project

Domain	Domain Code	Pass	Dip Direction	Dip	Search Distance X	Y	Z	Min Comps	Max Comps	Max Comps per DDH
		1	D.A.	D.A.	60	60	20	4	12	3
Main	101	2	D.A.	D.A.	120	120	40	4	12	3
		3	D.A.	D.A.	200	200	66	3	12	6
		1	270	-35	60	60	20	4	12	3
Main_A	102	2	270	-35	90	90	30	4	12	3
		3	270	-35	120	120	40	1	8	3
		1	270	-35	60	60	20	4	12	3
Main_B	103	2	270	-35	90	90	30	4	12	3
		3	270	-35	120	120	40	1	8	3
		1	270	-35	60	60	20	4	12	3
Main_C	104	2	270	-35	90	90	30	4	12	3
		3	270	-35	120	120	40	1	8	3
		1	270	-35	60	60	20	4	12	3
Main_D	105	2	270	-35	90	90	30	4	12	3
		3	270	-35	120	120	40	1	8	3
		1	270	-35	60	60	20	4	12	3
Main_E	106	2	270	-35	90	90	30	4	12	3
		3	270	-35	120	120	40	1	8	3
		1	300	-25	60	60	20	4	12	3
Main_F	107	2	300	-25	90	90	30	4	12	3
		3	300	-25	120	120	40	1	8	3
		1	220	-80	60	60	20	4	12	3
Cordova_01	108	2	220	-80	90	90	30	4	12	3
		3	220	-80	120	120	40	1	8	3
		1	220	-80	60	60	20	4	12	3
Cordova_02	109	2	220	-80	90	90	30	4	12	3
		3	220	-80	120	120	40	1	8	3

Domain	Domain Code	Pass	Dip Direction	Dip	Search Distance X	Y	Z	Min Comps	Max Comps	Max Comps per DDH
		1	220	-80	60	60	20	4	12	3
Cordova_03	110	2	220	-80	90	90	30	4	12	3
		3	220	-80	120	120	40	1	8	3
		1	220	-80	60	60	20	4	12	3
Cordova_04	111	2	220	-80	90	90	30	4	12	3
		3	220	-80	120	120	40	1	8	3
		1	220	-80	60	60	20	4	12	3
Cordova_Main	112	2	220	-80	90	90	30	4	12	3
		3	220	-80	120	120	40	1	8	3
		1	315	-35	60	60	20	4	12	3
Blue Whale	121	2	315	-35	90	90	30	4	12	3
		3	315	-35	120	120	40	1	8	3
Mesones_E_LG	170	1	0	-90	180	60	60	6	12	5
		2	0	-90	240	80	80	1	8	3
		1	0	-90	180	60	60	6	12	5
Mesones_E_HG	171	2	0	-90	240	80	80	1	8	3
Mesones_W_LG	172	1	0	-90	180	60	60	6	12	5
		2	0	-90	240	80	80	1	8	3
Mesones_W_HG	173	1	0	-90	180	60	60	6	12	5
		2	0	-90	240	80	80	1	8	3
		1	220	-80	60	60	20	4	12	3
Morrocoy_1	181	2	220	-80	90	90	30	4	12	3
		3	220	-80	120	120	40	1	8	3
		1	220	-80	60	60	20	4	12	3
Morrocoy_2	182	2	220	-80	90	90	30	4	12	3
		3	220	-80	120	120	40	1	8	3
		1	220	-80	60	60	20	4	12	3
Morrocoy_3	183	2	220	-80	90	90	30	4	12	3
		3	220	-80	120	120	40	1	8	3
		1	220	-80	60	60	20	4	12	3
Morrocoy_4	184	2	220	-80	90	90	30	4	12	3
		3	220	-80	120	120	40	1	8	3
		1	220	-80	60	60	20	4	12	3
Morrocoy_5	185	2	220	-80	90	90	30	4	12	3
		3	220	-80	120	120	40	1	8	3
		1	220	-80	60	60	20	4	12	3
Morrocoy_6	186	2	220	-80	90	90	30	4	12	3
		3	220	-80	120	120	40	1	8	3
		1	220	-80	60	60	20	4	12	3
Morrocoy_7	187	2	220	-80	90	90	30	4	12	3
		3	220	-80	120	120	40	1	8	3
		1	220	-80	60	60	20	4	12	3
Morrocoy_8	188	2	220	-80	90	90	30	4	12	3
		3	220	-80	120	120	40	1	8	3

TABLE 14-17 COPPER SAMPLE SELECTION STRATEGY

GR Engineering (Barbados), Inc. – Siembra Minera Project

Domain	Domain Code	Pass	Dip Direction	Dip	Search Distance X	Search Distance Y	Search Distance Z	Min Comps	Max Comps	Max Comps per DDH
Main	201	1	D.A.	D.A.	80	80	13	4	12	3
		2	D.A.	D.A.	120	120	10	4	12	3
		1	220	-80	60	60	20	4	12	3
Cordova_Main	230	2	220	-80	90	90	30	4	12	3
		3	220	-80	120	120	40	1	8	3
		1	220	-80	60	60	20	4	12	3
Cordova_01	231	2	220	-80	90	90	30	4	12	3
		3	220	-80	120	120	40	1	8	3
		1	220	-80	60	60	20	4	12	3
Cordova_02	232	2	220	-80	90	90	30	4	12	3
		3	220	-80	120	120	40	1	8	3
		1	220	-80	60	60	20	4	12	3
Cordova_03	233	2	220	-80	90	90	30	4	12	3
		3	220	-80	120	120	40	1	8	3
Mesones W	272	1	0	-90	120	120	40	5	6	12
		2	0	-90	240	240	80	1	8	3
Mesones E	270	1	0	-90	180	180	60	4	12	2
		2	0	-90	240	240	80	4	12	2
		1	D.A.	D.A.	80	80	13	4	12	3
Main	201	2	D.A.	D.A.	120	120	10	4	12	3

CLASSIFICATION

Definitions for resource categories used in this report are consistent with those defined by CIM (2014) and adopted by NI 43-101. In the CIM classification, a Mineral Resource is defined as “a concentration or occurrence of solid material of economic interest in or on the Earth’s crust in such form, grade or quality and quantity that there are reasonable prospects for eventual economic extraction”. Mineral Resources are classified into Measured, Indicated, and Inferred categories.

Blocks were classified as Measured, Indicated, and Inferred based on drill hole spacing and variograms. Flagging of the blocks by drill hole spacing was done initially by using cylinders centred on the drill holes traces of 25 m radius for Measured and 50 m radius for Indicated Mineral Resources. A clean-up process was subsequently carried out in cross section and plan view to create polylines that would comprise continuous areas of Measured or Indicated categories. Those portions of the mineralized wireframes that were not classified into either the Measured or Indicated categories were classified into the Inferred category. An isometric view showing the final classification is provided in Figure 14-13.

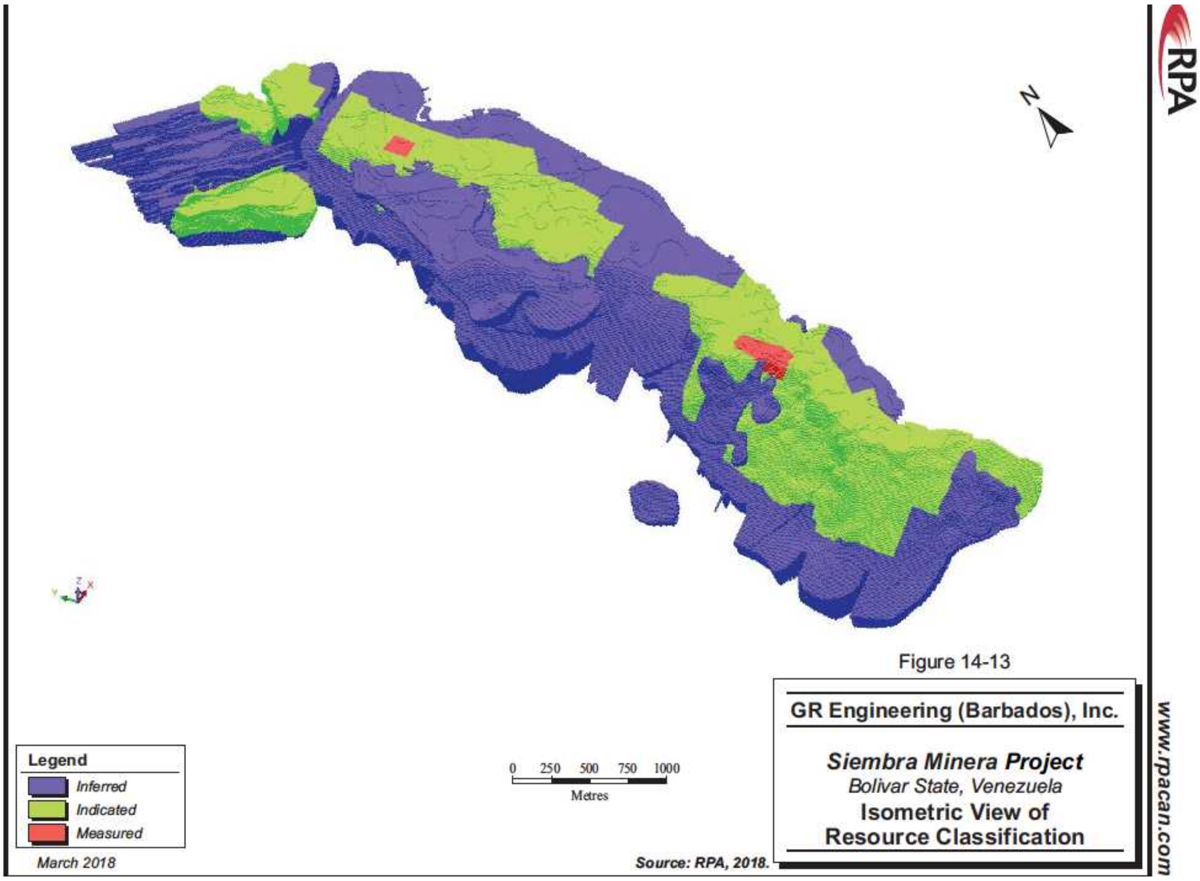


Figure 14-13

GR Engineering (Barbados), Inc.

Siembra Minera Project
Bolivar State, Venezuela

Isometric View of Resource Classification

NET SMELTER RETURN

Due to the fact that both gold and copper grades contribute to the value of the mineralization found at the Siembra Minera Project, RPA elected to adopt the Net Smelter Value (NSR) approach for reporting of the Mineral Resources. In this method, the dollar value that each metal contributes towards the overall total is calculated by applying an appropriate factor for each of the individual metals. At the end of the process, the sum of all of the two metal values is calculated and presented as one value referred to as the NSR value. The NSR value is the estimated dollar value per tonne of mineralized material after allowance for metallurgical recovery and consideration of smelter terms, including revenue from payable metals, treatment charges, refining charges, price participation, penalties, smelter losses, transportation, and sales charges. This NSR value is then used in preparation of the Mineral Resource statements. RPA proceeded to calculate the NSR value using the estimated gold and copper grades for each mineralized block within the block model. The key assumptions used to prepare the NSR factors are listed in Table 14-18, and the resulting NSR factors are presented in Table 14-19.

TABLE 14-18 KEY ASSUMPTIONS FOR CALCULATION OF NSR FACTORS

GR Engineering (Barbados), Inc. – Siembra Minera Project

Category	Input
Metal Prices	US$1,300/oz Au US$3.00/lb Cu
Metal Recovery	Au in Oxide Saprolite: 98% Au in Sulphide Saprolite and Hard Rock: 83% Cu in Oxide Saprolite: 0.0% Cu in Sulphide Saprolite and Hard Rock: 87% Au from Sulphide Sap: 133.82 g/t
Concentrate Grade	Au from Hard Rock: 113.53 g/t Cu from Sulphide Sap and Hard Rock: 24.0% Au from Gold Gravity: Maximum 100%
Payable Metal	Au in Cu Conc: Maximum 97.5% Cu in Cu Conc: Maximum 96.5% Au Doré: C$2.15/oz Au
Transportation	Cu Conc.:C$60.00/wmt Conc. Au Doré: US$1.00/oz
Treatment	Cu Concentrate: US$90.00/t conc. Au Doré: US$5.00/oz
Refining	Au in Concentrate: US$5.00/oz Cu in Concentrate: US$0.09/lb Cu
Royalties	6.0% NSR royalty

TABLE 14-19

SUMMARY OF NSR FACTORS

GR Engineering (Barbados), Inc. – Siembra Minera Project

	NSR Factors		
Metal	**Oxide Saprolite**	**Sulphide Saprolite**	**Hard Rock**
Gold	38.26 x Au (g/t)	31.75 x Au (g/t)	31.75 x Au (g/t)
Copper	n/a	45.50 x Cu (%)	45.50 x Cu (%)

CUT-OFF GRADE

To report Mineral Resources, a NSR cut-off value was estimated. To estimate the NSR cutoff value, the Project was envisaged as a 70,000 tpd mine. To estimate the COG, the following cost estimates were used:

- Mining: US$1.40/t moved
- Processing:
  - US$6.40/t milled for oxide saprolite,
  - US$4.20/t milled for sulphide saprolite,
  - US$4.20/t milled for hard rock,
- General and Administration (G&A): US$0.80/t milled

For the purpose of Mineral Resource reporting, operating costs were estimated at US$7.20/t for the oxide saprolite and US$5.00/t for the sulphide saprolite and fresh rock. This was the basis for the internal NSR cut-off grade using process and administrative costs.

TREATMENT OF ARTISANAL MINER ACTIVITY

The extent of historical workings and ground disturbance due to activity by artisanal miners in the Siembra Minera area is considerable, which has been an on-going activity for approximately 15 years (Figure 14-14). For the most part, the artisanal miners excavate material from the oxide saprolite and sulphide saprolite weathered layers using man-portable equipment and extract the gold by means of sluice boxes and the mercury amalgam process (Figure 14-15). Recent satellite imagery collected in 2017 shows that the cumulative extent of the impact of the artisanal miners is an area measuring greater than seven kilometres (north-south) by four kilometers (east-west).

It is important to note that creating an accurate representation of the volume of material excavated by the artisanal miners is currently impossible due to the presence of water-filled pits, and previously excavated pits that have since been back-filled with tailings from more recently completed excavations. However, the cumulative impact of these activities cannot be overlooked and must be considered when preparing an estimate of the Mineral Resources.

To achieve this, RPA used satellite photography and observations gained during the most recent field visit to create two polygons of disturbed ground (Figure 14-16). The larger polygon was created to represent an area of relatively shallower disturbance and the smaller polygon was created to represent the approximate area of deeper disturbance. Blocks within the shallower disturbance polygon were deemed to be void of mineralization within 10 m of the topographic surface, and blocks within the deeper disturbance polygon were deemed to be void of mineralization within 30 m of the topographic surface.

Figure 14-14

GR Engineering (Barbados), Inc.

Siembra Minera Project
Bolivar State, Venezuela

Overview of Artisinal Miner Excavations

March 2018 *Source: GR Engineering, 2017*

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Figure 14-15

GR Engineering (Barbados), Inc.

Siembra Minera Project

Bolivar State, Venezuela

Example of an Artisinal Miner Excavation

March 2018

Source: GR Engineering, 2017

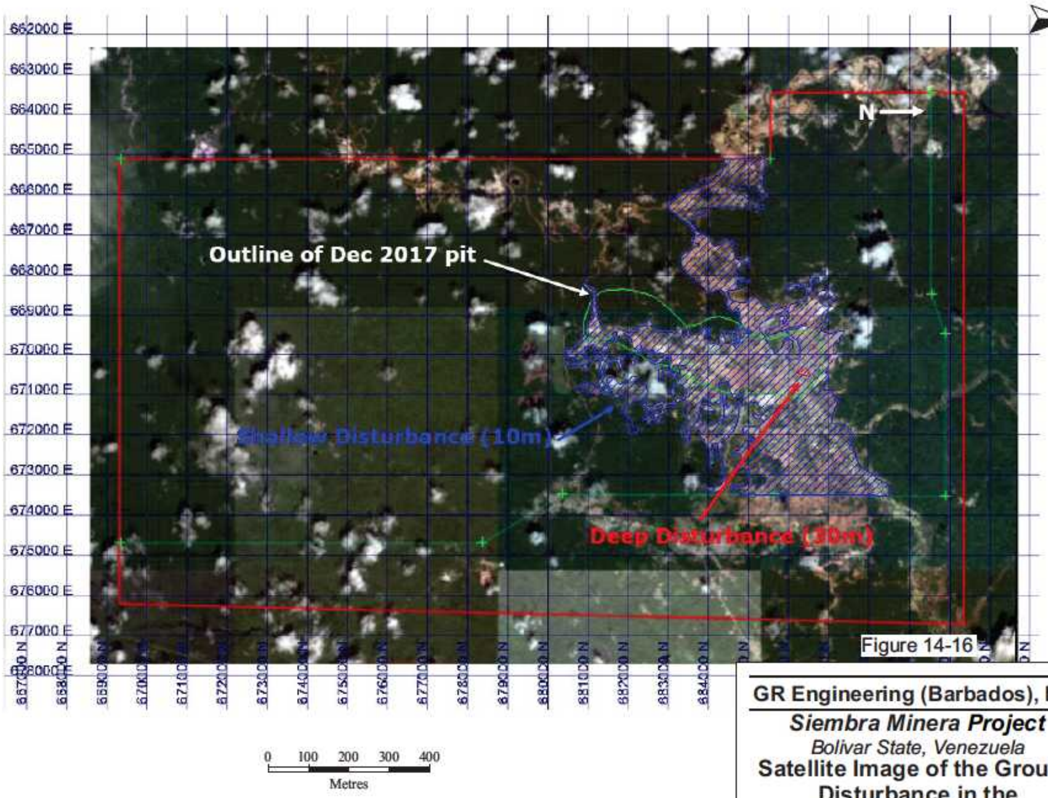

Figure 14-16

GR Engineering (Barbados), Inc.

*Siembra Minera **Project***

Bolivar State, Venezuela

Satellite Image of the Ground Disturbance in the Siembra Minera Area

March 2018

Source: RPA, 2018.

BLOCK MODEL VALIDATION

A number of validation steps were performed by RPA including:

- Global bias comparison between ID^2, ID, and NN grades within the pit on the main Brisas-Cristinas zone using a 0.2 g/t cut-off grade (Table 14-20).
- Local bias comparison between ID^2, ID, and NN grades (Figures 14-17 and 14-18).
- Visual inspection of composites versus block grades (example Figure 14-19).

In RPA's opinion the results presented look reasonable.

RPA recommends that some check estimates be carried out. Overall, the block grades are in reasonable agreement with the underlying composite grades on sections and plans.

TABLE 14-20 COMPARISON BETWEEN OK AND NN GRADES

GR Engineering (Barbados), Inc. – Siembra Minera Project

Category	Tonnes (Mt)	AUID2 g/t	Tonnes (Mt)	AUID1 g/t	Tonnes (Mt)	AUNN g/t
Measured	9	0.78	9	0.78	8	0.86
Indicated	1,018	0.73	1,029	0.73	930	0.79
Meas+Ind	1,026	0.72	1,037	0.72	937	0.78
Inferred	879	0.62	883	0.62	778	0.69

FIGURE 14-17 VALIDATION OF LOCAL BIAS FOR AU IN BRISAS

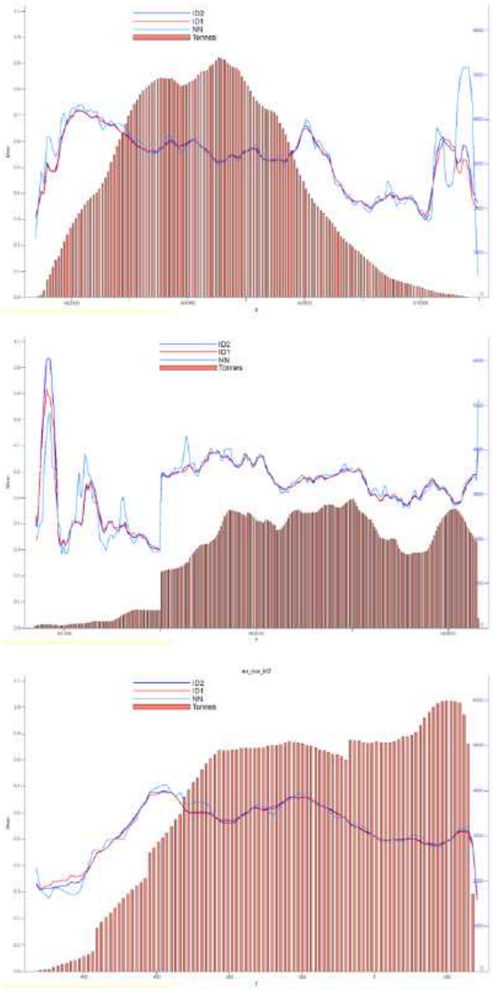

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FIGURE 14-18 VALIDATION OF LOCAL BIAS FOR AU IN CRISTINAS

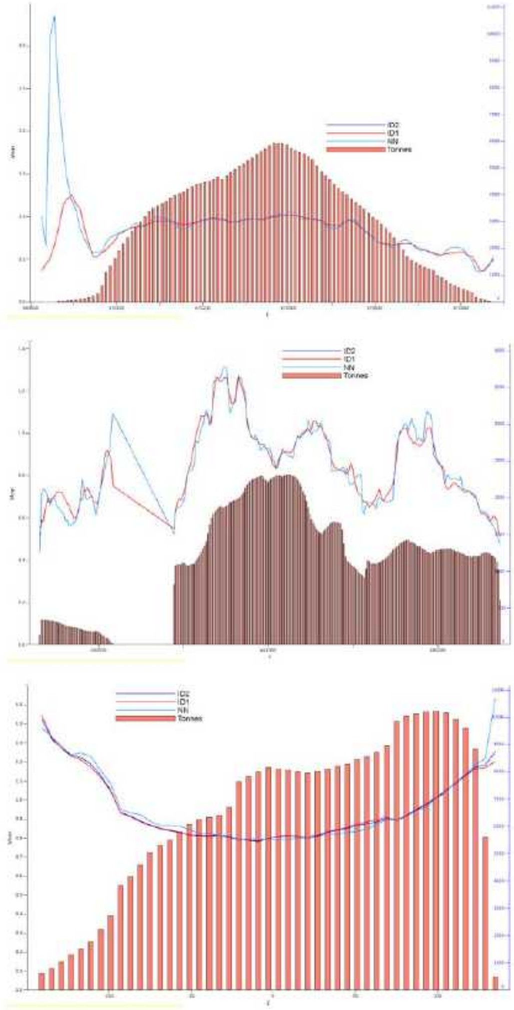
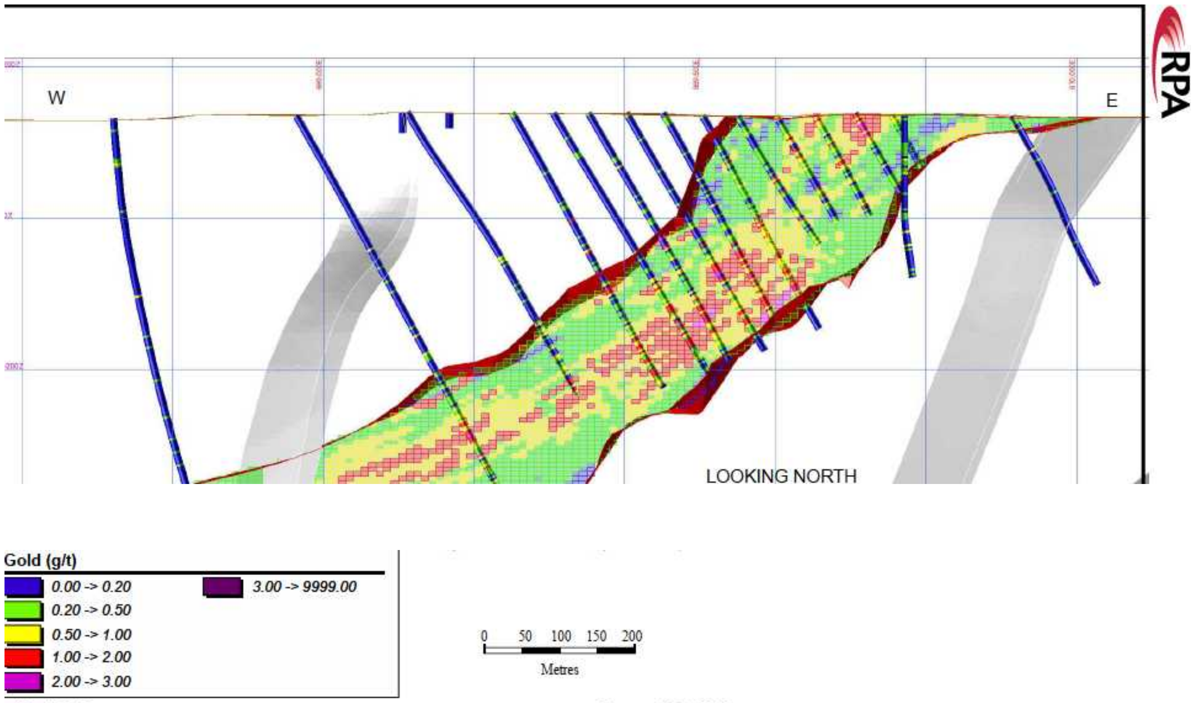


Figure 14-19



March 2018

Source: RPA, 2018.

GR Engineering (Barbados), Inc.
Siembra Minera Project
Bolivar State, Venezuela
Visual Inspection of Composite
Site GradesVs. Block Grades for
Au in Section 682250

OPEN PIT OPTIMIZATION

A preliminary open pit shell was created using the Whittle software package to determine those portions of the modelled mineralization that demonstrate the potential of extraction by means of open pit mining methods. The resulting open pit shell was used as a constraint in the preparation of the Mineral Resource statements. Details regarding the input parameters and results are discussed and presented in Section 16.

MINERAL RESOURCE ESTIMATE

The estimated Mineral Resources using the capped, composited samples are presented in Tables 14-21, 14-22, and 14-23. At a cut-off grade of US$7.20 per tonne for oxide saprolite material and US$5.00 per tonne for sulphide saprolite and fresh rock within the mineral resource pit shell, the Mineral Resources are estimated at 10 million tonnes at an average grade of 1.02 g/t Au and 0.18% Cu containing 318,000 ounces of gold and 17,000 tonnes of copper in the Measured category, 1,174 million tonnes at an average grade of 0.70 g/t Au and 0.10% Cu containing 26,504,000 ounces of gold and 1,202,000 tonnes of copper in the Indicated category. Mineral Resources in the Inferred category are estimated at 1,291 million tonnes at an average grade of 0.61 g/t Au and 0.08% Cu containing 25,388,000 ounces of gold and 1,044,000 tonnes of copper.

TABLE 14-21 SUMMARY OF MINERAL RESOURCES – DECEMBER 31, 2017

GR Engineering (Barbados), Inc. – Siembra Minera Project

Category	Tonnes	Grade	Grade	Contained Gold	Contained Copper	
	(Mt)	(g/t Au)	(% Cu)	(koz Au)	(kt Cu)	(Mlb Cu)
Measured	10	1.02	0.18	318	17	38
Indicated	1,174	0.70	0.10	26,504	1,202	2,649
Total Measured + Indicated	1,184	0.70	0.10	26,823	1,219	2,687
Inferred	1,291	0.61	0.08	25,389	1,044	2,300

Notes:

1. CIM (2014) definitions were followed for Mineral Resources.
2. Mineral Resources are estimated at an NSR cut-off value of US$7.20 per tonne for oxide-saprolite material and US$5.00 per tonne for sulphide-saprolite and fresh rock material.
3. Mineral Resources are constrained by a preliminary pit shell created using the Whittle software package.
4. Mineral Resources are estimated using a long-term gold price of US$1,300 per ounce, and a copper price of US$3.00 per pound.
5. Bulk density varies by material type.
6. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
7. Numbers may not add due to rounding.

TABLE 14-22 SUMMARY OF MINERAL RESOURCES BY MATERIAL TYPE – DECEMBER 31, 2017

GR Engineering (Barbados), Inc. – Siembra Minera Project

Material	Tonnes (Mt)	Grade (g/t Au)	(kg)	Contained Gold (koz)	Grade (%Cu)	(kt)	Contained Copper (Mlb)
			Measured				
Oxide Saprolite	1	0.89	575	18	-	-	-
Sulphide Saprolite	4	1.21	4,750	153	0.18	7	15
Hard Rock	5	0.90	4,579	147	0.20	10	23
Total, Measured	*10*	1.02	9,904	318	0.18	17	38
			Indicated				
Oxide Saprolite	20	0.75	14,857	478	-	-	-
Sulphide Saprolite	110	0.83	90,782	2,919	0.11	124	273
Hard Rock	1,045	0.69	718,736	23,108	0.10	1,078	2,376
Total, Indicated	*1,174*	0.70	824,374	26,504	0.10	1,202	2,649
				Measured + Indicated			
Oxide Saprolite	20	0.75	15,432	496	-	-	-
Sulphide Saprolite	114	0.84	95,531	3,071	0.12	131	289
Hard Rock	1,050	0.69	723,315	23,255	0.10	1,088	2,399
Sub-Total M&I	**1,184**	0.70	834,278	26,823	0.10	1,219	2,687
			Inferred				
Oxide Saprolite	24	0.53	12,528	403	-	-	-
Sulphide Saprolite	65	0.48	30,942	995	0.07	45	98
Hard Rock	1,202	0.62	746,201	23,991	0.08	999	2,202
Total Inferred	**1,291**	0.61	789,671	25,389	0.08	1,044	2,300

TABLE 14-23 SUMMARY OF MINERAL RESOURCES BY ZONE – DECEMBER 31, 2017

GR Engineering (Barbados), Inc. – Siembra Minera Project

Zone Name	Tonnes (Mt)	Grade (g/t Au)	Grade (%Cu)	(kg)	Contained Gold (koz)	Contained Copper (kt)	(Mlb)
			Measured				
Brisas	9	0.93	0.17	8,187	263	15	32
Cristinas	1	1.87	0.29	1,717	55	3	6
Mesones	-	-	-		-	-	-
Morrocoy	-	-	-		-	-	-
Cordova	-	-	-		-	-	-
Total, Measured	10	1.02	0.18	9,904	318	17	38
			Indicated				
Brisas	594	0.58	0.09	343,943	11,058	563	1,241
Cristinas	450	0.88	0.10	396,477	12,747	468	1,030
Mesones	76	0.65	0.22	49,221	1,582	164	361
Morrocoy	1	0.86	-	933	30	-	-

Zone Name	Tonnes (Mt)	Grade (g/t Au)	Grade (%Cu)	Contained Gold (kg)	(koz)	Contained Copper (kt)	(Mlb)
Cordova	53	0.63	0.01	33,800	1,087	8	17
Total, Indicated	1,174	0.70	0.10	824,374	26,504	1,202	2,649
Measured & Indicated							
Brisas	603	0.58	0.10	352,130	11,321	578	1,273
Cristinas	451	0.88	0.10	398,194	12,802	470	1,036
Mesones	76	0.65	0.22	49,221	1,582	164	361
Morrocoy	1	0.86	-	933	30	-	-
Cordova	53	0.63	0.01	33,800	1,087	8	17
Sub-Total, M&I	1,184	0.70	0.10	834,278	26,823	1,219	2,687
Inferred							
Brisas	364	0.47	0.12	170,731	5,489	441	971
Cristinas	761	0.70	0.07	530,775	17,065	517	1,140
Mesones	51	0.35	0.17	18,006	579	85	186
Morrocoy	92	0.60	-	55,046	1,770	-	-
Cordova	23	0.67	0.01	15,114	486	1	3
Total, Inferred	1,291	0.61	0.08	789,671	25,389	1,044	2,300

Categories of Inferred, Indicated, and Measured Mineral Resources are recognized in order of increasing geological confidence. However, Mineral Resources are not equivalent to Mineral Reserves and do not have demonstrated economic viability. There can be no assurance that Mineral Resources in a lower category may be converted to a higher category, or that Mineral Resources may be converted to Mineral Reserves. Inferred Mineral Resources cannot be converted into Mineral Reserves as the ability to assess geological continuity is not sufficient to demonstrate economic viability. Due to the uncertainty which may attach to Inferred Mineral Resources, there is no assurance that Inferred Mineral Resources will be upgraded to Indicated or Measured Mineral Resources with sufficient geological continuity to constitute Proven and Probable Mineral Reserves as a result of continued exploration.

There is a degree of uncertainty to the estimation of Mineral Reserves and Mineral Resources and corresponding grades being mined or dedicated to future production. The estimating of mineralization is a subjective process and the accuracy of estimates is a function of the accuracy, quantity, and quality of available data, the accuracy of statistical computations, and the assumptions used and judgments made in interpreting engineering and geological information. There is significant uncertainty in any Mineral Resource/Mineral Reserve estimate, and the actual deposits encountered and the economic viability of mining a deposit may differ significantly from these estimates. Until Mineral Reserves or Mineral Resources are actually mined and processed, the quantity of Mineral Resources/Mineral Reserves and their respective grades must be considered as estimates only. In addition, the quantity of Mineral Reserves and Mineral Resources may vary depending on, among other things, metal prices. Any material change in quantity of Mineral Reserves, Mineral Resources, grade, or stripping ratio may affect the economic viability of a property. In addition, there can be no assurance that recoveries in small scale laboratory tests will be duplicated in larger scale tests under on-site conditions or during production. Fluctuation in metal or commodity prices, results of additional drilling, metallurgical testing, receipt of new information, and production and the evaluation of mine plans subsequent to the date of any estimate may require revision of such estimate. The volume and grade of reserves mined and processed and recovery rates may not be the same as currently anticipated. Estimates may have to be re-estimated based on changes in mineral prices or further exploration or development activity. This could materially and adversely affect estimates of the volume or grade of mineralization, estimated recovery rates, or other important factors that influence estimates. Any material reductions in estimates of Mineral Reserves and Mineral Resources, or the ability to extract these mineral reserves, could have a material adverse effect on the Company's financial condition, results of operations, and future cash flows.

RPA has considered the impact of any environmental, permitting, legal, title, taxation, socioeconomic, marketing, political, or other relevant factors that could materially affect the Mineral Resource estimate. RPA agrees with Gold Reserve's view that a number of risk items could materially affect the Mineral Resource estimate. These items include:

- The effects of local political, labour and economic developments, instability and unrest,
- Significant or abrupt changes in the applicable regulatory or legal climate,
- Currency instability, hyper-inflation and the environment surrounding the financial markets and exchange rate in Venezuela,
- International response to Venezuelan domestic and international policies,
- Limitations on mineral exports,
- Invalidation, confiscation, expropriation or rescission of governmental orders, permits, agreements, or property rights,
- Exchange controls and export or sale restrictions,
- Currency fluctuations, repatriation restrictions and operation in a highly inflationary economy,

- Competition with companies from countries that are not subject to Canadian and U.S. laws and regulations,
- Laws or policies of foreign countries and Canada affecting trade, investment and taxation,
- Civil unrest, military actions, and crime,
- Corruption, requests for improper payments, or other actions that may violate Canadian and U.S. foreign corrupt practices acts, uncertain legal enforcement and physical security, and
- New or changes in regulations related to mining, environmental and social issues.

SENSITIVITY ANALYSIS

RPA notes that the Mineral Resources are sensitive to cut-off grade and there is a uniform reduction in tonnes as the cut-off grade is increased (Tables 14-24 and 14-25).

TABLE 14-24 M&I SENSITIVITY TO AU CUT-OFF GRADE BY CONCESSION

GR Engineering (Barbados), Inc. – Siembra Minera Project

Brisas

Cut-off Grade (g/t Au)	Tonnes (Mt)	Au (g/t)	Au (koz)
0.0	609	0.57	11,240
0.1	607	0.58	11,228
0.2	572	0.60	11,043
0.3	471	0.68	10,213
0.4	361	0.78	8,983
0.5	273	0.88	7,711
0.6	207	0.99	6,553
0.7	157	1.09	5,531
0.8	120	1.20	4,631
0.9	92	1.31	3,860
1.0	70	1.42	3,184

Cristinas

Cut-off Grade (g/t Au)	Tonnes (Mt)	Au (g/t)	Au (koz)
0.0	460	0.89	13,126
0.1	460	0.89	13,126
0.2	454	0.90	13,083
0.3	427	0.94	12,862
0.4	379	1.01	12,327
0.5	326	1.10	11,558
0.6	276	1.20	10,678
0.7	234	1.30	9,790
0.8	199	1.40	8,957
0.9	170	1.50	8,156
1.0	144	1.59	7,370

TABLE 14-25 M&I SENSITIVITY TO AU CUT-OFF GRADE

GR Engineering (Barbados), Inc. – Siembra Minera Project

	Measured and Indicated		
Cut-off Grade (g/t Au)	Tonnes (Mt)	Au (g/t)	Au (koz)
0.0	1,211	0.70	27,367
0.1	1,204	0.71	27,360
0.2	1,155	0.73	27,096
0.3	1,005	0.80	25,869
0.4	825	0.90	23,844
0.5	666	1.01	21,554
0.6	537	1.12	19,275
0.7	435	1.23	17,150
0.8	354	1.34	15,207
0.9	290	1.45	13,458
1.0	237	1.56	11,851

15 MINERAL RESERVE ESTIMATE

There are no current Mineral Reserves estimated for the Project at the current time.

