Coringa Project

NI 43-101 Technical Report
Coringa Project

State of Para, Brazil

Prepared for:

MAGELLAN
MINERALS LTD.

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Signature Date:
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Prepared by:

GRE
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ISSUANCE STATEMENT

This report is addressed to Magellan Minerals Ltd. ("Magellan"). Following the completion of a plan of arrangement in 2009, Chapleau Resources Ltd. (“Chapleau”) became a wholly owned subsidiary of Magellan. Chapleau is sole owner of the Coringa Property which is the basis of this report. Magellan is a reporting issuer in British Columbia, Alberta, Ontario and Nova Scotia, whose common shares are listed on the TSX Venture Exchange.

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CERTIFICATE of AUTHOR

I, Kevin J. Gunesch do hereby certify that:

1. I am currently employed as Principal Mining & Civil Engineer by Global Resource Engineering, Ltd at:

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2. I am a graduate of the Colorado School of Mines with a Bachelor of Science degree in Mining Engineering (2000).
3. I am a registered Professional Engineer in the State of Alabama (27448).
4. I have worked as a Mine Engineer for a total of 12 years since my graduation from university, as an employee as of several mining companies and as a consulting engineer.
5. 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.
6. I am responsible for the preparation of the technical report titled "NI 43-101 Technical Report Resource Estimate Update Coringa Project" with an effective date of December 6, 2012 (the "Technical Report") with specific responsibility for overall organization, preparation and review of this technical report. I conducted a personal 4.5-day visit of the subject property in January, 2012.
7. I have personally completed an independent review and analysis of the data and written information contained in this Technical Report.
8. As of the date of this certificate, to the best of my knowledge, information, and belief, this Technical Report contains all the scientific and technical information that is required to be disclosed to make this Technical Report not misleading.

9. I have not had prior involvement with the properties that are the subject of the Technical Report.
10. I am independent of the issuer applying all of the tests in Section 1.5 of National Instrument 43-101.
11. I have read National Instrument 43-101 and Form 43-101, and the Technical Report has been prepared in compliance with that instrument and form.
12. I consent to the filing of the Technical Report with any stock exchanges or other regulatory authority and any publication by them, including electronic publication in the public company files on the websites accessible by the public, of the Technical Report.

Dated this 6th day of December 2012

Kevin J. Gunesch (Signature)

Signature of Qualified Person

"Kevin J. Gunesch"

Print name of Qualified Person

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1.0 SUMMARY

1.1 Introduction

The Coringa Project is a shear-vein system, hosted in volcanic rocks from the Salustiano Formation (1.87 Ga) and in granites from the Maloquinha Suite (1.86 to 1.88 Ga) located in the Tapajos Gold Province. This Province was the site of a major gold rush by artisanal miners from the late 1970's until the late 1990's and had a historical production of at least 10 million ounces with actual production believed to be closer to 30 million ounces. Some 10 tonnes of gold has reportedly been produced from alluvial and primary sources at the Coringa property (Brazilian Department of Mineral Production - DNPM).

The Coringa Project covers an area of approximately 9 km by 11 km (9,900 ha). Diamond drilling has been completed on less than 20% (4,800 ha) of the total exploration permits and claims (23,754 ha) held by Magellan at Coringa. The Coringa Project is located at the center of eight (8) contiguous exploration permits or "alvaras." All the permits are held 100% by Chapleau Resources Ltd. which is a wholly owned subsidiary of Magellan. **Drawing 1** of **Annex 1** shows the project location and existing exploration permits.

From January 2007 to December 2008, Chapleau Exploração Mineral Ltda., carried out geological mapping, soil sampling, rock sampling, a first phase diamond-drilling program (1,774 m), a second phase diamond-drilling program (5,032 m), a third phase diamond-drilling program (1,979 m), an airborne magnetic-gradiometric-radiometric survey, and two IP surveys. Also in 2007, Chapleau placed a new exploration permit for 1,918 hectares for a total of 23,554 hectares.

In April 2009, Chapleau engaged Global Resource Engineering Ltd. (GRE) to complete a technical report of the gold resource contained at the Coringa Project to the standards of Canadian National Instrument 43-101. GRE completed this report in September 2009, "NI 43-101 Technical Report on the Coringa Project, Novo Progresso Brazil September 4th 2009." During the reporting process, Chapleau completed a plan of arrangement with Magellan Minerals Ltd. (Magellan) where it became a wholly owned subsidiary of Magellan. In December 2009, Magellan engaged GRE to complete a preliminary economic assessment (PEA) study based on the resource defined in the September 2009 technical report. GRE completed this report in April 2010, "NI 43-101 Technical Report, Preliminary Economic Assessment, Coringa Project, Novo Progresso Brazil, April 30th 2010."

In 2010, Magellan placed a new mining claim for 200 hectares making the total exploration permit and claim area 23,754 hectares. From January 2010 through December 2011, Magellan completed a fourth phase diamond-drilling program (15,051 m), several soil sampling surveys, and 14 trenches. In 2011, Magellan engaged GRE to complete an updated resource estimate on the property to include the drilling results from the fourth phase drilling program. The updated resource estimate was incorporated into the report in May 2012, "NI 43-101 Technical Report, Coringa Project, State of Para, Brazil, May 7th 2012." In July 2012, Magellan engaged GRE to update the Preliminary Economic Assessment (PEA) using the results of the updated resource. This report contains the PEA update.

1.2 Geology and Mineralization

The Coringa project area is located in the extreme southeastern part of the Tapajos Gold Province, which in turn is situated in the central part of the Amazon Craton and covers part of the Ventuari-Tapajos and Tapajos-Parima Provinces. The area is characterized by Proterozoic magmatism. The basement is formed by granite-gneisses of the Cuiú-Cuiú Complex, while other plutonic rocks are classified in the Creporizao Suite (1997 Ma), the Parauari Suite (1.89 Ga) and the Maloquinha Suite (1.88 Ga). The volcanic rocks widely distributed throughout the Province are grouped within the Iriri Group or Uatuma Supergroup. The Coringa Project area is underlain by granites, probably from the Maloquinha Suite (1.88 Ga). Overlying the granites, are rhyolites, from the Salustiano Formation (2.0 to 1.75 Ga).

Regional shears with a NW-SE orientation cut the crystalline units and many of the known gold occurrences are controlled by this tectonic event. Gold mineralization in the Tapajos is pluton related and shear hosted. Native gold occurs in quartz-carbonate-sulfides, and to a lesser extent gold is associated with disseminated sulfides. Pyrite is by far the dominant sulfide mineral, with subordinate sphalerite, chalcopyrite and galena. Host rocks are coarse grained, felsic (usually granitic) intrusives which are cut by fine grained andesitic dykes, although the minority of deposits is hosted in volcanic lithologies. Mineralization is frequently well developed at the contact between the dykes and the host intrusive. Although rare, post-mineral dykes and intrusive bodies do occur.

The main structural trend on the Coringa Property is northwest (310°) similar to the one seen on the regional scale. The main mineralized structure at Coringa is related to a set of 345° trending fault features interpreted as R-shears, related to the 310° regional trend and is hosted both in the granite and the rhyolite. The strike length of the main shear/vein system is about seven (7) km. There are at least four (4) more mineralized structures, and in total Coringa has mineralized structures extending over at least eleven (11) km strike length. The orientation of the structures varies from 310° to 345°. Dips are variable from 75° to the northeast and up to 85° to the southwest. The system has typical lenticular vein morphology of a shear hosted quartz vein, with three types of veins: extensional, fault vein fills, and stock work/sheeted veins. High-grade gold mineralization is associated with each vein type. The width of the veins or shears varies from 15 cm to 4.0 m with up to 15 m in respect the stock work/sheeted vein style of mineralization. The alteration is mainly sericitization (as wide as 0.5 m in granites, and 1 m in rhyolites), with some silicification and chloritization-hemtatization and generally occurs as haloes up to 10 meters in granites and 50 meters in rhyolites.

1.3 Exploration Program

The exploration program initially focused on determining drilling targets. These targets were identified through artisanal workings, geologic mapping, airborne geophysics, ground IP surveys, along with rock and soil sampling. A total of 4 drilling phases have been completed on targets identified at the project site for a total of 24,093 m of HQ core in 160 exploration holes. The first phase drilling (1,774 m in 22 holes) was for early stage exploration, and was drilled under the main “garimpos”, or artisanal workings. The second phase drilling (5,032 m in 44 holes) focused on further defining the resource in the Galena-Boca, Eloy-Juara-Mae de Leite, Serra, and Guaxebinha-Meio-Onza blocks. The third phase drilling (1,979 m in 15 holes) focused further in defining the resource in the Guaxebinha-Meio-Onza block, and tested two of the IP targets. A fourth phase of drilling (15,308 m in 79 holes) program proved the continuity of the Mae de Leite structure at depth

and along strike, as well as the continuity of the Meio Zone along strike to the north and south. In addition, the fourth phase drilling program targeted the lower portions of the two main zones, Serra and Meio, as well as a step-out drilling, and testing new targets Valdette, Demetrio, and Domingo.

1.4 Resource Estimation

The Mineral Resource estimate for the project was completed independently by GRE using Magellan's exploration database. Prior to completing the calculation, GRE verified the data contained within the database which included drill hole locations and orientations, original geologic logs, core stored onsite, sample Quality Assurance/Quality Control (QA/QC) procedures, sample chain of custody documentation, and assay results. A site visit was performed by Mr. Kevin Gunesch PE of GRE in May 2009 and again in January 2012. It is GRE's opinion that the data used for the resource calculation describes the deposit sufficiently to support this technical assessment of the resource.

A block model was developed using the drill-hole data, assay results, and geological structure of the deposit. Gold grades were estimated using the inverse distance squared method and were confined to the mineralized zones which are comprised of gold veins within the "shear" areas of the project. **Table 1.1** below presents the results of the gold resource estimation at two cutoff grades. Although the project contains significant values of silver, lead, copper, and zinc, these metals were not included in the resource estimation. The estimate below represents a substantial increase in the resource since 2009.

Table 1.1 Gold Resources, All Areas

Cutoff Grade Au gpt	Unit	1 gpt	2 gpt
Measured	tonnes	1,151,000	931,000
	grade gpt	6.81	8.06
	grams, Au	7,835,000	7,505,000
	troy ounces, Au	252,000	242,000
Indicated	tonnes	2,023,000	1,290,000
	grade gpt	4.75	6.63
	grams, Au	9,613,000	8,551,000
	troy ounces, Au	309,000	274,000
Measured & Indicated	tonnes	3,174,000	2,221,000
	grade gpt	5.50	7.23
	grams, Au	17,448,000	16,056,000
	troy ounces, Au	561,000	516,000
Inferred	tonnes	5,511,000	2,720,000
	grade gpt	3.01	4.67
	grams, Au	16,614,000	12,714,000
	troy ounces, Au	534,000	409,000

1.5 Preliminary Economic Assessment

The first preliminary economic assessment (PEA) was completed by GRE for the Coringa property in 2010 based on the 2009 resource estimate. See previous reports "NI 43-101 Technical Report on the Coringa Project, Novo Progresso Brazil September 4th 2009" and "NI 43-101 Technical Report, Preliminary Economic Assessment, Coringa Project, Novo Progresso Brazil, April 30th 2010." The second updated PEA was completed in 2012 and is the latest addition to this Technical Report. This updated PEA was commissioned to include the increased resource base from the May 2012 NI 43-101 report, leading to a significant change in the planned throughput, increasing it from 400 tonnes per day (tpd) to 750 tpd.

A detailed mine layout and sequence was completed to estimate the project cash flows on a yearly basis. The mine was divided into separate areas based on the 3 primary resource areas identified which contain 92% of the gold resource at the 4 gpt Au cutoff: Serra (41%), Meio-Come Quieto (MCQ) (33%), and Galena-Mae de Leite (GAMDL) (17%). The PEA considers mining 70% of the resource in these areas. Detailed mine layouts were completed for each resource block based on the surface topography and mineralized area above the cutoff grade. The mineralized area for each block was divided into discrete stopes. A mine sequence was developed for sequentially mining each resource block over the mine life with 11 active stopes during steady state production.

Metallurgical testing was completed on two composites (Serra and Meio), representing 75% of the mined resource. This testing indicates the ore is free milling with a 99% Au recovery for the Serra composite and 98% recovery for the Meio composite. Plant recoveries were estimated at 5% below the testing results or 94% and 93% respectively.

The following list of key parameters was used in the study:

- Cutoff Grade of 4 gpt Au
- Cut and Fill Mine Method
- Primary Mining Equipment
    - Slushers within the narrow 1.5 m wide stopes
    - Rubber tire equipment within the crosscuts and haulage levels
- Ramp Grade of 10%
- Production Rate of 750 tonnes per day (tpd) with 2 year ramp up period.
- Average Plant Recovery of 93.5%, based on current testing
- Average Stope Recovery of 94%
- Gold Price of $US 1,350/troy oz.

Capital costs were estimated for all facilities, mine equipment, preproduction development, permitting and working capital based on published average costs for 2011, 2012 manufacturer and vendor quotes, and professional experience. A detailed design was completed for a tailings storage facility (TSF) along with a subsequent capital estimate based on quantities and unit costs. Closure costs for the TSF were estimated based on this detailed design.

Operating costs were estimated on a unit basis for all mine operations by estimating equipment hours and materials for a set operating unit. Labour costs were estimated separately from unit operations costs by calculating the required number of employees for one man coverage and multiplying by the required number for each workforce. Labour costs were sourced from a Brazilian salary survey from Catho, an ISO 9001:2008 certified company, for the 3 quarter of 2012. The average operating costs over the life of the mine are shown in **Table 1.2 below**.

Table 1.2 Average Operating Costs

Average cost	$US
Operating cost per tonne mined	$106.35
Total cost per tonne mined	$163.99
Operating cost per ounce gold recovered	$530.94
Total cost per ounce gold recovered	$819.68

The results of the PEA study are presented **Table 1.3** below. This preliminary assessment is preliminary in nature. It includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves. There is no certainty that the results of the preliminary economic assessment will be realized. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

Table 1.3 Economic Model Results

Result Category	$US 1,000s
(Cumulative Cash Flow) NPV@0%	$157, 150
NPV@5%	$109,870
IRR	33%

1.6 Conclusions and Recommendations

The 2012 preliminary economic model indicates an internal rate of return (IRR) of 33% and a net present value (NPV) at a 5% discount rate of $US 109.9M. The project payback period is 2.9 years. The project is very robust and is still economic under any one of the following scenarios: either a gold price of $US 1,000/troy oz., a capital cost increase of 50%, or an operating cost increase of 50%. The preliminary assessment is preliminary in nature. It includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves. There is no certainty that the results of the preliminary economic assessment will be realized. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

The updated PEA confirmed the positive economics from the previous study and demonstrated the expansion potential of the project by incorporating additional defined resources along strike and dip from the fourth drilling campaign. The resource continues to be open along strike and at depth.

GRE concludes the following:

- A total of 24,093 m of HQ core in 160 exploration holes has been drilled on the Coringa Project since 2007. The density of the exploration drilling is adequate to estimate measured and indicated resources between most drill holes. The spacing between drill holes was increased during the 2011 drill program and focused on identifying additional inferred resources. This expanded spacing should be continued for further exploration along the vein system.
- The potential exists to expand resources through additional drilling along the strike and dip of the defined resource areas.
- The geostatistical analysis completed on the Serra resource block indicates a variogram range of 100 m. This range was used to define measured and indicated resources. A 75 m search distance and minimum 2 assay intercept limit was used to define measured resources. A 100 m search distance and minimum 1 assay intercept was used to define indicated resources. Infill drilling with the purpose of upgrading inferred resources should take into account these resource categorization parameters.
- The vein system is open both at depth and along strike. The Galena-Mae de Leite and Meio-Come Quieto resource blocks demonstrate that the shear vein zone is more or less continuous between blocks. It is probable that the shear system also continues along strike beyond the current extent of the resource blocks. There is also good potential to encounter additional parallel vein structures within the exploration area held by Magellan as demonstrated by the Serra and Demetrio resource blocks.
- A resource model to the standards of Canadian National Instrument NI-43-101 was completed for the Coringa Project defining an undiluted gold resource of 561,000 measured and indicated troy ounces of gold and 534,000 inferred troy ounces of gold at a 1 gpt cutoff grade capped at 50 gpt gold. The additional drilling completed during the fourth drilling program increased the previous resource by 108% in the Measured and Indicated Category and 444% in the Inferred category.
- The sample QA/QC program completed by Magellan produced representative samples of the deposit. Check assays completed by GRE on the onsite core show acceptable correlation with previous results. The QA/QC program meets industry standards with the insertion of 1 blank, 2 standards, and 0 to 3 duplicates every 42 samples with the exception of creating field duplicates. The blank samples consist of non-gold bearing granite sourced from a local quarry borrow and verify that sample contamination due to sample preparation did not occur.
- Creating duplicates onsite at a large particle size creates a bias in the samples and therefore does not meet the purpose of duplicate sampling. Field duplicates should be created by quartering the core along the principal structure to create the best duplicate sample. Laboratory duplicates prepared at a small particle size show excellent correlation for duplicate assay samples.
- A conventional carbon in leach (CIL) milling circuit with pre-aeration at a pH of 11, using hydrated lime, is an effective method for treating the Coringa ores. However due to the

presence of silver and overall high grade of combined gold and silver, a Merrill Crowe circuit may be more economical. Recent metallurgic test work (RDi, March 2010) performed on composite samples from the Serra block and from the Meio block produced laboratory gold recoveries of 99.0% and 97.7% respectively using the CIL approach.

- The PEA process flowsheet considered a Merrill Crowe circuit with three stages of crushing that will feed a ball mill to produce a P_{80} 200 Mesh product. Prior to leaching the pH would be adjusted with lime followed by aeration to reduce reagent consumption and improve recoveries. The aerated slurry would be leached with cyanide, the supernatant clarified and de-aerated, followed by zinc precipitation of the precious metals. The tailings stream would be detoxified to produce mine backfill.
- During the resource estimation, GRE noted that the data collected by Magellan was well organized and accurate and that the exploration camp was well run and has good facilities for continued exploration. These observations reflect the high quality of Magellan's exploration program. In addition, internal audits completed by Magellan on their sample chain of custody documents, illustrate their commitment to running a well-organized and comprehensive exploration program.
- Preliminary geochemical test work to evaluate acid rock drainage (ARD) potential indicates that a single rock type (altered vein deposits with quartz, pyrite, and galena) appears to have the highest ARD risk. In addition, the tailings from the processed ore will likely be potentially acid generating (PAG) material since the defined process flow diagram does not include oxidation of the sulfide minerals. However, it is also evident that some samples have acid neutralizing potential. As a result, it appears that the Coringa project may have ARD potential, but also the ability to control ARD with the acid neutralizing potential inherent in some of the waste rock. Additional studies are required to substantiate this hypothesis.
- An underground mine would be the most efficient and economical method for mineral extraction for the majority of the resource blocks since it would provide a means to selectively mine the higher grade vein zones. Additionally, an underground mine would minimize surface impacts. The Valdette resource block is significantly lower grade with more disseminated mineralization. A surface mine would be more applicable to the mineralization type and grade present in Valdette.
- The overhand cut and fill mining method was selected for the PEA study due to its prominence as a preferred mining method, the ability to provide a high degree of grade control, the lower preproduction development requirements, and the fact that it is more applicable to the untested ground conditions at Coringa. The method also allows the backfilling of stopes throughout the mine life with a tailings slurry, reducing external tailings facility storage requirements. The backfilled stopes have the potential to reduce acid rock drainage (ARD) potential relative to other mining methods that leave open or caved stopes.
- Mine production was scheduled at 750 tpd, allowing for ramp-up to full production during the first two years. Rubber-tired (LHD and truck) haulage was selected based on its predominance in the industry at present and to provide a conservative estimation of project economics. Mine access would be via ramp systems daylighting at valley portals for each resource area.

- A portion of the process tailings would be used as backfill underground with the remainder being placed within a thickened tailings impoundment. The tailings impoundment includes an earth fill embankment allowing for construction without the need for mine development rock. The feasibility of the unlined impoundment assumed for the PEA study must be demonstrated by field investigation and additional analyses.
- Project initial capital costs for the plant complex, mine equipment, surface and underground facilities, preproduction development, permitting, additional engineering studies, site investigations, and working capital are estimated at $US 64.5M. Sustaining capital including additional mine equipment, required rebuilds, and new portal facilities totals $US 18.5M. Concurrent development requirements total $US 26.9M over the mine life. Closures costs are estimated at $US 3.1M. Total project capital is estimated at $US 115.1M.
- The project workforce was estimated separately for mine operations, mine maintenance, mill operations, and salary personnel. Labour rates reflect a recent salary survey from an ISO 9001:2008 Brazilian company. A total workforce of 309 hourly employees and 22 salary employees is estimated for the mine and mill operations during full operations.
- Operating costs are estimated at $US 106 per tonne mined ($US 531 per ounce gold recovered). Controllable operating costs were estimated for each mining operation based on the materials and equipment required to complete each unit operation such as stope production, crosscut development, surface ore haulage, etc. Non-controllable operating costs included taxes, royalties, permit fees, and G&A expenses.
- Total costs (including capital and operating) are estimated at $US 164 per tonne mined ($US 819 per ounce of gold recovered).
- The preliminary economic model includes a 21% contingency on capital costs and a 15% contingency on operating costs.
- The preliminary economic model indicates a robust IRR of 33% and a NPV at a 5% discount rate of $US 109.9M. The project payback period is 2.9 years. This preliminary assessment is preliminary in nature. It includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves. There is no certainty that the results of the preliminary economic assessment will be realized. Mineral resources that are not mineral reserves do not have demonstrated economic viability.
- Three separate sensitivity analyses were completed on the economic model. The project is very robust and is still economic under any one of the following scenarios: either a gold price of $US 1,000/troy oz., a capital cost increase of 50%, or an operating cost increase of 50%.

In light of the positive results of the updated PEA, Magellan should advance the project, initially completing site investigations to gather data required for additional studies. The following sections list the recommendations for data acquisition and completion of further engineering studies along with the associated cost for each.

Data Acquisition– Total Cost $US 2,240,000

The following items are recommended as part of the data acquisition process. They are listed below along with their estimated cost.

- Resource definition drilling to upgrade targeted sections from inferred resources to measured and indicated resources.
    - Estimated Cost $US 600,000
- Update the resource model with the results of the resource definition drilling. The geostatistical analysis used in the resource categorization should be used to define the appropriate drill hole spacing.
    - Estimated Cost $US 50,000
- Condemnation drilling in the proposed areas for the tailings storage facility, plant site complex, and camp area.
    - Estimated Cost $US 200,000
- Complete a topographic update of the property with satellite photographic topography that would tie into the existing LIDAR topography. The areas for the Valdette, Come Quieto, and the access route to the project should be surveyed.
    - Estimated Cost $US 15,000.
- Additional metallurgical testing should be completed to increase the sample density in relation to spatial, mineralogical, and grade distribution of the mineralized zones as well as test the new areas of the resource. Additional work is also required to define the optimum recovery method as both carbon leaching and Merrill Crowe processes are applicable to the project. Perform the additional test work to determine the items listed below:
    - Bond Work Index
    - Abrasion index
    - Testing of high-grade and low-grade ores to confirm recoveries similar to the composite testing
    - Confirm the samples tested
    - Grind size optimization
    - Define pre-aeration conditions
    - Leach conditions, time and reagent concentrations
    - Complete filtration and settling tests
    - Tailings detoxification for placement as underground fill during the mining cycle
        - Estimated Cost for all above metallurgical testing $US 200,000.
- Perform hydrogeological testing to determine the availability of well water as a source for plant makeup water and other freshwater project needs. Two streams within a few kilometers of the proposed plant site appear to be viable alternative sources of freshwater. However, potential seasonal stream flow variations, permitting requirements, the complexity of pioneering access through the jungle, power supply issues, and the probable need to clarify and/or treat the water prior to use indicate groundwater supply wells are the preferred alternative. Establishing the feasibility of groundwater well supply could be critical

to the project timeline because developing the surface water alternatives may be a lengthy process.

  - Estimated Cost $US 150,000

- Perform an assessment of the project area groundwater compliance points and install monitoring wells.

  - Estimated Cost $US 175,000

- Perform geochemical, rheological, and physical property testing on remaining sample material from the April 2010 metallurgical testing. The current assumptions for the tailings properties are inferred from the geology, from testing performed on waste rock samples, and from industry experience. Characterization of the tailings to be impounded is critical for detailed design of the tailings facility, including establishing the feasibility of an unlined impoundment. The testing will also provide a better understanding of potential water treatment requirements. A comprehensive program to define the geochemical properties of the waste material should be completed which includes expanding the current testing already completed and completing onsite kinetic testing.

  - Estimated Cost $US 150,000

- Complete a mine dewatering study using the planned infill drill holes for upgrading targeted resource areas from inferred to measured and indicated. This study will evaluate the characteristics of the hanging wall and footwall to determine mine dewatering requirements.

  - Estimated Cost $US 100,000

- Completing a structural geologic mapping of the tailings facility to determine preferential fault, joint, and bedding planes that should be targeted during the site investigation programs.

  - Estimated Cost $US 25,000

- Initiate tailings facility geotechnical and hydrogeology site investigation programs. This is critical to establish the feasibility of an unlined tailings impoundment. Permitting of the tailings facility is expected to be lengthy process, and as such it is recommended that this work commences as soon as possible to ensure the overall project timeline is met.

  - Estimated Cost $US 300,000

- Complete a borrow materials characterization to determine the suitability and quantity of construction materials for the tailings storage facility.

  - Estimated Cost $US 100,000

- Complete a plant site geotechnical investigation to verify the suitability of the proposed plant site and determine the foundation requirements.

  - Estimated Cost $US 75,000

- Implement an environmental baseline data collection program which will include site climate data, groundwater quality, and surface water quantity and quality. The data from

the program will be important for permitting and is necessary to establish the pre-development site conditions.

  - Estimated Cost $US 50,000

- Perform bathymetric surveys of the historic garimpos to support detailed mine design and dewatering studies.

  - Estimated Cost $US 50,000

Additional Engineering Studies - $US 1,000,000

The following recommendations are the engineering analysis items for further study. The total cost estimate for all studies is $US 1,000,000.

Recommendations are:

- Update the site climate data and analyze the surface water flow measurements to determine the site storm parameters and surface water base flow data.
- Refine all capital cost estimates by sourcing hard vendor quotes with all duties, taxes, and transportation costs to the site included. Complete new estimates for capital items that were previously estimated as lump sums by estimating each component separately. Add detail estimates for engineering procurement and construction management (EPCM) for all fixed capital.
- Refine all operating costs by adding greater detail to material quantities, material costs, creating a reagent schedule for the process plant.
- Complete a geotechnical study on the existing core to estimate the mine support requirements.
- Complete a mass balance study for the process plant and detailed process plant design.
- Refine TSF discharge water treatment costs by completing a detailed (monthly) water balance for the tailings impoundment as well as creating a detailed treatment plant design.
- Perform a detailed power study to define the capital and operating costs of line power supply. Although capital costs for bringing national electrical grid service to the project will be high, the PEA indicates diesel electric generation will be prohibitively costly with the planned production rate. Contract negotiations with power companies will be required for a definitive evaluation of this alternative.
- Perform trade off studies to evaluate other mining methods and haulage options which may include quantifying potential cost savings and determining the best option.
- Conduct an expanded geochemical engineering analysis of all major rock types, mineralization types, and tailings. This analysis will predict the tailings impoundment discharge water quality that is a critical input to the water treatment system design.
- Create a predictive model of the local groundwater system to determine mine dewatering requirements.
- Develop a 3D mine model to quantify all mining and development areas. Perform a cutoff grade sensitivity analysis to determine the optimal cutoff grade and mine configuration.

Determine the process cutoff grade to be used for mineralized development rock that must be mined to provide access to the defined stopes.

- Optimize the layout of all facilities based on the new LIDAR topography.

2.0 INTRODUCTION

This report is addressed to Magellan Minerals Ltd. ("Magellan"). Following the completion of a plan of arrangement in 2009, Chapleau Resources Ltd. (“Chapleau”) became a wholly owned subsidiary of Magellan. Chapleau is sole owner of the Coringa Property which is the basis of this report. Magellan is a reporting issuer in British Columbia, Alberta, Ontario and Nova Scotia, whose common shares are listed on the TSX Venture Exchange.

The Coringa property was first visited by Chapleau geologists in May 2006 and on August 14, 2006, Chapleau Resources Ltd. signed an agreement with Tamin Mineração Ltda., a Brazilian company controlling the mineral rights to the area, to acquire an undivided 100 percent interest in the 19,093 hectares claim block (two exploration permits or “alvaras” (850567/1990 of 10,000 hectares and 850568/1990 of 9,093 hectares cover the area) which is located in the Tapajos Gold Province. The final payment for these minerals rights was completed in March 2010.

During 2006, four new exploration permits were placed by Chapleau Exploração Mineral Ltda., a subsidiary of Chapleau Resources Ltd., covering an additional 6,507 hectares. The 6,507 hectares were subsequently reduced due to overlapping of areas for a total of 21,636 hectares.

From January 2007 to December 2008, Chapleau Exploração Mineral Ltda., carried out geological mapping, soil sampling, rock sampling, a first phase diamond-drilling program (1,774 m), a second phase diamond-drilling program (5,032 m), a third phase diamond-drilling program (1,979 m), an airborne magnetic-gradiometric-radiometric survey, and two IP surveys. Also in 2007, Chapleau placed a new exploration permit for 1,918 hectares for a total of 23,554 hectares.

In April 2009, Chapleau engaged Global Resource Engineering Ltd. (GRE) to complete a technical report of the gold resource contained at the Coringa Project to the standards of Canadian National Instrument 43-101. GRE completed this report in September 2009, “NI 43-101 Technical Report on the Coringa Project, Novo Progresso Brazil September 4th 2009.” During the reporting process, Chapleau completed a plan of arrangement with Magellan Minerals Ltd. (Magellan) where it became a wholly owned subsidiary of Magellan. In December 2009, Magellan engaged GRE to complete a preliminary economic assessment (PEA) study based on the resource defined in the September 2009 technical report. GRE completed this report in April 2010, “NI 43-101 Technical Report, Preliminary Economic Assessment, Coringa Project, Novo Progresso Brazil, April 30th 2010.”

In 2010, Magellan placed a new mining claim for 200 hectares making the total exploration permit and claim area 23,754 hectares. From January 2010 through December 2011, Magellan completed a fourth phase diamond-drilling program (15,051 m), several soil sampling surveys, and 14 trenches. In 2011, Magellan engaged GRE to complete an updated resource estimate on the property to include the drilling results from the fourth phase drilling program. The updated resource estimate was incorporated into the report in May 2012, “NI 43-101 Technical Report, Coringa Project, State of Para, Brazil, May 7th 2012.” In July 2012, Magellan engaged GRE to update the Preliminary Economic Assessment (PEA) using the results of the updated resource. This report contains the PEA update.

3.0 RELIANCE ON OTHER EXPERTS

The authors are not qualified to provide comprehensive opinions regarding the legal, environmental, and political issues that may affect the property. All comments set forth in this technical report that pertain to these issues are the opinion of the authors based on many years of experience in the minerals exploration and mining business and should not be relied upon except in that context.

The primary data source for resource estimate contained within this Technical Report is Magellan's exploration database. The database was compiled under the direction of Magellan's regional geologist Guillermo Hughes and Magellan's chief geologists Felix Huber and Leo Arrais. Sections 7, 8, 9, and 10 of this report rely on analysis and interpretation provided by Mr. Hughes, Mr. Huber, and Mr. Arrais; however, this information has been reviewed by GRE.

Mr. Hughes is a graduate of Geological Sciences from the Universidad Nacional de Sur-Argentina and a member of AusIMM. He has 27 years of experience in the mineral exploration industry. During his career he was underground exploration manager in the Huemules Gold Project, Argentina, Chief Geologist of the Geological Survey of Chubut Province in Argentina and General Director of Mines and Geology of the Chubut Province in Argentina. Mr. Hughes is member of the Association of Professional Geologist of Chubut Province, Argentina. Mr. Huber has 10 years of experience working in exploration geology in private companies involved in geophysical surveys and grass-root diamond and gold exploration. Mr. Arrais has 10 years of experience working in private companies involved industrial minerals production and gold exploration.

GRE has critically reviewed the all data provided by Magellan before incorporating it into this technical report.

4.0 PROPERTY DESCRIPTION AND LOCATION

4.1 Location

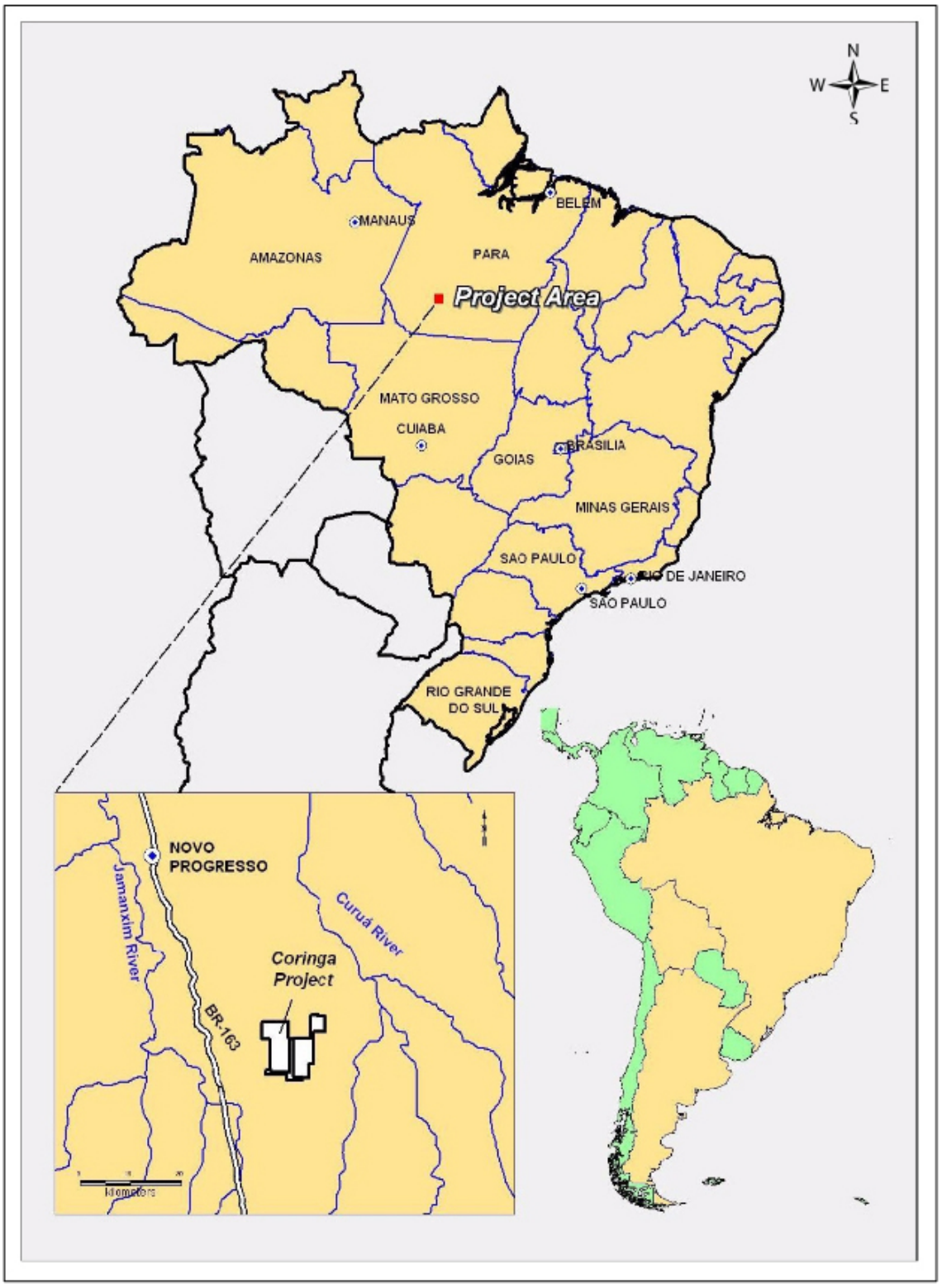
The Coringa project is located within the Amazon Basin, at an approximate latitude of 7° 32' S and longitude 55° 2' W (UTM coordinates 9,165,400N; 716,600E). The property is within the municipalities of Novo Progresso and Altamira, State of Para, 1,202 km NW of Brasilia, 1,984 km NNW of Sao Paulo, and 995 km SW of the Atlantic coast port city of Belem (**See Figure 4.1**).

Elevations range between 250 and 450 m a.s.l. Coringa has somewhat incised topography forming north-westerly trending ridges which rise up to 150 meters above the low-lying valleys (**See Photo 4.1**).

The project has a camp facility located in the center of the project area. Old artisanal surface workings are scattered throughout the project area.

Photo 4.1 Typical Topography

Figure 4.1 Project Location Map



4.2 Exploration Permits

In Brazil, priority is given to exploration license applications on a "first come, first serve" basis. The application must follow a DNPM (National Department of Mineral Production) format and must include a plan or map with the coordinates of the claim. Once the DNPM has evaluated an application, it is sent to Brasilia for publication and an "alvara de pesquisa" or exploration permit is awarded. The geographic limits of the exploration permits can be checked on the official web page.

(https://sistemas.dnpm.gov.br/SCM/extra/site/admin/dadosProcesso.aspx)

The Coringa Project is located at the center of eight (8) contiguous exploration permits or "alvaras" (23,754.44 ha) **(See Figure 4.2)**. All the permits are held 100% by Chapleau Resources Ltd. which is a wholly owned subsidiary of Magellan. **Table 4.1** shows a list of the exploration permits and claims that comprise the property along with their respective registration codes, areas, and legal status. Chapleau has submitted the exploration report of work completed during the past 3 years and the formal documentation to the DNPM office in Belem for the extension of the exploration permits ("alvaras") for an additional 3 years, and is presently awaiting notification of permit renewal.

Table 4.1 Coringa Property Exploration Permits or "Alvaras"

Claim DNPM	Area – Ha	Phase	Permit/claim date	Expiry Date	Renewal Status
850567-1990	10,000	Exploration Permit	28/09/2006	01/02/2014	-
850568-1990	9,092.69	Exploration Permit	14/12/2006	05/07/2014	-
850981-2006	637.56	Exploration Permit	13/12/2007	20/04/2014	-
850979-2006	381.41	Exploration Permit	13/12/2007	20/04/2014	-
850982-2006	347.64	Exploration Permit	13/12/2007	20/04/2014	-
850980-2006	1,177.07	Exploration Permit	13/12/2007	20/04/2014	-
850978/2007	1,917.64	Exploration Permit	16/09/2009	16/09/2012	Submitted, Pending DNPM approval
851161/2011	200.43	Claim	05/10/2011	-	-

4.2.1 Exploration Rights and Obligations

Rights:

- The right to perform the work necessary to define the deposits within the maximum allowable term of the license, which is three (3) years. It can be extended for no longer than another equal period, at the DNPM's discretion and in full compliance with the conditions stipulated by the Mining Code.
- To negotiate the deed. The holder may grant or transfer it before the approval of the final report of the exploration work, only requiring the prior consent of DNPM.
- The holder has a right to relinquish the title, without detriment to meeting the obligations arising from the Mining Code.

Obligations:

- The holder has an obligation to start exploration work no later than sixty days after the license has been published in the official gazette and to not stop or interrupt the work, without due reason, for more than three months or 120 non-consecutive days.
- The holder has an obligation to perform exploration work under the responsibility of a geologist or mining engineer whom is legally qualified in Brazil.
- The holder has an obligation to inform the DNPM of the occurrence of any other mineral substance not included in the license agreement, as well as the start or resumption of the exploration work and any possible interruptions.
- The holder has an obligation to perform the respective exploration work and submit a detailed report on the exploration work carried out, to DNPM, before the expiration of the license or its extension.
- The holder has an obligation to pay the Federal Government annual fees of R$ 3.83 per hectare.

4.3 Agreements

On August 14, 2006 Chapleau Resources Ltd. signed an option agreement with Tamin Mineraçao Ltda. to acquire a 100 percent interest in the 19,093 hectares which comprised the Coringa property. The exploration permits 850567/1990 of 10,000 hectares and 850568/1990 of 9,093 hectares applied to this agreement (**See Figure 4.2**).

Under the terms of the agreement, Chapleau paid a total of $US 1,640,000 and issued a total of 2,130,000 shares over a 3 year period to earn a 100% interest in the property. Payments under this agreement were completed in March 2010 and the option was exercised with Tamin Mineraçao Ltda. retaining a 2.5 % net smelter return (NSR) royalty on the property.

There are no other encumbrances on the project besides those pertaining to the 2.5% NSR held by Sandstorm Gold Ltd.

In May 2010, Magellan signed a farm-in agreement on an additional claim of 1,917 hectares located adjacent to and east of the Coringa exploration claims. Under the terms of the agreement Magellan paid $US 25,000 and issue 25,000 shares to the owner of the claim. Further payments of $US

165,000 and 225,000 shares are required over a three year period in order for Magellan to earn a 100% interest.

4.4 Surface Rights

Surface rights in Brazil are not associated with title to either a mining lease or a claim and must be negotiated with the landowner. The landowner's right to participate in the profit from a mine is documented in Article 11, letter b, from the Federal Mining Code. The text reads as follows: "The participation will be 50% of what is payable to the States, Municipalities, and Administrative Organism, as a Financial Compensation for the exploitation of a mineral resource". The Financial Compensation (CFEM) is calculated from the mineral sales value, minus taxes, transport costs, and insurances. The percentage of this "Financial Compensation" varies with the mineral; in the case of gold it is 1%.

In addition, Chapleau Resources Ltd. has signed an agreement with Mr. Benedicto Goncalves (on behalf of the landowners that are included in the Exploration Permits DNPM 850.567/90 and DNPM 850.568/90), which grants the right to carry on exploration and also the first option to obtain the surface rights. The amount agreed for this right is a monthly payment of $5,000 reais, starting on January 1st, 2007. The agreement ends once the exploration is finished. An agreement for the surface rights pertaining to DNPN 850.981/2006 is being pursued by Magellan.

4.5 General Description of the Property

The Coringa Project covers an area of approximately 9 km by 11 km (9,900 ha), and consists of ten main "zones" (**See Figure 4.3**). The majority of the recent exploration drilling has been focused on seven of these ten zones: Valdette, Mae de Leite, Serra, Guachebinha-Meio–Onca, Demetrio, Sr. Domingo, and Come Quieto zones. Most of this technical report focuses on these seven zones, with some discussion of the other exploration targets.

The remaining exploration area (13854.44 ha) surrounding the Coringa Project, has been subjected to limited exploration and includes an airborne geophysical magnetic, airborne gradiometric, radiometric and EM surveys, regional mapping, stream sediment sampling and some soil survey. No drilling has been conducted thus far.

Figure 4.2 Coringa Exploration Permit Map

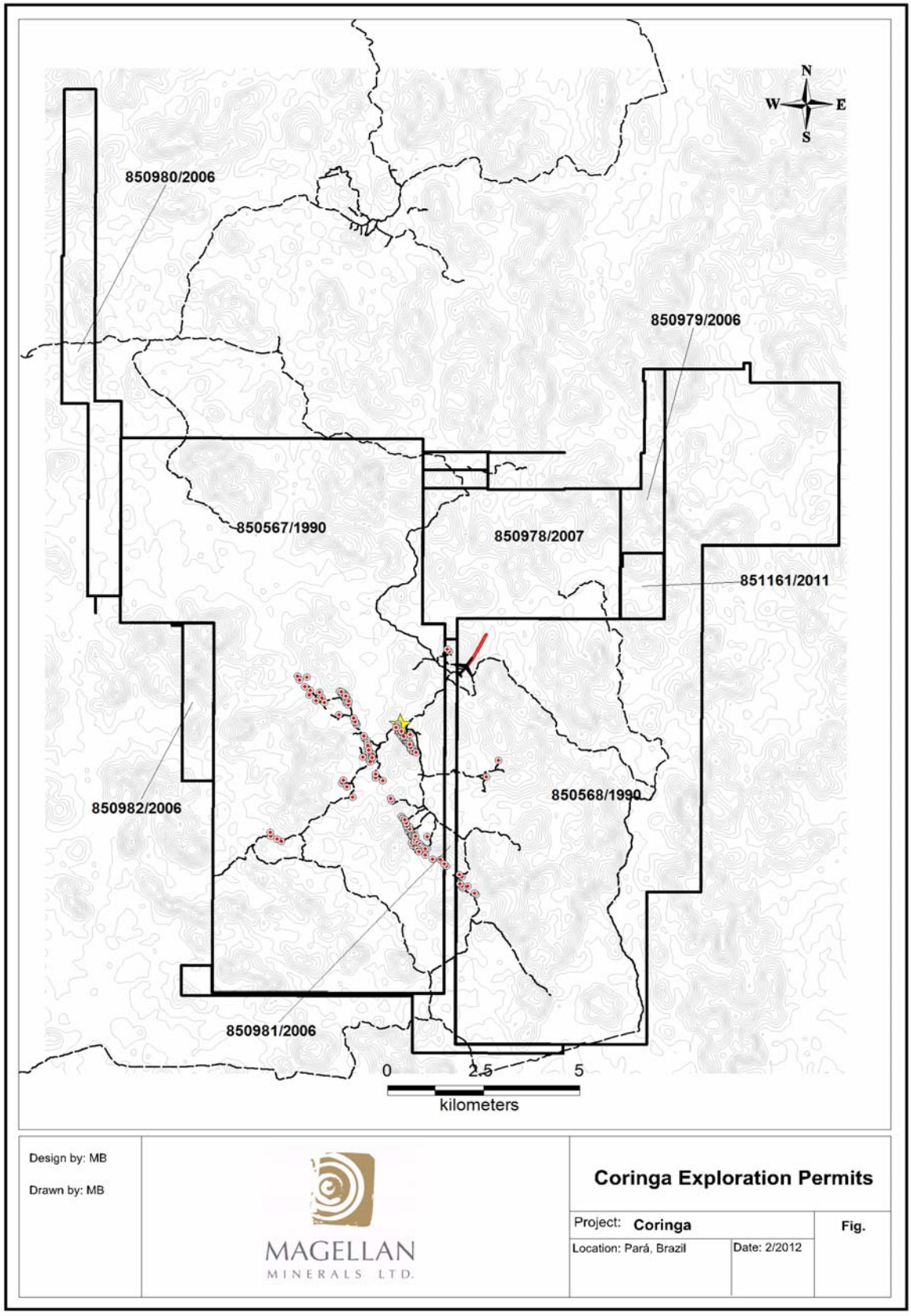
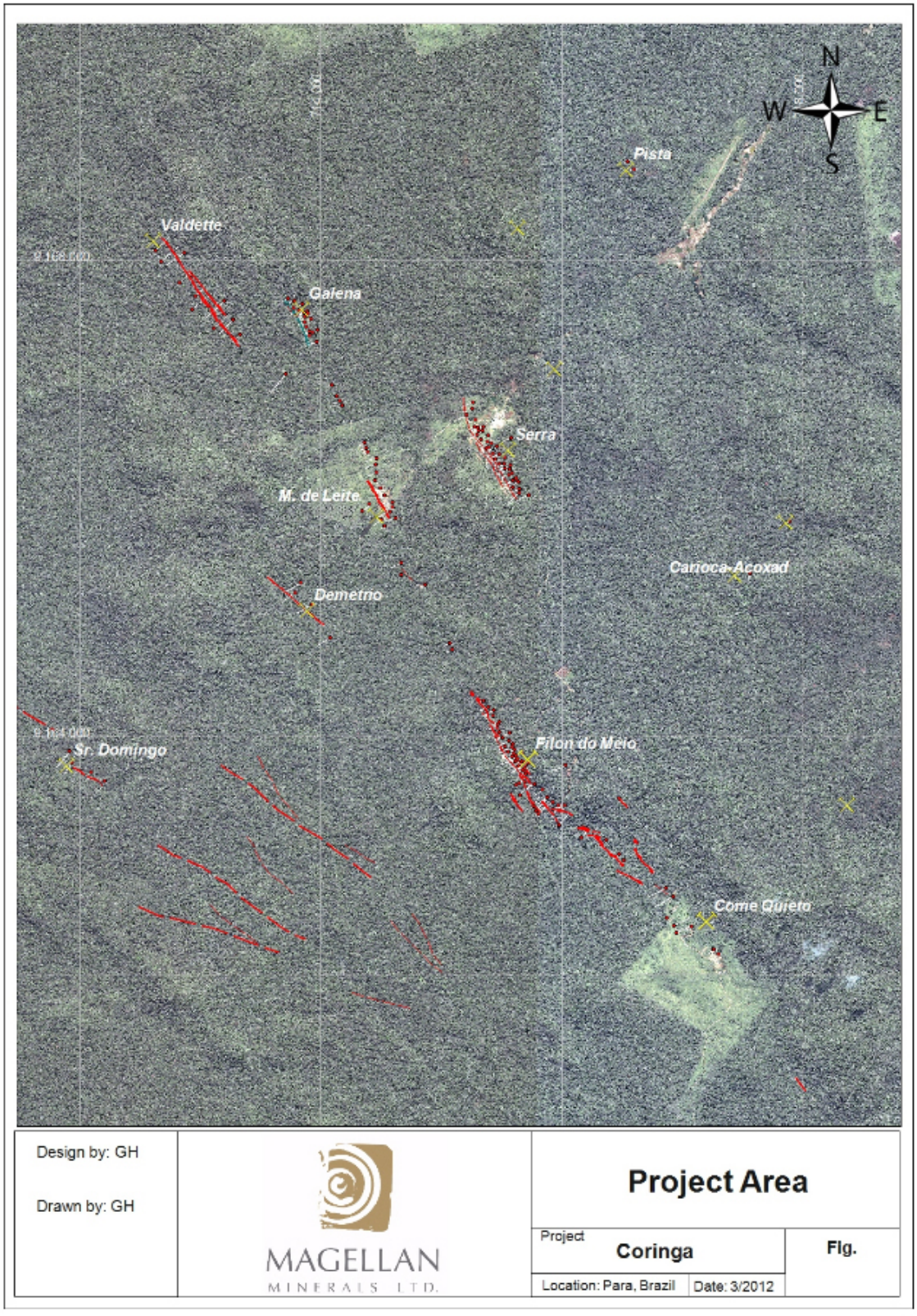


Figure 4.3 Coringa Project Area Map



4.6 Environmental Regulations

The Federal law 6938/1981 outlines the Environmental policy and the requirements of Environmental permits for any activity with contaminant potential, or involved with natural resources.

The Environmental permit is divided into three (3) stages/ categories:

- Previous Permitting (LP: "Licenca Previa"): this permit deals with the selection of the best place for developing the activity.
- Installation Permitting (LI: "Licenca de Instalacao"): this permit deals with the construction of the project, according to a technical project which has been previously approved.
- Operation Permitting (LO: "Licenca de Operacao"): this permit allows for the commencement of mining activities.

The environmental permitting process generally involves a six (6) month time period, from the time of the application to the approval or rejection of the permit.

If the required permit needs an Environmental Study (EIA/RIMA) where a Public Audience is requested, this approval period may extend to twelve (12) months. In the case of the Para State, this period is not legislated.

The National Environmental Council (CONAMA) has established a list of activities, which needs to be presented for a Previous Environmental Impact Study (EPIA/RIMA). Mining activities are included in that list. Generally speaking a Public Audience is requested.

The Environmental Permit needs to be renewed every 1 to 5 years. In the case of the Coringa Project, a permit application or "Term of Reference" was prepared and submitted to the Environmental Secretary of Para State (SEMA/PARA) for exploration and the construction of a decline, adit and/or shaft. This "Term of Reference" defined the scope of the Previous Environmental Study (EPIA/RIMA), which was carried out by Chapleau Resources Ltd, through Terra Meio Ambiente, an environmental consulting firm based in Belem, Para State. This original study was presented on January 12th, 2009, and approved for exploration (including diamond drilling) on August 5th, 2009. On March 10, 2010 the Pará State Environmental Authority (SEMA) granted Magellan's wholly owned Brazilian subsidiary Chapleau Exploração Mineral Ltda. the permit to extract 50,000 tonnes of ore for metallurgical studies at Coringa. This permit is valid for one year and does not allow for the installation of a plant to process any of the ore. In March 2012 an application was made for the renewal of the Environmental permit.

4.7 Environmental Liabilities

The main environmental liability of the project is an old milling area used by the artisanal miners which covers approximately 1.3 ha. The area was regraded and revegetated with tree seedlings by Chapleau in 2008; however, some of the tree seedlings did not survive. The surface regrade appeared stable during both site visits by Mr. Kevin Gunesch PE of Global Resource Engineering (GRE) in May 2009 and January 2012. The surviving seedlings experienced notable growth from 2009 to 2012. Some areas showed additional erosion due to storm runoff and were confined to small areas at the slope bottoms which were at a slightly steeper grade. This erosion could be controlled by decreasing the slope grade at the bottom to produce a concave slope and also

revegetation of the area with grasses. Acid Rock Drainage (ARD) was noted in the stream flowing through this area. The source of the ARD appears to be the exposed ore which was left at the site by the old miners. A scoping level geochemical characterization of the waste rock and tailings was completed for the PEA study which provides an initial estimate of the ARD risk for future mining operations.