16 MINING METHODS

The Siembra Minera Project is an open-pit gold-copper mining project. The mine will be a conventional truck and shovel open pit mining operation, which will utilize hydraulic shovels and 236-tonne trucks as the primary mining equipment. Ancillary activities will include, however not be limited to, road maintenance, road dust control, site dewatering, dump and stockpile maintenance, and grade control.

RESOURCE OPEN PIT OPTIMIZATION

Open pit optimization was conducted on the Mineral Resources using US$1,300/oz Au and US$3.00/lb Cu for the resource pit. Whittle software version 4.5.5 was used for open pit optimization.

The optimization parameters used for the PEA are listed in Table 16-1. These parameters were used in the generation of the Whittle pit shell for resources and may differ from the final economic parameters used in the economic model.

The Resource Pit Optimization was developed by RPA based on RPA's 2017 Mineral Resource estimate. Blocks classified as Measured, Indicated, and Inferred Mineral Resources were included in the resource pit optimization process for the Siembra Minera deposit.

Figure 16-1 presents the resource pit geometry connecting the two deposits in one pit.

GEOTECHNICAL ASSESSMENTS

A geotechnical assessment was carried out on the Brisas Project in July 2007 by Sergio Brito Consultoria Ltda (SBC) in conjunction with Vector Colorado, LLC (Vector) completing a geotechnical slope stability study for the open pit.

The Cristinas pit slope analysis was part of the SNC-Lavalin (2005) study, including plant site foundations, TMF site foundation, open pit slopes, waste dump and stockpiles, open pit hydrogeology and dewatering, infrastructure foundations, haulage and service roads, diversion channel, water management ponds, landfill and airstrip.

Brisas inter-ramp pit slopes based on Vector (2007) range from 22° to 30° in saprolite, 36.2° to 46.6° in the east wall, and 46.6° to 52° in the west wall. Cristinas pit slopes based on SNC-Lavalin (2005) are 31° for saprolite, 45° for east wall, and 50° to 55° for hard rock on the west and south wall.

The pit slopes used for RPA's pit optimization were 36° for oxide saprolite, 46° for sulphide saprolite, and 48° for hard rock for the entire deposit.

TABLE 16-1 PEA OPEN PIT OPTIMIZATION PARAMETERS

GR Engineering (Barbados), Inc. – Siembra Minera Project

Pit Optimization Parameter	Units	Values
Vulcan Block Size	m	10x10x6
Whittle Block Size	m	10x10x12
Oxide	°	36
Sulphide	°	46
Hard Rock	°	48
Gold Price	US$/oz	1,300
Oxide (CIP)		
Payable Gold	%	100.0
Au Doré Transport (Selling Cost)	US$/oz	2.15
Au Doré Treatment (Selling Cost)	US$/oz	1.00
Au Doré Refining (Selling Cost)	US$/oz	5.00
Gold Gravity Recovery	%	98.0
Flotation		
Payable Gold / Payable Copper	%	97.5 / 96.5
Copper Concentrate Transport (Selling Cost)	US$/wmt conc.	60.0
Copper Concentrate Treatment (Selling Cost)	US$/t conc.	90.0
Copper Concentrate Refining (Selling Cost)	US$/oz Au	5.00
Copper Concentrate Refining (Selling Cost)	US$/lb Cu	0.09
Gold Recovery	%	83.0
Copper Recovery	%	87.0
Royalty	%	6.0
Costs		
Reference Mining Cost	US$/t mined	1.10
Mining Cost - Incremental	US$/t/12m bench	0.008
Process Oxide	US$/t processed	6.4
Process Sulphide and Hard Rock	US$/t processed	4.2
G&A Cost	US$/t processed	0.80

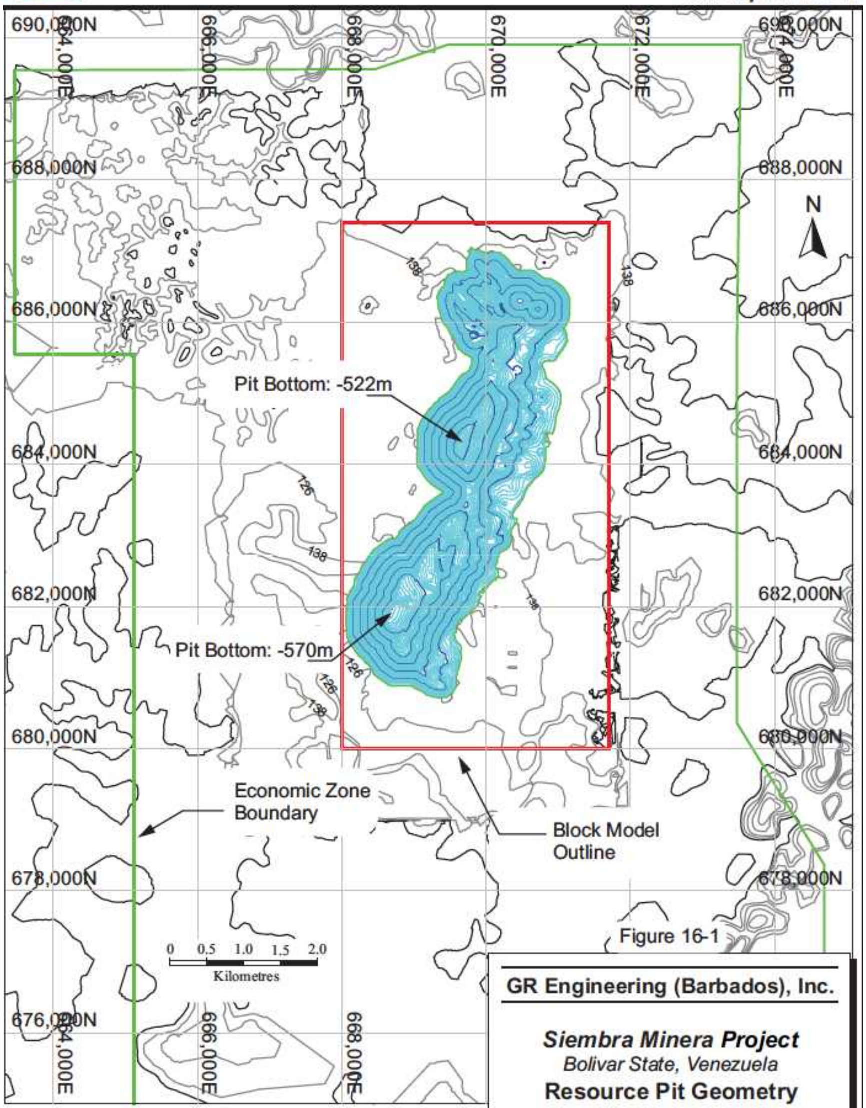


Figure 16-1

GR Engineering (Barbados), Inc.

Siembra Minera Project

Bolivar State, Venezuela

Resource Pit Geometry

March 2018

Source: RPA., 2018.

The resource pit is approximately 6,000 m long and 1,900 m wide with a maximum depth of approximately 700 m. The total material including waste is 5,910 million tonnes. The pit slope on the east wall follows the mineralization with slopes from 36° to 38°; the west wall final pit has overall pit slopes ranging from 48° to 50°.

PRODUCTION SCHEDULE

A mine design and production schedule was developed to support the cash flow at US$1,300 oz Au and US$3.0 lb Cu.

MINE PLAN PIT OPTIMIZATION

The pit optimization analyses for the mine plan were run on the Measured, Indicated, and Inferred Mineral Resources to determine the economics of extraction by open pit methods using US$1,300/oz Au and US$3.00/lb Cu prices for the Revenue Factor (RF) of 1.0. The parameters used in the pit optimization runs, using Whittle software, are presented above in Table 16-1. This optimization includes Measured, Indicated, and Inferred Mineral Resources in the Whittle analysis, and the potential mine plan.

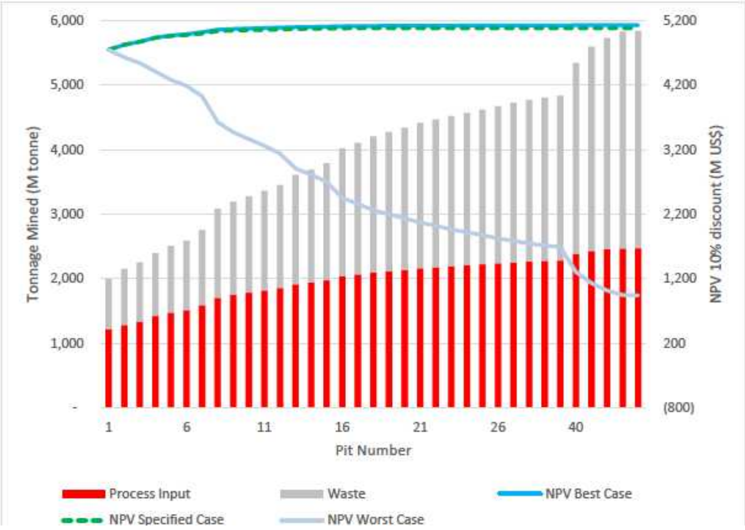
The Net Present Values (NPV) were analyzed in Whittle using a discount rate of 10%. Whittle produces a best, specified, and worst case scenario for mining. The best case assumes that mining can be carried out in thin pushbacks allowing earlier access to the mineralized material while the worst case assumes mining the entire bench from the top down, where more waste is mined in the early years, negatively impacting the NPV. The specified case combines groups of pit shells to work more closely to pushbacks, identifying steps on pit waste increments analyzing the NPV.

Pit 19 was selected as a guideline to design the smooth pit at an RF of 0.48 on gold and copper price. Analyzing the specified case, the NPV does not increase significantly after pit 19 as presented in Table 16-2.

TABLE 16-2 MINE PLAN OPEN PIT OPTIMIZATION
GR Engineering (Barbados), Inc. – Siembra Minera Project

Pit #	RF	Best NPV (10%)	Cash Flow Specified NPV (10%)	Worst NPV (10%)	Total Mined (Mt)	Waste Mined (Mt)	Process Input (Mt)	Au Grade (g/t)	Cu Grade (%)
1	0.30	4,751	4,751	4,751	1,997	781	1,216	0.822	0.089
2	0.31	4,829	4,829	4,635	2,151	868	1,283	0.814	0.089
3	0.32	4,876	4,874	4,545	2,252	918	1,334	0.805	0.089
4	0.33	4,941	4,936	4,415	2,396	977	1,419	0.785	0.095
5	0.34	4,973	4,963	4,283	2,512	1,039	1,473	0.777	0.095
6	0.35	4,992	4,979	4,193	2,587	1,076	1,511	0.771	0.095
7	0.36	5,025	5,002	4,030	2,757	1,171	1,586	0.760	0.096
8	0.37	5,063	5,040	3,631	3,087	1,386	1,702	0.747	0.096
9	0.38	5,075	5,049	3,472	3,195	1,446	1,750	0.741	0.096
10	0.39	5,082	5,055	3,367	3,279	1,494	1,785	0.736	0.096
11	0.40	5,089	5,061	3,262	3,363	1,544	1,819	0.732	0.096
12	0.41	5,095	5,065	3,143	3,451	1,594	1,857	0.726	0.096
13	0.42	5,103	5,072	2,909	3,609	1,695	1,914	0.721	0.095
14	0.43	5,106	5,076	2,813	3,694	1,752	1,942	0.717	0.095
15	0.44	5,110	5,079	2,702	3,793	1,817	1,976	0.713	0.095
16	0.45	5,116	5,083	2,455	4,022	1,986	2,036	0.709	0.094
17	0.46	5,119	5,084	2,362	4,108	2,044	2,064	0.706	0.094
18	0.47	5,121	5,085	2,261	4,210	2,112	2,098	0.702	0.094
19	0.48	5,122	5,086	2,201	4,277	2,158	2,119	0.700	0.094
20	0.49	5,123	5,086	2,139	4,344	2,206	2,138	0.697	0.094
30	0.59	5,129	5,085	1,694	4,837	2,554	2,283	0.680	0.093
40	0.69	5,132	5,086	1,306	5,346	2,958	2,389	0.667	0.093
50	0.79	5,132	5,086	1,132	5,599	3,166	2,433	0.662	0.093
60	0.89	5,132	5,086	1,017	5,733	3,276	2,457	0.659	0.093
70	0.99	5,132	5,086	947	5,836	3,365	2,471	0.657	0.092
71	1.00	5,132	5,086	943	5,842	3,370	2,471	0.657	0.092

FIGURE 16-2 PIT BY PIT GRAPH WITH NPV



Pit shells after pit 19 will continue increasing the process input along with the total waste as presented in Figure 16-2, however, pit 19 (RF 0.48) was selected in order to maximize the NPV as shown in Table 16-2, processing material that adds economic value to the Project under current assumptions.

PHASE DESIGN

Phases are commonly designed by targeting the highest grade first, limiting the stripping ratio and following minimum mining width constraints. Maximizing the metal content in early years provides the best sequence for the discounted cash flow. The phase sequence was derived using Whittle pit shells as a guideline, starting from lower revenue factors, maintaining minimum mining width, and ensuring continuity in the bench progression.

The projected processing plant construction schedule will require that saprolite material be mined in early years to feed the oxide plant. Three phases following the oxide material were designed in order to provide this material to the oxide plant while the flotation plant is under construction. Table 16-3 summarizes the process material, gold and copper grades, and stripping ratio.

TABLE 16-3

MINE PHASES SUMMARY

GR Engineering (Barbados), Inc. – Siembra Minera Project

Phase	Process Input (kt)	Au (g/t)	Cu (%)	Waste (kt)	Total (kt)	Ratio W:O
1	38,696	0.721	0.060	40,318	79,014	1.04
2	28,016	0.686	0.063	33,661	61,677	1.20
3	15,124	0.565	0.030	10,753	25,877	0.71
4	103,101	1.153	0.085	11,089	114,190	0.11
5	275,616	0.895	0.081	93,199	368,815	0.34
6	209,691	0.886	0.070	269,424	479,115	1.28
7	156,667	0.630	0.095	86,527	243,194	0.55
8	295,418	0.584	0.116	395,484	690,902	1.34
9	155,139	0.592	0.103	372,702	527,841	2.40
10	389,032	0.630	0.099	351,324	740,356	0.90
11	49,165	0.458	0.118	298,028	347,193	6.06
12	123,264	0.745	0.066	196,656	319,920	1.60
13	165,811	0.501	0.106	161,186	326,997	0.97
Total	**2,004,741**	**0.705**	**0.092**	**2,320,350**	**4,325,091**	**1.16**

Phase design was developed using Vulcan's Automated Pit Designer tool to smooth and connect isolated blocks. The phase design does not include access ramps, however, the final pit design accounts for access ramps as presented in Figure 16-3.

The final pit design includes 35 m ramps width at a 10% grade. The west wall was designed to include 20 m step out berms every 120 m in elevation with inter-ramp slopes of 54°, resulting in overall slopes of 50°. Overall slopes on the east wall were defined by the mineralization and access ramps resulting in an angle of approximately 34°.

Figure 16-3 also shows the proposed waste dump location. The waste dumps were designed to verify if the economic boundary was able to accommodate the waste required by the mine production schedule.

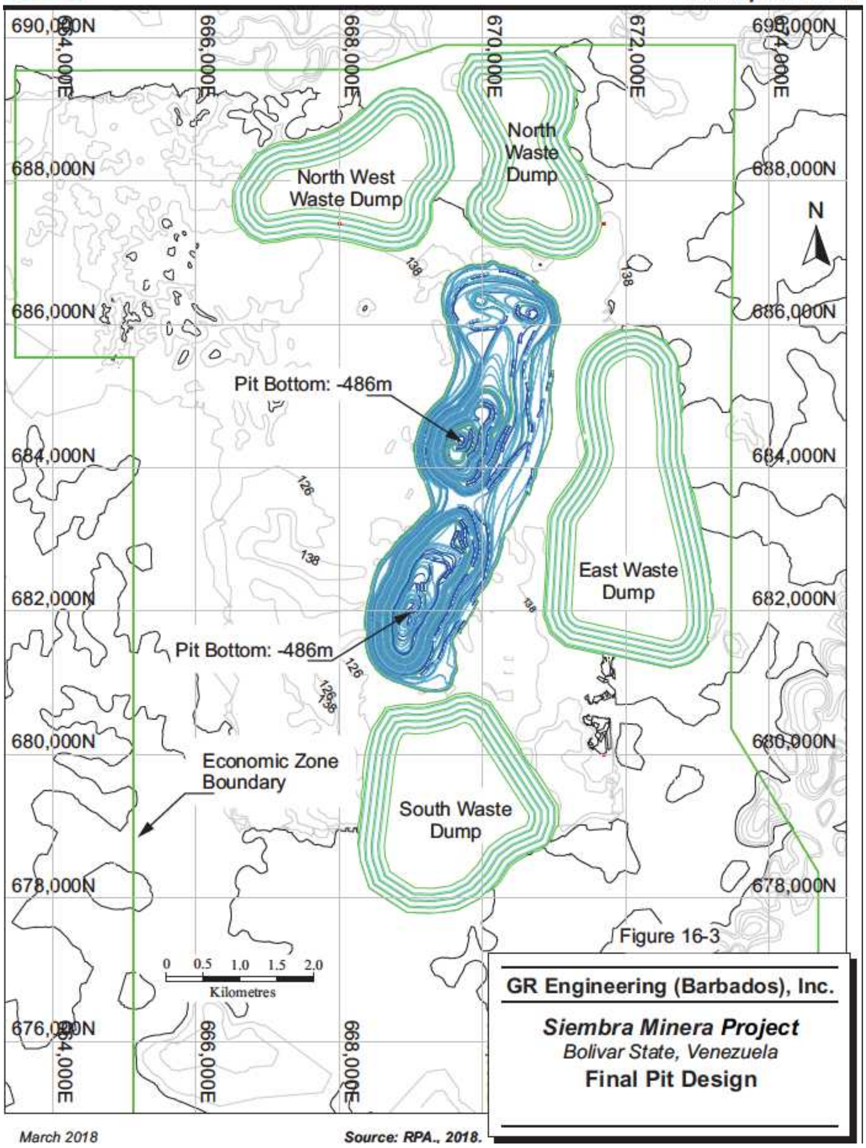


Figure 16-3

GR Engineering (Barbados), Inc.

Siembra Minera Project

Bolivar State, Venezuela

Final Pit Design

March 2018

Source: RPA., 2018.

Table 16-4 presents the waste dump capacity based on 1.8 t/m^3 loose density. Waste dump designs are based on 20 m berms for every 50 m lift and face angles of 35°. Dumps are located within the economic zone boundary and at a distance of more than 100 m from the final pit design.

TABLE 16-4

WASTE DUMP CAPACITY

GR Engineering (Barbados), Inc. – Siembra Minera Project

Dump	Volume (Mm^3)	Capacity (Mt)
North-West	668	1,202
North	536	965
East	1,240	2,233
South	1,038	1,868
Total	**3,482**	**6,268**

PRODUCTION SCHEDULE

Mine production was scheduled to be carried out at a maximum mining rate ranging from 330 ktpd to 380 ktpd of total material. Stripping ratios are expected to average 1.16 over the Life of Mine (LoM) plan. The production schedule was produced using Whittle software to guide the mining sequence; Vulcan to design phases, waste dumps and the final pit; and XPAC to schedule the phases following the processing requirements.

During the first ten years of the Project, 5.8 Mtpa of oxide saprolite that does not require grinding will be processed in the oxide saprolite plant. The flotation plant starts two years after the oxide plant. Feed to the flotation mill is scheduled to be 58.0 Mtpa tonnes for years 3 to 10, while softer high copper sulphide saprolite material is available. In year 11, one quarter of the flotation mill (12.25 Mtpa) is converted to oxide to accommodate the harder grinding low copper hard rock materials. The other 36.75 Mtpa of capacity in the mill will be used for the harder grinding higher copper material in the flotation. The oxide plant will start processing with a combination of saprolite and low copper hard rock using the leach tanks from the oxide saprolite plant and additional leach tanks required for processing. The hard rock and sulphide saprolite was divided into high copper and low copper using a 0.02% Cu threshold.

In order to supply the processing input required in the first 10 years of production, the total material mined must achieve up to 120 Mtpa from a combination of the mining phases. The mining rate will change depending on stockpile size, increasing total mining rate to 140 Mtpa in year 20.

The periods after year 25 were scheduled using 5-year periods as presented in the mine production schedule summarized in Table 16-5.

Table 16-6 presents the processing plant production schedule including the oxide processing material and the flotation processing material. The processing schedule also has 5-year periods at the end of the Project life. The processing rate starts at 5.8 Mtpa for the oxide saprolite plant only, increasing to 63.8 Mtpa from year 4 to 10 when flotation and oxide plants are working, and decreasing to 49 Mtpa after year 11 when the processing circuit modification is required to accommodate the low copper hard rock material that bypasses the flotation plant and goes directly to leaching.

TABLE 16-5

MINE PRODUCTION SCHEDULE

GR Engineering (Barbados), Inc. – Siembra Minera Project

Period	Process Material (kt)	Grade Au (g/t)	Grade Cu (%)	Contained Au (Moz)	Contained Cu (Mlb)	Waste (kt)	Total (kt)	Ratio W:O
-1						25,000	25,000	
1	15,718	0.824	0.059	0.42	20	8,282	24,000	0.53
2	30,223	1.048	0.100	1.02	67	9,777	40,000	0.32
3	62,406	0.844	0.067	1.69	92	45,607	108,014	0.73
4	75,083	0.867	0.069	2.09	114	44,917	120,000	0.60
5	73,267	0.873	0.086	2.06	139	46,733	120,000	0.64
6	50,218	0.835	0.092	1.35	102	69,782	120,000	1.39
7	49,950	0.693	0.091	1.11	101	70,050	120,000	1.40
8	45,994	0.701	0.074	1.04	75	74,006	120,000	1.61
9	73,610	0.659	0.076	1.56	123	46,390	120,000	0.63
10	68,275	0.614	0.083	1.35	125	51,725	120,000	0.76
11	62,233	0.774	0.074	1.55	102	47,767	110,000	0.77
12	58,966	0.938	0.087	1.78	114	31,034	90,000	0.53
13	52,540	1.155	0.090	1.95	104	37,460	90,000	0.71
14	29,456	0.722	0.083	0.68	54	60,544	90,000	2.06
15	39,655	0.804	0.079	1.03	69	70,345	110,000	1.77
16	54,261	0.875	0.082	1.53	98	65,739	120,000	1.21
17	45,395	0.909	0.084	1.33	84	74,605	120,000	1.64
18	26,792	0.519	0.089	0.45	52	93,208	120,000	3.48
19	37,715	0.542	0.108	0.66	90	82,285	120,000	2.18
20	48,739	0.572	0.096	0.90	103	91,261	140,000	1.87

Period	Process Material (kt)	Grade Au (g/t)	Grade Cu (%)	Contained Au (Moz)	Contained Cu (Mlb)	Waste (kt)	Total (kt)	Ratio W:O
21	55,939	0.581	0.107	1.04	132	84,061	140,000	1.50
22	52,627	0.614	0.111	1.04	129	87,373	140,000	1.66
23	60,390	0.583	0.110	1.13	147	79,610	140,000	1.32
24	55,088	0.694	0.135	1.23	164	44,912	100,000	0.82
25	41,849	0.593	0.087	0.80	80	58,151	100,000	1.39
26-30	224,778	0.645	0.100	4.66	498	175,222	400,000	0.78
31-35	190,929	0.623	0.105	3.82	443	209,071	400,000	1.10
36-40	168,035	0.681	0.081	3.68	300	301,965	470,000	1.80
41	154,610	0.500	0.107	2.49	365	133,467	288,077	0.86
TOTAL	**2,004,741**	**0.705**	**0.092**	**45.42**	**4,085**	**2,320,350**	**4,325,091**	**1.16**

TABLE 16-6 PROCESS PRODUCTION SCHEDULE

GR Engineering (Barbados), Inc. – Siembra Minera Project

Period	Oxide Process Material (kt	Grade Au (g/t)	Grade Cu (%)	Flotation Process Material (kt)	Grade Au (g/t)	Grade Cu (%)	Total Process Material (kt)	Grade Au (g/t)	Grade Cu (%)
-1	-			-		-	-		
1	5,162	0.628	0.034	-	0.000	0.000	5,162	0.63	0.03
2	5,800	0.894	0.075	-	0.000	0.000	5,800	0.89	0.08
3	5,800	0.671	0.069	40,890	1.113	0.092	46,690	1.06	0.09
4	5,800	0.643	0.066	58,000	1.052	0.082	63,800	1.01	0.08
5	5,800	0.609	0.020	58,000	0.937	0.102	63,800	0.91	0.09
6	5,800	0.622	0.024	58,000	0.829	0.096	63,800	0.81	0.09
7	5,800	0.565	0.042	58,000	0.757	0.093	63,800	0.74	0.09
8	5,800	0.507	0.015	58,000	0.683	0.078	63,800	0.67	0.07
9	5,800	0.630	0.002	58,000	0.692	0.092	63,800	0.69	0.08
10	5,800	0.573	0.011	58,000	0.618	0.094	63,800	0.61	0.09
11	12,250	0.504	0.016	36,750	0.826	0.088	49,000	0.75	0.07
12	12,250	0.530	0.016	36,750	0.934	0.094	49,000	0.83	0.07
13	12,250	0.549	0.015	36,750	1.050	0.097	49,000	0.92	0.08
14	12,250	0.550	0.015	36,750	1.145	0.098	49,000	1.00	0.08
15	12,250	0.492	0.015	36,750	0.911	0.096	49,000	0.81	0.08
16	12,250	0.531	0.015	36,750	0.920	0.099	49,000	0.82	0.08
17	12,250	0.585	0.015	36,750	0.977	0.100	49,000	0.88	0.08
18	12,250	0.495	0.016	36,750	0.870	0.104	49,000	0.78	0.08
19	12,250	0.622	0.015	36,750	0.507	0.131	49,000	0.54	0.10
20	12,250	0.593	0.016	36,750	0.561	0.128	49,000	0.57	0.10
21	12,250	0.499	0.022	36,750	0.598	0.128	49,000	0.57	0.10
22	11,954	0.518	0.012	36,750	0.633	0.131	48,704	0.60	0.10
23	9,915	0.560	0.008	36,750	0.609	0.132	46,665	0.60	0.11
24	2,355	0.630	0.017	36,750	0.647	0.136	39,105	0.65	0.13

Period	Oxide Process Material (kt	Grade Au (g/t)	Grade Cu (%)	Flotation Process Material (kt)	Grade Au (g/t)	Grade Cu (%)	Total Process Material (kt)	Grade Au (g/t)	Grade Cu (%)
25	9,887	0.577	0.008	36,750	0.690	0.139	46,637	0.67	0.11
26-30	39,290	0.600	0.002	183,750	0.643	0.120	223,040	0.64	0.10
31-35	21,278	0.620	0.005	183,750	0.631	0.119	205,028	0.63	0.11
36-40	11,931	0.676	0.019	183,750	0.666	0.090	195,681	0.67	0.09
41-	3,475	0.698	0.016	153,155	0.496	0.109	156,630	0.50	0.11
TOTAL	**302,195**	**0.581**	**0.017**	**1,702,545**	**0.727**	**0.106**	**2,004,741**	**0.705**	**0.092**

Plans are for the mine to operate two 12-hour shifts per day, 7 days per week for a total of 14 shifts per week. The mine operation schedule allows for 26 shifts per year being lost due to weather delays in the mine. It is envisioned that mining would occur during both shifts to minimize stockpiling and rehandling. Scheduled work time is 10.5 hours per shift, allowing 30 minutes for meals, 30 minutes of delays, and 30 minutes lost during shift change.

MINE EQUIPMENT

Mine equipment requirements were developed from the annual mine production schedule, based on the mine operation schedule, equipment availability, and equipment productivities. The mine equipment fleet will include 30 m^3 hydraulic shovels, 18 m^3 wheel loaders, 236-tonne class haul trucks, and 251 mm diameter track-mounted rotary drills.

Equipment productivities were determined for drills, shovels, and loaders. Haul truck productivity was dependent on annual cycle times. Production hours were calculated for the trucks, loaders, and support equipment. Annual operating requirements, such as fleet size, fleet utilization, and labour requirements, were then output from the production hours. Annual operating requirements for auxiliary equipment were based on haul truck hours for graders and water trucks and the operating shifts and loader hours for dozer support. A summary of the total fleet requirements for the major mine equipment is presented in given in Table 16-7.

A separate equipment fleet of smaller excavators and articulated dump trucks is include in the mining capital for saprolite mining in the first 10 years. Typically, undisturbed saprolite material can be difficult to mine as the moisture creates operation problems. As the Project area has essentially been disturbed (see Figure 14-16), RPA has assumed most saprolite is handled by the larger equipment fleet. Further review of saprolite mining is recommended.

TABLE 16-7 MAJOR MINE EQUIPMENT REQUIREMENTS

GR Engineering (Barbados), Inc. – Siembra Minera Project

QUANTITIES	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13	Y14	Y15	Y16	Y17	Y18	Y19	Y20	Y21	Y22	Y23	Y24	Y25-Y29	Y30-Y34	Y35-Yr39	Y40-Y45
Major Mining Equipment																													
Blast Hole Drill Cat MD6250	2	2	3	5	6	6	6	6	5	5	5	5	4	4	4	4	4	4	4	5	6	6	6	5	5	4	3	3	4
Hydraulic Shovel Cat 6050	3	3	3	7	7	7	7	7	8	8	8	8	8	6	6	7	7	7	7	8	9	9	9	9	8	8	5	5	5
Front End Loader Cat 994K	2	2	2	6	6	6	6	6	4	4	4	3	3	3	4	4	4	4	5	4	3	4	4	4	4	4	3	2	3
Haulage Truck Cat 793F	12	12	15	43	45	50	57	57	59	59	59	59	59	59	59	64	67	70	75	72	81	82	81	81	81	79	79	74	56
Support Mining Equipment																													
Pit In-Fill Drill Cat MD6250	0	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Dozer Cat D11T	2	4	4	4	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	4	4	4
Dozer Cat D10T	2	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Wheel Dozer Cat 834K	1	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Motor Grader Cat 18M	1	1	1	4	4	4	5	5	5	5	5	5	5	5	5	5	6	6	6	6	7	7	7	7	7	7	7	6	5
Motor Grader Cat 16M	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Water Truck Cat 777G	1	1	1	4	4	4	5	5	5	5	5	5	5	5	5	5	6	6	6	6	7	7	7	7	7	7	7	6	6

UNIT OPERATIONS AND PRODUCTIVITY

DRILLING AND BLASTING

The blast hole drills consist of a fleet of crawler-mounted diesel-powered units in the 334kN (75,000 lb) pulldown class (e.g., Sandvik D75KS, Caterpillar MD6310, or Atlas Copco Pit Viper 271) equipped with high pressure compressors (2,400 Mpa). Each unit has the ability to drill a single pass of 13 m. Drilling will be achieved with a 251 mm bit.

A 7.5 m by 7.5 m drill pattern will be used for all areas. All areas will be sampled, but only harder saprolite and hard rock areas will be blasted. Drill penetration rate is 34m per hour. Production rates vary with density and are estimated to be 2,700 tonnes per hour for oxide saprolites, 3,270 tonnes per hour for sulphide saprolites, and 4,650 tonnes per hour for hard rock. Other assumptions used to develop drill requirements are:

- All in-pit material is drilled with the blast hole drills.
- All drill hole bench heights are 12 m high with a 1 m subdrill. Sampling may be provided in 6 m intervals if required.
- An average powder factor of 0.16 kg of explosive per tonne of rock was used.
- For water conditions it was assumed that 33% of the holes are wet and will be loaded with a higher density slurry product. The remaining holes will be dry and loaded with an Ammonium Nitrate Fuel Oil (ANFO) product.

The drill productivity and production tonnage was used to calculate the number of hours required in a given time period. Drill utilization was not allowed to exceed 80% to reflect lost time during a shift for blast moves.

LOADING

The mine loading fleet consists of 30 m^3 capacity hydraulic shovels and 18 m^3 wheel loaders. The loading fleet requirements diminish over the mine life based on lower overall tonnage in the later years.

Hydraulic Shovel (30 m^3)

Diesel-powered hydraulic shovels equipped with standard 30 m^3 rock buckets, such as the Hitachi EX5600 and the Caterpillar 6050, will be used to load material into rear dump haul trucks. The shovels will be particularly important for digging areas of boulders in the lower saprolite unit, sorting small blocks of mineralized material, initiating drop-cuts, and digging in areas with bad ground conditions.

Four passes are used to load the 236-tonne trucks. Hourly production is 2,700 tonnes per hour in the oxide saprolite, 3,000 tonnes per hour in the sulphide saprolite, and 3,400 tonnes per hour in the hard rock.

<u>Wheel Loader (18 m^3)</u>

Rubber-tired front-end loaders equipped with a standard 18 m^3 rock bucket are used to load the fleet of 236-tonne capacity haul trucks. Twenty percent of the total material was allocated to the wheel loaders. Caterpillar 994's and Komatsu WA-1200's are examples of this type of machine. Hourly production is 1,400 tonnes per hour in the oxide saprolite, 1,600 tonnes per hour in the sulphide saprolite, and 1,800 tonnes per hour in the hard rock. Six passes are used to load the 236-tonne trucks. Wheel loaders provide flexibility and mobility for mining and will provide pit wall clean when required.

HAULING

A single haulage fleet consisting of mechanical drive rear-dump haul trucks in the 236-tonne payload class was selected to minimize mining costs while still providing selectivity. Caterpillar 793F's and Komatsu 830E's are examples of this type of machine. The trucks match up with the 30 m^3 class hydraulic shovels with a nominal four passes per truck and with the18 m^3 class wheel loaders with a nominal six passes per truck. The rated payload capacity used was 150.6 dry tonnes for oxide saprolite, 177.0 dry tonnes for sulphide saprolites, and 233.4 dry tonnes for hard rock.

Haulage requirements were calculated based on an average annual truck cycle time. Cycle times were calculated based the annual production requirement. Haulage profiles were calculated for each bench based on the reserves for the bench and the destination of the material. Separate profiles were measured for each material type and destination. Truck cycle times provided the input to determine the number of truck hours required.

The fixed time for trucks going to the crusher or waste dump area is broken out into the following components:

- Spot and load 4.0 minutes
- Turn and dump 1.5 minutes
- Miscellaneous delays 2.0 minutes
- Total 7.5 minutes

Miscellaneous delays are any delays less than five minutes in length, including shovels cleaning the face, rest stops, bunching delays, and blasting delays.

Haul road widths are presently designed at 35 m. A left-hand traffic pattern is proposed for safety and operational considerations.

MAINTENANCE SHOP

A mechanical maintenance shop will be constructed. This shop is envisaged to handle all maintenance requirements, and will include a welding area, tire bay, wash bay, lubricants area, tool storage, and training area. Major equipment rebuilds would be sent off-site.

POWDER MAGAZINE

Two storage areas are designed with one for explosives and the second for accessories. Both facilities will meet all local and federal requirements. The powder magazine will be operated by the mine owner.

STOCKPILES

Stockpiles are required for blending the process feed to achieve sufficient copper grades in flotation to produce a copper concentrate above 20%. Stockpiles fluctuate year to year, but achieve maximum capacity of just over 70 million tonnes (see Figure 18-1).

17 RECOVERY METHODS

INTRODUCTION

The conceptual plant design includes a 15,000 tpd oxide cyanidation plant that is designed to recover gold from oxide saprolite and sulphide saprolite that contains low concentrations of copper and a flotation concentrator that is designed to process 140,000 tpd of hard rock material. The oxide leach plant recovers the gold as doré from gravity concentrate and leaching of the gravity tailings. The flotation plant recovers gold as doré from a gravity concentrate and leaching of the cleaner scavenger tailings. Gold and copper are recovered in a copper concentrate.

OXIDE CYANIDATION PLANT

A conceptual 15,000 tpd cyanidation plant design was completed by Samuel Engineering to support this PEA. A simplified process flow diagram is provided in Figure 17-1.

Saprolite will be excavated and loaded onto trucks. The material will be transported to a stockpile located adjacent to a saprolite crushing plant that is located adjacent to the mine.

A front-end loader will remove material from the stockpile and feed it into the crusher feed bin. The feed bin will be equipped with a static grizzly to remove any large debris or rocks that may cause problems in the double roll crusher. Oversize material will be rejected and placed in a stockpile. The oversize material will be periodically removed from the stockpile and placed in a waste dump if the gold grade is low or moved to a stockpile if it is rock that can be economically processed in the future.

Mineralized material from the feed bin is fed to the crusher using an apron feeder. The discharge from the crusher is then transferred by conveyor to a vibrating screen where any debris that can plug a pump is removed. The screen undersize discharges into a mix tank where it is mixed with water to prepare a slurry that can be pumped through a high density polyethylene (HDPE) pipeline to a surge tank that is located in the grinding circuit at the plant site which is approximately five kilometres to six kilometres away.

SAPROLITE
CRUSHER
ORE STOCKPILES
BRISAS CRISTINAS SURFACE MINE (PIT)
PIPE LINE
SLURRY
TRANSFER TANK
CYCLONES
PRE-LEACH
THICKENER
PROCESS WATER
GRINDING
RECYCLE TANK
GRAVITY
COLD
CIRCUIT
STRIP (Cu)
CARBON STRIP
REFINERY
MILL SURGE TANK
ELECTROWWINNING
SLUDGE FILTER
COPPER
RETORT FURNACE
PRECIPITATE

SMELTING VAULT
FURNACE
FRESH CARBON
LOADED CARBON
TAILINGS
THICKENER
AGITATED LEACH
C.I.P.
CN WATER
RECYCLE TANK
CYANIDE DESTRUCTION
RECLAIM WATER
TAILINGS STORAGE FACILITY

Figure 17-1

GR Engineering (Barbados), Inc.

Siembra Minera Project

Bolivar State, Venezuela

Simplified Process Flowsheet for the Oxide Cyanidation Plant

March 2018

Source: Gold Reserves Inc., 2017.

At the oxide plant, the slurry is pumped to a ball mill for a nominal reduction in size. The ball mill discharge is pumped to a cyclone cluster where material with a particle size P_{80} of 50 ¼m reports to the overflow of the cyclone and discharges into a pre-leach thickener. Coarser material reports to the cyclone underflow and is returned by gravity to the ball mill feed chute for further size reduction.

A fraction of the cyclone underflow will be fed to the gravity concentration circuit. Batch centrifugal concentrators were selected for the conceptual design. They can handle coarser sized material but they also require cleaning between batches. The concentrators will alternate operation to simulate a continuous processing circuit.

Tailings from the gravity concentrators are returned to the grinding circuit. The concentrate from the centrifugal concentrators is fed to a storage tank located in the gold room. The concentrate is transferred to a feed tank and the primary gravity concentration table to further separate the gold from the gangue material and increase the grade of the gravity concentrate.

Concentrate from the primary gravity table will be collected and stored before being fed to the secondary gravity table for final cleaning and upgrading. The secondary gravity table concentrate is collected and stored in a decant tank where free water is removed. Tailings from the gravity tables are returned to the grinding circuit.

A pre-leach thickener is provided to separate the solids in the slurry from the liquid to produce an optimum slurry density for the leach circuit. Overflow from the pre-leach thickener is collected and pumped to the process water tank. Underflow from the pre-leach thickener is pumped to the cyanide leach circuit.

The leach circuit consists of six tanks operated in series to provide a retention time of 18 hours. Lime slurry is added to the tanks to maintain the proper pH (i.e., 10.0 to 11.0). Slurry that discharges from the leach circuit will be fed to a carbon-in-pulp (CIP) circuit consisting of six tanks operated in series to provide a retention time of eight hours. Activated carbon is advanced in the CIP circuit in a counter-current direction to the slurry flow to recover dissolved gold from the slurry by adsorption. Slurry handling and carbon advancement will be accomplished using Kemix pump cell technology. Loaded carbon is removed from the circuit and transferred to the elution circuit.

Loaded carbon will be treated using the Anglo-American Research Laboratory (AARL) method recover the gold. Samuel Engineering has included an industry standard cold strip method to remove copper from the activated carbon, however, RPA is of the opinion that optimization work completed by Crystallex indicates that copper adsorption can be mitigated by management of cyanide in the leach circuit including maintaining proper cyanide concentrations and staged additions of cyanide.

Loaded carbon will be transferred from the CIP circuit to the acid wash vessel where it will be soaked in three percent hydrochloric acid solution to condition the carbon for subsequent metal elution. The acid-treated carbon will be rinsed with fresh water and transferred to the elution column where it can undergo a cold strip with caustic-cyanide solution to remove copper, if needed. The cold strip solution that contains copper will be bled to the cyanide detoxification circuit. The carbon is then soaked in a circulating hot solution of two percent sodium hydroxide and two to three percent sodium cyanide solution to remove gold and silver from the carbon. After elution, the carbon will be rinsed with hot fresh water to recover the metals.

To achieve the hot solution temperatures, the caustic cyanide solution will be pumped through two heat exchangers that are heated by a diesel fired boiler. The concentrated pregnant solution will be cooled in the pregnant solution tank.

Pregnant solution will be pumped from the pregnant solution tank to a single electrowinning cell equipped with stainless steel cathodes. Gold is removed from solution and forms a sludge on the cathodes and in the bottom of the electrowinning cell. After the electrowinning cycle, the cathodes will be washed to remove the metal bearing sludge. The sludge will be pumped from the electrowinning cells to a receiving tank. From the tank, the sludge will be filtered and dried. Dewatered sludge and gravity concentrates are transferred to pans that will be placed in a mercury retort to remove mercury and dry the materials prior to smelting. Any mercury will be recovered in a flask and will be disposed of per Venezuelan environmental regulation.

The dried concentrate is mixed with flux and transferred to an induction furnace to separate the precious metal from the slag and produce doré. The doré will be weighed and stored in a vault for shipment off-site for further processing.

Stripped carbon will be removed from the elution vessel and transferred to a dewatering screen and fed to a carbon regeneration kiln to be thermally regenerated to remove organic contaminants. Undersize from the dewatering screen will flow by gravity to the carbon fines clarifier.

Particulates from the carbon regeneration kiln will be collected in a wet scrubber that uses process water. Carbon will discharge from the kiln to the carbon quench tank where the carbon will be cooled and the thermal processes will be stopped. The quenched activated carbon will be pumped to the carbon sizing screen where undersize from the screen will flow by gravity to the carbon fines clarifier. Oversize carbon from the sizing screen will flow by gravity to the regenerated carbon tank. Fresh carbon will also be added as needed to maintain an adequate concentration of activated carbon in the CIP circuit.

A single tailings thickener is included in the circuit to increase the solids density of the slurry and to recover solution that will be recycled to the leach plant in order to reduce the cyanide consumption. The thickened underflow is processed in a sulphur dioxide (SO_2) - air cyanide detoxification circuit before being discharged (by gravity) to the TMF via an 11 km HDPE pipeline.

FLOTATION CONCENTRATOR

Aker Kvaerner completed a plant design to support a feasibility study in 2005. Subsequently, SNC-Lavalin completed some minor modifications to the plant design during detailed design during 2006 and 2007. The changes included:

- Changed the leach circuit from CIP to CIL
- Increased the size of the SAG mills from 10.97 m diameter to 11.58 m diameter
- Added intensive cyanide leach to process the gravity recovered gold

A simplified process flowsheet is provided in Figure 17-2.

Hard rock will be crushed in two gyratory crushers (1,473 mm by 1,905 mm) that operate in parallel at locations near the open pit mine. Discharge from the crushers falls into hoppers. Variable speed apron feeders transfer the crushed material from the hoppers to primary crusher discharge conveyors which, in turn, transfer to an overland stockpile feed conveyor.

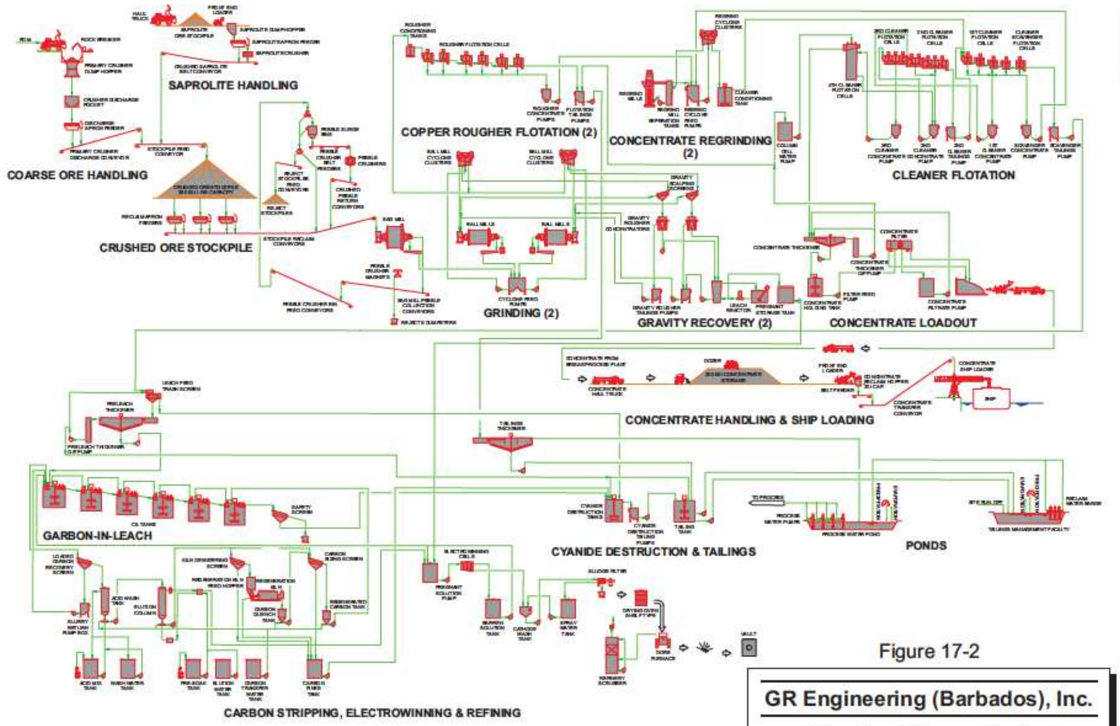


Figure 17-2

GR Engineering (Barbados), Inc.

Siembra Minera Project

Bolivar State, Venezuela

Simplified Process Flowsheet for the Flotation Concentrator

March 2018

Source: pincock, allen & holt, 2008.

Oxide saprolite and sulphide saprolite that contains low concentrations of copper is processed in the oxide cyanidation plant.

The stockpile feed conveyor transports crushed hard rock and sulphide saprolite that contains greater than 0.07% Cu to the crushed material stockpile. The stockpile is an elongated stockpile that is fed by a tripper conveyor. Apron feeders reclaim crushed material from the stockpile and transfer it to the semi-autogenous grinding (SAG) mill feed conveyors.

As a base case, the numbers of the pieces of equipment used in the SNC-Lavalin basic engineering design for the flotation concentrator was doubled in most cases. The exception is that an optimized plant layout for the stockpiles and feeders was developed by Samuel Engineering.

Four parallel grinding lines are included in the plant design. Each line consists of one SAG mill and two ball mills. Crushed material is conveyed to the SAG mill feed hoppers by the stockpile reclaim conveyors. Water is also added to the feed hoppers to create a slurry density of approximately 70% solids by weight. Slurry discharges from the SAG mills through trommel screens. Oversize from the trommel screens is directed to the pebble collection conveyor. Undersize from the trommel screens discharges to the cyclone feed sumps. Each sump receives the discharge from one SAG mill and two ball mills. The slurry is pumped from the sump to hydrocyclones. Overflow from the cyclones is the product from the grinding circuit. The grinding circuit is designed to produce a particle size that is P_{80} 100 µm. Underflow from the cyclones discharges to the ball mill feed chutes. A portion of the cyclone underflow is fed to the gravity gold recovery circuit.

Each of the four grinding lines includes a gravity gold recovery circuit. The gravity gold recovery circuits include two gravity scalping screens and two centrifugal concentrators. The gravity concentrators are batch concentrators that are shut down every four hours to flush the gravity concentrate from the concentrators. Tailings from the gravity gold recovery circuit are returned to the ball mill feed chutes. Concentrate from the centrifugal concentrators is processed in an intensive cyanide leach reactor.

Overflow from the cyclones flows by gravity into the rougher flotation conditioning tanks. Four rougher flotation lines, each of which contain two banks of flotation cells that provide 960 m^3 of capacity to provide 20 minutes of retention time. Rougher flotation is performed at a neutral pH. Tailings from the rougher flotation circuit are collected in a pump box and pumped to the tailings thickener feed tank.

Rougher flotation concentrate will be pumped to the regrind cyclone pump boxes. The slurry is pumped to the regrind cyclones for classification. Overflow from the cyclones is the product from the regrind circuit with a particle size of P_{80} 37 µm. Underflow from the cyclones flows to the vertical regrind mills for grinding. Four parallel regrind circuits are provided in the design. Four stages of cleaner flotation are provided to improve the grade of the flotation concentrate. The circuits are operated at a pH between 11.5 and 12.0 to reject pyrite from the sulphide flotation concentrate.

The first cleaner flotation circuit includes two circuits that operate in parallel to provide 390 m^3 of capacity to provide 10 minutes of retention time. The first cleaner concentrate progresses through the second and third cleaner flotation circuits. Tailings from the first cleaner flotation circuit flow by gravity to the cleaner scavenger circuit which consists of two parallel circuits that provide 260 m^3 of capacity. Concentrate from the cleaner scavenger circuit is returned to the regrind cyclone pump box where it combines with the rougher flotation concentrate. Tailings from the cleaner scavenger circuit are pumped to the cyanide leach circuit.

Second stage cleaner flotation circuit consists of two parallel circuits that provide approximately 102 m^3 of capacity. The third stage of cleaner flotation consists of two circuits that operate in parallel to provide approximately 51 m^3 of capacity. The fourth stage of cleaner flotation is conducted in four pairs of column flotation cells that operate in parallel. Tailings from each cleaner flotation stage are returned to the feed of the previous cleaner flotation stage.

The final flotation concentrate from the two parallel flotation circuits is pumped to two 9 m diameter concentrate thickeners that also operate in parallel. Overflow from the thickeners is pumped to the process water pond. Underflow from the thickeners will be transferred to concentrate holding tanks. From the holding tanks the concentrate will be filtered in automated horizontal plate filter presses to produce a target moisture concentration of 8% solids by weight. The final concentrate will be stored in a stockpile and trucked to a port facility for overseas transport.

The first cleaner scavenger tailings will be leached in a CIL circuit. A trash screen is provided to remove trash from the slurry. Underflow from the screen will flow into one of two 30 m diameter pre-leach thickeners that operate in parallel. Two parallel CIL circuits that each have six agitated, 13.6 m diameter by 14.0 m high tanks are provided for leaching. Slurry will flow by gravity sequentially from tank one to tank two to tank three and to tank four. Lime slurry will be added to increase the pH to between 10.0 and 10.5. Sodium cyanide will also be added to the beginning of the circuit. New or re-activated carbon will be added to the sixth tanks of the parallel CIL circuits. The carbon will be advance counter-currently to the slurry flow. That is from tank six to tank five to tank three and so on. Loaded carbon will be removed from tank one and sent to adsorption desorption recovery (ADR) circuit to produce gold doré.

Slurry discharging from the CIL circuit will flow to the cyanide destruction circuit where the SO_2 – air process will be used to reduce the WAD cyanide concentration to less than 0.6 mg/L.

Discharge from the cyanide destruction circuit will be pumped to the TMF.

The current design assumes that two parallel circuits are provided in the ADR. Loaded carbon that is removed from tank one of each of the CIL circuits will flow across the loaded carbon screen to the loaded carbon surge bin. Carbon will discharge from the surge bin to the acid wash tank where it will be washed with hydrochloric acid to remove inorganic contaminants. After acid washing, the carbon will be washed with fresh water and neutralized with dilute sodium hydroxide solution. After neutralization, the loaded carbon will be transferred to the elution columns which have a capacity of 6,500 kg of carbon. The batch AARL elution process is utilized in the ADR circuit. The carbon will be pre-soaked in hot solution containing cyanide and sodium hydroxide. Following the pre-soak cycle, carbon elution will begin. The elution cycle will continue until four bed volumes of solution is collected in the pregnant solution storage tank. Pregnant solution will be pumped from the pregnant solution tank to two electrowinning cells with stainless steel mesh cathodes that operate in series. Gold is removed from the pregnant solution as sludge on the cathodes and in the bottom of the electrowinning cells. Sludge will be removed from the cells and washed from the cathodes on a daily basis. It will be pumped to the sludge filter for dewatering. After filtration, the gold-bearing sludge will be dried in a drying oven. After drying, it will be mixed with fluxes and smelted in a furnace to produce doré that will be shipped off site for further processing. The refinery includes a slag handling circuit.

After elution, the carbon will be reactivated in a regeneration kiln that is provided to remove organic contaminants. Hot carbon will discharge from the kiln into a quench tank. From the quench tank, the carbon will be transferred to carbon attrition tanks where the carbon fines will be removed. The carbon will advance from the carbon attrition tanks to carbon sizing screens for final removal of carbon fines before being returned to the four tanks of the CIL circuits.

Rougher flotation tailings will be dewatered in the tailings thickeners to produce an underflow slurry density of 55% solids by weight. From the thickener underflow the tailings will be pumped to the TMF. Overflow from the tailings thickeners will report to the process water pond.

The plant design includes all reagent handling facilities, utilities, and auxiliary facilities required to operate the facility.

At the times of the feasibility study and the detailed design, a number of trade-off studies were completed to select the optimum circuit configurations and designs, however, the conditions and assumptions made over twenty years ago were much different than current conditions so RPA recommends that some of these studies be re-evaluated using current metal prices, equipment sizes, and costs.

PLANT TRANSITIONS AND RECONFIGURATION

The production schedule is based on initially processing oxide saprolite through a 15,000 tpd cyanide leach plant. The crushing and screening plant feed is approximately 10% higher assuming that some of the material will be rejected due to oversize and/or rock material. Starting in year 7, the majority of the oxide saprolite is depleted and sulphide saprolite that contains low concentrations of copper will also be fed to the plant. In years 9 and 10, only low copper sulphide saprolite will be fed to the plant.

In year 4, the flotation concentrator will be commissioned. The feed to the plant includes sulphide saprolite that contains higher concentration of copper and a combination of high and low copper hard rock material at a nominal rate of 140,000 tpd although the actual feed rate is somewhat higher due to the presence of sulphide saprolite.

In year 11, the quantity of hard rock with suitable copper grades to produce acceptable concentrates in the flotation plant diminishes so the plant will be re-configured to process less material through the flotation plant and additional material through the oxide leach plant. The conceptual plan is to reduce the feed to the flotation concentrator to approximately 105,000 tpd and increase the tonnage to the oxide leach plant to 35,000 tpd. The low copper hard rock material will be ground in the existing milling circuit in the flotation plant and the leach plant will be expanded to accommodate the higher tonnage of material. The ball mill in the oxide leach plant, which is only sized to process saprolite, can be decommissioned or used to grind saprolite that is pumped from the open pit mine to the oxide leach plant.

18 PROJECT INFRASTRUCTURE

The information in this section is taken from the Aker Kvaerner FS, the Samuel Engineering Oxide Saprolite Order of Magnitude Capital Cost Basis of Estimate, and from the PEA TMF design that was completed by Tierra Group International, Ltd.

There are two distinct areas of activity, as shown on Figure 18-1, the general site plan. The first is the Mine and the Crushing Plants and the second is the area surrounding the Processing Facilities and the TMF. Other facilities and infrastructure will be located in both areas and at an intermediate location where the camp will be located.

HIGHWAY ACCESS ROADS

There is a main paved road, Highway 10 that connects Puerto Ordaz to El Dorado. Highway 10 passes through the small town of Las Claritas near the Siembra Minera Property. There is an existing access road, approximately 2.5 km long, from Highway 10 to the Project site, which requires improvements. A second access road is planned to connect from Highway 10 north of Las Claritas substation and around the substation to the south to intersect with the main access road. The road will follow the existing road east to the mine area, continue over the conveyor and haul road, around the waste dump area, and to the Cuyuni River to provide local access to the river.

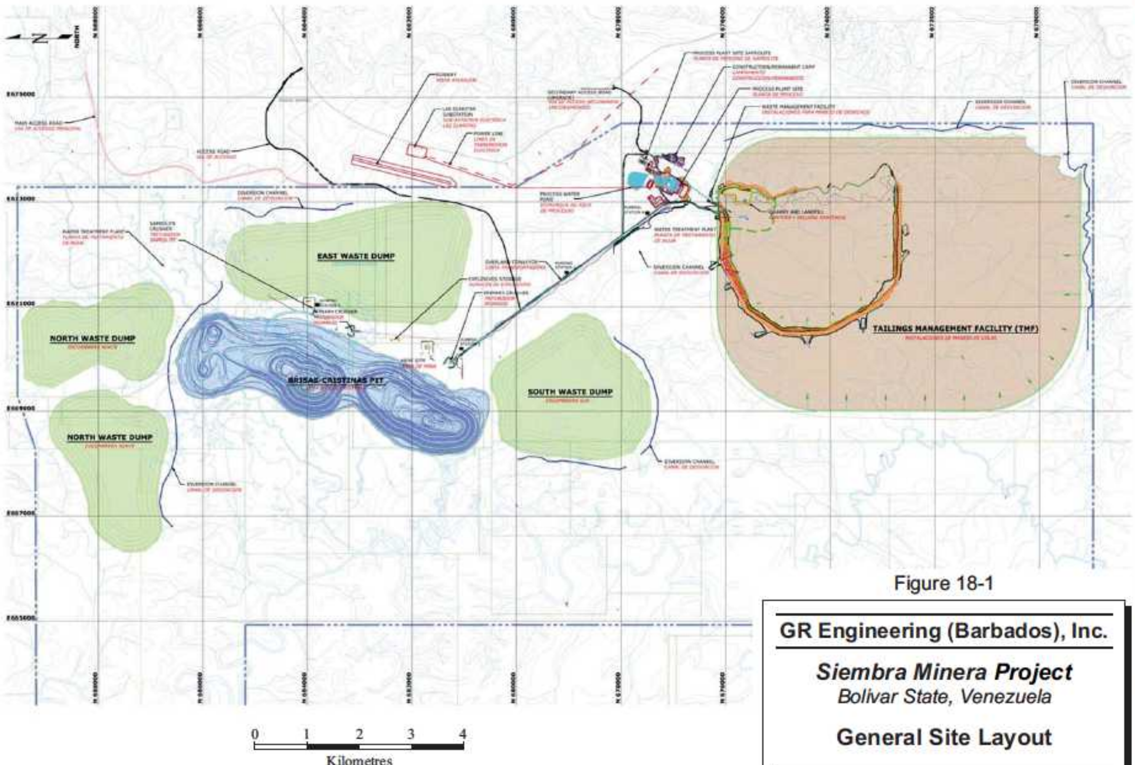


Figure 18-1

GR Engineering (Barbados), Inc.

Siembra Minera Project
Bolivar State, Venezuela

General Site Layout

March 2018

Source: Samuel Engineering, 2017.

OXIDE PLANT

The conceptual design completed by Samuel Engineering and the PEA-level capital cost estimate includes:

- Process Building
- Administration Building including Furnishings
- Laboratory and Equipment
- Warehouse/Maintenance Shop
- Truck Shop
- Camp
- Medical/Security/Fire Truck Building
- Mine Break Room
- On-site Diesel Power Generators
- Electrical Distribution Systems

In addition to the costs estimated by Samuel Engineering, RPA allocated a portion of the infrastructure costs associated with the larger plant to the early years of the Project development.

FLOTATION PLANT

Infrastructure costs estimated by SNC-Lavalin include:

- Administration Building
- Plant Maintenance and Warehouse Building
- Laboratory
- Mine Truck Shop and Warehouse
- Reagent Storage Building
- Fueling Stations
- Explosive Storage
- Solid Waste Disposal Facilities
- Camp
- Port Facilities
- Electric Power Supply and Distribution
- Water Supplies
- Compressed Air Supplies

- Sewage Treatment Facilities
- Safety and Fire Protection Systems
- Guard Houses

TAILINGS

Tierra Group plans two TMFs for the Project designated as Stage 1 and Stage 2. Stage 1 has a capacity of 138 Mt as designed by SNC-Lavalin during basic engineering that uses the Centerline raise methodology. The larger Stage 2 TMF footprint is approximately 7 km by 5 km that will inundate the Stage 1 design. Table 18-1 shows the anticipated capacity for the two stages. The second stage has three years more capacity that the potential mine plan.

TABLE 18-1 TAILINGS MANAGEMENT FACILITY CAPACITY

GR Engineering (Barbados), Inc. – Siembra Minera Project

Parameter	**Value**
Stage 1 Crest Elevation	159 m
Stage 1 Dam Capacity	135 Mt (minimum)
Stage 1 Dam Life	3 to 4 years
Stage 1 and 2 Dam Capacity	2,100 Mt
Stage 2 Crest Elevation	197 m
Stage 1 and 2 Dam Life	48 years

The conceptual design and cost estimate includes diversion dams and ditches and other items needed for water management.

19 MARKET STUDIES AND CONTRACTS

MARKETS

The principal commodities at the Project are freely traded, at prices that are widely known, so that prospects for sale of any production are virtually assured. For the Base Case, RPA used a gold price of $1,300 per ounce; a silver price of $17.00 per ounce; and a copper price of $3.00 per pound. These price points are in line with standard metal pricing metrics as shown in Table 19-1.

TABLE 19-1 METAL PRICE COMPARATIVE ANALYSIS

GR Engineering (Barbados), Inc. – Brisas-Las Cristinas Gold Project

Commodity	Units	This Study	Dec 31 2017 Spot	3 Yr Trailing Avg
Gold	US$/oz	$ 1,300	$ 1,291	$ 1,222
Silver	US$/oz	$ 17.00	$ 16.87	$ 16.62
Copper	US$/lb	$ 3.00	$ 3.25	$ 2.49

Per Table 19-2, the Project is expected to sell an annual average of 330,000 troy ounces gold and 95,000 troy ounces silver over the LoM in the form of doré per year. In addition, based on a total of 6.4 million tonnes of concentrate with average grades of 115 g/t Au, 65 g/t Ag, and 23.6% Cu, the Project is expected to sell an annual average of 149,000 dry tonnes per year of concentrate during the LoM containing 540,000 troy ounces of gold, 291,000 troy ounces of silver, and 33.7 million tonnes of copper.

TABLE 19-2 METAL SALES

GR Engineering (Barbados), Inc. – Brisas-Las Cristinas Gold Project

Process Facility	Commodity	Units	Yrs 1 & 2	Yrs 3 to 18	Yrs 19 to 45	LoM Avg	LoM Total (koz)
Leach Plant - Doré							
Direct Feed	Gold	koz/y	132.8	144.0	91.2	111.8	5,026.0
From Concentrator[1]	Gold	koz/y	-	334.5	150.0	218.6	9,392.4
Subtotal	Gold	koz/y	132.8	478.5	241.2	330.4	14,418.4
Direct Feed	Silver	koz/y	26.4	43.5	26.2	32.4	1,428.2
From Concentrator[1]	Silver	koz/y	-	73.0	56.1	62.4	2,682.2
Subtotal	Silver	koz/y	26.4	116.5	82.3	94.8	4,110.4
Concentrator							
	Concentrate	kt/y (dry)	-	151.7	147.9	149.2	6,419.8
	Gold	koz/y	-	735.3	424.3	540.0	23,220.7
	Silver	koz/y	-	340.1	261.6	290.8	12,504.3
	Copper	kt/y	-	33.5	33.9	33.7	1,450.4
Grand Total							
	Gold	koz/y	132.8	1,213.3	665.2	836.0	37,639.1
	Silver	koz/y	26.4	455.7	343.4	369.2	16,614.7
	Copper	kt/y	-	33.5	33.9	33.7	1,450.4

Note: [1]Gold and silver recovered by gravity and from cyanidation of concentrator tailings are added into the final doré product.

During years 3 to 18, which coincides with mining in areas with the highest gold grades, the Siembra Minera Project is also expected to sell an annual average of 152,000 dry tonnes per year of copper concentrate containing 735,000 ounces of gold and 33.5 million tonnes (74 million pounds) of copper. A further 479,000 ounces of gold per year will be produced during the first fifteen years in the form of doré for a grand total of 1.2 million ounces of gold a year during that 15-year period.

CONTRACTS

RPA is not aware of any forward sales or hedging contracts for the Project's metal production as of the date of this report. Cost assumptions are discussed in Section 22.

DORÉ HANDLING AND TRANSPORT

Doré will be shipped to the United States, Canada or Europe for refining by one of the internationally-established refiners.

CONCENTRATE HANDLING AND TRANSPORT

Concentrate will be trucked from site to Puerto Ordaz. A 40,000 metric tonne concentrate storage and ship loading facility will be constructed in Puerto Ordaz. The concentrate will be loaded for ocean shipment to a smelter, most likely in Europe.

For this Technical Report, charges for road freight and concentrate storage are a nominal $28/t and ocean freight has been assumed at $75/t. Concentrate treatment charges of $95/t and refining charges of $0.095/lb for copper, $3.00/oz for gold and $0.20/oz for silver have been used based on current smelter contracts for similar projects.

DORÉ AND CONCENTRATE MARKETING

Siembra Minera will be authorized to export and sell its doré and concentrate containing gold, copper, silver, and other strategic minerals outside of Venezuela and maintain proceeds from such sales in an offshore US dollar account.

SITE OPERATIONS

All cost estimates for operating costs are based on factoring and budgetary supplier quotes. RPA is not aware of contractual arrangements with any suppliers at this time. RPA notes that operating and capital cost estimates are impacted favourably by Venezuela's diesel costs of US$0.02/L and power costs of US$0.038/kWh which are well below industry norms, particularly diesel fuel pricing.

RPA is not aware of any operational contracts for the Project as of the date of this report. Cost assumptions are discussed in Section 21.

20 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

This section summarizes the current Project's environmental setting and the corporate, regulatory, and international framework within which the Project is being developed. GRI has prepared, or is in the process of preparing, environmental reports and programs to meet municipal, provincial, and national regulatory requirements, as well as generally-accepted international standards.

The description of the Project's current status on permitting, environmental, and social considerations is based on:

- the ongoing Venezuelan and International Environmental and Social Impact Assessments (ESIAs);
- the ongoing Preliminary Economic Assessment (PEA);
- field environmental and social data collected during previous comprehensive studies at the Project site, as well as, in 2016 and 2017; and,
- inspections of the site and environs, observations and reports by AATA International, Inc. and others conducted during July 2017.

ENVIRONMENTAL STUDIES

PHYSICAL ENVIRONMENT

The Project area is located at the foot of the Sierra de Lema high plateau; and the topography is moderately homogenous, dominated by plains with some rolling hills. Elevations range from 127 MASL to 218 MASL, with higher elevations near the east and southeast margins of the Project area. The Project site layout is presented in Figure 18-1.

The climate is tropical with January through March being drier months and June through July being wetter months. Humidity is high (monthly average from 80% to 87%) and annual precipitation is over 3,000 mm. Daily temperatures range from 21ºC to 38ºC. Prevailing winds are from the west – southwest, with a speed mostly from 0.5 m/sec to 2.1 m/sec. Particulate matter (PM_{10}) in the air based upon sampling in 2005 to 2006, is in the range of 10 µg/m^3 to 25 µg/m^3, with no significant seasonal changes.

Detailed studies on geology, mineralization, and geotechnical characteristics of the Project area have been conducted and the results are presented in previous sections of this report.

The soils of the Project area are dominated by udic ultisols (udults), with clay or cambic horizons and well-drained inceptisols (dystrudepts). The parent oxide saprolite material extends to a typical depth of 10 m to 30 m. Top soils (0 cm to 30 cm depth) of the Project area are characterized by sandy loams with variable organic matter content (from very low at 0.6% to very high at 7.2%).

Sediments in the Project area are predominantly made up of sands (>90%) and significantly coarser than the soils (sands ≤70% mostly). As a result, organic matter content is relatively low compared with other soils.

Due to the high rainfall in the region, a relatively large number of streams and smaller tributaries have formed in the gently sloping watershed that drains towards the west into the Cuyuni River, which functions as the mainstream river in the region. The Cuyuni River flows south to north through the western portion of the Project area, with flows normally at 20 m^3/sec to 35 m^3/sec, and a maximum rate of 64.3 m^3/sec measured during the 2004 to 2005 baseline surveys. Tributaries of the Cuyuni River in the Project area include (from south to north): the Uey River (normal flows in the range of 10 m^3/sec to 20 m^3/sec), the Aymara Creek (0.5 m^3/sec to 3 m^3/sec), and the Amarilla Creek (5 m^3/sec to 15 m^3/sec).

Considering groundwater resources, four hydrologic units have been identified: saprolite, transitional rock, fractured rock and fresh hard rock. While the saprolite unit (average thickness 55 m) contributes water to the underlying aquifer through leakage, horizontal water movement is extremely limited due to high clay content. The transition rock unit and the fractured rock unit (with average thickness of 15 m and 125 m respectively) are both characterized by the highest hydraulic conductivity values and thus are considered the best aquifers in the Project area. The groundwater table is typically very high in the Project area (0 m to 3 m below ground surface in most places) and most of the pits from previous artisanal mining activities by others are filled with water. Significant efforts will be required to dewater the mine pit for the Siembra Minera Project.

CHEMICAL ENVIRONMENT

Project regional air quality is representative of global background concentrations in a reasonably undeveloped rainforest setting. There are no large industrial facilities in the immediate area that contribute large-scale emissions to the Project air-shed. The local service industry is mainly geared towards the needs of small-scale gold mining and agricultural activities and is, therefore, mostly free of large air contaminant discharges to the air-shed. Therefore, background levels of sulphur dioxide (SO_2), nitrogen oxides (NO_x), carbon monoxide (CO), ozone (O_3), and hydrocarbons are estimated to be insignificant. Local vehicular traffic, burning of trash and deforestation refuse, and cooking contribute typical air pollutants. There are known releases of gaseous mercury to the local environment due to the refining of amalgamated gold ore in make-shift devices and heated furnaces by small-scale miners.

Soils in undisturbed areas are acidic, with pH values in the range of 3.5 to 6.0. Based upon previous site-specific soils studies, the Sodium Adsorption Ratio (SAR) values were less than 0.03, indicating that the soils are not sodic. Nutrients were mostly low for agricultural production purposes. Metals are mostly in normal ranges, though contents of copper and a few other trace metals were high in some locations since this is a mineralized area.

Sediments from locations in the Cuyuni and Uey Rivers upstream of Aymara Creek had low concentrations of metals, while samples from Aymara Creek, Amarilla Creek and the Cuyuni River downstream from Amarilla Creek showed relatively high baseline concentrations of mercury, lead, arsenic, aluminum, chromium, and copper. These results are indicative of existing impacts from small-scale mining activities in the Project area.

Surface water quality in non-impacted waters (i.e., upstream of disturbed areas) are generally good, with pH values in the range of 6.6 to 6.9, total suspended solids (TSS) less than 15 mg/L, and trace metal contents mostly in normal ranges. Samples from drainages within or downstream of the disturbed areas (specifically the Amarilla Creek watershed, Cuyuni River downstream of Amarilla Creek, and certain portions of the Aymara Creek watershed) showed lower pH values, higher TSS, and higher concentrations of trace metals (noticeably, Al, Cu, Cr, Pb, Zn, and Hg), which clearly indicate impacts from small scale mining activities by others in the Project area, Historical monitoring data from the Project area indicated good groundwater quality in the Project area, with pH values ranging from 6.5 to 7.9, mostly low in dissolved metals (and Hg was below or at detection limit of 0.0002 mg/L), and low in nutrients. Surveys of current groundwater quality, especially within the disturbed areas, are being planned and results will be presented in the Project ESIA report.

BIOLOGICAL ENVIRONMENT

FLORA

A large part of the total surface area of the Project area is covered by tropical evergreen forests (>80%) with distinct ranges of height and density, depending primarily upon geomorphology, soil characteristics, and the degree of anthropogenic disturbance. This forest has a dense canopy and a height of approximately 30 m to 40 m. There are two predominant vegetation types:

1. An ombrophilous, macrothermic forest or tropical, evergreen forest associated with a peneplain landscape. The forests developed on this landscape exhibit a medium- density physiognomy associated with high-density forest types. This association develops in areas of low relief, meadows, colluvial slopes, and small groups of hills.
2. An ombrophilous, sub-mesothermic forest, premontane or submontane, evergreen forest associated with the piedmont landscape. These forests are generally less than 25 m in height and with medium to dense crown cover. Epiphytes, mosses, lichens, and vines are abundant in these forests.

In general, woody species are of relatively low commercial value with tall stems and small diameters. A significant amount of deforestation has occurred throughout the Project area, which totals more than 100 km^2. Deforestation continues in many areas inside and outside of the Project boundary.