In September 2008, local inspectors from the Brazilian Institute of Environment and Renewable Natural Resources (IBAMA) issued an assessment notice to Chapleau for a fine and work embargo on the basis that Chapleau was completing exploration activities without an environmental permit. Chapleau challenged this assessment notice with the Executive Manager of IBAMA in the local office of Santarem, arguing that the assessment was in error as a permit is only required when mineral extraction is performed. The executive manager of IBAMA in the Santarem office agreed with Chapleau's position and lifted the work embargo on the condition that mineralized ore is not removed in quantities that characterize extraction and that the exploration does not produce an environmental impact. Further to this, Chapleau applied for an "Operation Licence" to conduct further exploration activities on the Coringa project. The "Operation Licence" was granted to Chapleau on August 05, 2009. The current "Operation Licence" for exploration is valid until 2014. Although the formal administrative closure of the assessment notice provided to Chapleau for the fine is still pending, no further action by IBAMA on the matter is anticipated.

5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Accessibility

The project is located within 21 km of the main road, the BR-163 highway heading north from Cuiaba to Nova Progresso for some 1,100 kilometers. The road from Cuiaba to Nova Progresso is asphalted except for 130 km (**See Photo 5.1**). The BR-163 must be followed for some 1,020 kilometers before reaching the turn off to the east along a 38 kilometer farm trail to access the project, often in thick jungle (**See Figure 5.1**). The road from Novo Progresso south to the Coringa project area (some 80 kilometers to the turn off) is being paved.

Photo 5.1 Road, Novo Progresso to Cuiaba

5.2 Climate

The Coringa Project is located within the Amazon Basin where the dry season normally begins around late May and continues through to November, although there are occasional rains in June, October and November. Temperatures vary between a minimum of 17 degrees centigrade in June to a maximum of 44 degrees centigrade in January with average annual temperatures of 26 degrees centigrade. Temperatures are typically cooler in the high jungle where humidity is also constantly higher throughout the year. Average annual precipitation is between 1,500 and 2,000 mm.

Although the rains fall heavily from February to May, exploration work can be carried out on the project all year round.

Magellan has collected climate data from their onsite weather station since October 2008. **Tables 5.1 through 5.5** below summarize the data collected. The weather station was in disrepair from March 2010 through June 2011; no data was collected during that period.

Table 5.1 Climate Data 2008

Data	Oct*	Nov	Dec
Temperature, °C (Avg.)	26.2	24.9	24.9
Relative Humidity, % (Avg.)	75.1	84.4	85.2
Wind speed, m/s (Avg.)	0.99	0.79	0.62
Precipitation, mm (Total)	84.9	338.8	367.3

** - Partial Month*

Table 5.2 Climate Data 2009

Data	Jan	Feb	Mar	Apr	May	Jun	Jul*	Aug*	Sep*	Oct	Nov	Dec
Temperature, °C (Avg.)	25.2	24.9	24.8	24.9	24.7	24.4	24.6	25.5	26.2	26.2	26.5	ID
Relative Humidity, % (Avg.)	84.7	87.4	89.6	89.4	88.7	85.7	79.2	77.9	84.1	84.4	85.9	ID
Wind speed, m/s (Avg.)	0.04	0.09	0.26	0.60	0.20	0.19	0.35	0.30	0.03	0.08	0.14	ID
Precipitation, mm (Total)	273	401	332	243	169	187	3	85	32	229	123	ID

** - Partial Month, ID - Insufficient data*

Table 5.3 Climate Data 2010

Data	Jan	Feb	Mar*	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec
Temperature, °C (Avg.)	ID	27.8	27.6	ND	ND	ND	ND	ND	ID	ID	ND	ND
Relative Humidity % (Avg.)	ID	90.8	1.1	ND	ND	ND	ND	ND	ID	ID	ND	ND
Wind speed, m/s (Avg.)	ID	0.42	0.10	ND	ND	ND	ND	ND	ID	ID	ND	ND
Precipitation, mm (Total)	ID	424	52	ND	ND	ND	ND	ND	ID	ID	ND	ND

** - Partial Month, ID - Insufficient data, ND - No Data*

Table 5.4 Climate Data 2011

Data	Jan	Feb	Mar	Apr	May	Jun	Jul*	Aug*	Sep	Oct*	Nov*	Dec
Temperature, °C (Avg.)	ND	ND	ND	ND	ND	ND	24.7	27.3	ID	27	25.1	ID
Relative Humidity, % (Avg.)	ND	ND	ND	ND	ND	ND	69.4	65.3	ID	74.85	82.56	ID
Wind speed, m/s (Avg.)	ND	ND	ND	ND	ND	ND	1.52	0.73	ID	0.37	0.27	ID
Precipitation, mm (Total)	ND	ND	ND	ND	ND	ND	0	31	ID	221	252	ID

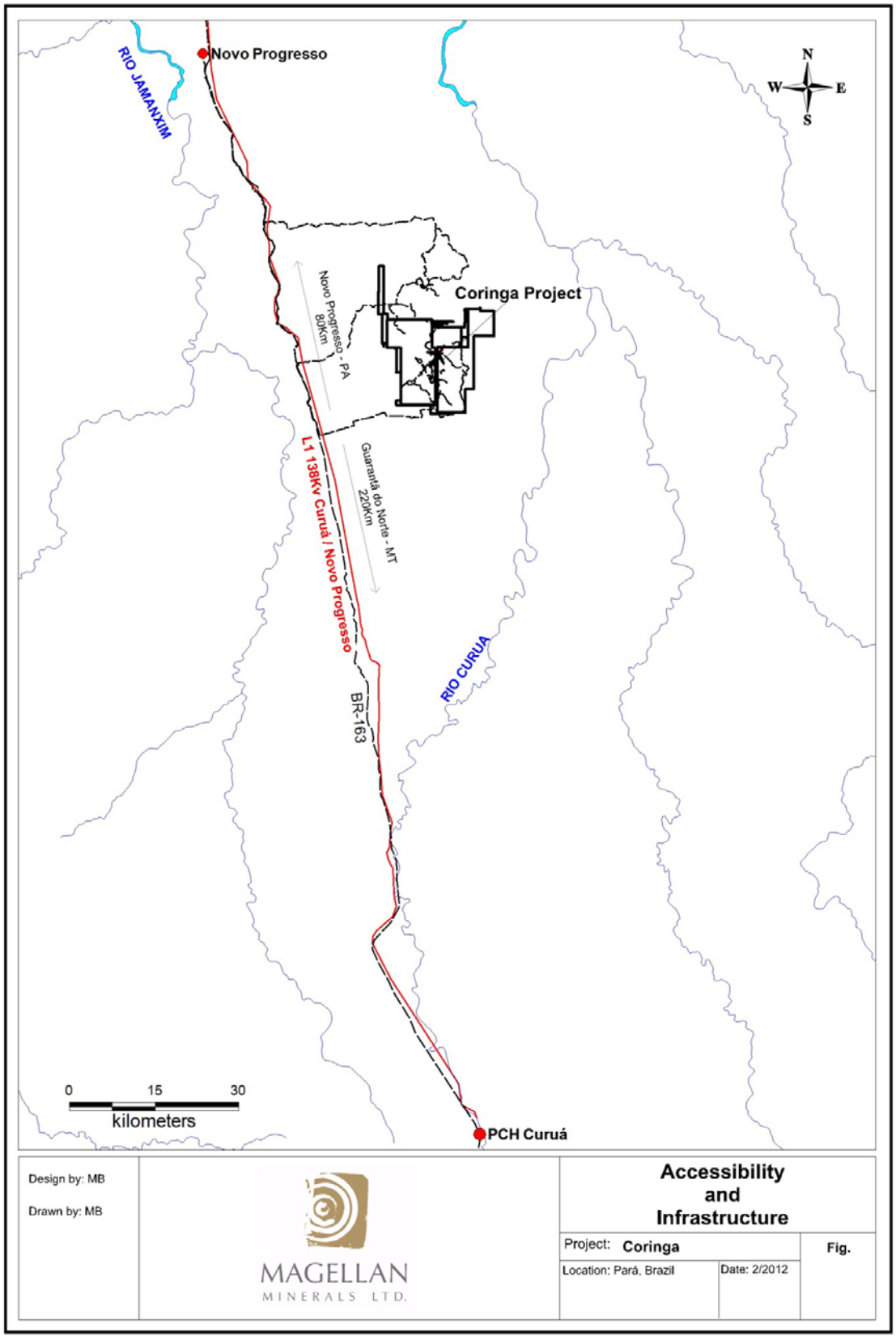
** - Partial Month, ID - Insufficient data, ND - No Data*

Table 5.5 Climate Data 2012

Data	Jan*	Feb*	Mar*	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov
Temperature, °C (Avg.)	24.1	23.9	24.8	ID	22.4	25.0	24.3	24.8	26.6	ID	ID
Relative Humidity, % (Avg.)	85.4	85.2	83.7	ID	90.1	81.3	75.3	68.3	73.8	ID	ID
Wind speed, m/s (Avg.)	0.25	0.16	0.14	ID	1.83	0.22	0.32	0.41	0.57	ID	ID
Precipitation, mm (Total)	410	165	185	ID	89	26	2.1	0	27	ID	ID

** - Partial Month, ID - Insufficient data, ND - No Data*

Figure 5.1 Accessibility and Infrastructure



5.3 Local Resources and Infrastructure

The town of Novo Progresso has a population of some 30,000 people and is the center for all basic fuel and food supplies. There are good hotels and an adequate internet connection in the town.

Light, single-motor planes are available for rental by the hour from the city of Novo Progresso. On the project, there is a 1,100 meter long laterite air strip located some 5 kilometers from the Coringa camp **(See Photo 5.2**).

Photo 5.2 Project Air Strip

Within the property, several cattle farms ("fazendas") have limited infrastructure. There is no national grid electricity in the project area, but a national grid high voltage power line has recently been completed from Guaranta do Norte to Novo Progresso which passes some 21 kilometers west of the project area **(See Figure 5.1 and Photo 5.3)**.

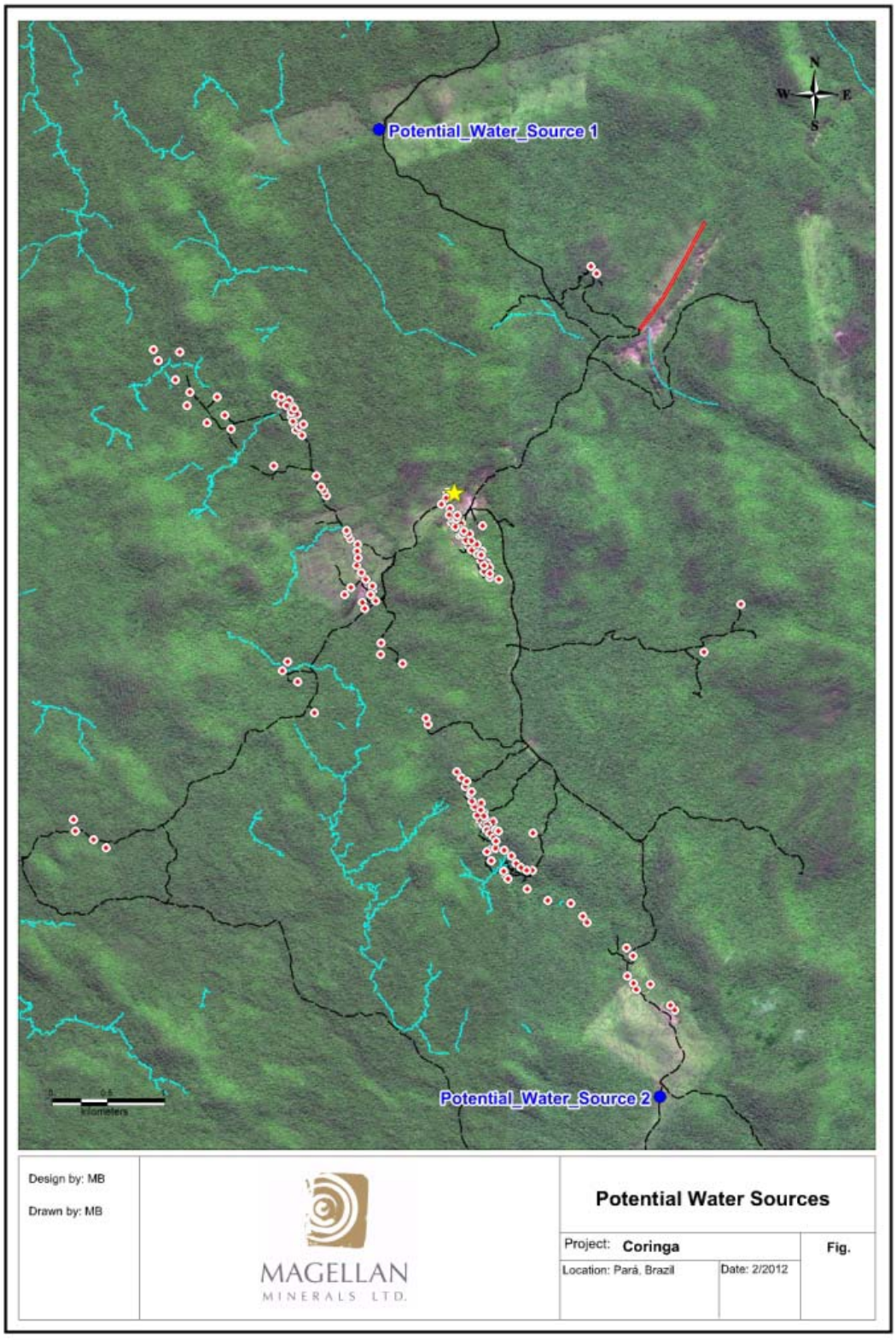
Photo 5.3 High Voltage Power Line near Coringa

Magellan's exploration camp, located on the Coringa farm is powered by a diesel generator that runs refrigerators, freezers, air conditioners in the office area and the television, which captures several channels via a satellite dish. The camp has a telephone, which is connected via a radio link to a fixed line in Novo Progresso. Basic internet is provided at the camp via satellite dish. The telephone provides communication between the Mato Velho and Coringa projects and a short wave radio system provides communication within the project area. The camp has sufficient space to house up to 70 people and a core shed has been constructed with capacity to store 30,000 meters of core. A water well provides the camp with drinking water and septic tanks and leach field provide for sewage waste disposal **(See Photo 5.4)**.

Two streams that flow throughout the year have been identified, (**See Figure 5.2**), as possible water sources for the process plant and camp facilities. In addition to these water sources, a tributary of the Curua River, located approximately 10 km east of the project site, could be used as a source of fresh water for mineral processing.

Photo 5.4 Coringa Exploration Camp

Figure 5.2 Potential Water Sources



5.4 Physiography and Flora

Coringa has steeply incised topography forming north-westerly trending ridges which rise up to 150 meters above the low-lying valleys. The valleys have a marked stream flow during the wet season and most dry up in the winter months (July to November). Vegetation over more than 98 percent of the project area is tropical jungle with the tree canopy reaching some 30 meters high in places (**See Photo 5.5**). The rest of the property has been cleared in the last 5 years and tropical grasses have been sown, which can be over two meters high. Within both the cleared areas and the areas of virgin jungle, the remains of the old workings of the "garimpeiros", or artisanal workers, can still be clearly seen, with many of them now flooded (**See Photo 5.6**). Alluvial gold workings on the river Coringa and in the many streams in the area along with primary gold workings on the four shear-vein structures are seen as linear trench-like excavations.

Photo 5.5 Typical Jungle Canopy

Photo 5.6 Old Garimpeiro Working, Now Flooded

6.0 HISTORY

The Tapajos Gold Province was the site of a major gold rush by artisanal miners from the late 1970's until the late 1990's and had a historical production of at least 10 million ounces of gold (Brazilian Department of Mineral Production - DNPM).

The Coringa project is located within the Tapajos Gold Province, and has produced gold via artisanal methods since the late 1970's. It is estimated that some 10 tonnes of gold have been produced from alluvial and primary sources on the property. At the height of mining activities in the 1980's, some 3,000 "garimpeiros", or artisanal workers, were on the property. No other exploration activities are known in the area, except for those completed by the garimpeiros. Gold was produced from both the many mineralized drainages which dissect the area and from shear-vein zones with deep saprolitic profiles where oxidation of the sulfides allowed the garimpeiros to mine down to 15 meters, using high pressure hoses to wash the clay-rich saprolitic rhyolite or simply by hand cobbing the high-grade oxidized surface portions of the shear-zones. Some deeper, more recent workings on the Mae de Leite vein go down as deep as 70 meters **(See Photo 6.1 and 6.2 and Figure 6.1)**.

Photo 6.1 Artisanal Working

Photo 6.2 Artisanal Working

With the devaluation of the monetary system in Brazil in 1991, the artificially high internal gold price in Brazil fell and the price of diesel oil rose substantially, causing an immediate reduction in mining activity in the area.

Joseph Haraoui initially staked the area in 1990 in the name of Tamin Mineraçao, his family business; no other exploration permits are known. On August 14, 2006 Chapleau Resources Ltd. entered into an option agreement to be able to earn a 100 percent interest in the property over a three-year period. The two exploration permits are part of an 180,000 hectare package over which Chapleau has first right of refusal **(See Figure 6.2)**.

Since the formation of the Brazilian subsidiary of Chapleau Resources Ltd in 2006, the property package has been expanded with the acquisition of four new exploration permits (850979/2006, 850980/2006, 850981/2006 and 850982/2006).

In September 2009, Magellan Minerals acquired all the issued and outstanding shares of Chapleau Resources. Upon completion of the plan of arrangement, Chapleau became a wholly-owned subsidiary of Magellan. In March 2010, Magellan completed the final payment to Tamin Mineracao Ltda., and earned a 100% interest in the property. (**See Section 4.3 Agreements**).

In May 2010, Magellan optioned a 100% interest in an additional claim of 1,917 hectares immediately adjacent to and east of the Coringa exploration claims (exploration permit 850978/2007) (**See Figure 4.2**). In 2011 Magellan staked an area of 200ha to the West of the project.

Figure 6.1 Locations of Artisanal Workings

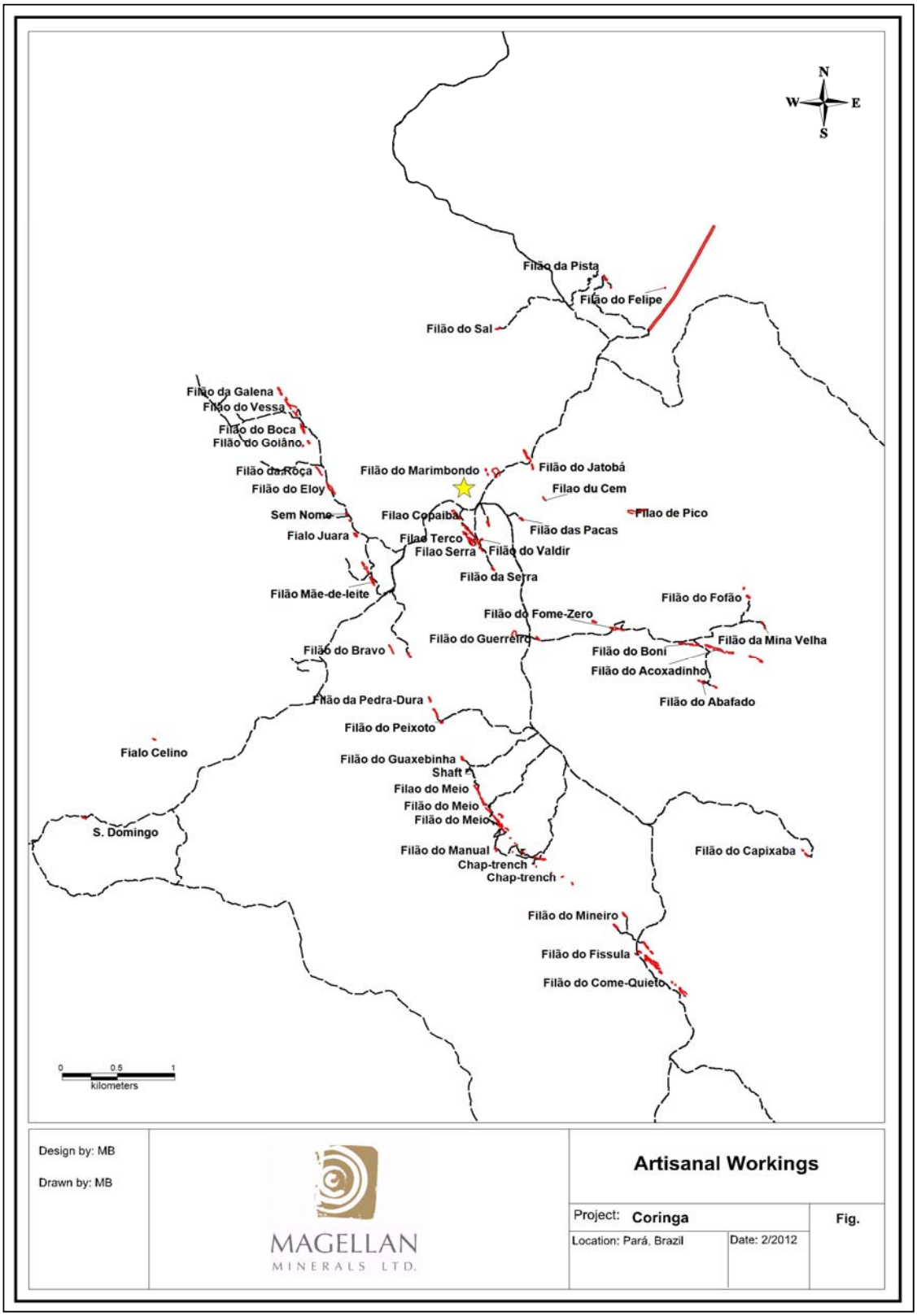
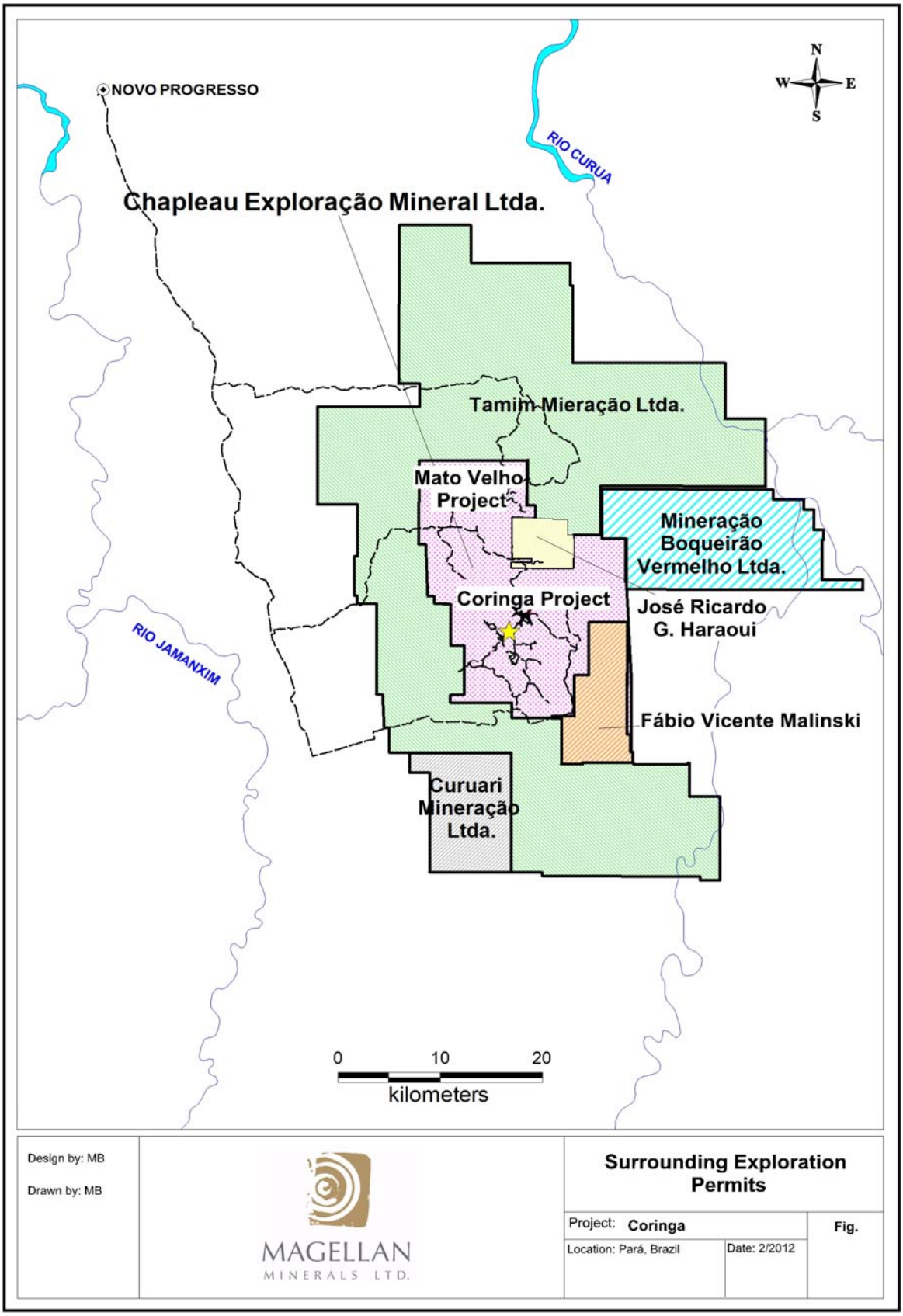


Figure 6.2 Surrounding Exploration Permits



7.0 GEOLOGICAL SETTING AND MINERALIZATION

7.1 Summary

The following description of the regional geology and district geology was taken mainly from work done by the Brazil Geological Survey (Geologia e Recursos Minerais da Folha Vila Riozinho). The description of the deposit geology and geological model was taken from information obtained from field mapping, drilling, and other similar gold occurrences such as Palito, Tocantizinho, Sao Jorge and Cuiú-Cuiú in the Tapajos Gold Province.

7.2 Regional Geology

The Coringa project area is located in the extreme southeastern part of the Tapajos Gold Province, which in turn is situated in the central part of the Amazon Craton and covers part of the Ventuari-Tapajos and Tapajos-Parima Provinces. The area is characterized by Proterozoic magmatism **(See Figure 7.1)**.

The basement is principally composed of granite-gneisses of the Cuiú-Cuiú Complex of 2015 Ma, while other plutonic rocks are classified as the Creporizão Suite (1997 Ma), the Parauari Suite (1890 Ma) and the Maloquinha Suite (1880 Ma). The Cuiú-Cuiú complex is intruded by granitoids of the Parauari Suite consisting of monzonite-granites and granodiorites, with little or no deformation. The Creporizao Suite includes syenite-granites, monzonite-granites, tonolites and granodiorites, all of which are generally deformed. The Maloquinha Suite is characterized by syenite-granites and alkaline–feldspar granites. The Maloquinha granite is sometimes intrusive into the Paruari Granite.

Volcanic rocks are widely distributed throughout the Province and are grouped within the Iriri Group. Two sequences are described which are both temporally and geochemically distinct. The Vila Riozinho Formation (2.0 Ga) is of calc-alkaline affinity, formed by intermediate and acid volcanics such as andesites, basalts, trachytes and rhyolites. The Moraes de Almeida Formation (1.88 Ga) has an alkaline affinity and is formed by ignimbrites, rhyolites and trachyites. The Salustiano Formation are rhyolites and rhyodacites, with and calc-alkaline affinity (2.0- 1.75 Ga).

The Iriri-Maloquinha igneous event is associated with a strong extensional period.

Mafic rocks from the Ingarana Suite and the Cachoeira Seca Suite are part of the Crepori diabase and are seen intimately associated with many of the mineralized structures in the area.

Regional shears with a NW-SE orientation cut the crystalline units and many of the known gold occurrences are controlled by this tectonic event such as Palito, Tocantinzinho, União, Coringa, and Mato Velho. Gold occurrences such as Cuiú-Cuiú, Tocantizinho, Mamoal and Palito are all aligned along this trend (Reconsult Geofísica - 2008), which has been previously described as the Tocantizinho Trend. These crystalline units offer many possibilities for gold and base metal mineral deposits and since the alluvial gold rush of the 1980s and 1990s, various types of primary mineralization have been identified.

Gold mineralization in the Tapajos is pluton related and shear hosted. Native gold occurs in quartz-carbonate-sulfides, and to a lesser extent gold is associated with disseminated sulfides. Pyrite is by far the dominant sulfide mineral, with subordinate sphalerite, chalcopyrite and galena. Host rocks are coarse grained, felsic (usually granitic) intrusives which are cut by fine grained andesitic dykes, although a minority of deposits is hosted in volcanic lithologies. Mineralization is frequently well developed at the contact between the dykes and the host intrusive, although rare, post-mineral dykes and intrusive bodies occur. It appears the deposits are genetically "magmatic-hydrothermal" as opposed to "orogenic/greenstone" types (D. Moore, ProExplo 2011 Lima Peru).

7.3 Coringa Property Geology

The Coringa Project area is largely underlain by intrusive bodies often seen as circular expressions within granites, probably of the Maloquinha Suite (1.88 Ga). In hand specimen, the granite has a typical granular texture, with pink feldspar and quartz crystals. In thin section the granite has a hypidiomorphic texture, with quartz (40%) and feldspar (55%) intergrowths. Most of the granite outcrops as rounded topographic highs **(See Figure 7.2)**.

Overlying the granites are rhyolites, probably from the Salustiano Formation (2.0 to 1.75 Ga). These have a typical porphyritic texture, with white phenocrystals, in a dark matrix. This matrix turns into a reddish colour when it is altered, close to the veins. The mineral composition varies from over 75% of sanidine crystals, to 50% of sanidine and 30% of plagioclase, in a matrix formed by quartz and sanidine crystals. Biotite is a minor constituent (5%), and is normally altered to chlorite. The rock is strongly magnetic (Universidad Nacional del Sur -2007).

Figure 7.1 Regional Geology

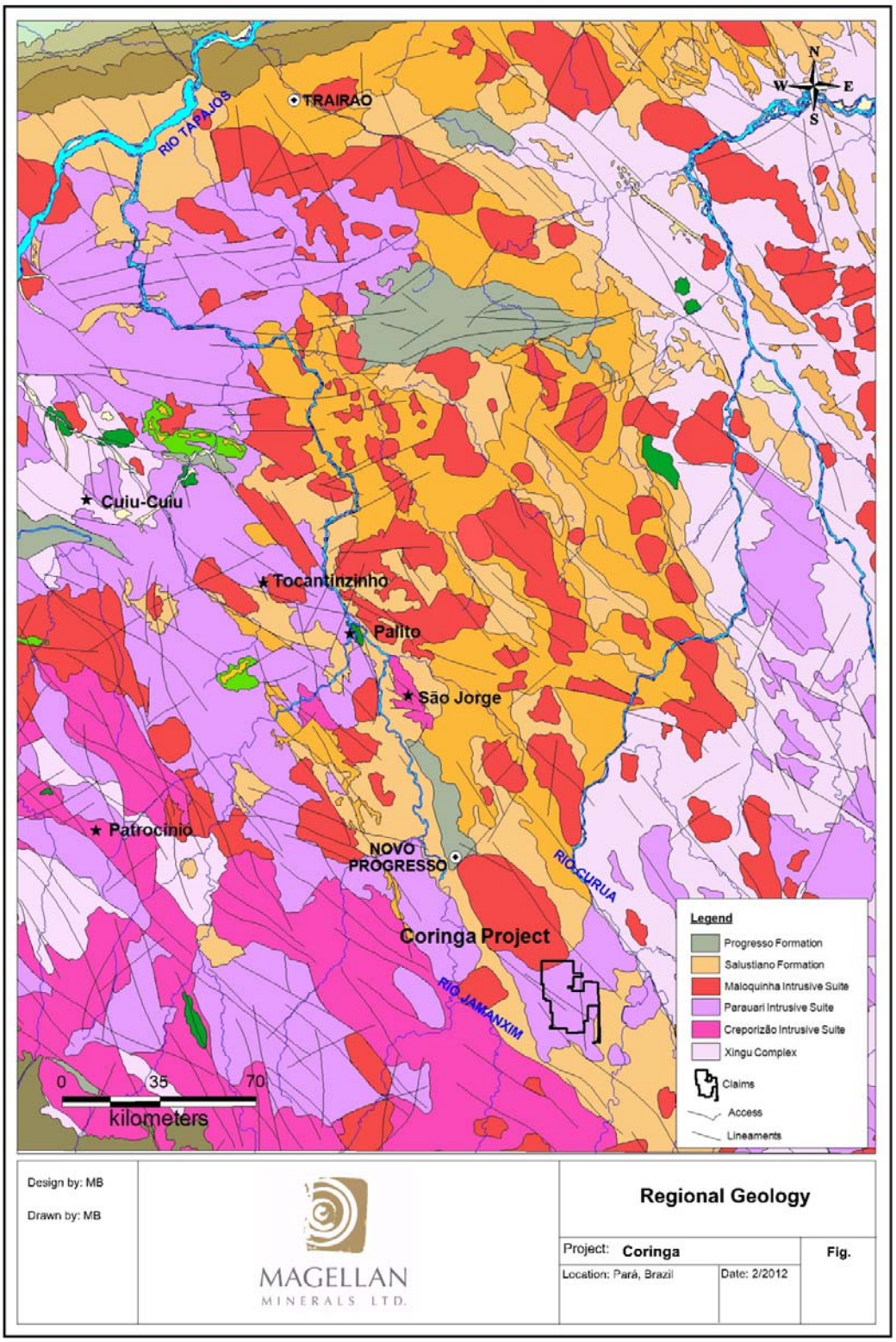
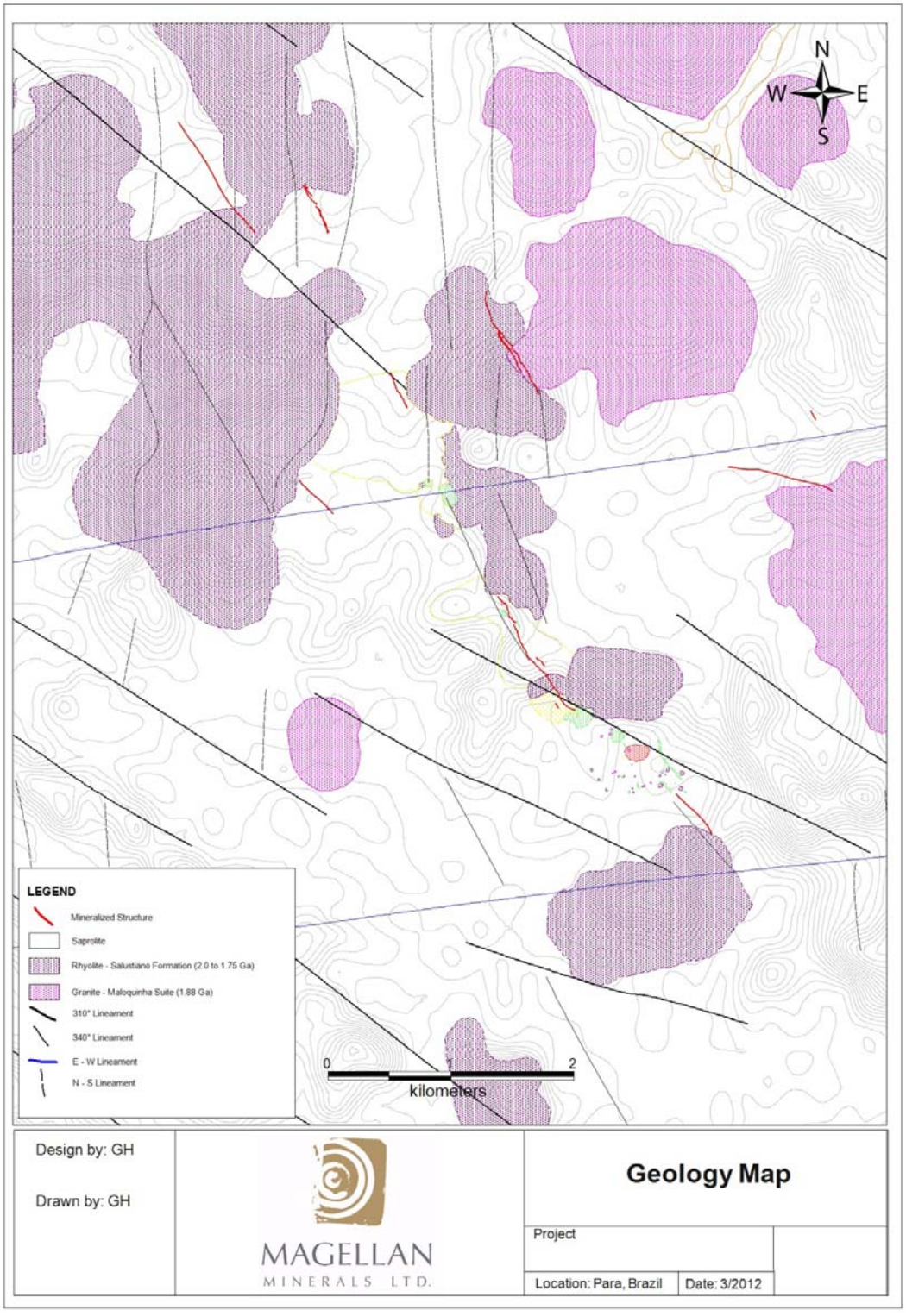


Figure 7.2 Geology Map



7.4 Structure

Since more than 90% of the area is covered by tropical forest, and almost 60% is covered by saprolite, structural interpretation is based on airborne geophysics (Buks Lubbe -Coringa Airborne Geophysics interpretation), and Ikonos satellite images.

The 310° regional structural trend is pervasive throughout the property. These regional structures are interpreted as strike slip faults **(See Figure 7.3)**, with probably locally a dextral (right lateral) sense of displacement. The main mineralized structure at Coringa is related to a set of 345° trending fault features, related to the 310° regional trend and could be interpreted as R-shears. Both features (310° and 345° structures) are cut by a younger East-West structure. Finally there is a set of North-South trending features that could be interpreted as faults, but could also represent contacts (intrusive granites / rhyolites). These features are cut by the East-West trending structure and the 310° regional trend. These North-South features are interpreted as the oldest structural features in the area.

7.4.1 Structural Model

Although the main structural feature at Coringa is the regional 310° trend, most of the known mineralized structures are slightly oblique to this structure (345° strike) **(See Figure 7.4)**. The veins dip vertically to sub vertically (75° to 90° dip), suggesting that they originated as strike-slip structures.

The Coringa shear zone may have been developed as an R (Riedel) shear in a dextral strike-slip setting off the main trend (310°). This right lateral movement created dilational jogs and the formation of open spaces. This can be seen on a macro scale at Meio Zone **(See Figure 7.5)**.

The Valdette, Galena, Mae de Leite, Meio, Come Quieto, and Serra zones sit on these R-shears, while the Sr. Domingo zone appears to be located along one of the main 310° strike-slip faults. **(See Figure 7.6)**.

Figure 7.3 Principal Structural Lineaments

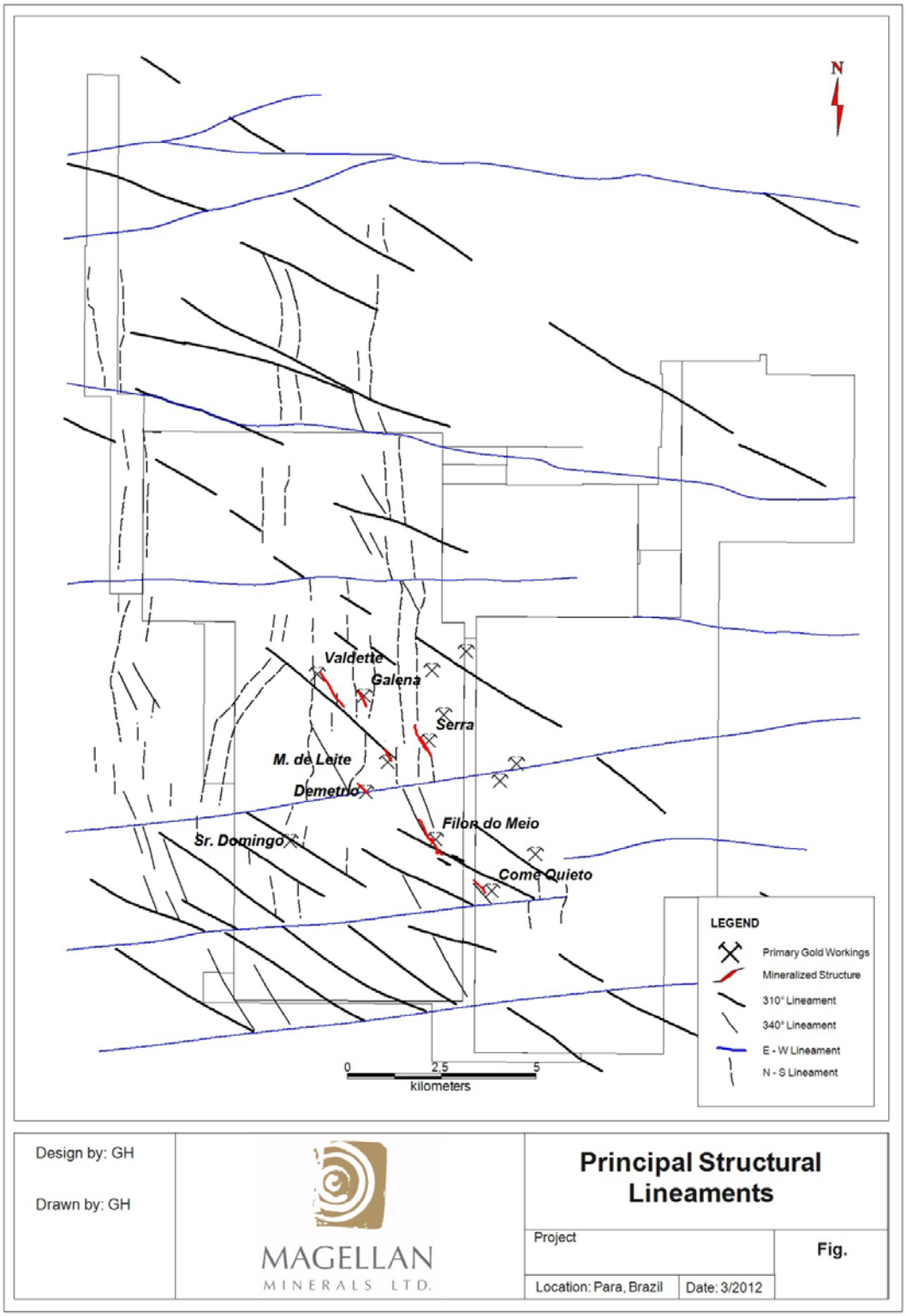


Figure 7.4 Structural Model

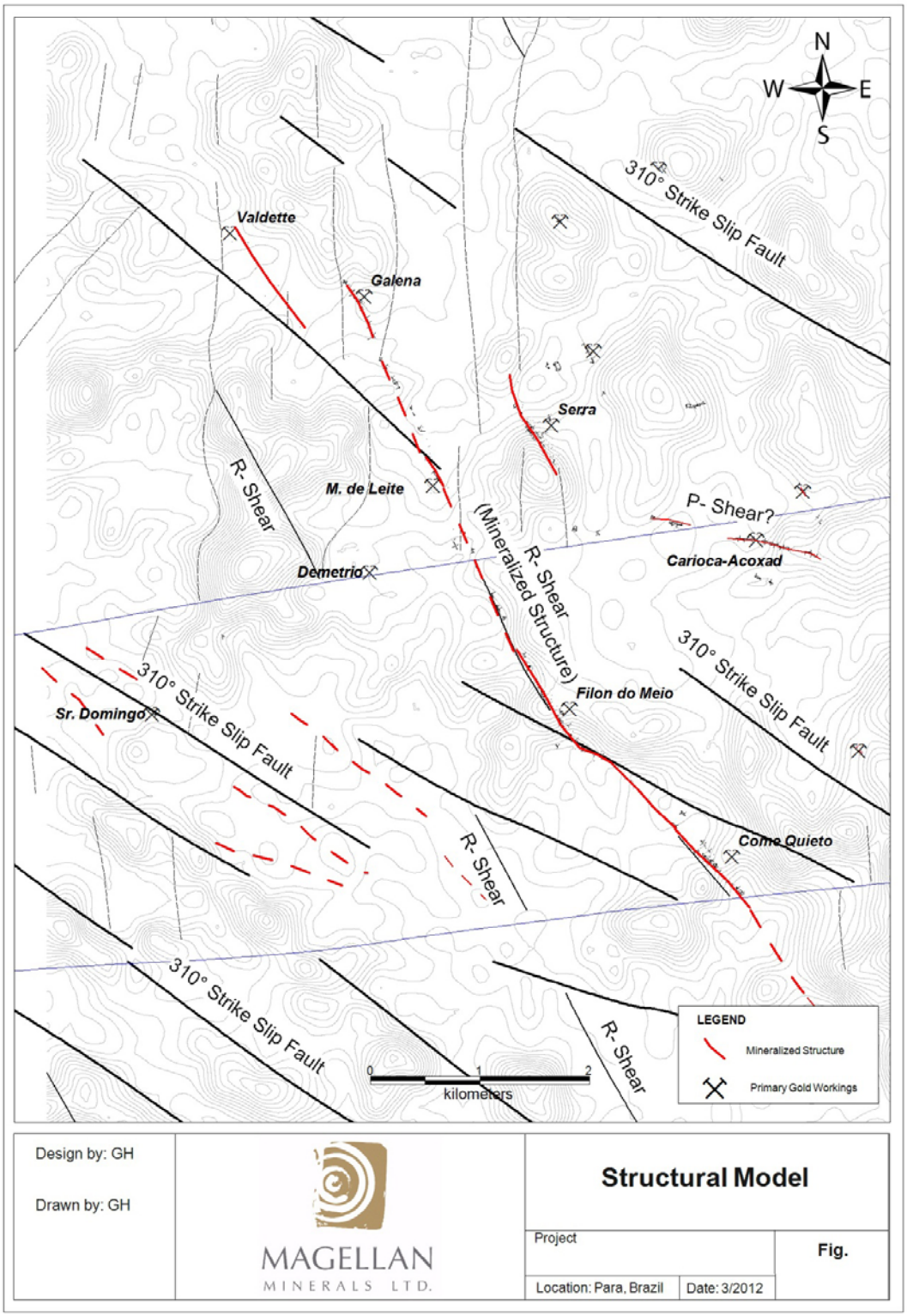


Figure 7.5 Meio Zone Structural Interpretation

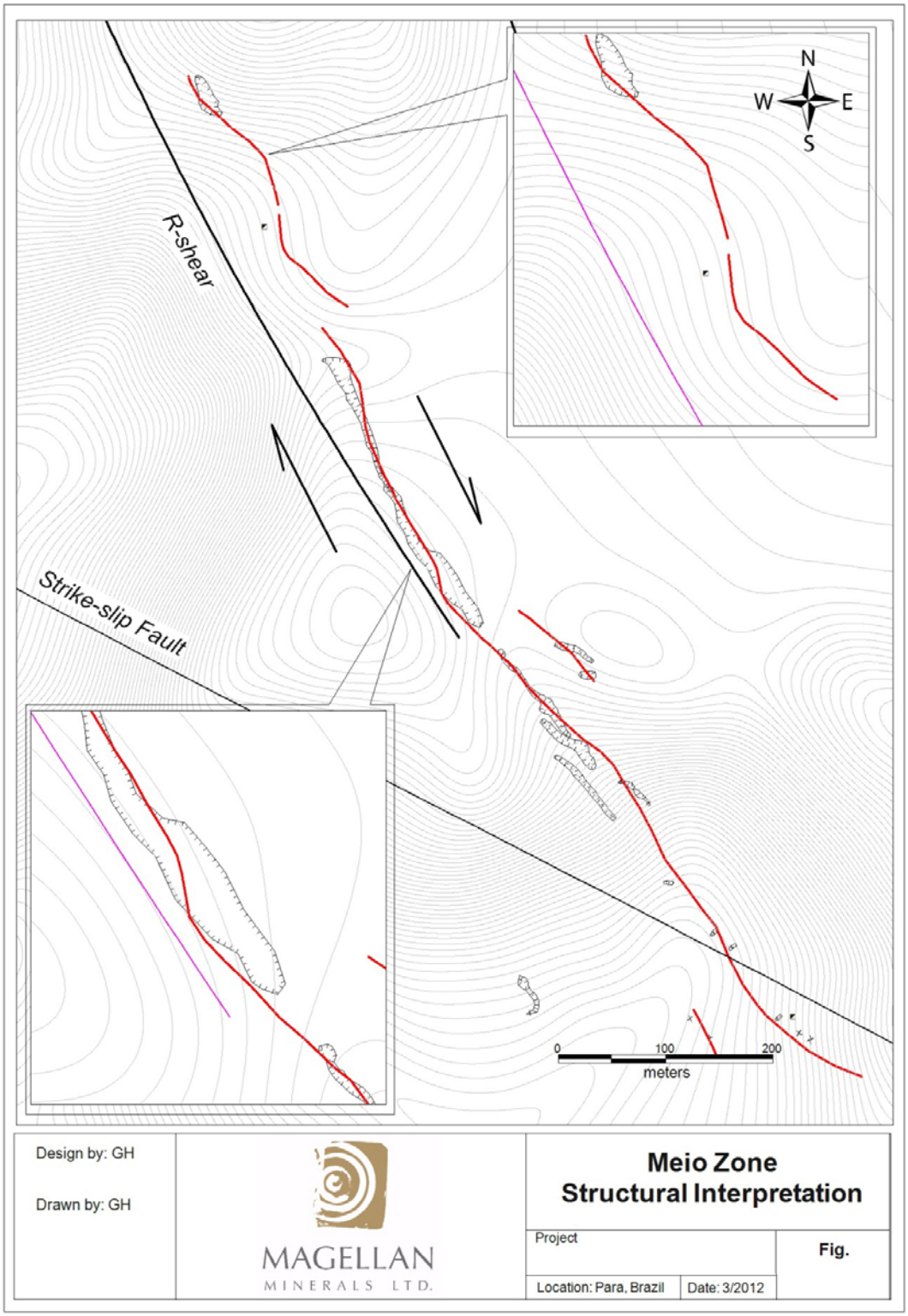
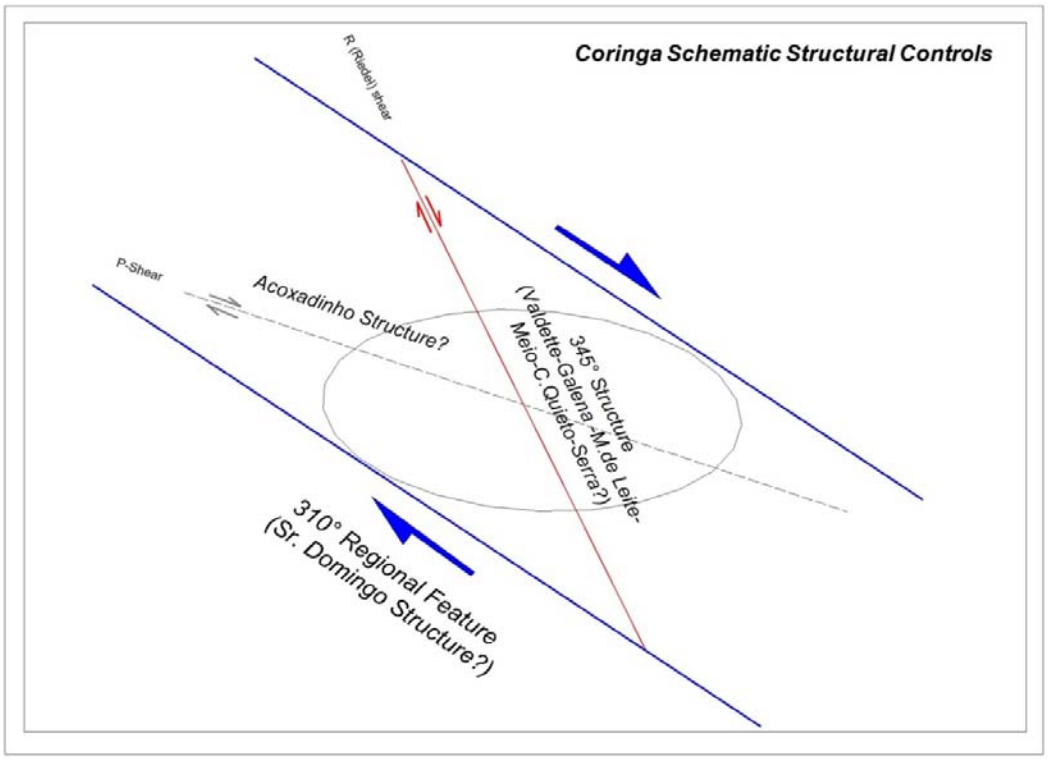


Figure 7.6 Coringa Schematic Structural Controls



7.5 Alteration

Mineralized veins at Coringa are surrounded by zones of wall-rock alteration which is not texturally destructive and consists of a distal chlorite-hematite alteration with a proximal sericite-pyrite alteration assemblage next to the veins **(See Photo 7.1).**

The chlorite-hematite alteration can be as wide as 10 meters in granite and up to 50 meters in rhyolites, where they surround multiples veins or veinlets.