FAUNA

Despite diverse anthropogenic activity and disturbances over the past few decades in the Km 88 region, including historic mineral exploration, localized small-scale and artisanal mining, localized deforestation, traffic, and several settlements, the Project area continues to support a diverse faunal population.

The number of registered species (collected or observed) to date in the southern sector of the Upper Cuyuni River Basin (where the Project is located) includes: 118 mammals, 338 birds, 61 reptiles, and 40 amphibians. Collections and surveys in the Project area have reported 95 species of mammals, 177 species of birds, 27 species of reptiles, and 12 species of amphibians.

The Cuyuni River supports a diverse fish fauna with at least 187 taxa reported.

Two species that were considered endemic to the region were identified in the Project area: the Escalera Tree Frog (*Hyla sibleszi*) and a small characid fish, the Redtailed Bryconops (*Bryconops colaroja*). The Escalera tree frog in the Project area is included in the *Least Concern* category by the International Union for the Conservation of Nature (IUCN) (i.e., widespread and not threatened). This frog is stable in its native range and is not considered to be threatened by the Siembra Minera Project. *Bryconops colaroja* was found in Las Claritas Creek, Aymara Creek, and is likely to be found in many areas of the upper Cuyuni River. It appears to be fairly abundant based on previous surveys. Impacts to *Bryconops colaroja* are possible due to its occurrence in Aymara Creek. Measures may be taken to identify nearby refugia and protect portions of habitat within the Project area for this small tropical fish.

THREATENED, ENDANGERED AND SENSITIVE SPECIES

No species of plants from the IUCN *Red List* of species were found in the Project area. Seven mammals and one reptile with *Special International Conservation Status* have been identified in the Project area.

Two mammals, whose range includes the Project area, are listed as *Endangered* in the IUCN Red List (1994, 2001). These are the giant Armadillo (*Priodontes maximus*) and the giant river otter (*Pteronura braziliensis*). Based on the lack of presence in the Project area and the overall size of the range of these two endangered species, the Project will have no significant negative impacts on their conservation status.

Five species of mammals are listed by IUCN as *Vulnerable*, whose ranges include the Project area, of which only the tapir (*Tapirus terrestris*) is known from the Project area, and its status is common in Venezuela. There is a small potential for negative impacts to tapirs in the Project area due to opening of areas where they may be hunted.

One reptile is listed as *Vulnerable* by IUCN whose range includes the Project area: the Brazilian tortoise (*Geochelone denticulata*). This tortoise is very common in Venezuela.

There were no threatened or endangered species of fish known to occur or reported from waters within the Project site or area. A detailed Rapid Assessment Program was conducted by Conservation International and results published in 2008 (Evaluación Rápida de la Biodiversidad de los Ecosistemas Acuáticos de la Cuenca Alta del Río Cuyuní, Guayana Venezolana). The findings complemented the ecological baseline information generated by GRI. Planning for ecological education, ecotours, and scientific cooperation had been considered previously as part of stakeholder engagement and public outreach.

SOCIAL CONDITIONS

The Km 88 Mining District, where the Project is located, has seen several decades of planned and unplanned natural resource extraction activities (particularly mining and logging); in-migration and settlements; government-led regional development activities (including transmission lines and road construction to Brazil, housing programs); with associated environmental and social impacts.

Geopolitically, the Project area belongs to the San Isidro Parish, with major population centers just east of the Project boundary. Total population of the San Isidro Parish has been estimated to be 16,000 to 20,000 persons, more than half of whom live in five population centers immediately east of the Project area: Las Claritas, Santo Domingo, Ciudad Dorada, St. Lucia Inaway and Km 88 (TECMIN, 2017).

Approximately 80% of the local population in the San Isidro Parish is described as *criollo*, which are ethnically mixed communities originating from Venezuela and neighboring countries, and are primarily associated with the informal artisanal mining sector. There are currently several thousand indigent miners, over 100 small processing facilities (mills), and more than 300 gold-trading offices that are actively operating at the Project site and in the surrounding area (TECMIN, 2017).

Many of the *criollo* communities were created as a result of periodic in- and out-migration driven by “gold-rush” events. Thus, the social baseline conditions of the area are dominated by artisanal small mining related issues. These include unplanned urbanization; community health concerns (access to safe water, endemic malaria, and high levels of gastrointestinal and sexually transmitted diseases); limited or ineffective social infrastructure and services; and land-use conflicts. Other issues of concerns are associated with limited or overstretched educational facilities and high levels of local unemployment. Lack of educational opportunity, access to information, and general public services are key social factors.

Indigenous communities represent approximately 20% of the local population. The vast majority of indigenous communities in the Project region are settled within larger, planned village structures ranging in size from several hundred to over 1,000 inhabitants. The indigenous communities follow their traditional political and decision-making structures, and mainly consist of Pemón, and to a lesser extent Kariña and Arawak-speaking communities. Many members of these communities are multilingual (indigenous language(s), Spanish and – those with Guyanese descent – also English). The indigenous communities enjoy special privileges under the Venezuelan constitution, and those in the Project vicinity are not isolated or disconnected from the overall, larger, socioeconomic structure of the area. Compared with their *criollo* counterparts, indigenous communities are more permanent and enjoy a notably better quality of life.

As of 2013, there were five pre-school plus primary schools, two high schools (one with pre-school + primary) and one technical school in the San Isidro Parish, with a total enrollment of near 1,700. The technical school in San Miguel de Betania, ETA Integral Pemón Samarayi, enrolled 75 students as of 2013 (TECMIN, 2017). There are seven clinics in the San Isidro Parish. Although trained medical professionals are working at the clinics, shortages of adequate equipment and other medical supplies are apparent. Nearly 80% of the patients who visited the clinics were diagnosed with malaria, an endemic characteristic of the Project area (TECMIN, 2017).

The needs of and opportunities for *criollo* and indigenous communities are all being considered as part of the ESIA, Public Consultation and Disclosure Plan (PCDP), the Community Development Plan (CDP) and the Resettlement Action Plan (RAP) for the Project, which are all currently in preparation. A conceptual management plan for small-scale mining has been prepared, and a summary of this plan is presented in the Environmental and Social section of this chapter.

No archaeological sites or other cultural resources were identified in the Project area. Should any archaeological and cultural resources be identified during construction and operation of the Project, prompt measures will be taken to protect the resource and minimize impact, if any. A Cultural Heritage Plan will be prepared for the I-ESIA. The Piedra de Virgen (Rock of the

Virgin) is a popular tourist attraction located at Km 98 on the main Highway 10 south of Las Claritas, but is not within the Project boundary.

PERMITTING: REGULATORY APPROVAL PROCESS

An ESIA is required by the Venezuelan Constitution for the Project. Procedures for the preparation and approval of the ESIA, as well as additional permits that must be obtained, are defined by Decree 1257 of March 13, 1996 (*Normas Sobre Evaluación Ambiental de Actividades Susceptibles de Degradar el Ambiente*) which establishes the regulations for developing ESIAs.

The Ministry of People's Power for Ecosocialism and Water (Ministerio del Poder Popular para el Ecosocialismo y Aguas - known as MINEA) is responsible for the approval and monitoring of ESIAs for mining projects. It is important to note that the ESIA provides the umbrella permission for virtually all other environmental impacts created as a result of mining activities, including air emissions, effluent discharges, and the storage, control, and management of hazardous wastes.

In addition to an ESIA, an Authorization to Occupy the Territory (AOT - *Autorización de Ocupación del Territorio*) and an Authorization to Affect Natural Resources (AANR -

Autorización de Afectación de Recursos Naturales) must also be obtained. Both permits (AOT and AANR) are issued by MINEA.

The AOT certifies that the proposed use of the land by the Project is compatible with the land use provisions designated for the area. To complete the ESIA for the Project, a Term of Reference (TDR) that defines the scope and contents of the ESIA must be submitted to MINEA. Upon the approval of the TDR, the proponent will prepare and submit the ESIA to MINEA. Once the ESIA is approved, and the performance bond is paid, MINEA will then issue an AANR for the Project. Finally, an environmental supervision (monitoring) plan must be submitted and approved by MINEA before starting any onsite exploitation activities. The implementation of this plan will be supervised by MINEA.

There are a number of international standards and guidelines that will also be considered by GRI in the design, construction, operation, and closure of the Project. These international guidelines and standards will be reviewed by the combined company's technical and administrative staff, to determine applicability and extent to which they will be applied. International standards to be reviewed include the following: the Performance Standards on Environmental and Social Sustainability (IFC, 2012a) of the International Finance Corporation, a unit of the World Bank Group, including IFC PS Guidance Notes (IFC, 2012b); IFC's General Environmental, Health, and Safety Guidelines (2007a); IFC's Environmental, Health, and Safety Guidelines for Mining (2007b); IFC's Policy on Disclosure of Information (2006); the World Bank's Anti-Corruption Strategy (2007 and 2012); the Voluntary Principles on Security and Human Rights (2000); and the Equator Principles III (2013) .

These international guidelines and standards provide a project owner with: guidelines for conducting an International ESIA (I-ESIA); a set of specific environmental quality standards, including both "end of pipe" discharge limits and acceptable ambient levels for various parameters; extensive operating management practices (known as "good international industry practices" or GIIP); standards of performance for the design, construction, operation, and closure of a mine project; and, a system for formal documentation of social and environmental studies, programs and practices. GRI is committed to voluntary consideration of these international guidelines and standards for the Project as may be applicable.

GRE has submitted the application for an AOT, and the TDR for the Project will be submitted as soon as the AOT is approved. The Project ESIA is in the process of being prepared. Application of the AANR for exploitation will be submitted as soon as the Project ESIA is approved, which is expected to be in 2018.

ENVIRONMENTAL AND SOCIAL IMPACT ASSESSMENT (ESIA)

Two separate but parallel ESIAs are being prepared for the Project. The ESIAs are intended to meet Venezuelan regulatory requirements and international standards and guidelines. The Venezuelan ESIA (VZ-ESIA) is expected to be completed and submitted to the MINEA in 2018; and, the International ESIA (I-ESIA) will be completed soon thereafter.

ENVIRONMENTAL AND SOCIAL MANAGEMENT

In addition to the ESIAs, GRE is in the process of developing a series of environmental and social management plans and programs. Thousands of small-scale miners are actively working in the Project area and adequate management of small-scale mining is critical to the success of the Project. A conceptual plan for small-scale mining management has been developed and is summarized below. Summaries of the Closure and Reclamation Plan and Waste Management Plan that are currently in preparation are also presented below. Other environmental and social management plans and programs being developed, including among others: Indigenous Peoples Plan, Cultural Heritage Plan, Community Development Plan, Public Consultation and Disclosure Plan, Resettlement Action Plan, Environmental Management Plan, Small-Scale & Artisanal Miners Plan, Occupational Health and Safety Plan, Environmental Protection Plan, Erosion and Sediment Control Plan, Environmental Monitoring Plan, Emergency Response Plan, Hazardous Materials Management Plan, Hazardous Waste Management Program, Cyanide Management Plan, Site Water Management Plan, and, Biodiversity Management Plan.

It is impractical and politically untenable to forcibly remove the artisanal miners from the Project area. GRE has developed a conceptual plan to relocate these miners to the Oro concession area north of Diversion Channel #4. The conceptual plan includes an oxide saprolite processing and stockpile area with concrete tailings ponds that collect and transport tailings from the artisanal mining operations to the Project tailings storage facility.

The fundamental goals of this proposed plan are as follows:

- Encourage the small-scale miners to relocate to an area away from the active, large scale mining operations.
- Improve the health and safety of the small-scale miners and their families.
- Decrease the sediment and mercury contamination impacts to the river system.
- Discourage additional small-scale miners from moving into the area.
- Do not commit CVG or GRM to post-mining public support (welfare) type programs and limit or avoid contingent liabilities.
- Limit continued physical impacts from the small-scale miners to the oxide saprolite section of the open pit, which makes open pit mining operations difficult.

The proposed approach to the small-scale miner program is to develop an area to the northeast of the planned pit area where oxide saprolite can be mined with conventional open pit methods and stockpiled for future use by the small-scale miner community. The current group of small-scale miners who are actively mining would be given areas around this low profile stockpile where they could continue their placer mining operations. Oxide saprolite would be delivered and stockpiled to a plant that is collocated adjacent to the main plant, in relatively low, stable stockpiles for processing.

The tailings launder would deliver tailings and sediments to the main tailings pond, followed by a sediment impoundment constructed above the Cuyuni River. This impoundment would also collect tailings and sediment from upstream miners who are not actively mining in the Project area, effectively cleaning up the river. The sediment may require flocculant to meet discharge requirements for suspended solids.

Each small-scale miner and their family would have a small area to process material (6 m by 30 m) plus an area for constructing temporary living quarters. Water for the mining operations and potable water would be provided from the water supply pipeline, which would be supplied either by the pit dewatering operations or an upstream river. Drinking water will be treated, if required, to meet applicable standards. Sewage would likely be discharged into the tailings pond, which is physically separated from the public, limiting contact and cross-contamination. In addition, significant dilution of sewage would be provided by the tailings and processing water.

GRE would also construct a mercury retort at the processing location in order to return gold to each small-scale miner while removing mercury from the environment in a safe and controlled manner. The goal is to turn operation of the retort over to either a small-scale miner cooperative or to the government. Benefits include:

- Small-scale miners are able to process more material than they can currently.
- Improved physical safety of the miners since they would not be working in pits with stability problems and water drainage issues.
- Improved human safety, through better water supply and wastewater systems.
- Dramatically improved river system water quality by limiting the discharge of sediments and mercury into local drainages.

Significant positive cumulative impacts would also be expected, which may help improve the existing adverse baseline conditions over time. These include an increase of wages and incomes; skill and capacity building; demand for and improved quality of goods and services; company and local government driven investments in social infrastructure and services; and, additional sources of tax and royalty incomes. Coordination between the Project, local government agencies, local communities and other stakeholders can help to further mitigate any adverse cumulative impacts, as well as, leverage and accelerate positive impacts.

Much of the Project area has been deforested and hydraulically mined by the artisanal miners. As a result, there are numerous water-filled pits and large areas of tailings material. Some areas of poorly consolidated tailings are unstable and cannot support the weight of a vehicle, making access to some areas extremely difficult or impossible.

GRE is committed to providing technical assistance to small-scale miners including the identification of suitable areas where small-scale mining can continue to mine up to 365,000 tpa of oxide saprolite.

CLOSURE AND RECLAMATION PLAN

Based on the current Project design, reclamation activities will commence soon after construction begins, and will continue throughout the life of the Project. Closure activities will continue for three years after the end of the mine life in year 45. Some intermittent reclamation would also take place during the mine operations, when areas are no longer needed. Total expenditures for reclamation and closure are currently estimated to be US$150 million.

The objectives, criteria, and conceptual plans proposed in the Reclamation and Closure Plan for the Project will be the subject of future mine management and planning and, as such, subject to continuing refinement. GRM is committed to continuous reclamation of disturbed areas throughout the life of the Project and will implement an advanced, modern environmental management and monitoring program to include reclamation and closure activities. The mine, all equipment, and all facilities will revert to the Government of Venezuela at the end of the Project.

WASTE MANAGEMENT PLAN

Wherever possible, re-use of recoverable material in all operations will be considered. Domestic and industrial liquid wastes generated at Project site will be collected, properly treated prior to disposal. There exists a large amount of uncontrolled waste disposal throughout the site and environs which will require cleanup.

A large proportion of typical domestic solid wastes (about 60-70%) will be combustible and could be disposed of by incineration. Domestic solid wastes are largely inert materials and therefore may be landfilled. Landfillable materials will be disposed of by burial in the mine waste rock deposition area.

Mine waste rock and tailings are the most significant solid wastes to be generated from the Project. Specific maintenance and monitoring programs for the operations of mine waste rock and tailings deposition areas are being developed and will be implemented to minimize environmental impacts. The management of other solid industrial wastes includes minimization, recycling, and source separation between non-hazardous solid wastes and hazardous chemical wastes. Scrap metal and packing materials will be collected and stored for recycling, where practical. A handling procedure for used drums and oil filters will be established to prevent spillage, loss, or damage. All used containers, construction materials, and equipment will be returned to the suppliers. Other non-hazardous solid wastes will be separated and disposed of by incineration and/or landfill.

Hazardous wastes will be treated and handled according to applicable Venezuelan regulations, as well as generally-accepted international standards. Hazardous wastes associated with the Project will include used equipment lubrication oils, automobile batteries, paints, solvents, caustic or acid cleaners, pesticide wastes, used oil filters, hydraulic fluids (coolant), and miscellaneous chemicals and solid wastes. Hazardous wastes will be collected and temporarily stored in closed containers as soon as they are generated. The containers will then be transported periodically to approved recycle/disposal facilities, which may need to be constructed at or near the Project site. All containers with hazardous waste will be clearly marked and posted with warning signs. A more detailed Hazardous Waste Management Plan will be developed for the Project during the I-ESIA process.

ENVIRONMENTAL AND SOCIAL SUMMARY

Modern mining techniques and practices, including advanced environmental and social management, will result in great improvements in the lives of the local people, and will yield major benefits to the environment. Health and safety improvements notwithstanding (e.g., malaria and disease control, health services, modern safety procedures and practices, support of clinics, etc.), the capture and control of mercury, elimination of mercury use, sediment control, and modern operations, reclamation, and restoration will have significantly positive effects on the entire area, and downstream on the Cuyuni River. The dramatic environmental impacts of small miner activities will be reversed with application of highly managed modern mining techniques, practices, and procedures.

21 CAPITAL AND OPERATING COSTS

All capital and operating costs are in Q4 2017 US dollars (US$). Due to inflationary conditions in Venezuela, all costs are estimated in US dollars.

CAPITAL COSTS

The mine capital costs are estimated by RPA with updated quotes received from Venequip in November 2017 for most of the Caterpillar mine equipment.

The process and infrastructure capital costs for the CIP plant are taken from the PEA Oxide Saprolite Basis of Estimate that was completed by Samuel Engineering in 2017.

The process and infrastructure capital costs for the flotation concentrator are estimated using the SNC-Lavalin definitive detailed capital costs estimate for the 70,000 tpd plant dated March 31, 2008. The cost was factored to increase the tonnage to 140,000 tpd and the costs were escalated from 2008 to 2017. The plant layout was optimized to provide a plant feed of 140,000 tpd and the mechanical equipment costs and labour costs were also escalated to 2017 values using price escalations and fluctuations in currency exchange rates.

The tailings dam capital costs are estimated by the Tierra Group in 2017. The Owner's Cost estimate is estimated jointly by GRE and RPA.

Based on the available information used in the study, under Association for the Advancement of Cost Engineering (AACE) guidelines, the capital estimate is considered a Class 4 estimate (scoping study). RPA considers the accuracy of the overall estimate to be +35% to -15%.

DEVELOPMENT CAPITAL

Initial capital estimates for both the CIP and concentrator processing plants plus related activities are combined into a single development capital estimate. The leach plant construction is planned to commence in Q1 of Year -2 and is scheduled to begin commercial production in Q1 of Year 1. The concentrator plant construction is planned to commence in

Q1 Year 1 and is scheduled to begin commercial production in Q1 of Year 3. Table 21-1 presents a summary breakdown of the $2,571 million development capital costs.

TABLE 21-1 DEVELOPMENT CAPITAL COST SUMMARY
GR Engineering (Barbados), Inc. – Siembra Minera Project

Description	Total US$ M	Yr -2	Yr -1	Yr 1	Yr 2	Yr 3
Direct Costs						
Mining	436.6	0.0	174.1	41.7	15.1	205.7
Processing	923.5	60.5	221.2	187.9	452.2	1.8
Engineering & Geology	15.9	2.7	8.1	3.5	1.6	0.0
ARD Plant	2.3	0.0	2.3	0.0	0.0	0.0
Site Infrastructure	111.8	16.1	52.3	21.1	22.3	0.0
Subtotal Direct Costs	**1,490.1**	**79.3**	**457.9**	**254.2**	**491.3**	**207.5**
Indirect Costs						
Construction Indirects	312.3	10.7	70.8	64.0	166.8	0.0
Owner's Cost	310.4	45.0	88.1	66.2	85.0	26.1
Subtotal Indirect Costs	**622.7**	**55.7**	**158.9**	**130.1**	**251.9**	**26.1**
Contingency	457.8	37.1	128.7	90.7	182.8	18.5
Total	**2,570.6**	**172.1**	**745.5**	**475.0**	**925.9**	**252.1**

Contingency has been applied to the estimate as a deterministic assessment of the level of confidence in each of the defining inputs to the item cost being engineering, estimate basis and vendor or contractor information. Contingency values applied ranged from 5% to 30%, for an overall Project contingency of approximately 22%.

Tables 21-2 through 21-4 show the details behind the development capital cost estimate.

TABLE 21-2 DEVELOPMENT CAPITAL DIRECT COST DETAILS

GR Engineering (Barbados), Inc. – Siembra Minera Project

Description	Total US$ M	Yr -2	Yr -1	Yr 1	Yr 2	Yr 3
Pre-Stripping	40.6	0.0	40.6	0.0	0.0	0.0
Mining Equipment	396.0	0.0	133.4	41.7	15.1	205.7
Subtotal Mining	**436.6**	**0.0**	**174.1**	**41.7**	**15.1**	**205.7**
Processing - CIP	97.0	14.6	82.5	0.0	0.0	0.0
Processing - Concentrator	696.8	13.9	104.5	160.3	418.1	0.0
Processing - Tailings Dam	54.9	19.8	19.8	7.5	7.7	0.0
Processing - Port	19.7	0.0	0.0	4.0	13.9	1.8
Processing - Cristinas Diversion	32.8	11.7	10.5	10.6	0.0	0.0
Processing - Other	22.3	0.4	3.9	5.5	12.5	0.0
Subtotal Processing	**923.5**	**60.5**	**221.2**	**187.9**	**452.2**	**1.8**
Engineering & Geology	15.9	2.7	8.1	3.5	1.6	0.0
ARD Plant	2.3	0.0	2.3	0.0	0.0	0.0
Infra: Buildings/Roads/Utilities	52.5	7.8	26.4	9.2	9.2	0.0
Infra: Earthworks - TMF	13.1	2.1	8.8	1.7	0.5	0.0
Infra: Earthworks - Other	26.1	5.9	14.1	5.6	0.6	0.0
Infra: Electrical/Power	20.2	0.4	3.0	4.6	12.1	0.0
Subtotal Site Infrastructure	**111.8**	**16.1**	**52.3**	**21.1**	**22.3**	**0.0**
Total	**1,490.1**	**79.3**	**457.9**	**254.2**	**491.3**	**207.5**

Construction indirect cost estimates for the CIP plant and concentrator are estimated by Samuel Engineering. The Owner's Cost estimate is composed of Owner's construction and CIP plant start-up team as well as allowances for feasibility study costs and pre-production reserve conversion drilling. Light vehicles and the GRE management fee (5% of annual direct plus indirect capital) are also added.

TABLE 21-3 DEVELOPMENT CAPITAL INDIRECT COST DETAILS

GR Engineering (Barbados), Inc. – Siembra Minera Project

Description	Total US$ M	Yr -2	Yr -1	Yr 1	Yr 2	Yr 3
CIP Plant						
Temporary Facilities	3.2	0.5	2.7	0.0	0.0	0.0
Temporary Services	6.5	1.0	5.5	0.0	0.0	0.0
Camp Facility and Catering	1.3	0.2	1.1	0.0	0.0	0.0
First Fills/Critical Spares	1.8	0.3	1.5	0.0	0.0	0.0
Freight	5.3	0.8	4.5	0.0	0.0	0.0
Taxes	0.0	0.0	0.0	0.0	0.0	0.0
Vendor Reps	0.7	0.1	0.6	0.0	0.0	0.0
EPCM	15.5	2.3	13.2	0.0	0.0	0.0
Subtotal CIP Plant Indirects	**34.3**	**5.1**	**29.1**	**0.0**	**0.0**	**0.0**
Concentrator						
Temporary Facilities	7.1	0.1	1.1	1.6	4.3	0.0
Temporary Services	17.9	0.4	2.7	4.1	10.8	0.0
Camp Facility and Catering	44.9	0.9	6.7	10.3	26.9	0.0
First Fills/Critical Spares	12.2	0.2	1.8	2.8	7.3	0.0
Freight	54.7	1.1	8.2	12.6	32.8	0.0
Taxes	0.0	0.0	0.0	0.0	0.0	0.0
Vendor Reps	8.2	0.2	1.2	1.9	4.9	0.0
EPCM	133.0	2.7	20.0	30.6	79.8	0.0
Subtotal Concentrator Indirects	**278.1**	**5.6**	**41.7**	**64.0**	**166.8**	**0.0**
Total Construction Indirects	**312.3**	**10.7**	**70.8**	**64.0**	**166.8**	**0.0**
Owner's Cost						
Feasibility Studies	30.0	6.0	15.0	4.5	4.5	0.0
Annual Reserve Conversion Drilling	27.5	15.0	7.5	2.5	2.5	0.0
Owner's Construction Team	39.7	15.9	12.9	3.0	1.7	6.2
Owner's Operations Team - CIP	78.6	0.0	15.8	30.8	32.1	0.0
Lt Vehicles	13.8	0.0	1.9	3.1	0.8	8.1
GRM Management Fee	120.7	8.1	35.0	22.3	43.5	11.8
Total Owner's Cost	**310.4**	**45.0**	**88.1**	**66.2**	**85.0**	**26.1**
Total	**622.7**	**55.7**	**158.9**	**130.1**	**251.9**	**26.1**

The buildup of development capital contingency of 22% is shown in Table 21-4. Approximately 58% of the overall contingency for development capital is related to the process plant construction. In addition to the quoted 20% and 24.4% contingency from Samuel Engineering for the CIP and concentrator plants, RPA included additional contingency in various line items as it deemed appropriate for a PEA study.

TABLE 21-4 DEVELOPMENT CAPITAL CONTINGENCY DETAILS

GR Engineering (Barbados), Inc. – Siembra Minera Project

Description	Factor	% of Total	US$ M
Pre-Stripping	25%	2%	10.2
Mining Equipment	5%	4%	19.8
CIP (D+I)	20%	6%	26.3
Concentrator (D+I)	24%	52%	238.6
Tailings Dam	30%	4%	16.5
Port	20%	1%	3.9
Cristinas Diversion	30%	2%	9.8
Processing Other (Vehicles)	20%	1%	4.5
Eng and Geo	20%	1%	3.2
ARD Plant	20%	0%	0.5
Infra - Buildings/Roads/Utilities	30%	3%	15.7
Infra - All Earthworks	30%	3%	11.8
Infra - Electrical	20%	1%	4.0
Owner's Cost	30%	20%	93.1
Total	**21.7%**	**100.0%**	**457.8**

SUSTAINING CAPITAL

Sustaining capital totals approximately $1.941.7 million over the LoM, starting in Year 3. Sustaining capital costs account for equipment that needs to be replaced over the LoM, TMF expansions, leach plant conversion to 35 ktpd, and new infrastructure construction. Table 21-5 summarizes the sustaining capital costs.

TABLE 21-5 SUSTAINING CAPITAL COST SUMMARY

GR Engineering (Barbados), Inc. – Siembra Minera Project

Description	US$ M
Annual Reserve Conversion Drilling	100.0
Mining Equipment	1,212.6
Processing - Plant Sustaining	11.0
Processing - TMF Raises	322.5
Processing - Concentrate Trucks	34.2
Processing - Leach Plant Conversion to 35ktpd	35.0
Engineering & Geology	30.1
Earthworks - TMF	9.5
Lt Vehicles (pickups/vans/buses)	156.8
GRM Management Fee (5% of Annual non-Mining Capital Cost)	30.0
Total	**1,941.7**

The annual reserve conversion drilling category is the $2.5 million per year cost to upgrade the current Inferred Mineral Resources in the mine schedule to Measured and Indicated status which then can be converted to Proven and Probable Mineral Reserves. Mining infrastructure and equipment is largely composed of replacement mine equipment, specifically haul trucks, as detailed in Section 16. Details behind the sustaining capital estimate for the TMF raises and processing plant are presented in Sections 17 and 18.

CLOSURE COSTS

The Project economic analysis has a $150 million LoM closure cost estimate. The origin of this estimate is from the 2006 SNC-Lavalin estimate for Brisas of $52 million, which has been escalated and factored based on the area of disturbance and the cost is spread over the LoM as concurrent reclamation as well as final closure activities. Due to the long life of the Project, a final end of mine (EoM) closure cost expenditure has negligible effect on Project economics, however, a closure plan with a revised cost estimate should be included in the next stage of study.

SALVAGE VALUE

No salvage value was estimated as part of the Project economic analysis.

WORKING CAPITAL

RPA developed a $195 million working capital estimate during the first four years of commercial operations. This estimate includes the following assumptions:

- Accounts Receivable: 5 days doré; 45 days/90% concentrate, 90 days/10% concentrate
- Accounts Payable: 45 days on all operating costs (except labour)
- Inventories: 30 days worth of inventorial mining and processing consumables (< 1-year useful life)

All working capital expenditures are recaptured by the end of the mine life and thus net to zero. Note that this estimate does not include first fills/critical spares which are included in Samuel Engineering's CIP plant and concentrator indirect capital costs ($1.8 million and $12.2 million, respectively). Those costs are capitalized and thus depreciable whereas working capital represents annual adjustments to operating revenue and expense and is not depreciable.

EXCLUSIONS

The following items were excluded from the capital cost estimate:

- Project financing and interest charges
- Escalation during construction
- Permits and licences
- Joint venture sunk costs of $7 million actuals through September 2017 and an estimated additional $400,000 through the remainder of 2017
- Exchange rate variations

OPERATING COSTS

A summary of the LoM operating costs is presented in Table 21-6.

TABLE 21-6

OPERATING COST SUMMARY

GR Engineering (Barbados), Inc.– Siembra Minera Project

Description	LoM Cost US$/t milled
Mining (1.36/t mined)	2.89
Process	4.93
G&A	1.32
Other Infrastructure	0.14
Direct Operating Costs	**9.29**
Concentrate Freight	0.36
Off-site Costs	0.54
Total	**10.19**

Operating costs for the Project have been estimated from first principles. Labour costs were estimated using Project specific staffing, salary, wage, and benefit requirements. Unit consumption of materials, supplies, power, water, and delivered supply costs were also estimated.

The operating costs presented are based upon ownership of all Project production equipment and site facilities, as well as the Owner employing and directing all operating, maintenance, and support personnel.

<u>Consumable Pricing</u>

The consumable costs were estimated from supplier quotes, inputs from GRE, and in-house references. The Project's major consumable pricing and basis is as follows:

- Diesel – The Project has a pre-production and LoM diesel price of $0.02/L delivered to site. Diesel is primarily used in the mining cost estimate and the pre-production phase as well as the first two years of production at the leach plant (2020-2021) where power is supplied by an onsite genset farm.
- Power – Both genset and grid power costs were estimated at $0.038/kWh. Genset power is utilized during pre-production period and first two years of leach plant operation. Grid power comes online in 2022 with the start of the concentrator plant.

Consumption levels of the major consumables are presented in Sections 16, 17, and 18, for mining, processing, and infrastructure respectively.

<u>Manpower</u>

Table 21-7 shows the total Project headcount of 1,549 for Year 5 by area.

TABLE 21-7 YEAR 5 ANNUAL HEADCOUNT DETAIL

GR Engineering (Barbados), Inc. – Siembra Minera Project

Area	Headcount
Mine	975
Process	305
On-Site G&A	177
Off-Site G&A	31
Port	61
Total	**1,549**

Staffing assumptions are presented in Sections 16, 17, and 18 for mining, processing, and infrastructure.

MINE OPERATING COSTS

Mine operating costs are based on the equipment requirements as discussed in Section 16 and the operating cost per hour. Equipment hourly operating costs were updated to current prices using InfoMine's 2016 Mine Cost Service. Mine unit operating costs are presented in Table 21-8.

TABLE 21-8 MINE UNIT OPERATING COSTS ($/T)

GR Engineering (Barbados), Inc. – Siembra Minera Project

Mining	US$/tonne mined
Labour	
Supervision Labour	0.082
Operating Labour	0.148
Maintenance Labour	0.060
Blasting Supplies	0.157
Equipment	
Diesel Fuel	0.020
Tires	0.160
Oil & Grease	0.182
Supplies & Parts	0.138
Shop Supplies	0.170
Major Repairs	0.197
Ground Engaging Tools & Wear Plates	0.043
Light Vehicles	0.004
Mining Total	1.358

PROCESS OPERATING COSTS

Process operating costs for the flotation concentrator are estimated using the same reagent and consumables consumptions that were used for the Aker-Kvaerner feasibility study and SNC-Lavalin's basic engineering. The consumptions for the oxide leach plant are estimated using data from the Cristinas Technical Report (MDA, 2007). RPA updated costs for reagents and consumables using data from similar projects.

OXIDE LEACH PLANT

Table 21-9 summarizes the costs to leach the oxide ore.

TABLE 21-9 REAGENT AND CONSUMABLES COSTS FOR LEACHING OXIDE SAPROLITE

GR Engineering (Barbados), Inc. – Siembra Minera Project

Item	Consumption, kg/t	Cost US$	Units	US$/t
Saprolite Crusher Wear Parts	0.009	1.80	kg	0.016
Ball Mill Liners	0.010	1.90	kg	0.019
Ball Mill Balls	0.131	1.09	kg	0.143
NaCN	0.520	2.68	kg	1.396
Lime	1.640	0.26	kg	0.420
HCl	0.050	0.55	kg	0.028

Item	Consumption, kg/t	Cost US$	Units	US$/t
NaOH	0.040	1.05	kg	0.042
Carbon	0.010	3.35	kg	0.034
Sodium Metabisulfite	0.700	0.96	kg	0.675
Flocculent	0.050	3.53	kg	0.177
Diesel for Refinery (L/t)	0.140	0.03	L	0.004
Refinery Fluxes and Supplies	150,000	annual allowance		0.029
Maintenance Supplies	5%	equipment cost		0.434
Operating Supplies	15%	maintenance supplies		0.065
Analytical Supplies	500,000	annual allowance		0.095
Total				**3.575**

Table 21-10 summarizes the costs to leach sulphide saprolite.

TABLE 21-10 REAGENT AND CONSUMABLES COSTS FOR LEACHING SULPHIDE SAPROLITE

GR Engineering (Barbados), Inc. – Siembra Minera Project

Item	Consumption, kg/t	Cost US$	Units	US$/t
Saprolite Crusher Wear Parts	0.009	1.80	kg	0.016
Ball Mill Liners	0.010	1.90	kg	0.019
Ball Mill Balls	0.131	1.09	kg	0.143
NaCN	0.700	2.68	kg	1.879
Lime	2.180	0.26	kg	0.230
HCl	0.392	0.55	kg	0.216
NaOH	0.383	1.05	kg	0.400
Carbon	0.038	3.35	kg	0.126
Sodium Metabisulfite	0.700	0.96	kg	0.675
Flocculent	0.050	3.53	kg	0.177
Diesel for Refinery (L/t)	0.140	0.02	L	0.003
Refinery Fluxes and Supplies	150,000	annual allowance		0.029
Maintenance Supplies	5%	equipment cost		0.465
Operating Supplies	15%	maintenance supplies		0.070
Analytical Supplies	500,000	annual allowance		0.095
Total				**4.542**

Table 21-11 summarizes the costs to leach hard rock.

TABLE 21-11 REAGENT AND CONSUMABLES COSTS FOR LEACHING HARD ROCK

GR Engineering (Barbados), Inc. – Siembra Minera Project

Item	Consumption, kg/t	Cost US$	Units	US$/t
SAG Mill Liners	0.052	1.90	kg	0.098
Ball Mill Liners	0.020	1.90	kg	0.038
Crusher Liners	0.008	1.80	kg	0.014
SAG Mill Balls	0.323	1.16	kg	0.374
Ball Mill Balls	0.295	1.09	kg	0.322
NaCN	0.520	2.68	kg	1.396
Lime	1.640	0.26	kg	0.420
HCl	0.050	0.55	kg	0.028
NaOH	0.040	1.05	kg	0.042
Carbon	0.010	3.35	kg	0.034
Sodium Metabisulfite	0.700	0.96	kg	0.675
Flocculent	0.050	3.53	kg	0.177
Diesel for Refinery (L/t)	0.140	0.02	L	0.003
Refinery Fluxes and Supplies		annual allowance		0.029
Maintenance Supplies	5%	equipment cost		0.434
Operating Supplies	15%	maintenance supplies		0.065
Analytical Supplies	500,000	annual allowance		0.095
Total				**4.242**

FLOTATION CONCENTRATOR

Table 21-12 summarizes the costs to process sulphide saprolite or hard rock in the flotation concentrator.

TABLE 21-12 REAGENT AND CONSUMABLES COSTS FOR FLOTATION OF SULPHIDE SAPROLITE AND HARD ROCK

GR Engineering (Barbados), Inc. – Siembra Minera Project

Item	Consumption, kg/t	Cost US$	Units	US$/t
SAG Mill Liners	0.052	1.90	kg	0.098
Ball Mill Liners	0.020	1.90	kg	0.038
Crusher Liners	0.008	1.80	kg	0.014
SAG Mill Balls	0.323	1.16	kg	0.374
Ball Mill Balls	0.295	1.09	kg	0.322
Regrind Mill Balls	0.015	1.09	kg	0.016
Lime	0.900	0.26	kg	0.230
NaCN (leach feed)	1.270	2.68	kg	3.409
Flocculant	0.089	3.53	kg	0.315
AP3477	0.022	2.90	kg	0.062
PAX	0.019	3.52	kg	0.066
Frother	0.032	2.55	kg	0.082

Item	Consumption, kg/t	Cost US$	Units	US$/t
Sodium Metabisulfite	0.707	0.96	kg	0.682
Water / Boiler Treatment	20,000	annual allowance		0.0004
Maintenance Supplies	5%	equipment cost		0.400
Operating Supplies	15%	maintenance supplies		0.060
Analytical Supplies	2,000,000	annual allowance		0.041
Total				**6.210**

PROCESS LABOUR COSTS

The process labour costs were estimated by developing a conceptual organizational chart for the operation and estimating the number of personnel who are required for each position. The cost estimates take into account the number of personnel required to operate the oxide leach plant in the first two years and the transition to operating both the leach plant and the flotation plant in the later years of the operation. The positions, number of people, and salary estimates are summarized in Table 21-13.

TABLE 21-13 SUMMARY OF PROCESS LABOUR COSTS
GR Engineering (Barbados), Inc. – Siembra Minera Project

Item	Salary US$	Years 1 - 2 Number	Annual Cost US$	Years 3 - 41 Number	Annual Cost US$
Leach Plant					
Plant Manager (expat)	504,000		-		-
Metallurgy & Laboratory Manager (expat)	360,000	2	720,000		-
Metallurgy & Laboratory Professionals	62,288	2	124,577		-
Metallurgy & Laboratory Staff	24,539	4	98,156		-
Sample Preparation	6,435	12	77,217		-
Analytical Clerks	10,168	2	20,336		-
Production Superintendent (expat)	432,000	1	432,000		
Mill General Foremen (expats)	288,000	-	-		-
Shift Foreman	24,539	4	98,156	4	98,156
Plant Operators	10,227	32	327,275	32	327,275
Operation Helpers	6,435	32	205,913	32	205,913
Tailings Dam Foreman[1]	50,233	-	-		-
Tailings Dam Technicians[1]	6,434	-	-		-
Maintenance Superintendent (expat)	331,200	1	321,600		-
Maintenance General Foremen (expats)	321,600	-	-		-
Shift Foreman	24,539	2	20,455		-
Mechanics & Electricians	10,227	14	436,199	8	249,256
Maintenance Planners & Other	31,157	2	12,870		-
Maintenance Helpers	6,435	14	90,087	8	-
Total Oxide Plant		**124**	**2,984,840**	**84**	**880,601**

Item	Salary US$	Years 1 - 2 Number	Annual Cost US$	Years 3 - 41 Number	Annual Cost US$
Flotation Plant					
Plant Manager	504,000		-	1	504,000
Metallurgy & Laboratory Manager (expat)	360,000		-	1	360,000
Metallurgy & Laboratory Professionals	62,288		-	2	124,577
Metallurgy & Laboratory Staff	24,539		-	25	613,474
Sample Preparation	6,435		-	27	173,739
Analytical Clerks	10,168		-	3	30,504
Production Superintendent (expat)	432,000		-	1	432,000
Mill General Foremen (expats)	288,000		-	2	576,000
Shift Foreman	24,539		-	8	196,312
Plant Operators	10,227		-	28	286,366
Operation Helpers	6,435		-	48	308,870
Tailings Dam Foreman[2]	50,233		-	2	100,465
Tailings Dam Technicians[2]	6,434		-	8	51,475
Maintenance Superintendent (expat)	331,200		-	1	331,200
Maintenance General Foremen (expats)	321,600		-	2	643,200
Shift Foreman	24,539		-	8	196,312
Mechanics & Electricians	10,227		-	24	245,456
Maintenance Planners & Other	31,157		-	6	186,942
Maintenance Helpers	6,435		-	24	154,435
Total Concentrator		0	-	221	5,515,327
Total Both Plants		**124**	**2,984,840**	**305**	**6,395,928**

Notes:

[1]RPA and SE determined that additional tailings dam foreman and operator positions were not needed in Years 1 and 2 as their duties could be handled by the shift foreman and other operators/helpers, respectively.

[2]For simplicity, RPA and SE placed tailings dam foreman and technician positions within the larger flotation plant headcount in Years 3 to 41.

POWER COSTS

Power costs were estimated by using the electrical load lists from the Samuel Engineering conceptual design that serves as the basis for the PEA and the SNC-Lavalin basic engineering design for the flotation concentrator. The electrical loads and power consumption for the flotation concentrator are doubled to support this PEA with the exception of the items listed as the “Other Facilities”. They are multiplied by 1.3. The power cost in Venezuela is estimated to be $38 per MW. The power consumption estimates are provided in Table 21-14.

TABLE 21-14 SUMMARY OF POWER CONSUMPTION ESTIMATES
GR Engineering (Barbados), Inc. – Siembra Minera Project

Area	Average kW	Annual MW
Leach Plant:		
Mine Area Material Handling 250 MCC 100	445	5,251
Slurry Transfer 250 MVMCC 100	352	5,865
Grinding and Gravity 400 MCC 100	526	6,847
CIP 450 MCC 101	555	8,437
Refinery 650 MCC 102	534	6,248
General Site 100 SWGR 100	236	3,932
Tailings 500 MCC 200	559	6,246
Cyanide Destruction 700 MCC 201	458	4,580
Reagents 750 MCC 202	200	2,978
General Site 100 MVMCC 100	4,369	72,711
550 MVMCC 100	338	4,553
Total Leach Plant	8,571	127,648

Flotation Plant:	Average kW 70,000 tpd	Annual MW 70,000 tpd	Annual MW 140,000 tpd
Process Areas Services	800	6,912	13,824
Pebble Crushers & Stockpile Reclaim	720	5,700	11,400
Grinding Mills	56,000	443,520	887,040
Grinding Area Process	4,800	38,016	76,032
Flotation	4,700	37,224	74,448
Regrinding	1,580	12,516	25,032
Tailings	2,240	17,736	35,472
Leach & CIL	1,328	10,512	21,024
Refinery	1,160	9,192	18,384
Reagents	128	1,008	2,016
Water Ponds	160	696	1,392
Reclaim Barge	360	2,856	5,712
Metallurgical Laboratory	400	1,728	3,456
Primary Crusher	640	4,608	9,216
Overland Conveyor	1,760	12,672	25,344
Power Loss	2,100	18,147	36,294
Total	78,876	623,043	1,246,086
Other Facilities:			
Truck Shop/ Wash/ Fuel	480	4,152	5,398
Administration Building	140	600	780
Warehouse & Maintenance Shop	200	1,728	2,246
Camp	400	3,456	4,493
Power Loss	35	298	388
Total Other Facilities		10,234	13,304
Port Site Power (from March 2006 est.)	500	4,320	8,640

The average annual power costs, based on $38 per MW, are provided in Table 21-15.

TABLE 21-15 SUMMARY OF AVERAGE ANNUAL POWER COSTS

GR Engineering (Barbados), Inc. – Siembra Minera Project

Area	Annual Costs US$
Leach Plant	2,425,308
Flotation Plant	23,675,629
Other Facilities	388,895
Port Site	164,160

The cost estimates were not changed in year 10 when more hard rock will go to the leach plant and less hard rock will be processed in the flotation plant since grinding the hard rock is planned in the existing flotation plant and none of the site facilities are expected to change.

Also, in years 1 and 2, when only the leach plant will be operating and powered only by the genset farm, RPA estimates that the power cost unit rate would equal the grid power rate of $38/MWh, also expressed as $0.46 per tonne.

The total process operating costs are summarized in Table 21-16.

TABLE 21-16 SUMMARY OF PROCESS OPERATING COSTS ($/t)

GR Engineering (Barbados), Inc. – Siembra Minera Project

Item	Leach Oxide Saprolite	Leach Sulphide Saprolite	Flotation Saprolite	Flotation Hard Rock
Steel	0.18	0.18	0.85	0.86
Reagents & Supplies	3.40	4.36	3.72	5.35
Power	0.46	0.46	0.97	0.97
Labour	0.57	0.57	0.57	0.13
Total	**4.61**	**5.57**	**6.10**	**7.31**

Additional work is required in future studies in order to improve the accuracy of the operating costs. For example, the cost to treat sulphide saprolite in the flotation plant is expected to be lower than the costs listed due to lower power costs for grinding the softer material.

G&A OPERATING COSTS

General and administrative costs run a nominal $38 million to $42 million per year. Unit operating costs are presented in Table 21-17.

TABLE 21-17 SUMMARY OF G&A OPERATING COSTS ($/t) **GR Engineering (Barbados), Inc. – Siembra Minera Project**

Description	US$/tonne processed
Site Administration Labour Costs	0.271
G&A Off-Site Labour Costs	0.023
Total G&A Labour & Supervision	**0.294**
G&A Expenses	0.692
Maintenance Consumables	0.052
Electricity	0.013
Light Vehicles	0.014
Management Fee	0.522
Total G&A Costs	**1.586**
Capitalized Pre-production Costs & Adjustments.	(0.262)
G & A Operating Costs Total	**1.324**

OTHER INFRASTRUCTURE OPERATING COSTS

Other Infrastructure costs run a nominal $6.5 million per year and include Engineering and Geology (pit dewatering costs plus engineering/geology team labour and systems support) and acid rock drainage (ARD) treatment costs. Unit operating costs are presented in Table 21-18.

TABLE 21-18 SUMMARY OF OTHER INFRASTRUCTURE OPERATING COSTS ($/t)
GR Engineering (Barbados), Inc. – Siembra Minera Project

Description	US$/tonne processed
Engineering & Geology Labour	0.049
Pit Dewatering Operating Costs	0.072
Pit Dewatering Labour	0.014
Total Systems Support	0.003
Total Engineering and Geology Costs	**0.138**
Total ARD Treatment Costs	**0.007**
Total	**0.145**

OFF-SITE OPERATING COSTS

Doré/concentrate freight costs and smelter/refining charges are described elsewhere in the report.

22 ECONOMIC ANALYSIS

The economic analysis contained in this report is based, in part, on Inferred Mineral Resources, and is preliminary in nature. Inferred Mineral Resources are considered too geologically speculative to have mining and economic considerations applied to them that would enable them to be categorized as Mineral Reserves. There is no certainty that economic forecasts on which this PEA is based will be realized.

A Cash Flow Projection has been generated from the LoM production schedule and capital and operating cost estimates, and is summarized in Table 22-3. A summary of the key criteria is provided below.

ECONOMIC CRITERIA

PRODUCTION

- The LoM production plan assumes that leach plant detailed engineering/early earthworks will commence in Q1 of Year -2.
- The LoM production plan assumes concentrator plant detailed engineering will commence in Q1 of Year -2.
- A 2-year pre-production period for the leach plant, 2 additional years for completion of the flotation concentrator and a 45-year overall mine life.
- The leach plant has nameplate capacity of 15,000 tpd from year 1 through year 10, which increases in year 11 to 35,000 tpd through year 45 EoM (5.8 Mtpa to 12.25 Mtpa, respectively).
- The concentrator plant has nameplate capacity of 140,000 tpd from year 3 through year 10, which decreases in year 11 to 105,000 tpd through year 45 EoM (58 Mtpa to 36.75 Mtpa, respectively).
- Total combined leach and concentrator production is 2.0 billion tonnes, at a grade of 0.70 g/t Au, 0.50 g/t Ag, and 0.090% Cu.
- The copper head grades in the mine plan are 302 Mt at 0.017% Cu and 1,703 Mt at 0.106% Cu for the leach and concentrator plants, respectively. However, the leach plant does not recover copper, thus the overall average copper head grade in the total mill feed is 2,005 Mt at 0.090% Cu.
- Average overall metal recovery of 84% Au, 53% Ag, and 84% Cu.
- Total recovered metal of 38.1 Moz Au, 17.1 Moz Ag, and 3.3 billion lb Cu.

- Average LoM annual recovered metal production of 847 koz Au, 380 koz Ag, and 78 million lb Cu.
- Average annual recovered metal production in Years 3 through 18 of 1,229 koz Au, 469 koz Ag, and 77 million lb Cu.
- Average annual recovered metal production in Years 19 through 45 EoM of 674 koz Au, 353 koz Ag, and 78 million lb Cu.

REVENUE

- Doré payable factors at refinery are 99.9% Au and 98% Ag.
- Copper concentrate average payable factors at smelter are 98% Au, 97% Ag, and 95.8% Cu.
- Payable metal sales for the Project total 37.6 Moz Au, 16.6 Moz Ag, and 3.2 billion lb Cu split as follows:
  - From Doré: 14.4 Moz Au and 4.1 Moz Ag.
  - From Concentrate: 23.2 Moz Au, 12.5 Moz Ag, and 3.2 billion lb Cu.
- Metal prices: US$1,300 per troy ounce Au; US$17 per troy ounce Ag and US$3.00 per pound Cu.
- NSR for doré includes transport and refining costs of $0.50 per ounce doré and $6 per ounce gold/$0.40 per ounce silver, respectively.
- NSR for copper concentrate includes:
  - Cost Insurance and Freight (CIF) charge of $103 per wet tonne concentrate (8% moisture content) consisting of:
    - Road Transport (350 km one way): $11/t
    - Port Charges (Puerto Ordaz) : $17/t
    - Ocean Transport (Europe): $75/t.
  - Smelter treatment charge of $95 per dry tonne concentrate.
  - Smelter refining charges of $0.095/lb Cu, $6/oz Au, and $0.40/oz Ag.
  - Copper price participation is not included.

COSTS

- Pre-production period to CIP plant First Production: 24 months (January Year -2 to December Year -1).
- Pre-production period to concentrator First Production: 48 months (January Year -2 to December Year 2).
- Project development capital totals $2.57 billion, including $459 million in contingency (22% of direct and indirect capital).
- Sustaining capital of $1.42 billion.

- Average unit operating costs in $/t milled over the mine life:

o Mine (US$1.36/t mined):		2.89
o Process:		4.93
o G&A:		1.32
o Other Infrastructure:		0.14
o **Direct Operating Costs**		**9.29**
o Concentrate Freight		0.36
o Off-site Costs		0.54
o **Total**	**$**	**10.19**

ROYALTIES AND GOVERNMENT PAYMENTS

Royalties and other government payments total $5.6 billion or $2.77/t milled over the LoM as shown in Table 22-1.

TABLE 22-1 ROYALTIES AND GOVERNMENT PAYMENTS

GR Engineering (Barbados), Inc. – Siembra Minera Project

Item	US$ M	US$/t milled
NSR Royalty	3,262.8	1.63
Special Advantages Tax	1,710.0	0.85
Science, Technology and Innovation Contributions	588.1	0.29
Total	**5,560.9**	**2.77**

The Project will pay an annual NSR royalty to Venezuela on the sale of gold, copper, and silver and any other strategic minerals of 5% for the first ten years of commercial production and 6% thereafter.

The Project is subject to an additional 3% NSR annual royalty called Special Advantages Tax which is a national social welfare fund.

The Project is subject to a 1% gross revenue levy as part of the Science, Technology and Innovation Contributions fund (LOCTI).

Customs duties and Value Added Taxes (VAT) are assumed to be waived for the Project.

INCOME TAXES, WORKING CAPITAL, AND OTHER

Income taxes/contributions, upfront working capital, and reclamation/closure costs total $8.3 billion as shown in Table 22-2. Withholding taxes on corporate dividends and interest payments are not incorporated into the Project economic analysis.

TABLE 22-2 INCOME TAXES, WORKING CAPITAL, AND OTHER

GR Engineering (Barbados), Inc. – Siembra Minera Project

Item	US$ M
Anti-Drug Contributions	283.9
Sports Contributions	283.9
Corp. Income Taxes Paid	7,373.8
Upfront Working Capital (Yrs 1 to 4)	195.4
Reclamation and Closure	150.0
Salvage Value	0
Total	**8,286.9**

Anti-drug and Sport Contributions

These profit-based taxes are assessed at 1% of current year and previous year operating income, respectively. The annual operating margin is calculated by taking annual gross revenues and deducting all operating costs and depreciation/amortization allowances.

Corporate Income Tax

The Project economic analysis incorporates a sliding scale of tax rates applicable on income based on Project phases starting in Year 1 of commercial production as follows:

- Years 1 through 5: 14%
- Years 6 through 10: 19%
- Years 11 through 15: 24%
- Years 16 through 20: 29%
- Years 21+: 34%

Year 1 is the first year of gold production, after commissioning of the 15,000 tpd oxide plant.

Deductions from income for the purpose of estimating income subject to tax include the following items:

- Operating Expense
  Expensed operating costs are deducted 100% in year incurred.
- Stockpile adjustments
  As a result of large stockpiles of mill feed being generated during the life of the mine, the Project economic analysis includes annual adjustments to EBITDA to match mining costs with recognized revenue. The net effect of these adjustments over the life of the mine is zero but the adjustments increase EBITDA in years where stockpiling exceeds processing and inversely decrease EBITDA when processing stockpile material exceeds stockpile placement amounts.

- Depreciation/Amortization
  - All prior expenditures before January 2018 are considered sunk with respect to this analysis.
  - Depreciation commences once the facilities are placed into service and the mine and mill are operating.
  - Heavy mine fleet equipment capital is depreciated using 8-year straight line (SL) method. Light vehicle capital is depreciated using 5-year SL method.
  - All process and infrastructure capital are depreciated using the Units of Production (UoP) method.
  - Capitalized pre-production activities such as pre-stripping and water management are amortized the UoP method.
  - The Project economic analysis incorporates an accelerated depreciation methodology which combines the first 12 years of annual SL depreciation allowances with the standard UoP cost basis. The resulting combined UoP/SL basis is then re-calculated using the UoP method. After 12 years, the depreciation allowances come directly from each UoP or SL category.
  - Reclamation costs are amortized during the LoM by an annual accrual of $0.035/t mined ($150 million cost divided by 4.33 billion tonnes mined). This allowance is adjusted annually by periodic reclamation capital expenditures during the LoM.
- Other Deductions

  Other deductions from income for the purposes of estimating taxable income include management fees which amount to 5% of annual operating and capital costs. The annual management fees derived from operating costs are within the G&A opex category and thus expensed 100% in the year incurred while the annual fees derived from capital costs are amortized using the UoP method starting in the year they are incurred.
- Loss Carry Forwards

  Income tax losses may be carried forward indefinitely but may not be used for prior tax years.

Upfront Working Capital

A total of $195 million has been allocated for upfront working capital in Years 1 to 4. This amount covers year over year changes in accounts receivable and payable plus consumable inventory.

Reclamation/Closure Costs

The Project economic analysis has a $150 million LoM closure cost estimate.

Salvage

No salvage value was estimated as part of the Project economic analysis.

CASH FLOW ANALYSIS

The Project as currently designed has significant variations in the mining schedule, processing methods, and head grades over its planned 45-year life. These variations are shown in Figures 22-1 and 22-2 and the resulting impact on the pre-tax free cash flow profile is shown in Figure 22-3.

FIGURE 22-1

MINE VS. MILL PRODUCTION

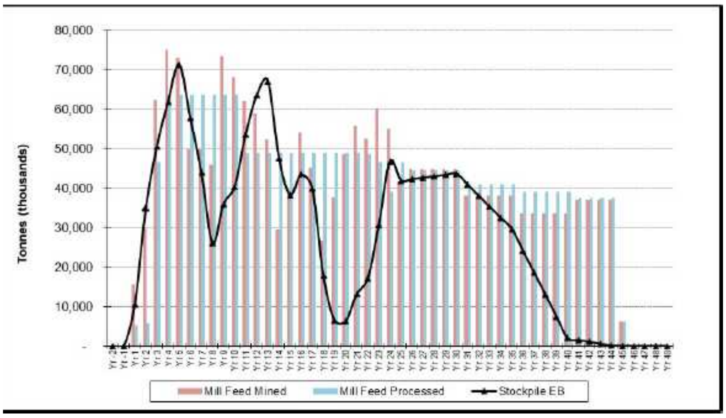


FIGURE 22-2 MILL PRODUCTION PROFILE BY PLANT

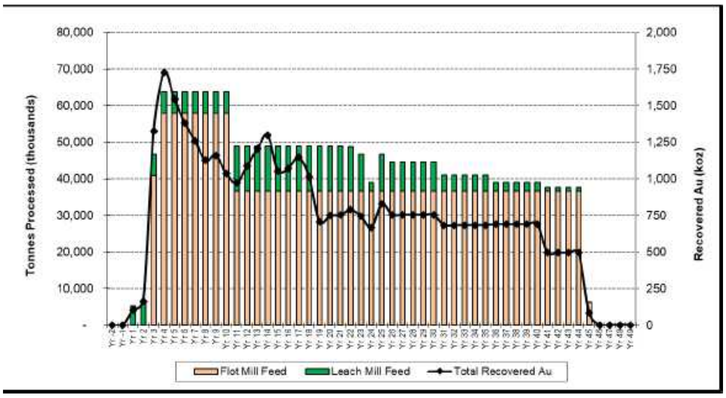


FIGURE 22-3 PROJECT PRE-TAX METRICS SUMMARY

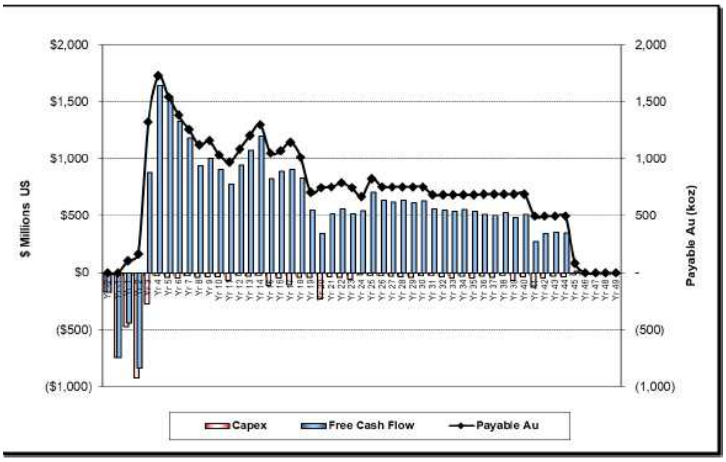


Table 22-3 shows the LoM total metrics for the Project as currently designed. Due to the length of the 45-year mine life, the full annual cash flow model is presented in Appendix 1.

TABLE 22-3 INDICATIVE PROJECT ECONOMICS

GR Engineering (Barbados), Inc. – Siembra Minera Project

Item	Unit	Value
Realized Market Prices		
Au	US$/oz	1,300
Ag	US$/oz	17.00
Cu	US$/lb	3.00
Payable Metal		
Au	Moz	37.6
Ag	Moz	16.6
Cu	Mlb	3,197.6
Total Gross Revenue	**US$ M**	**58,806.2**
Mining Cost	US$ M	(5,790.9)
Process Cost	US$ M	(9,881.0)
G & A Cost	US$ M	(2,653.6)
Other Infrastructure Cost	US$ M	(288.9)
Concentrate Freight Cost	US$ M	(728.0)
Off-site Costs	US$ M	(1,076.5)
NSR Royalty Cost	US$ M	(3,262.8)
Special Advantages Tax Cost	US$ M	(1,710.0)
Science (LOCTI) Contributions	US$ M	(588.1)
Total Operating Costs	**US$ M**	**(25,979.7)**
Operating Margin (EBITDA)	**US$ M**	**32,826.5**
Anti-Drug Contributions	US$ M	(283.9)
Sport Contributions	US$ M	(283.9)
Effective Tax Rate	%	22.5%
Income Tax	US$ M	(7,373.8)
Total Taxes	**US$ M**	**(7,941.5)**
Working Capital ($195 M in Years 1 to 4)	US$ M	0
Operating Cash Flow	**US$ M**	**24,885.0**
Development Capital	US$ M	(2,570.6)
Sustaining Capital	US$ M	(1,941.7)
Closure/Reclamation Capital	US$ M	(150.0)
Total Capital	US$ M	**(4,662.3)**
Pre-tax Free Cash Flow	US$ M	28,164.2
Pre-tax NPV @ 5%	US$ M	**11,209.4**
Pre-tax NPV @ 10%	**US$ M**	**5,534.5**
Pre-tax IRR	**%**	**36.8%**
After-tax Simple Payback	Years	3.8
After-tax Free Cash Flow	US$ M	20,222.7
After-tax NPV @ 5%	US$ M	**8,101.2**
After-tax NPV @ 10%	**US$ M**	**3,930.1**
After-tax IRR	**%**	**31.1%**
After-tax Simple Payback	Years	4.1

On a pre-tax basis, the undiscounted cash flow totals $28,164 million over the mine life. The pre-tax Internal Rate of Return (IRR) is 36.8%, and simple payback from start of commercial production occurs in 3.8 years. The pre-tax Net Present Values (NPV) are:

- $11,209 million at a 5% discount rate.
- $5,534 million at a 10% discount rate.

On an after-tax basis, the undiscounted cash flow totals $20,223 million over the mine life, the IRR is 31.1%, and simple payback from start of commercial production occurs in 4.1 years. The after-tax NPVs are:

- $8,101 million at a 5% discount rate.
- $3,930 million at a 10% discount rate.

The average annual gold sales during the forty-five years of operation is 836 koz per year (37.6 Moz over the LoM) at an average all in sustaining cost (AISC) of $483 per ounce. Table 22-4 shows the AISC build up which is net of a $262/oz copper and silver by-product credit (nbp).

TABLE 22-4 ALL-IN SUSTAINING COSTS COMPOSITION

GR Engineering (Barbados), Inc. – Siembra Minera Project

Item	US$M	US$/oz Au
Mining	5,790.9	154
Process	9,881.0	263
G & A	2,653.6	71
Other Infrastructure	288.9	8
Subtotal Site Costs	**18,614.3**	**495**
Transportation	728.0	19
Off-site Treatment	1,076.5	29
Subtotal Off-site Costs	**1,804.5**	**48**
Direct Cash Costs	**20,418.8**	**542**
Ag and Cu By-Product Credit	(9,875.4)	(262)
Total Direct Cash Costs (nbp)	**10,543.4**	**280**
NSR Royalty	3,262.8	87
Special Advantages Tax	1,710.0	45
STI Contributions	588.1	16
Total Indirect Cash Costs	**5,560.9**	**148**
Total Production Costs	**16,104.3**	**428**
Sustaining Capital Cost	1,941.7	52
Closure/Reclamation Capital	150.0	4

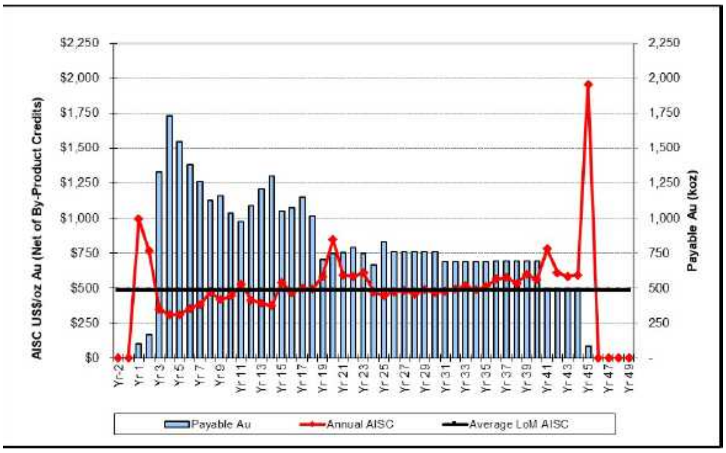
Item	US$M	US$/oz Au
Corporate G&A	0.0	0
Off-mine Exploration	0.0	0
Total Sustaining Costs	**2,091.7**	**56**
Total All-in Sustaining Costs	**18,196.0**	**483**

Figure 22-4 shows the annual AISC trend during the mine operations against an overall average AISC of $483/payable oz over the 45-year LoM at an annual production rate of 836 koz Au per year. The AISC variations are mainly driven changes in grades, mine schedule, and processing methods. The AISC metric can range from $309/oz to $992/oz Au in a given year (excluding final year spike in Year 45 of $1,956/oz) but can be subdivided into three distinct phases:

- Phase 1: Years 1 and 2 (CIP only) - 133 koz/yr Au at $853/oz.
- Phase 2: Years 3 through 18 (mining highest grades) - 1,191 koz/yr Au at $411/oz.
- Phase 3: Years 19 through 45 EoM (mining lower grades) - 665 koz/yr Au at $554/oz.

FIGURE 22-4

ANNUAL AISC CURVE PROFILE



SENSITIVITY ANALYSIS

Project risks can be identified in both economic and non-economic terms. Key economic risks were examined by running cash flow sensitivities:

- Head grade
- Gold recovery
- Gold price
- Operating costs
- Capital costs
- Discount rates

Pre-tax NPV and IRR sensitivities over the base case has been calculated for -20% to +20% variations metal-related categories. For operating costs and capital costs, the sensitivities over the base case has been calculated at -15% to +35% variation. The sensitivities are shown in Table 22-5 and in Figures 22-5 and 22-6, respectively.

TABLE 22-5 PRE-TAX SENSITIVITY ANALYSIS

GR Engineering (Barbados), Inc. – Siembra Minera Project

Factor Change	Head Grade (g/t Au)	NPV at 10% (US$ M)	IRR (%)
0.8	0.56	3,477.3	28.3%
0.9	0.63	4,505.8	32.7%
1	**0.70**	**5,534.5**	**36.8%**
1.1	0.78	6,563.2	40.6%
1.2	0.85	7,591.9	44.3%

Factor Change	Recovery (% Au)	NPV at 10% (US$ M)	IRR (%)
0.8	67	3,477.3	28.3%
0.9	76	4,505.8	32.7%
1	**84**	**5,534.5**	36.8%
1.1	92	6,563.2	40.6%
1.2	100	7,489.0	44.0%

Factor Change	Metal Price (US$/oz Au)	NPV at 10% (US$ M)	IRR (%)
0.8	1,040	3,166.4	27.2%
0.9	1,170	4,350.4	32.2%
1	**1,300**	**5,534.5**	**36.8%**
1.1	1,430	6,718.5	41.1%
1.2	1,560	7,902.5	45.1%

Factor Change		Operating Costs (US$/t milled)	NPV at 10% (US$ M)	IRR (%)
0.85	$	11.57	6,068.2	38.6%
0.93	$	12.27	5,801.3	37.7%
1.00	**$**	**12.96**	**5,534.5**	**36.8%**
1.18	$	14.59	4,911.7	34.6%
1.35	$	16.21	4,289.0	32.3%

Factor Change		Capital Costs (US$ M)	NPV at 10% (US$ M)	IRR (%)
0.85	$	4,222	5,812.0	41.1%
0.93	$	4,385	5,673.2	38.8%
1.00	**$**	**4,547**	**5,534.5**	**36.8%**
1.18	$	4,927	5,210.7	32.7%
1.35	$	5,306	4,886.9	29.3%

FIGURE 22-5 PRE-TAX NPV 10% SENSITIVITY ANALYSIS

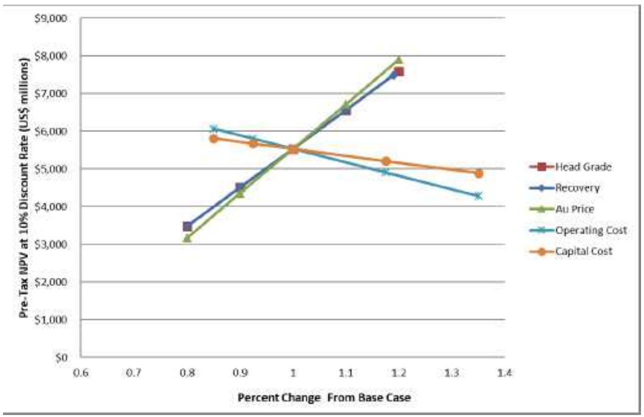
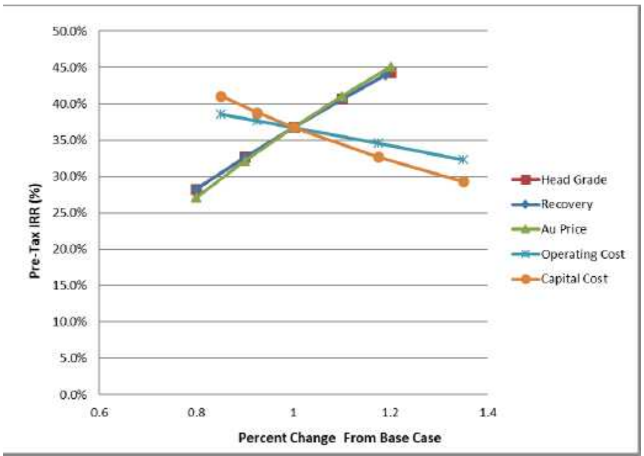


FIGURE 22-6 PRE-TAX IRR SENSITIVITY ANALYSIS



A sensitivity analysis of discount rates is presented in Figure 22-7 and 22-8 and shows that the Project as currently designed would be NPV positive through a 20% discount rate.

FIGURE 22-7 PRE-TAX DISCOUNT RATE SENSITIVITY ANALYSIS

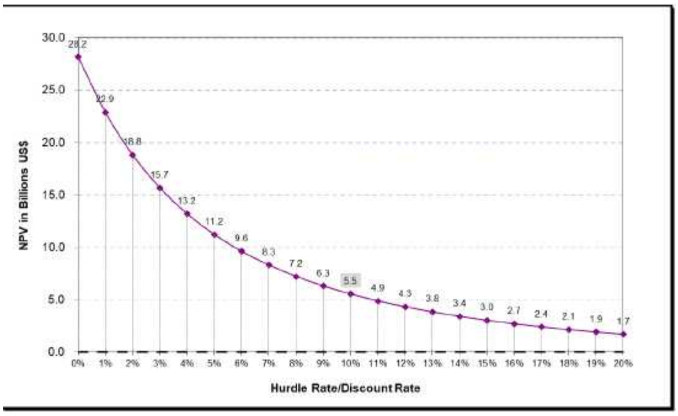
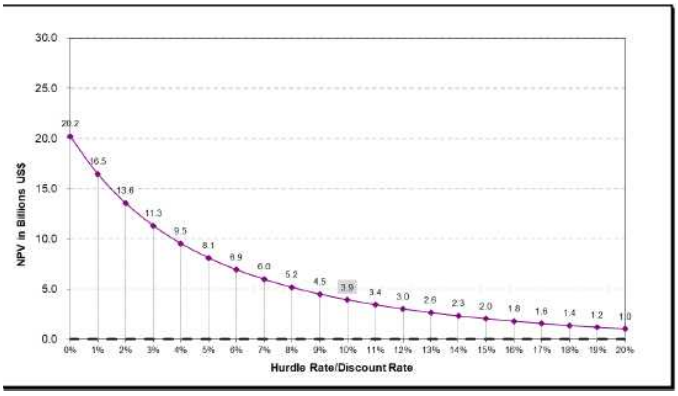


FIGURE 22-8 AFTER-TAX DISCOUNT RATE SENSITIVITY ANALYSIS



23 ADJACENT PROPERTIES

There are no adjacent properties to report in this section.

24 OTHER RELEVANT DATA AND INFORMATION

No additional information or explanation is necessary to make this Technical Report understandable and not misleading.

25 INTERPRETATION AND CONCLUSIONS

RPA offers the following conclusions by area.