The sericite-pyrite alteration can be as wide as 0.5 m in granites, and 1 m in rhyolites. There is no apparent relationship between the width of the vein and the width of the alteration. There are veins of 15 cm with 1 m of sericitic alteration and veins of 1.5 m with 10 cm of sericitic alteration. Similar observations hold for the chloritic-hematitic alteration.

In both the granitic and rhyolitic host rocks, the feldspar is strongly altered to sericite in a pervasive style. Calcite also exists as an alteration mineral, mainly in fractures. Chlorite appears in veinlets.

The alteration in the rhyolite is stronger, with a wider halo. In the granite, the sericitic alteration is just a few centimeters wide as seen in **Photo 7.2**.

The sericitization appears as a pale green colour, and it mainly affects the sanidine and plagioclase crystals. It also appears in veinlets with alteration minerals such as quartz, and chlorite (Universidad Nacional del Sur -2007) **(See Photo 7.3)**.

It is also usual to see the dark rhyolite matrix turn a reddish colour when altered. This is believed to be the result of staining of the plagioclase crystals by microcrystalline hematite **(See Photo 7.4)**.

Photo 7.1 Alteration of Rhyolite

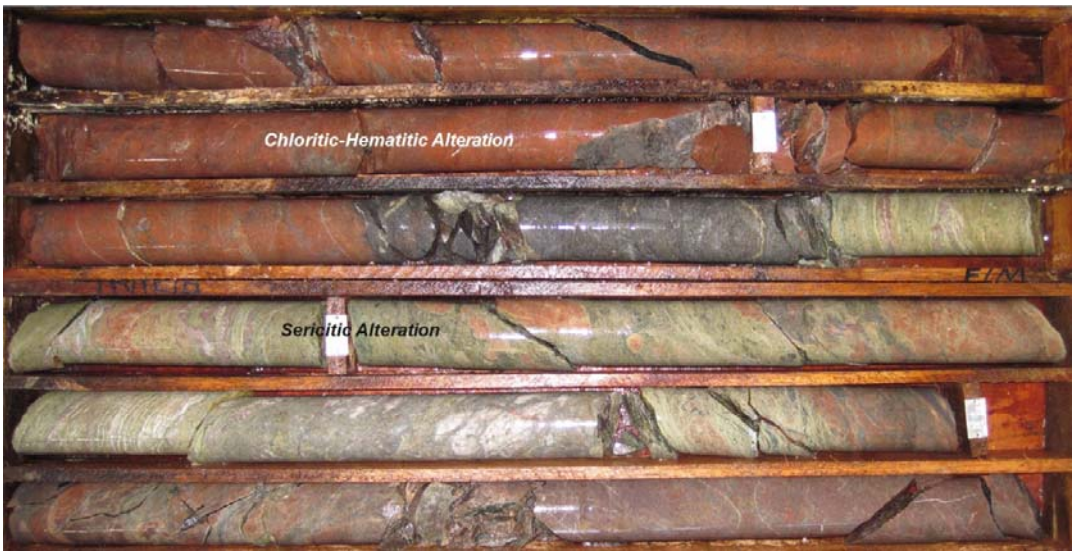


Photo 7.2 Sericitic Alteration of Granite

Photo 7.3 Sericitic Alteration in Rhyolite

Photo 7.4 Chloritic-Hematitic Alteration of Rhyolite

7.6 Mineralization

7.6.1 Introduction

Mineralization zones at Coringa are associated with a shear-vein system, defined either by historic surface "garimpeiro", or artisanal workers, workings or by soil anomalies. The orientation of the structures varies from 310° to 345°. Dips are variable from 75° to the northeast and up to 85° to the southwest. Mineralized shears vary in thickness from a few cm to 4.0 m in the veins, and up to 15 m with respect to the stock work/sheeted vein style of mineralization. The Coringa shear-vein system is hosted by both granites and rhyolites. Outcrops of the shear-hosted and/or vein mineralization are rare.

7.6.2 Coringa System

The Coringa system occurs within a north-northwest/south-southeast orientation, in granites from the Malaquinho Suite and rhyolites from the Salustiano formation. The system has typical lenticular vein morphology of a shear hosted quartz vein, with three types of veins, extensional (Galena-Mae de Leite zone, Serra zone, and Come Quieto), fault vein fills (Meio zone), and stockwork/sheeted veins (Valdette, Demetrio and Sr. Domingo zones).

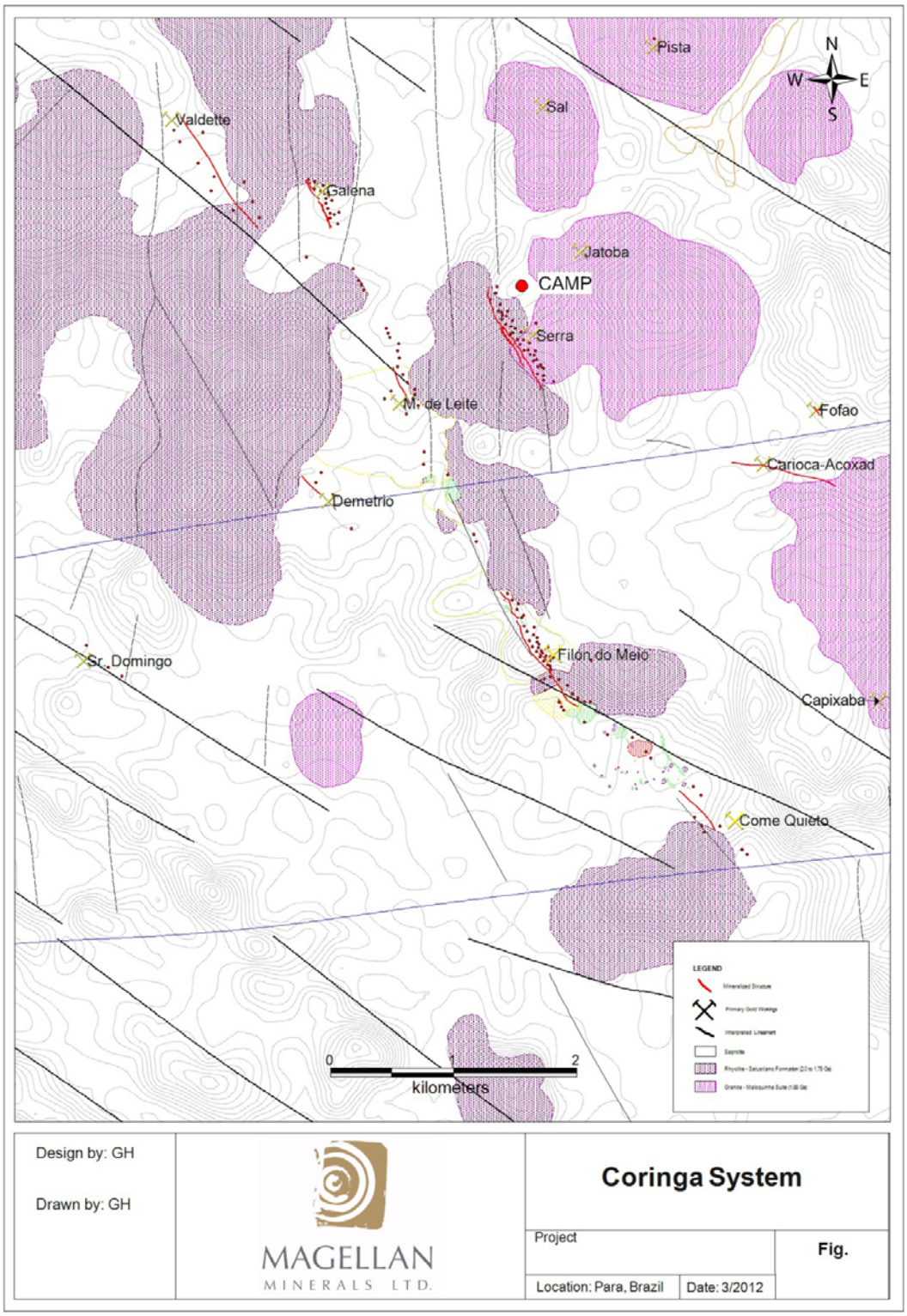
The main structure, which extends from Galena in the north to Come Quieto in the south, can be traced over 7,000 m along strike, and is intermittently exposed. There are sub-parallel zones (Valdette, Serra, Demetrio and Fofao). All these zones (the main structure and the sub parallel zones) are interpreted as R- shears in an en-echelon array. The main Coringa structure has a strike of 345°. The Sr. Domingo zone (310° strike) has received very limited exploration work thus far but is probably sitting on one of the main strike slip faults, and is not part of the Coringa main R-shear. One last zone, Carioca-Acoxadinho, with a 210° strike orientation has been interpreted as a P- Shear. P- shears usually develop after the establishment of R- shear, and their development is probably related to temporal variations in the local stress field along the shear zone (Structural Geology - Haakon Fossen-Univ. of Bergen) **(See Figure 7.7)**.

The Serra zone dips 75° to the NE, while the rest of the zones are vertical to sub vertical. So far, over 11 km of mineralized structure have been identified by drilling.

As outlined above, there are almost no outcrops of mineralization. The only exposures are within the artisanal workings, in which there is sometimes mineralized material evident to a depth of 18 to 20 m **(See Photo 7.5)**.

Photo 7.5 Mined Out Artisanal Workings (Mae de Leite)

Figure 7.7 Coringa System



Within the Coringa system, the high-grade mineralization is restricted to ore shoots. The exact spacing, size and depth of these shoots are still to be determined. In Serra and Meio zones ore shoot seems to be plunging 70° to the south and in all zones mineralization is open in depth.

Gold contents vary between 1 to 230.5 ppm and have a positive correlation with the percentage of sulfide in the ore **(See Photo 7.6 and 7.7)**. The ratio of gold to base metal varies between mineralized zones (CLM Petrografia Ltda., 2007) as shown in **Table 7.1**.

Table 7.1 Ratio of Au to Other Base Metals by Mineralized zone

Zone	Ag:Au	Cu:Au	Pb:Au	Zn:Au
Galena	1.7	172	996	1595
Serra	3.35	125	343	333
Meio	4.2	232	2700	1845
Come- Quieto	0.4	17	220	76

Photo 7.6 Visible Gold Shown in Drill Core

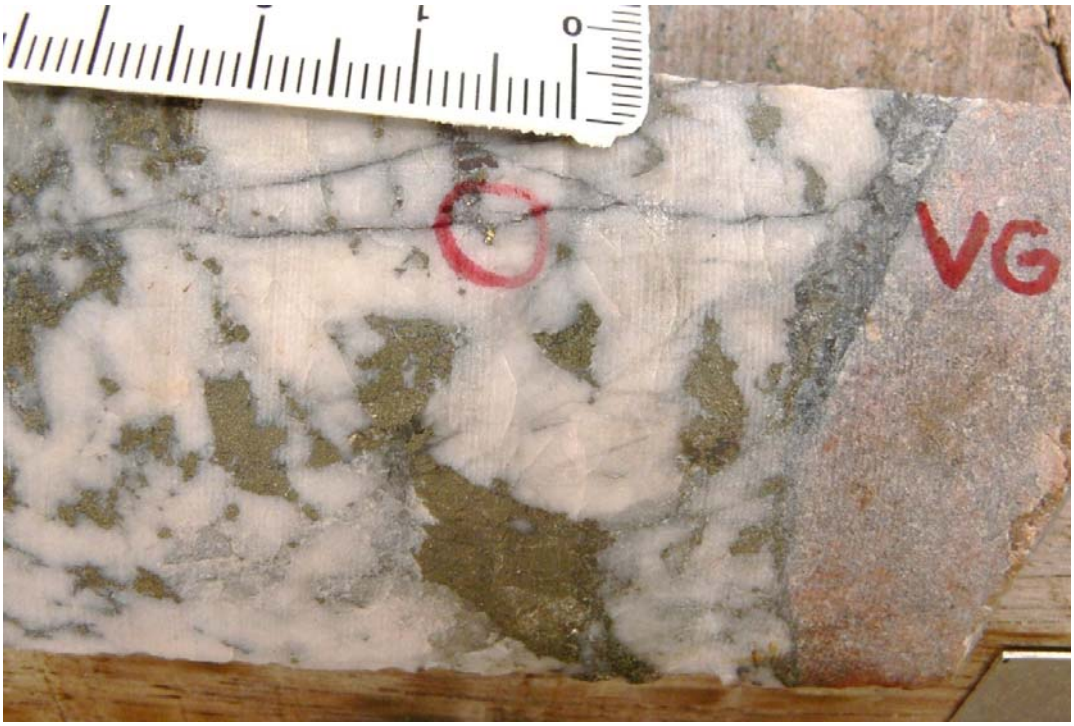
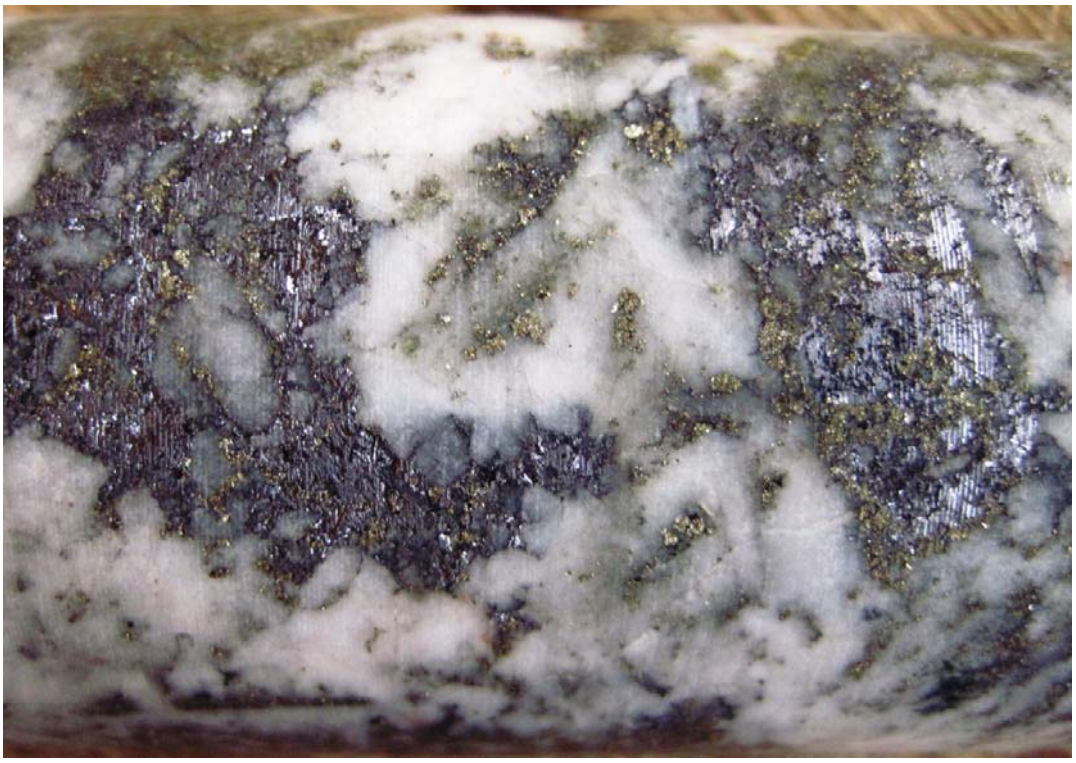


Photo 7.7 Drill Core Showing Sulfide Mineralization



The mineralogy of the veins is simple, although it does vary from one location to another. The Galena zone comprises extensional veins, which are mainly composed of quartz (commonly 60 to 90% volume), and sulfide minerals (commonly 5 – 30% by volume). Pyrite, galena and sphalerite and to a lesser extent chalcopyrite comprise more than 90% of sulfide by volume.

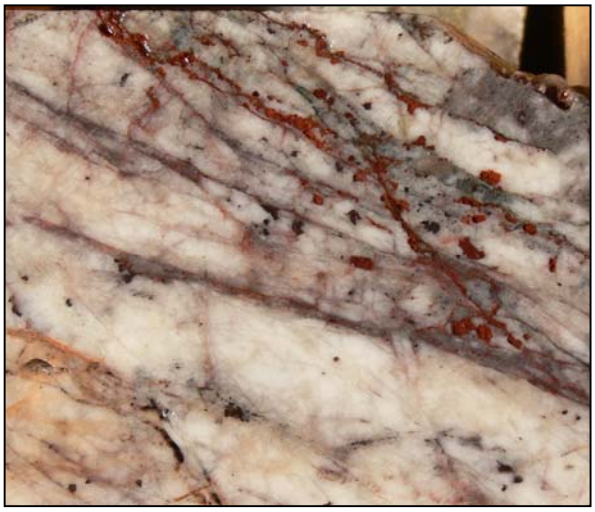
The extensional veins at Serra are of similar composition (60 – 90% of quartz by volume), with a slight difference in the percentage of sulfide minerals (commonly 2 – 20% by volume).

In the fault fill veins of the Meio zone, the main structure is a hydrothermal breccia, with quartz clasts and a grey quartz-sulfide rich matrix comprised, mainly of galena, sphalerite, pyrite (commonly 3 – 45% by volume), and lesser amounts of chalcopyrite.

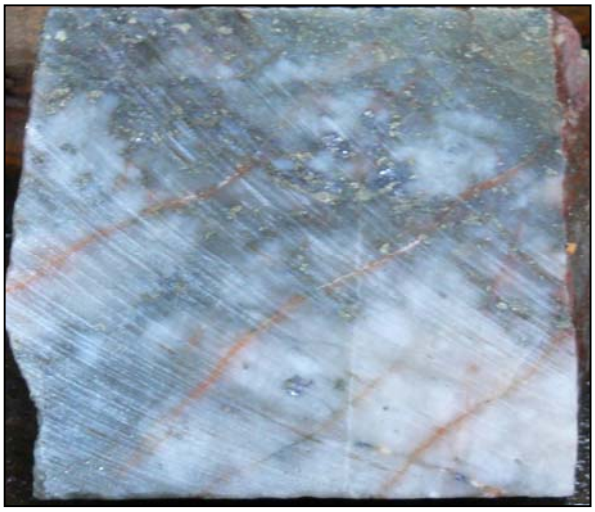
Ten different units have been mapped and logged in the Mineralized Domain and are as follows:

- Quartz veins.
- Quartz veins with base metal.
- Quartz vein with pyrite.
- Stockwork with base metal.
- Hydrothermal breccias with base metal matrix.
- Hydrothermal breccias with grey quartz matrix.
- Sericitic alteration.

- Sericitic alteration with base metal
- Chloritic-hematitic alteration.
- Quartz breccias with hematite matrix.

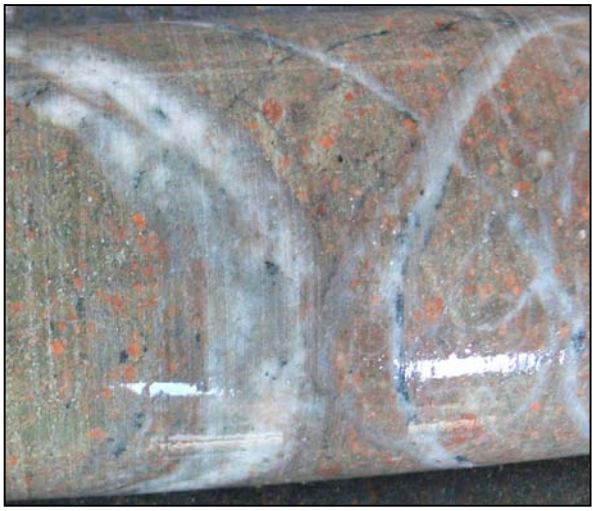


Quartz Vein

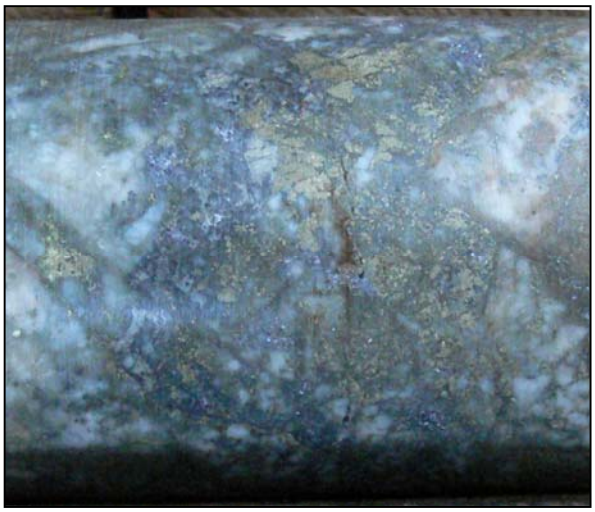


Quartz vein with base metal

Quartz vein with pyrite



Stockwork with base metal



Hydrothermal breccias with base metal matrix

Hydrothermal breccias with grey quartz matrix

Sericitic alteration

Sericitic alteration with base metal

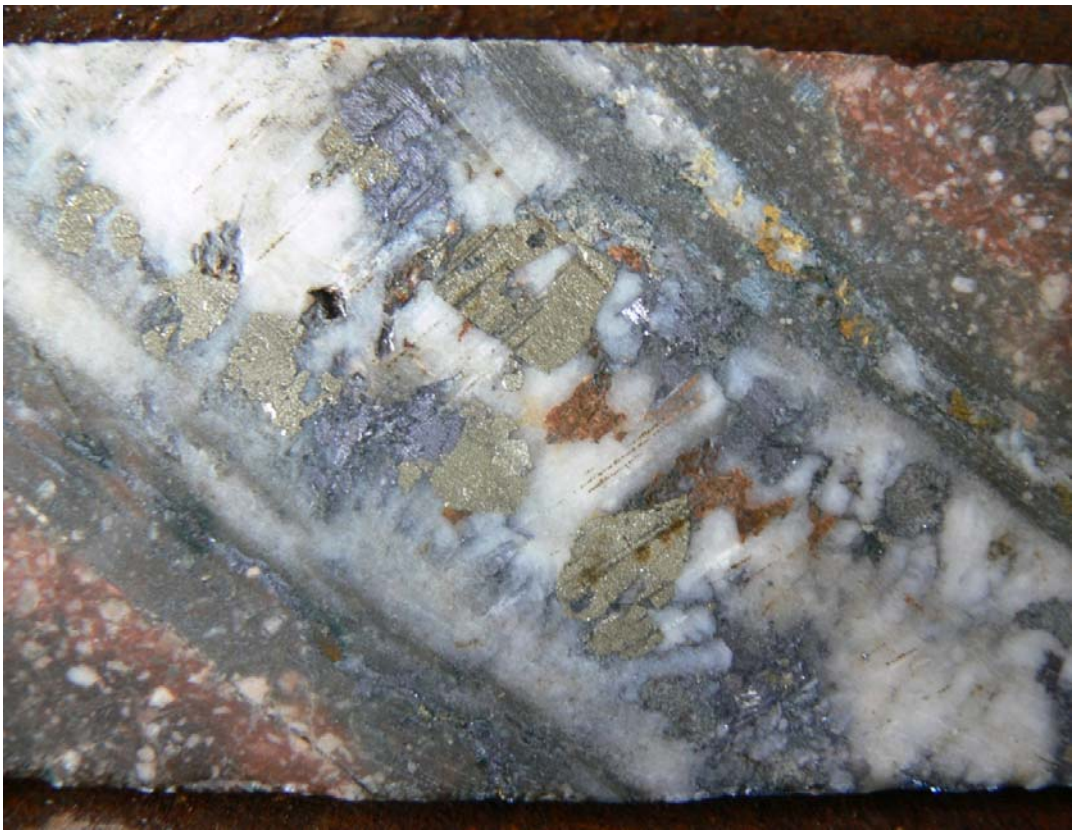
Chloritic-hematitic alteration.

Quartz breccias with hematite matrix.

7.6.3 Mineralization Styles and Paragenetic Sequence

The style of the veining varies from open, banded to druzy, textured quartz veins (Serra), to strongly sheared/fractures and milled quartz sulfide (Meio). Base metal sulfides are later than the initial phase of druzy quartz as the sulfides envelop hexagonal quartz crystals **(See Photo 7.8)**. The lack of grade in the massive quartz that is not infilled with later sulfide suggests that the massive quartz represents an unrelated pre-mineralization event. (Telluris Consulting-Field Structural Review of the Cuiú-Cuiú and Coringa District).

Photo 7.8 Base Metals Sulfides and Quartz



A paragenetic sequence has been recognized through a petrographic study by CLM PETROGRAFIA LTDA. (2007). The summary of this study is described below.

"This report presents the result of the petrographic study of 21 (twenty-one) polished sections of rocks requested by CHAPLEAU EXPLORAÇÃO MINERAL LTDA.

The polished section study showed pyrite as the most frequent sulfide, followed by sphalerite, galena, chalcopyrite, and in smaller amount covellite.

Pyrite usually forms euhedral crystals and was the first sulfide to crystallize in the paragenesis. Chalcopyrite forms anhedral aggregates and is replacing pyrite suggesting its later crystallization. Sphalerite replaces both, pyrite and chalcopyrite. Galena was the last sulfide to crystallize in the paragenesis and replaces all the other sulfides. Covellite is an alteration product of chalcopyrite and was locally observed replacing the later.

Native gold was detected in seven of the sections analyzed, 5μ – 220μ in size, included mostly in pyrite or at the junction between pyrite and sphalerite, galena or chalcopyrite. A smaller amount of grains were detected in the silicate gangue or at the junction between sulfide and silicate. Sixty-one grains of native gold were detected."

8.0 DEPOSIT TYPES

8.1 Shear Zone Deposits

Shears zone deposits are zones of anomalously high strain, and form linear mappable units. These linear units occur in generally predictable orientations, and are located in certain preferred settings.

In these mappable units, lithologies may be rotated (A), folded (B), dislocated (C), truncated (D), thinned (E), thickened (F), repeated (G), transposed (H), deformed parallel to bedding at outcrops (I), or lozenge shaped in lithons and enclaves (J) (**See Figure 8.1**).

Figure 8.1 Block Diagram of Conceptual Shear Deformation Zone

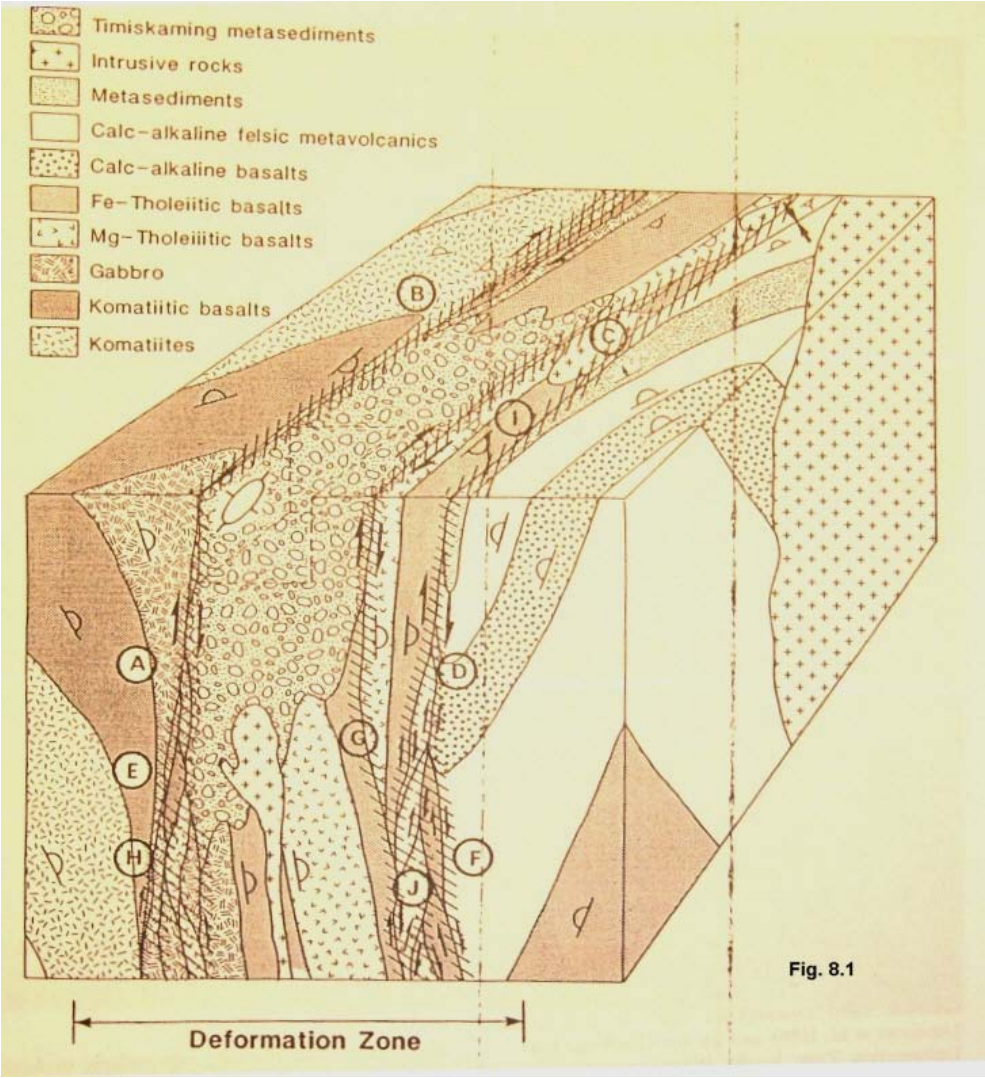


Figure 8.1 shows a schematic block diagram indicating large-scale features of deformation zones which occur in shear zones (Ontario Geological Survey – 1988). The large scale zone consists of many individual shear zones that are rarely linear and continuous, but which anastomose, bifurcate, or are discontinuous in both vertical and horizontal dimensions.

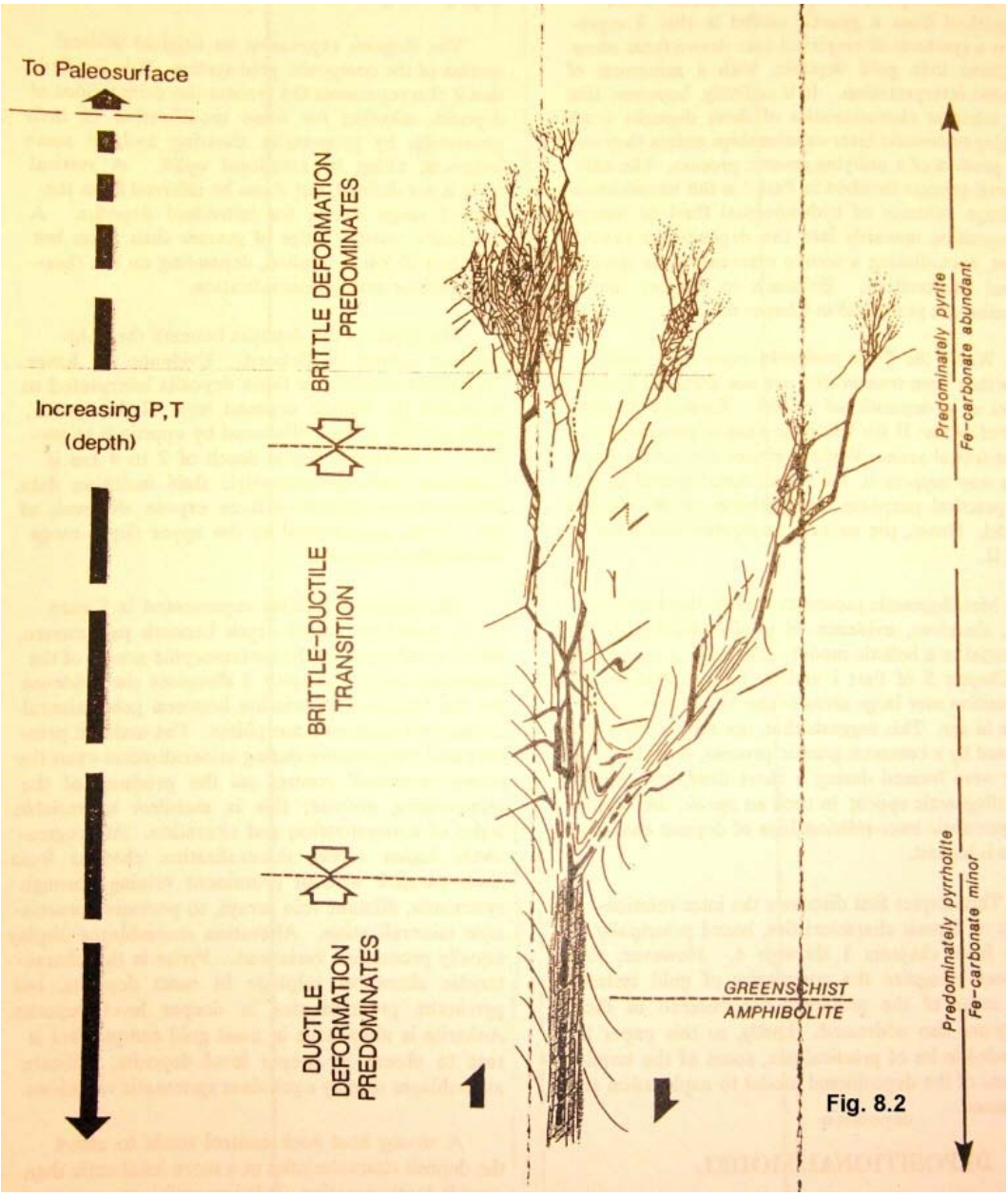
Many deformation zones are characterized by structures of both ductile and brittle deformation, which may be coeval or occurred at different times. These zones generally contain altered rocks, carbonate and hydrous minerals, as well as quartz and quartz-carbonates veins (low Pressure-Temperature [P-T] conditions), or aluminosilicate and garnet alteration (higher P-T condition). The higher P-T alteration conditions would not appear to be characteristic of the Coringa deposit. Syn- to late- tectonic, intermediate to felsic, silica-saturated and under saturated dikes and stocks may appear intruding the deformation zones.

Gold mineralization is hosted in small-scale structures within larger deformation zones. These structures are highly permeable zones, which are preferred sites for mineralization. A change from the brittle to ductile environment occurs with increasing depth. Gold is associated with large volumes of fluid that have altered the host rock. This fluid, through its influence on the ductile-brittle behavior of the rocks, will affect the form and distribution of gold bearing structures.

Ore zones are lenticular, tabular or irregular shaped bodies composed of veins, breccias zones, and/or stockwork systems. Veins transect lithological contacts, and are not restricted to a specific rock type. Veins can be classified as replacement veins, extension veins, or breccias and fracture veins.

There is also a vertical zonation of gold deposits, which reflects a change in deformation style, from brittle to brittle-ductile. For example breccia veins occur principally within brittle deformation and replacement veins typify ductile zones (**See Figure 8.2**).

Figure 8.2 Idealized Deposit Model



Idealized composite depositional model for Archean load gold deposits – Ontario Geological Survey – 1988.

Minerals common to the gold related alteration zone include:

- Carbonates
- Potassic phyllosilicates (Sericite and biotite)
- Alkali feldspar (albite and potash feldspar)
- Chlorite associated calcite and dolomite

- Iron sulfides (pyrite)
- Quartz
- Chloritoid

The most distinctive occurrence of gold is in quartz veins. In some deposits this is the principal location of ore, although gold occurs in the altered rocks. In other deposits, gold is associated with alteration sulfides in the wall rock.

Deposits similar to the Coringa Gold Project can be found in Ontario's Archean gold district in Eastern Canada. One characteristic of the gold deposits in this district is their occurrence within major tectonic zones which comprise linear shear systems. All of the major gold camps in the Superior Province of Canada, including Rice Lake, Red Lake, Hemlo, Wawa, Timmins, Kirkland Lake, Val D'or-Malartic and Casa Berardi are associated with deformation zones. (cf. Hurst 1935, Gunning and Ambrose 1937, Newhouse 1940).

The Campbell Red Lake and Dickenson Mines (A. W. White), the Conchenour-Williams Mine at Red Lake (Sanborn-Barrie 1987), the Sigma Mine at Val d'Or, and the Duport deposit (Smith 1986) on Lake of the Woods are some examples of gold mineralization associated with shear veins developed during brittle and brittle-ductile deformation.

Breccia veins similar to Filon do Meio Block (Coringa gold project) can be seen in the Macassa Mine at Kirkland Lake (Watson y Kerrich, 1983), and the Duport deposit.

Fracture filling veins, similar to Filon do Serra block (Coringa Gold Project), and Galena-Boca Block (Coringa Gold Project), were mined at the Canadian Arrow and Murphy-Garrison Mines (Cherry, 1983).

Table 8.1 shows a list of some of the Archean, shear-hosted type deposits known to date in the Ontario Province of Eastern Canada and with which the Coringa gold deposit can be compared due to the similar style of mineralization (Ontario Geological Survey).

Table 8.1 Shear Hosted Deposits, Ontario Geological Survey

Mine	Gold (Millions Oz)
Page Williams, Hemlo	9.1
Lake Shore, Kirkland Lake	8.5
Golden Giant, Hemlo	6.7
Campbell Red Lake, Red Lake	5.5
Wright-Hargreaves, Kirkland Lake	4.8
Lamaque, Val d'Or	4.7
Teck-Hughes, Kirkland Lake	3.7
Sigma, Val d'Or	3.3
David Bell, Hemlo	3.0
Pamour (no.1), Porcupine	2.9
East Malartic, Val d'Or	2.9
Macassa, Kirkland Lake	2.6
Aunor (no.3), Porcupine	2.5
Madsen, Red Lake	2.4
Dickenson, Red Lake	2.3

9.0 EXPLORATION

9.1 Introduction

The following section presents the different phases of the exploration program carried out in defining the resources and gold potential of the Coringa Project.

9.2 Chapleau Resources Exploration (January 2007 to June 2007)

Between January 2007 and June 2007, Chapleau completed an exploration program over an area of 6.5 km by 8 km on the Coringa project.

Since there was very little outcrop and no previous exploration work in the area (except for the Garimpeiros workings), a structural interpretation based on an Ikonos Satellite Image was carried out, together with some rock sampling (mainly grab samples), and a survey of the main "garimpos", or artisanal workings. A small soil sample program was carried out on the Serra zone. This work led to the recognition of the main targets to be drilled.

After the first-phase drilling, samples were taken for petrographic and petrologic studies, in order to understand the paragenetic sequence and the mineralization style.

The diamond-drill holes showed the presence of high-grade narrow veins which were interpreted to be associated with northwest-trending structural lineaments.

Details of the exploration activity are as follows:

- **Structural Interpretation/Geological Mapping**: An Ikonos Image base structural interpretation, followed by geological mapping was carried out, together with a detailed survey of all the garimpos.
- **Rock Sampling:** Rock samples (62) were taken wherever possible as most of the area is covered by jungle or saprolite. The samples were sent to ALS Chemex Laboratory in Belo Horizonte, Brazil, for preparation and assayed for gold and 34 other elements.
- **Soil Sampling:** Soil sampling was carried out over the Serra structure using a 100 by 25 m sampling grid. A total of 218 soil samples were collected, which were submitted to the ALS Chemex laboratories in Brazil for preparation and gold assay and 34 element analyses.
- **Diamond Drilling:** A total of 1,774 m of HQ core were drilled in 22 hole diamond-drill holes by Geoserv Pesquisas Geológicas S.A. (Boart Longyear).
- **Petrologic Studies:** Four (4) thin sections from drill core samples were prepared and reported on by the Universidad Nacional del Sur, Bahia Blanca, Province of Buenos Aires, Argentina.
- **Petrographic Studies**: Twenty one (21) polished sections from drill core samples were taken and studied by CLM PETROGRAFIA LTDA., in Rio de Janeiro, Brazil.
- **Drill Core Sampling**: A total of 1,215 samples were taken from the 1,774 m of HQ core, which was sent to SGS Geosol laboratory in Belo Horizonte for sample preparation, followed by gold and 34 element analyses.

9.3 Chapleau Resources Exploration (June 2007 to March 2008)

Continuing with the exploration program, a 548 square kilometer airborne magnetic-gradiometric-radiometric survey was carried out, followed by an interpretation of the survey by RECONSULT Geofisica. This showed several new regional targets and a clear definition of the different lithologic units (granite-rhyolite). An Induced polarization dipole-dipole geophysical survey over Galena-Boca block, Eloy-Juara-Mae de Leite, and Serra blocks, showed a high chargeability anomaly coincident with the main structure, and after the drilling program a relationship between high chargeability with high grades was interpreted.

The drilling program was designed to follow the main Galena-Mae de Leite structure and the Serra structure (50 m spacing along strike). Three holes were also drilled in the Meio Block, which showed a high-grade zone, with more of a massive base metal component. The diamond-drill holes showed:

The Galena-Boca structure is continuous up to BR_COR_DDH31, and is interpreted to have two higher grade ore shoots,

The Juara-Mae de Leite zone has some discontinuity, and

The Serra zone proved to be continuous for almost 700 m, with two higher grade ore shoots along strike.

Details of the exploration program are as follows:

- **Airborne Magnetic – Gradiometric and Gamma Ray Survey**: During the period of June 26th to June 28th 2007, a high resolution airborne magnetic gradiometric survey was carried out by LASA ENGENHARIA E PROSPECCOES S.A. The airborne survey covered 549 square kilometres with a 200 m grid spacing, at an altitude of 100 m.
- **IP (induced polarization) dipole-dipole survey**: A 34 km line induced polarization (IP) dipole-dipole geophysical survey over the Galena–Mae de Leite zone (except the Mae de Leite Garimpo), and Serra zone was carried out by Laboratorio de Pesquisa em Geofisica Aplicada – UFPR.
- **Metallurgical Testing**: Two composite gold samples, prepared from drill core samples, were tested at SGS Geosol Mineral Laboratory in Belo Horizonte, Brazil for definition of the processing route. Test work included size by size analysis as well as gravimetric separation, leaching and flotation.
- **Diamond Drilling:** A total of 5,032 m of HQ core was drilled in a 44 hole diamond-drill program Geoserv Pesquisas Geológicas S.A. (Boart Longyear).
- **Drill Core Sampling:** A total of 2,437 samples were taken from the 5,032 m of HQ core, which were sent to SGS Geosol laboratory in Belo Horizonte for sample preparation, followed by gold and 34 element analyses.
- **Interpretation of the Airborne Magnetic + Gamma Ray Survey:** An interpretation of the airborne geophysical data was carried out by RECONSULT Geofisica. A new set of enhanced images were generated, and new exploration targets were defined.

9.4 Chapleau Resources Exploration (March 2008 to December 2008)

A second ground IP survey of 70.7 km line was carried out by INTERGEO GEOFISICA over the remaining structures to the south of the Galena-Mae de Leite zone. An 860 km line EM airborne geophysical survey of was flown. The third diamond-drill program was designed to follow the Meio Structure, and to test one or two of the IP anomalies. The Meio structure proved to be continuous for at least 450 m, and shows three plunging higher grade ore shoots.

Details of the exploration program are as follows:

- **IP (induced polarization) dipole-dipole survey:** A 70.7 km line IP (induced polarization) dipole-dipole, geophysical survey over approximately 7.0 km of gold bearing structures was completed by INTERGEO GEOFISICA.
- **EM geophysical survey:** A time domain electromagnetic geophysical survey of 860 km line was flown by GEOTECH AIRBORNE LIMITED & MICROSURVEY AEROGEOFISICA E CONSULTORIA CIENTIFICA LTDA. to cover all identified pan concentrate and gold in soil anomalies.
- **Diamond Drilling:** A total of 1,979 m of HQ core was drilled in a 15 hole diamond-drill program by Geoserv Pesquisas Geológicas S.A. (Boart Longyear).
- **Drill Core Sampling:** A total of 386 samples were taken from the 1,979 m of HQ core, which were sent to SGS Geosol laboratory in Belo Horizonte for sample preparation, followed by gold and 34 element analyses.

9.5 Chapleau Resources Exploration (January 2009 to September 2009)

A detailed geological survey was carried out from the Eloy-Juara-Mae de Leite block up to the Come Quieto block, together with trenches and soil sampling. Field results from this survey showed continuity of the main structure along 3,800 m, with rock chip samples up to 67.91 gpt of Au.

Details of the exploration program are as follows:

- **Geological Survey and Trenching:** A detailed geological survey was carried on an area of 3 km^2 (3.8 km by 08 km), together with 18 trenches and 76 rock chip samples. The survey covered the southern extent of Mae de Leite artisanal working, in the north, and extended south to Come Quieto.
- **Soil Survey:** 36 soil lines were opened with lengths ranging from 200 m up to 800 m. Spacing between samples ranged from 12 m up to 25 m, with a total of 624 samples.

9.6 Magellan Minerals Exploration (September 2009 to December 2009)

A soil survey was carried out over two new targets, identified during the stream sediment program. These two soil programs were carried out north of the Galena zone, east of the

Juara-Mae de Leite zone, and north of the Come Quieto zone. The fourth diamond-drill program of a proposed 2,500 m was designed to:

1. Delineate the southeast and northwest extension of the Meio gold zone which forms part of the inferred and combined indicated and measured gold resource at Coringa.
2. Confirm gold mineralization at depth in the new discovery zone approximately 900 m to the southeast of the Meio zone.
3. Test the northwest extension of a 2 km gold-in-soil anomaly extending from the Meio zone that has numerous artisanal workings along its strike length.

Details of the exploration program are as follows:

- **Soil Sampling:** Soil sampling was carried out over 2 new targets, north of Galena (100 m by 25 m grid for the first eleven lines, and a 200 m by 25 m grid for the following five lines), east of Mae de Leite-Juara (200 m by 25 m grid), and north of Come-Quieto (100 m by 25 m grid). A total of 2,275 soil samples were collected of which 987 samples were submitted to SGS Geosol laboratory in Itaituba for preparation and SGS-Geosol in Belo Horizonte for gold assay. 1,288 Soil samples were collected.

Rock and soil samples were submitted to ACME laboratories in Itaituba, Brazil for preparation and ACME Santiago de Chile for gold assay.

9.7 Magellan Minerals Exploration (January 2010 to December-2010)

A soil survey was carried out to the north and NW of Galena Zone, and to the west of the main Coringa mineralized structure, together with some trenches. The fourth diamond-drill program started, totalizing 3,396 m. By Quarter 2, a Preliminary Economic Assessment on the Coringa Project was released.

- **Soil Sampling:** Soil survey was carried out to the N and NW of Galena zone and to the W of Coringa main structure (200 m by 25 m grid). A total of 4,004 soil samples were collected.

- **Diamond Drilling:** A total of 3,396 m of HQ core was drilled in 28 diamond-drill holes by Layne do Brasil Sondagems Ltda. A total of 835 samples were taken. **Core, rock and soil samples were** submitted to ACME laboratories in Itaituba, Brazil for preparation and ACME Santiago de Chile for gold assay. The program proved the continuity of the Mae de Leite structure at depth and along strike, as well as the continuity of the Meio Zone along strike to the North and South.

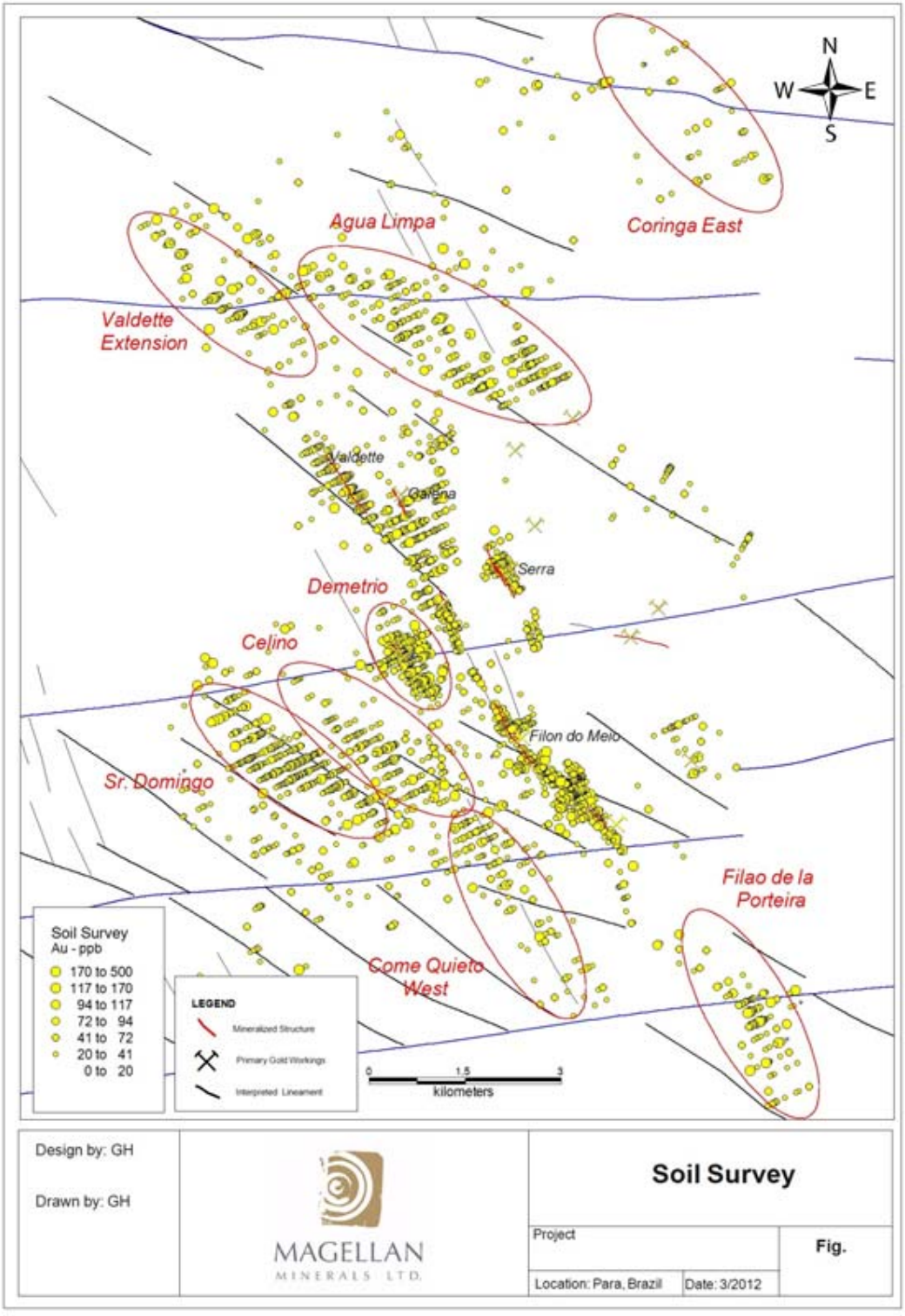
9.8 Magellan Minerals Exploration (January 2011 to December-2011)

The soil survey that started in 2010 went through 2011, to the SW and NE of the main Coringa structure. Trenches were placed at Valdette and Demetrio new soil anomalies. The fifth diamond-drill program started, with 3 rigs on site, totalizing 11,912 m.

- **Soil Sampling:** Soil survey was carried out to the W and SW of Demetrio zone, and to the N and NE of Galena zone on a 200 m by 25 m grid. A total of 10,833 soil samples were collected. Five new soil anomalies were identified from the sampling (**See Figure 9.1):**
    - **Celino:** 2.8 km long**,** located to the W of Demetrio,
    - **Sr. Domingo:** 3.8 km long, located to the W of Celino
    - **Come Quieto West:** 2.3 km long, located to the W of Come Quieto
    - **Agua Limpa:** 3.9 km long, located to the NW of Galena.
    - **Valdette Extension:** 3.2 km long, possible the N extension of Valdette.
- **Trenches:** Fourteen trenches were dug on Valdette zones and 3 on Demetrio zones. Over 1,000 samples were taken. Results showed the continuity of both structures along strike.
- **Diamond Drilling:** A total of 11,912 m of core was drilled in 51 holes by Layne do Brasil Sondagem Ltda. and Geosol-Geologia e Sondagem S/A. The program was targeting the lower portions of the two main zones, Serra and Meio, as well as a step-out-drilling, and testing new targets as Valdette and Demetrio. A total of 4,392 samples were taken.

Core, rock and soil samples were submitted to ACME laboratories in Itaituba, Brazil for preparation and ACME Santiago de Chile for gold assay.