GEOLOGY AND MINERAL RESOURCES

- A number of exploration programs completed by Placer and GRI were successful in locating and defining the extents of the various mineralized zones on each of their respective property holdings. The recently established Siembra Minera Economic Zone has unified the land tenure.
- The geology of the deposit is well understood in general. RPA is of the opinion that the distribution of high grade areas in the Main Zone should be studied in more detail.
  In the southern two-thirds of the Cristinas concessions and the entirety of the Brisas concessions, the mineralization occurs in a large tabular body, which strikes approximately north-south and dips moderately to the west. In the northern third of the Cristinas concessions, the mineralization can occur as pipe-shaped forms, and as thinner tabular forms with sub-vertical dips and strikes to the southeast.
- The large tabular, strataform mineralized zone (the Main Zone) forms most of the Mineral Resource. The Main Zone has a minimum thickness of 10 m at the south end and reaches a maximum thickness of 350 m. The average thickness is approximately 200 m. While the southern limits of the Main Zone have been outlined by the existing drilling pattern with a reasonable degree of confidence, the down-dip limits have not been defined by drilling. The northern limits of the Main Zone are also reasonably well defined by the existing drilling pattern.
- The drill hole information collected by Placer and GRI was merged into one master database that was then used to prepare the Mineral Resource estimate. Additional drill hole information collected by Crystallex on the Cristinas concessions could not be used to prepare the current estimate of the Mineral Resources, as the detailed information required was not available. The drill hole data from Placer contained drilling information and analytical results up to 1997 while the drill hole data from GRI included information up to 2006.
- In RPA's opinion, the drill hole data is adequate for use in the preparation of Mineral Resource estimates.
- The outline of the gold mineralization was created by drawing wireframes using approximately a 0.20 g/t Au cut-off grade and the copper mineralization was outlined using broad wireframes based on approximately a 0.04% Cu cut-off grade. A total of 24 wireframes were constructed to represent the gold mineralization zones and six wireframes to represent the copper mineralization zones. RPA also prepared wireframe surfaces to represent the three main weathering profiles for the mineralized zones: oxide saprolite, sulphide saprolite, and hard rock.
- RPA applied variable capping values for gold and copper grades for each of the mineralized wireframe domains. The capped assay values were composited into three

metre lengths. The composites were then used to estimate the gold and copper grades into a grade-block model that used block sizes of 10 m by 10 m by 6 m. Gold and copper grades were estimated into blocks using inverse distance squared and dynamic anisotropy with the Surpac v.6.8 software package. The estimated gold and copper grades were used to calculate NSR values for each mineralized block.

- Mineral Resources were prepared using an NSR cut-off value of US$7.20/t for the oxide saprolite and US$5.00/t for the sulphide saprolite and fresh rock. An open pit shell was created using the Whittle software package to constrain reporting of the Mineral Resources.
- The Mineral Resource estimate conforms to CIM (2014).
- The Mineral Resources are estimated at 10 million tonnes at an average grade of 1.02 g/t Au and 0.18% Cu containing 318,000 ounces of gold and 17,000 tonnes of copper in the Measured category, 1.17 billion tonnes at an average grade of 0.70 g/t Au and 0.10% Cu containing 26.5 million ounces of gold and 1.2 million tonnes of copper in the Indicated category. Mineral Resources in the Inferred category are estimated at 1.30 billion tonnes at an average grade of 0.61 g/t Au and 0.08% Cu containing 25.4 million ounces of gold and 1.0 million tonnes of copper.

MINING

- Mine production is scheduled to be carried out at a maximum mining rate ranging from 330 ktpd to 380 ktpd of total material.
- Stripping ratios are expected to average 1.16 over the LoM plan
- A separate equipment fleet of smaller excavators and articulated dump trucks is included in the mining capital for saprolite mining in the first 10 years. Typically, undisturbed saprolite material can be difficult to mine as the moisture creates operation problems. As the Project area has essentially been disturbed, RPA has assumed most saprolite is handled by the larger equipment fleet. The larger mine fleet is more productive and prior experience at Cristinas shows that rigid frame trucks can operate in the saprolite.
- Stockpiles are required for blending the process feed to achieve sufficient copper grades in flotation to produce a copper concentrate above 20%. Stockpiles fluctuate year to year, but achieve maximum capacity of just over 70 million tonnes

MINERAL PROCESSING

- Both Brisas and Cristinas were developed to the feasibility-level stage and beyond in 2006 to 2007 so the quantity of information available is greater than would typically be available at the PEA stage of a project.
- The material to be mined from Siembra Minera is demonstrated to be amenable to both cyanide leaching and to sulphide flotation. For materials that contain lower concentrations of copper, cyanide leaching is more cost effective and for material that contains higher concentrations of copper, sulphide flotation is more cost effective.

- The prior metallurgical test work met industry standards at the time the studies were completed, however, technology has progressed in the subsequent ten plus years and some of the industry standards have evolved. Current standards include testing of a large number of variability samples and development of geometallurgical models, as opposed to testing composite samples to represent “average” material to be processed, which was the emphasis for the Brisas test program.

ENVIRONMENT

- GRE is in the process of preparing environmental reports and programs to meet municipal, provincial, and national regulatory requirements, as well as generally accepted international standards.
- Two separate but parallel ESIAs are being prepared for the Project, one that meets Venezuelan regulatory requirements and one that meets international standards and guidelines.
- A conceptual plan for small-scale mining management is in place. The conceptual plan includes relocation of the artisanal miners away from the active, large scale mining operations and establishment of an oxide saprolite processing and stockpile area with concrete tailings ponds that collect and transport tailings from the artisanal mining operations to the Project TMF.

26 RECOMMENDATIONS

Given the positive economic results presented in this report, RPA recommends that the Project be advanced to the next stage of engineering study and permitting.

RPA offers the following recommendations.

GEOLOGY AND MINERAL RESOURCES

- Acquire new topographic data.
- Drill approximately 150 to 200 drill holes totaling approximately 75 km to 100 km. This drilling would have a number of objectives including:
  - Conversion of Inferred Mineral Resources to Indicated with priority set on Inferred Mineral Resources situated in the 5 and 10 year pit shells.
  - Drilling to determine the extent of mineralization at depth in the Main Zone as this will determine the limits of the largest possible pit and help with the location of features such as dumps and roads.
  - Better definition of the copper mineralization in the Main Zone footwall.
  - Improving preliminary artisanal mining sterilization assumptions.
  - Condemnation drilling of proposed waste rock storage sites.
  - Closer spaced drilling in the El Potaso area between Brisas and Cristinas.
  - Drilling on the northwest extensions of the mineralization in the Morrocoy and Cordova areas.
  - Drilling on the Cristinas Main Zone for density measurements.
- Improve understanding of the geological and structural controls on the shapes and local trends of high grade lenses in the Main Zone. Northwest striking cross-faults need to be identified and modelled and structural sub-domains built to improve future variography studies and dynamic anisotropy trend surfaces. This will improve the local accuracy of future gold and copper grade models.
- Carry out additional 3D mineralization trend analysis studies, domain modelling, and variography work should be carried out for the gold and copper mineralization. This will also assist in evaluating if additional 5-spot drill holes are needed to support the Indicated classification in some areas with more complex geology.
- Depending on the outcome of new variography work, build gold and copper models using ordinary kriging.
- Develop a new lithology model once new drill holes have been drilled so that an improved material densities model can be created.
- Build a structural model.

- For the proposed drilling, implement field and coarse duplicate sampling programs at Siembra Minera at a rate of 1 in 50.
- Acquire three or four matrix matched CRMs that approximate the cut-off grade, average grade and high-grade material and insert them in all future drill programs at the Project at a rate of approximately 1 in 25.
- Implement external laboratory check assays at a rate of approximately 1 in 10.

MINING

- RPA is of the opinion that one of the most important factors influencing mining will be the amount of water entering the pit. RPA recommends contracting a groundwater hydrologist to evaluate the combined Project based on past work.
- A LoM schedule should be generated for the mining and processing of the Siembra Minera mineralized material. This study should include optimization and blending of the materials to achieve a sufficiently high copper grade to produce a copper concentrate grade above 20%.
- A trade-off study should be completed for the backfilling of the open pit with waste rock and/or neutralized tailings.
- A geotechnical investigation program should be carried out to confirm the subsurface conditions under the proposed new open pit, waste dump locations, and stability analysis undertaken to verify design recommendations.

MINERAL PROCESSING

- Every effort should be made to acquire access to the detailed metallurgical and plant data for Cristinas. In the absence of that data, detailed metallurgical sampling and testing are required to provide the information required to design the oxide leaching plant.
- Additional test work should be conducted for the flotation plant using variability samples taken from throughout the deposits with particular emphasis on Cristinas where limited variability testing was done using the flotation flowsheet. Currently, industry standard emphasizes the use of variability samples as opposed to the composite samples that were predominantly used in previous flotation testing.
- RPA is of the opinion that there is considerable potential for optimization of the flowsheet of the Siembra Minera Project. Examples include:
    - Increased efficiency if larger equipment sizes are utilized in the design. Due to cost savings and enhanced performance, the sizes for grinding mills and flotation cells have increased substantially. As examples, semi-autogenous grinding (SAG) mills that are now available are as large as 12.2 m diameter by 8.8 m long as opposed to the 11.6 m by 6.7 m that are in the current design and flotation cells now have capacities of 600 m^3 instead of the 160 m^3 that are in the current design. The larger pieces of equipment result in a reduced footprint and fewer pieces of equipment and, therefore, lower installed costs.

- The use of an adsorption desorption recovery (ADR) that is designed for the combined Project will probably result in less cost than merely doubling the size of the current design. In addition to this, consolidating the ADR from the oxide leach plant into a plant that can later be expanded to process the doré from the flotation plant has the potential to not only cut costs but also reduce security concerns and efforts.

- RPA is of the opinion that the current conceptual design for the oxide leach plant does not include the best options for Siembra Minera. Areas that require detailed evaluations include:
  - Use of CIL instead of CIP particularly since the plant designs for both Cristinas and Brisas were changed to CIL from CIP during previous studies.
  - Investigate elimination of the copper circuits. Data from the Cristinas feasibility study shows that copper is only soluble in the sulphide saprolite and that it is not soluble in material that has lower copper concentrations. Therefore, the copper circuit should not be needed as the sulphide saprolite that contains higher concentrations of copper will be processed in the flotation plant and not in the oxide leach plant.
  - Changes to the gravity separation circuit. The use of continuous centrifugal concentrators instead of batch units to eliminate manual labour and reduce potential for theft. Use intensive cyanide leaching to process the gravity gold concentrate instead of shaking tables. Prior studies showed that intensive cyanide leaching was preferable for treatment of the gravity concentrate for both Brisas and Cristinas.
  - Selection of designs that are appropriate for processing clay-like saprolitic material, including:
    - Appropriate tank sizing using slurry densities that are consistent with the material that has a low specific gravity and is viscous in nature
    - Proper agitator selection
    - Selection of pumps and design of piping
- Design of the TMF for the combined Project is preliminary. Further detailed geotechnical work is required to complete a design for the final tailings. Preliminary plans are to use the feasibility level design from the SNC-Lavalin 2007 study as Stage 1 of construction with the final tailings inundating the Stage 1 structure.

ENVIRONMENT

- GRI has held discussions with the small miners, indigenous groups, and local people. RPA recommends continuing discussions with these groups.
- Due to the increase in mineral resources, additional work is required for the increased WRD and TMF, and redesign/update of the ARD mitigation measures.
- A new ESIA will be required for the combined project with an updated project plan and in conjunction with detail design and feasibility study.

COSTS AND ECONOMICS

- After the designs are complete for the Siembra Minera Project, a new capital and operating cost estimate should be completed.

- An updated copper concentrate marketing study should be completed. Recent changes in the world copper concentrate supply have reduced treatment and refining charges for copper and reduced participation charges.

PROPOSED PROGRAM AND BUDGET

RPA's proposed program for the next stage of study is summarized in Table 26-1.

TABLE 26-1 PROPOSED PROGRAM

GR Engineering (Barbados), Inc. – Siembra Minera Project

Description	Cost (US$ M)
Drilling to upgrade Inferred Mineral Resources – 150 to 200 holes	20
Geotechnical Studies	2
Hydrogeology Study	1
Metallurgical Studies	2
Pre-feasibility/Feasibility Study	5
ESIA and Permitting	2
Total	**32**

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28 DATE AND SIGNATURE PAGE

This report titled "Technical Report on the Siembra Minera Project, Bolivar State, Venezuela" and dated March 16, 2018 was prepared and signed by the following authors:

	(Signed and Sealed) "*Richard J. Lambert*"
Dated at Lakewood, CO	Richard J. Lambert, P.E., P.Eng. Principal Mining Engineer
March 16, 2018	**(Signed and Sealed) "*José Texidor Carlsson*"**
Dated at Toronto, ON	José Texidor Carlsson, P.Geo. Senior Geologist
March 16, 2018	**(Signed and Sealed) "*Hugo Miranda*"**
Dated at Lakewood, CO	Hugo Miranda, ChCM (RM) Principal Mining Engineer
March 16, 2018	**(Signed and Sealed) "*Kathleen Ann Altman*"**
Dated at Lakewood, CO	Kathleen Ann Altman, P.E., Ph.D. Principal Metallurgist
March 16, 2018	**(Signed and Sealed) "*Grant Malensek*"**
Dated at Lakewood, CO	Grant Malensek, P.Geo., P.Eng. Principal Valuation Engineer
March 16, 2018	

29 CERTIFICATE OF QUALIFIED PERSON

RICHARD J. LAMBERT

I, Richard J. Lambert, P.Eng., as an author of this report titled "Technical Report on the Siembra Minera Project, Bolivar State, Venezuela" prepared for GR Engineering (Barbados), and dated March 16, 2018, do hereby certify that:

1. I am Principal Mining Consultant with Roscoe Postle Associates Inc. of Suite 505, 143
   - Boulevard, Lakewood, CO, USA 80227.
2. I am a graduate of Mackay School of Mines, University of Nevada, Reno, U.S.A., with
   - Bachelor of Science degree in Mining Engineering in 1980, and Boise State
   - with a Masters of Business Administration degree in 1995.
3. I am a Registered Professional Engineer in the state of Wyoming (#4857) and the state
   - Montana (#11475). I am licensed as a Professional Engineer in the Province of
   - (Reg. #100139998). I have been a member of the Society for Mining,
   - and Exploration (SME) since 1975, and a Registered Member
   - since May 2006. I have worked as a mining engineer for a total of 37
   - since my graduation. My relevant experience for the purpose of the Technical
   - is:
   - Review and report as a consultant on numerous mining projects for due diligence and regulatory requirements
   - Mine engineering, mine management, mine operations and mine financial analyses, involving copper, gold, silver, nickel, cobalt, uranium, oil shale, phosphates, coal and base metals located in the United States, Canada, Zambia, Madagascar, Turkey, Bolivia, Chile, Brazil, Serbia, Australia, Russia and Venezuela.
4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI
   - and certify that by reason of my education, affiliation with a professional
   - (as defined in NI 43-101) and past relevant work experience, I fulfill the
   - to be a "qualified person" for the purposes of NI 43-101.
5. I visited the Brisas Project site in February 2008. During the visit I observed the
   - pit, process plant, mine shop, tailings facility and waste dump areas. I
   - the drill core.
6. I am responsible for the preparation of Sections 15, 16, 19 and 20 and collaborated
   - my co-authors on Sections 1, 2, 3, 18, 21, 24, 25, 26, and 27 of the Technical
   -
7. I am independent of the Issuer applying the test set out in Section 1.5 of NI 43-101.
8. I prepared a previous Technical Report on the Brisas Project dated March 31, 2008.
9. I have read NI 43-101, and the Technical Report has been prepared in compliance with
   - 43-101 and Form 43-101F1.

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

Dated this 16th day of March, 2018

(Signed and Sealed) "*Richard J. Lambert*"

Richard J. Lambert, P.Eng.

JOSÉ TEXIDOR CARLSSON

I, José Texidor Carlsson, P.Geo., as an author of this report entitled "Technical Report on the Siembra Minera Project, Bolivar State, Venezuela" prepared for GR Engineering (Barbados), Inc., and dated March 16, 2018, do hereby certify that:

1. I am a Senior Geologist with Roscoe Postle Associates Inc. of Suite 501, 55 University Ave
   - ON, M5J 2H7.
2. I am a graduate of University of Surrey, United Kingdom, in 1998 with a Master of
   - Electronic and Electrical degree and Acadia University, Nova Scotia, in 2007
   - an M.Sc. degree in Geology.
3. I am registered as a Professional Geologist in the Province of Ontario (Reg. #2143). I have
   - as a geologist for a total of 10 years since my graduation. My relevant experience
   - the purpose of the Technical Report is:
   - Mineral Resource estimation and NI 43-101 reporting
   - Supervision of exploration properties and active mines in Canada, Mexico, and South America
   - Experienced user of geological and resource modelling software
4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-
   - and certify that by reason of my education, affiliation with a professional association
   - defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be
   - "qualified person" for the purposes of NI 43-101.
5. I did not visit the Siembra Minera Project.
6. I am responsible for Sections 4 to 12 and 14 and share responsibility for Sections 1, 2, 23,
   - 25, 26, and 27 of the Technical Report.
7. I am independent of the Issuer applying the test set out in Section 1.5 of NI 43-101.
8. I have had no prior involvement with the property that is the subject of the Technical Report.
9. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI
   - and Form 43-101F1.
10. At the effective date of the Technical Report, to the best of my knowledge, information, and
    - the Technical Report contains all scientific and technical information that is required
    - be disclosed to make the Technical Report not misleading.

Dated this 16th day of March, 2018

(Signed and Sealed) "*José Texidor Carlsson*"

José Texidor Carlsson, M.Sc., P.Geo.

www.rpacan.com

HUGO M. MIRANDA

I, Hugo M. Miranda, ChCM (RM), as an author of this report entitled "Technical Report on the Siembra Minera Project, Bolivar State, Venezuela" prepared for GR Engineering (Barbados), Inc., and dated March 16, 2018, do hereby certify that:

1. I am a Principal Mining Engineer with RPA (USA) Ltd. of 143 Union Boulevard, Suite 505,
   - Colorado, USA 80228.
2. I am a graduate of the Santiago University of Chile, with a B.Sc. degree in Mining
   - in 1993, and a Masters of Business Administration degree in 2004. I'm also a
   - of the Colorado School of Mines with a Master of Engineering (Engineer of Mines)
   - in 2015.
3. I am registered as a Competent Person of the Chilean Mining Commission (Registered
   - #0031). I am a Registered Member (#4149165) with the Society for Mining,
   - and Exploration (SME). I have worked as a mining engineer for a total of 23
   - since my graduation. My relevant experience for the purpose of the Technical Report
   -
   - Principal Mining Engineer - RPA in Colorado. Review and report as a consultant on mining operations and mining projects. Mine engineering including mine plan and pit optimization, pit design and economic evaluation.
   - Principal Mining Consultant – Pincock, Allen and Holt in Colorado, USA. Review and report as a consultant on numerous development and production mining projects.
   - Mine Planning Chief, El Tesoro Open Pit Mine - Antofagasta Minerals in Chile.
   - Open Pit Planning Engineer, Radomiro Tomic Mine, CODELCO – Chile.
   - Open Pit Planning Engineer, Andina Mine, CODELCO - Chile.
4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-
   - and certify that by reason of my education, affiliation with a professional association
   - defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be
   - "qualified person" for the purposes of NI 43-101.
5. I visited the Project on September 19, 2017.
6. I am responsible for parts of Section 16 and share responsibility with my co-authors for
   - 1, 2, 3, 24, 25, and 26 of the Technical Report.
7. I am independent of the Issuer applying the test set out in Section 1.5 of NI 43-101.
8. I have had no prior involvement with the property that is the subject of the Technical Report.
9. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI
   - and Form 43-101F1.

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

Dated this 16th day of March, 2018

(Signed and Sealed) "*Hugo Miranda*"

Hugo M. Miranda, C.P.

KATHLEEN ANN ALTMAN

I, Kathleen Ann Altman, P.E., as an author of this report titled “Technical Report on the Siembra Minera Project, Bolivar State, Venezuela” prepared for GR Engineering (Barbados), Inc., and dated March 16, 2018, do hereby certify that:

1. I am Principal Metallurgist with RPA (USA) Ltd. of Suite 505, 143 Union Boulevard,
   - Co., USA 80228.
2. I am a graduate of the Colorado School of Mines in 1980 with a B.S. in Metallurgical
   - I am a graduate of the University of Nevada, Reno Mackay School of Mines
   - an M.S. in Metallurgical Engineering in 1994 and a Ph.D. in Metallurgical Engineering
   - 1999.
3. I am registered as a Professional Engineer in the State of Colorado (Reg. #37556) and a
   - Professional Member of the Mining and Metallurgical Society of America
   - #01321QP). I have worked as a metallurgical engineer for a total of 37 years
   - my graduation. My relevant experience for the purpose of the Technical Report is:
   - Review and report as a metallurgical consultant on numerous mining operations and projects around the world for due diligence and regulatory requirements.
   - I have worked for operating companies, including the Climax Molybdenum Company, Barrick Goldstrike, and FMC Gold in a series of positions of increasing responsibility.
   - I have worked as a consulting engineer on mining projects for approximately 15 years in roles such a process engineer, process manager, project engineer, area manager, study manager, and project manager. Projects have included scoping, prefeasibility and feasibility studies, basic engineering, detailed engineering and start-up and commissioning of new projects.
   - I was the Newmont Professor for Extractive Mineral Process Engineering in the Mining Engineering Department of the Mackay School of Earth Sciences and Engineering at the University of Nevada, Reno from 2005 to 2009.
4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-
   - and certify that by reason of my education, affiliation with a professional association
   - defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be
   - "qualified person" for the purposes of NI 43-101.
5. I did not visit the Siembra Minera Project.
6. I am responsible for Sections 13 and 17 and share responsibility for Sections 1, 18, 20, 21,
   - 25, 26, and 27 of the Technical Report.
7. I am independent of the Issuer applying the test set out in Section 1.5 of NI 43-101.
8. I have had no prior involvement with the property that is the subject of the Technical Report.
9. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI
   - and Form 43-101F1.

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

Dated this 16th day of March, 2018

(Signed and Sealed) "*Kathleen Ann Altman*"

Kathleen Ann Altman, P.E.

GRANT A. MALENSEK

I, Grant A. Malensek, P.Eng., P.Geo., as an author of this report entitled "Technical Report on the Siembra Minera Project, Bolivar State, Venezuela" prepared for GR Engineering (Barbados), and dated March 16, 2018, do hereby certify that:

1. I am Principal Engineer - Valuations with Roscoe Postle Associates Inc. of Suite 505, 143
    - Boulevard, Lakewood, CO, USA 80227.
2. I am a graduate of University of British Columbia, Vancouver Canada in 1987 with a
    - degree in Geological Sciences. In addition, I have obtained a Master of
    - in Geological Engineering from the Colorado School of Mines in 1997 and a
    - Business Certificate in Finance from the University of Denver – Daniels College
    - Business in 2011.
3. I am registered as a Professional Engineer/Geologist in the Province of British Columbia
    - 23905). I have worked as a mining engineer/geologist for a total of 22 years
    - my graduation. My relevant experience for the purpose of the Technical Report is:
    - Numerous mining project technical-economic modeling assignments.
    - Review and report as a consultant on numerous mining projects for due diligence and regulatory requirements
    - I have worked for operating entities, including Rio Tinto Group, Freeport McMoRan Copper and Gold Inc., and Newmont Mining Company on a variety of exploration and advanced development projects as well as operations in a number of countries.
4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-
    - and certify that by reason of my education, affiliation with a professional association
    - defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be
    - "qualified person" for the purposes of NI 43-101.
5. I did not visit the Siembra Minera Project.
6. I am responsible for Sections 19 and 22 and collaborated with my co-authors on Sections
    - and 21 of the Technical Report.
7. I am independent of the Issuer applying the test set out in Section 1.5 of NI 43-101.
8. I have had no prior involvement with the property that is the subject of the Technical Report.
9. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI
    - and Form 43-101F1.
10. At the effective date of the Technical Report, to the best of my knowledge, information, and
    - the Technical Report sections for which I am responsible contains all scientific and
    - information that is required to be disclosed to make the Technical Report not
    -

Dated 16th day of March, 2018

(Signed and Sealed) "*Grant Malensek*"

Grant A. Malensek, P.Eng., P.Geo.

30 APPENDIX 1

CASH FLOW PROJECTION

Economic Model Annual Summary

	Company	GR Engineering (Barbados)													
	Project Name	Brisas/Cristinas													
	Scenario Name	15CIP_140Flot_V303 BM52													
	Analysis Type	PEA					<== 15kt/d CIP Plant		<==Grid Power <== 140kt/d Flot Plant						
Project Timeline in Years					1	2	3	4	5	6	7	8	9	10	11
Commercial Production Timeline in Years					-2	-1	1	2	3	4	5	6	7	8	9
Time Until Closure In Years		US$ & Metric Units LoM Avg / Total			47	46	45	44	43	42	41	40	39	38	37
Market Prices															
Gold		US$/oz	$	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300
Silver		US$/oz	$	17.00	17.00	17.00	17.00	17.00	17.00	17.00	17.00	17.00	17.00	17.00	17.00
Copper		US$/lb	$	3.00	3.00	3.00	3.00	3.00	3.00	3.00	3.00	3.00	3.00	3.00	3.00
Physicals															
Total Mill Feed Mined			kt	2,004,741	-	-	15,718	30,223	62,406	75,083	73,267	50,218	49,950	45,994	73,610
Total Waste Mined			kt	2,320,350	-	25,000	8,282	9,777	45,607	44,917	46,733	69,782	70,050	74,006	46,390
Total Material Mined			kt	4,325,091	-	25,000	24,000	40,000	108,014	120,000	120,000	120,000	120,000	120,000	120,000
Strip Ratio		W:O		1.16	-	-	0.53	0.32	0.73	0.60	0.64	1.39	1.40	1.61	0.63
CIP Plant Feed Processed			kt	302,195	-	-	5,162	5,800	5,800	5,800	5,800	5,800	5,800	5,800	5,800
Flotation Plant Feed Processed			kt	1,702,545	-	-	-	-	40,890	58,000	58,000	58,000	58,000	58,000	58,000
Total Mill Feed Processed			kt	2,004,741	-	-	5,162	5,800	46,690	63,800	63,800	63,800	63,800	63,800	63,800
Gold Grade, Processed		g/t		0.70	-	-	0.63	0.89	1.06	1.01	0.91	0.81	0.74	0.67	0.69
Silver Grade, Processed		g/t		0.50	-	-	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50
Copper Grade, Processed			%	0.090	-	-	-	-	0.081	0.075	0.093	0.087	0.085	0.071	0.083
Contained Gold, Processed			koz	45,420	-	-	104	167	1,588	2,081	1,861	1,662	1,516	1,367	1,409
Contained Silver, Processed			koz	32,227	-	-	83	93	751	1,026	1,026	1,026	1,026	1,026	1,026
Contained Copper, Processed		klb		3,974,514	-	-	-	-	83,328	105,031	130,431	122,530	118,862	100,330	117,105
Average Recovery, Gold			%	83.9 % -	- -	-	98.0 %	98.0 %	84.4 %	84.1 %	84.1 %	84.2 %	84.0 %	83.5 %	83.5 %
Average Recovery, Silver			%	53.0 % -	- -	-	30.0 %	30.0 %	53.7 %	54.6 %	54.6 %	54.6 %	54.6 %	54.6 %	54.6 %
Average Recovery, Copper			%	84.0 % -	- -	- -	- -	-	67.0 %	71.4 %	77.9 %	77.8 %	70.9 %	76.2 %	86.1 %
Recovered Gold			koz	38,127	-	-	102	163	1,340	1,749	1,565	1,400	1,275	1,141	1,176
Recovered Silver			koz	17,085	-	-	25	28	403	560	560	560	560	560	560
Recovered Copper		klb		3,339,179	-	-	-	-	55,801	75,013	101,636	95,375	84,304	76,410	100,883
Payable Gold			koz	37,639	-	-	102.0	163.2	1,326.0	1,728.2	1,545.1	1,382.9	1,259.9	1,126.5	1,159.7
Payable Silver			koz	16,615	-	-	24.4	27.4	392.1	544.8	544.8	544.8	544.8	544.8	544.8
Payable Copper	1,450.44	klb		3,197,647	-	-	-	-	53,194.6	71,600.3	97,237.1	91,140.6	80,561.5	73,094.3	96,573.1
Cash Flow															
Gold Gross Revenue	83 %	$	000 s	48,930,808	-	-	132,632	212,184	1,723,844	2,246,631	2,008,655	1,797,816	1,637,806	1,464,455	1,507,610
Silver Gross Revenue	0.5 %	$	000 s	282,450	-	-	415	466	6,666	9,261	9,261	9,261	9,261	9,261	9,261
Copper Gross Revenue	16 %	$	000 s	9,592,942	-	-	-	-	159,584	214,801	291,711	273,422	241,684	219,283	289,719
Gross Revenue Before By-Product Credits	**100.0 %**	**$**	**000 s**	**58,806,200**	**-**	**-**	**133,047**	**212,650**	**1,890,095**	**2,470,693**	**2,309,628**	**2,080,499**	**1,888,751**	**1,692,999**	**1,806,590**
Gold Gross Revenue		$	000 s	48,930,808	-	-	132,632	212,184	1,723,844	2,246,631	2,008,655	1,797,816	1,637,806	1,464,455	1,507,610
Silver Gross Revenue		$	000 s	-	-	-	-	-	-	-	-	-	-	-	-
Copper Gross Revenue		$	000 s	-	-	-	-	-	-	-	-	-	-	-	-
Gross Revenue After By-Product Credits		**$**	**000 s**	**48,930,808**	**-**	**-**	**132,632**	**212,184**	**1,723,844**	**2,246,631**	**2,008,655**	**1,797,816**	**1,637,806**	**1,464,455**	**1,507,610**
Mining Cost		$	000 s	(5,790,854)	-	-	(48,750)	(62,176)	(123,127)	(128,247)	(135,569)	(148,550)	(148,461)	(152,813)	(147,992)
Process Cost		$	000 s	(9,880,955)	-	-	(27,395)	(28,807)	(214,379)	(263,401)	(267,770)	(265,501)	(270,122)	(297,197)	(304,788)
G&A Cost		$	000 s	(2,653,575)	-	-	(7,695)	(8,021)	(61,596)	(64,549)	(65,272)	(65,919)	(66,145)	(67,704)	(67,755)
Engineering & Geology Cost		$	000 s	(276,752)	-	-	(4,999)	(6,117)	(6,117)	(6,179)	(6,179)	(6,179)	(6,179)	(6,179)	(6,179)
ARD Plant Cost		$	000 s	(12,130)	-	-	(215)	(245)	(245)	(368)	(368)	(353)	(338)	(323)	(323)
Transportation Cost		$	000 s	(728,023)	-	-	(64)	(96)	(13,624)	(17,762)	(22,705)	(21,851)	(19,350)	(17,108)	(22,130)
Offsite Treatment Cost		$	000 s	(1,076,520)	-	-	(623)	(991)	(22,483)	(29,140)	(34,874)	(33,008)	(29,486)	(26,087)	(32,552)
NSR Royalty		$	000 s	(3,262,770)	-	-	(6,618)	(10,578)	(92,699)	(121,190)	(112,602)	(101,282)	(91,996)	(82,490)	(87,595)
Special Advantages Tax		$	000 s	(1,710,050)	-	-	(3,971)	(6,347)	(55,620)	(72,714)	(67,561)	(60,769)	(55,197)	(49,494)	(52,557)
LOCTI (Science) Contributions		$	000 s	(588,062)	-	-	(1,330)	(2,127)	(18,901)	(24,707)	(23,096)	(20,805)	(18,888)	(16,930)	(18,066)
Subtotal Cash Costs Before By-Product Credits		$	000 s	(25,979,692)	-	-	(101,659)	(125,506)	(608,791)	(728,256)	(735,996)	(724,216)	(706,161)	(716,324)	(739,937)
By-Product Credits		$	000 s	9,875,392	-	-	415	466	166,250	224,062	300,972	282,683	250,945	228,544	298,980
Total Cash Costs After By-Product Credits		**$**	**000 s**	**(16,104,300)**	**-**	**-**	**(101,244)**	**(125,040)**	**(442,541)**	**(504,194)**	**(435,024)**	**(441,534)**	**(455,215)**	**(487,780)**	**(440,957)**
Operating Margin	**56 %**	**$**	**000 s**	**32,826,508**	**-**	**-**	**31,388**	**87,144**	**1,281,303**	**1,742,436**	**1,573,631**	**1,356,283**	**1,182,590**	**976,674**	**1,066,653**
EBITDA		**$**	**000 s**	**32,826,508**	**-**	**-**	**31,388**	**87,144**	**1,281,303**	**1,742,436**	**1,573,631**	**1,356,283**	**1,182,590**	**976,674**	**1,066,653**
Stockpile Adjustments		$	000 s	0	-	-	36,550	49,806	6,736	(8,726)	12,990	3,954	(17,847)	(26,892)	3,131
Capital Depreciation Allowance		$	000 s	(4,212,377)	-	-	(2,852)	(8,371)	(71,071)	(96,113)	(89,297)	(83,277)	(79,089)	(73,141)	(77,884)
Amortization Allowance		$	000 s	(266,315)	-	-	-	-	-	(750)	(1,256)	(10,638)	(13,889)	(12,225)	(10,784)
Reclamation Amortization		$	000 s	(150,000)	-	(867)	(832)	(1,387)	(3,746)	(4,162)	(4,162)	(4,162)	(4,162)	(4,162)	(4,162)
Loss Carry Forward Credit		$	000 s	(867)	-	-	(867)	-	-	-	-	-	-	-	-
Earnings Before Taxes		**$**	**000 s**	**28,196,950**	**-**	**(867)**	**63,386**	**127,192**	**1,213,223**	**1,632,685**	**1,491,905**	**1,262,161**	**1,067,604**	**860,255**	**976,955**
Anti-Drug Contributions		$	000 s	(283,851)	-	-	(277)	(774)	(12,065)	(16,414)	(14,789)	(12,582)	(10,855)	(8,871)	(9,738)
Sport Contributions		$	000 s	(283,851)	-	-	-	(277)	(774)	(12,065)	(16,414)	(14,789)	(12,582)	(10,855)	(8,871)
Corp. Income Tax @ Effective Rate of:	22.5 %	$	000 s	(7,373,821)	-	-	(8,874)	(17,807)	(169,851)	(228,576)	(208,867)	(239,811)	(202,845)	(163,448)	(185,621)
Net Income		**$**	**000 s**	**20,255,427**	**-**	**(867)**	**54,235**	**108,334**	**1,030,533**	**1,375,630**	**1,251,835**	**994,979**	**841,322**	**677,080**	**772,723**
Non-Cash Add Back - Stockpile Adjustments		$	000 s	(0)	-	-	(36,550)	(49,806)	(6,736)	8,726	(12,990)	(3,954)	17,847	26,892	(3,131)
Non-Cash Add Back - Depreciation		$	000 s	4,212,377	-	-	2,852	8,371	71,071	96,113	89,297	83,277	79,089	73,141	77,884
Non-Cash Add Back - Amortization		$	000 s	266,315	-	-	-	-	-	750	1,256	10,638	13,889	12,225	10,784
Non-Cash Add Back - Reclamation Amortization		$	000 s	150,000	-	867	832	1,387	3,746	4,162	4,162	4,162	4,162	4,162	4,162
Non-Cash Add Back - LCF Credit		$	000 s	867	-	-	867	-	-	-	-	-	-	-	-
Working Capital		$	000 s	0	-	-	2,260	(517)	(128,870)	(68,277)	2,610	27,520	24,907	3,979	(22,254)
Operating Cash Flow		**$**	**000 s**	**24,884,985**	**-**	**-**	**24,496**	**67,769**	**969,743**	**1,417,105**	**1,336,171**	**1,116,621**	**981,216**	**797,479**	**840,168**
Development Capital		$	000 s	(2,570,611)	(172,074)	(745,476)	(475,015)	(925,938)	(252,107)	-	-	-	-	-	-
Sustaining Capital		$	000 s	(1,941,696)	-	-	-	-	(18,567)	(30,278)	(43,987)	(50,357)	(26,443)	(43,391)	(40,934)
Closure/Reclamation Capital		$	000 s	(150,000)	-	-	-	-	-	-	-	-	-	-	-
Total Capital		**$**	**000 s**	**(4,662,307)**	**(172,074)**	**(745,476)**	**(475,015)**	**(925,938)**	**(270,674)**	**(30,278)**	**(43,987)**	**(50,357)**	**(26,443)**	**(43,391)**	**(40,934)**