Figure 9.1 Soil Survey Anomalies



10.0 DRILLING

10.1 Introduction

A total of 24,093 m of diamond drilling core in 160 exploration holes has been completed on the Coringa Project since 2007. The first-phase drilling (1,774 m in 22 holes) consisting of early stage exploration targeted the main "garimpos", or artisanal workings. The second-phase drill program (5,032 m in 44 holes) focused on the Galena- Boca, Eloy-Juara-Mae Leite, Serra, and Guaxebinha-Meio-Onza zones. The third-phase drill program (1,979 m in 15 holes) focused the Guaxebinha-Meio-Onza zone, and also aimed at testing two of the IP targets. The fourth phase of drilling (15,308 m in 79 holes) was initiated in March 2010, and was focused on increasing the 43-101 Compliance Resources released in October 2009 by targeting the lower portion of the two main zones, Serra and Meio, as well as step-out-drilling and testing new targets as Valdette, Demetrio and Sr. Domingo.

Table 10.1 summarizes the drill holes completed within each phase of drilling. Details of holes drilled by zones are shown in **Table 10.2**.

Sections 10.2 through 10.5 provide a summary of the drilling by phase. Section 10.6 details the core logging procedures. Section 10.7 discusses the drilling interpretations and results by zone.

Drawings showing the drill hole locations are shown in Annex 1.

Table 10.1 Drilling Summary by Phase

Phase	Holes	Holes with downhole survey	Meters	Samples	Meters sampled
1	22	0	1,774	1,922	1,717
2	44	42	5,032	1,711	1,370
3	15	12	1,979	434	333
4	79	66	15,308	5227	4752,83
Total	**160**	**120**	**24,093**	**9,294**	**8,172.83**

Table 10.2 Drilling Summary by zones

Zone	Number of Drill Holes	BR_COR_DDH#	Meters Drilled
Galena-Boca	17	3; 4; 5; 6; 23; 24; 25; 26; 27; 28; 29; 30; 31; 34; 36; 58; 60	1956.35
Eloy-Juara-Mae de Leite	21	17; 32; 33; 35; 40; 44; 51; 53; 54; 56; 96; 98; 99; 100; 101; 102; 103; 104; 105; 106; 118	2097.7
Serra	37	1; 2; 19; 20; 37; 38; 39; 41; 42; 43; 45; 46; 47; 48; 49; 50; 52; 55; 57; 59; 61; 64; 66; 121; 124; 127; 129; 132; 135; 138; 139; 141; 145; 148; 150; 153; 160	5624.3
Bravo-Escorpion-Peixoto	5	16; 22; 97; 108; 109	475.87
Guaxebinha-Meio-Onza	48	11; 12; 13; 14; 62; 63; 65;67;68;69;70;71;72;73;74;75;76;77;78;79;80;81; 82; 83; 84; 85; 86; 87; 88; 89; 90; 91; 92; 93; 94; 95; 130; 134; 137; 140; 144; 149; 152; 155; 156; 157; 158; 159	7660.6
Come Quieto	9	7; 8; 9; 10; 120; 122; 126; 151; 154	1567.65
Fofao	1	15	59
Pista	2	18; 21	105.75
Acoxadinho	1	107	101.43
Demetrio	4	111; 113; 115; 116	897.4
Valdette	11	110; 112; 114; 117; 119; 123; 125; 128; 131; 133; 136	2843.1
Sr. Domingo	4	142; 143; 146; 147	703.58

10.2 March 2007 - June 2007: Phase 1 Diamond Drilling Program

The phase 1 drilling program on the property was completed by Chapleau in 2007. A total of 22 widely spaced HQ core holes (BR_COR_DDH1 to 22), totaling 1,774 m were drilled to test the mineralized structure under the most significant garimpos. The drilling tested the following zones:

- 4 holes (totaling 393.2 m) within the Serra zone
- 1 hole (totaling 66.1 m) in the Eloy-Juara-Mae Leite zone
- 4 holes (totaling 343.6 m) in the Galena-Boca zone.
- 1 hole (totaling 59 m) in the Fofao zone.
- 2 holes (totaling 105.75 m) in the Pista zone

- 2 holes (totaling 115.43 m) in the Bravo-Peixoto zone.
- 4 holes (totaling 362.65 m) in the Come Quieto zone.
- 4 holes (totaling 328.4 m) in the Guaxebinha-Meio-Onza zone

The holes were drilled at a dip angle of 50° with lengths ranging between 44 m to 115 m. All drilling was completed by Geoserv Pesquisas Geológicas S.A. (Boart Longyear) using a Longyear 38 skid-mounted hydraulic rig.

All drill-hole collars were surveyed with a differential GPS at the completion of the second-phase drilling, and the collars were marked with a concrete marker.

Down-hole deviation tests were not taken during the initial phase of drilling.

Core recovery ranged from 86% to 100%.

10.3 October 2007 - March 2008: Phase 2 Diamond Drilling Program

During 2007 and 2008, 44 HQ core holes (BR_COR_DDH23 to 66) totaling 5,032 m were completed at the Coringa Project. Holes were drilled at a dip angle ranging between 47° to 60° (most of them at 50°), with lengths ranging from 59 m to 159 m. The drilling tested the following zones:

- 13 holes (totaling 1,612.75 m) in the Galena- Boca zone
- 9 holes (totaling 769.5 m) in the Eloy-Juara-Mae Leite
- 19 holes (totaling 2,333.45 m) in the Serra zone.
- 3 holes (totaling 316.6 m) in the Guaxebinha-Meio-Onza zone

In this phase of drilling down hole surveys were carried out.

10.4 September 2008 - December 2008: Phase 3 Diamond Drilling Program

During 2008, 15 HQ core holes (BR_COR_DDH67 to 81) totaling 1,979 m were completed at the Coringa Project testing the Guaxebinha-Meio-Onza zone. Holes were drilled at a dip angle ranging between 50° to 60°, with lengths ranging from 74 m to 238 m. These holes were used to further define known structures, except BR_COR_DDH74 and BR_COR_DDH77, which tested two IP anomalies.

10.5 March 2010 - December 2011: Phase 4 Diamond Drilling Program

In 2010, 28 HQ core holes, totaling 3,396 m were drilled. Sixteen of them tested the continuity along strike to the South and North of the Meio zone. The Mae de Leite zone was drilled for the first time (11 holes). All holes were drilled at a dip angle ranging from 50° to 60°, with lengths ranging from 80 m to 175 m. Average core recovery was over 93%.

The distribution of the holes was as follows:

- 11 holes (totaling 1,253 m) in the Mae de Leite zone
- 16 holes (totaling 2,042 m) in the Meio zone
- 1 hole (totaling 101 m) in the Acoxadinho zone

During 2011, 51 holes totaling 11,911.53 m were completed. Holes were drilled at dip angles between 50 and 70°, with length ranging from 110 m and 330 m. The drilling tested the extension along strike and at depth of Meio, Come Quieto and Serra, as well as new targets at Valdette, Demetrio and Sr. Domingo. The distribution of the holes was as follows:

- 14 holes (totaling 2,897 m) in the Serra zone
- 12 holes (totaling 3,209 m) in the Meio-Onca zone
- 11 holes (totaling 1737 m) in the Valdette zone
- 4 holes (totaling 897 m) in the Demetrio zone
- 4 holes (totaling 703 m) in the Sr. Domingo zone
- 5 holes (totaling 1,205 m) in the Come Quieto zone
- 1 hole (totaling 155 m) in the Mae de Leite zone

Average core recoveries were over 99%

10.6 Standard Logging Procedure

The following is a summary of the logging protocols in place.

- Core logging took place in a well-lit and secure facility (**See Photo 10.1**).
- The drilling contractor provided core recovery and Magellan's technician checked and verified the information.
- Core photography was completed at this stage.
- A project geologist logged lithology, alteration, mineralogy, structures and marked the core samples.
- Data from the core logging was added to the drill hole data base (Microsoft Access).
- The core was stored in secured, well labeled racks.

Drill core logs contain the following information:

- Drilling header information: drill-hole number, collar coordinates and elevation, location, azimuth, dip, length, geologist, drilling dates, and core diameter.
- Core recovery

- Sample data: sample number with from-to intervals.
- Graphic log: columns for displaying the lithology.
- Letter codes for digital data base for lithology (rock type, composition, form, and texture), alteration (type, style, intensity, and mineralogy), mineralization (type, style, mineralogy, and %), and structures (type and angle to core).

Photo 10.1 Core Shack

10.7 Mineralized Zones

10.7.1 Valdette zone

Eleven holes were drilled at the Valdette zone, six of them with 45° azimuth and 50° dip, and five with a 225° azimuth and 50° dip. The lengths of the holes vary between 158 m and 345 m, with a spacing between holes ranging from 100 m to 240 m. The drill holes intersected the mineralized structure approximately perpendicular to the strike. The Valdette mineralized structure dips from 80° to 90° to the NE. So far, 1,130 m of mineralized structure has been identified by drilling and trenching. **Table 10.3** shows the main intercepts.

Table 10.3 Valdette Main Intercepts

BHID	FROM (m)	TO (m)	SAMPLE_NO	AU_PPB
BR_COR_DDH110	94,95	95,95	DS0031143	2376
BR_COR_DDH110	110,55	111,55	DS0031163	1919
BR_COR_DDH110	142,55	143,05	DS0031181	1686
BR_COR_DDH110	183,75	184,25	DS0031233	2472
BR_COR_DDH110	184,25	184,75	DS0031234	3138
BR_COR_DDH112	157,5	158	DS0031304	2104
BR_COR_DDH112	211	211,5	DS0031332	1446
BR_COR_DDH112	212	212,5	DS0031334	1304
BR_COR_DDH112	212,5	213	DS0031335	3234
BR_COR_DDH112	213	213,5	DS0031336	1619
BR_COR_DDH112	213,5	214	DS0031337	1300
BR_COR_DDH112	214	214,5	DS0031338	1572
BR_COR_DDH112	216	217	DS0031342	2959
BR_COR_DDH112	217	218	DS0031343	1206
BR_COR_DDH112	220	221	DS0031346	1131
BR_COR_DDH114	184,45	185,45	DS0031629	4386
BR_COR_DDH114	185,45	186,45	DS0031630	1469
BR_COR_DDH114	188,45	189,45	DS0031633	1314
BR_COR_DDH114	189,45	190,45	DS0031634	1464
BR_COR_DDH114	190,45	191,45	DS0031635	1334
BR_COR_DDH114	195,65	196,15	DS0031643	1893
BR_COR_DDH114	203,15	203,65	DS0031660	1512
BR_COR_DDH114	204,15	204,65	DS0031662	2808
BR_COR_DDH114	207,65	208,15	DS0031670	1686
BR_COR_DDH114	208,15	209,15	DS0031671	1213
BR_COR_DDH114	218,15	219,15	DS0031681	1068
BR_COR_DDH114	278,5	279,5	DS0031701	1210
BR_COR_DDH114	283,75	284,75	DS0031703	1317
BR_COR_DDH114	288,5	289,7	DS0031705	1287
BR_COR_DDH114	297	298,2	DS0031708	2219
BR_COR_DDH114	300,65	301,65	DS0031709	1951
BR_COR_DDH119	121	121,5	DS0032018	1076
BR_COR_DDH125	52,7	53,2	DS0032511	2178
BR_COR_DDH125	85,7	86,7	DS0032529	5433
BR_COR_DDH125	86,7	87,7	DS0032530	1149
BR_COR_DDH125	87,7	88,7	DS0032531	1675
BR_COR_DDH125	92,5	93	DS0032536	1158
BR_COR_DDH125	96	97	DS0032540	1840
BR_COR_DDH125	188,55	189,55	DS0032616	2396
BR_COR_DDH128	84	84,5	DS0032842	2024
BR_COR_DDH128	84,5	85	DS0032843	3636
BR_COR_DDH128	85	85,5	DS0032844	1600
BR_COR_DDH128	119	120	DS0032906	1521
BR_COR_DDH131	164	164,5	DS0033111	2448
BR_COR_DDH131	164,5	165	DS0033112	1437
BR_COR_DDH131	165,5	166	DS0033114	2177
BR_COR_DDH133	189	189,5	DS0033306	1338
BR_COR_DDH133	191	192	DS0033310	2350
BR_COR_DDH133	216	216,5	DS0033353	1769

10.7.2 Galena zone

In this zone, seventeen holes were drilled at Galena, with an orientation ranging from 240° to 250° and dipping 50°. The lengths of the holes vary from 68 m to 173 m. Drill holes intersected the structure at approximately 90° and the estimated true widths of all of the mineralized intervals are approximately 82% of the intersected widths in the holes. **Table 10.4** shows the main intercepts.

Table 10.4 Galena Main Intercepts

BHID	FROM (m)	TO (m)	SAMPLE_NO	AU_PPB
BR_COR_DDH03	33	33.5	DS0021981	1133
BR_COR_DDH03	32	32.5	DS0021979	1289
BR_COR_DDH03	32.5	33	DS0021980	3931
BR_COR_DDH03	35	35.5	DS0021985	11579
BR_COR_DDH04	3	4	DS0027048	904
BR_COR_DDH04	74	74.5	DS0027130	4676
BR_COR_DDH04	74.5	75	DS0027131	57918
BR_COR_DDH05	26.5	27	DS0027190	1043
BR_COR_DDH05	27.5	28	DS0027193	1001
BR_COR_DDH05	27	27.5	DS0027192	3025
BR_COR_DDH05	32.5	33	DS0027203	8763
BR_COR_DDH05	33	33.5	DS0027204	10653
BR_COR_DDH06	12.5	13	DS0027270	1571
BR_COR_DDH06	30.5	31	DS0027291	582
BR_COR_DDH23	123.5	124	DS0023006	1474
BR_COR_DDH24	72.5	73	DS0023026	4934
BR_COR_DDH25	131	131.5	DS0023102	3047
BR_COR_DDH25	133	134	DS0023105	1376
BR_COR_DDH26	52.1	52.6	DS0023063	5066
BR_COR_DDH28	75.8	76.3	DS0023201	882
BR_COR_DDH29	45	45.5	DS0023210	2071
BR_COR_DDH29	45.5	46	DS0023211	41151
BR_COR_DDH30	51.1	51.6	DS0023249	2590
BR_COR_DDH30	50.1	50.6	DS0023247	3097
BR_COR_DDH30	49.6	50.1	DS0023246	54493
BR_COR_DDH30	53.1	53.6	DS0023253	8603
BR_COR_DDH34	65.9	66.4	DS0023365	19978
BR_COR_DDH34	66.9	67.4	DS0023367	43029
BR_COR_DDH34	67.4	67.9	DS0023368	447
BR_COR_DDH34	67.9	68.4	DS0023369	1257
BR_COR_DDH34	85.9	86.4	DS0023389	6317
BR_COR_DDH36	78	78.5	DS0023416	1385
BR_COR_DDH36	78.5	79	DS0023417	981

BHID	FROM (m)	TO (m)	SAMPLE_NO	AU_PPB
BR_COR_DDH36	79.4	79.9	DS0023419	23917
BR_COR_DDH58	138	138.5	DS0024450	3674
BR_COR_DDH60	82.5	83	DS0024460	2111

10.7.3 Eloy-Juara Zone

Ten holes were drilled in this zone, with a general orientation ranging from 225° to 254° and dipping at 50°. The lengths of the holes vary from 66 m to 131 m. The intersection was perpendicular to the structure, and the estimated true widths of all the mineralized intervals are approximately 85% of the intersected widths in the holes. **Table 10.5** shows the main intercepts.

Table 10.5 Eloy-Juara Main Intercepts

BHID	FROM (m)	TO (m)	SAMPLE_NO	AU_PPB
BR_COR_DDH17	51.5	52	DS0028827	27799
BR_COR_DDH17	52	52.5	DS0028828	31482
BR_COR_DDH17	52.5	53	DS0028829	7861
BR_COR_DDH33	119.3	119.8	DS0023348	1402
BR_COR_DDH33	120.8	121.3	DS0023351	2749
BR_COR_DDH33	121.3	121.8	DS0023352	1665
BR_COR_DDH53	44.4	44.9	DS0024229	7470
BR_COR_DDH56	56	57	DS0024327	1188

10.7.4 Mae de Leite zone

Twelve holes were drilled at the Mae de Leite zone; seven of them (azimuth 250° and dip ranging from 50° to 70°) tested and proved the continuity of the Mae de Leite stucture at depth and along strike. Four holes were drilled to test a soil anomaly 200 m to the west of the Mae de Leite garimpo. The vein structure dips vertically to sub vertically (75° to 90° to the NE). Approximately 370 m of mineralized structure has been identified so far. **Table 10.6** shows the main intercepts.

Table 10.6 Mae de Leite Main Intercepts

BHID	FROM (m)	TO (m)	SAMPLE_NO	AU_PPB
BR_COR_DDH100	85,4	85,9	DS0031009	1317
BR_COR_DDH101	74,5	75,1	DS0031029	2351
BR_COR_DDH101	75,6	76,1	DS0031031	1112
BR_COR_DDH101	81,7	82,3	DS0031041	4487
BR_COR_DDH106	110,2	110,8	DS0031078	1651
BR_COR_DDH96	123,2	123,85	DS0030869	6270
BR_COR_DDH96	123,85	124,5	DS0030870	9517
BR_COR_DDH98	46	46,5	DS0030932	1339
BR_COR_DDH98	48,5	49	DS0030938	18500
BR_COR_DDH98	49	49,5	DS0030939	9227
BR_COR_DDH98	52,5	53	DS0030947	1688
BR_COR_DDH98	54,5	55	DS0030951	1306

BHID	FROM (m)	TO (m)	SAMPLE_NO	AU_PPB
BR_COR_DDH99	75,5	76	DS0030975	2171
BR_COR_DDH99	76	76,5	DS0030976	1515
BR_COR_DDH99	76,5	77	DS0030977	3963
BR_COR_DDH99	77	77,5	DS0030978	2659

10.7.5 Meio-Onca zone

Forty eight holes were drilled at Meio-Onca, eight of them with an azimuth ranging from 30° to 58° and forty with azimuth ranging from 210° to 245°. The dip of the holes ranged from 50° to 75° and length from 60 m to 290 m. The mineralized structure dips between 80° to the SW to 80° to the NE. The Meio zone has been traced for approximately 2,000 m. **Table 10.7** shows the main intercepts

Table 10.7 Meio-Onca Main Intercepts

BHID	FROM (m)	TO (m)	SAMPLE_NO	AU_PPB
BR_COR_DDH108	69,8	70,3	DS0031107	6643
BR_COR_DDH11	50	50,5	DS0028354	25194
BR_COR_DDH11	50,5	51	DS0028355	55107
BR_COR_DDH11	51	51,5	DS0028356	13456
BR_COR_DDH12	49	50	DS0028424	3429
BR_COR_DDH13	76	76,5	DS0028558	11224
BR_COR_DDH130	40	40,5	DS0032964	22700
BR_COR_DDH130	197	197,5	DS0033073	8800
BR_COR_DDH130	246	246,5	DS0033081	5313
BR_COR_DDH130	246,5	247	DS0033082	8400
BR_COR_DDH130	255,55	256,05	DS0033091	52200
BR_COR_DDH134	198	198,5	DS0033417	1007
BR_COR_DDH137	158,3	158,9	DS0033620	1984
BR_COR_DDH14	44	44,5	DS0028621	1218
BR_COR_DDH14	55,5	56	DS0028646	2320
BR_COR_DDH14	56	56,5	DS0028647	1402
BR_COR_DDH144	262	262,5	DS0034214	9163
BR_COR_DDH144	262,5	263	DS0034215	23900
BR_COR_DDH144	263	263,5	DS0034216	14900
BR_COR_DDH149	127,5	128	DS0034260	1897
BR_COR_DDH152	75	75,5	DS0034481	3782
BR_COR_DDH152	279,2	279,7	DS0035131	1214
BR_COR_DDH155	91	91,5	DS0035182	2415
BR_COR_DDH155	91,5	92	DS0035184	1480
BR_COR_DDH155	92	92,5	DS0035185	1409
BR_COR_DDH155	92,5	93	DS0035187	23000
BR_COR_DDH155	93	93,5	DS0035188	1116
BR_COR_DDH155	95	96	DS0035191	2563
BR_COR_DDH156	165,5	166	DS0035328	2622
BR_COR_DDH157	141,5	142	DS0036397	1094
BR_COR_DDH157	142,5	143	DS0036399	16000
BR_COR_DDH157	144	145	DS0036402	1997
BR_COR_DDH159	114,7	115,2	DS0036507	2543
BR_COR_DDH62	56	56,5	DS0024500	3469
BR_COR_DDH62	56,5	57	DS0024501	59798
BR_COR_DDH62	57	57,5	DS0024502	25556
BR_COR_DDH62	57,5	58	DS0024503	149798
BR_COR_DDH62	58	58,5	DS0024504	82310
BR_COR_DDH62	58,5	59	DS0024505	73701
BR_COR_DDH62	62	62,5	DS0024512	1043

BHID	FROM (m)	TO (m)	SAMPLE_NO	AU_PPB
BR_COR_DDH65	113,5	114	DS0024710	3191
BR_COR_DDH65	114	114,5	DS0024711	3608
BR_COR_DDH65	114,5	115	DS0024712	4157
BR_COR_DDH65	116	116,5	DS0024715	1857
BR_COR_DDH65	116,5	117	DS0024716	10808
BR_COR_DDH65	117	117,5	DS0024718	4524
BR_COR_DDH65	117,5	118	DS0024719	6200
BR_COR_DDH65	118	118,5	DS0024720	1295
BR_COR_DDH67	20,5	21	DS0029707	1183
BR_COR_DDH67	22	22,5	DS0029710	3839
BR_COR_DDH67	22,5	23	DS0029711	1427
BR_COR_DDH67	23,5	24	DS0029713	1083
BR_COR_DDH67	24	24,5	DS0029714	1184
BR_COR_DDH67	107,5	108	DS0029750	20039
BR_COR_DDH67	108	108,5	DS0029751	13675
BR_COR_DDH67	108,5	109	DS0029752	36653
BR_COR_DDH67	115,5	116	DS0029760	1871
BR_COR_DDH67	116	116,5	DS0029762	1691
BR_COR_DDH68	149	149,5	DS0029789	4817
BR_COR_DDH68	149,5	150	DS0029790	3560
BR_COR_DDH68	152,5	153	DS0029796	3464
BR_COR_DDH68	153	153,5	DS0029797	2543
BR_COR_DDH68	153,5	154	DS0029798	1917
BR_COR_DDH68	154	154,5	DS0029799	23933
BR_COR_DDH68	154,5	155	DS0029800	9736
BR_COR_DDH69	169,8	170,3	DS0029858	1149
BR_COR_DDH70	123,5	124	DS0029889	1182,25
BR_COR_DDH71	58,9	59,9	DS0029923	2016
BR_COR_DDH71	60,4	60,9	DS0029925	27964
BR_COR_DDH71	60,4	60,9	DS0029925	28483
BR_COR_DDH71	60,9	61,4	DS0029926	16836
BR_COR_DDH71	61,9	62,4	DS0029928	2728
BR_COR_DDH71	62,4	62,9	DS0029929	7357
BR_COR_DDH72	123,8	124,4	DS0029979	11403
BR_COR_DDH76	129,35	130,1	DS0030047	3018
BR_COR_DDH76	130,85	131,6	DS0030049	1569
BR_COR_DDH77	8,2	8,7	DS0030057	3120
BR_COR_DDH78	32,5	33	DS0030055	1722
BR_COR_DDH78	127,4	127,8	DS0030092	34309
BR_COR_DDH78	127,8	128,2	DS0030093	32951
BR_COR_DDH78	128,6	129,1	DS0030095	25191
BR_COR_DDH79	57,6	58,1	DS0030080	23875
BR_COR_DDH79	58,1	58,6	DS0030081	10139
BR_COR_DDH79	58,6	59,1	DS0030082	68730
BR_COR_DDH79	59,1	59,6	DS0030083	20692
BR_COR_DDH79	59,6	60,1	DS0030084	3393
BR_COR_DDH80	61,5	62	DS0030106	1500
BR_COR_DDH80	227,13	227,63	DS0030126	1115
BR_COR_DDH81	61,05	61,55	DS0030112	11455
BR_COR_DDH81	61,55	62,05	DS0030113	6228
BR_COR_DDH81	62,05	62,55	DS0030114	4168
BR_COR_DDH82	36	36,5	DS0030210	6838
BR_COR_DDH82	36,5	37	DS0030212	7484
BR_COR_DDH82	37	37,5	DS0030213	23600
BR_COR_DDH82	37,5	38	DS0030215	2593
BR_COR_DDH83	88,5	89,1	DS0030272	18200
BR_COR_DDH85	123,7	124,2	DS0030331	1310
BR_COR_DDH85	136,2	136,7	DS0030340	2276

BHID	FROM (m)	TO (m)	SAMPLE_NO	AU_PPB
BR_COR_DDH86	86,9	87,4	DS0030380	1685
BR_COR_DDH86	96,4	96,9	DS0030401	1920
BR_COR_DDH86	97,4	98	DS0030403	3408
BR_COR_DDH88	43,8	44,3	DS0030456	1611
BR_COR_DDH88	44,3	44,8	DS0030457	1477
BR_COR_DDH89	62	62,5	DS0030472	2722
BR_COR_DDH89	62,5	63	DS0030473	1526
BR_COR_DDH93	64	64,5	DS0030647	3800
BR_COR_DDH93	99,6	100,1	DS0030683	1130
BR_COR_DDH94	26,6	27,1	DS0030701	2571
BR_COR_DDH94	27,7	28,2	DS0030703	8045
BR_COR_DDH94	28,2	28,7	DS0030704	11800
BR_COR_DDH94	28,7	29,2	DS0030705	3434
BR_COR_DDH94	119,7	120,3	DS0030739	3191
BR_COR_DDH95	64,7	65,2	DS0030768	6820

10.7.6 Come Quieto zone

Nine holes were drilled at the Come Quieto zone, six of them with azimuth ranging from 225° to 240° with a 50° dip and three with azimuth ranging from 45° to 65° with a 50° dip. The length of the holes varies from 85 m to 269 m. The mineralized structure dips vertically to sub-vertically. The Come Quieto zone has been traced for approximately 850 m. **Table 10.8** shows the main intercepts.

Table 10.8 Come Quieto Main Intercepts

BHID	FROM (m)	TO (m)	SAMPLE_NO	AU_PPB
BR_COR_DDH07	42,5	43	DS0027393	1407
BR_COR_DDH07	43	43,5	DS0027394	4408
BR_COR_DDH07	44	44,5	DS0027396	1537
BR_COR_DDH07	44,5	45	DS0027397	1925
BR_COR_DDH07	46	46,5	DS0027400	4144
BR_COR_DDH09	19	19,5	DS0028070	1026
BR_COR_DDH120	152,8	153,4	DS0032134	1521
BR_COR_DDH122	51	51,5	DS0032277	9600
BR_COR_DDH122	132,95	133,95	DS0032303	1827
BR_COR_DDH122	134,85	135,45	DS0032305	21400
BR_COR_DDH126	115,5	116	DS0032703	2185
BR_COR_DDH126	116	116,5	DS0032704	3560
BR_COR_DDH126	129	129,5	DS0032724	1622
BR_COR_DDH126	129,5	130	DS0032725	2995
BR_COR_DDH126	130	130,5	DS0032726	4775
BR_COR_DDH126	130,5	131	DS0032727	9861
BR_COR_DDH126	131	131,5	DS0032728	3442
BR_COR_DDH126	131,5	132	DS0032729	3345
BR_COR_DDH126	132,5	133	DS0032731	1368
BR_COR_DDH126	134	134,5	DS0032736	1151
BR_COR_DDH126	140	141	DS0032743	1947
BR_COR_DDH126	152	153	DS0032757	1690
BR_COR_DDH126	171	171,5	DS0032783	1069
BR_COR_DDH126	185	185,5	DS0032797	10100
BR_COR_DDH151	142,75	143,25	DS0036122	1145
BR_COR_DDH151	143,25	143,75	DS0036123	22600
BR_COR_DDH154	50,6	51	DS0036198	3994
BR_COR_DDH154	158,5	159	DS0036249	6183
BR_COR_DDH154	160,2	160,8	DS0036252	1685

BHID	FROM (m)	TO (m)	SAMPLE_NO	AU_PPB
BR_COR_DDH154	160,8	161,4	DS0036253	1846
BR_COR_DDH154	163	163,5	DS0036257	2028
BR_COR_DDH154	179	180	DS0036278	2863

10.7.7 Serra zone

Thirty seven holes were drilled at the Serra zone, with azimuths ranging from 240° to 250° and dip from 50° to 75°. The lengths vary from 65 m to 282 m. The Serra veins dip sub-vertically (70° to 80° to the NE) and over 1,100 m of mineralized structure has been identified so far. **Table 10.9** shows the main intercepts.

Table 10.9 Serra Main Intercepts

BHID	FROM (m)	TO (m)	SAMPLE_NO	AU_PPB
BR_COR_DDH01	34	34,5	DS0021138	86915
BR_COR_DDH01	47	48	DS0021154	7746
BR_COR_DDH01	61	61,5	DS0021170	15513
BR_COR_DDH02	22,5	23	DS0021859	1286
BR_COR_DDH02	60	60,5	DS0021906	1654
BR_COR_DDH121	46	47	DS0032212	4420
BR_COR_DDH121	90,25	90,75	DS0032236	230500
BR_COR_DDH121	90,75	91,25	DS0032237	6041
BR_COR_DDH124	94	94,5	DS0032458	1253
BR_COR_DDH124	95	95,5	DS0032460	1162
BR_COR_DDH124	100	100,5	DS0032472	6495
BR_COR_DDH124	106,5	107	DS0032482	1370
BR_COR_DDH124	115	115,5	DS0032494	1778
BR_COR_DDH124	119,5	120	DS0032500	2910
BR_COR_DDH124	120	120,5	DS0032501	2376
BR_COR_DDH124	120,5	121	DS0032502	25400
BR_COR_DDH127	102,36	102,86	DS0032812	32900
BR_COR_DDH127	103,86	104,36	DS0032815	9430
BR_COR_DDH132	117	117,5	DS0033220	4404
BR_COR_DDH132	136,4	136,9	DS0033233	2906
BR_COR_DDH132	136,9	137,4	DS0033234	17200
BR_COR_DDH132	137,4	137,9	DS0033235	4385
BR_COR_DDH132	139,4	139,9	DS0033239	30400
BR_COR_DDH132	139,9	140,4	DS0033241	2278
BR_COR_DDH135	84,8	85,4	DS0033451	4840
BR_COR_DDH135	127	127,5	DS0033469	12100
BR_COR_DDH138	146	146,5	DS0033717	1312
BR_COR_DDH138	150	150,5	DS0033727	1005
BR_COR_DDH139	168,1	168,7	DS0033745	17600
BR_COR_DDH139	179,7	180,2	DS0033757	2189
BR_COR_DDH141	183	183,6	DS0033768	1495
BR_COR_DDH141	187,6	188,6	DS0033774	1125
BR_COR_DDH141	188,6	189,1	DS0033775	2777
BR_COR_DDH145	207,5	208	DS0033791	1309
BR_COR_DDH145	209	209,5	DS0033794	2085
BR_COR_DDH145	209,5	210	DS0033795	2197
BR_COR_DDH145	211,95	212,5	DS0033799	3138
BR_COR_DDH145	212,5	213	DS0033800	1470
BR_COR_DDH145	213	213,5	DS0033801	10000
BR_COR_DDH145	213,5	214	DS0033802	42600
BR_COR_DDH145	214	214,5	DS0033803	10200
BR_COR_DDH145	214,5	215	DS0033804	9349

BHID	FROM (m)	TO (m)	SAMPLE_NO	AU_PPB
BR_COR_DDH145	215,5	216	DS0033806	1912
BR_COR_DDH148	142,5	143	DS0033839	34500
BR_COR_DDH153	189,3	189,8	DS0033897	1898
BR_COR_DDH160	98,1	98,6	DS0033973	5837
BR_COR_DDH160	98,6	99,1	DS0033974	59400
BR_COR_DDH19	40	40,5	DS0028946	1586
BR_COR_DDH20	36	36,5	DS0029503	34729
BR_COR_DDH20	37	37,5	DS0029505	1861
BR_COR_DDH20	39	39,5	DS0029509	20683
BR_COR_DDH37	55	55,5	DS0023462	4237
BR_COR_DDH37	55,5	56	DS0023463	2674
BR_COR_DDH37	73	73,6	DS0023486	3908
BR_COR_DDH37	81	81,5	DS0023500	2630
BR_COR_DDH37	87	88	DS0023504	2326
BR_COR_DDH37	92	92,5	DS0023513	1181
BR_COR_DDH38	33,2	33,7	DS0023538	1129
BR_COR_DDH38	33,7	34,2	DS0023540	1640
BR_COR_DDH38	36,2	36,8	DS0023545	22303
BR_COR_DDH38	36,8	37,3	DS0023546	1970
BR_COR_DDH38	75,7	76,2	DS0023579	7874
BR_COR_DDH39	19	19,5	DS0023641	3075
BR_COR_DDH39	19,5	20	DS0023642	2068
BR_COR_DDH39	66,6	67,1	DS0023697	14944
BR_COR_DDH41	60,5	61,1	DS0023727	35114
BR_COR_DDH41	61,1	61,5	DS0023728	9245
BR_COR_DDH42	38	38,5	DS0023749	3794
BR_COR_DDH42	39	39,5	DS0023751	63522
BR_COR_DDH42	76	76,5	DS0023800	2654
BR_COR_DDH43	56	57	DS0023867	1740
BR_COR_DDH43	57	57,5	DS0023868	3949
BR_COR_DDH43	60	60,7	DS0023874	6680
BR_COR_DDH45	28,2	28,7	DS0023935	3870
BR_COR_DDH45	44,5	45	DS0023957	5056
BR_COR_DDH45	81	81,5	DS0023991	24687
BR_COR_DDH46	112,1	112,7	DS0024011	19500
BR_COR_DDH49	62,6	63,1	DS0024121	7312
BR_COR_DDH50	42,6	42,8	DS0024178	6489
BR_COR_DDH55	56	57	DS0024274	44507
BR_COR_DDH55	88,5	89	DS0024282	1275
BR_COR_DDH55	92,5	93	DS0024288	3782
BR_COR_DDH55	103	103,5	DS0024294	7030
BR_COR_DDH55	104,5	105	DS0024297	1730
BR_COR_DDH55	108,4	109	DS0024304	15851
BR_COR_DDH57	98	98,5	DS0024350	48095
BR_COR_DDH59	96,5	97,2	DS0024410	12722
BR_COR_DDH59	111,5	112,1	DS0024425	1191
BR_COR_DDH59	112,1	112,6	DS0024426	8128
BR_COR_DDH61	63,5	64	DS0024540	6417
BR_COR_DDH61	98,5	99	DS0024565	42981
BR_COR_DDH61	99	100	DS0024566	2471
BR_COR_DDH61	101,7	102,2	DS0024569	33555
BR_COR_DDH64	100,5	101	DS0024627	3769
BR_COR_DDH64	101	101,5	DS0024628	1631
BR_COR_DDH64	157,5	158	DS0024649	1557

10.7.8 Demetrio zone

Four holes were drilled at the Demetrio zone with an azimuth of 235° and dip of 50°. Lengths ranged from 198 m to 281 m depth. The mineralized structure dips 85° to the SW, and has been traced for 600 m along strike in drill holes. **Table 10.10** shows the main intercepts.

Table 10.10 Demetrio Main Intercepts

BHID	FROM (m)	TO (m)	SAMPLE_NO	AU_PPB
BR_COR_DDH111	118,2	118,8	DS0031246	9500
BR_COR_DDH111	141	142	DS0031265	4375
BR_COR_DDH111	159	160	DS0031276	1158
BR_COR_DDH113	119	119,5	DS0031515	1093
BR_COR_DDH113	119,5	120	DS0031516	2232
BR_COR_DDH113	120,5	121	DS0031518	1644
BR_COR_DDH113	121,5	122	DS0031521	1790
BR_COR_DDH115	81,5	82	DS0031724	1429
BR_COR_DDH115	82	82,5	DS0031725	3297
BR_COR_DDH115	82,5	83	DS0031726	6970
BR_COR_DDH115	83	83,5	DS0031727	34900
BR_COR_DDH115	83,5	84	DS0031728	2546
BR_COR_DDH115	84	84,5	DS0031729	2819
BR_COR_DDH115	84,5	85	DS0031730	2148
BR_COR_DDH115	85	85,5	DS0031731	5436
BR_COR_DDH115	85,5	86	DS0031732	8040
BR_COR_DDH115	86	86,5	DS0031733	1673
BR_COR_DDH115	86,5	87	DS0031734	2110
BR_COR_DDH116	262,9	263,9	DS0031791	1065
BR_COR_DDH116	264,9	265,4	DS0031793	2903
BR_COR_DDH116	265,4	265,9	DS0031794	2108

10.7.9 Sr. Domingo zone

Four holes were drilled at the Sr. Domingo zone, (on the northern edge of the Domingo soil anomaly), with 210° azimuths and dips of 50°. Length ranged from 111 m to 275 m and the average spacing between holes was 140 m. **Table 10.11** shows the main intercepts;

Table 10.11 Sr. Domingo Main Intercepts

BHID	FROM (m)	TO (m)	SAMPLE_NO	AU_PPB
BR_COR_DDH142	59,7	60,2	DS0034610	2780
BR_COR_DDH142	62,7	63,2	DS0034616	1054
BR_COR_DDH142	64,2	64,7	DS0034619	1317
BR_COR_DDH143	53,8	54,3	DS0034703	1136
BR_COR_DDH147	168,5	169	DS0034994	1203

11.0 SAMPLE PREPARATION, ANALYSES AND SECURITY

11.1 Sample Preparation

11.1.1 March – June 2007 – 1st Phase Drilling Program

Magellan diamond drill hole samples were prepared on site at the Coringa Project using the following protocols (**See Photo 11.1**):

- The holes were continuously sampled at intervals of approximately 0.5 m (mineralized zones) to 1 m (non-mineralized zones).
- Magellan's Geologist or Technician was responsible for the core handling procedures at the drill rigs verifying:
    - Core is removed from the core barrel without any loss.
    - Core is properly reassembled and placed in the core box in the correct orientation. After each drill run, the depth of the hole is marked with a wooden block.
    - Full core boxes are securely covered and transported at the end of each shift to the core shack at the camp.
- Core boxes have the drill-hole number, box number "from – to" meter noted on an aluminum tag attached to the front of the box.
- Each box is photographed to provide a visual record of the core.
- Core recoveries are calculated by the drilling contractor (Boart Longyear) and verified by Magellan's technician.

A cutting/splitting guide line is marked on the core by the geologist to ensure that the mineralized structure is equally divided.

The core is then cut in half along the indicated line with the onsite core saw. The saw contains a metal blade suitable for cutting stone, and a guide table to ensure the core is cut evenly. Water lines run directly to the blade to cool the saw and prevent sample contamination.

- Both halves are placed in the core box and placed on a logging table.
- During the logging, the sample intervals down the hole are marked on the box. An aluminum tag showing the sample number is attached to the core box.
- The core was sampled at 1 m intervals, and within the mineralized zone the sample interval was reduced to 0.5 m.
- One half of the core is returned to the core box while the other half is placed in the numbered and tagged sample bag.
- The bags are immediately sealed with a plastic fastener.
- The samples tags have pre-assigned sample numbers to account for the insertion of the one blank, 2 standards, and 2 duplicates every 42 samples.

- Samples are then entered in the database.
- Groups of bagged samples are placed in larger sacks, which are marked showing samples numbers.
- Samples are shipped using a secure private transport company to Cuiaba (phase 1 drilling) or Novo Progresso (phase 2 & 3 drilling).
- In Cuiaba or Novo Progresso, samples are checked and registered by Magellan personnel before being dispatched by secure truck (Cooperativa Burbure) to SGS Geosol Laboratory in Belo Horizonte.

Photo 11.1 Sampling Procedure

11.1.2 October 2007 – March 2008: Phase 2 Diamond-Drilling Program and September 2008 – December 2008: Phase 3 Diamond Drilling.

Sample procedures stayed the same for the Second Phase Drilling Program, with the difference being holes were not continuously sampled. Samples were only taken from the mineralized zones and 2 m to 4 m before and after the structure. The mineralized zone was sampled using a 0.5 m sampling interval.

During the Third Phase Drilling, the 0.5 m sampling interval started where the high-grade mineralization was thought to start, and kept sampling 0.5 m samples up to the end of the mineralized structure.

Mineralized zones for both the second and third drilling campaign were readily identified by the presence of the vein (hydrothermal breccias) or sericitic alteration of the host rock which was identified during the first drilling campaign.

During the second phase and third phase drilling programs, drill core samples were sent from the field camp to Magellan's office in Novo Progresso before shipping via secure transport to SGS preparation laboratory in Itaituba. From Itaituba, SGS sent the pulp samples to Belo Horizonte for assaying.

11.1.3 March 2010 – December 2011: Phase 4 Diamond-Drilling Program

Sample procedures stayed the same for the Fourth Phase Drilling Program.

During the fourth phase program, samples were sent from the field camp to Chapleau-Magellan's office in Novo Progresso before shipping via secure transport to ACME preparation laboratory in Itaituba. From Itaituba, ACME sent the sample pulps to Santiago de Chile for assaying.

11.2 Sample Analysis

All core samples from the first to third drill programs were sent to SGS Geosol laboratories in Belo Horizonte and/or Itaituba for sample preparation and then sent to SGS Geosol. Each hole was submitted as a separate lot to the laboratory. The core samples were then crushed down to 2 mm, a split of 250 to 300 g was taken and pulverized to better than 95% minus 150 mesh. Gold analysis was by fire assay of a 30 g sample and an atomic absorption finish. Silver and 32 additional elements, using aqua regia acid digestion, were analyzed by Inductively Coupled Plasma (ICP). SGS Geosol, which is an ISO 9001:2000 and ISO 14001: 2004 registered laboratory, has a quality control program in place which includes standards, blanks, repeats, and duplicates.

Core samples for the fourth phase of drilling were sent to the ACME laboratory in Itaituba for sample preparation and then the sample pulps were sent to the ACME assay laboratory in Santiago de Chile. Core samples were crushed down to 2 mm and a 1 kg split taken and pulverized to better than 85% minus 200 mesh. Gold analysis was by fire assay of a 30 g sample with inductively coupled plasma atomic emission spectroscopy (ICP-ES), atomic adsorption spectroscopy (ASS), or gravimetric finish. Some samples were also assayed for Silver and 35 additional elements, using a 4 acid digestion, and analyzed by Inductively Coupled Plasma Mass Spectrometry (ICP-MS). ACME is an ISO 9001 registered laboratory and has a quality control program in place which includes standards, blanks, repeats, and duplicates.

11.3 Quality Assurance/Quality Control (QA/QC) Programs

At the time the core samples are bagged, duplicate, standard and blank samples were inserted in the sample sequence with the normal core samples to monitor sampling variances, laboratory precision and accuracy, and to identify problems caused by poor sampling, preparation and other assaying practices, possible sample contamination, and other parameters.

The QA/QC program included the insertion of two standards, two duplicates and one blank every 42 samples. In some cases duplicates samples were not inserted consistently into the data set with 0 to three duplicates added every 42 samples. This was due to the control documents which did not contain a defined sample number for the duplicate but rather relied on the technician to select the appropriate number of duplicate samples. Assay results for the first through third drilling programs were monitored internally and by Dr. Owen Miller, an independent qualified geologist contracted by Magellan. Assay results for the fourth drilling campaign were monitored directly by Magellan. **Table 11.1** shows a summary list of control samples.

Table 11.1 Summary QA/QC Program

Type Sample	Description	Number of Samples
	Total number of samples	9139
	Number of control samples	1922 (21.03%)
Sampling	Duplicates	315 (3.44%)
	Standards	421 (4.60%)
	Blanks	212 (2.32%)
Assaying	Lab - Duplicates	353 (3.86%)
	Lab - Repetition	275 (3.00%)
	Second Lab Checking	346 (3.78%)

SGS Geosol in Belo Horizonte was used as the primary laboratory for the first 3 drilling programs, and ALS Chemex, in Belo Horizonte was used as a secondary laboratory. ACME was used as the primary laboratory for the fourth drilling program and SGS Geosol was used as the secondary laboratory.

11.4 QA/QC Review

GRE completed a review of the sample quality assurance and control (QA/QC) program employed by Magellan during all four phases of drilling. The QA/QC program included the insertion of two standards, two duplicates and one blank every 42 samples. In some cases, duplicates samples were not inserted consistently into the data set with 0 to three duplicates added every 42 samples. GRE believes the program reflects that of a modern exploration company completing work to industry standards.

During the review of the QA/QC program, two items of concern were noted with respect to the blank and duplicate samples. During the first three drilling programs, Magellan sent sample 'blanks' to the lab in the form of pulverized cement powder. Although previously pulverized blanks can verify that the final fire assay was free of contamination and correct calibration of

the equipment, it cannot confirm that the sample preparation method was free of contamination. However, the sample set from the first three drilling campaigns has several samples that contain high gold values followed by samples below the detection limit. This sequence of results would tend to indicate that grade cross-contamination did not occur in the sample preparation and testing process; therefore it is GRE's opinion that the sample preparation procedure for the first three drilling campaigns produced uncontaminated samples. This procedure was changed during the fourth drilling campaign to use inert granite sourced from a local granite quarry and sent in aggregate form to the lab as blank samples. This material had previously been analyzed and returned results below the detection limit for gold. Sending blank material in aggregate form and not powder provided a means to check cross contamination between samples due to the laboratory sample preparation process.

Magellan created sample 'duplicates' on site utilizing a method that creates a disparity in the duplicates. The procedure employed was to place a one-half core split in a plastic bag and to then crush the core with a hammer. The resulting crushed material was mechanically hand-mixed, and then divided into equal portions for shipment. The use of this procedure introduces a bias in the sampled since they are divided at a very coarse particle size. The bias is compounded by the fact that the majority of the mineralization is represented in discrete veins which are typically represented by a fraction of the particles at the large particle size. Due to this procedure, comparison of the duplicates created by Magellan is poor. In contrast, the laboratory preparation duplicates compare well with each other showing the sampling program was unbiased. **Figure 11.1** below shows a comparison graph of the laboratory duplicates. **Figure 11.2** shows a comparison of the laboratory repeat assays which also compare well.

Figure 11.1 Laboratory Duplicate Comparison

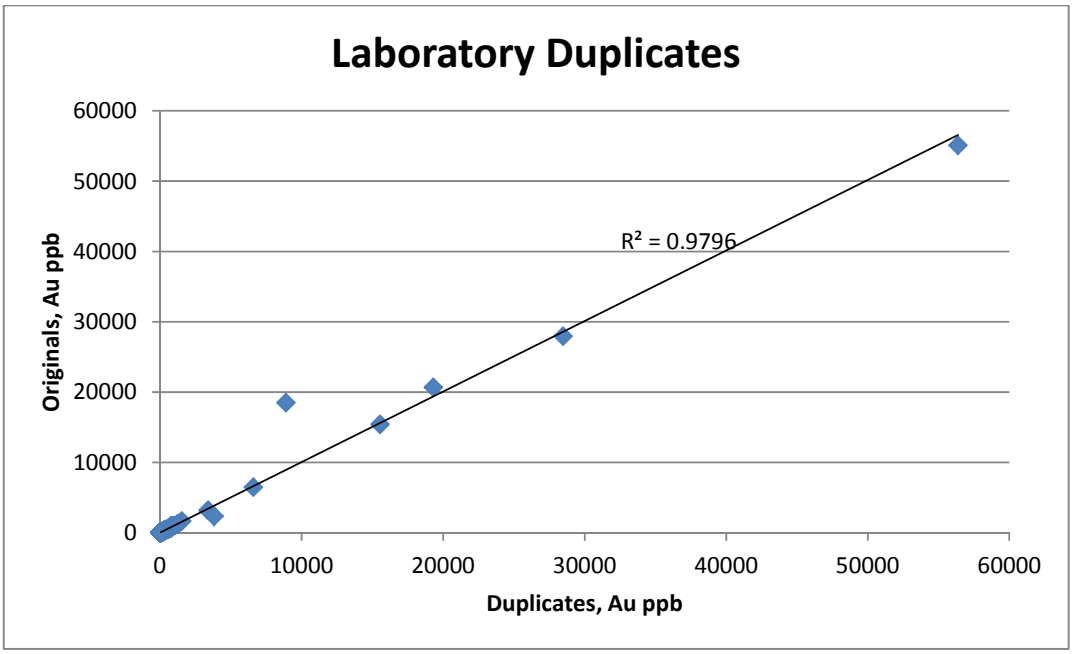


Figure 11.2 Laboratory Repeat Comparison

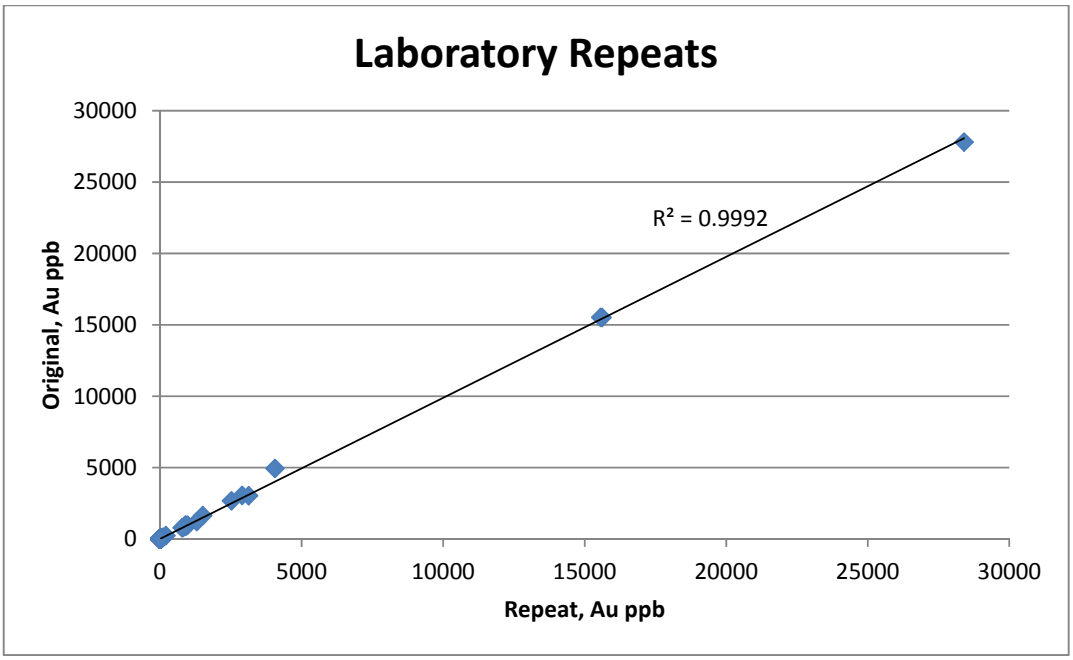
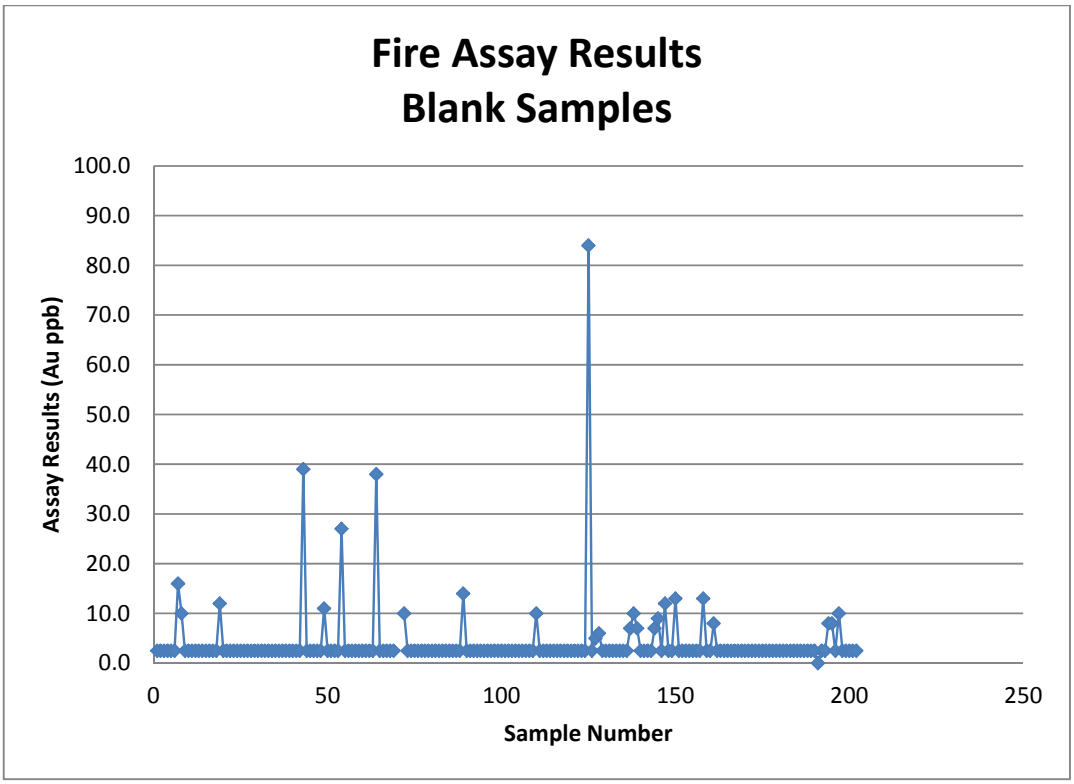


Figure 11.3 below shows the assay results of the blanks used in the QA/QC program. A total of 26 blanks returned detectable gold values. In all cases, the assay result was below 100 ppb Au. These discrepancies do not greatly affect the resource calculation since the average grade of the resource is 3.92 gpt Au or 47 times the highest gold value returned for a blank sample.

Figure 11.3 Fire Assay Results Blank Samples



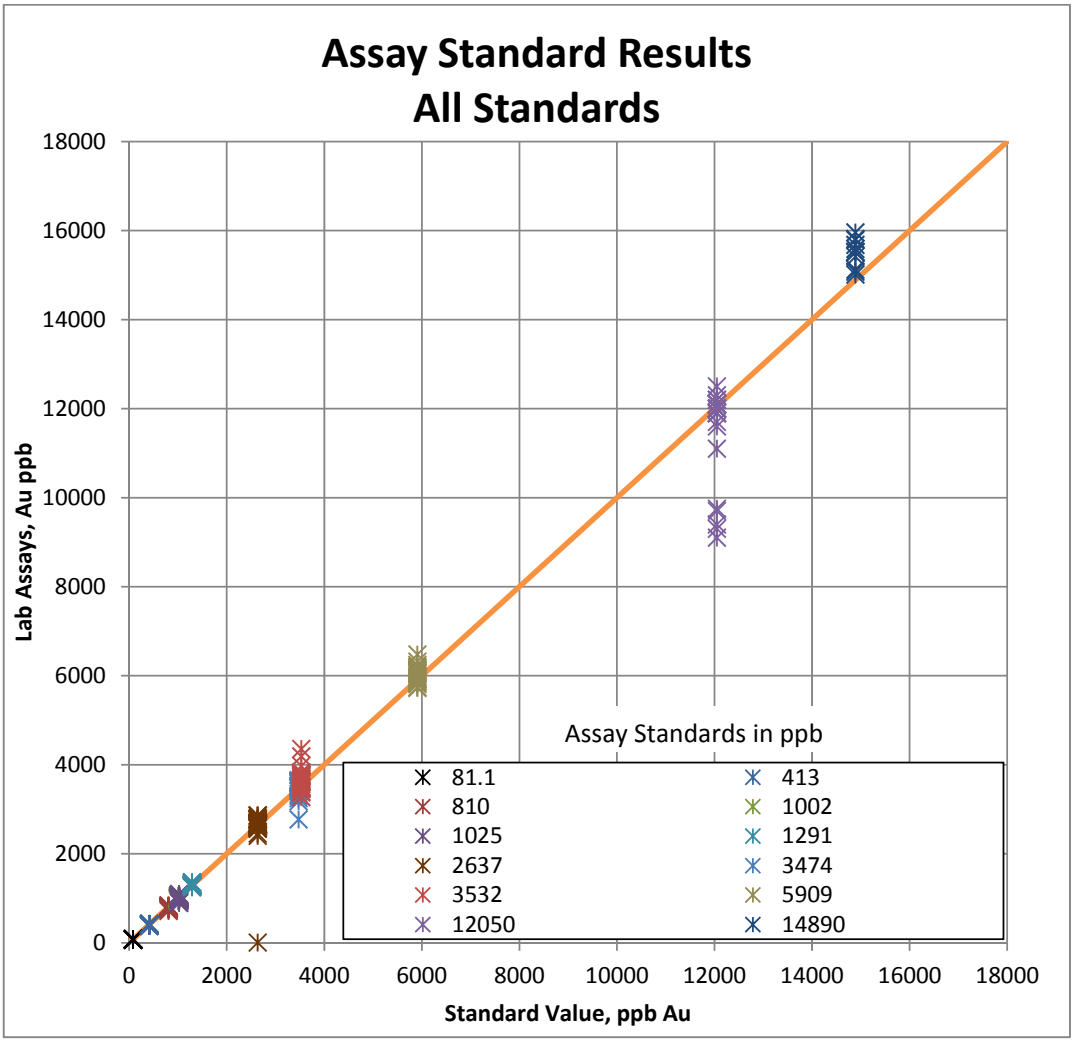
Note: value of 2.5 reflects value below detection limits.

In addition to blanks and duplicates, standards were used to check the accuracy of the assay results. A total of 12 different standards provided by Rocklabs Ltd. of New Zealand were used. The gold values of the assay standards cover the variation of the average gold grade for the resource.

- 0.081 ppm Au
- 0.413 ppm Au
- 0.810 ppm Au
- 1.002 ppm Au
- 1.025 ppm Au
- 1.291 ppm Au
- 2.637 ppm Au
- 3.474 ppm Au
- 3.532 ppm Au
- 5.909 ppm Au
- 12.050 ppm Au
- 14.89 ppm Au

Figure 11.4 shows a scatter plot of the certified value for each assay standard compared to the multiple values obtained from the laboratory assay of the standards which was used to evaluate the accuracy of the lab results. A 45 degree line represents the point at which the lab would return the certified value. This equal value line passes through all but one of the sample sets with the majority of the points directly adjacent to the line indicating acceptable accuracy performance for the majority of the standards. One assay of the 2.637 ppm Au returned gold grade below the detection limit of 0.005 ppm Au. Assay standards above 3.000 ppm Au have a slightly higher scatter around the equal value line with some standards having tail values on either side.

Figure 11.4 Assay Standards Results, All Standards



The 12.050 and 14.890 ppm Au standards did not perform as expected. The 12.050 standard shows that 5 of the 31 samples reported in the 9.500 ppm Au range which may indicate that the lab was under reporting the gold grade. These samples pertain to one particular assay certificate from the fourth drilling campaign. The samples for the assay run were not

reanalyzed by the lab. The difference for the apparent underreporting of the gold grade for that particular assay run is unknown.

The 14.890 ppm Au standard shows that all of the 12 samples report above the certified standard value which may indicate that the lab was over reporting the gold grade. The certificate of analysis for the standard was reviewed to determine the possible cause of the difference. The certificate shows that gold grade for the 14.890 ppm Au standard was determined by laboratory consensus which represented the average of 8 sub-samples sets analyzed by 11 different laboratories. The average value of all samples from all labs was 14.890 ppm Au which was determined to be the assay standard value. However, the average value of the 8 sub-samples from each lab varied from 13.913 to 15.708 ppm Au with coefficients of variation from 0.2% to 2.5%. In other words, each of the 11 labs had consistent values in the range of 13.913 to 15.708 ppm Au. The average value of lab assays by ACME labs was 15.501 ppm Au with a 2% coefficient of variation. This is comparable to the individual lab runs for the sub-sample sets for the standard and indicates acceptable accuracy performance of the standard despite the fact that all samples return assays higher than the certified value.

A statistical analysis of the standard assay results shows that acceptable precision or repeatability was also attained for each standard resulting in a coefficient of variation of 5% or below, except for four of the standards:

- 0.0811 ppm Au
- 2.637 ppm Au
- 3.474 ppm Au
- 3.532 ppm Au
- 12.050 ppm Au

The assays for the 0.0811 ppm Au standard contains a range of assay values from 0.064 ppm Au to 0.090 ppm Au resulting in a higher than desired variation in from the mean value with a coefficient of variation of 8%. The variation does not greatly affect the resource calculation since the average grade of the resource is 3.92 gpt Au or 48 times the value standard.

The assays for the 2.637 ppm Au standard contain one assay that was below the detection limit. Removing this sample from the data set results in a 3% coefficient of variation indicating acceptable precision for the standard.

The assays for the 3.474 ppm Au standard contain one assay at 2.771 ppm Au. Removing this sample from the data set results in a 4% coefficient of variation indicating acceptable precision for the standard.

The assays for the 3.532 ppm Au standard contain one assay at 4.354 ppm Au. Removing this sample from the data set results in a 5% coefficient of variation indicating acceptable precision for the standard.

The assays for the 12.050 ppm Au standard contain 5 assays with values ranging from 9.100 ppm Au to 9.7525 ppm Au. As previously mentioned, these samples represent one particular assay run from the fourth drilling campaign. Removing these samples from the data set results in a 2% coefficient of variation indicating acceptable precision for the standard.

Overall, the QA/QC program for sample assays indicates acceptable performance of all standards and blanks with only a few minor discrepancies. The propensity of these discrepancies might indicate that the laboratory was under reporting the gold grades for the samples; however, no adjustment to the gold assays was made during the resource calculation. **Table 11.2** below summarizes the statistical analysis of the assay standards.

Table 11.2 Statistical Analysis, Assay Standards

0.0811 ppm Au Standard	
Mean	74.4
Median	73.0
Stand Dev.	5.7
Mean - 2 SD	62.9
Mean + 2 SD	85.9
Min	64
Max	90
# of Records	25
Coefficient of Variation	8%

0.413 ppm Au Standard	
Mean	403.1
Median	405.0
Stand Dev.	15.4
Mean - 2 SD	372.2
Mean + 2 SD	434.0
Min	377
Max	430
# of Records	29
Coefficient of Variation	4%

0.810 ppm Au Standard	
Mean	789.8
Median	792.5
Stand Dev.	25.3
Mean - 2 SD	739.2
Mean + 2 SD	840.4
Min	714
Max	842
# of Records	36
Coefficient of Variation	3%

1.002 ppm Au Standard	
Mean	1,001.1
Median	1,004.5
Stand Dev.	28.9
Mean - 2 SD	943.3
Mean + 2 SD	1,058.9
Min	938
Max	1055
# of Records	20
Coefficient of Variation	3%

1.025 ppm Au Standard	
Mean	1,015.2
Median	1,018.0
Stand Dev.	36.1
Mean - 2 SD	942.9
Mean + 2 SD	1,087.5
Min	895
Max	1097
# of Records	76
Coefficient of Variation	4%

1.291 ppm Au Standard	
Mean	1,296.7
Median	1,295.0
Stand Dev.	30.5
Mean - 2 SD	1,235.8
Mean + 2 SD	1,357.6
Min	1245
Max	1359
# of Records	24
Coefficient of Variation	2%

2.637 ppm Au Standard	
Mean	2,619.0
Median	2,655.0
Stand Dev.	376.5
Mean - 2 SD	1,866.0
Mean + 2 SD	3,372.0
Min	2.5
Max	2866
# of Records	52
Coefficient of Variation	14%

3.474 ppm Au Standard	
Mean	3,452.8
Median	3,502.5
Stand Dev.	194.3
Mean - 2 SD	3,064.3
Mean + 2 SD	3,841.3
Min	2771
Max	3648
# of Records	22
Coefficient of Variation	6%

3.532 ppm Au Standard	
Mean	3,663.3
Median	3,629.0
Stand Dev.	217.7
Mean - 2 SD	3,227.8
Mean + 2 SD	4,098.8
Min	3274
Max	4354
# of Records	31
Coefficient of Variation	6%

5.909 ppm Au Standard	
Mean	6,022.8
Median	6,016.5
Stand Dev.	170.2
Mean - 2 SD	5,682.3
Mean + 2 SD	6,363.3
Min	5724
Max	6478
# of Records	28
Coefficient of Variation	3%

12.05 ppm Au Standard	
Mean	11,582.3
Median	12,000.0
Stand Dev.	967.1
Mean - 2 SD	9,648.1
Mean + 2 SD	13,516.6
Min	9100
Max	12500
# of Records	31
Coefficient of Variation	8%

14.89 ppm Au Standard	
Mean	15,501.4
Median	15,523.0
Stand Dev.	288.4
Mean - 2 SD	14,924.5
Mean + 2 SD	16,078.3
Min	15014
Max	15960
# of Records	12
Coefficient of Variation	2%

12.0 DATA VERIFICATION

12.1 Drill Collar Review

Drill collar location was initially determined using a Garmin 65SX GPS, which proved to have an acceptable accuracy in horizontal coordinates, but not in elevation. The procedure for determining the coordinates of the collar was taking three (3) GPS readings on different days, and using an average of the three readings.

After finishing the Second Phase Drilling, Magellan used the services of Guara Agroservicios, a local Survey Consulting Company for the exact location of the collars. The collars of the third and fourth phase drilling programs were located by compass and metric tape measuring the distances from a surveyed hole. Magellan is presently locating the holes with a differential GPS.

12.2 Down hole Survey Review

All the diamond-drill holes from the second phase (BR_COR_DDH23 to BR_COR_DDH66) have been surveyed with a Sperry Sun single-shot camera supplied by the drilling contractor, at 30 m intervals. Holes from the third phase (BR_COR_DDH67 to BR_COR_DDH81) were surveyed with the REFLEX method. Holes from the fourth phase of the drilling (BR_COR_DDH83 to BR_COR_DDH_103) were surveyed with the REFLEX EZ SHOT. Holes BR_COR_DDH104 to BR_COR_DDH110, BR_COR_DDH122, BR_COR_DDH125, BR_COR_DDH130, BR_COR_DDH131 and BR_COR_DDH143 were not surveyed. Holes from BR-COR_DDH110 to BR_COR_DDH160 were surveyed with REFLEX MAXIBOR II method or DEVICO EZ SHOT method. Most holes (142) were drilled on an approximate 190° to 255° azimuth. Eighteen holes were drilled on an approximate 45° to 65° azimuth. Inclinations range between -47° to -70°. Of the 160 drill holes, 36 holes (22%) do not have down-hole survey data, however only nine of the thirty six are greater than 100 m in length.

One of the host rock units (Rhyolite) proved to be magnetic and therefore measurements could be subject to interference.

Magellan reviewed the collar readings after the drilling program was completed, to check for abrupt azimuth or dip changes that may suggest the presence of false magnetic interferences or inappropriately collected readings (See **Photos 12.1** and **12.2**).

This review found several which should be addressed. **Tables 12.1** and **12.2 (Figures 12.1** and **12.2)** summarize the collar survey and the first Sperry Sun measurements, and **Tables 12.3** and **12.4 (Figure 12.3** and **12.4)** the REFLEX and DEVICO measurements.

Photo 12.1 REFLEX Equipment used at Project

Photo 12.2 Reviewing Azimuth and dip readings

Table 12.1 Collar Azimuth Survey and Sperry Sun Azimuth Reading

BR_COR_DDH#	Host Rock	Collar Az.	First Sperry Sun	Difference
23	Rhyolite	252		
24	Rhyolite	245	253	8
25	Rhyolite		250	
26	Rhyolite	246	254	8
27	Rhyolite	246	254	8
28	Rhyolite	245	253	8
29	Rhyolite	248	254	6
30	Rhyolite	245	253	8
31	Rhyolite	250	251	1
32	Rhyolite	246		
33	Rhyolite	252	242	10
34	Rhyolite	244	252	8
35	Rhyolite	254	250	4
36	Rhyolite	246	245	1
37	Rhyolite	245	252	7
38	Granite	248		
39	Rhyolite	245	251	6
40	Rhyolite	245	257	12
41	Rhyolite	240	250	10
42	Granite	246	234	12
43	Granite	242	253	11
44	Rhyolite	243	252	9
45	Rhyolite	250	238	12
46	Granite	248	249	1
48	Granite	242	255	13
49	Granite	242	255	13
51	Rhyolite	252	252	0
53	Rhyolite	244	254	10
54	Rhyolite	248	252	4
55	Granite	245	249	4
56	Rhyolite	250	248	2
57	Granite	245	253	8
58	Rhyolite	245	247	2
59	Rhyolite	248	249	1
60	Rhyolite	245	239	6
61	Rhyolite	240	254	14
64	Granite	240	251	5

Table 12.2 Collar Dip Survey and Sperry Sun Dip Reading

BR_COR_DDH#	Host Rock	Collar Dip	First Sperry Sun	Difference
23	Rhyolite	50		
24	Rhyolite	52	50	2
25	Rhyolite	52	47	5
26	Rhyolite	56	49	7
27	Rhyolite	52	50	2
28	Rhyolite	50	50	0
29	Rhyolite	56	49	7
30	Rhyolite	50	49	1
31	Rhyolite	54	50	4
32	Rhyolite	48		
33	Rhyolite	48	48	0
34	Rhyolite	50	48	2
35	Rhyolite	48	50	2
36	Rhyolite	52	45	7
37	Rhyolite	50	50	0
38	Granite	60		
39	Rhyolite	47	48	1
40	Rhyolite	48	50	2
41	Rhyolite	49	51	2
42	Granite	60	60	0
43	Granite	52	49	3
44	Rhyolite	54	51	3
45	Rhyolite	50	49	1
46	Granite	50	50	0
48	Granite	50	50	0
49	Granite	50	50	0
51	Rhyolite	56	51	5
53	Rhyolite	50	50	0
54	Rhyolite	54	50	4
55	Granite	52	50	2
56	Rhyolite	48	51	3
57	Granite	52	47	5
58	Rhyolite	58	50	8
59	Rhyolite	53	50	3
60	Rhyolite	60	51	9
61	Rhyolite	52	49	3
64	Granite	52	50	2

Down-hole deviations in azimuth ranges from 0° to 14°, and deviations in dip ranges from 0° to 9°.

Figure 12.1 Azimuth Survey, Sperry Sun

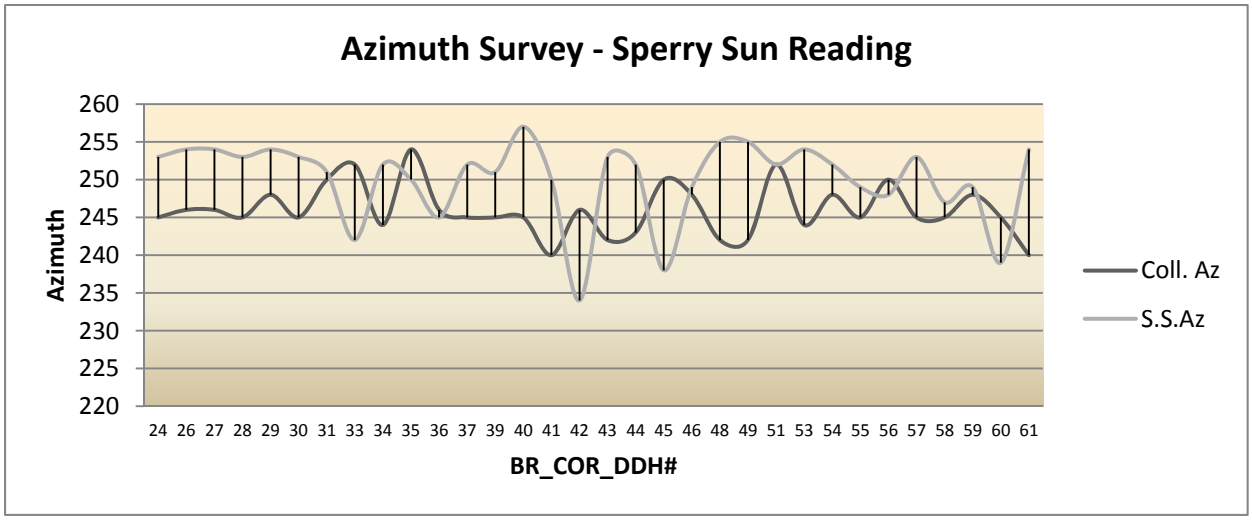


Figure 12.2 Dip Survey, Sperry Sun

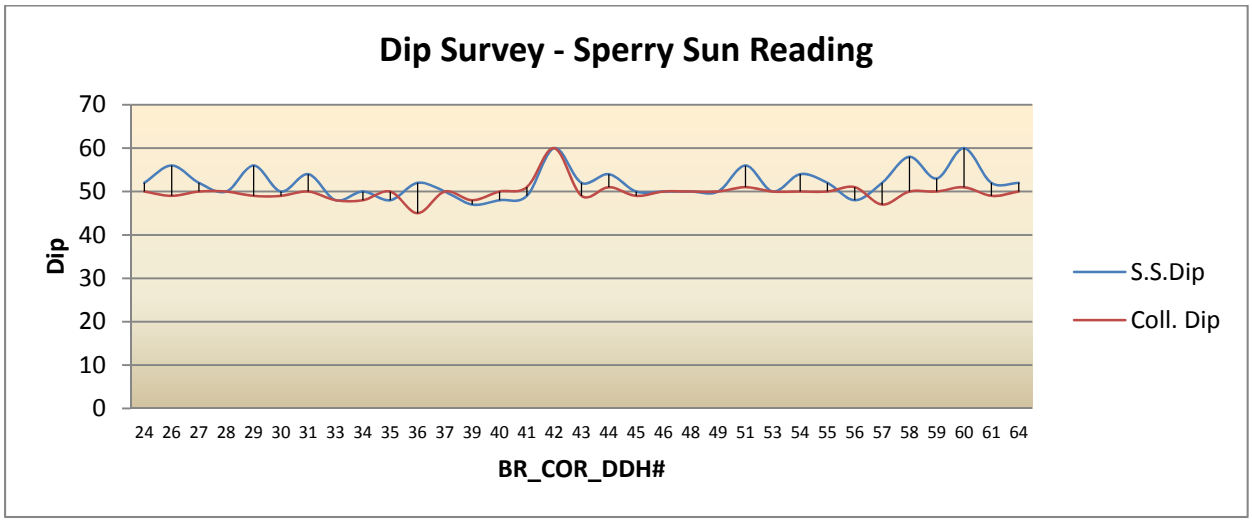


Table 12.3 Collar Azimuth Survey and REFLEX Azimuth Reading

BR_COR_DDH#	Host rock	Collar Az Survey	Reflex/Devico Survey	Difference
67	Rhyolite	248	245	-3
68	Rhyolite	238	240.5	2.5
69	Rhyolite	240	246.7	6.7
70	Rhyolite	241	243	2
71	Rhyolite	251	242.6	-8.4
72	Rhyolite	245	247.1	2.1
73	Rhyolite	245	247	2
74	Rhyolite	245	240.9	-4.1
75	Rhyolite	245	248.5	3.5
76	Rhyolite	245	242.7	-2.3
77	Rhyolite	245	245.3	0.3
78	Rhyolite	245	247.3	2.3
79	Rhyolite	245	242.7	-2.3
80	Rhyolite	245	245.3	0.3
81	Rhyolite	245	247.3	2.3
83	Rhyolite	58	58.9	0.9
84	Rhyolite	235	233.2	-1.8
85	Rhyolite	235	229.9	-5.1
86	Rhyolite	235	239.5	4.5
87	Rhyolite	220	221.3	1.3
88	Rhyolite	210	209.3	-0.7
89	Rhyolite	230	231.1	1.1
90	Rhyolite	240	236	-4
91	Rhyolite	245	247.3	2.3
92	Rhyolite	245	244	-1
93	Rhyolite	230	232.7	2.7
94	Rhyolite	30	30.3	0.3
95	Rhyolite	210	209.9	-0.1
96	Rhyolite	250	247.9	-2.1
97	Rhyolite	250	247.4	-2.6
98	Rhyolite	250	249.3	-0.7
99	Rhyolite	250	247.2	-2.8
100	Rhyolite	230	231.5	1.5
101	Rhyolite	250	249.9	-0.1
102	Rhyolite	250	252.5	2.5
103	Rhyolite	245	242.4	-2.6
111	Rhyolite	235	235.15	0.15
112	Rhyolite	225	225	0
113	Rhyolite	235	235	0

BR_COR_DDH#	Host rock	Collar Az Survey	Reflex/Devico Survey	Difference
114	Rhyolite	45	47.6	2.6
115	Rhyolite	235	234.07	-0.93
116	Rhyolite	235	236.68	1.68
117	Rhyolite	45	46.9	1.9
118	Rhyolite	50	51.35	1.35
119	Rhyolite	235	232.06	-2.94
120	Rhyolite	65	66.64	1.64
121	Rhyolite	245	247.7	2.7
123	Rhyolite	45	47.97	2.97
124	Granite	245	244.06	-0.94
126	Rhyolite	45	44.31	-0.69
127	Granite	245	246.54	1.54
128	Rhyolite	225	226.14	1.14
129	Rhyolite	225	227.74	2.74
132	Rhyolite	245	245.34	0.34
133	Rhyolite	45	48.82	3.82
134	Rhyolite	45	45	0
135	Rhyolite	245	245.03	0.03
136	Rhyolite	215	215.92	0.92
137	Rhyolite	58	58	0
138	Granite	245	244.43	-0.57
139	Granite	245	246.79	1.79
140	Rhyolite	245	242.81	-2.19
141	Granite	245	245.28	0.28
142	Rhyolite	210	210	0
144	Rhyolite	240	240.92	0.92
145	Granite	245	238.93	-6.07
146	Rhyolite	210	210	0
147	Rhyolite	210	210	0
148	Rhyolite	245	244.9	-0.1
149	Rhyolite	240	239.56	-0.44
150	Rhyolite	245	243.99	-1.01
151	Rhyolite	240	240	0
152	Rhyolite	240	239.81	-0.19
153	Rhyolite	245	242.13	-2.87
154	Rhyolite	240	240	0
155	Rhyolite	248	248.08	0.08
156	Rhyolite	240	241.44	1.44
157	Rhyolite	40	40	0
158	Rhyolite	40	40	0

BR_COR_DDH#	Host rock	Collar Az Survey	Reflex/Devico Survey	Difference
159	Rhyolite	40	40	0
160	Rhyolite	245	245	0

Table 12.4 Collar Dip Survey and REFLEX Dip Reading

BR_COR_DDH#	Host rock	Collar Dip Survey	Reflex/Devico Survey	Difference
67	Rhyolite	50	50	0
68	Rhyolite	59.2	60	0.8
69	Rhyolite	48.3	50	1.7
70	Rhyolite	47.4	50	2.6
71	Rhyolite	50.5	50	-0.5
72	Rhyolite	48.3	50	1.7
73	Rhyolite	50.3	50	-0.3
74	Rhyolite	50.7	50	-0.7
75	Rhyolite	46.8	50	3.2
76	Rhyolite	58.8	60	1.2
77	Rhyolite	58.4	60	1.6
78	Rhyolite	60.1	60	-0.1
79	Rhyolite	60	58.8	-1.2
80	Rhyolite	60	58.4	-1.6
81	Rhyolite	60	60.1	0.1
83	Rhyolite	50	48.6	-1.4
84	Rhyolite	60	51.5	-8.5
85	Rhyolite	50	61.6	11.6
86	Rhyolite	50	51.2	1.2
87	Rhyolite	50	52.9	2.9
88	Rhyolite	50	48.5	-1.5
89	Rhyolite	60	61	1
90	Rhyolite	50	50	0

BR_COR_DDH#	Host rock	Collar Dip Survey	Reflex/Devico Survey	Difference
91	Rhyolite	50	49.7	-0.3
92	Rhyolite	50	49.3	-0.7
93	Rhyolite	50	49.6	-0.4
94	Rhyolite	50	48.6	-1.4
95	Rhyolite	50	50.9	0.9
96	Rhyolite	60	59.9	-0.1
97	Rhyolite	50	46.6	-3.4
98	Rhyolite	60	58.3	-1.7
99	Rhyolite	50	50.2	0.2
100	Rhyolite	50	50	0
101	Rhyolite	50	50	0
102	Rhyolite	50	51	1
103	Rhyolite	50	49.8	-0.2
111	Rhyolite	50	50.7	0.7
112	Rhyolite	50	50.7	0.7
113	Rhyolite	50	49.6	-0.4
114	Rhyolite	50	51.68	1.68
115	Rhyolite	50	49.37	-0.63
116	Rhyolite	50	51.39	1.39
117	Rhyolite	50	49.81	-0.19
118	Rhyolite	50	48.69	-1.31
119	Rhyolite	50	53.68	3.68
120	Rhyolite	50	53.49	3.49
121	Rhyolite	60	61.26	1.26
123	Rhyolite	50	51.85	1.85
124	Granite	60	59.35	-0.65
126	Rhyolite	50	51.57	1.57

BR_COR_DDH#	Host rock	Collar Dip Survey	Reflex/Devico Survey	Difference
127	Granite	60	59.94	-0.06
128	Rhyolite	50	49.24	-0.76
129	Rhyolite	50	50.44	0.44
132	Rhyolite	65	60.01	-4.99
133	Rhyolite	50	47.95	-2.05
134	Rhyolite	65	65.4	0.4
135	Rhyolite	65	64.55	-0.45
136	Rhyolite	50	49.03	-0.97
137	Rhyolite	65	65.03	0.03
138	Granite	65	63.39	-1.61
139	Granite	65	64.89	-0.11
140	Rhyolite	65	65.64	0.64
141	Granite	65	65.73	0.73
142	Rhyolite	50	50.3	0.3
144	Rhyolite	70	69.58	-0.42
145	Granite	75	74.97	-0.03
146	Rhyolite	50	52.1	2.1
147	Rhyolite	50	50.7	0.7
148	Rhyolite	70	69.21	-0.79
149	Rhyolite	65	64.23	-0.77
150	Rhyolite	70	69.23	-0.77
151	Rhyolite	50	49.31	-0.69
152	Rhyolite	60	60.73	0.73
153	Rhyolite	55	58.22	3.22
154	Rhyolite	50	52.97	2.97
155	Rhyolite	70	71.49	1.49
156	Rhyolite	70	70.89	0.89

BR_COR_DDH#	Host rock	Collar Dip Survey	Reflex/Devico Survey	Difference
157	Rhyolite	50	49.66	-0.34
158	Rhyolite	50	52	2
159	Rhyolite	50	52.03	2.03
160	Rhyolite	50	48.9	-1.1

Figure 12.3 Azimuth Survey, REFLEX

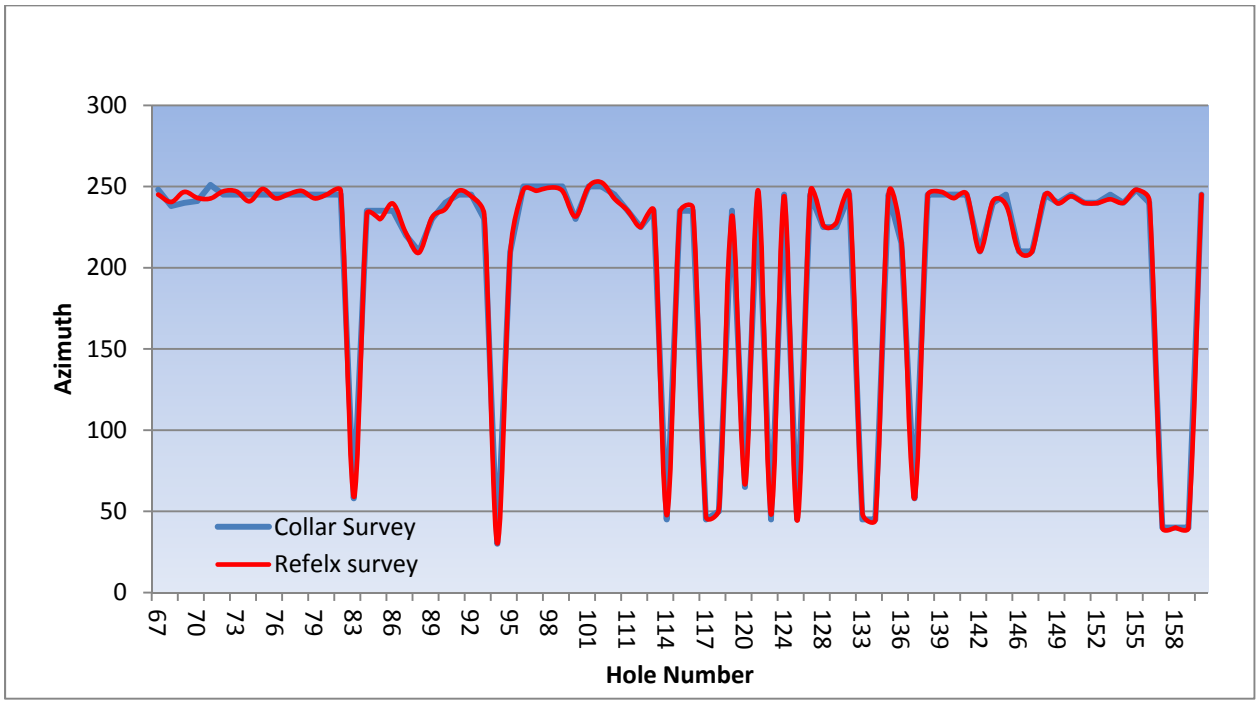
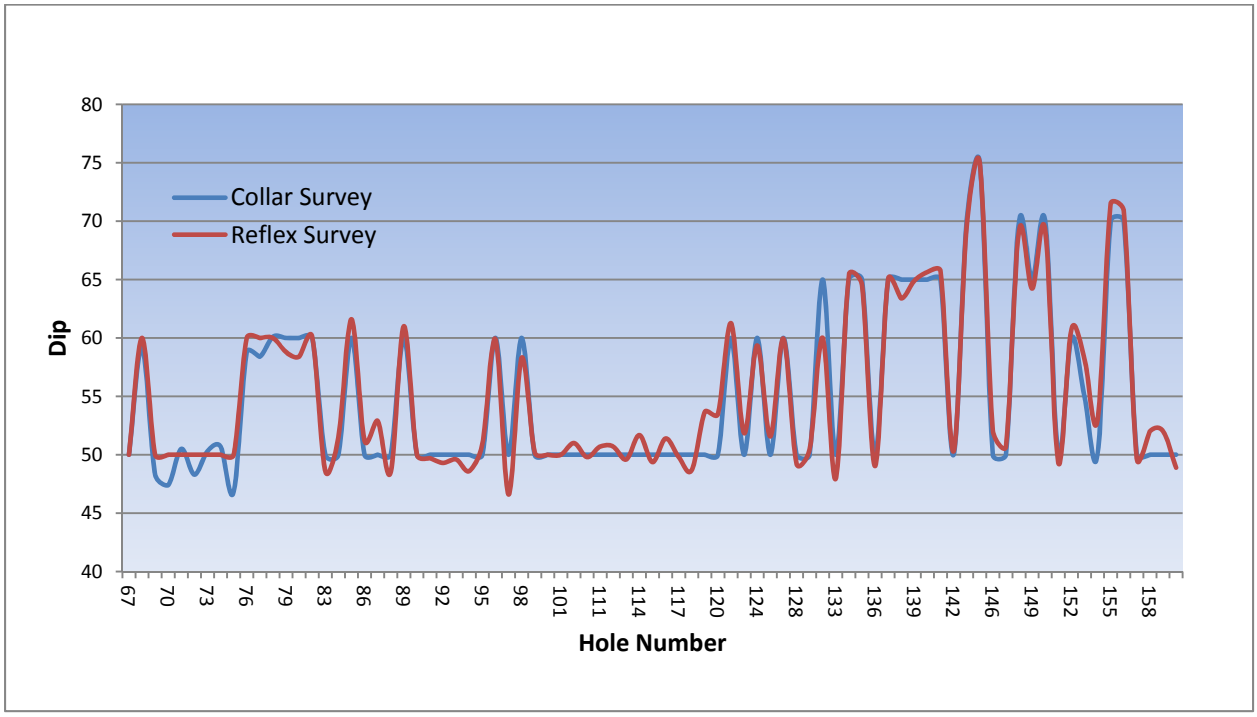


Figure 12.4 Dip Survey, REFLEX



From **Tables 12.1** and **12.2** and the graphics we can see that the Sperry Sun results for hole numbers 33, 40, 41, 42, 43, 45, 48, 49, 53, and 61 show significant azimuth differences between the collar readings and the first down-hole readings, ranging from 10° to 14°.

The holes that show significant dip differences are: 26, 29, 36, 58, and 60; which range from 7° to 9°.

From **Tables 12.3** and **12.4** and the graphics we can see the REFLEX method shows less variation, reaching a maximum of 8°.

The dip survey does not show any significant difference.

The magnetic nature of some of the host rocks may have some influence on the azimuth readings of the Sperry Sun method; however the dip readings appear to be more precise and reliable.

12.3 Density Review

Magellan conducted a density-sampling program, taking 207 samples from 50 holes (of 160 available holes). Small core pieces, 5 cm to 10 cm long, were taken from quartz veins with sulphides, quartz breccia, hydrothermal breccia, strong sericitic alteration, and host rock (rhyolite and granite).

Ten of these samples were sent to SGS facilities in Belo Horizonte and compared with the measurement done at the project by Magellan geologists. The average difference from SGS Lab measurements and Magellan measurements was 0.049 t/m^3.

The standard water displacement method was implemented by measuring the sample mass in the air (M_a) and submerged in water (M_w).

The density statistics for the main lithologies are presented in **Table 12.5**.

Table 12.5 Density Statistics from core samples

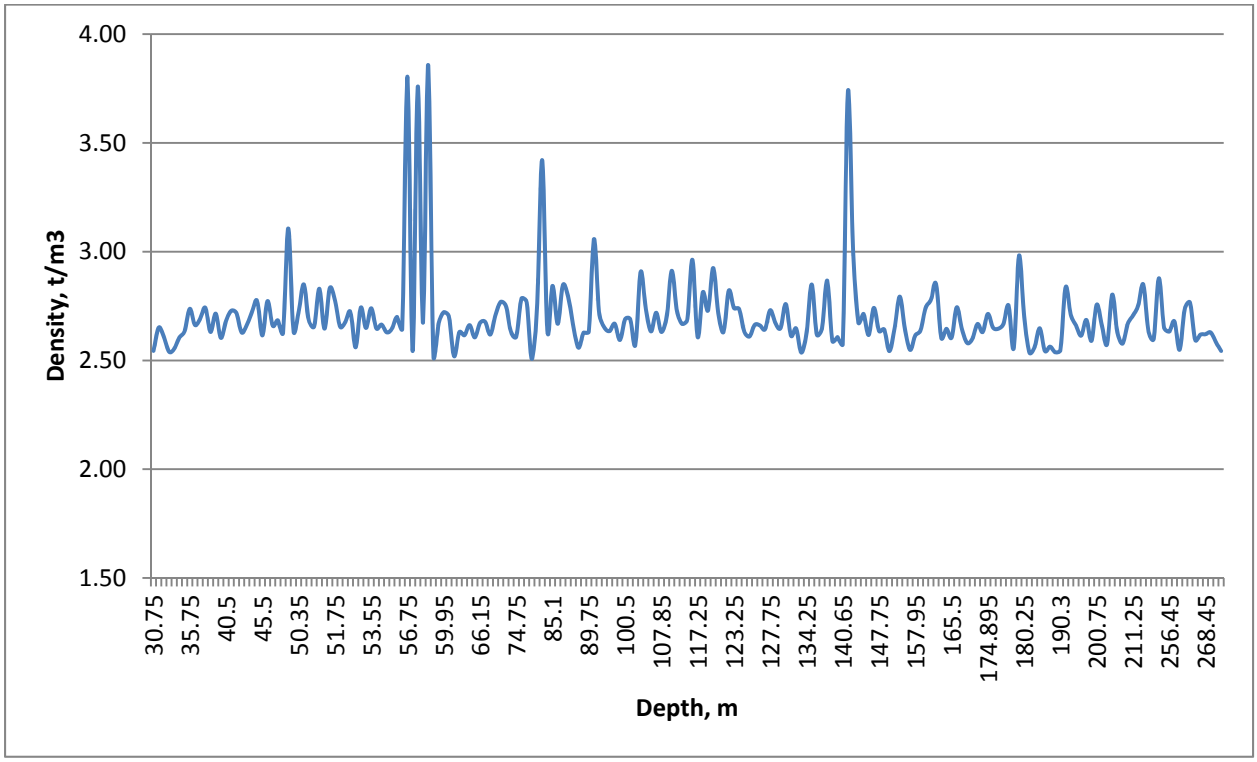
	Quartz Veins with Sulphides	Quartz Breccia	Sericitic Alteration	Hydrothermal Breccia	Granite	Rhyolite
Density(t/m^3)	2.73	2.74	2.77	2.88	2.60	2.64
SD (t/m^3)	0.20	0.14	0.20	0.39	0.04	0.06
CV	7.4%	5.0%	7.3%	13.6%	1.5%	2.3%
Count	55	10	35	15	31	61
Min (t/m^3)	2.54	2.60	2.51	2.52	2.53	2.52
Max (t/m^3)	3.74	3.01	3.80	3.86	2.67	2.91

Note: Updated to include all drilling through 2011

The shear zones and the host rocks exhibit low coefficients of variation, calculated as the percent ratio between the standard deviation and the average density (Max 13.6%). The hydrothermal breccia have a bigger coefficient of variation (13.6%), due to the high contents of sulphide (Filon do Meio).

The depth of the density samples ranged from 30 m to 132 m. There is not significant variation of density with depth **(Figure 12.5)**.

Figure 12.5 Density vs. Depth



12.4 Site Visits by GRE

A 6-day and a 4.5-day site visit were completed by Mr. Kevin Gunesch, PE of GRE during May 2009 and January 2012, respectively. A summary of the items verified by Mr. Gunesch during the site visits are presented below. In both site visits, the data reviewed onsite was found to be representative of the original records and matches well with the core samples. The onsite core provides an excellent physical record of the exploration drilling and the visible gold noted in the core further authenticates the presence of gold within the deposit.

12.4.1 Infrastructure

January 2012

During the trip to the site, Mr. Gunesch noted many major improvements to the public highway BR-163 since 2009. Large reinforced concrete bridges were being constructed at each major water crossing with smaller concrete culverts located at minor drainages. A few of the bridges were completed while most were under construction. Several sections of the highway south of Novo Progresso were paved with the highway being completely paved and finished with appropriate signage and lane markings about one hour south of Novo Progresso just before the turn to the project.

12.4.2 Drill Hole Location, Azimuth, and Dip

May 2009

Mr. Gunesch completed field surveys of 24 of the 81 diamond-drill holes located at the project site, or 30%, using a handheld GPS, model Garmin 60CSx and a Brunton Geo transit. The easting, northing, azimuth, and dip were recorded for each field survey and then compared to the drill-hole database. Two errors were found in the azimuths of holes 17 and 18, which were later corrected in the database.

January 2012

Since Mr. Gunesch's previous visit, a total of 79 additional drill holes were completed (holes 82 through 160). Mr. Gunesch completed field surveys of 49 of the 79 additional holes, representing 62% of the new diamond-drill holes located at the project site. The holes were located using a handheld GPS, model Garmin 62s and a Brunton Geo transit. The easting, northing, azimuth, and dip were recorded for each field survey and then compared to the drill-hole database. Discrepancies were noted in the field azimuth measurements of DH72, DH132, DH134 and DH 155. These apparent errors were later discounted after comparing the database azimuth to the down-hole survey azimuth. Variations in the field measurements for horizontal locations were attributed to accuracy of hand-held GPS units operating under thick jungle canopy.

12.4.3 Density

May 2009

A density test utilizing the water displacement method was completed on a piece of core during the site visit. The result of this test, 2.75 t/m^3, compares well with the results obtained by Magellan and SGS.

12.4.4 Original Assay Certificates

May 2009

A portion of the original assay certificates by SGS were available onsite with the remainder being provided by Magellan after the site visit for the first through third drilling programs. Assay certificates for all drill holes have been accounted for by GRE. Mr. Gunesch completed a comparison of the assay results for 20 of the drill holes which contained 457 assays or 19% of the total 4,492 assays to the values contained in Magellan's drill-hole database. Only one error was found where the results for samples DS23574 and DS 23575 had been transposed. The 5 highest Au assay results of the database were contained within assays checked by Mr. Gunesch.

January 2012

Mr. Gunesch completed a review of all assay certificates for the fourth drilling program which contained 4,930 assays. The electronic versions of the certificates sent by the lab were compared with Magellan's database. Two assays were noted that did not match the certificate and were corrected in the database. Four assays were not correctly labeled as blanks or standards and were corrected in the database. 123 assays were incorrectly entered into the repeat column of the database and were moved to the correct column.

12.4.5 Sample Preparation

May 2009

Mr. Gunesch reviewed the entire sample preparation method which consisted of receiving and logging the core, selecting the appropriate cut line to bisect the principal structure, splitting the core with the core saw, selecting the sample interval, assigning the sample number, bagging the sample, and completing the control documentation for sample transport to the laboratory for analysis. The sampling method employed by Magellan meets industry standards.

12.4.6 Assay QA/QC Program

May 2009

Mr. Gunesch reviewed the original sample control sheets that were used to insert two duplicates, two standards, and one blank every 42 samples and compared these to the samples analyzed. No discrepancies were found; however, the sample control sheet for drill hole 52 could not be located. Magellan sent sample 'blanks' to the lab in the form of pulverized cement powder. Previously pulverized blank samples cannot confirm that the sample preparation method was free of contamination. Mr. Gunesch recommended using inert rock material from a local quarry borrow for blank samples in future QA/QA programs.

January 2012

Mr. Gunesch again reviewed the original sample control process along with the quality control sheets used to determine the insertion of duplicates, standards, and blanks. Two standards and one blank are inserted every 42 samples. Duplicates appeared to be chosen at random with zero to three duplicates every 42 samples. It was noted that one sample, DS30091, had been previously identified as incorrectly labeled and had not updated in the database to reflect that sample contained 0.0811 ppm Au standard and not 4.113 ppm Au standard.

Mr. Gunesch reviewed the standards and material for blanks used for the quality control program. Standards consisted of two forms, bulk powder and individual sample packets, both prepared by Rock Labs Ltd. in New Zealand. Approximately 100 grams or two packets of material were sent to the labs as standard samples. Material for blanks was sourced from a local granite quarry and sent in aggregate form to the lab as blank samples. This material had previously been analyzed and returned results below the detection limit for gold. Sending blank material in aggregate form and not powder provides a means to check cross contamination between samples due to the laboratory sample preparation process.

12.4.7 Chain of Custody Documentation

May 2009

The chain of custody documentation found during the site visit was incomplete; however the remaining documentation was located after completion of the site visit. Two types of documents exist: documents for transporting the samples from the project site to Novo Progresso, and documents for transporting

the samples from Novo Progresso to SGS Laboratories in Belo Horizonte. Mr. Gunesch was informed that confirmation of receipt of the samples by SGS was via telephone.

January 2012

The chain of custody documentation for the 2010 and 2011 drilling programs was found to be in proper order. Magellan had recently performed an internal audit of its program and identified 48 missing control documents (DS31090 through DS31137). The missing documents represent less than 1% of the 5,277 drill core samples. The missing receipts for transportation to Itiatuba were located and on file at the Novo Progresso office.

12.4.8 Assay Verification

May 2009

Mr. Gunesch selected two samples to be sent to Inspectorate Labs in Lima Peru for independent assay verification. These samples reflect a vein sample from the Serra block and a shear-zone altered sample from the Filao do Meio block. The assay results of these selected samples compare well with the previous results. **Tables 12.6 and 12.7** below detail the results.

Table 12.6 Samples Selected for Analysis at Inspectorate Labs Lima Peru

Previous Sample Number	New Sample Number	Drill Hole	From	To	Box Number	Block
23579	30129	DDH38	75.7	76.2	27	Serra
29800	30130	DDH68	154.5	155	57	Filao do Meio

Table 12.7 Check Assays Results Compared to Original Assays

Bore Hole ID	From	To	Magellan Sample No	Au PPB	GRE Sample No	Au PPB	Avg PPB	Error PPB	% Error
BR_COR_DDH68	154.5	155	29800	9736	30130	10,114	9,925	189	1.9%
BR_COR_DDH38	75.7	76.2	23579	7874	30129	6,682	7,278	-596	-8.2%*

* higher percent error due to inherent error created from large gold particles contained within the vein zone

January 2012

Mr. Gunesch selected two samples for independent assay verification from two of the newly drilled areas; the Valdette block and the Demetrio block. The samples were sent to Florin Analytical services in Reno, NV and assayed using a metal screen assay with the plus fraction at +150 mesh. The assay results

of these selected samples are within acceptable ranges for correlation between the samples. **Tables 12.8 and 12.9** below detail the results.

Table 12.8 Samples Selected for Independent Analysis

Previous Sample Number	New Sample Number	Drill Hole	From	To	Box Number	Block
32844	35701	DH128	85.0	85.5	N/A	Valdette
31731	35702	DH115	85.0	85.5	N/A	Demetrio

Table 12.9 Check Assays Results Compared to Original Assays

Bore Hole ID	From	To	Magellan Sample No	Au PPB	GRE Sample No	Au PPB	Avg PPB	Error PPB	% Error
DH128	85.0	85.5	32844	1600	35701	1291	TBD	-154.5	-11%
DH115	85.0	85.5	31731	5436	35702	4327	TBD	-554.5	-11%

The results indicate an acceptable correlation between the previous assay results and the verification assays. A percent error for each assay was calculated using the average value as the correct assay value. The metal screen assays indicate a -11% error from the original assays. This is most likely due to the difference between the assay methods (previous assays were not metal screen assays) and that coarse gold was contained within the assay split of the original samples. Although both assays indicate the same negative percent error, it is just as probable for coarse gold to be contained within the assay split as to not be contained with the split.

12.4.9 Geologic Logs and Core

May 2009

During the site visit, Mr. Gunesch compared the original geologic logs to the core stored onsite. Mr. Gunesch reviewed 55 mineralized drill intercepts representing 66% of the 81 drill holes. No discrepancies were found. Mr. Gunesch confirmed a visible gold specimen in drill hole 79 and discovered a new visible gold specimen in drill hole 57.

January 2012

During the site visit, Mr. Gunesch compared the original geologic logs to the core stored onsite. Mr. Gunesch reviewed 51 mineralized drill holes representing 65% of the 79 new drill holes. Mr. Gunesch confirmed a visible gold specimen in DH121. A minor discrepancy was noted for DH128; the log was incorrectly labeled as DH126. The error was noted for correction. No other discrepancies were noted.

13.0 MINERAL PROCESSING AND METALLURGICAL TESTING

Preliminary scoping tests were completed at Resource Development Incorporated (RDi), Wheat Ridge, CO, on two composite samples to develop fundamental metallurgical data for the Coringa project. Prior metallurgical work on the project included some limited gravity and flotation tests (SGS Geosol, Belo Horizonte, Brazil, March 2008). These tests indicated that reasonable recoveries would be obtained on the Coringa material. As a follow up to this initial work, additional metallurgical tests were completed by Resource Development Incorporated (RDi) in March 2010, which included indirect bond work index, grinding studies, gravity tests, flotation tests and whole ore cyanide leach.

13.1 Composite samples

Available exploration samples were used to create two metallurgical composite samples. Nineteen samples were included within the 33 kilogram Serra composite. Fifty-two samples were combined to create the Meio 78 kilogram composite. Within the available samples, two high-grade samples, 64 and 48 grams Au per tonne, were excluded from the Serra composite. The samples were withheld to generate a composite grade similar to the reserve grade. Likewise, two high-grade samples were withheld from the Meio composite, 150 and 82 grams Au per tonne. The samples selected from the Meio block are representative of the entire block with a good spatial and grade distribution through the resource block. The samples selected from Serra are not as numerous as the Meio samples and therefore do not cover the entire extent though there were selected from all three identified veins of the resource block.

Table 13.1 Composite Sample Analysis

ELEMENT	SERRA COMPOSITE	MEIO COMPOSITE
Gold, gpt	8.2	11.1
Silver, gpt	14.9	16.0
Lead, %	0.260	1.605
Zinc, %	0.125	0.515
Organic carbon, %	0.045	0.110
Sulphur total, %	2.56	2.86

As shown in **Table 13.1**, organic carbon is present within the sample composites, which had an impact on the test program and selection of the process route.

13.2 Bond Work Index

Whole rock samples were not available to directly generate a Bond ball mill work index. To obtain a relative idea of the work index, RDi was able to calculate an indirect Bond index using grinding information from known samples. The Coringa ore appears to be very hard. The test results indicate the indirect Bond index for the Serra and Meio composite are 18.82 and 22.25. These values are estimates and require confirmation by actual testing. A work index of 20 was used for the grinding energy calculations within the operating costs.

13.3 Grind Studies

A series of grinding tests were performed to obtain the necessary grind times to produce a P_{80} of 65, 100, 150 and 200 Mesh products. This information was used for subsequent metallurgical tests to generate the proper size material for subsequent tests. A laboratory rod mill was used for sample grinding as it better simulates a full size ball mill operating in closed circuit.

13.4 Gravity tests

Gravity tests were performed at three grind sizes of P_{80} of 65, 100 and 150 mesh. Following grinding, the slurry was fed to a 3.5 inch diameter Knelson Concentrator. After filtering and drying of the products the material was processed over a Gemini Concentration Table to produce a final concentrate and tailings product.

- At a relatively coarse grind of P_{80} of 100 mesh or coarser, gravity concentration methods will recover 40 to 50% of the gold in a concentrate assaying greater than 400 gpt gold. The concentrate was less than 1% of the total mass.
- As the primary grind becomes finer, the gold tends to report to the Knelson tailing.
- The silver recovery in the gravity process was 13 to 23%. Most of the silver was lost to the tailings.

13.5 Flotation tests

Two flotation approaches were evaluated for the recovery of precious metals. First a bulk sulphide float was used to collect all products, gold, silver, lead, zinc and pyrite into a single concentrate. The second approach consisted of differential flotation of lead, zinc and pyrite concentrates.

- Bulk sulphide flotation recovered 90 to 95% of the gold and 70 to 80% of the silver into one concentrate assaying 90 to 97 gpt gold and 106 to 150 gpt silver. The concentrate weight was 8 to 14% of the original mass. Recovery of lead was 60 to 80% while zinc was 68 to 87%. A combined concentrate of this nature would not only be difficult to market but would have a low smelter return on contained value.