Cash Flow Adj./Reimbursements	**$**	**000 s**		**-**	-	-	-	-	-	-	-	-	-	-	-

LoM Metrics

Economic Metrics															
Discount Factors	EOP @	10 %			1.0000	0.9091	0.8264	0.7513	0.6830	0.6209	0.5645	0.5132	0.4665	0.4241	0.3855
a) Pre-Tax															
Free Cash Flow	$	000 s		**28,164,202**	(172,074)	(745,476)	(441,368)	(839,311)	881,759	1,643,882	1,532,255	1,333,446	1,181,055	937,263	1,003,465
Cumulative Free Cash Flow	$	000 s			(172,074)	(917,551)	(1,358,918)	(2,198,229)	(1,316,470)	327,412	1,859,667	3,193,112	4,374,167	5,311,430	6,314,895
NPV @ 10%	$	000 s		**5,534,458**	(172,074)	(677,706)	(364,767)	(630,587)	602,253	1,020,721	864,918	684,268	550,971	397,491	386,879
Cumulative NPV	$	000 s			(172,074)	(849,780)	(1,214,547)	(1,845,133)	(1,242,880)	(222,159)	642,759	1,327,028	1,877,998	2,275,489	2,662,369
IRR		%		**36.8 %**											
Undiscounted Payback From Start of Comm. Prod.		Years		**3.8**	-	-	-	-	-	3.8	3.8	3.8	3.8	3.8	3.8
PI @ 10%		NPV / (PW of TC)		**2.26**	172,074	677,706	392,574	695,671	184,874	18,800	24,829	25,841	12,336	18,402	15,782
b) After-Tax															
Free Cash Flow	$	000 s		**20,222,678**	(172,074)	(745,476)	(450,519)	(858,169)	699,069	1,386,827	1,292,185	1,066,264	954,774	754,088	799,234
Cumulative Free Cash Flow	$	000 s			(172,074)	(917,551)	(1,368,069)	(2,226,238)	(1,527,169)	(140,342)	1,151,843	2,218,107	3,172,880	3,926,969	4,726,203
NPV @ 10%	$	000 s		**3,930,067**	(172,074)	(677,706)	(372,330)	(644,755)	477,474	861,110	729,405	547,162	445,409	319,807	308,139
Cumulative NPV	$	000 s			(172,074)	(849,780)	(1,222,110)	(1,866,864)	(1,389,391)	(528,280)	201,124	748,286	1,193,695	1,513,502	1,821,642
IRR		%		**31.1 %**											
Undiscounted Payback from Start of Comm. Prod.		Years		**4.1**	-	-	-	-	-	-	4.1	4.1	4.1	4.1	4.1
PI @ 10%		NPV / (PW of TC)		**1.61**	172,074	677,706	392,574	695,671	184,874	18,800	24,829	25,841	12,336	18,402	15,782
Operating Metrics															
Mine Life		Years		**45**											
Maximum Daily Mining Rate		t/d mined		**400,000**	-	71,429	68,571	114,286	308,611	342,857	342,857	342,857	342,857	342,857.14	342,857
Maximum Daily Processing Rate - CIP		t/d milled		**35,000**	-	-	14,749	16,571	16,571	16,571	16,571	16,571	16,571	16,571	16,571
Maximum Daily Processing Rate - Concentrator		t/d milled		**166,000**	-	-	-	-	116,829	165,714	165,714	165,714	165,714	165,714	165,714
Maximum Daily Processing Rate - Combined		t/d milled		**182,000**	-	-	14,749	16,571	133,400	182,286	182,286	182,286	182,286	182,286	182,286
Mining Cost		$ / t mined	**$**	**1.35**	-	-	2.03	1.55	1.14	1.07	1.13	1.24	1.24	1.27	1.23
Mining Cost		$ / t milled	**$**	**2.89**	-	-	3.10	2.06	1.97	1.71	1.85	2.96	2.97	3.32	2.01
Processing Cost		$ / t milled	**$**	**4.93**	-	-	5.31	4.97	4.59	4.13	4.20	4.16	4.23	4.66	4.78
G&A Cost	17 %	$ / t milled	**$**	**1.32**	-	-	1.49	1.38	1.32	1.01	1.02	1.03	1.04	1.06	1.06
Other Infrastructure Cost		$ / t milled	**$**	**0.14**	-	-	1.01	1.10	0.14	0.10	0.10	0.10	0.10	0.10	0.10
Transportation Cost		$ / t milled	**$**	**0.36**	-	-	0.01	0.02	0.29	0.28	0.36	0.34	0.30	0.27	0.35
Offsite Costs		$ / t milled	**$**	**0.54**	-	-	0.12	0.17	0.48	0.46	0.55	0.52	0.46	0.41	0.51
NSR Royalty		$ / t milled	**$**	**1.63**	-	-	1.28	1.82	1.99	1.90	1.76	1.59	1.44	1.29	1.37
Special Advantages Tax Cost		$ / t milled	**$**	**0.85**	-	-	0.77	1.09	1.19	1.14	1.06	0.95	0.87	0.78	0.82
Science Contributions (ITC)		$ / t milled	**$**	**0.29**	-	-	0.26	0.37	0.40	0.39	0.36	0.33	0.30	0.27	0.28
Total Cost		$ / t milled	**$**	**12.96**	-	-	13.35	12.98	12.37	11.11	11.26	11.98	11.71	12.15	11.29
Sales Metrics															
Au Sales		koz		**37,639**	**-**	-	102	163.22	1,326	1,728	1,545	1,383	1,260	1,127	1,160
Total AISC	$	000 s		**28,071,387**	**-**	-	101,659	125,506	627,358	758,534	779,983	774,573	732,603	759,715	780,871
Less Ag and Cu By-Product Credits	$	000 s		**(9,875,392)**	**-**	-	(415)	(466)	(166,250)	(224,062)	(300,972)	(282,683)	(250,945)	(228,544)	(298,980)
AISC After By-Product Credits	$	000 s		**18,195,996**	-	-	101,244	125,040	461,108	534,472	479,011	491,890	481,658	531,171	481,891
AISC / oz Au (net of Ag and Cu byproduct credit)		$ / oz Au	**$**	**483**	-	-	992	766	348	309	310	356	382	472	416
AuEq Sales		koz		**45,236**	-	-	102	164	1,454	1,901	1,777	1,600	1,453	1,302	1,390
AISC / oz AuEq		$ / oz AuEq	**$**	**621**	-	-	993	767	431	399	439	484	504	583	562

YEARS 10-
21

Economic Model Annual Summary

				12	13	14	15	16	17	18	19	20	21	22	23
					<== 105kt/d Flot Plant + 35kt/d CIP										
Project Timeline in Years				12	13	14	15	16	17	18	19	20	21	22	23
Commercial Production Timeline in Years				10	11	12	13	14	15	16	17	18	19	20	21
Time Until Closure In Years		US$ & Metric Units		36	35	34	33	32	31	30	29	28	27	26	25
Market Prices															
Gold		US$/oz		1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300
Silver		US$/oz		17.00	17.00	17.00	17.00	17.00	17.00	17.00	17.00	17.00	17.00	17.00	17.00
Copper		US$/lb		3.00	3.00	3.00	3.00	3.00	3.00	3.00	3.00	3.00	3.00	3.00	3.00
Physicals															
Total Mill Feed Mined			kt	68,275	62,233	58,966	52,540	29,456	39,655	54,261	45,395	26,792	37,715	48,739	55,939
Total Waste Mined			kt	51,725	47,767	31,034	37,460	60,544	70,345	65,739	74,605	93,208	82,285	91,261	84,061
Total Material Mined			kt	120,000	110,000	90,000	90,000	90,000	110,000	120,000	120,000	120,000	120,000	140,000	140,000
Strip Ratio		W:O		0.76	0.77	0.53	0.71	2.06	1.77	1.21	1.64	3.48	2.18	1.87	1.50
CIP Plant Feed Processed			kt	5,800	12,250	12,250	12,250	12,250	12,250	12,250	12,250	12,250	12,250	12,250	12,250
Flotation Plant Feed Processed			kt	58,000	36,750	36,750	36,750	36,750	36,750	36,750	36,750	36,750	36,750	36,750	36,750
Total Mill Feed Processed			kt	63,800	49,000	49,000	49,000	49,000	49,000	49,000	49,000	49,000	49,000	49,000	49,000
Gold Grade, Processed		g/t		0.61	0.75	0.83	0.92	1.00	0.81	0.82	0.88	0.78	0.54	0.57	0.57
Silver Grade, Processed		g/t		0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50
Copper Grade, Processed			%	0.085	0.066	0.071	0.073	0.074	0.072	0.074	0.075	0.078	0.098	0.096	0.096
Contained Gold, Processed			koz	1,259	1,174	1,313	1,457	1,569	1,270	1,296	1,385	1,223	844	897	903
Contained Silver, Processed			koz	1,026	788	788	788	788	788	788	788	788	788	788	788
Contained Copper, Processed		klb		120,160	71,623	76,419	78,446	79,452	77,868	80,101	81,147	84,281	106,181	103,457	103,687
Average Recovery, Gold			%	83.5%	83.9%	83.9%	83.9%	83.8%	83.9%	83.9%	83.9%	83.9%	84.5%	84.3%	84.4%
Average Recovery, Silver			%	54.6%	50.3%	50.3%	50.3%	50.3%	50.3%	50.3%	50.3%	50.3%	50.3%	50.3%	50.3%
Average Recovery, Copper			%	83.8%	87.0%	86.5%	87.0%	84.0%	87.0%	87.0%	87.0%	87.0%	87.0%	80.5%	87.0%
Recovered Gold			koz	1,051	985	1,101	1,221	1,315	1,065	1,087	1,162	1,026	713	756	762
Recovered Silver			koz	560	396	396	396	396	396	396	396	396	396	396	396
Recovered Copper		klb		100,695	62,312	66,095	68,248	66,747	67,745	69,688	70,598	73,325	92,378	83,317	90,208
Payable Gold			koz	1,036.6	972.7	1,087.0	1,205.3	1,298.3	1,050.9	1,073.1	1,147.4	1,013.1	705.2	748.6	752.7
Payable Silver			koz	544.8	385.7	385.7	385.7	385.7	385.7	385.7	385.7	385.7	385.7	385.7	385.7
Payable Copper	1,450.44	klb		96,391.0	59,635.4	63,271.4	65,345.8	63,891.2	64,861.9	66,731.4	67,607.1	70,230.7	88,576.8	79,806.0	86,486.6
Cash Flow															
Gold Gross Revenue	83%	$	000s	1,347,632	1,264,510	1,413,155	1,566,839	1,687,748	1,366,154	1,395,043	1,491,623	1,316,970	916,779	973,217	978,515
Silver Gross Revenue	0.5%	$	000s	9,261	6,557	6,557	6,557	6,557	6,557	6,557	6,557	6,557	6,557	6,557	6,557
Copper Gross Revenue	16%	$	000s	289,173	178,906	189,814	196,037	191,674	194,586	200,194	202,821	210,692	265,731	239,418	259,460
Gross Revenue Before By-Product Credits	**100.0%**	**$**	**000s**	**1,646,066**	**1,449,973**	**1,609,526**	**1,769,433**	**1,885,978**	**1,567,297**	**1,601,794**	**1,701,001**	**1,534,219**	**1,189,066**	**1,219,192**	**1,244,531**
Gold Gross Revenue		$	000s	1,347,632	1,264,510	1,413,155	1,566,839	1,687,748	1,366,154	1,395,043	1,491,623	1,316,970	916,779	973,217	978,515
Silver Gross Revenue		$	000s	-	-	-	-	-	-	-	-	-	-	-	-
Copper Gross Revenue		$	000s	-	-	-	-	-	-	-	-	-	-	-	-
Gross Revenue After By-Product Credits		**$**	**000s**	**1,347,632**	**1,264,510**	**1,413,155**	**1,566,839**	**1,687,748**	**1,366,154**	**1,395,043**	**1,491,623**	**1,316,970**	**916,779**	**973,217**	**978,515**
Mining Cost		$	000s	(147,059)	(139,344)	(125,476)	(123,151)	(130,511)	(151,423)	(158,394)	(163,443)	(172,031)	(164,832)	(183,629)	(184,191)
Process Cost		$	000s	(302,620)	(239,285)	(237,485)	(238,303)	(236,239)	(240,232)	(238,148)	(238,357)	(237,931)	(240,895)	(232,499)	(248,773)
G&A Cost		$	000s	(65,404)	(61,710)	(60,873)	(61,106)	(61,408)	(62,842)	(63,219)	(63,565)	(64,148)	(63,932)	(64,829)	(65,736)
Engineering & Geology Cost		$	000s	(6,179)	(6,179)	(6,179)	(6,179)	(6,179)	(6,179)	(6,179)	(6,179)	(6,179)	(6,179)	(6,179)	(6,179)
ARD Plant Cost		$	000s	(323)	(323)	(323)	(323)	(323)	(323)	(323)	(323)	(323)	(323)	(323)	(323)
Transportation Cost		$	000s	(22,092)	(13,835)	(14,602)	(15,017)	(14,808)	(14,894)	(15,275)	(15,466)	(15,960)	(19,529)	(18,086)	(19,115)
Offsite Treatment Cost		$	000s	(32,090)	(21,383)	(22,826)	(23,805)	(23,919)	(23,054)	(23,663)	(24,215)	(24,334)	(28,071)	(26,287)	(27,615)
NSR Royalty		$	000s	(79,594)	(84,885)	(94,326)	(103,837)	(110,835)	(91,761)	(93,771)	(99,679)	(89,636)	(68,488)	(70,489)	(71,868)
Special Advantages Tax		$	000s	(47,756)	(42,443)	(47,163)	(51,918)	(55,418)	(45,880)	(46,886)	(49,840)	(44,818)	(34,244)	(35,245)	(35,934)
LOCTI (Science) Contributions		$	000s	(16,461)	(14,500)	(16,095)	(17,694)	(18,860)	(15,673)	(16,018)	(17,010)	(15,342)	(11,891)	(12,192)	(12,445)
Subtotal Cash Costs Before By-Product Credits		$	000s	(719,578)	(623,887)	(625,347)	(641,333)	(658,500)	(652,262)	(661,875)	(678,076)	(670,700)	(638,383)	(649,757)	(672,178)
By-Product Credits		$	000s	298,434	185,463	196,371	202,594	198,230	201,143	206,751	209,378	217,249	272,287	245,975	266,017
Total Cash Costs After By-Product Credits		**$**	**000s**	**(421,144)**	**(438,424)**	**(428,976)**	**(438,739)**	**(460,270)**	**(451,119)**	**(455,124)**	**(468,697)**	**(453,452)**	**(366,096)**	**(403,782)**	**(406,161)**
Operating Margin	**56%**	**$**	**000s**	**926,488**	**826,087**	**984,179**	**1,128,100**	**1,227,478**	**915,035**	**939,919**	**1,022,926**	**863,518**	**550,683**	**569,435**	**572,353**
EBITDA		**$**	**000s**	**926,488**	**826,087**	**984,179**	**1,128,100**	**1,227,478**	**915,035**	**939,919**	**1,022,926**	**863,518**	**550,683**	**569,435**	**572,353**
Stockpile Adjustments		$	000s	12,106	27,595	20,927	20,728	(27,929)	(5,124)	14,742	596	(69,571)	(33,834)	(2,308)	22,748
Capital Depreciation Allowance		$	000s	(71,969)	(69,058)	(78,652)	(111,482)	(114,765)	(109,706)	(109,704)	(125,734)	(116,902)	(97,835)	(125,235)	(127,764)
Amortization Allowance		$	000s	(9,745)	(8,641)	(8,843)	(7,881)	(7,448)	(8,445)	(9,348)	(10,003)	(8,033)	(8,185)	(8,565)	(7,380)
Reclamation Amortization		$	000s	(4,162)	(3,815)	(3,121)	(3,121)	(3,121)	(3,815)	(4,162)	(4,162)	(4,162)	(4,162)	(4,855)	(4,855)
Loss Carry Forward Credit		$	000s	-	-	-	-	-	-	-	-	-	-	-	-
Earnings Before Taxes		**$**	**000s**	**852,718**	**772,167**	**914,490**	**1,026,344**	**1,074,215**	**787,945**	**831,447**	**883,622**	**664,850**	**406,666**	**428,471**	**455,102**
Anti-Drug Contributions		$	000s	(8,406)	(7,446)	(8,936)	(10,056)	(11,021)	(7,931)	(8,167)	(8,830)	(7,344)	(4,405)	(4,308)	(4,324)
Sport Contributions		$	000s	(9,738)	(8,406)	(7,446)	(8,936)	(10,056)	(11,021)	(7,931)	(8,167)	(8,830)	(7,344)	(4,405)	(4,308)
Corp. Income Tax @ Effective Rate of:	22.5%	$	000s	(162,017)	(185,320)	(219,478)	(246,323)	(257,812)	(189,107)	(241,120)	(256,250)	(192,807)	(117,933)	(124,257)	(154,735)
Net Income		**$**	**000s**	**672,558**	**570,995**	**678,631**	**761,030**	**795,326**	**579,886**	**574,229**	**610,375**	**455,869**	**276,984**	**295,502**	**291,736**
Non-Cash Add Back - Stockpile Adjustments		$	000s	(12,106)	(27,595)	(20,927)	(20,728)	27,929	5,124	(14,742)	(596)	69,571	33,834	2,308	(22,748)
Non-Cash Add Back - Depreciation		$	000s	71,969	69,058	78,652	111,482	114,765	109,706	109,704	125,734	116,902	97,835	125,235	127,764
Non-Cash Add Back - Amortization		$	000s	9,745	8,641	8,843	7,881	7,448	8,445	9,348	10,003	8,033	8,185	8,565	7,380
Non-Cash Add Back - Reclamation Amortization		$	000s	4,162	3,815	3,121	3,121	3,121	3,815	4,162	4,162	4,162	4,162	4,855	4,855
Non-Cash Add Back - LCF Credit		$	000s	-	-	-	-	-	-	-	-	-	-	-	-
Working Capital		$	000s	19,529	23,836	(16,248)	(17,648)	(6,759)	27,615	(2,037)	(8,383)	14,673	42,695	4,635	(14,518)
Operating Cash Flow		**$**	**000s**	**765,856**	**648,751**	**732,072**	**845,138**	**941,830**	**734,591**	**680,664**	**741,294**	**669,210**	**463,696**	**441,100**	**394,469**
Development Capital		$	000s	-	-	-	-	-	-	-	-	-	-	-	-
Sustaining Capital		$	000s	(35,301)	(72,713)	(22,522)	(35,269)	(22,989)	(114,194)	(48,088)	(107,355)	(45,940)	(45,000)	(227,328)	(40,219)
Closure/Reclamation Capital		$	000s	(4,000)	-	-	-	-	-	-	-	-	-	(4,000)	-
Total Capital		**$**	**000s**	**(39,301)**	**(72,713)**	**(22,522)**	**(35,269)**	**(22,989)**	**(114,194)**	**(48,088)**	**(107,355)**	**(45,940)**	**(45,000)**	**(231,328)**	**(40,219)**
Cash Flow		**$**	**000s**	-	-	-	-	-	-	-	-	-	-	-	-

Adj./Reimbursements

LoM Metrics

Economic Metrics

Discount Factors		EOP @	10 %	0.3505	0.3186	0.2897	0.2633	0.2394	0.2176	0.1978	0.1799	0.1635	0.1486	0.1351	0.1228
a) Pre-Tax															
Free Cash Flow		$	000 s	906,716	777,211	945,410	1,075,183	1,197,730	828,455	889,793	907,187	832,251	548,378	342,741	517,616
Cumulative Free Cash Flow		$	000 s	7,221,611	7,998,822	8,944,231	10,019,414	11,217,144	12,045,600	12,935,393	13,842,580	14,674,831	15,223,209	15,565,950	16,083,566
NPV @ 10%		$	000 s	317,798	247,643	273,852	283,129	286,727	180,296	176,041	163,166	136,080	81,513	46,315	63,587
Cumulative NPV		$	000 s	2,980,167	3,227,810	3,501,662	3,784,791	4,071,518	4,251,814	4,427,855	4,591,021	4,727,100	4,808,613	4,854,928	4,918,515
IRR			%												
Undiscounted Payback From Start of Comm. Prod.			Years	3.8	3.8	3.8	3.8	3.8	3.8	3.8	3.8	3.8	3.8	3.8	3.8
PI @ 10%			NPV / (PW of TC)	13,775	23,168	6,524	9,287	5,503	24,852	9,514	19,309	7,512	6,689	31,260	4,941
b) After-Tax															
Free Cash Flow		$	000 s	726,555	576,039	709,551	809,869	918,840	620,397	632,576	633,940	623,270	418,695	209,772	354,250
Cumulative Free Cash Flow		$	000 s	5,452,758	6,028,797	6,738,348	7,548,216	8,467,057	9,087,453	9,720,029	10,353,969	10,977,238	11,395,934	11,605,706	11,959,956
NPV @ 10%		$	000 s	254,653	183,544	205,532	213,264	219,963	135,016	125,152	114,020	101,910	62,236	28,347	43,518
Cumulative NPV		$	000 s	2,076,295	2,259,838	2,465,370	2,678,634	2,898,597	3,033,613	3,158,765	3,272,785	3,374,694	3,436,931	3,465,277	3,508,795
IRR			%												
Undiscounted Payback from Start of Comm. Prod.			Years	4.1	4.1	4.1	4.1	4.1	4.1	4.1	4.1	4.1	4.1	4.1	4.1
PI @ 10%			NPV / (PW of TC)	13,775	23,168	6,524	9,287	5,503	24,852	9,514	19,309	7,512	6,689	31,260	4,941
Operating Metrics															
Mine Life			Years												
Maximum Daily Mining Rate			t/d mined	342,857	314,286	257,143	257,143	257,143	314,286	342,857	342,857	342,857	342,857	400,000	400,000
Maximum Daily Processing Rate - CIP			t/d milled	16,571	35,000	35,000	35,000	35,000	35,000	35,000	35,000	35,000	35,000	35,000	35,000
Maximum Daily Processing Rate - Concentrator			t/d milled	165,714	105,000	105,000	105,000	105,000	105,000	105,000	105,000	105,000	105,000	105,000	105,000
Maximum Daily Processing Rate - Combined			t/d milled	182,286	140,000	140,000	140,000	140,000	140,000	140,000	140,000	140,000	140,000	140,000	140,000
Mining Cost			$ / t mined	1.23	1.27	1.39	1.37	1.45	1.38	1.32	1.36	1.43	1.37	1.31	1.32
Mining Cost			$ / t milled	2.15	2.24	2.13	2.34	4.43	3.82	2.92	3.60	6.42	4.37	3.77	3.29
Processing Cost			$ / t milled	4.74	4.88	4.85	4.86	4.82	4.90	4.86	4.86	4.86	4.92	4.74	5.08
G&A Cost	17 %		$ / t milled	1.03	1.26	1.24	1.25	1.25	1.28	1.29	1.30	1.31	1.30	1.32	1.34
Other Infrastructure Cost			$ / t milled	0.10	0.13	0.13	0.13	0.13	0.13	0.13	0.13	0.13	0.13	0.13	0.13
Transportation Cost			$ / t milled	0.35	0.28	0.30	0.31	0.30	0.30	0.31	0.32	0.33	0.40	0.37	0.39
Offsite Costs			$ / t milled	0.50	0.44	0.47	0.49	0.49	0.47	0.48	0.49	0.50	0.57	0.54	0.56
NSR Royalty			$ / t milled	1.25	1.73	1.93	2.12	2.26	1.87	1.91	2.03	1.83	1.40	1.44	1.47
Special Advantages Tax Cost			$ / t milled	0.75	0.87	0.96	1.06	1.13	0.94	0.96	1.02	0.91	0.70	0.72	0.73
Science Contributions (ITC)			$ / t milled	0.26	0.30	0.33	0.36	0.38	0.32	0.33	0.35	0.31	0.24	0.25	0.25
Total Cost			$ / t milled	11.13	12.13	12.33	12.92	15.21	14.04	13.19	14.10	16.60	14.03	13.28	13.25
Sales Metrics															
Au Sales			koz	1,037	973	1,087	1,205	1,298	1,051	1,073	1,147	1,013	705	749	753
Total AISC		$	000 s	758,879	696,599	647,868	676,602	681,489	766,456	709,964	785,430	716,641	683,384	881,086	712,397
Less Ag and Cu By-Product Credits		$	000 s	(298,434)	(185,463)	(196,371)	(202,594)	(198,230)	(201,143)	(206,751)	(209,378)	(217,249)	(272,287)	(245,975)	(266,017)
AISC After By-Product Credits		$	000 s	460,445	511,136	451,497	474,008	483,259	565,313	503,213	576,052	499,392	411,096	635,111	446,381
AISC / oz Au (net of Ag and Cu byproduct credit)			$ / oz Au	444	525	415	393	372	538	469	502	493	583	848	593
AuEq Sales			koz	1,266	1,115	1,238	1,361	1,451	1,206	1,232	1,308	1,180	915	938	957
AISC / oz AuEq			$ / oz AuEq	599	625	523	497	470	636	576	600	607	747	939	744

YEARS 22-33

Economic Model Annual Summary

Project Timeline in Years				24	25	26	27	28	29	30	31	32	33	34	35
Commercial Production Timeline in Years				22	23	24	25	26	27	28	29	30	31	32	33
Time Until Closure In Years			US$ & Metric Units	24	23	22	21	20	19	18	17	16	15	14	13
Market Prices															
Gold			US$/oz	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300
Silver			US$/oz	17.00	17.00	17.00	17.00	17.00	17.00	17.00	17.00	17.00	17.00	17.00	17.00
Copper			US$/lb	3.00	3.00	3.00	3.00	3.00	3.00	3.00	3.00	3.00	3.00	3.00	3.00
Physicals															
Total Mill Feed Mined			kt	52,627	60,390	55,088	41,849	44,956	44,956	44,956	44,956	44,956	38,186	38,186	38,186
Total Waste Mined			kt	87,373	79,610	44,912	58,151	35,044	35,044	35,044	35,044	35,044	41,814	41,814	41,814
Total Material Mined			kt	140,000	140,000	100,000	100,000	80,000	80,000	80,000	80,000	80,000	80,000	80,000	80,000
Strip Ratio			W:O	1.66	1.32	0.82	1.39	0.78	0.78	0.78	0.78	0.78	1.10	1.10	1.10
CIP Plant Feed Processed			kt	11,954	9,915	2,355	9,887	7,858	7,858	7,858	7,858	7,858	4,256	4,256	4,256
Flotation Plant Feed Processed			kt	36,750	36,750	36,750	36,750	36,750	36,750	36,750	36,750	36,750	36,750	36,750	36,750
Total Mill Feed Processed			kt	48,704	46,665	39,105	46,637	44,608	44,608	44,608	44,608	44,608	41,006	41,006	41,006
Gold Grade, Processed			g/t	0.60	0.60	0.65	0.67	0.64	0.64	0.64	0.64	0.64	0.63	0.63	0.63
Silver Grade, Processed			g/t	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50
Copper Grade, Processed			%	0.099	0.104	0.128	0.110	0.099	0.099	0.099	0.099	0.099	0.106	0.106	0.106
Contained Gold, Processed			koz	946	898	812	998	911	911	911	911	911	830	830	830
Contained Silver, Processed			koz	783	750	629	750	717	717	717	717	717	659	659	659
Contained Copper, Processed			klb	105,791	107,038	110,305	112,907	97,048	97,048	97,048	97,048	97,048	96,182	96,182	96,182
Average Recovery, Gold			%	84.3 %	84.1 %	83.5 %	84.0 %	83.9 %	83.9 %	83.9 %	83.9 %	83.9 %	83.6 %	83.6 %	83.6 %
Average Recovery, Silver			%	50.4 %	51.3 %	55.5 %	51.4 %	52.3 %	52.3 %	52.3 %	52.3 %	52.3 %	54.3 %	54.3 %	54.3 %
Average Recovery, Copper			%	87.0 %	87.0 %	87.0 %	85.8 %	85.8 %	85.8 %	85.8 %	85.8 %	85.8 %	87.0 %	87.0 %	87.0 %
Recovered Gold			koz	798	755	677	839	765	765	765	765	765	694	694	694
Recovered Silver			koz	395	385	349	385	375	375	375	375	375	358	358	358
Recovered Copper			klb	92,038	93,123	95,966	96,890	83,224	83,224	83,224	83,224	83,224	83,679	83,679	83,679
Payable Gold			koz	788.4	745.7	667.5	828.3	755.3	755.3	755.3	755.3	755.3	684.6	684.6	684.6
Payable Silver			koz	384.3	374.7	338.9	374.5	364.9	364.9	364.9	364.9	364.9	347.9	347.9	347.9
Payable Copper	1,450.44		klb	88,249.7	89,294.5	92,033.4	92,916.2	79,752.6	79,752.6	79,752.6	79,752.6	79,752.6	80,198.2	80,198.2	80,198.2
Cash Flow															
Gold Gross Revenue	83 %	$	000 s	1,024,977	969,440	867,812	1,076,739	981,936	981,936	981,936	981,936	981,936	889,992	889,992	889,992
Silver Gross Revenue	0.5 %	$	000 s	6,533	6,369	5,762	6,367	6,204	6,204	6,204	6,204	6,204	5,915	5,915	5,915
Copper Gross Revenue	16 %	$	000 s	264,749	267,883	276,100	278,749	239,258	239,258	239,258	239,258	239,258	240,595	240,595	240,595
Gross Revenue Before By-Product Credits	**100.0 %**	**$**	**000 s**	**1,296,259**	**1,243,693**	**1,149,674**	**1,361,854**	**1,227,398**	**1,227,398**	**1,227,398**	**1,227,398**	**1,227,398**	**1,136,502**	**1,136,502**	**1,136,502**
Gold Gross Revenue		$	000 s	1,024,977	969,440	867,812	1,076,739	981,936	981,936	981,936	981,936	981,936	889,992	889,992	889,992
Silver Gross Revenue		$	000 s	-	-	-	-	-	-	-	-	-	-	-	-
Copper Gross Revenue		$	000 s	-	-	-	-	-	-	-	-	-	-	-	-
Gross Revenue After By-Product Credits		**$**	**000 s**	**1,024,977**	**969,440**	**867,812**	**1,076,739**	**981,936**	**981,936**	**981,936**	**981,936**	**981,936**	**889,992**	**889,992**	**889,992**
Mining Cost		$	000 s	(182,617)	(184,361)	(144,211)	(145,168)	(122,336)	(123,705)	(101,016)	(115,784)	(115,784)	(116,111)	(116,032)	(115,963)
Process Cost		$	000 s	(259,522)	(241,538)	(205,759)	(233,408)	(222,755)	(224,414)	(222,952)	(223,269)	(222,952)	(213,225)	(212,284)	(212,493)
G&A Cost		$	000 s	(66,167)	(65,301)	(61,437)	(62,758)	(60,966)	(61,056)	(59,571)	(60,441)	(60,426)	(59,918)	(59,789)	(59,796)
Engineering & Geology Cost		$	000 s	(6,179)	(6,179)	(6,179)	(6,179)	(6,179)	(6,179)	(6,179)	(6,179)	(6,179)	(6,179)	(6,179)	(6,179)
ARD Plant Cost		$	000 s	(323)	(323)	(220)	(220)	(220)	(220)	(220)	(220)	(220)	(220)	(220)	(220)
Transportation Cost		$	000 s	(19,461)	(19,645)	(20,102)	(20,400)	(17,824)	(17,824)	(17,824)	(17,824)	(17,824)	(17,826)	(17,826)	(17,826)
Offsite Treatment Cost		$	000 s	(28,210)	(28,272)	(28,414)	(29,598)	(25,838)	(25,838)	(25,838)	(25,838)	(25,838)	(25,484)	(25,484)	(25,484)
NSR Royalty		$	000 s	(74,915)	(71,747)	(66,069)	(78,711)	(71,024)	(71,024)	(71,024)	(71,024)	(71,024)	(65,592)	(65,592)	(65,592)
Special Advantages Tax		$	000 s	(37,458)	(35,873)	(33,035)	(39,356)	(35,512)	(35,512)	(35,512)	(35,512)	(35,512)	(32,796)	(32,796)	(32,796)
LOCTI (Science) Contributions		$	000 s	(12,963)	(12,437)	(11,497)	(13,619)	(12,274)	(12,274)	(12,274)	(12,274)	(12,274)	(11,365)	(11,365)	(11,365)
Subtotal Cash Costs Before By-Product Credits		$	000 s	(687,814)	(665,675)	(576,923)	(629,417)	(574,928)	(578,046)	(552,409)	(568,365)	(568,032)	(548,716)	(547,566)	(547,713)
By-Product Credits		$	000 s	271,282	274,253	281,862	285,116	245,462	245,462	245,462	245,462	245,462	246,509	246,509	246,509
Total Cash Costs After By-Product Credits		**$**	**000 s**	**(416,531)**	**(391,422)**	**(295,061)**	**(344,301)**	**(329,466)**	**(332,584)**	**(306,947)**	**(322,903)**	**(322,570)**	**(302,206)**	**(301,057)**	**(301,203)**
Operating Margin	**56 %**	**$**	**000 s**	**608,446**	**578,018**	**572,750**	**732,438**	**652,470**	**649,352**	**674,988**	**659,032**	**659,365**	**587,786**	**588,936**	**588,789**
EBITDA		**$**	**000 s**	**608,446**	**578,018**	**572,750**	**732,438**	**652,470**	**649,352**	**674,988**	**659,032**	**659,365**	**587,786**	**588,936**	**588,789**
Stockpile Adjustments		$	000 s	15,847	35,479	36,828	(9,238)	(6,928)	(1,052)	(9,848)	3,696	3,125	2,602	(3,512)	(6,783)
Capital Depreciation Allowance		$	000 s	(132,610)	(126,030)	(114,455)	(118,043)	(109,259)	(111,173)	(86,073)	(88,900)	(87,095)	(80,253)	(80,946)	(85,719)
Amortization Allowance		$	000 s	(5,052)	(5,393)	(5,421)	(5,643)	(5,354)	(4,797)	(5,998)	(5,432)	(5,432)	(5,432)	(5,385)	(5,335)
Reclamation Amortization		$	000 s	(4,855)	(4,855)	(3,468)	(3,468)	(2,775)	(2,775)	(2,775)	(2,775)	(2,775)	(2,775)	(2,775)	(2,775)
Loss Carry Forward Credit		$	000 s	-	-	-	-	-	-	-	-	-	-	-	-
Earnings Before Taxes		**$**	**000 s**	**481,775**	**477,218**	**486,234**	**596,045**	**528,154**	**529,556**	**570,295**	**565,621**	**567,189**	**501,929**	**496,318**	**488,177**
Anti-Drug Contributions		$	000 s	(4,659)	(4,417)	(4,494)	(6,053)	(5,351)	(5,306)	(5,801)	(5,619)	(5,641)	(4,993)	(4,998)	(4,950)
Sport Contributions		$	000 s	(4,324)	(4,659)	(4,417)	(4,494)	(6,053)	(5,351)	(5,306)	(5,801)	(5,619)	(5,641)	(4,993)	(4,998)
Corp. Income Tax @ Effective Rate of:	22.5 %	$	000 s	(163,804)	(162,254)	(165,320)	(202,655)	(179,572)	(180,049)	(193,900)	(192,311)	(192,844)	(170,656)	(168,748)	(165,980)
Net Income		**$**	**000 s**	**308,989**	**305,887**	**312,003**	**382,843**	**337,178**	**338,850**	**365,287**	**361,889**	**363,085**	**320,639**	**317,578**	**312,249**
Non-Cash Add Back - Stockpile Adjustments		$	000 s	(15,847)	(35,479)	(36,828)	9,238	6,928	1,052	9,848	(3,696)	(3,125)	(2,602)	3,512	6,783
Non-Cash Add Back - Depreciation		$	000 s	132,610	126,030	114,455	118,043	109,259	111,173	86,073	88,900	87,095	80,253	80,946	85,719
Non-Cash Add Back - Amortization		$	000 s	5,052	5,393	5,421	5,643	5,354	4,797	5,998	5,432	5,432	5,432	5,385	5,335
Non-Cash Add Back - Reclamation Amortization		$	000 s	4,855	4,855	3,468	3,468	2,775	2,775	2,775	2,775	2,775	2,775	2,775	2,775
Non-Cash Add Back - LCF Credit		$	000 s	-	-	-	-	-	-	-	-	-	-	-	-
Working Capital		$	000 s	(4,850)	2,200	(7,582)	(4,740)	9,949	142	(896)	539	(19)	331	(66)	10
Operating Cash Flow		**$**	**000 s**	**430,810**	**408,888**	**390,938**	**514,495**	**471,443**	**458,788**	**469,085**	**455,839**	**455,242**	**406,828**	**410,130**	**412,871**
Development Capital		$	000 s	-	-	-	-	-	-	-	-	-	-	-	-
Sustaining Capital		$	000 s	(42,443)	(62,739)	(19,228)	(24,507)	(27,534)	(31,447)	(39,531)	(46,500)	(26,067)	(25,257)	(37,345)	(45,003)
Closure/Reclamation Capital		$	000 s	-	-	-	-	-	-	-	-	(3,000)	-	-	(6,000)
Total Capital		**$**	**000 s**	**(42,443)**	**(62,739)**	**(19,228)**	**(24,507)**	**(27,534)**	**(31,447)**	**(39,531)**	**(46,500)**	**(29,067)**	**(25,257)**	**(37,345)**	**(51,003)**