- Differential flotation resulted in the precious metals being distributed across the lead, zinc and pyrite concentrates. Meio lead and zinc concentrates were marketable, the concentrates produced from the Serra would be difficult to market.
- Results indicate the best flotation route would be as a bulk sulphide float for the recovery of precious metals; however this route does not indicate the best economics for the project.

13.6 Whole ore leach tests

Two series of carbon-in-leach (CIL) and whole ore cyanidation tests were performed on both the Coringa composites. The first series was completed at a grind of P_{80} 200 mesh, 1 g/L cyanide at 40% solids for 48 hours. These same conditions were used for both the CIL and whole ore cyanidation tests for both Serra and Meio composites. **Table 13.2** presents the results of these tests.

Table 13.2 Initial Cyanide Leach Tests – 200 MESH

Test Parameter	Serra		Meio	
	Cyanidation	CIL	Cyanidation	CIL
48 Hour Leach Gold Recovery	91.7%	98.9%	86.5%	86.0%
NaCN Consumption kg/t	1.785	2.002	2.384	2.664
Lime Consumption kg/t	10.236	7.747	6.779	7.765

Initial results indicate the ore is free milling and non-refractory. The Serra ore has some pre-robbing characteristics indicated by the 7% difference in recovery between the direct cyanidation and CIL tests. The reagent consumptions were high and yet acceptable due to the relatively high grade of the ore.

Additional CIL tests were performed to confirm the initial leach results, test the impact of grind size, and test pre-aeration as a potential method to reduce reagent consumption. The additional test conditions were the same as the prior tests with the following changes; the tests were conducted at P_{80} 100 and 200 mesh, and a pre-aeration of 4 hours. Prior to aeration the sample was slurried to 40% solids and the pH was adjusted to 11 using hydrated lime.

The results of the first series of leach tests were confirmed by the second series. **Table 13.3** presents the results from the second series of leach tests.

Table 13.3 CIL Leach Tests with Pre-Aeration

Test Parameter	Serra		Meio	
	100 MESH	200 MESH	100 MESH	200 MESH
48 Hour Leach Gold Recovery	97.2%	99.0%	92.5%	97.7%
NaCN Consumption kg/t	1.121	1.173	1.117	1.766
Lime Consumption kg/t	4.458	5.081	5.672	6.195

As can be seen from the results in **Table 13.3,** the CIL recovery at both grind sizes was excellent. Based upon these results, it does appear that Meio would benefit from the finer grind size. Pre-aeration reduced reagent consumption significantly for both ore types. An added benefit was increasing the recovery of Meio by 11% when compared to the CIL test without pre-aeration.

Based upon the excellent CIL results, and the relatively low value of the floatable base metals, a conventional CIL milling circuit with pre-aeration would be a feasible method for treating the Coringa ores. However, due to the silver content within the resource and resultant overall high grade of combined gold and silver, a Merrill-Crowe circuit may be required.

13.7 Proposed additional work

The excellent results from the preliminary scoping tests indicate the processing of the Coringa ores should be relatively straight forward using known processing methodologies. As the project is advanced, additional metallurgical testing should be completed. The following is suggested;

- Bond Work Index
- Abrasion index
- Testing of high-grade and low-grade ores to confirm recoveries similar to the composite testing
- Confirm the samples tested, and test new samples which are more spatially and mineralogically representative of the deposit
- Grind size optimization
- Define pre-aeration conditions
- Leach conditions, time and reagent concentrations
- Complete filtration and settling tests
- Tailings detoxification for placement as underground fill during the mining cycle

14.0 MINERAL RESOURCE ESTIMATES

14.1 Introduction

The mineral resource estimate reported for the Coringa project was completed by Zachary J. Black, E.I.T., Geological Engineer of Gustavson Associates (Gustavson). At the time of this report, sufficient data was not available to estimate mineral resources for all of the exploration targets. Gustavson has estimated mineral resources for the Valdette, Meio, Mae de Leite, Come Quieto, Serra, Galena, Demetrio, and Sr. Domingo shear zones.

Brazil has a defined environmental impact and permitting procedure for mineral extraction, an established export market for gold, and a stable political and legal environment which supports mining exploration and exploitation. For these reasons there are no anticipated factors relating to environmental, permitting, legal, socio-economic, marketing, political or other factors that would materially affect or preclude the potential development of the estimated resource at the Coringa Project.

Drawings illustrating the layout of the resource blocks estimated along with plan and section views of the vein systems are located in **Annex 1** of this report.

14.2 Deposit Geology Pertinent to Resource Estimation

Mineralization at Coringa project is encountered in a series of en-echelon strike-slip structural zones related a predominant dextral strike-slip fault. Although the main structural feature at Coringa has a regional 310° trend, most of the known mineralized structures are slightly oblique (345° strike) to this structure. The shear zones dip vertically to sub vertically (75° to 90° dip) and are known as the Valdette, Galena, Mae de Leite, Meio, Come Quieto, and Serra zones. The Sr. Domingo zone strikes 310° and is interpreted to be part of the main strike-slip structural zone responsible for the en-echelon Coringa system. Limited exploration work has been completed on this structure to date.

Gold mineralization at the Coringa project occurs within the north-northwest striking structural zones within granite and rhyolite. The system is defined by typical shear hosted lenticular quartz veins. Three types of veins have been identified on the project.

- Extensional (Galena, Mae de Leite zone, Serra zone, and Come Quieto),
- Fault vein fills (Meio zone)
- Stockwork/sheeted veins (Valdette, Demetrio and Sr. Domingo zones)

14.3 Data Used for Resource Estimation

Gustavson created a model for estimating mineral resources at Coringa from data provided by GRE. Drill hole data representing collar coordinates, down-hole surveys, sample assay intervals, and geologic logs were provided as Microsoft® comma delimitated files (csv). GRE was responsible for the validation of the data provided. Gustavson database validation was limited to checking for sample interval overlaps,

missing intervals, and erroneous values. Geology surface maps, geophysical studies, and cross-sections detailing the geology interpretations from each drill hole were provided in electronic format.

The Coringa drill hole database provided to Gustavson contains gold and silver assay analytical information for 11,907 sample intervals. The drill hole spacing is generally on nominal 100 meter centers. The azimuth and inclination of drill holes are approximately perpendicular to the vein. The majority of the drilling has been focused on testing the zones of interest throughout the district. **Drawing 2** in **Annex 1** illustrates the distribution of the drill holes in the resource area.

GRE utilized Leapfrog 3D® geological modeling software version 2.4.5.17 to create 3D solids representing the individual veins, and to define the Valdette structural zone. Magellan geologists defined the structure, stratigraphy, and alteration for each drill hole on cross-sections oriented along the strike of the drill hole, to best account for orientation of the vein target. GRE combined Magellan's subsurface interpretations with the drill hole geology to create 3D solids representing the veins and the Valdette structural zone (**Drawing 2 Annex 1**). The structural model in Leapfrog defined the footwall and hangingwall of the discrete veins or the shear zone using the assay, alteration, alteration intensity, and lithology intercepts. The intercepts were connected in a logical manner by viewing the data in three dimensions.

The Valdette zone was modeled as a shear zone. Similarly to the other resources areas, the assay intercepts were viewed in three dimensions to determine the structural limits of the shear zone. The Valdette area is bound by a footwall and hangingwall structure with intermittent assays within the mineralized zone. The shear zone model indicates that vein structures occur erratically throughout the area.

14.4 Method

Gustavson estimated the mineral resources by evaluating the 3D model provided by GRE for each vein with sufficient data, and geostatistically analyzing the available data to define the estimation parameters used to interpolate grades into a 3D block model. The mineral resources were estimated using CAE Datamine mining software version 3.20.6140.0.

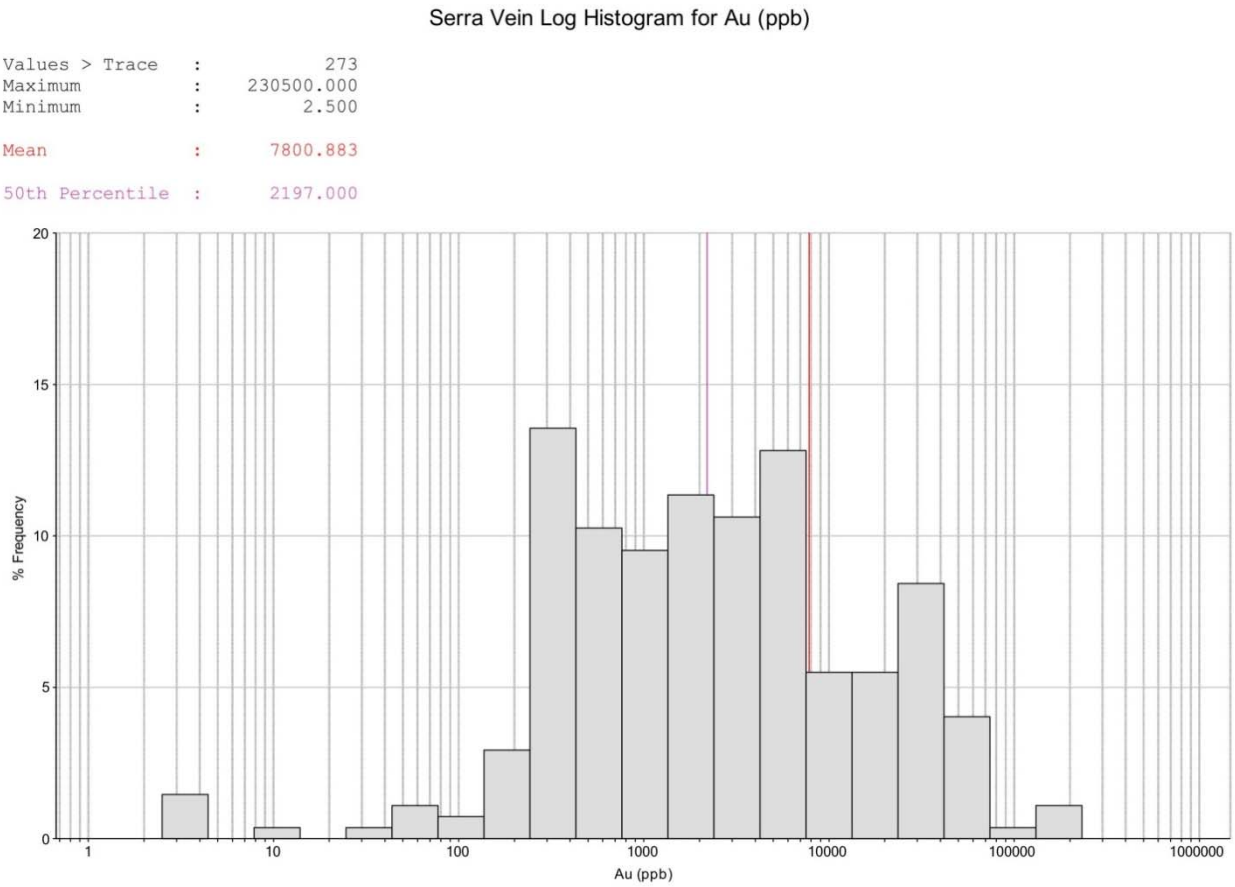
14.5 Modeled Zones

For estimation purposes, the Coringa deposit was separated by vein or structural zone. A 3D block model for each vein and structural zone was created. Each block model was oriented perpendicular to the strike of the vein system being modeled. Six separate zones were modeled.

- Galena-Mae de Leite zone
- Serra zone
- Come Quieto-Meio zone
- Valdette zone
- Demetrio zone
- Sr. Domingo

Assay sample data that intersected the modeled solids were separated for estimation. The descriptive statistics for each zone were calculated to ensure that the data selected within each zone was statistically related (**Figure 14.1**).

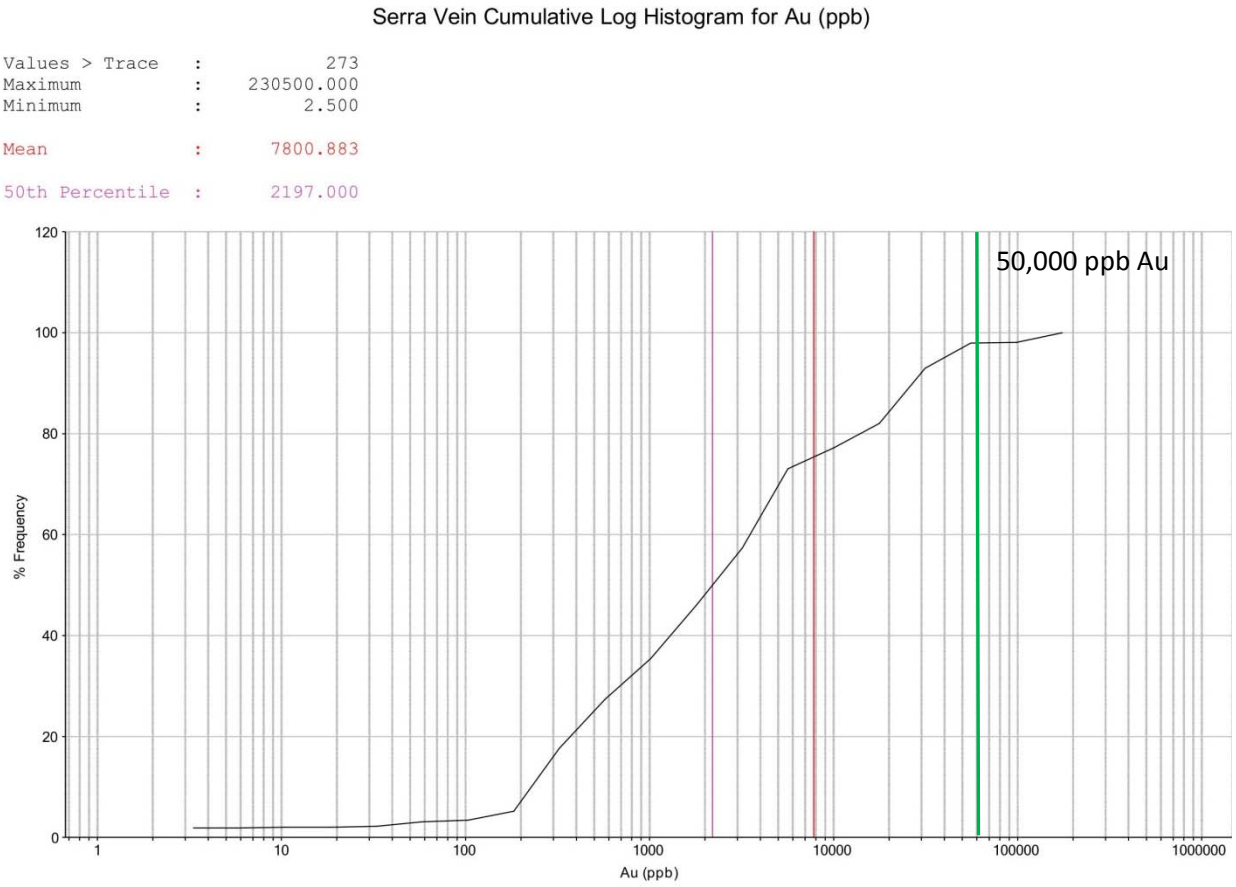
Figure 14.1 Serra Vein Gold Samples Log Histogram



14.6 Capping

Gustavson evaluated the Cumulative Histogram Plot (CHP) for both gold and silver in each of the zones with adequate data, and as a total population. The cumulative histogram plot for the total population was used to select an appropriate cap for both gold and silver. This was assumed to be indicative of all zones as the same break in the data as identified in the Serra zone CHP (**Figure 14.2**). Gold drill hole sample values were capped at 50 gpt (50,000 ppb) and the silver sample values were capped a 60 gpt.

Figure 14.2 Serra Vein Cumulative Log Histogram for Gold Samples



14.7 Compositing

The drill hole samples were all composited into true thickness intervals for the discrete vein models. The composite length of the intervals was calculated using the strike and dip of the modeled vein and the length of the sample interval. The composites were then used for the grade capping analysis and variography for each zone. The Valdette zone was composited using a gold indicator of ≥ 1,000 ppb to code samples as a 1. The samples were then composited into equal 1.5 meter intervals based on the indicator code. **Table 14.1** below summarizes the composited statistics for the individual zones.

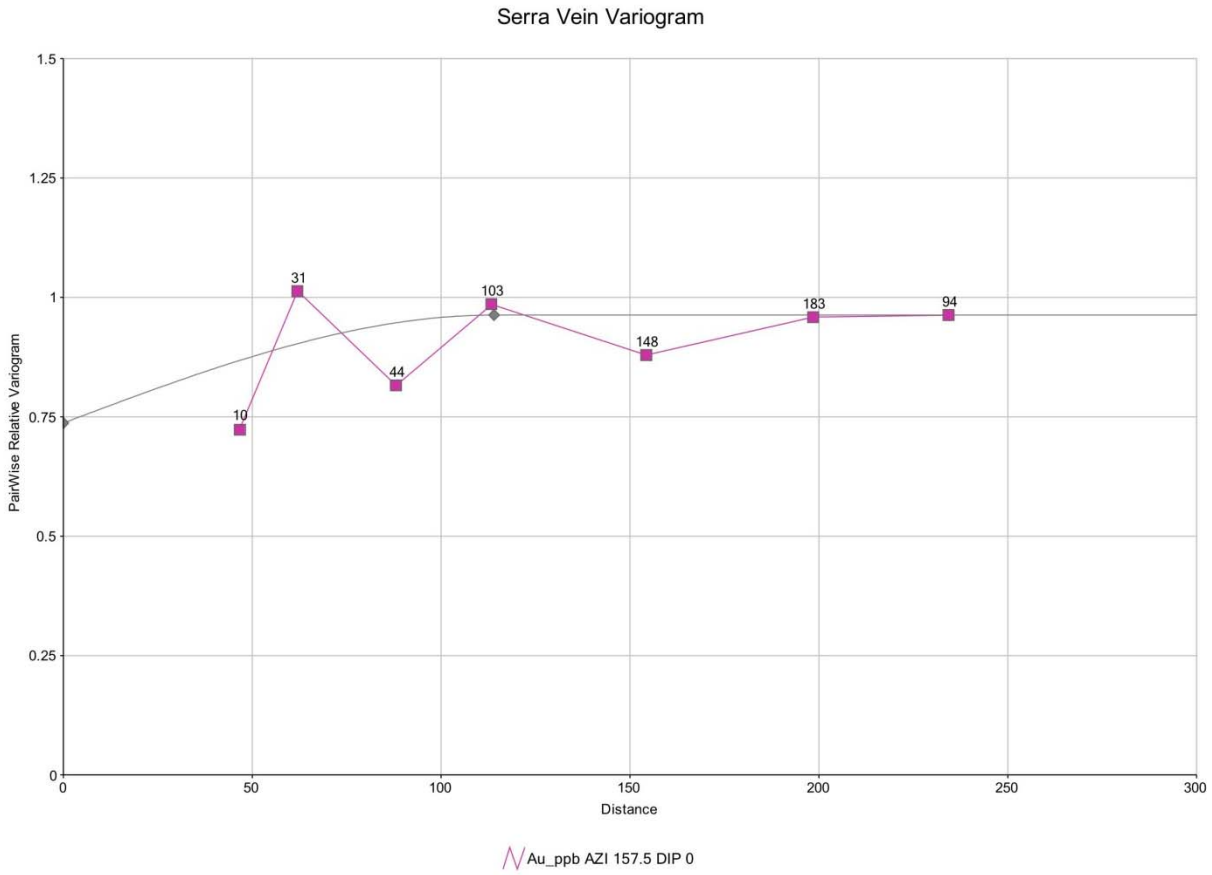
Table 14.1 Composite Descriptive Summary Statistics

Demetrio Zone Composite Statistics					
Field	Samples	Min	Max	Mean	Std. Dev.
Gold (ppb)	4	1,193	9,500	4,609	3,548
True Thickness (m)	4	0.33	3.15	1.73	1.00
Galena-Mae de Leite Zone Composite Statistics					
Gold (ppb)	32	3	21,611	4,607	5,652
True Thickness (m)	32	0.21	4.50	1.02	0.93
Come Quieto-Meio Zone Composite Statistics					
Gold (ppb)	52	7	65,772	6,194	10,532
True Thickness (m)	52	0.24	3.46	1.06	0.90
Serra Zone Composite Statistics					
Gold (ppb)	75	45	118,271	9,265	16,148
True Thickness (m)	75	0.13	2.05	0.61	0.38
Valdette Zone < 1000 ppb Au Composite Statistics					
Gold (ppb)	478	3	869	145	171
Valdette Zone Indicator ≥ 1000 ppb Au Composite Statistics					
Gold (ppb)	25	1,068	4,005	1,946	711

14.8 Variography

An analysis of the Variography was completed for the Serra zone to establish the spatial variability of the mineralization. Variography establishes the appropriate contribution that any specific composite should have when estimating the value of a block volume within a model. This procedure is performed by comparing the orientation and distance used for the estimation with the variability of other samples with similar relative directions and distances. An example of a spherical variogram constructed from the major axis of the Serra Zone using a "Pairwise Relative" method of organizing of the variance pairs is shown in **Figure 14.3**. The variogram model used to define search ranges for classifying mineral resources was determined to have a range of 100 m. This is assumed to be indicative of all zones at Coringa.

Figure 14.3 Serra Vein Spherical Variogram



14.9 Estimation Methodology

GRE was provided with the drill hole database in Microsoft Access file format which was verified by GRE prior to the resource calculation.

Gustavson estimated the mineral resources for the six discrete zones provided by GRE: Demetrio, Sr. Domingo, Galena-Mae de Leite, Meio-Come Quieto, Serra, and Valdette. A block model was created for each vein system using blocks that are 1 m long, 10 m wide and 5 m high, oriented along the strike of the vein being modeled (**Table 14.2**). Blocks that the vein passed through were divided into sub-blocks in all directions to better define the boundaries and volume of the zone being modeled. Each of the blocks was assigned attributes of estimated gold, silver, resource classification, rock density, and vein code. The first five areas were modeled as discrete vein systems while Valdette was modeled a shear zone using an indicator method.

Demetrio, Sr. Domingo, Galena-Mae de Leite, and Meio-Come Quieto were modeled as single vein. Three continuous veins were modeled in the Serra zone and are defined as the footwall vein, hangingwall vein, and intermediary vein which traverses a path from the footwall to the hanging wall (**Drawing 3 Annex 1**).

Table 14.2 Individual Zone Block Model Parameters

Zone	Origin			Number of Blocks			Increment			Rotation
	X (m)	Y (m)	Z (m)	X	Y	Z	X (m)	Y (m)	Z (m)	degrees
Demetrio	714,120	9,164,560	-10	148	93	76	1	10	5	320
Sr. Domingo	712,224	9,163,563	89	70	48	39	1	10	5	295
Galena-Mae de Leite	714,617	9,165,733	-10	233	239	77	1	10	5	330
Come Quieto-Meio	716,716	9,161,589	-10	815	510	86	1	10	5	330
Serra	715,768	9,165,698	-10	352	139	84	1	10	5	320
Valdette	713,657	9,166,805	-10	241	175	71	1	10	5	320

The Valdette zone is characterized by erratically distributed high grade gold and silver bearing veins, within a more continuous lower grade mineralized zone, and low grade mineralization disseminated between the veins. Gustavson chose a single stage indicator approach to model the portion of high grade vein material within each block, and then a separate estimate for the halo and inter-vein disseminated mineralized grade. This type of estimate limits the extrapolation of high grade mineralization into lower grade hangingwall and footwall units. Intervals with composite values greater than 1,000 ppb Au were assumed to be vein material for this estimate.

The gold grade was estimated from 1.5 meter down hole composites using an Inverse Distance Squared method. Composites assigned a 0 or 1 vein code, where a value of 1 means the composite is vein, 0 meaning it was not vein (wallrock). The estimate of vein percentage within each block was performed from the 0 and 1 vein codes. The grade of the vein portion of each block was estimated from the composites that had been coded with a 1 (vein). The final estimate of the grade representing the wallrock portion of the block, was estimated in a similar fashion, using only wallrock composites (vein code = 0).

The average grade of the block was then calculated using the percentage of the block that is vein, times the grade of the vein portion of the block, plus the percentage of the block that is wallrock, times the grade of the wallrock portion of the block.

The gold and silver grades within the veins and the Valdette zone were estimated and each block was assigned a classification of indicated or inferred. The resource classification of each block was based upon a factor of the distance in an anisotropic direction as established by the variogram ellipsoid for the vein being estimated. The measured class was estimated using a ¾ variogram search distance (75 m x 75 m x 7.5 m) with at least 2 composites from two different drill holes with no more than 5 total composites. The indicated class was estimated using a full variogram search distance (100 m x 100 m x 10 m) with a single composite required and no more than 5 total composites. The Inferred class was set

at 3 times the variogram distance (300 m x 300 m x 30 m) with a single composite required and no more than 5 total composites.

Inverse Distance Squared (ID^2) was used to estimate grade for all domains. Estimation search directions for each of the domains are given in **Table 14.3**.

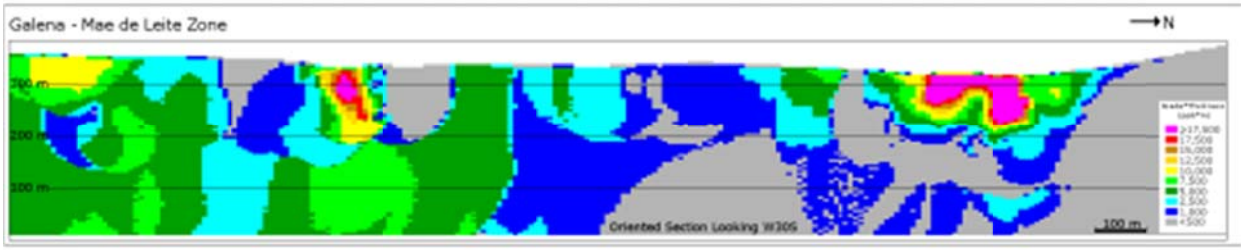
Table 14.2 Individual Zone Orientations

Zone	Dip Direction	Dip
Demetrio	320	-85
Sr. Domingo	295	-85
Galena-Mae de Leite	240	-85
Come Quieto-Meio	330	-85
Serra	320	-75
Valdette	320	-80

14.10 Estimate Validation

The model was validated by examining the blocks with actual drill hole assay data to determine if the estimated blocks fit the geologic parameters of the various domains of the deposit. Both assay and geological constraints were visually examined. Gustavson also created 2-dimensional grade thickness models to validate the estimated grades and true thickness of the individual vein models (**Figure 14.4**). The individual models, true thicknesses, grades, and tonnages compared favorably.

Figure 14.3 Grade Thickness Map of the Galena-Mae de Leite Zone



14.11 Mineral Resource Classification

The resource classification of each block was based upon a factor of the distance in an anisotropic direction as established by the variogram ellipsoid for the vein being estimated. The measured class was estimated using a ¾ variogram search distance (75 m x 75 m x 7.5 m) with at least 2 composites from two different drill holes with no more than 5 total composites. The indicated class was estimated using a full variogram search distance (100 m x 100 m x 10 m) with a single composite required and no more

than 5 total composites. The Inferred class was set at 3 times the variogram distance (300 m x 300 m x 30 m) with a single composite required and no more than 5 total composites.

14.12 Mineral Resource Tabulation

The mineral resource estimate for the Coringa Project is summarized in **Table 14.4**. A density of 2.7 tonnes per cubic meter was used in the estimation. Gustavson knows of no existing environmental, permitting, legal, socio-economic, marketing, political, or other factors that might materially affect the mineral resource estimate. The base case economic cut-off (2 gpt) reported in **Tables 14.4** through **14.6** were calculated assuming a 94 percent recovery and a mining cost of $US 75/tonne with a three-year trailing average gold price of $US 1,255 per ounce.

$$Cutoff = \frac{Mining\ Cost}{Price\ of\ Gold * Recovery}$$

Additionally, a sensitivity analysis was run to evaluate the potential open pit resources assuming a mining cost of $US 35/tonne, resulting in an economic cut-off of 1 gpt. This was completed to evaluate the potential of the lower grade higher tonnage mineralization encountered in the newer drilling on the Valdette zone. No economic analysis has been completed on the open pit scenario.

Table 14.4 Summary Resources, All Areas

Cutoff Grade Au gpt	Unit	1 gpt	2 gpt
Measured	tonnes	1,151,000	931,000
	grade gpt	6.81	8.06
	grams, Au	7,835,000	7,505,000
	troy ounces, Au	252,000	242,000
Indicated	tonnes	2,023,000	1,290,000
	grade gpt	4.75	6.63
	grams, Au	9,613,000	8,551,000
	troy ounces, Au	309,000	274,000
Measured & Indicated	tonnes	3,174,000	2,221,000
	grade gpt	5.50	7.23
	grams, Au	17,448,000	16,056,000
	troy ounces, Au	561,000	516,000
Inferred	tonnes	5,511,000	2,720,000
	grade gpt	3.01	4.67
	grams, Au	16,614,000	12,714,000
	troy ounces, Au	534,000	409,000

Table 14.5 Summary Resources by Area, 2 gpt Au cutoff

Zone	Measured				Indicated				Measured Plus Indicated				Inferred			
	Gold				Gold				Gold				Gold			
	tonnes	gpt	grams	troy oz.	tonnes	gpt	grams	troy oz.	tonnes	gpt	grams	troy oz.	tonnes	gpt	grams	troy oz.
Serra	446,000	8.27	3,688,000	119,000	324,000	10.59	3,432,000	110,000	770,000	9.25	7,120,000	229,000	408,000	7.41	3,025,000	97,000
Valdette	---	---	---	---	---	---	---	---	---	---	---	---	60,000	2.32	139,000	4,000
Meio-Come Quieto	285,000	8.14	2,319,000	75,000	844,000	5.13	4,333,000	139,000	1,129,000	5.89	6,652,000	214,000	1,355,000	3.64	4,936,000	159,000
Galena-Mae de Leite	200,000	7.49	1,498,000	48,000	122,000	6.44	786,000	25,000	322,000	7.09	2,284,000	73,000	449,000	4.9	2,199,000	71,000
Demetrio	---	---	---	---	---	---	---	---	---	---	---	---	448,000	5.39	2,415,000	78,000
Total	931,000	8.06	7,505,000	242,000	1,290,000	6.63	8,551,000	274,000	2,221,000	7.23	16,056,000	516,000	2,720,000	4.67	12,714,000	409,000

Table 14.6 Summary Resources by Area, 1 gpt Au cutoff

Zone	Measured				Indicated				Measured Plus Indicated				Inferred			
	Gold				Gold				Gold				Gold			
	tonnes	gpt	grams	troy oz.	tonnes	gpt	grams	troy oz.	tonnes	gpt	grams	troy oz.	tonnes	gpt	grams	troy oz.
Serra	520,000	7.29	3,793,000	122,000	383,000	9.23	3,534,000	114,000	903,000	8.11	7,327,000	236,000	482,000	6.55	3,156,000	101,000
Valdette	18,000	1.22	22,000	1,000	227,000	1.33	303,000	10,000	245,000	1.33	325,000	11,000	1,661,000	1.33	2,216,000	71,000
Meio-Come Quieto	335,000	7.15	2,395,000	77,000	1,196,000	4.06	4,859,000	156,000	1,531,000	4.74	7,254,000	233,000	1,948,000	3.01	5,865,000	189,000
Galena-Mae de Leite	278,000	5.85	1,625,000	52,000	217,000	4.23	917,000	29,000	495,000	5.14	2,542,000	81,000	564,000	4.2	2,367,000	76,000
Demetrio	---	---	---	---	---	---	---	---	---	---	---	---	856,000	3.52	3,010,000	97,000
Total	1,151,000	6.81	7,835,000	252,000	2,023,000	4.75	9,613,000	309,000	3,174,000	5.5	17,448,000	561,000	5,511,000	3.01	16,614,000	534,000

All numbers have been rounded to their appropriate level of accuracy

15.0 MINERAL RESERVE ESTIMATES

No mineral reserve estimates have been calculated for the project.

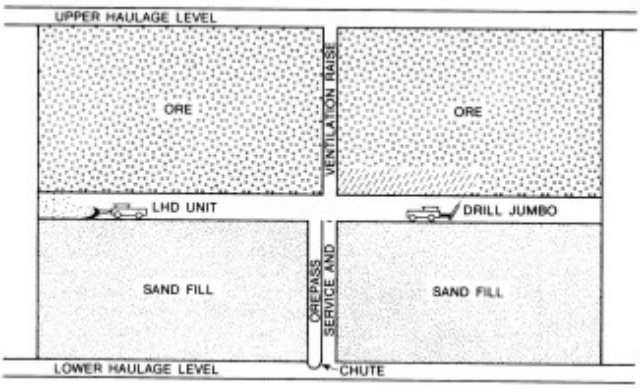
16.0 MINING METHODS

16.1 Mining Method Selection

The overhand cut and fill mining method was selected for the updated PEA study due to its prominence as a preferred mining method, the ability to provide a high degree of grade control, the lower preproduction development requirements, better ability to mine difficult or broken ground, and the availability to begin backfilling the stopes early in the mine life with a tailings slurry.

Cut and fill mining offers a high degree of flexibility during the mining process. This method completes a defined cut within the stope and then fills that void before each successive cut. Overhand cut and fill progresses from the bottom of the stope to the top and uses the fill as the working platform for the following cut. Overhand cut and fill requires a lower strength fill relative to underhand cut and fill since the fill acts as the working platform. In contrast, underhand cut and fill requires a higher strength fill but does allow for mining the deposit from the top down which decreases the required development. The main benefit to cut and fill mining is mining selectivity. Although steady state operating costs are higher than shrinkage stoping, cut and fill requires less preproduction development and is better able to mine difficult or broken ground. **Figure 16.1** shows a conceptual overhand cut and fill stope.

Figure 16.1 Overhand Cut and Fill



Hartman, Howard L. SME Mining Engineering Handbook. Aug – 1992. Sacramento, CA.

16.1.1 Production Rate & Mine Life

The mine production rate and mine life was determined from the new resource estimate completed in May 2012. The current resource has expanded greatly from the first PEA study, extending the defined resource at depth and along strike. Preliminary mining blocks were created using long sections delimiting the resource above 3 gpt gold. Each potential mining block was then broken out into the resource categories: measured, indicated, and inferred. An infill drilling plan was developed to upgrade the logical portions of the inferred resources within each block to measured and indicated. New mining

blocks were then calculated including measured, indicated, and only the inferred resources to be upgraded through the infill drilling program. Given the substantial increase in tonnage, an increase in the mine production rate from the previous study to 750 tonnes per day (tpd) was used. Additional cutoff grades of 4 and 5 gpt gold were evaluated to provide a rough estimate of gold production throughout the mine life using 350 working days per year. The base case scenario was chosen using a 4 gpt cutoff grade and a 750 tpd production rate. There is a two year ramp up period to 400 tpd by the end of year 1 and 750 by the end of year 2. The ramp up period during year 1 allows for required development and training of the new mine workforce. The further increase to steady state production in the second year is in line with other known operations with similar mining conditions. The resultant mine life is 8.5 years.

16.1.2 Mine Layout and Design

The mine layout was created on longitudinal sections. The mine was divided into separate areas based on the 3 primary resource areas identified which contain 92% of the gold resource at the 4 gpt Au cutoff: Serra (41%), Meio-Come Quieto (MCQ) (33%), and Galena-Mae de Leite (GAMDL) (17%). The PEA considers mining 70% of the resource in these areas. Resource blocks Valdette and Demetrio are not mined in the PEA. Each area will have a main portal where rubber tired mining equipment can enter the resource block ramp system to access each mine level. Ancillary portals are required for the MCQ and GAMDL mining areas to provide additional access points since they cover a larger extent along strike. A minimum of two ventilation raises near the ends of each area will allow for adequate ventilation of each level and minimize the amount of dead air space that requires ventilation through auxiliary fans and ventilation tubing. A third ventilation raise was included for mining areas with a larger extent along strike. Development ramps will be driven at 10% maximum grade to access all areas of the mine. Crosscuts from each main haulage level will provide access to the stopes with a total of 3 crosscuts per 100 m long stope. Only one vein is mined in the MCQ and GAMDL blocks while 3 separate veins are mined in the Serra block. Crosscuts within the Serra block will connect one or more veins to the main haulage level to minimize required development. Due to the narrow mining width of 1.5 m within most stopes, slushers will be utilized within each stope to pull ore to the closest ore chute where rubber tired load haul dump (LHD) and trucks will be used to load and haul ore out of the mine. Backfilling of the stopes with a tailings slurry will be completed on one half of the stope while mining progresses on the other half so that stope production is never delayed by backfilling. The processing facilities will be located closest to the Serra block which contains the bulk of the resource. Existing roads between the resource blocks and the process plant will be improved to allow for surface ore haulage to the plant at 10% maximum road grades. A thickened tailings storage facility is located to the southwest of the process plant. Thickened tailings will be deposited in this facility through a gravity assisted tailings disposal pipeline. **Drawing 02** shows the general facility layout of the project. **Drawings 3 through 11** shows the plan and section views of each mining block. **Drawing 12** illustrates the typical stope and development sections. **Drawings 13 through 15** illustrate the tailings impoundment plan, section, and details.

16.1.3 Design Criteria

A detailed set of mine design criteria is shown in the tables below.

Table 16.1 Mine Method

Design Item	Criteria
Mine Method	Overhand Cut and Fill
Stope cut Type	Back Stoping
Haulage Type	Rubber Tired LHD and Truck
Main Mining Equipment	Slusher, Jackleg Drill, Stoping Drill

Table 16.2 Mine Layout

Design Item	Criteria
Stope Length	100 - 150 m
Stope Min./Max./Avg. Height	16 m/100 m/ 66 m
Stope cut Height	2.3 m
Cribbed ore chutes	4 per 100 m stope
Crosscuts	3 per 100 m stope
Wood Slides	1 per stope
Cribbed Manways	1 per stope
Sill Pillar Thickness	6 m
Definition drilling stations	2 per stope
Crown Pillar Thickness	5 m
Min. Mining Width	1.5 m
Ramp Heading Cross Section (width x height)	4.0 m x 4.0 m
Crosscut Heading Cross Section (width x height)	3.0 m x 3.0 m
Stope Powder Factor	0.44 kg / tonne
Development Powder Factor	1.35 kg / tonne
Ventilation	Exhaust raise - Axial Fan, 2 minimum per resource

Design Item	Criteria
	block
Small underground shop	5 total: 1 Serra, 2 GAMDL, 2 MCQ
Underground lunch room	5 total: 1 Serra, 2 GAMDL, 2 MCQ
Underground warehouse	5 total: 1 Serra, 2 GAMDL, 2 MCQ
Safety Refuge	5 total: 1 Serra, 2 GAMDL, 2 MCQ

Table 16.3 Operational Recovery and Dilution

Design Item	Criteria
Overall Recovery	94%, based on average stope height and remaining pillars
Final Recovery of Sill Pillar	67%
Final Recovery of Sill Pillar	60%
Fill dilution due to mining cemented backfill	5%
Loss into cemented backfill	5%

Table 16.4 Fill Design

Design Item	Criteria
Cemented Top Thickness	0.5 m
Uncemented Thickness	1.8 m

Table 16.5 Production Rate

Design Item	Criteria
Tonnes per Day	750 tpd, 0-200 tpd Year 1; 200-750 tpd Year 2
Weeks off per Year	2 wks
Mine Operating Days per Year	350 days
Annual Tonnage	262,500 tonnes
Total Tonnes Mined	2,000,000 tonnes
Mine life	8.5 yrs.

Table 16.6 Material Properties

Design Item	Criteria
Ore Density	2.7 t/m^3
Waste Density	2.6 t/m^3
Swell	40%

Table 16.7 Major Surface Mine Equipment

Equipment	Quantity
2.3 CM Wheel Loader - Plant	1
6 CM Highway Dump Truck	1
3.7 M Grader (Surface)	1
22.7 kL Water Truck	1

Table 16.8 Major Underground Mine Equipment

Equipment	Quantity
3 CM LHD	8
20 TONNE HAUL TRUCK	5
2 BOOM HYD JUMBO	2
Underground Grader	2
134 CFM JACKLEG	2
160 CFM Stoper	15
30 HP Slusher	13

17.0 RECOVERY METHODS

17.1 General

The following sections contain excerpts and summary information from the NI 43-101 Technical Report, Preliminary Economic Assessment, Coringa Project Novo Progresso, Brazil – April 30 2010.

17.2 Process Plant Design

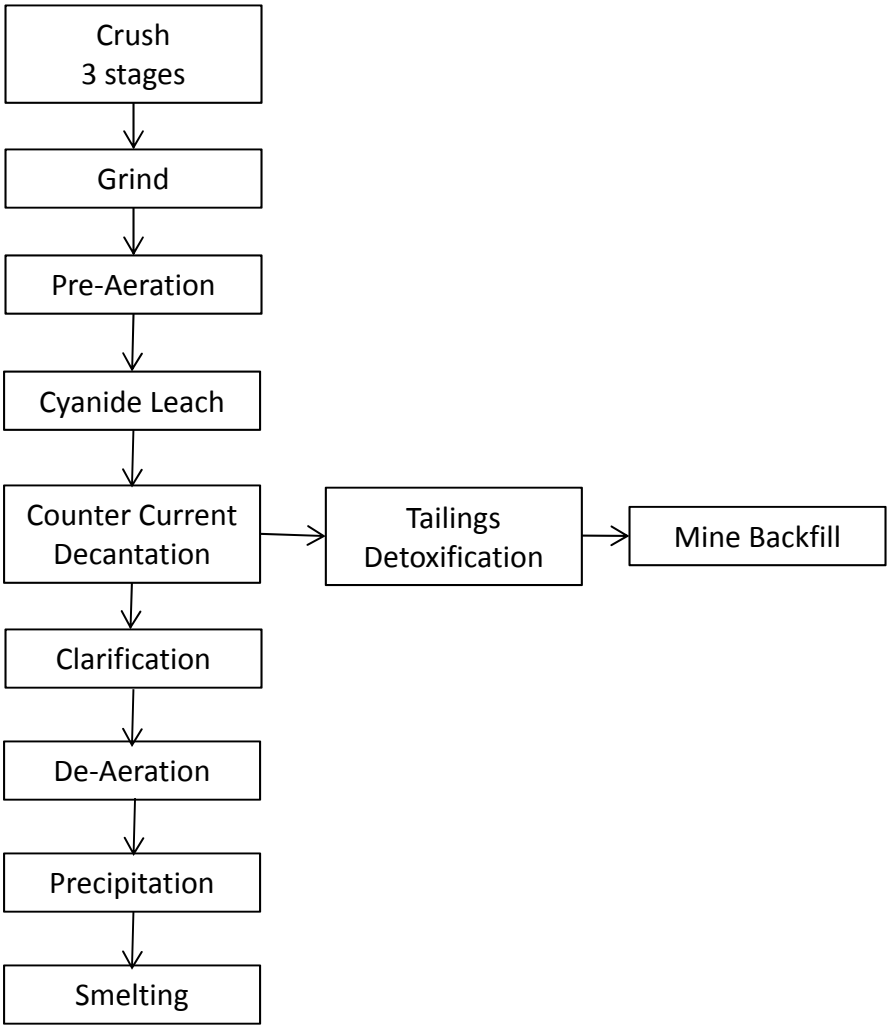
17.2.1 Metallurgical testing / process selection

Preliminary scoping tests were completed at Resource Development Incorporated (RDi), Wheatridge, CO, on two composite samples. The samples represented the Serra and Meio deposits. Laboratory work included grinding, gravity, flotation and cyanidation test, (**See Section 13**). All tests gave reasonable to good gold recoveries. The carbon-in-leach (CIL) process resulted in excellent gold recoveries greater than 90%. The Serra ore is slightly pregnant solution robbing, and the use of the CIL process during testing resolved this issue. The CIL process route is a feasible processing method based on these results. However, due to the silver content within the resource and overall high grade of combined gold and silver, a Merrill-Crowe circuit may be required. A Merrill-Crowe circuit was used in the PEA.

17.2.2 Processing description

Based upon the preliminary metallurgical scoping work completed to date and the significant presence of silver within the resource, a Merrill-Crowe processing circuit was estimated for the Coringa project. Three stages of crushing (12.5 mm) would feed a ball mill to produce a 200 Mesh product. Prior to leaching the pH would be adjusted with lime to pH 11 followed by four hours of aeration to reduce reagent consumption and improve recoveries. The aerated slurry will be leached with cyanide to produce a pregnant solution. The solution would be clarified to remove solids and then de-aerated. Zinc powder would be added to the resultant solution to precipitate the precious metals. It would then be filtered to produce a filter cake and smelted onsite to produce a doré. The tailings would undergo detoxification prior to disposal. A portion of the tails will be used as backfill underground with the remainder being placed within a tailings impoundment. Figure 17.1 shows the conceptual process flow sheet. The requirement for supplies which include electricity, water, and processing materials is projected to be comparable to the 500 tonne per day agitated cyanide hydrometallurgical model published by Western Mine Engineering Inc.

Figure 17.1 Conceptual Process Flow Sheet



17.2.3 Design Criteria

The design criteria for the proposed process plant are shown in the following table.

Table 17.1 Processing Design Criteria

AREA	CRITERIA
Crushing	Three stages, P_{80} 12.5 mm
Grinding	P_{80} 200 Mesh
Pre-aeration	4 hours, pH 11
Leaching	40% solids, 48 hours retention, 1g/l cyanide
Tailings	70% solids
Tailings detoxification	TBD

18.0 PROJECT INFRASTRUCTURE

18.1 Infrastructure Requirements

The project will require additional infrastructure and upgrading of existing facilities before production can commence. The existing infrastructure includes a main access road from the paved highway BR-163, site access roads, exploration camp, and airstrip. The main access road will need to be improved to allow access for commercial trucks bringing supplies to the project. This will consist of widening the road and constructing surface water ditches over its entire length, installing culverts and other water management structures, and upgrading the stream crossings with designed bridge structures. The site access roads will need to be upgraded to allow access for surface ore hauling from the mining areas to the process plant. This will consist of widening the road and reducing the maximum grade in some areas. Surface water ditches, culverts, low water crossings, and other water management structures will be required along these access routes to provide dependable year round access. The existing camp must be upgraded to provide facilities for additional personnel. The airstrip should be graded to drain with surface water ditches constructed along its entire length to provide a dependable landing surface for light aircraft.

Additional infrastructure is needed at the project site which includes:

- High voltage power line
- Water supply
- Tailings storage facility (TSF)
- Plant site complex
- Mine portals

A high voltage power line along the main access road to the project site will be constructed. Power line spurs will be constructed to each mining area and the tailings storage facility from this main line. This will include a substation at the tie-in to the main power line just east of intersection of the main access road with the highway BR-163. Additional substations are required at the camp connection, plant site complex, and each mining area.

A raw water supply for the project has yet to be defined. Both surface and underground water supplies are feasible for the site. There are 2 defined surface water sources with the project and a river located to the east as described in **Section 5**. Alternatively, a series of water wells constructed in the overlying saprolite could be constructed to provide water. As described below, the TSF will provide mix of raw and recycle water that will be used in the process plant.

A TSF will be constructed on the southern end of the project to the west of the Meio-Come Quieto resource block. This will consistent of an earth-fill embankment, wet tailings impoundment, and a tailings supply line and water return line to the plant site complex. The TSF impoundment will capture

some raw water which will be available for use in the process plant. Additionally, recycle water from tailings consolidation will be available for use in the process plant.

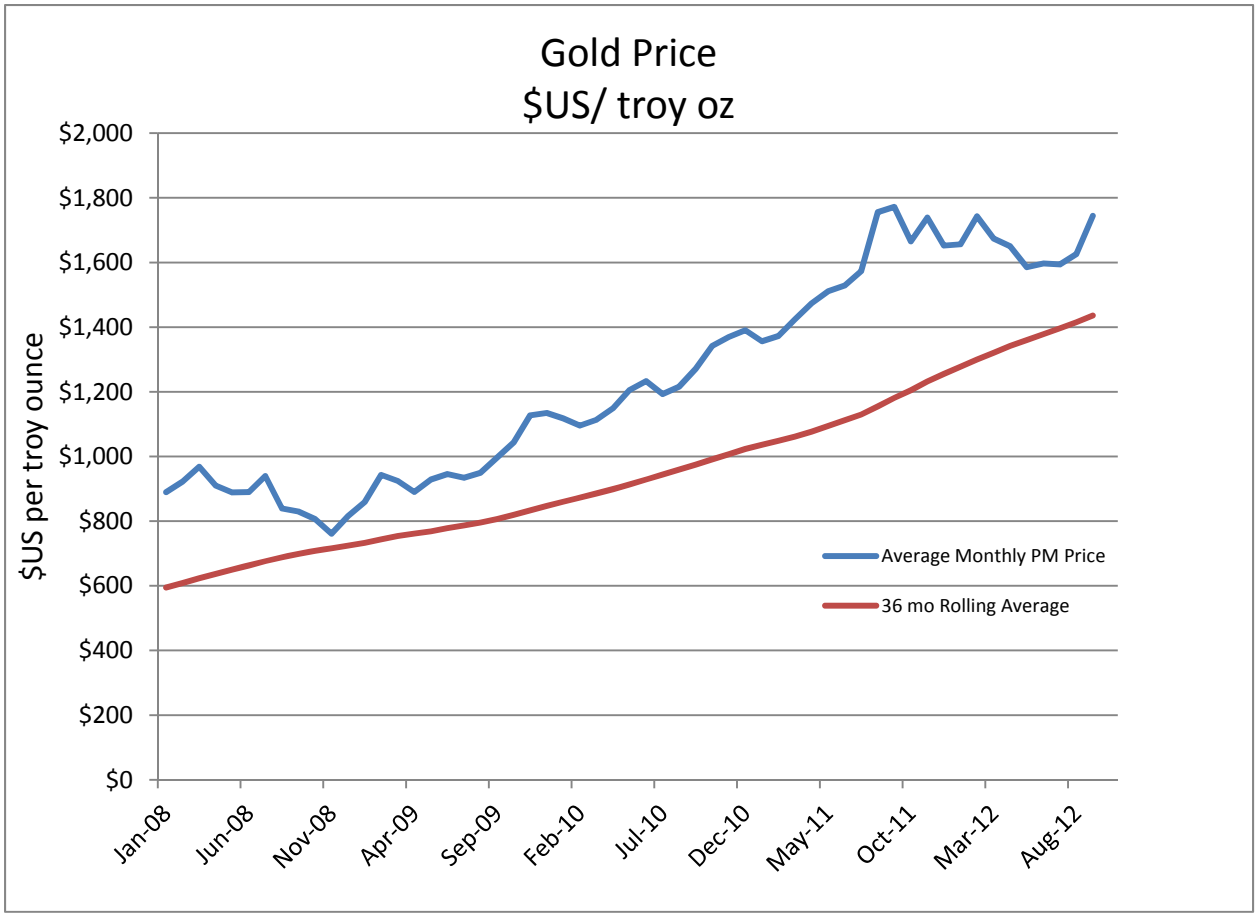
A plant site complex will be constructed between the Serra and Meio-Come Quieto resource blocks along the existing access road. The site complex will include the process plant, assay lab, change room, maintenance shop, warehouse, management offices, backfill plant, and fuel depot. At a safe distance from the plant site complex, a secure location with surrounding safety berm and security fence will be constructed for placement of the high explosives magazine and ANFO storage bin.

Mine portals will be constructed at each mining block. Mining blocks will have one main portal and ancillary portals as necessary to provide access to underground workings. Each portal location shall consist of a graded area with sufficient dimensions for a covered stockpile, laydown yard, ore storage bin, tool storage container, and sediment pond. The main portals will have additional area for an office building, power center, and compressor station.

19.0 MARKET STUDIES AND CONTRACTS

Brazil is a significant producer of gold with several well-known gold companies currently operating in the country. There are smaller firms which are focused on developing projects in the state of Para, the same host state as the Coringa project. Due to the dramatic increase in gold price since 2000, gold has become Brazil's second most important mining export after iron ore. Brazil's main gold producing states are Minas Gerais, Goias, Bahia and Para accounting for 64%, 11%, 11% and 3% of overall production respectively. The three main gold producers in Brazil are AngloGold Ashanti, Yamana Gold, and Kinross. Smaller firms such as Jaguar Mining and Eldorado Gold make up the remaining share of production. Eldorado Gold is focused on bringing their Tocantinzinho project in Para state into production. They are continuing with the preparation of an environmental impact assessment and additional technical studies to allow them make a decision for construction in 2012. **Figure 19.1** presents the average monthly gold price since 2008 with a 36-month (3-year) rolling average.

Figure 19.1 Gold Price



20.0 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 Environmental Impact Study

An Environmental Impact Study (EIS) was completed during 2008 by Terra Meio Ambiente for the exploration phase, which involved Claims N ° DNPM 850567/1990, 850568/1990, and 850981/2006. The EIS provided:

- An environmental synopsis of the area of influence
- Description of the activities
- Evaluation of the impacts
- Proposed control, mitigation and compensation measures.