Cash Flow Adj./Reimbursements		**$**	**000 s**	-	-	-	-	-	-	-	-	-	-	-	-
LoM Metrics															
Economic Metrics															
Discount Factors		EOP @	10 %	0.1117	0.1015	0.0923	0.0839	0.0763	0.0693	0.0630	0.0573	0.0521	0.0474	0.0431	0.0391
a) Pre-Tax															
Free Cash Flow		$	000 s	561,153	517,479	545,940	703,190	634,884	618,046	634,562	613,072	630,280	562,860	551,524	537,796
Cumulative Free Cash Flow		$	000 s	16,644,719	17,162,198	17,708,138	18,411,328	19,046,213	19,664,259	20,298,820	20,911,892	21,542,171	22,105,031	22,656,556	23,194,352
NPV @ 10%		$	000 s	62,668	52,537	50,388	59,001	48,428	42,857	40,002	35,134	32,837	26,658	23,747	21,051
Cumulative NPV		$	000 s	4,981,183	5,033,721	5,084,109	5,143,110	5,191,538	5,234,395	5,274,398	5,309,532	5,342,369	5,369,027	5,392,774	5,413,825
IRR			%												
Undiscounted Payback From Start of Comm. Prod.			Years	3.8	3.8	3.8	3.8	3.8	3.8	3.8	3.8	3.8	3.8	3.8	3.8
PI @ 10%			NPV / (PW of TC)	4,740	6,370	1,775	2,056	2,100	2,181	2,492	2,665	1,514	1,196	1,608	1,996
b) After-Tax															
Free Cash Flow		$	000 s	388,366	346,149	371,709	489,988	443,908	427,340	429,554	409,340	426,176	381,570	372,785	361,868
Cumulative Free Cash Flow		$	000 s	12,348,322	12,694,470	13,066,180	13,556,168	14,000,076	14,427,416	14,856,970	15,266,309	15,692,485	16,074,055	16,446,840	16,808,708
NPV @ 10%		$	000 s	43,372	35,143	34,307	41,113	33,860	29,633	27,079	23,459	22,203	18,072	16,051	14,164
Cumulative NPV		$	000 s	3,552,167	3,587,310	3,621,618	3,662,730	3,696,591	3,726,224	3,753,303	3,776,761	3,798,964	3,817,037	3,833,087	3,847,252
IRR			%												
Undiscounted Payback from Start of Comm. Prod.			Years	4.1	4.1	4.1	4.1	4.1	4.1	4.1	4.1	4.1	4.1	4.1	4.1
PI @ 10%			NPV / (PW of TC)	4,740	6,370	1,775	2,056	2,100	2,181	2,492	2,665	1,514	1,196	1,608	1,996
Operating Metrics															
Mine Life			Years												
Maximum Daily Mining Rate			t/d mined	400,000	400,000	285,714	285,714	228,571	228,571	228,571	228,571	228,571	228,571	228,571	228,571
Maximum Daily Processing Rate - CIP			t/d milled	34,154	28,327	6,727	28,248	22,451	22,451	22,451	22,451	22,451	12,159	12,159	12,159
Maximum Daily Processing Rate - Concentrator			t/d milled	105,000	105,000	105,000	105,000	105,000	105,000	105,000	105,000	105,000	105,000	105,000	105,000
Maximum Daily Processing Rate - Combined			t/d milled	139,154	133,327	111,727	133,248	127,451	127,451	127,451	127,451	127,451	117,159	117,159	117,159
Mining Cost			$ / t mined	1.30	1.32	1.44	1.45	1.53	1.55	1.26	1.45	1.45	1.45	1.45	1.45
Mining Cost			$ / t milled	3.47	3.05	2.62	3.47	2.72	2.75	2.25	2.58	2.58	3.04	3.04	3.04
Processing Cost			$ / t milled	5.33	5.18	5.26	5.00	4.99	5.03	5.00	5.01	5.00	5.20	5.18	5.18
G&A Cost	17 %		$ / t milled	1.36	1.40	1.57	1.35	1.37	1.37	1.34	1.35	1.35	1.46	1.46	1.46
Other Infrastructure Cost			$ / t milled	0.13	0.14	0.16	0.14	0.14	0.14	0.14	0.14	0.14	0.16	0.16	0.16
Transportation Cost			$ / t milled	0.40	0.42	0.51	0.44	0.40	0.40	0.40	0.40	0.40	0.43	0.43	0.43
Offsite Costs			$ / t milled	0.58	0.61	0.73	0.63	0.58	0.58	0.58	0.58	0.58	0.62	0.62	0.62
NSR Royalty			$ / t milled	1.54	1.54	1.69	1.69	1.59	1.59	1.59	1.59	1.59	1.60	1.60	1.60
Special Advantages Tax Cost			$ / t milled	0.77	0.77	0.84	0.84	0.80	0.80	0.80	0.80	0.80	0.80	0.80	0.80
Science Contributions (ITC)			$ / t milled	0.27	0.27	0.29	0.29	0.28	0.28	0.28	0.28	0.28	0.28	0.28	0.28
Total Cost			$ / t milled	13.84	13.37	13.68	13.85	12.87	12.94	12.37	12.72	12.71	13.59	13.56	13.57
Sales Metrics															
Au Sales			koz	788	746	668	828	755	755	755	755	755	685	685	685
Total AISC		$	000 s	730,257	728,414	596,152	653,924	602,462	609,493	591,940	614,865	597,099	573,973	584,911	598,716
Less Ag and Cu By-Product Credits		$	000 s	(271,282)	(274,253)	(281,862)	(285,116)	(245,462)	(245,462)	(245,462)	(245,462)	(245,462)	(246,509)	(246,509)	(246,509)
AISC After By-Product Credits		$	000 s	458,975	454,161	314,289	368,808	357,000	364,031	346,478	369,403	351,637	327,463	338,401	352,206
AISC / oz Au (net of Ag and Cu byproduct credit)			$ / oz Au	582	609	471	445	473	482	459	489	466	478	494	514
AuEq Sales			koz	997	957	884	1,048	944	944	944	944	944	874	874	874
AISC / oz AuEq			$ / oz AuEq	732	761	674	624	638	646	627	651	632	657	669	685

YEARS 34-45

Economic Model Annual Summary

Company	GR Engineering (Ba	
Project Name	Brisas/Cristinas	
Scenario Name	15CIP_140Flot_V30	
Analysis Type	PEA	

Project Timeline in Years				36	37	38	39	40	41	42	43	44	45	46	47
Commercial Production Timeline in Years				34	35	36	37	38	39	40	41	42	43	44	45
Time Until Closure In Years			US$ & Metric Units	12	11	10	9	8	7	6	5	4	3	2	1
Market Prices															
Gold			US$/oz	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300
Silver			US$/oz	17.00	17.00	17.00	17.00	17.00	17.00	17.00	17.00	17.00	17.00	17.00	17.00
Copper			US$/lb	3.00	3.00	3.00	3.00	3.00	3.00	3.00	3.00	3.00	3.00	3.00	3.00
Physicals															
Total Mill Feed Mined			kt	38,186	38,186	33,607	33,607	33,607	33,607	33,607	37,099	37,099	37,099	37,099	6,214
Total Waste Mined			kt	41,814	41,814	60,393	60,393	60,393	60,393	60,393	32,026	32,026	32,026	32,026	5,364
Total Material Mined			kt	80,000	80,000	94,000	94,000	94,000	94,000	94,000	69,125	69,125	69,125	69,125	11,578
Strip Ratio			W:O	1.10	1.10	1.80	1.80	1.80	1.80	1.80	0.86	0.86	0.86	0.86	0.86
CIP Plant Feed Processed			kt	4,256	4,256	2,386	2,386	2,386	2,386	2,386	834	834	834	834	140
Flotation Plant Feed Processed			kt	36,750	36,750	36,750	36,750	36,750	36,750	36,750	36,750	36,750	36,750	36,750	6,155
Total Mill Feed Processed			kt	41,006	41,006	39,136	39,136	39,136	39,136	39,136	37,584	37,584	37,584	37,584	6,295
Gold Grade, Processed			g/t	0.63	0.63	0.67	0.67	0.67	0.67	0.67	0.50	0.50	0.50	0.50	0.50
Silver Grade, Processed			g/t	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50
Copper Grade, Processed			%	0.106	0.106	0.084	0.084	0.084	0.084	0.084	0.106	0.106	0.106	0.106	0.106
Contained Gold, Processed			koz	830	830	839	839	839	839	839	605	605	605	605	101
Contained Silver, Processed			koz	659	659	629	629	629	629	629	604	604	604	604	101
Contained Copper, Processed			klb	96,182	96,182	72,863	72,863	72,863	72,863	72,863	88,200	88,200	88,200	88,200	14,773
Average Recovery, Gold			%	83.6 %	83.6 %	83.5 %	83.5 %	83.5 %	83.5 %	83.5 %	83.3 %	83.3 %	83.3 %	83.3 %	83.3 %
Average Recovery, Silver			%	54.3 %	54.3 %	55.4 %	55.4 %	55.4 %	55.4 %	55.4 %	56.5 %	56.5 %	56.5 %	56.5 %	56.5 %
Average Recovery, Copper			%	87.0 %	87.0 %	84.4 %	84.4 %	84.4 %	84.4 %	84.4 %	86.6 %	86.6 %	86.6 %	86.6 %	86.6 %
Recovered Gold			koz	694	694	701	701	701	701	701	504	504	504	504	84
Recovered Silver			koz	358	358	349	349	349	349	349	341	341	341	341	57
Recovered Copper			klb	83,679	83,679	61,531	61,531	61,531	61,531	61,531	76,357	76,357	76,357	76,357	12,789
Payable Gold			koz	684.6	684.6	691.1	691.1	691.1	691.1	691.1	496.5	496.5	496.5	496.5	83.2
Payable Silver			koz	347.9	347.9	339.1	339.1	339.1	339.1	339.1	331.7	331.7	331.7	331.7	55.6
Payable Copper	1,450.44		klb	80,198.2	80,198.2	58,865.4	58,865.4	58,865.4	58,865.4	58,865.4	73,146.0	73,146.0	73,146.0	73,146.0	12,251.3
Cash Flow															
Gold Gross Revenue	83 %	$	000 s	889,992	889,992	898,478	898,478	898,478	898,478	898,478	645,470	645,470	645,470	645,470	108,110
Silver Gross Revenue	0.5 %	$	000 s	5,915	5,915	5,764	5,764	5,764	5,764	5,764	5,640	5,640	5,640	5,640	945
Copper Gross Revenue	16 %	$	000 s	240,595	240,595	176,596	176,596	176,596	176,596	176,596	219,438	219,438	219,438	219,438	36,754
Gross Revenue Before By-Product Credits	**100.0 %**	**$**	**000 s**	**1,136,502**	**1,136,502**	**1,080,839**	**1,080,839**	**1,080,839**	**1,080,839**	**1,080,839**	**870,548**	**870,548**	**870,548**	**870,548**	**145,809**
Gold Gross Revenue		$	000 s	889,992	889,992	898,478	898,478	898,478	898,478	898,478	645,470	645,470	645,470	645,470	108,110
Silver Gross Revenue		$	000 s	-	-	-	-	-	-	-	-	-	-	-	-
Copper Gross Revenue		$	000 s	-	-	-	-	-	-	-	-	-	-	-	-
Gross Revenue After By-Product Credits		**$**	**000 s**	**889,992**	**889,992**	**898,478**	**898,478**	**898,478**	**898,478**	**898,478**	**645,470**	**645,470**	**645,470**	**645,470**	**108,110**
Mining Cost		$	000 s	(116,253)	(114,648)	(127,383)	(127,201)	(119,432)	(119,530)	(119,533)	(95,759)	(95,759)	(95,857)	(95,564)	(41,677)
Process Cost		$	000 s	(211,925)	(213,368)	(204,268)	(204,802)	(204,268)	(205,372)	(204,214)	(200,909)	(200,341)	(200,731)	(200,537)	(65,521)
G&A Cost		$	000 s	(59,782)	(59,591)	(59,864)	(59,780)	(57,697)	(57,757)	(57,699)	(56,006)	(55,978)	(56,002)	(55,905)	(46,459)
Engineering & Geology Cost		$	000 s	(6,179)	(6,179)	(6,179)	(6,179)	(6,179)	(6,179)	(6,179)	(6,179)	(6,179)	(6,179)	(6,179)	(6,179)
ARD Plant Cost		$	000 s	(220)	(220)	(220)	(220)	(220)	(220)	(220)	(220)	(220)	(220)	(220)	(220)
Transportation Cost		$	000 s	(17,826)	(17,826)	(13,684)	(13,684)	(13,684)	(13,684)	(13,684)	(16,407)	(16,407)	(16,407)	(16,407)	(2,748)
Offsite Treatment Cost		$	000 s	(25,484)	(25,484)	(19,961)	(19,961)	(19,961)	(19,961)	(19,961)	(22,822)	(22,822)	(22,822)	(22,822)	(3,822)
NSR Royalty		$	000 s	(65,592)	(65,592)	(62,832)	(62,832)	(62,832)	(62,832)	(62,832)	(49,879)	(49,879)	(49,879)	(49,879)	(8,354)
Special Advantages Tax		$	000 s	(32,796)	(32,796)	(31,416)	(31,416)	(31,416)	(31,416)	(31,416)	(24,940)	(24,940)	(24,940)	(24,940)	(4,177)
LOCTI (Science) Contributions		$	000 s	(11,365)	(11,365)	(10,808)	(10,808)	(10,808)	(10,808)	(10,808)	(8,705)	(8,705)	(8,705)	(8,705)	(1,458)
Subtotal Cash Costs Before By-Product Credits		$	000 s	(547,421)	(547,068)	(536,616)	(536,883)	(526,497)	(527,759)	(526,546)	(481,827)	(481,230)	(481,743)	(481,158)	(180,617)
By-Product Credits		$	000 s	246,509	246,509	182,361	182,361	182,361	182,361	182,361	225,078	225,078	225,078	225,078	37,698
Total Cash Costs After By-Product Credits		**$**	**000 s**	**(300,912)**	**(300,559)**	**(354,255)**	**(354,522)**	**(344,137)**	**(345,398)**	**(344,186)**	**(256,749)**	**(256,153)**	**(256,665)**	**(256,081)**	**(142,918)**
Operating Margin	**56 %**	**$**	**000 s**	**589,081**	**589,434**	**544,224**	**543,957**	**554,342**	**553,080**	**554,293**	**388,722**	**389,318**	**388,806**	**389,390**	**(34,808)**
EBITDA		**$**	**000 s**	**589,081**	**589,434**	**544,224**	**543,957**	**554,342**	**553,080**	**554,293**	**388,722**	**389,318**	**388,806**	**389,390**	**(34,808)**
Stockpile Adjustments		$	000 s	(8,312)	(9,848)	(9,569)	(14,771)	(18,577)	(17,169)	(16,809)	(1,377)	(1,316)	(1,311)	(1,315)	(219)
Capital Depreciation Allowance		$	000 s	(85,266)	(90,882)	(89,209)	(89,995)	(89,211)	(98,138)	(96,726)	(88,041)	(90,327)	(91,724)	(92,833)	(149,568)
Amortization Allowance		$	000 s	(4,786)	(4,786)	(4,786)	(4,792)	(4,800)	(4,855)	(4,855)	(4,855)	(4,254)	(3,485)	(1,663)	(1,663)
Reclamation Amortization		$	000 s	(2,775)	(2,775)	(3,260)	(3,260)	(3,260)	(3,260)	(3,260)	(2,397)	(2,397)	(2,397)	(2,397)	(402)
Loss Carry Forward Credit		$	000 s	-	-	-	-	-	-	-	-	-	-	-	-
Earnings Before Taxes		**$**	**000 s**	**487,942**	**481,144**	**437,400**	**431,138**	**438,494**	**429,658**	**432,643**	**292,051**	**291,024**	**289,888**	**291,182**	**(186,660)**
Anti-Drug Contributions		$	000 s	(4,963)	(4,910)	(4,470)	(4,459)	(4,571)	(4,468)	(4,495)	(2,934)	(2,923)	(2,912)	(2,925)	-
Sport Contributions		$	000 s	(4,950)	(4,963)	(4,910)	(4,470)	(4,459)	(4,571)	(4,468)	(4,495)	(2,934)	(2,923)	(2,912)	(2,925)
Corp. Income Tax @ Effective Rate of:	22.5 %	$	000 s	(165,900)	(163,589)	(148,716)	(146,587)	(149,088)	(146,084)	(147,099)	(99,297)	(98,948)	(98,562)	(99,002)	-
Net Income		**$**	**000 s**	**312,129**	**307,682**	**279,304**	**275,622**	**280,376**	**274,535**	**276,582**	**185,325**	**186,218**	**185,490**	**186,343**	**(189,585)**
Non-Cash Add Back - Stockpile Adjustments		$	000 s	8,312	9,848	9,569	14,771	18,577	17,169	16,809	1,377	1,316	1,311	1,315	219
Non-Cash Add Back - Depreciation		$	000 s	85,266	90,882	89,209	89,995	89,211	98,138	96,726	88,041	90,327	91,724	92,833	149,568
Non-Cash Add Back - Amortization		$	000 s	4,786	4,786	4,786	4,792	4,800	4,855	4,855	4,855	4,254	3,485	1,663	1,663
Non-Cash Add Back - Reclamation Amortization		$	000 s	2,775	2,775	3,260	3,260	3,260	3,260	3,260	2,397	2,397	2,397	2,397	402
Non-Cash Add Back - LCF Credit		$	000 s	-	-	-	-	-	-	-	-	-	-	-	-
Working Capital		$	000 s	(24)	15	6,298	17	(450)	70	(70)	15,236	(34)	27	(28)	68,953
Operating Cash Flow		**$**	**000 s**	**413,244**	**415,988**	**392,426**	**388,457**	**395,774**	**398,027**	**398,162**	**297,232**	**284,478**	**284,435**	**284,523**	**31,221**
Development Capital		$	000 s	-	-	-	-	-	-	-	-	-	-	-	-
Sustaining Capital		$	000 s	(25,994)	(42,572)	(25,210)	(31,857)	(13,761)	(58,290)	(30,217)	(119,841)	(36,437)	(21,385)	(28,414)	(9,242)
Closure/Reclamation Capital		$	000 s	(7,500)	(7,500)	(10,500)	(10,500)	(10,500)	(10,500)	(10,500)	(10,500)	(10,500)	(10,500)	(10,500)	(10,500)
Total Capital		**$**	**000 s**	**(33,494)**	**(50,072)**	**(35,710)**	**(42,357)**	**(24,261)**	**(68,790)**	**(40,717)**	**(130,341)**	**(46,937)**	**(31,885)**	**(38,914)**	**(19,742)**

Cash Flow Adj./Reimbursements		**$**	**000 s**	-	-	-	-	-	-	-	-	-	-	-	-
LoM Metrics															
Economic Metrics															
Discount Factors		EOP @	10 %	0.0356	0.0323	0.0294	0.0267	0.0243	0.0221	0.0201	0.0183	0.0166	0.0151	0.0137	0.0125
a) Pre-Tax															
Free Cash Flow		$	000 s	555,562	539,376	514,812	501,617	529,630	484,360	513,506	273,617	342,347	356,948	350,448	14,404
Cumulative Free Cash Flow		$	000 s	23,749,914	24,289,290	24,804,102	25,305,719	25,835,349	26,319,708	26,833,215	27,106,831	27,449,178	27,806,126	28,156,574	28,170,977
NPV @ 10%		$	000 s	19,769	17,448	15,140	13,411	12,872	10,702	10,314	4,996	5,683	5,387	4,808	180
Cumulative NPV		$	000 s	5,433,594	5,451,042	5,466,182	5,479,593	5,492,465	5,503,167	5,513,481	5,518,478	5,524,161	5,529,547	5,534,355	5,534,535
IRR			%												
Undiscounted Payback From Start of Comm. Prod.			Years	3.8	3.8	3.8	3.8	3.8	3.8	3.8	3.8	3.8	3.8	3.8	3.8
PI @ 10%			NPV / (PW of TC)	1,192	1,620	1,050	1,132	590	1,520	818	2,380	779	481	534	246
b) After-Tax															
Free Cash Flow		$	000 s	379,750	365,915	356,716	346,101	371,512	329,237	357,445	166,891	237,541	252,551	245,609	11,479
Cumulative Free Cash Flow		$	000 s	17,188,458	17,554,373	17,911,089	18,257,190	18,628,702	18,957,939	19,315,384	19,482,275	19,719,816	19,972,366	20,217,975	20,229,454
NPV @ 10%		$	000 s	13,513	11,837	10,490	9,253	9,029	7,274	7,180	3,047	3,943	3,811	3,370	143
Cumulative NPV		$	000 s	3,860,765	3,872,602	3,883,092	3,892,345	3,901,375	3,908,649	3,915,829	3,918,876	3,922,820	3,926,631	3,930,001	3,930,144
IRR			%												
Undiscounted Payback from Start of Comm. Prod.			Years	4.1	4.1	4.1	4.1	4.1	4.1	4.1	4.1	4.1	4.1	4.1	4.1
PI @ 10%			NPV / (PW of TC)	1,192	1,620	1,050	1,132	590	1,520	818	2,380	779	481	534	246
Operating Metrics															
Mine Life			Years												
Maximum Daily Mining Rate			t/d mined	228,571	228,571	268,571	268,571	268,571	268,571	268,571	197,499	197,499	197,499	197,499	33,079
Maximum Daily Processing Rate - CIP			t/d milled	12,159	12,159	6,818	6,818	6,818	6,818	6,818	2,382	2,382	2,382	2,382	399
Maximum Daily Processing Rate - Concentrator			t/d milled	105,000	105,000	105,000	105,000	105,000	105,000	105,000	105,000	105,000	105,000	105,000	17,587
Maximum Daily Processing Rate - Combined			t/d milled	117,159	117,159	111,818	111,818	111,818	111,818	111,818	107,382	107,382	107,382	107,382	17,986
Mining Cost			$ / t mined	1.45	1.43	1.36	1.35	1.27	1.27	1.27	1.39	1.39	1.39	1.38	3.60
Mining Cost			$ / t milled	3.04	3.00	3.79	3.78	3.55	3.56	3.56	2.58	2.58	2.58	2.58	6.71
Processing Cost			$ / t milled	5.17	5.20	5.22	5.23	5.22	5.25	5.22	5.35	5.33	5.34	5.34	10.41
G&A Cost	17 %		$ / t milled	1.46	1.45	1.53	1.53	1.47	1.48	1.47	1.49	1.49	1.49	1.49	7.38
Other Infrastructure Cost			$ / t milled	0.16	0.16	0.16	0.16	0.16	0.16	0.16	0.17	0.17	0.17	0.17	1.02
Transportation Cost			$ / t milled	0.43	0.43	0.35	0.35	0.35	0.35	0.35	0.44	0.44	0.44	0.44	0.44
Offsite Costs			$ / t milled	0.62	0.62	0.51	0.51	0.51	0.51	0.51	0.61	0.61	0.61	0.61	0.61
NSR Royalty			$ / t milled	1.60	1.60	1.61	1.61	1.61	1.61	1.61	1.33	1.33	1.33	1.33	1.33
Special Advantages Tax Cost			$ / t milled	0.80	0.80	0.80	0.80	0.80	0.80	0.80	0.66	0.66	0.66	0.66	0.66
Science Contributions (ITC)			$ / t milled	0.28	0.28	0.28	0.28	0.28	0.28	0.28	0.23	0.23	0.23	0.23	0.23
Total Cost			$ / t milled	13.56	13.55	14.25	14.25	13.96	13.99	13.96	12.85	12.84	12.85	12.84	28.78
Sales Metrics															
Au Sales			koz	685	685	691	691	691	691	691	497	497	497	497	83
Total AISC		$	000 s	580,915	597,140	572,326	579,239	550,759	596,549	567,263	612,168	528,167	513,627	520,072	200,359
Less Ag and Cu By-Product Credits		$	000 s	(246,509)	(246,509)	(182,361)	(182,361)	(182,361)	(182,361)	(182,361)	(225,078)	(225,078)	(225,078)	(225,078)	(37,698)
AISC After By-Product Credits		$	000 s	334,406	350,631	389,965	396,878	368,398	414,189	384,903	387,090	303,089	288,550	294,994	162,660
AISC / oz Au (net of Ag and Cu byproduct credit)			$ / oz Au	488	512	564	574	533	599	557	780	610	581	594	1,956
AuEq Sales			koz	874	874	831	831	831	831	831	670	670	670	670	112
AISC / oz AuEq			$ / oz AuEq	664	683	688	697	662	718	682	914	789	767	777	1,786

RICHARD J. LAMBERT

I, Richard J. Lambert, P.Eng., as an author of this report titled "Technical Report on the Siembra Minera Project, Bolivar State, Venezuela" prepared for GR Engineering (Barbados), and dated March 16, 2018, do hereby certify that:

1. I am Principal Mining Consultant with Roscoe Postle Associates Inc. of Suite 505, 143 Union Boulevard, Lakewood, CO, USA 80227.

2. I am a graduate of Mackay School of Mines, University of Nevada, Reno, U.S.A., with a Bachelor of Science degree in Mining Engineering in 1980, and Boise State University, with a Masters of Business Administration degree in 1995.

3. I am a Registered Professional Engineer in the state of Wyoming (#4857) and the state of Montana (#11475). I am licensed as a Professional Engineer in the Province of Ontario (Reg. #100139998). I have been a member of the Society for Mining, Metallurgy, and Exploration (SME) since 1975, and a Registered Member (RM#1825610) since May 2006. I have worked as a mining engineer for a total of 37 years since my graduation. My relevant experience for the purpose of the Technical Report is:
   - Review and report as a consultant on numerous mining projects for due diligence and regulatory requirements
   - Mine engineering, mine management, mine operations and mine financial analyses, involving copper, gold, silver, nickel, cobalt, uranium, oil shale, phosphates, coal and base metals located in the United States, Canada, Zambia, Madagascar, Turkey, Bolivia, Chile, Brazil, Serbia, Australia, Russia and Venezuela.

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 fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

5. I visited the Brisas Project site in February 2008. During the visit I observed the planned pit, process plant, mine shop, tailings facility and waste dump areas. I reviewed the drill core.

6. I am responsible for the preparation of Sections 15, 16, 19 and 20 and collaborated with my co-authors on Sections 1, 2, 3, 18, 21, 24, 25, 26, and 27 of the Technical Report.

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

8. I prepared a previous Technical Report on the Brisas Project dated March 31, 2008.

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

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

Dated this 16th day of March, 2018

(Signed and Sealed) "*Richard J. Lambert*"

Richard J. Lambert, P.Eng.

JOSÉ TEXIDOR CARLSSON

I, José Texidor Carlsson, P.Geo., as an author of this report entitled "Technical Report on the Siembra Minera Project, Bolivar State, Venezuela" prepared for GR Engineering (Barbados), Inc., and dated March 16, 2018, do hereby certify that:

1. I am a Senior Geologist with Roscoe Postle Associates Inc. of Suite 501, 55 University Ave Toronto, ON, M5J 2H7.

2. I am a graduate of University of Surrey, United Kingdom, in 1998 with a Master of Engineering, Electronic and Electrical degree and Acadia University, Nova Scotia, in 2007 with an M.Sc. degree in Geology.

3. I am registered as a Professional Geologist in the Province of Ontario (Reg. #2143). I have worked as a geologist for a total of 10 years since my graduation. My relevant experience for the purpose of the Technical Report is:
   - Mineral Resource estimation and NI 43-101 reporting
   - Supervision of exploration properties and active mines in Canada, Mexico, and South America
   - Experienced user of geological and resource modelling software

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 fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

5. I did not visit the Siembra Minera Project.

6. I am responsible for Sections 4 to 12 and 14 and share responsibility for Sections 1, 2, 23, 24, 25, 26, and 27 of the Technical Report.

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

8. I have had no prior involvement with the property that is the subject of the Technical Report.

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

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

Dated this 16th day of March, 2018

(Signed and Sealed) "*José Texidor Carlsson*"

José Texidor Carlsson, M.Sc., P.Geo.

HUGO M. MIRANDA

I, Hugo M. Miranda, ChCM (RM), as an author of this report entitled "Technical Report on the Siembra Minera Project, Bolivar State, Venezuela" prepared for GR Engineering (Barbados), Inc., and dated March 16, 2018, do hereby certify that:

1. I am a Principal Mining Engineer with RPA (USA) Ltd. of 143 Union Boulevard, Suite 505, Lakewood, Colorado, USA 80228.

2. I am a graduate of the Santiago University of Chile, with a B.Sc. degree in Mining Engineering in 1993, and a Masters of Business Administration degree in 2004. I'm also a graduate of the Colorado School of Mines with a Master of Engineering (Engineer of Mines) degree in 2015.

3. I am registered as a Competent Person of the Chilean Mining Commission (Registered Member #0031). I am a Registered Member (#4149165) with the Society for Mining, Metallurgy, and Exploration (SME). I have worked as a mining engineer for a total of 23 years since my graduation. My relevant experience for the purpose of the Technical Report is:
   - Principal Mining Engineer - RPA in Colorado. Review and report as a consultant on mining operations and mining projects. Mine engineering including mine plan and pit optimization, pit design and economic evaluation.
   - Principal Mining Consultant – Pincock, Allen and Holt in Colorado, USA. Review and report as a consultant on numerous development and production mining projects.
   - Mine Planning Chief, El Tesoro Open Pit Mine - Antofagasta Minerals in Chile.
   - Open Pit Planning Engineer, Radomiro Tomic Mine, CODELCO – Chile.
   - Open Pit Planning Engineer, Andina Mine, CODELCO - Chile.

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 fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

5. I visited the Project on September 19, 2017.

6. I am responsible for parts of Section 16 and share responsibility with my co-authors for Sections 1, 2, 3, 24, 25, and 26 of the Technical Report.

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

8. I have had no prior involvement with the property that is the subject of the Technical Report.

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

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

Dated this 16th day of March, 2018

(Signed and Sealed) "*Hugo Miranda*"

Hugo M. Miranda, C.P.

KATHLEEN ANN ALTMAN

I, Kathleen Ann Altman, P.E., as an author of this report titled "Technical Report on the Siembra Minera Project, Bolivar State, Venezuela" prepared for GR Engineering (Barbados), Inc., and dated March 16, 2018, do hereby certify that:

1. I am Principal Metallurgist with RPA (USA) Ltd. of Suite 505, 143 Union Boulevard, Lakewood, Co., USA 80228.

2. I am a graduate of the Colorado School of Mines in 1980 with a B.S. in Metallurgical Engineering. I am a graduate of the University of Nevada, Reno Mackay School of Mines with an M.S. in Metallurgical Engineering in 1994 and a Ph.D. in Metallurgical Engineering in 1999.

3. I am registered as a Professional Engineer in the State of Colorado (Reg. #37556) and a Qualified Professional Member of the Mining and Metallurgical Society of America (Member #01321QP). I have worked as a metallurgical engineer for a total of 37 years since my graduation. My relevant experience for the purpose of the Technical Report is:
   - Review and report as a metallurgical consultant on numerous mining operations and projects around the world for due diligence and regulatory requirements.
   - I have worked for operating companies, including the Climax Molybdenum Company, Barrick Goldstrike, and FMC Gold in a series of positions of increasing responsibility.
   - I have worked as a consulting engineer on mining projects for approximately 15 years in roles such a process engineer, process manager, project engineer, area manager, study manager, and project manager. Projects have included scoping, prefeasibility and feasibility studies, basic engineering, detailed engineering and start-up and commissioning of new projects.
   - I was the Newmont Professor for Extractive Mineral Process Engineering in the Mining Engineering Department of the Mackay School of Earth Sciences and Engineering at the University of Nevada, Reno from 2005 to 2009.

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 fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

5. I did not visit the Siembra Minera Project.

6. I am responsible for Sections 13 and 17 and share responsibility for Sections 1, 18, 20, 21, 24, 25, 26, and 27 of the Technical Report.

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

8. I have had no prior involvement with the property that is the subject of the Technical Report.

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

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

Dated this 16th day of March, 2018

(Signed and Sealed) "*Kathleen Ann Altman*"

Kathleen Ann Altman, P.E.

GRANT A. MALENSEK

I, Grant A. Malensek, P.Eng., P.Geo., as an author of this report entitled "Technical Report on the Siembra Minera Project, Bolivar State, Venezuela" prepared for GR Engineering (Barbados), and dated March 16, 2018, do hereby certify that:

1. I am Principal Engineer - Valuations with Roscoe Postle Associates Inc. of Suite 505, 143 Union Boulevard, Lakewood, CO, USA 80227.

2. I am a graduate of University of British Columbia, Vancouver Canada in 1987 with a Bachelor's degree in Geological Sciences. In addition, I have obtained a Master of Engineering in Geological Engineering from the Colorado School of Mines in 1997 and a Graduate Business Certificate in Finance from the University of Denver – Daniels College of Business in 2011.

3. I am registered as a Professional Engineer/Geologist in the Province of British Columbia (Licence# 23905). I have worked as a mining engineer/geologist for a total of 22 years since my graduation. My relevant experience for the purpose of the Technical Report is:
   - Numerous mining project technical-economic modeling assignments.
   - Review and report as a consultant on numerous mining projects for due diligence and regulatory requirements
   - I have worked for operating entities, including Rio Tinto Group, Freeport McMoRan Copper and Gold Inc., and Newmont Mining Company on a variety of exploration and advanced development projects as well as operations in a number of countries.

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 fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

5. I did not visit the Siembra Minera Project.

6. I am responsible for Sections 19 and 22 and collaborated with my co-authors on Sections 1 and 21 of the Technical Report.

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

8. I have had no prior involvement with the property that is the subject of the Technical Report.

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

10. At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the Technical Report sections for which I am responsible contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated 16th day of March, 2018

(Signed and Sealed) "*Grant Malensek*"
Grant A. Malensek, P.Eng., P.Geo

CONSENT OF QUALIFIED PERSON

April 5, 2018

I, Richard J. Lambert, P.Eng., do hereby consent to the public filing of the report titled "Technical Report on the Siembra Minera Project, Bolivar State, Venezuela" (the Technical Report), prepared for Gold Reserve Inc. and dated March 16, 2018, and to the use of extracts from, or the summary of, the Technical Report in the press release of Gold Reserve Inc. dated March 19, 2018 (the Press Release).

I also certify that I have read the Press Release and that it fairly and accurately represents the information in the Technical Report that supports the Press Release.

(Signed) "*Richard J. Lambert*"

Richard J. Lambert, P.Eng.
Principal Mining Engineer

RPA 143 Union Boulevard Suite 505 | Lakewood, CO, USA 80228 | **T** +1 (303) 330 0950 **www.rpacan.com**

Exhibit 99.8

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CONSENT OF QUALIFIED PERSON

April 5, 2018

I, José Texidor Carlsson, P.Geo. do hereby consent to the public filing of the report titled "Technical Report on the Siembra Minera Project, Bolivar State, Venezuela" (the Technical Report), prepared for Gold Reserve Inc. and dated March 16, 2018, and to the use of extracts from, or the summary of, the Technical Report in the press release of Gold Reserve Inc. dated March 19, 2018 (the Press Release).

I also certify that I have read the Press Release and that it fairly and accurately represents the information in the Technical Report that supports the Press Release.

(Signed) "*José Texidor Carlsson*"

José Texidor Carlsson, P.Geo.
Senior Geologist

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Exhibit 99.9

CONSENT OF QUALIFIED PERSON

April 5, 2018

I, Hugo Miranda, ChMC(RM), do hereby consent to the public filing of the report titled “Technical Report on the Siembra Minera Project, Bolivar State, Venezuela” (the Technical Report), prepared for Gold Reserve Inc. and dated March 16, 2018, and to the use of extracts from, or the summary of, the Technical Report in the press release of Gold Reserve Inc. dated March 19, 2018 (the Press Release).

I also certify that I have read the Press Release and that it fairly and accurately represents the information in the Technical Report that supports the Press Release.

(Signed) ***“Hugo Miranda”***

Hugo Miranda, ChMC(RM)
Principal Mining Engineer

RPA 143 Union Boulevard Suite 505 | Lakewood, CO, USA 80228 | **T** +1 (303) 330 0950 **www.rpacan.com**

CONSENT OF QUALIFIED PERSON

April 5, 2018

I, Kathleen A. Altman, Ph.D., P.E., do hereby consent to the public filing of the report titled "Technical Report on the Siembra Minera Project, Bolivar State, Venezuela" (the Technical Report), prepared for Gold Reserve Inc. and dated March 16, 2018, and to the use of extracts from, or the summary of, the Technical Report in the press release of Gold Reserve Inc. dated March 19, 2018 (the Press Release).

I also certify that I have read the Press Release and that it fairly and accurately represents the information in the Technical Report that supports the Press Release.

(Signed) "*Kathleen A. Altman*"

Kathleen A. Altman, Ph.D., P.E.
Principal Metallurgist

RPA 143 Union Boulevard Suite 505 | Lakewood, CO, USA 80228 | **T** +1 (303) 330 0950 **www.rpacan.com**

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CONSENT OF QUALIFIED PERSON

April 5, 2018

I, Grant A. Malensek, P.Eng. do hereby consent to the public filing of the report titled “Technical Report on the Siembra Minera Project, Bolivar State, Venezuela” (the Technical Report), prepared for Gold Reserve Inc. and dated March 16, 2018, and to the use of extracts from, or the summary of, the Technical Report in the press release of Gold Reserve Inc. dated March 19, 2018 (the Press Release).

I also certify that I have read the Press Release and that it fairly and accurately represents the information in the Technical Report that supports the Press Release.

(Signed) “*Grant A. Malensek*”

Grant A. Malensek, P.Eng.
Principal Engineer - Valuations