The study in its final consideration highlights two scenarios. The first one considers the project area without the exploration/mining potential. In this scenario, the area is deemed to be used solely for farming with livestock, which is a very important activity for the region and for Para State. Livestock farming, however, has a low rate of wealth creation per km^2, compared to mining. Projects like the Palito mine in the Para State generate employment for 400 people, in an area of no more than 250 ha. compared to the employment of less than 10 people for farming livestock in a similar area.

The second scenario, considers potential gold mining as an activity co-existing with livestock farming on the 19,730 ha. Such scenarios can be seen in other areas of the region.

20.2 Environmental Liabilities

The main environmental liability of the project is an old milling area used by the artisanal miners which covers approximately 1.3 ha. The area was regraded and revegetated with tree seedlings by Chapleau in 2008; however, some of the tree seedlings did not survive. The surface regrade appeared stable during both site visits by Mr. Kevin Gunesch PE of Global Resource Engineering (GRE) in May 2009 and January 2012. The surviving seedlings experienced notable growth from 2009 to 2012. Some areas showed additional erosion due to storm runoff and were confined to small areas at the slope bottoms which were at a slightly steeper grade. This erosion could be controlled by decreasing the slope grade at the bottom to produce a concave slope and also revegetation of the area with grasses. Acid Rock Drainage (ARD) was noted in the stream flowing through this area. The source of the ARD appears to be the exposed ore which was left at the site by the old miners. A scoping level geochemical characterization of the waste rock and tailings was completed for the PEA study which provides an initial estimate of the ARD risk for future mining operations.

20.3 Site Water Management

A site wide water management plan will be required for the property. Baseline groundwater and surface water samples should be collected. A year of data is typically accepted to substantiate baseline

conditions. Samples should be collected quarterly or more often depending on site specific conditions. The groundwater sampling will require the installation of a monitoring well network up gradient and down gradient of the project area.

20.4 Environmental Permitting

The Federal law 6938/1981 outlines the Environmental policy and the requirements of Environmental permits for any activity with contaminant potential, or involved with natural resources.

The Environmental permit is divided into three (3) stages/ categories:

Previous Permitting (LP: "Licenca Previa"): this permit deals with the selection of the best place for developing the activity.

Installation Permitting (LI: "Licenca de Instalacao"): this permit deals with the construction of the project, according to a technical project previously approved.

Operation Permitting (LO: "Licenca de Operacao"): this permit allows you to begin the mining activities.

The environmental organisation has a six (6) month period to approve or deny the permitting, from the time of the application.

If the required permit needs an Environmental Study (EIA/RIMA) where a Public Audience is asked, this approval period may extend to twelve (12) months. In the case of the Para State, this period is not legislated.

The National Environmental Council (CONAMA) has established a list of activities, which needs to be presented for a Previous Environmental Impact Study (EPIA/RIMA). The mining activity is included in that list. Generally speaking a Public Audience is asked to attend.

The Environmental Permitting needs to be renewed every 1 to 5 years. In the case of the Coringa Project, for exploration and the construction of a decline, adit and/or shaft, a "Term of Reference" was prepared for the Environmental Secretary of Para State (SEMA/PARA). This "Term of Reference" defined the scope of the Previous Environmental Study (EPIA/RIMA), which was carried out by Chapleau Resources Ltd, through Terra Meio Ambiente, an environmental consulting firm based in Belem, Para State. The study was presented on January the 12th, 2009, and approved for exploration (including diamond drilling) on August the 5th, 2009. On March 10, 2010 the Pará State Environmental Authority (SEMA) granted Magellan's wholly owned Brazilian subsidiary Chapleau Exploração Mineral Ltda. the permit to extract 50,000 tons of ore for metallurgical studies at Coringa. The permit is valid for one year and does not allow for the installation of a plant to process any of the ore. In March 2012 an application was made for the renewal of this Environmental permit.

20.5 Tailings Disposal

Both dry filter and thickened tailings disposal options have been evaluated for the project. A thickened tailings storage facility (TSF) was selected and design for disposal of tailings from the project through a weighted advantage/disadvantage matrix analysis. The primary reasons for selecting thickened tailings disposal were:

- Sub-aqueous disposal of tailings greatly reduces the potential for acid rock drainage (ARD) from the tailings.
- Tailings impoundment provides a disposal location for potentially acid generating (PAG) development and waste rock.
- Impoundment will provide some storm water storage which can be used in the process plant.
- Low capital and operating costs.

The site location for the TSF is on the southern end of the property to the west of the Meio-Come Quieto resource block. This site was selected as the preferred construction site from 4 separate options. The main reasons for the site selected were the lower impoundment area and location away from known resource areas. A topographic saddle separates the TSF from the plant site complex with a 15 m elevation difference between the saddle and the plant site complex and a 70 m elevation difference between the saddle and the TSF impoundment. In addition to tailings storage, the TSF is envisioned as a storage and encapsulation facility for co-disposal of PAG waste rock.

The density of the settled tailings product after deposition in the impoundment is estimated at 1.2 t/m^3, or 70% solids by weight. Based on this density, 1.0 million cubic meters capacity is required for storing 60% of the total 2.0 million tonnes of tailings. 40% of the tailings will be used in backfilling the mined out stopes. The current design includes 400,000 cubic meters of additional capacity for PAG material or 1.0 million tonnes. The current development tonnes are estimated at 1.4 million tonnes of which only a small portion would be PAG material. The current location allows for an expansion of the TSF to a total of 3.8 million cubic meters (4.5 million tonnes at 1.2 t/m^3) of storage by increasing the crest elevation from 301 m to 320 m.

A detailed set of design criteria was developed to design the TSF, shown in tables below.

Table 20.1 Operation and Base Engineering

Design Item	Criteria or Approach
Total Tailings Production	2,000,000 tonnes
Specific Gravity	2.7 t/m^3
Settled % Solids by weight	70%
Tailings Settled Dry Density	1.2. t/m^3
Total Storage Capacity, 60% of total production	1,200,000 t (1,000,000 m^3)
Design Storm	100-yr, 24-hr + 2 m freeboard

Table 20.2 Tailings Embankment

Design Item	Criteria or Approach
Dam Construction Type	Downstream raise earth fill.
Diversion	Uphill perimeter diversion to divert rainwater from collecting in impoundment.
Operational Discharge	Operational discharge by floating barge pump
Spillway – Emergency Discharge	Spillway capacity for 100 yr. storm event
Cyanide Limits WAD/Total	0.2 ppm / 1.0 ppm
Dam Slopes	1.5H:1V upstream and 2.5:1 downstream slopes
Dam Crest Width	10 m
Perimeter Road width	10 m minimum width on perimeter benches for tailings disposal line
Underdrain	Underdrain installed beneath embankment in main valley drainage

Table 20.3 Tailings Impoundment

Design Item	Criteria or Approach
Impoundment Type	Unlined – Planned cyanide destruction and water treatment of tailings discharge to remove heavy metals
Tailings Disposal Line	Perimeter and embankment disposal to control location of water pool, away from dam at all times
Water Return	Barge pump with flexible connection pipe for free movement within the water pool area

Table 20.4 Closure Requirements

Design Item	Criteria or Approach
Impoundment	Tailings surface graded to drain toward spillway with wetland established on surface
Impoundment	Tailings surfaced capped with 1 m inert waste rock and topsoil for revegetation

20.6 Mine Closure

Mine closure will be completed during the year following the end of mine operations. This will consist of reclaiming the tailings storage facility, removal of all surface facilities and equipment, closing and sealing of all underground mine workings and allowing them to flood, as well as regrading and revegetating any areas disturbed during the mine life. The estimated closure costs are presented in **Section 21**.

21.0 CAPITAL AND OPERATING COSTS

21.1 Capital Costs

Capital cost estimates were completed using estimates from Mine and Mill Equipment Cost Guide 2011 published by Infomine (Infomine 2011), manufacturer and supplier quotes from US companies, past professional experience, and quotes for key mine equipment from suppliers in Brazil. Where appropriate, adjustments have been made to the Infomine estimates to account for Brazilian labour and construction costs. **Table 21.1** below provides a summary of the start-up capital costs for the project.

Table 21.1 Start-Up Capital Summary

Capital Category	Total Capital Cost ($US)
Plant Site Complex	$27,300,000
Surface & Underground Facilities	$19,200,000
Mine Equipment	$11,600,000
Additional Engineering Studies & Site Investigations	$3,200,000
Preproduction Development	$1,900,000
Permitting	$700,000
Working Capital	$600,000
Total	$64,500,000

All capital costs include a 21% contingency except for preproduction development which includes a 15% contingency since they were calculated through detailed unit cost estimates.

21.1.1 Surface and Underground Facilities

Initial surface facilities including the process plant, warehouse, assay lab, and water treatment plant were estimated as a lump-sum item (plant site complex) and are discussed in the following section. Initial surface facilities that were estimated separately are the power line, maintenance shop, office complex, change room, existing camp expansion, fuel depot, tailings storage facility, backfill plant, explosive magazine, ANFO storage bin, and main and ancillary mine portals.

The tailings storage facility (TSF) estimate was completed in detail using estimated quantities and unit costs and adjusted for local contractor costs for excavation, placement, and compaction of engineered fill. Expansion of the TSF is required by year 4 following production and is included in sustaining capital. The remaining facilities were estimated from Infomine (Infomine 2009) and professional experience.

Initial underground facilities include only the underground shop and refuge and the main portal for the initial mining block. Additional main and ancillary portals are required throughout the mine life and are included in sustaining capital.

A listing of initial surface and underground facilities and their estimated costs are shown in **Table 21.2** below.

Table 21.2 Initial Surface & Underground Facilities

Surface Facility	Quantity	Unit Capital Cost ($US)	Total Capital Cost ($US)
Power line	1	$7,600,000	$7,600,000
Start-up Tailings Storage Facility	1	$4,000,000	$4,000,000
Maintenance Shop	1	$661,000	$661,000
Office Complex 40' by 100 ' steel	1	$339,000	$339,000
Camp Expansion	1	$91,000	$91,000
Change room 40' by 60' steel	1	$203,000	$203,000
Backfill Plant	1	$1,900,000	$1,900,000
60k gallon Fuel Depot	1	$378,000	$378,000
Explosive Magazine	2	$30,500	$61,000
27.2 tonne ANFO storage bin	2	$59,000	$118,000
50m Water Supply Wells	2	$500,000	$1,000,000
Rounded Total			$16,400,000
Underground Facility	**Quantity**	**Unit Capital Cost ($US)**	**Total Capital Cost ($US)**
Main Portal	1	$1,600,000	$1,600,000
UG Shop/Refuge	5	$235,000	$1,175,000
Rounded Total			$2,800,000
All	**Quantity**	**Unit Capital Cost ($US)**	**Total Capital Cost ($US)**
Rounded Total			$19,200,000

21.1.2 Process Plant

The capital cost estimate for the process plant was sourced from Infomine (Infomine 2009) for an Agitated Cyanide Leach mill operation based on the metallurgical testing. This estimate includes the purchase and installation of all equipment required for the plant unit operations. All wiring, piping, foundations, on-site utilities, mill buildings, offices, labs, tailings disposal, and coarse ore storage are included. The operations covered in this capital cost are as follows:

- Crushing
- Grinding
- Agitation Leaching

- Countercurrent decantation
- Pregnant solution holding
- Pressure clarification
- De-aeration
- Merrill-Crowe zinc precipitation
- Precious metal pressure filtration
- Carbon column scavenger recovery from bleed streams and tailing return water
- Acid pre-treatment of precipitates
- Bullion refining and casting
- Tailings disposal

Due to the ARD potential of the tailings and waste rock, a water treatment plant for tailings discharge to the TSF was added to the capital cost of the plant. The plant will consist of a high-density sludge (HDS) treatment system that will effectively neutralize any potential ARD and will remove dissolved heavy metals. The capital cost of a treatment plant with a discharge rate of 10 l/s is estimated at $US 550,000. This estimate is based on information from SGS Lakefield in Vancouver for a similar project. Plant construction costs were adjusted to reflect Brazilian labour rates. An adjustment to the capital cost was also made for a larger ball mill than that provided in the Infomine estimate. An indirect work index of 20 kwh/ton (22.1 kwh/tonne) has been calculated for the project indicating the ore in much harder than the 11.7 kwh/tonne work index used in the Infomine model.

The total estimate for the plant complex is $US 22,500,000.

21.1.3 Initial Mine Equipment

Required equipment for the surface, underground, and process plant operations were determined by calculating the operational hours required for each piece of equipment and divided by the total available operating hours per piece of equipment. This calculation included an allowance for 85% equipment availability and 77% efficiency since the mine trucks and LHDs will move between different stopes during operations which requires unproductive tramming time. The requirement for small equipment was based on the number of active concurrent resource blocks and stopes during the mine life. Initial equipment capital was summarized into surface and underground categories. **Tables 21.3 and 21.4** present these capital costs. A larger mine equipment fleet is required once during steady state production. These equipment purchases are included in sustaining capital.

Table 21.3 Initial Surface Mine Equipment

Equipment	Quantity	Unit Capital Cost ($US)	Total Capital Cost ($US)
2.3 CM Wheel Loader - Plant	1	$213,000	$213,000
6 CM Highway Dump Truck	1	$113,000	$113,000
3.7 M Grader (Surface)	1	$384,000	$384,000
22.7 kL Water Truck	1	$472,000	$472,000
Rounded Total			$1,200,000

Table 21.4 Initial Underground Mine Equipment

Equipment	Quantity	Unit Capital Cost ($US)	Total Capital Cost ($US)
Main Fan (700 HP) with motor	1	$561,000	$561,000
Aux. fan 14000 cfm	2	$20,000	$40,000
3 cubic meter LHD	2	$867,000	$1,733,000
20mt underground truck	2	$1,069,000	$2,138,000
Jumbo (2-boom) 7m x 5m face	1	$ 973,000	$973,000
Jack Leg Drill (134 cfm)	2	$12,000	$24,000
Stoper Drill (160 cfm)	5	$7,000	$34,000
Scissor Lift Truck (3.7m)	1	$335,000	$335,000
Major Pump (Submersible-2500 gpm@100' head:150Hp)	2	$102,000	$204,000
Aux. Pump (350 gpm:12.5Hp)	2	$9,000	$17,000
Air Compressor (750 cfm-80Hp)	1	$ 91,000	$91,000
UG Grader	1	$369,000	$369,000
Pick-up	5	$34,000	$169,000
SUV	2	$34,000	$68,000
Mechanics truck	2	$355,000	$709,000
Lube truck	2	$342,000	$685,000

Equipment	Quantity	Unit Capital Cost ($US)	Total Capital Cost ($US)
Explosives Truck	2	$465,000	$929,000
Anfo air loader	1	$22,000	$22,000
Raise Climber	2	$672,000	$1,343,000
Rounded Total			$10,400,000

21.1.4 Sustaining Capital

Sustaining capital consists of additional equipment purchases, required rebuilds, and new portal facilities throughout the mine life. The initial capital equipment requirement in Year -1 consists of only development mine equipment. The initial production mine equipment is purchased in Year 1 with additional purchases through Year 4. The main mining fleet during steady state production consists of 8 LHDs, 5 trucks, 15 stoper drills, and 13 slushers. Rebuilds for the LHDs are required in Years 6 and 7. New portal facilities are required as mining transitions to the second and third mining blocks. Each mining block contains a main portal facility and up to two ancillary portals. Sustaining capital totals $US 18.5M over the mine life and is summarized by category in **Table 21.5** below.

Table 21.5 Sustaining Capital

Type	$US
UG Mine Equipment, Additional Purchases	$11,600,000
UG Mine Equipment, Rebuilds	$1,200,000
Portal Facilities	$5,700,000
Total	$18,500,000

21.1.5 Working Capital

Three months of operating and development costs were added for working capital and is broken out into 2 categories: cash or liquid working capital and inventory or tangible working capital. This provides a 3 month inventory of raw materials, spare parts, and supplies along with a 3 month cash buffer for cash costs such as general and administrative costs, labour, and loan interest. This amount increases until steady state production is realized in Year 3 of the project. 100% of the cash working capital is recaptured at the end of the project life along with 50% of the tangible working capital since these items would be resold to the market.

21.1.6 Preproduction Mine Development

Costs for mine development were estimated by the actual meters of development scheduled for each year multiplied by the development unit cost for each activity. Mine development consists of 7 types of development activities:

1. Ramp
2. Level
3. Crosscut with Utilities
4. Crosscut without Utilities
5. Ventilation Raise
6. Ore Compartment Raise (ore chute)
7. Three Compartment Raise (manway with 2 ore chutes)

The unit cost for each development activity was determined by estimating the equipment hours and materials required for one blasting round of each operation and then dividing by the meters of advance. Labour was not included in these unit costs (or other unit operating costs) but was estimated for the project as a whole by workforce type. Two types of crosscuts were estimate, one with and one without utilities. Utilities such including power, air, water, backfill pipe, and ventilation tubing are only required in one crosscut per stope. **Table 21.6** below lists the development unit costs used to estimate the mine development capital. A more detailed discussion of each development unit operation is included in section 21.3.2.

Table 21.6 Unit Development Costs

Development Type	$US/m
Ramp & Level	$692
Crosscut with Utilities	$624
Crosscut without Utilities	$345
Ventilation Raise	$707
Ore Chute Raise	$1,384
Manway and 2 Ore Chute Raise	$3,198

21.1.7 Closure

Closure costs for the project consist of reclaiming the tailings storage facility, removing all surface facilities, closing access to all underground workings, and revegetation of any disturbed lands. A detailed closure estimate for the tailings storage facility was completed which includes capping the entire surface with 1 meter of inert rock/soil, grading the area to drain out a permanent spillway, and constructing riprap channels along the regarded surface to prevent erosion. These operations totalled $US 2.5M. An additional $US 600K was added to the closure estimate to account for the remaining closure costs. Ongoing geochemical analysis will better define long-term environmental risks and their potential impact on closure costs.

21.2 Controllable Operating Costs

Controllable costs were estimated for each unit operation, excluding labour. Labour costs for mine operations, mine maintenance, mill operations, and salary labour were estimated separately. Hourly labour rates and annual salaries were obtained from a Brazilian salary survey from Catho, an ISO 9001:2008 certified company, for the 3 quarter of 2012. An exchange rate of 2.03 Brazilian reais to 1 USD was to convert labour costs to USD. Local costs for diesel were used in all estimates. A total workforce of 309 hourly employees and 22 salary employees is estimated for the mine and mill operations during steady state operations.

All operating costs include a 15% contingency.

21.2.1 Hourly and Salary Labour

Hourly and salary labour were estimated by determining the number of people within each workforce category, the required number of employees for one man coverage of mine and mill operations, and the average hourly rate or average annual salary. A burden rate of 40% was added to both salary and hourly labour costs. Overtime allowances for mine and mill hourly labour costs are 5% and 9% respectively to account for hours worked over 44 hours per week.

21.2.1.1 Mine Production

The mine will be operated 350 days per year with 2 seven hour shifts per day for a total of 4,900 operating hours per year. Mine operations employees will work a 6 day on 2 day off rotation. Removing hours for paid public holidays (12 days), vacation (22 days), sick leave (5 days), and training (3 days), each mine operator works a total of 1,622 available hours. Therefore to have one-man coverage for the entire 4,900 hours of operations time, a total of 4 employees are required. Each active stope or development heading requires a total of 2 mine operators. The average annual number of active stopes and development headings were estimated through the mine life. **Table 21.7** below shows the detailed year by year requirement for mine operations employees. The highest number of employees is required during the period 2018 through 2022 which represents steady state production and concurrent development. An average hourly labour rate of $US 5.00/hr. was estimated all mine operations employees. An hour rate of $US 5.75/hr. which includes a 15% contingency was used in the economic model.

Table 21.7 Mine Operations Employees by Year

Year	Period	Active Stopes	Active Development Headings	Employees per Active Area	Total Employees Required
2015	Pre-production development	0	2	2	16
2016	Ramp up Period	4	3	2	56
2017	Ramp up Period	9	3	2	96
2018-2022	Steady State	11	4	2	120
2023	Concurrent Development Decreases	11	2	2	104
2024	Cessation of Mining	5	1	2	48

21.2.1.2 Mine Maintenance

The mine maintenance workforce is estimated to contain a total of 26 employees: 8 labourers, 2 lead mechanic, 10 mechanics, 2 lead electrician, and 4 electricians. This is based on the estimate required for one mining block and that 2 mining blocks will be active during most of the mine life. Again multiplying the number of employees required by 4 employees for one person coverage of mine operations gives a total of 104 maintenance workers. **Table 21.8** illustrates the maintenance workforce and average hourly rate calculation of $US 5.52/hr. An hour rate of $US 6.35/hr. which includes a 15% contingency was used in the economic model.

Table 21.8 Mine Maintenance Workforce

Mine Maintenance Workforce	Hourly Wage $US	Per Crew	Total Employees Required
Labourer	$2.93	8	32
Lead Mechanic	$8.98	2	8
Mechanic	$5.73	10	40
Lead Electrician	$9.19	2	8
Electrician	$6.61	4	16
	Total Crew Employees	26	104
	Average rate	$5.52	

21.2.1.3 Mill Operations

The mill will be operated 365 days per year with 3 eight hour shifts per day for a total of 8,760 operating hours per year. Mill operations will work a 6 day on 2 day off rotation. Going through the same calculation as for mine employees results in a total of 1,854 available work hours per mill employee per year. This results in 5 employees required for one-man coverage. The mill workforce is comprised of both operations and maintenance personnel with a total of 17 employees: 1 lead mechanic, 2 mechanics, 1 lead electrician, 2 electricians, 6 mill workers, 2 lab technicians, and 3 mill helpers. Taking the required employees for one person coverage and multiplying by 17 gives us a total of 85 mill employees. **Table 21.9** shows the mill workforce and average hourly rate calculation of $US 6.54/hr. An hour rate of $US 7.52/hr. which includes a 15% contingency was used in the economic model.

Table 21.9 Mill Workforce

Mill Workforce	Hourly Wage $US	Per Crew	Total Employees Required
Lead Mill Mechanic	$9.88	1	5
Mill Mechanic	$6.30	2	10
Lead Mill Electrician	$10.11	1	5
Mill Electrician	$7.27	2	10
Mill Worker	$7.58	6	30
Lab Tech	$4.48	2	10
Mill Helper	$3.22	3	15
	Total Crew Employees	17	85
	Average rate	$6.54	

The salary workforce is estimated to be comprised of the following 22 individuals. **Table 21.10** presents the salary workforce and average annual salary calculation of $US 43,000. An average annual salary of $US 49,450 which includes a 15% contingency was used in the economic model.

Table 21.10 Salary Workforce

Salary Workforce	Monthly Wage $US	Number
General Manager	$11,108	1
Mine Manager/Superintendent	$7,510	1
Plant Manager/Superintendent	$7,510	1
Chief Mine Engineer	$7,510	1
Chief Geologist	$6,658	1
Metallurgist	$4,094	1
Plant Supervisor	$5,248	2
Accountant	$1,714	1
Purchase Agent	$1,714	1
Surveyor	$1,591	2
Mine Foreman	$1,726	2
Mine Maintenance Foreman	$1,726	2
Safety and Env. Engineer	$3,497	1
Doctor	$3,209	1
Secretary	$482	4
	Total Employees	22
	Average rate	$3,502
	Annual Rate	$42,019
	Rounded Annual Rate	$43,000.00

21.2.2 Unit Operations

Unit operating costs were estimated for each mining operation based on the materials and equipment hours to complete one work cycle. For each operation, a complete work cycle was estimated and then divided by the number of tonnes produced or meters of advance. **Table 21.11** summarizes the unit costs for each operation which include the 15% contingency.

Table 21.11 Unit Operating Costs

Unit Operation	$US/tonne	$US/m
Stope Production	$10.07	
Ramp and Level Development		$692
Crosscut with Utilities Development		$624
Crosscut without Utilities Development		$345
Ventilation Raise Development		$707
Ore Chute Raise Development		$1,384
Manway & 2 Ore Chute Raise Development		$3,198
Stope Backfilling	$17.31	
Surface Ore Haul	$1.43	
Power	$24.00	
Miscellaneous Equipment	$2.58	

Stope production unit costs considered a production block of 1.7 m wide by 4.8 meters long and 2.5 meters deep. The width was based on the average width of all stopes. The production block is drilled with pneumatic stopper drills, blasted, and then mucked to the ore pass by a 0.4 m^3 slusher. Once the ore has been mucked to the ore pass, LHD's remuck the ore hauling it out the crosscut and load it into a low profile haulage truck for transport out of the mine. The loaded ore trucks dump the broken ore into a storage bin of a batch loadout system located at each portal. If the bin is at capacity, the trucks can dump under the covered storage area provided by the fabric building at each portal.

Ramp and level development considered a production block 4.0 m wide by 4.0 m high and 3.6 m deep. The heading is drilled with a 2-boom jumbo, blasted and mucked with an LHD to a muck bay cut out along the ramp or level approximately 30 meters from the active face. Once the face is clear, roof bolts are installed in the new portion of the heading as required and drilling commences again. Remucking of the ore from the muck bay into haulage trucks by LHD's occurs simultaneously with roof bolting and the beginning of drilling for the new development face. Utility installation is included in the unit cost estimate.

Crosscut development considered a production block 3.0 m wide by 3.0 m high and 2.9 m deep. Operations for the crosscut development are similar to ramp and level development with the difference that mucking of the crosscut heading by the LHD brings the ore out to a muck bay located on the main

haulage level instead of a muck bay interior to the crosscut. Utility installation occurs in only one crosscut per stope.

One compartment raise development (ore chute) considered a 2.1 m by 2.6 meter cross section with 2.3 meters of depth driven by conventional methods with a raised platform. The face is drilled, blasted, and mucked from below by an LHD to a muck bay within the main haulage level. From there, the waste or ore is remucked with the LHD and loaded into a haulage truck. Simultaneously with remucking, cribbing is installed within the ore chute with angle iron covering the upper edges of each timber to protect the cribbing as ore falls down the chute.

Three compartment raise development (manway and 2 ore chutes) considered a 6.9 m by 2.6 m cross section with 2.3 m of depth. These raises are developed conventionally in the same manner as ore chute raises. However, they are cribbed in a fashion to provide 2 ore chutes with a central manway between them. Utilities are placed in the manway to provide power, air, water, and access for the backfill pipe to the working stope level. Access ladders and slides within the manway provide for worker access and the ability to bring materials to the working stope level.

Ventilation raises are completed using an Alimak raiser climber and considered a 2.0 m by 3.8 m cross section with a 2.3 m depth. The raise climber is trammed up a track installed on the back to the working face. The face is then drilled from the Alimak platform, blasted and then mucked from below by an LHD to an adjacent muck bay. The waste is then remucked from with an LHD into a haulage truck and transported to the surface. A steel ladder with landing platforms every 50 m is installed in ventilation raises to serve a secondary escape way.

Stope backfilling considers backfilling one half of a typical stope length or 50 m. The backfill volume is equal to the displaced ore volume within the stope allowing for a simple calculation of cost per tonne of ore. Backfilling is completed for an entire ore cut of 2.3 m high. During backfilling, the ore chutes and manway are raised above the fill level and wrapped in burlap. A PVC pipe is extended to end of the area to be backfilled with a second pipe installed in the floor to serve as a drain. The backfill slurry is pumped into the fill area from a slurry pump on the surface. The top 0.5 m of backfill contains 10% cement by volume to create a stable working surface to mitigate ore loss into the backfill during slushing operations. 5% ore loss is applied to each stope to account for ore loss in the cemented fill. 5% dilution is applied to each stope to account for dilution due to backfill that is inadvertently mined.

The unit cost for surface ore hauling took into account hauling 750 tonnes per day (tpd) to the process plant. Trucks are loaded through the batch loading system at each portal location. The tonne weighted average haul distance of 2.3 km was used to determine the truck productivity and required number of trucks during the mine life. One 10 cubic meter dump truck has sufficient production capacity to haul the ore from each portal to the plant operating 2 eight-hour shifts per day.

The unit cost for power considers line power supply at $US 0.165/kwh. The plant power was estimated as a whole and increased to account for a larger ball mill and addition of a backfill plant. A 50% allowance of the calculated plant power was added to account for the additional electrical demand from the office buildings, maintenance shop, change room, water supply wells, and other demands. The mine power requirement was estimated by line item, which includes the main and auxiliary fans, main and

secondary pumps, underground shop, slushers, and air compressor stations. The annual power demand was calculated by multiplying the mill power demand by 24 hours per day and the mine demand by 14 hours per day. This demand was divided by the annual production tonnage to determine the cost per tonne mined.

Finally, all miscellaneous mine equipment were combined to calculate a unit operation cost. This included the support loader, pickups, SUV's for mine personnel transport, fuel & lube truck, mechanics truck, explosive truck, mine dewatering pumps, auxiliary fans, jacklegs, and air compressors.

21.3 Non-Controllable Operating Costs

21.3.1 Taxes

GRE relied on Magellan to provide tax information for the project. There are a total of 2 separate taxes which are assessed to the project: Corporate Income Tax (IRJP) and Social Contribution on Profits (CSLL). Projects located in the North of Brazil, in the jurisdiction of the Superintendence for Development of North Brazil (SUDAM) are able to receive tax benefits applicable to the operational phase. A qualified project, considered by the Ministry of National Integration priority for the development of the North region of Brazil receives a 75% reduction of the IRPJ for a 10 year period. This benefit is available to projects that receive approval by December, 2013. For a project to be approved, it must be producing at least 20% of its projected capacity by December 2013. This tax benefit has been considered in the economic evaluation of the Coringa property and thus reduces the IRPJ from 25% to 6.25%. The CSLL at 9% is still applied to the project, making the overall tax rate 15.25%. As of the effective date of this Technical Report, the legal framework authorizing the extension of such tax benefits still exists. However, it depends on a case by case analysis carried out by SUDAM. It is not known if a specific extension would be granted for the Coringa Project. The reader is cautioned to take this fact into account when reviewing the presented economics.

Rural Land Tax "ITR" or property tax is paid by the property owner. Although Magellan is in active negotiations to acquire the surface rights, it does not currently own the property and therefore is not obligated to pay the ITR.

21.3.2 Royalties

Three set of royalties are levied on the project revenue:

1. The Financial Compensation for the Exploitation of Mineral Resources (CFEM) of 1% payable to the government.
2. Statutory royalty of half the CFEM of 0.5% payable to the property owner.
3. A NSR royalty of 2.5% paid to Sandstorm Gold Ltd.

21.3.3 Exploration Permit Fees

An annual fee per hectare is assessed to the area within the exploration license (23,754.44 hectares). In the case of Magellan, the rate of R$ 3.83 per hectare is applicable.

21.3.4 General and Administrative Expenses

General and Administrative expenses were estimated based on 25% of the annual salary labour cost including burden. This equates to roughly $US 1.67 per tonne.

21.4 Additional Model Parameters

21.4.1 Gold Price

A constant gold price of $US 1,350/troy oz. was used throughout the model. This represents the 36 month rolling average of the London PM Fix gold price as of May 2012.

21.4.2 Cutoff grade

A cutoff grade of 4 grams Au per tonne was used in this analysis.

21.4.3 Process Plant Recoveries

Process plant recoveries were based on the most recent metallurgical testing. The test work was completed for 2 separate composites which represent 90% of the resource. The test results obtained were a 99% recovery from the Serra composite, and 98% recovery for the Meio composite. Plant recovery was estimated at 5% less than the test results or 94% for the Serra mining block and 93% for the Meio-Come Quieto mining block. Testing was not completed on ore from the Galena-Mae de Leite resource block; a plant recovery of 93% was assumed for this mining block.

21.4.4 Depreciation

All capital expenditures were depreciated on a straight line basis over 10 years from the date of expenditure. All remaining depreciation was taken at the end of operating life (Year 9).

22.0 ECONOMIC ANALYSIS

22.1 Project Forecast

An economic model was created using the above capital cost and operating cost estimates. These costs along with gold sale revenues were estimated on a yearly basis to generate net cash flows. The following subsections give a descriptive overview of the project on a yearly basis. The overall mine production sequence begins with the Serra resource block in Year 1, followed by the Meio-Come Quieto (MCQ) block in Year 3, and finally the Galena-Mae de Leite (GAMDL) block in Year 7.

22.1.1 Preproduction – Year -1

A preproduction period of one year is required for mine development. The development will be completed with mine operations personnel under the direction of expatriate mine engineer who will oversee the development. This includes completing the ramp and the 310 m level in Serra. The majority of the ramp between the 310 m and 210 m level is also completed along with the 2 vent raises on either side of the first block of production stopes. Construction of the process plant complex, surface and underground facilities, power line, and the starter tailings storage facility occur simultaneously with preproduction development. Besides mine development, there are minor equipment operating costs during Year -1 consisting of power and small mine equipment. Full mine labour crews are only required for the development headings. Mine salary labour is shown at 75% of its full requirement and mill operations labour is shown at a minimum reflecting only a small workforce to assist in the process plant construction, training, and commissioning. Two development crews are required during the year.

22.1.2 Start-Up Production & Ramp-Up Period – Year 1 & 2

Mine production begins in Year 1 increasing to 400 tonnes per day (tpd) at the end of the year. Mine production continues a ramp up period in Year 2 from 400 tpd to 750 tpd. The average mine production is shown as 251 and 638 tpd during Years 1 and 2 respectively. Stopes are brought online in a progressive manner starting with 2 stopes in the first quarter of Year 1. Additional stopes are brought online each quarter with 6 active stopes by the end of the year. Five additional stopes are brought online in Year 2 to achieve full production at the end of year with 11 active stopes. Development crews increase to 3 crews in Year 1 and are maintained at 3 in Year 2.

22.1.3 Steady State Production and Concurrent Development – Years 3 through 7

Full production is achieved in Year 3 at 750 tpd and remains constant until mine resources are depleted in Year 9. Development crews increase to 4 crews in Year 3 and remain constant through Year 7. The tailings storage facility is constructed to its full capacity in Year 3. Production begins from the 2nd resource area, MCQ, in Year 3 of the project. The Serra area is completely mined out in Year 6. Production begins from the 3rd resource area, GAMDL, in Year 7. Eleven active stopes are required during full production.

22.1.4 Declining Development at Full Production – Year 8

Development decreases in Year 8 while production is maintained at 750 tpd with 11 active stopes. Development crews are reduced to 2 in proportion to the decrease in required development. Both the MCQ and GAMDL areas are active.

22.1.5 Cessation of Mine Operations – Year 9

Mine operations cease in Year 9 of the project as the MCQ and GAMDL areas are mined out. Only one development crew is required during the year. Stope production continues at the designed rate, but the number of active stopes dwindles throughout the year, reducing the annual production rate to 315 tpd.

22.1.6 Mine Closure – Year 10

Mine closure will be completed during the final year of the project following cessation of mining. This will consist of reclaiming the tailings storage facility, removal of all surface facilities and equipment, closing and sealing of all underground mine workings and allowing them to flood, as well as regrading and revegetating any areas disturbed during the mine life.

22.2 Economic Model Results

The results of the economic modeling indicate the project is viable, robust and provides excellent return on investment at current and projected gold prices. **Table 22.1** shows the yearly summary cash flows for the economic model. **Table 22.2** lists the average operating costs and average total costs per tonne and per troy ounce of gold recovered. **Table 22.3** summaries the cumulative cash flow, net present value (NPV), and internal rate of return (IRR) for the project. The project payback period is 2.9 years.

Table 22.1 Economic Model by Year

Coringa Underground Mine - Economic Model												
Summary By Year	**Year -1**	**Year 1**	**Year 2**	**Year 3**	**Year 4**	**Year 5**	**Year 6**	**Year 7**	**Year 8**	**Year 9**	**Year 10**	
	2015	**2016**	**2017**	**2018**	**2019**	**2020**	**2021**	**2022**	**2023**	**2024**	**2025**	**Total/Avg**
Production												
Total Tonnes Mined	-	87,938	223,437	262,499	262,495	262,659	262,363	262,550	262,448	110,107	-	1,996,496
Tonnes Mined per day	-	251	638	750	750	750	750	750	750	315	-	634
Gold oz	-	24,195	58,323	59,635	62,987	52,718	46,052	50,166	50,693	22,775	-	427,544
Grade gpt	-	8.56	8.12	7.07	7.46	6.24	5.46	5.94	6.01	6.43	-	6.66
Process Recovery	-	94.0%	94.0%	93.9%	93.6%	93.8%	93.2%	93.0%	93.0%	93.0%	-	93.5%
Recovered Gold	-	22,743	54,823	56,011	58,983	49,453	42,913	46,654	47,144	21,180	-	399,906
Revenue												
Gold	$0	$30,703,618	$74,011,454	$75,614,392	$79,627,360	$66,761,869	$57,932,951	$62,983,421	$63,644,770	$28,593,538	$0	$539,873,373
Minus Royalty	$0	$460,554	$1,110,172	$1,134,216	$1,194,410	$1,001,428	$868,994	$944,751	$954,672	$428,903	$0	$8,098,101
Minus NSR	$0	$767,590	$1,850,286	$1,890,360	$1,990,684	$1,669,047	$1,448,324	$1,574,586	$1,591,119	$714,838	$0	$13,496,834
Minus Exploration Permit Fee	$39,552	$39,552	$39,552	$39,552	$39,552	$39,552	$39,552	$39,552	$39,552	$29,664	$0	$385,629
Revenue Less Royalty and Fees	($39,552)	$29,435,921	$71,011,444	$72,550,264	$76,402,714	$64,051,842	$55,576,081	$60,424,532	$61,059,428	$27,420,133	$0	$517,892,809
Surface Rights Payment	$60,000	$60,000	$60,000	$60,000	$60,000	$60,000	$60,000	$60,000	$60,000	$45,000	$0	$585,000
Net Revenue	($99,552)	$29,375,921	$70,951,444	$72,490,264	$76,342,714	$63,991,842	$55,516,081	$60,364,532	$60,999,428	$27,375,133	$0	$517,307,809
Operating Costs												
Mining Production	$0	$885,891	$2,250,903	$2,644,411	$2,644,375	$2,646,024	$2,643,044	$2,644,926	$2,643,903	$1,109,219	$0	$20,112,697
Mine Backfill	$0	$1,521,993	$3,867,134	$4,543,195	$4,543,132	$4,545,967	$4,540,846	$4,544,080	$4,542,322	$1,905,679	$0	$34,554,348
Transport	$0	$125,400	$318,621	$374,323	$374,318	$374,551	$374,130	$374,396	$374,251	$157,013	$0	$2,847,003
Power	$351,761	$2,110,565	$5,362,598	$6,300,098	$6,300,011	$6,303,942	$6,296,841	$6,301,325	$6,298,887	$2,642,627	$0	$48,268,655
Small Mine Equipment	$113,265	$226,529	$575,573	$676,196	$676,187	$676,609	$675,847	$676,328	$676,067	$283,636	$0	$5,256,237
Processing	$0	$1,653,461	$4,201,173	$4,935,630	$4,935,562	$4,938,642	$4,933,079	$4,936,591	$4,934,682	$2,070,290	$0	$37,539,109
G&A	$58,010	$146,691	$372,718	$437,877	$437,871	$438,145	$437,651	$437,963	$437,793	$183,671	$0	$3,388,391
Mine Operations Labor	$252,983	$885,442	$1,517,900	$1,897,375	$1,897,375	$1,897,375	$1,897,375	$1,897,375	$1,644,392	$758,950	$0	$14,546,541
Mine Maintenance Labor	$242,054	$847,191	$1,452,327	$1,815,408	$1,815,408	$1,815,408	$1,815,408	$1,815,408	$1,573,354	$726,163	$0	$13,918,131
Mill Labor	$246,735	$2,097,251	$2,097,251	$2,097,251	$2,097,251	$2,097,251	$2,097,251	$2,097,251	$2,097,251	$838,900	$0	$17,863,645
Salary Labor	$1,236,924	$1,523,060	$1,523,060	$1,523,060	$1,523,060	$1,523,060	$1,523,060	$1,523,060	$1,523,060	$609,224	$0	$14,030,628
Total Operating Costs	$2,501,733	$12,023,474	$23,539,259	$27,244,826	$27,244,551	$27,256,974	$27,234,531	$27,248,703	$26,745,962	$11,285,373	$0	$212,325,386
Unit Cash Cost $/oz	*$0*	*$529*	*$429*	*$486*	*$462*	*$551*	*$635*	*$584*	*$567*	*$533*	*$0*	*$531*
Before Tax Cash Flow	($2,601,285)	$17,352,447	$47,412,186	$45,245,438	$49,098,163	$36,734,868	$28,281,550	$33,115,829	$34,253,466	$16,089,760	$0	$304,982,423
Depreciation	$6,323,446	$6,896,840	$7,405,243	$8,469,484	$9,173,676	$9,450,067	$10,189,553	$10,694,419	$10,866,275	$32,306,462	$0	$111,775,466
Loss Carry Forward - max 30% of BTCF	$0	($5,205,734)	$0	$0	$0	$0	$0	$0	$0	$0	$0	($5,205,734)
Tax	$0	$800,606	$6,101,059	$5,608,333	$6,088,484	$4,160,932	$2,759,030	$3,419,265	$3,566,547	$257,783	$0	$32,762,039
After Tax Cash Flow	($2,601,285)	$16,551,842	$41,311,127	$39,637,105	$43,009,679	$32,573,936	$25,522,520	$29,696,564	$30,686,919	$15,831,977	$0	$272,220,384

Coringa Underground Mine - Economic Model												
Summary By Year	**Year -1**	**Year 1**	**Year 2**	**Year 3**	**Year 4**	**Year 5**	**Year 6**	**Year 7**	**Year 8**	**Year 9**	**Year 10**	
	2015	**2016**	**2017**	**2018**	**2019**	**2020**	**2021**	**2022**	**2023**	**2024**	**2025**	**Total**
Capital Costs												
Permitting	$700,000	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$700,000
Engineering Studies & Site Investigations	$3,240,000	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$3,240,000
Mine Development	$1,893,612	$2,752,308	$2,126,412	$3,513,015	$4,092,139	$2,763,904	$4,350,468	$3,715,596	$1,718,565	$7,715	$0	$26,933,735
Closure	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$3,105,000	$3,105,000
Mine Equipment	$11,625,086	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$11,625,086
Surface Facilities	$46,399,867	$0	$0	$1,965,384	$0	$0	$0	$0	$0	$0	$0	$48,365,251
Replacement/Sustaining Capital	$0	$3,057,433	$3,032,809	$4,937,359	$3,024,778	$0	$3,121,796	$1,366,949	$0	$0	$0	$18,541,124
Working Capital	$600,000	$2,200,000	$3,200,000	$1,400,000	$0	$0	$0	$0	$0	($4,839,733)	$0	$2,560,267
Total Capital Cost	$64,458,564	$8,009,741	$8,359,221	$11,815,758	$7,116,917	$2,763,904	$7,472,264	$5,082,545	$1,718,565	($4,832,018)	$3,105,000	$115,070,462
Net Cash Flow	($67,059,849)	$8,542,101	$32,951,906	$27,821,348	$35,892,762	$29,810,032	$18,050,257	$24,614,019	$28,968,354	$20,663,994	($3,105,000)	$157,149,922
Full Cash Cost $/oz												$ 819

Table 22.2 Average Operating Costs

Average cost	$US
Operating cost per tonne mined	$106.35
Total cost per tonne mined	$163.99
Operating cost per ounce gold recovered	$530.94
Total cost per ounce gold recovered	$818.68

Table 22.3 Economic Model Results

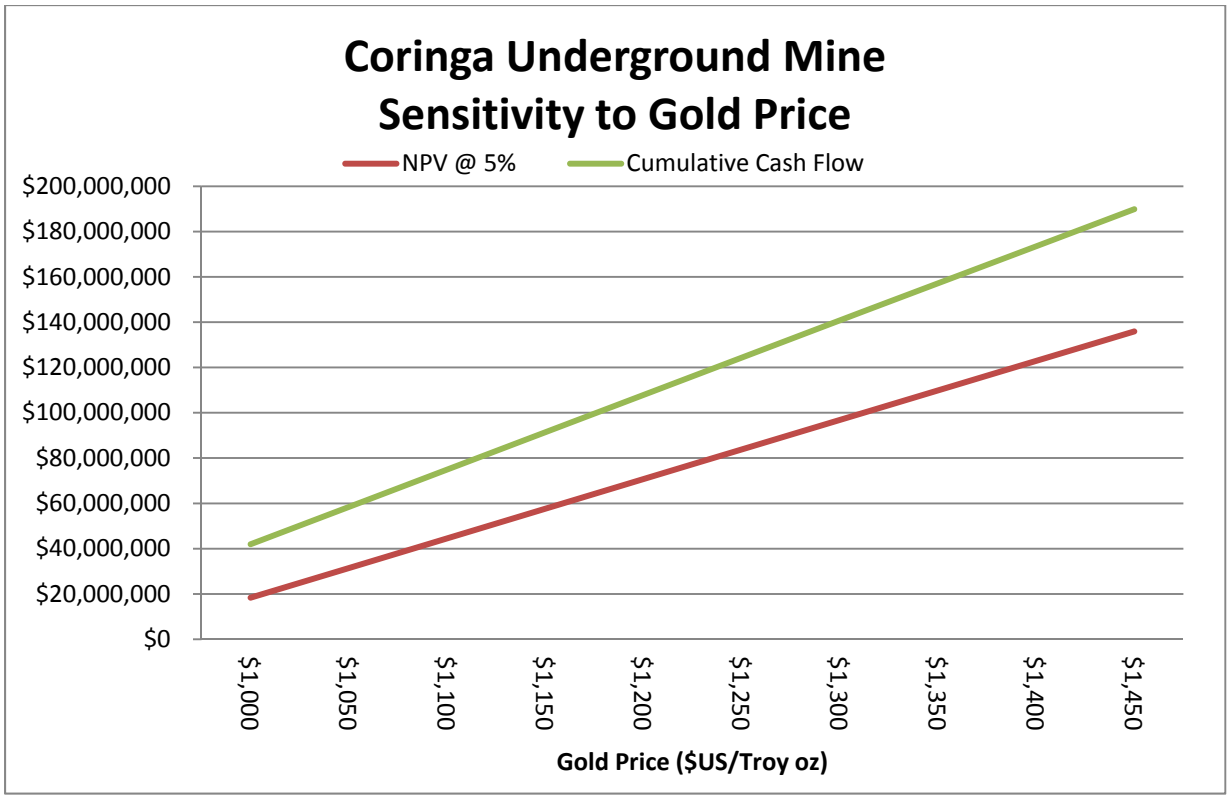
Result Category	$US 1,000s
(Cumulative Cash Flow) NPV@0%	$157, 150
NPV@5%	$109,870
IRR	33%

This preliminary assessment is preliminary in nature. It includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves. There is no certainty that the results of the preliminary economic assessment will be realized. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

22.3 Economic Model Sensitivity

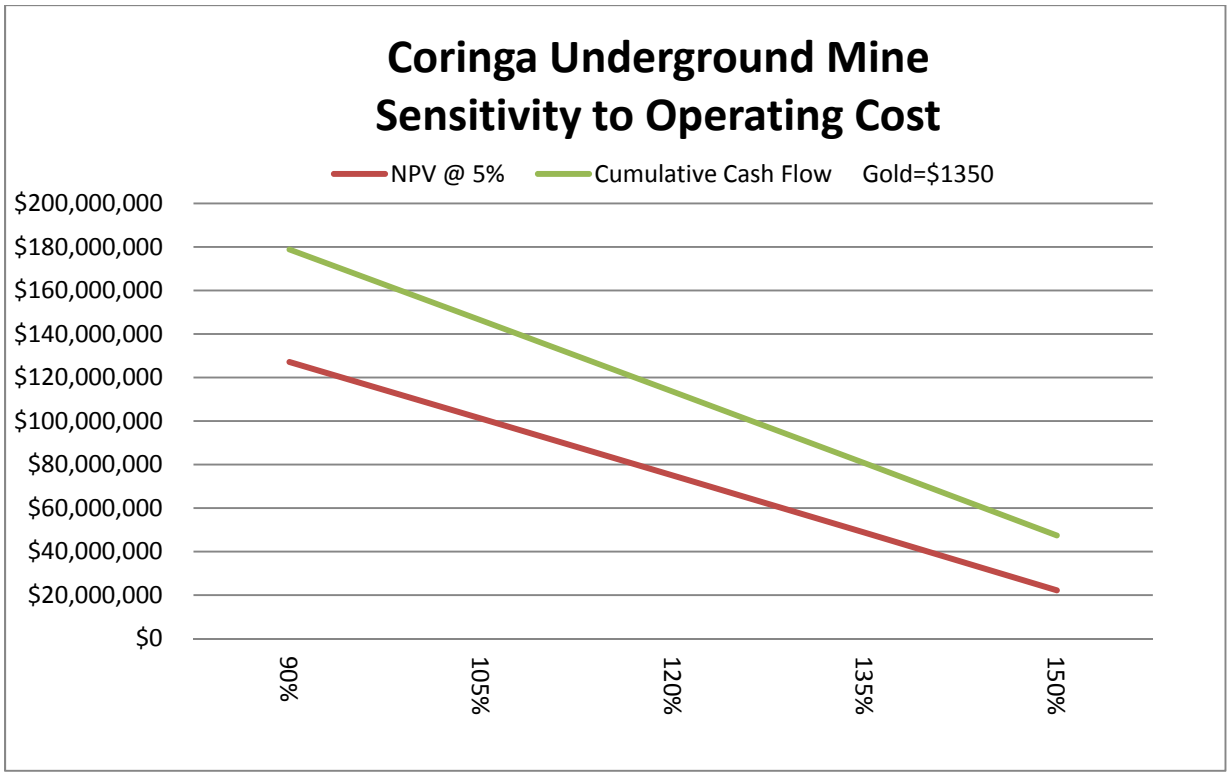
Three separate sensitivity analyses were completed for variations in gold price, capital costs, and operating costs. The model is most sensitive to gold price and least sensitive to capital costs. Figures depicting the model results by varying each of the above parameters are shown below. The project is very robust and is still economic under any one of the following scenarios: either a gold price of $US 1,000/troy oz., a capital cost increase of 50%, or an operating cost increase of 50%.

Figure 22.1 Sensitivity to Gold Price



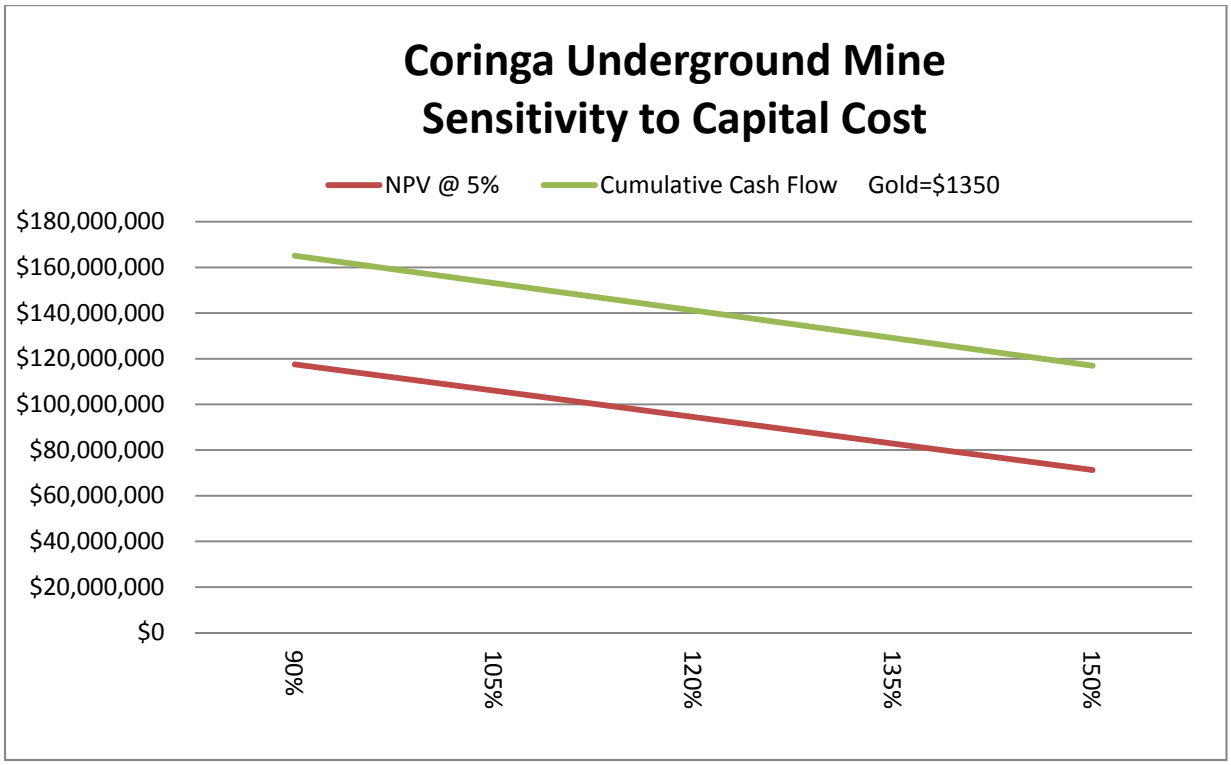
The above figure shows that project economics are most sensitive to gold price. Cumulative cash flow and NPV at a 5% discount rate vary from a low of $US 41.9M and $US 18.4M at a gold price of $US 1,000/oz. to $US 189.9M and $US 135.9M at a gold price of $US 1,450/oz. respectively. At the current gold price of $US 1,720/ oz. project economics increase to an NPV at 5% of $US 206.1 M.

Figure 22.2 Sensitivity to Operating Costs



The above figure demonstrates the model sensitivity to operating costs at a fixed gold price of $US 1,350/oz. Cumulative cash flow and NPV at a 5% discount rate vary from a low of $US 47.5M and $US 22.5M at 150% of the planned operating costs to a high of $US 178.8M and $US 127.2M at 90% of the planned operating costs respectively. This shows the project is very robust and still generates a good return (IRR of 11%) at 150% of the planned operating costs.

Figure 22.3 Sensitivity to Capital Costs



The above figure demonstrates the model sensitivity to capital costs at a fixed gold price of $US 1,350/oz. Cumulative cash flow and NPV at a 5% discount rate vary from a low of $US 116.0M and $US 71.2M at 150% of the planned capital costs to a high of $US 165.1M and $US 117.5M at 90% of the planned capital costs respectively. Again, this illustrates the resilience of the project as it still generates a very good return (IRR of 19%) at 150% of the planned operating costs.

22.4 Economic Model Optimization

Quantitative model optimization was completed for 2 separate issues: increasing the ramp grades and increasing the cutoff grade.

Two additional ramps grades were analyzed: 12% and 15%. Both will decrease the amount of development required during preproduction as well as during the mine life. The table below illustrates the results of a high level analysis of changing the ramp grades.

Table 22.4 Model Optimization – Ramp Grades

Ramp Grade	(base case) 10%	12%	15%
(Cumulative Cash Flow) NPV@0%, $US 1,000s	$157, 150	$158,810	$160,340
NPV@5%, $US 1,000s	$109,870	$111,240	$112,470
IRR	33%	33%	33%

As shown in the above table, increasing the ramp grades has a minimal impact on project economics.

An increase of the cutoff grade from 4 gpt Au to 5 gpt Au was also evaluated. Stope tonnages and grades were recalculated using the higher cutoff value and then re-sequenced over the mine life. Production rates and ramp up period were kept the same. Increasing the cutoff grade to 4 gpt reduces the mined resource from 2,000,000 tonnes at 6.66 gpt Au to 1,400,000 tonnes at 7.51 gpt Au. Project life is also reduced from 8.5 years to 6.1 years. The table below summarizes the expected economic changes of increasing the cutoff grade to 5 gpt Au.

Table 22.5 Model Optimization – Cutoff Grades

Cutoff Grade gpt Au	(base case) 4 gpt	5 gpt
(Cumulative Cash Flow) NPV@0%, $US 1,000s	$157, 150	$114,640
NPV@5%, $US 1,000s	$109,870	$83,232
IRR	33%	32%

As shown in the above table, increasing the cutoff grade has a small negative impact to the IRR. This is due to a minimal increase in grade which does not provide enough revenue to offset the accelerated development schedule and reduction in mine life.

23.0 ADJACENT PROPERTIES

The Coringa project is located in the state of Para in Brazil. The area has been the focal point for both alluvial and primary gold mining during the 1980's and 1990's and exploration claims cover large parts of the area. Many of the exploration claims surrounding the Magellan project area are held by Tamin Mineração and Magellan has first right of refusal on these blocks.

Figure 6.2, in **Section 6**, shows the Coringa property held by Magellan and the adjoining exploration claims.

24.0 OTHER RELEVANT DATA AND INFORMATION

Between March 2007 and 2012 Magellan carried out a stream sediment and soil sampling program. Lines with one km spacing were laid out across the project boundary, oriented NE-SW, for the stream sediment sampling program. In places where the stream samples contained significant free gold, the drainage was sampled upstream to locate the source. A total of 756 samples were collected.

Samples which had over 24 gold colour (90 Percentile), were considered anomalous. Those that had over 9 ppm in Au (90 percentile), were also considered anomalous **(See Figure 24.1 and 24.2).**

Based on those anomalies, a soil sampling program was carried out (**See Figure 9.1**). Soil samples over 41 ppb (90 percentile) are considered anomalous.

Figure 24.1 Stream Sediment Survey

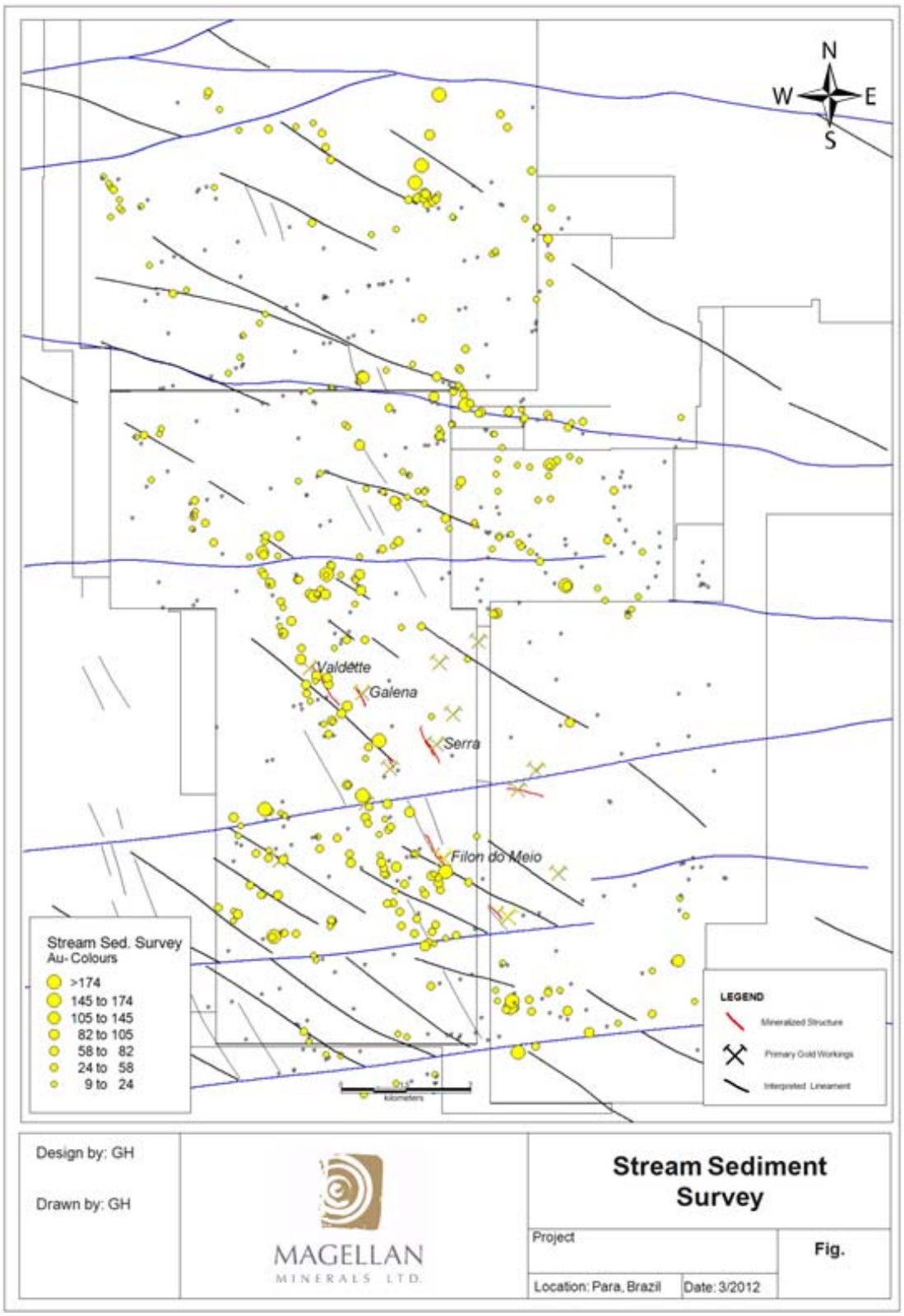
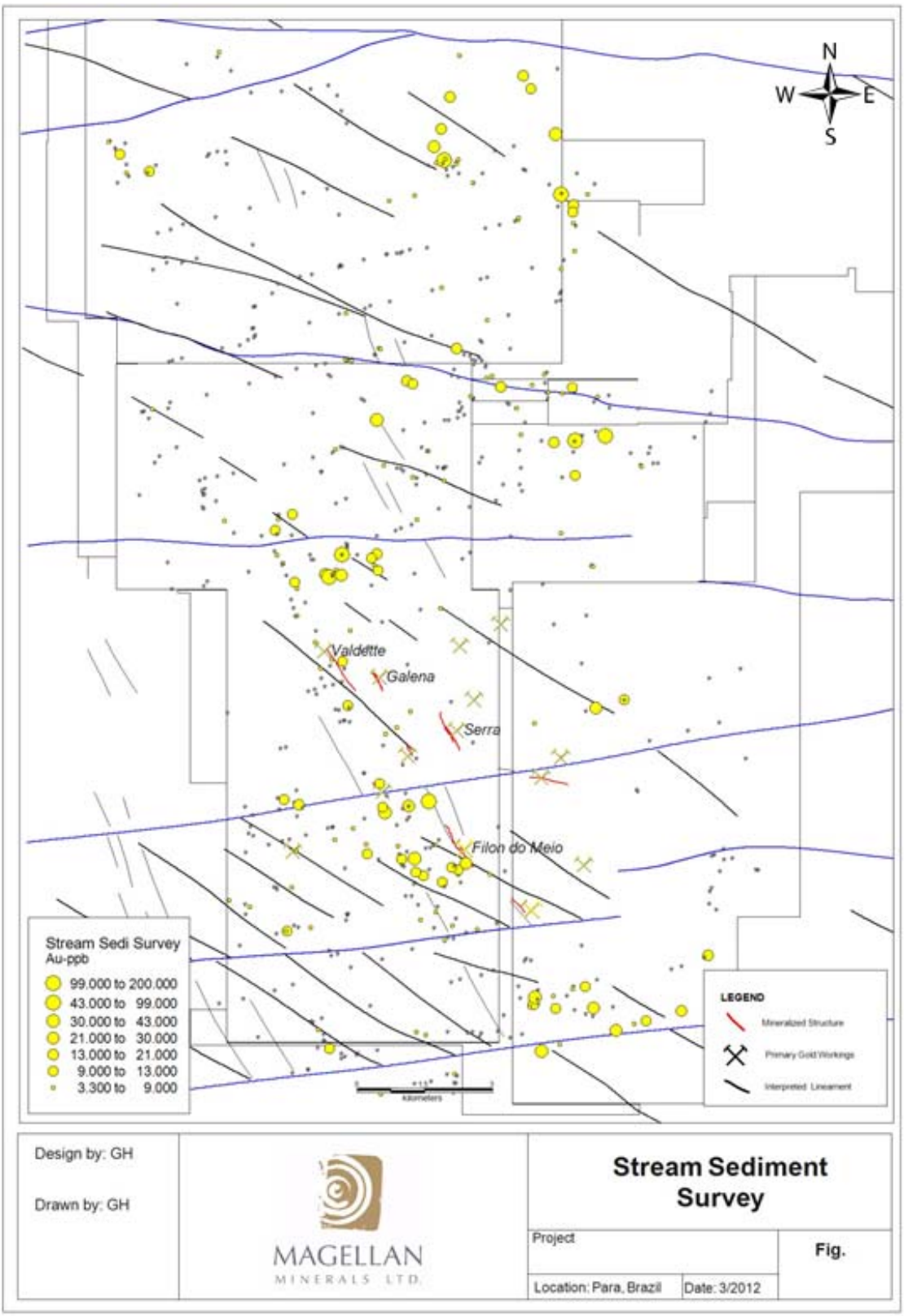


Figure 24.2 Stream Sediment Survey



25.0 INTERPRETATION AND CONCLUSIONS

The Coringa Project is a shear-vein system, hosted in volcanic rocks from the Salustiano Formation (1.87 Ga) and in granites from the Maloquinha Suite (1.86 to 1.88 Ga) located in the Tapajos Gold Province. This Province was the site of a major gold rush by artisanal miners from the late 1970's until the late 1990's and had a historical production of at least 10 million ounces with actual production believed to be closer to 30 million ounces. Some 10 tonnes of gold has reportedly been produced from alluvial and primary sources at the Coringa property (Brazilian Department of Mineral Production - DNPM).

The Coringa Project covers an area of approximately 9 km by 11 km (9,900 ha). Diamond drilling has been completed on less than 20% (4,800 ha) of the total exploration permits and claims (23,754 ha) held by Magellan at Coringa. The main shear zone extends for at least 7 km, and has an average strike of 345°. Sub-parallel and parallel shears add at least another 4.1 km to the strike length. The mineralized high grade vein structures contained within the shear zone vary in thickness from 15 cm to 4.0 m. A low grade mineralized shear zone has been identified at Valdette with an average width of 50 to 80 meters. The Coringa shear-vein system is hosted by both granites and rhyolites. Pyrite is the most common sulphide, followed by sphalerite, galena, chalcopyrite, and smaller amounts of covellite. Native gold occurs in particles, 5 - 220µ in size, mostly included within pyrite, or at the junction between pyrite and sphalerite, galena or chalcopyrite. The alteration is mainly sericitization (as wide as 0.5 m in granites, and 1 m in rhyolites), with some silicification and chloritization-hemtatization and generally occurs as haloes up to 10 meters in granites and 50 meters in rhyolites.

GRE concludes the following:

- A total of 24,093 m of HQ core in 160 exploration holes has been drilled on the Coringa Project since 2007. The density of the exploration drilling is adequate to estimate measured and indicated resources between most drill holes. The spacing between drill holes was increased during the 2011 drill program and focused on identifying additional inferred resources. This expanded spacing should be continued for further exploration along the vein system.
- The potential exists to expand resources through additional drilling along the strike and dip of the defined resource areas.
- The geostatistical analysis completed on the Serra resource block indicates a variogram range of 100 m. This range was used to define measured and indicated resources. A 75 m search distance and minimum 2 assay intercept limit was used to define measured resources. A 100 m search distance and minimum 1 assay intercept was used to define indicated resources. Infill drilling with the purpose of upgrading inferred resources should take into account these resource categorization parameters.
- The vein system is open both at depth and along strike. The Galena-Mae de Leite and Meio-Come Quieto resource blocks demonstrate that the shear vein zone is more or less continuous between blocks. It is probable that the shear system also continues along strike beyond the

current extent of the resource blocks. There is also good potential to encounter additional parallel vein structures within the exploration area held by Magellan as demonstrated by the Serra and Demetrio resource blocks.

- A resource model to the standards of Canadian National Instrument NI-43-101 was completed for the Coringa Project defining an undiluted gold resource of 561,000 measured and indicated troy ounces of gold and 534,000 inferred troy ounces of gold at a 1 gpt cutoff grade capped at 50 gpt gold. The additional drilling completed during the fourth drilling program increased the previous resource by 108% in the Measured and Indicated Category and 444% in the Inferred category.
- The sample QA/QC program completed by Magellan produced representative samples of the deposit. Check assays completed by GRE on the onsite core show acceptable correlation with previous results. The QA/QC program meets industry standards with the insertion of 1 blank, 2 standards, and 0 to 3 duplicates every 42 samples with the exception of creating field duplicates. The blank samples consist of non-gold bearing granite sourced from a local quarry borrow and verify that sample contamination due to sample preparation did not occur.
- Creating duplicates onsite at a large particle size creates a bias in the samples and therefore does not meet the purpose of duplicate sampling. Field duplicates should be created by quartering the core along the principal structure to create the best duplicate sample. Laboratory duplicates prepared at a small particle size show excellent correlation for duplicate assay samples.
- A conventional carbon in leach (CIL) milling circuit with pre-aeration at a pH of 11, using hydrated lime, is an effective method for treating the Coringa ores. However due to the presence of silver and overall high grade of combined gold and silver, a Merrill Crowe circuit may be more economical. Recent metallurgic test work (RDi, March 2010) performed on composite samples from the Serra block and from the Meio block produced laboratory gold recoveries of 99.0% and 97.7% respectively using the CIL approach.
- The PEA process flowsheet considered a Merrill Crowe circuit with three stages of crushing that will feed a ball mill to produce a P_{80} 200 Mesh product. Prior to leaching, the pH would be adjusted with lime followed by aeration to reduce reagent consumption and improve recoveries. The aerated slurry would be leached with cyanide, the supernatant clarified and de-aerated, followed by zinc precipitation of the precious metals. The tailings stream would be detoxified to produce mine backfill.
- During the resource estimation, GRE noted that the data collected by Magellan was well organized and accurate and that the exploration camp was well run and has good facilities for continued exploration. These observations reflect the high quality of Magellan's exploration program. In addition, internal audits completed by Magellan on their sample chain of custody documents, illustrate their commitment to running a well-organized and comprehensive exploration program.

- Preliminary geochemical test work to evaluate acid rock drainage (ARD) potential indicates that a single rock type (altered vein deposits with quartz, pyrite, and galena) appears to have the highest ARD risk. In addition, the tailings from the processed ore will likely be PAG material since the defined process flow diagram does not include oxidation of the sulfide minerals. However, it is also evident that some samples have acid neutralizing potential. As a result, it appears that the Coringa project may have ARD potential, but also the ability to control ARD with the acid neutralizing potential inherent in some of the waste rock. Additional studies are required to substantiate this hypothesis.
- An underground mine would be the most efficient and economical method for mineral extraction for the majority of the resource blocks since it would provide a means of selectively mine the higher grade vein zones. Additionally, an underground mine would minimize surface impacts. The Valdette resource block is significantly lower grade with more disseminated mineralization. A surface mine would be more applicable to the mineralization type and grade present in Valdette.
- The overhand cut and fill mining method was selected for the PEA study due to its prominence as a preferred mining method, the ability to provide a high degree of grade control, the lower preproduction development requirements, and the fact that it is more applicable to the untested ground conditions at Coringa. The method also allows the backfilling of stopes throughout the mine life with a tailings slurry, reducing external tailings facility storage requirements. The backfilled stopes have the potential to reduce ARD potential relative to other mining methods that leave open or caved stopes.
- Mine production was scheduled at 750 tpd, allowing for ramp-up to full production during the first two years. Rubber-tired (LHD and truck) haulage was selected based on its predominance in the industry at present and to provide a conservative estimation of project economics. Mine access would be via ramp systems daylighting at valley portals for each resource area.
- A portion of the process tailings would be used as backfill underground with the remainder being placed within a thickened tailings impoundment. The tailings impoundment includes an earth fill embankment allowing for construction without the need for mine development rock. The feasibility of the unlined impoundment assumed for the PEA study must be demonstrated by field investigation and additional analyses.
- Project initial capital costs for the plant complex, mine equipment, surface and underground facilities, preproduction development, permitting, additional engineering studies, site investigations, and working capital are estimated at US$ 64.5M. Sustaining capital including additional mine equipment, required rebuilds, and new portal facilities totals $US 18.5M. Concurrent development requirements total $US 26.9M over the mine life. Closures costs are estimated at $US 3.1M. Total project capital is estimated at $US 115.1M.
- The project workforce was estimated separately for mine operations, mine maintenance, mill operations, and salary personnel. Labour rates reflect a recent salary survey from an ISO

9001:2008 Brazilian company. A total workforce of 309 hourly employees and 22 salary employees is estimated for the mine and mill operations during full operations.

- Operating costs are estimated at $US 106 per tonne mined ($US 531 per ounce gold recovered). Controllable operating costs were estimated for each mining operation based on the materials and equipment required to complete each unit operation such as stope production, crosscut development, surface ore haulage, etc. Non-controllable operating costs included taxes, royalties, permit fees, and G&A expenses.
- Total costs (including capital and operating) are estimated at $US 164 per tonne mined ($US 819 per ounce of gold recovered).
- The preliminary economic model includes a 21% contingency on capital costs and a 15% contingency on operating costs.
- The preliminary economic model indicates a robust IRR of 33% and a NPV at a 5% discount rate of $US 109.9M. The project payback period is 2.9 years. This preliminary assessment is preliminary in nature. It includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves. There is no certainty that the results of the preliminary economic assessment will be realized. Mineral resources that are not mineral reserves do not have demonstrated economic viability.
- Three separate sensitivity analyses were completed on the economic model. The project is very robust and is still economic under any one of the following scenarios: either a gold price of $US 1,000/troy oz., a capital cost increase of 50%, or an operating cost increase of 50%.

26.0 RECOMMENDATIONS

The updated preliminary economic assessment (PEA) in November 2012 indicates a robust IRR of 33% and a NPV at a 5% discount rate of US$ 109.9M. The project payback period is 2.9 years. The PEA indicates positive economics under any one of the following scenarios: either a gold price of $US 1,000/troy oz., a capital cost increase of 50%, or an operating cost increase of 50%. This PEA is preliminary in nature. It includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves. There is no certainty that the results of the preliminary economic assessment will be realized. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

The updated PEA confirmed the positive economics from the previous study and demonstrated the expansion potential of the project by incorporating additional defined resources along strike and dip from the forth drilling campaign. The resource continues to be open along strike and at depth. The current PEA considers the updated resources defined in May 2012 using only the inferred resources that are planned to be upgraded through infill drilling. In light of the positive results of the updated PEA, Magellan should advance the project, initially completing site investigations to gather data required for additional studies. The following sections list the recommendations for data acquisition and completion of further engineering studies along with the associated cost for each.

Data Acquisition– Total Cost $US 2,240,000

The following items are recommended as part of the data acquisition process. They are listed below along with their estimated cost.

- Resource definition drilling to upgrade targeted sections from inferred resources to measured and indicated resources.
    - Estimated Cost $US 600,000
- Update the resource model with the results of the resource definition drilling. The geostatistical analysis used in the resource categorization should be used to define the appropriate drill hole spacing.
    - Estimated Cost $US 50,000
- Condemnation drilling in the proposed areas for the tailings storage facility, plant site complex, and camp area.
    - Estimated Cost $US 200,000
- Complete a topographic update of the property with satellite photographic topography that would tie into the existing LIDAR topography. The areas for the Valdette, Come Quieto, and the access route to the project should be surveyed.
    - Estimated Cost $US 15,000.

- Additional metallurgical testing should be completed to increase the sample density in relation to spatial, mineralogical, and grade distribution of the ore zones as well as test the new areas of the resource. Additional work is also required to define the optimum recovery method as both carbon leaching and Merrill Crowe processes are applicable to the project. Perform the additional test work to determine the items listed below:
    - Bond Work Index
    - Abrasion index
    - Testing of high-grade and low-grade ores to confirm recoveries similar to the composite testing
    - Confirm the samples tested
    - Grind size optimization
    - Define pre-aeration conditions
    - Leach conditions, time and reagent concentrations
    - Complete filtration and settling tests
    - Tailings detoxification for placement as underground fill during the mining cycle
        - Estimated Cost for all above metallurgical testing $US 200,000.
- Perform hydrogeological testing to determine the availability of well water as a source for plant makeup water and other freshwater project needs. Two streams within a few kilometers of the proposed plant site appear to be viable alternative sources of freshwater. However, potential seasonal stream flow variations, permitting requirements, the complexity of pioneering access through the jungle, power supply issues, and the probable need to clarify and/or treat the water prior to use indicate groundwater supply wells are the preferred alternative. Establishing the feasibility of groundwater well supply could be critical to the project timeline because developing the surface water alternatives may be a lengthy process.
    - Estimated Cost $US 150,000
- Perform an assessment of the project area groundwater compliance points and install monitoring wells.
    - Estimate Cost $US 175,000
- Perform geochemical, rheological, and physical property testing on remaining sample material from the April 2010 metallurgical testing. The current assumptions for the tailings properties are inferred from the geology, from testing performed on waste rock samples, and from industry experience. Characterization of the tailings to be impounded is critical for detailed design of the tailings facility, including establishing the feasibility of an unlined impoundment. The testing will also provide a better understanding of potential water treatment requirements. A comprehensive program to define the geochemical properties of the waste material should be

completed which includes expanding the current testing already completed and completing onsite kinetic testing.

  - Estimated Cost $US 150,000

- Complete a mine dewatering study using the planned infill drill holes for upgrading targeted resource areas from inferred to measured and indicated. This study will evaluate the characteristics of the hanging wall and footwall to determine mine dewatering requirements.
  - Estimated Cost $US 100,000
- Completing a structural geologic mapping of the tailings facility to determine preferential fault, joint, and bedding planes that should be targeted during the site investigation programs.
  - Estimated Cost $US 25,000
- Initiate tailings facility geotechnical and hydrogeology site investigation programs. This is critical to establish the feasibility of an unlined tailings impoundment. Permitting of the tailings facility is expected to be lengthy process, and as such it is recommended that this work commences as soon as possible to ensure the overall project timeline is met.
  - Estimated Cost $US 300,000
- Complete a borrow materials characterization to determine the suitability and quantity of construction materials for the tailings storage facility.
  - Estimated Cost $US 100,000
- Complete a plant site geotechnical investigation to verify the suitability of the proposed plant site and determine the foundation requirements.
  - Estimated Cost $US 75,000
- Implement an environmental baseline data collection program which will include site climate data, groundwater quality, and surface water quantity and quality. The data from the program will be important for permitting and is necessary to establish the pre-development site conditions.
  - Estimated Cost $US 50,000
- Perform bathymetric surveys of the historic garimpos to support detailed mine design and dewatering studies.
  - Estimated Cost $US 50,000

Additional Engineering Studies - $US 1,000,000

The following recommendations are the engineering analysis items for further study. The total cost estimate for all studies is $US 1,000,000.

Recommendations are:

- Update the site climate data and analyze the surface water flow measurements to determine the site storm parameters and surface water base flow data.
- Refine all capital cost estimates by sourcing hard vendor quotes with all duties, taxes, and transportation costs to the site included. Complete new estimates for capital items that were previously estimated as lump sums by estimating each component separately. Add detail estimates for engineering procurement and construction management (EPCM) for all fixed capital.
- Refine all operating costs by adding greater detail to material quantities, material costs, creating a reagent schedule for the process plant.
- Complete a geotechnical study on the existing core to estimate the mine support requirements.
- Complete a mass balance study for the process plant and detailed process plant design.
- Refine TSF discharge water treatment costs by completing a detailed (monthly) water balance for the tailings impoundment as well as creating a detailed treatment plant design.
- Perform a detailed power study to define the capital and operating costs of line power supply. Although capital costs for bringing national electrical grid service to the project will be high, the PEA indicates diesel electric generation will be prohibitively costly with the planned production rate. Contract negotiations with power companies will be required for a definitive evaluation of this alternative.
- Perform trade off studies to evaluate other mining method and haulage options which may include quantifying potential cost savings and determining the best option.
- Conduct an expanded geochemical engineering analysis of all major rock types, mineralization types, and tailings. This analysis will predict the tailings impoundment discharge water quality that is a critical input to the water treatment system design.
- Create a predictive model of the local groundwater system to determine mine dewatering requirements.
- Develop a 3D mine model to quantify all mining and development areas. Perform a cutoff grade sensitivity analysis to determine the optimal cutoff grade and mine configuration. Determine the process cutoff grade to be used for mineralized development rock that must be mined to provide access to the defined stopes.
- Optimize the layout of all facilities based on the new LIDAR topography.

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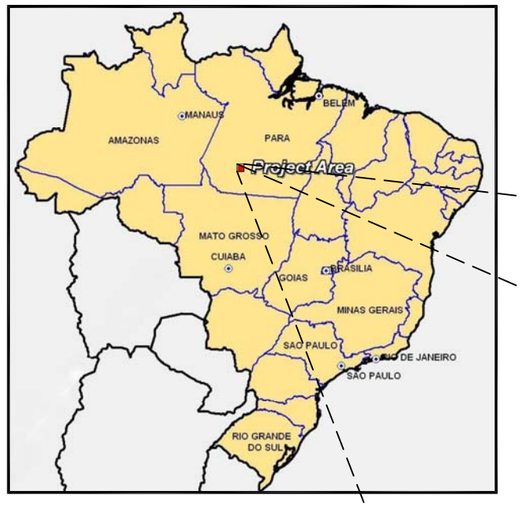
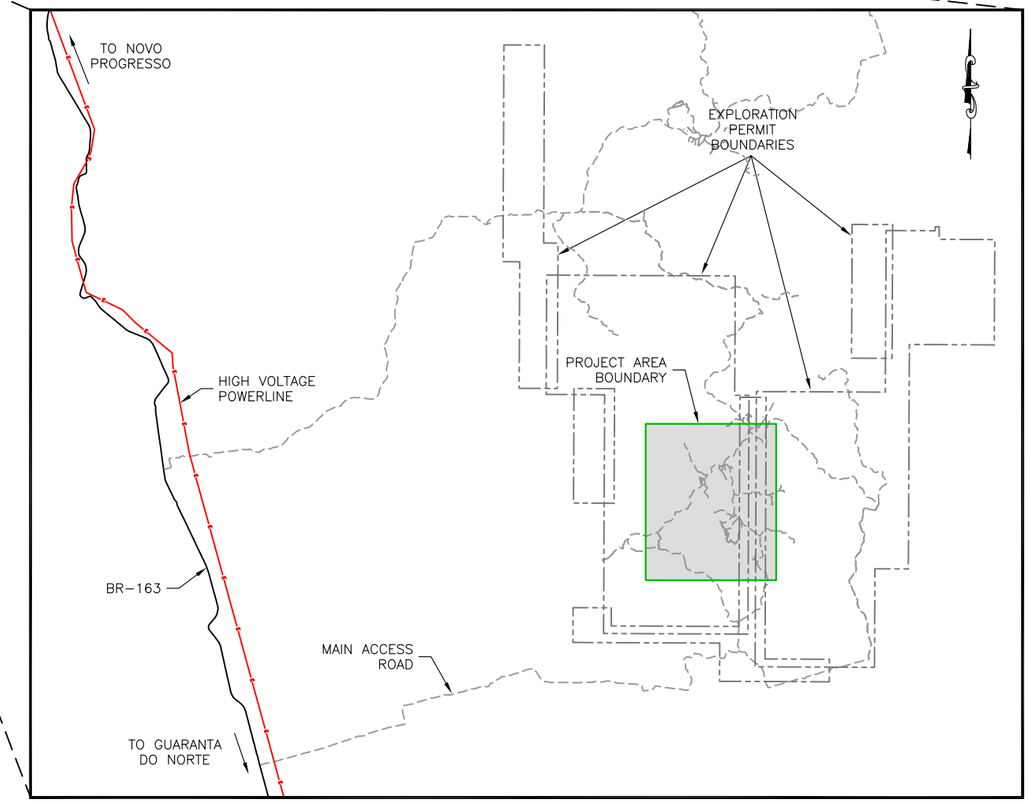
Annex 1

Drawings

NI 43-101 TECHNICAL REPORT
PRELIMINARY ECONOMIC ASSESMENT
CORINGA PROJECT
PARA, BRAZIL



BRAZIL
NOT TO SCALE



PROJECT LOCATION

0 2 4 6 8 10
SCALE IN KILOMETERS

LIST OF DRAWINGS:

01 - TITLE SHEET AND PROJECT LOCATION
02 - GENERAL FACILITIES ARRANGEMENT
03 - SERRA UNDERGROUND MINE LAYOUT PLAN
04 - SERRA FOOTWALL STOPES 4 gpt Au CUTOFF
05 - SERRA MIDDLE VEIN STOPES 4 gpt Au CUTOFF
06 - SERRA HANGING WALL STOPES 4 gpt Au CUTOFF
07 - MEIO-COME QUIETO UNDERGROUND MINE LAYOUT PLAN -1
08 - MEIO-COME QUIETO UNDERGROUND MINE LAYOUT PLAN -2
09 - MEIO-COME QUIETO STOPES 4 gpt Au CUTOFF
10 - GALENA MAE DE LEITE UNDERGROUND MINE LAYOUT PLAN
11 - GALENA MAE DE LEITE STOPES 4 gpt Au CUTOFF
12 - TYPICAL STOPE SECTIONS AND DETAILS
13 - TAILINGS IMPOUNDMENT PLAN
14 - TAILINGS DAM SECTIONS
15 - TAILINGS DAM DETAILS

REFERENCE	Rev	Description	By	Date
	B	ISSUED FOR REPORT	GRE	11/2012
	A	ISSUED FOR REVIEW	GRE	10/2012

REVISIONS

Designed by: GRE
Drawn by: DKS
Checked by: KJG
Approved by: KJG

Prepared by: GRE Global Resource Engineering

Prepared for: MAGELLAN MINERALS LTD.

Title: TITLE SHEET AND PROJECT LOCATION

B REVISION

Project: CORINGA | Project No.: 12.1032
Location: PARA, BRAZIL | Date: 11/2012
DRAWING 01

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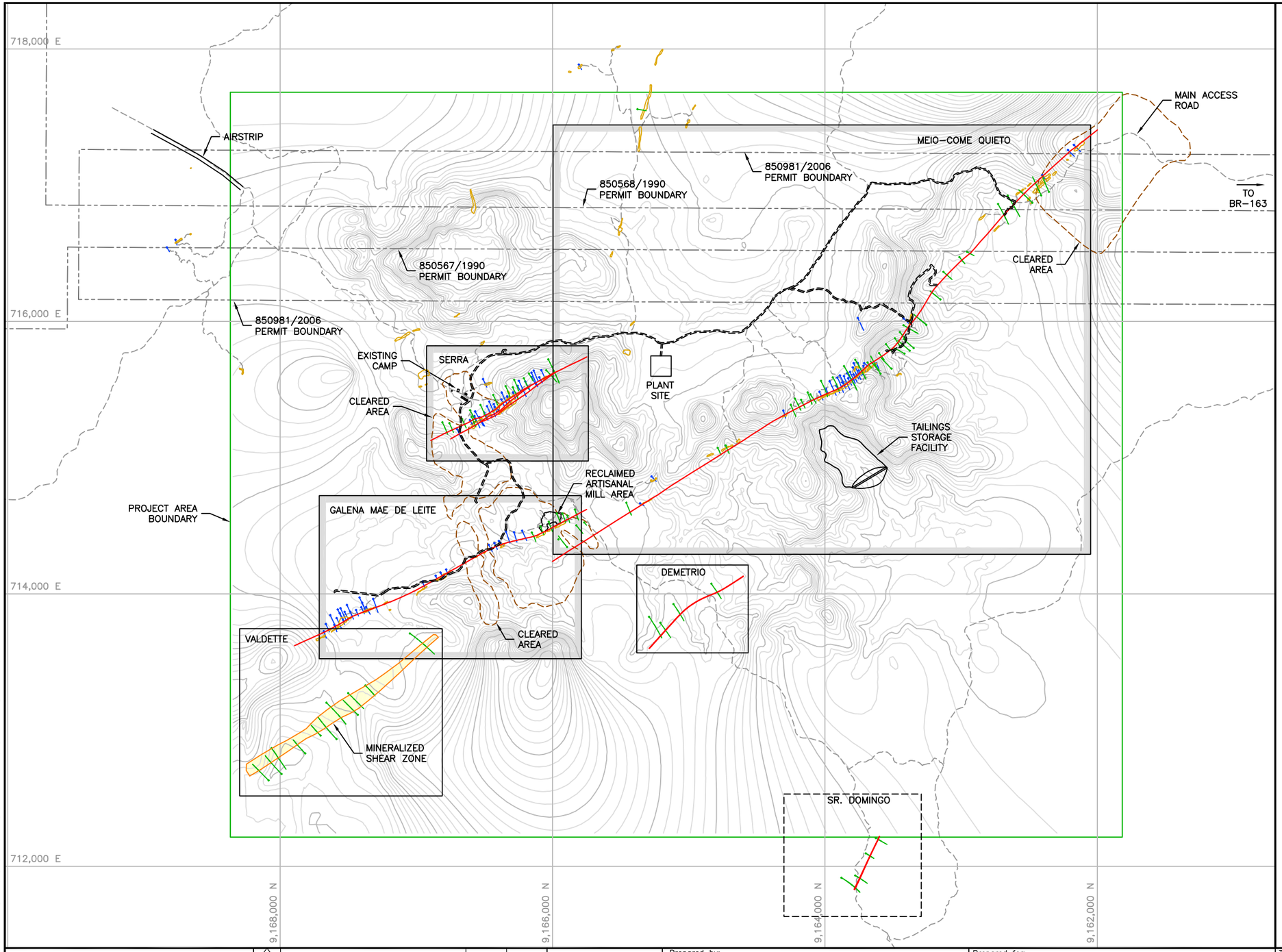

LEGEND

- MINED RESOURCE BLOCK
- ESTIMATED RESOURCE BLOCK
- POTENTIAL RESOURCE BLOCK
- CORE DRILLHOLE LOCATION AND PROJECTION (2007–2008)
- CORE DRILLHOLE LOCATION AND PROJECTION (2010–2011)
- MINERALIZED VEIN
- OLD ARTISANAL SURFACE WORKING (GARIMPO)
- EXISTING ACCESS ROAD
- UPGRADED/NEW ACCESS ROAD

NOTES:

1) EXISTING GROUND TOPOGRAPHY PROVIDED BY MAGELLAN MINERALS.

2) MINERALIZED VEINS SHOWN IN PLAN AT OUTCROP TO SURFACE.

N E S W

300 0 . 300 600 1200

SCALE IN METERS
CONTOUR INTERVAL: 5m

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Drawn by: DKS
Checked by: KJG
Approved by: KJG

Prepared by: GLOBAL RESOURCE ENGINEERING

Prepared for: MAGELLAN MINERALS LTD.

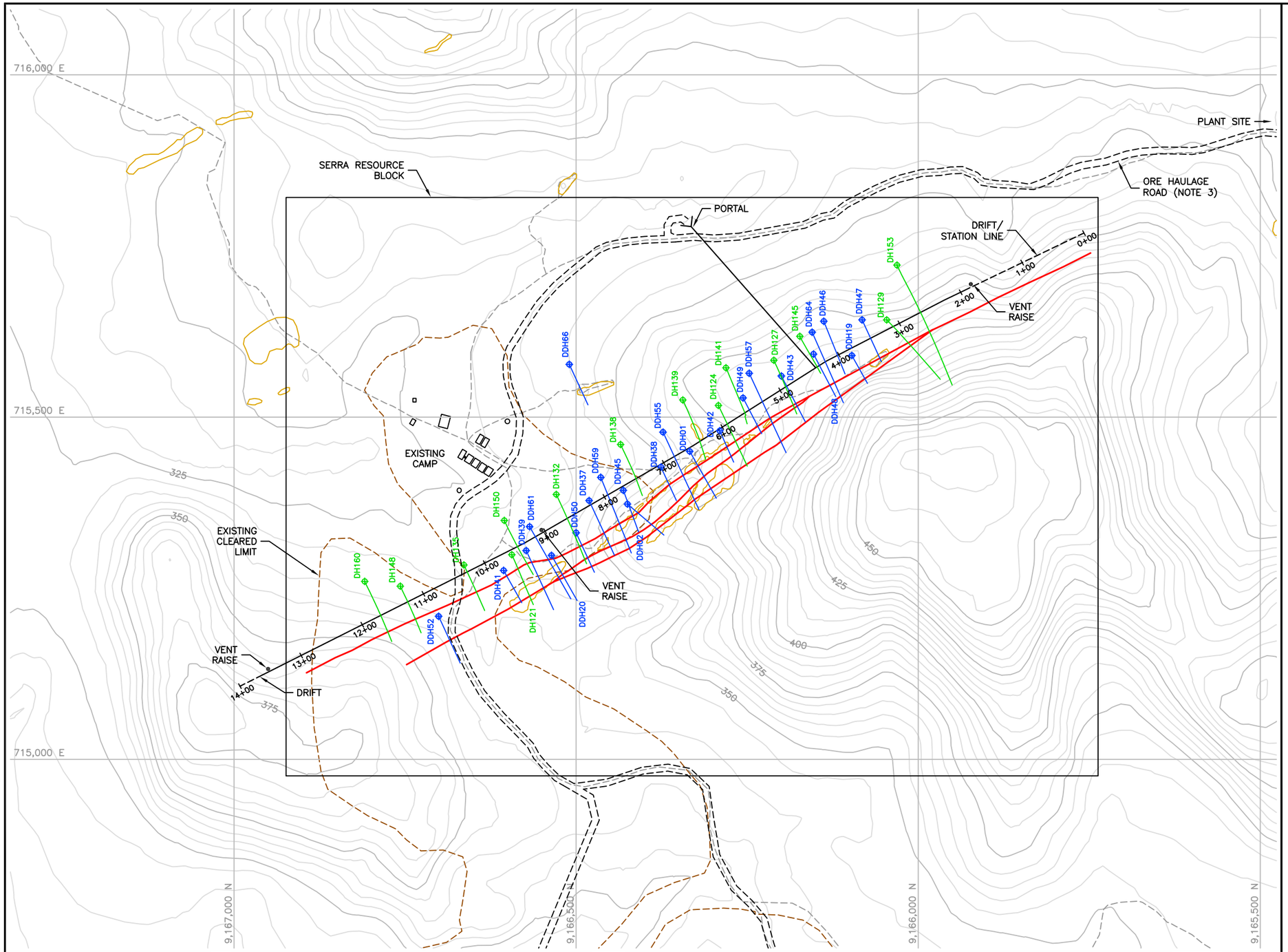
Title: GENERAL FACILITIES ARRANGEMENT

B REVISION

Project: CORINGA | Project No.: 12.1032

Location: PARA, BRAZIL | Date: 11/2012

DRAWING 02



LEGEND

- CORE DRILLHOLE LOCATION AND PROJECTION (2007–2008)
- CORE DRILLHOLE LOCATION AND PROJECTION (2010–2011)
- MINERALIZED VEIN
- OLD ARTISANAL SURFACE WORKING (GARIMPO)
- EXISTING ACCESS ROAD

NOTES:

1) EXISTING GROUND TOPOGRAPHY PROVIDED BY MAGELLAN.

2) MINERALIZED VEINS SHOWN IN PLAN AT OUTCROP TO SURFACE.

3) EXISTING ROAD TO BE IMPROVED FOR ORE HAULAGE AT 10% GRADE MAXIMUM.

N E S W

60 0 30 60 120 240

SCALE IN METERS
CONTOUR INTERVAL: 5m

Rev	Description	By	Date
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A	ISSUED FOR REVIEW	GRE	10/2012

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Designed by: GRE
Drawn by: DKS
Checked by: KJG
Approved by: KJG

Prepared by: GLOBAL RESOURCE ENGINEERING

Prepared for: MAGELLAN MINERALS LTD.

Title: SERRA UNDERGROUND MINE LAYOUT PLAN

B REVISION

Project: CORINGA
Project No.: 12.1032
Location: PARA, BRAZIL
Date: 11/2012
DRAWING 03

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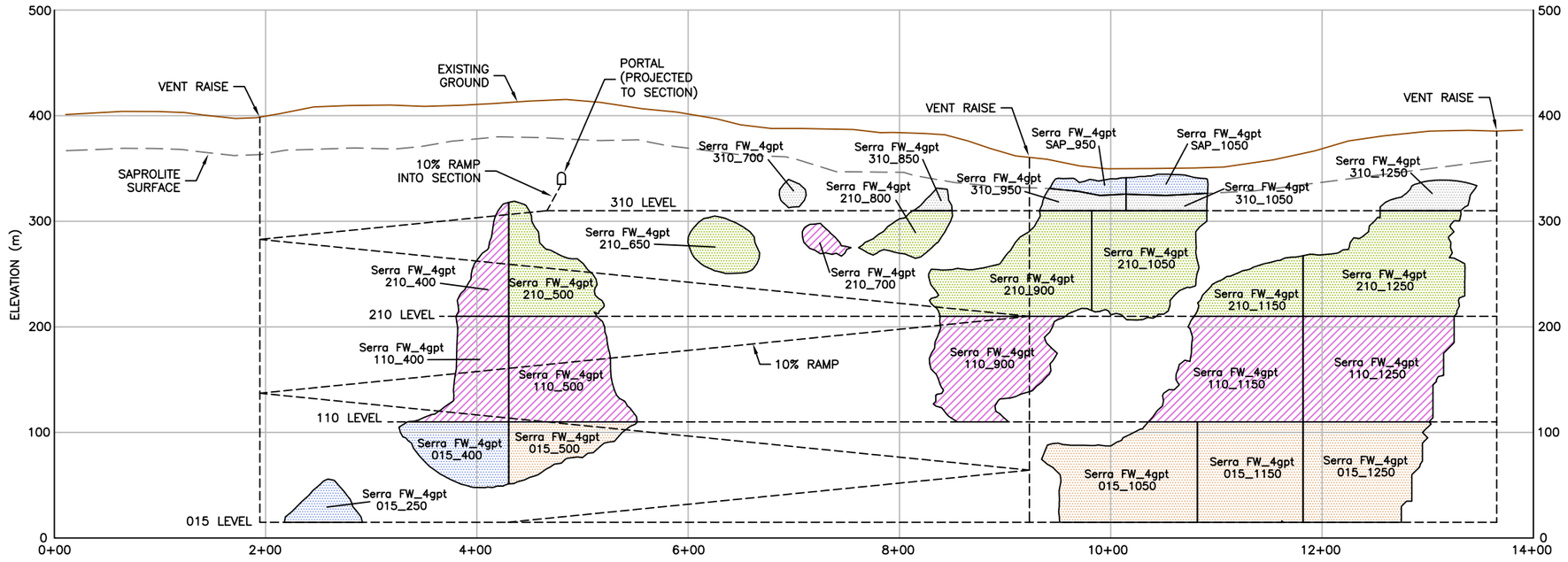

SERRA FOOTWALL SECTION

IN–SITU RESOURCES BY STOPE

Block	Stope	Cubic Meters	Tonnes	Grams Au	Grade g/tonne	Troy ounces
Serra FW_4gpt	015_1050	16,813	45,396	288,620	6.4	9,279
Serra FW_4gpt	015_1150	14,297	38,601	298,658	7.7	9,602
Serra FW_4gpt	015_1250	15,237	41,139	226,346	5.5	7,277
Serra FW_4gpt	015_250	2,707	7,309	32,048	4.4	1,030
Serra FW_4gpt	015_400	10,349	27,943	133,708	4.8	4,299
Serra FW_4gpt	015_500	10,819	29,210	159,582	5.5	5,131
Serra FW_4gpt	110_1150	17,729	47,868	425,625	8.9	13,684
Serra FW_4gpt	110_1250	19,730	53,271	396,777	7.4	12,757
Serra FW_4gpt	110_400	10,904	29,440	175,602	6.0	5,646
Serra FW_4gpt	110_500	26,992	72,880	428,572	5.9	13,779
Serra FW_4gpt	110_900	13,940	37,639	270,671	7.2	8,702
Serra FW_4gpt	210_1050	14,023	37,863	347,461	9.2	11,171
Serra FW_4gpt	210_1150	6,474	17,479	122,694	7.0	3,945
Serra FW_4gpt	210_1250	18,499	49,946	304,408	6.1	9,787
Serra FW_4gpt	210_400	4,736	12,786	68,880	5.4	2,215
Serra FW_4gpt	210_500	9,968	26,914	171,968	6.4	5,529
Serra FW_4gpt	210_650	4,133	11,159	62,901	5.6	2,022
Serra FW_4gpt	210_700	1,184	3,198	14,489	4.5	466
Serra FW_4gpt	210_800	3,479	9,394	53,235	5.7	1,712
Serra FW_4gpt	210_900	15,581	42,068	286,981	6.8	9,227
Serra FW_4gpt	310_1050	1,764	4,762	44,672	9.4	1,436
Serra FW_4gpt	310_1250	2,849	7,693	32,105	4.2	1,032
Serra FW_4gpt	310_700	730	1,970	8,443	4.3	271
Serra FW_4gpt	310_850	713	1,924	9,323	4.8	300
Serra FW_4gpt	310_950	2,056	5,550	31,756	5.7	1,021
Serra FW_4gpt	SAP_1050	1,942	5,245	42,234	8.1	1,358
Serra FW_4gpt	SAP_950	1,321	3,567	19,174	5.4	616
Serra FW_4gpt	Total	248,969	672,216	4,456,933	6.6	143,294

STOPE DESCRIPTION = LEVEL_SECTION LOCATION

YEAR STOPE PRODUCTION BEGINS

- 2016
- 2017
- 2018
- 2019
- 2020
- 2021
- 2022
- 2023
- 2024

Rev	Description	By	Date
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A	ISSUED FOR REVIEW	GRE	10/2012

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REVISIONS

Designed by: GRE
Drawn by: DKS
Checked by: KJG
Approved by: KJG

Prepared by: GRE GLOBAL RESOURCE ENGINEERING

Prepared for: MAGELLAN MINERALS LTD.

Title: SERRA FOOTWALL STOPES
4 gpt Au CUTOFF

REVISION: B

Project: CORINGA
Project No.: 12.1032
DRAWING: 04
Location: PARA, BRAZIL
Date: 11/2012

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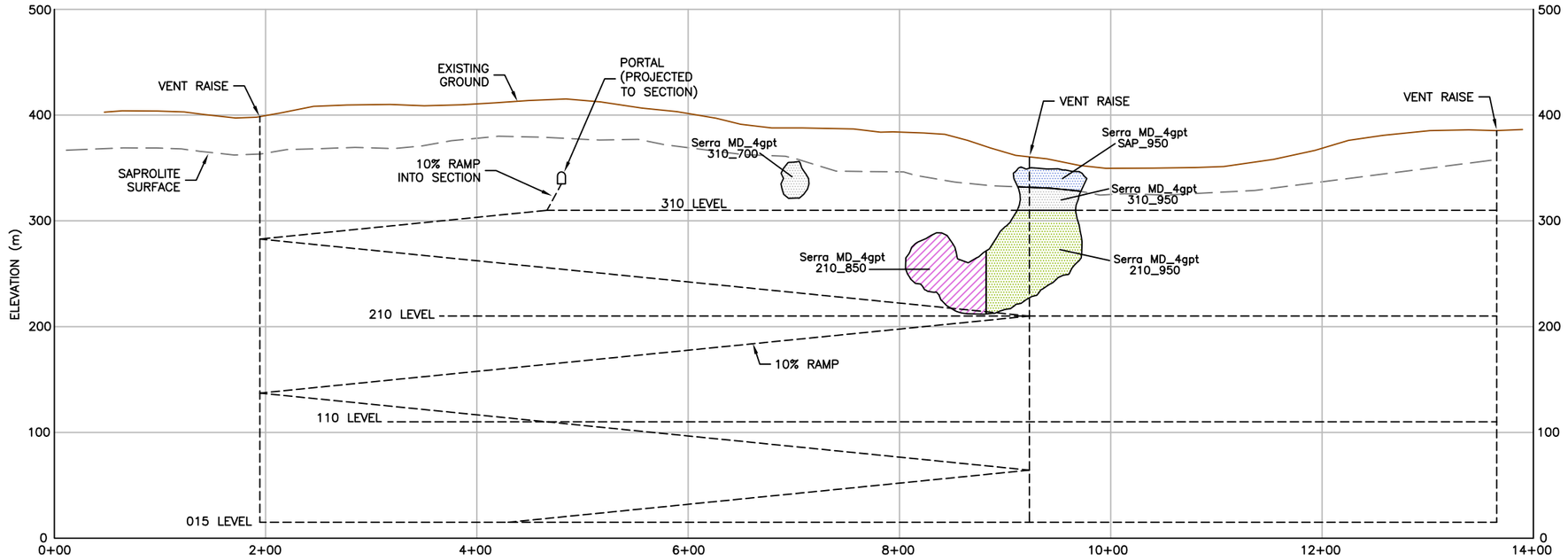

SERRA MIDDLE VEIN SECTION

IN–SITU RESOURCES BY STOPE

Block	Stope	Cubic Meters	Tonnes	Grams Au	Grade g/tonne	Troy ounces
Serra MD_4gpt	210_850	5,725	15,457	79,245	5.1	2,548
Serra MD_4gpt	210_950	9,435	25,473	240,603	9.4	7,736
Serra MD_4gpt	310_700	1,059	2,859	12,209	4.3	393
Serra MD_4gpt	310_950	1,786	4,821	40,647	8.4	1,307
Serra MD_4gpt	SAP_950	1,719	4,640	37,117	8.0	1,193
Serra MD_4gpt	Total	19,723	53,251	409,821	7.7	13,176

STOPE DESCRIPTION = LEVEL_SECTION LOCATION

YEAR STOPE PRODUCTION BEGINS

- 2016
- 2017
- 2018
- 2019
- 2020
- 2021
- 2022
- 2023
- 2024

Rev	Description	By	Date
B	ISSUED FOR REPORT	GRE	11/2012
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REVISIONS

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Drawn by: DKS
Checked by: KJG
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Prepared by: GRE Global Resource Engineering

Prepared for: MAGELLAN MINERALS LTD.

Title: SERRA MIDDLE VEIN STOPES
4 gpt Au CUTOFF

B REVISION

Project: CORINGA
Project No.: 12.1032
Location: PARA, BRAZIL
Date: 11/2012
DRAWING 05

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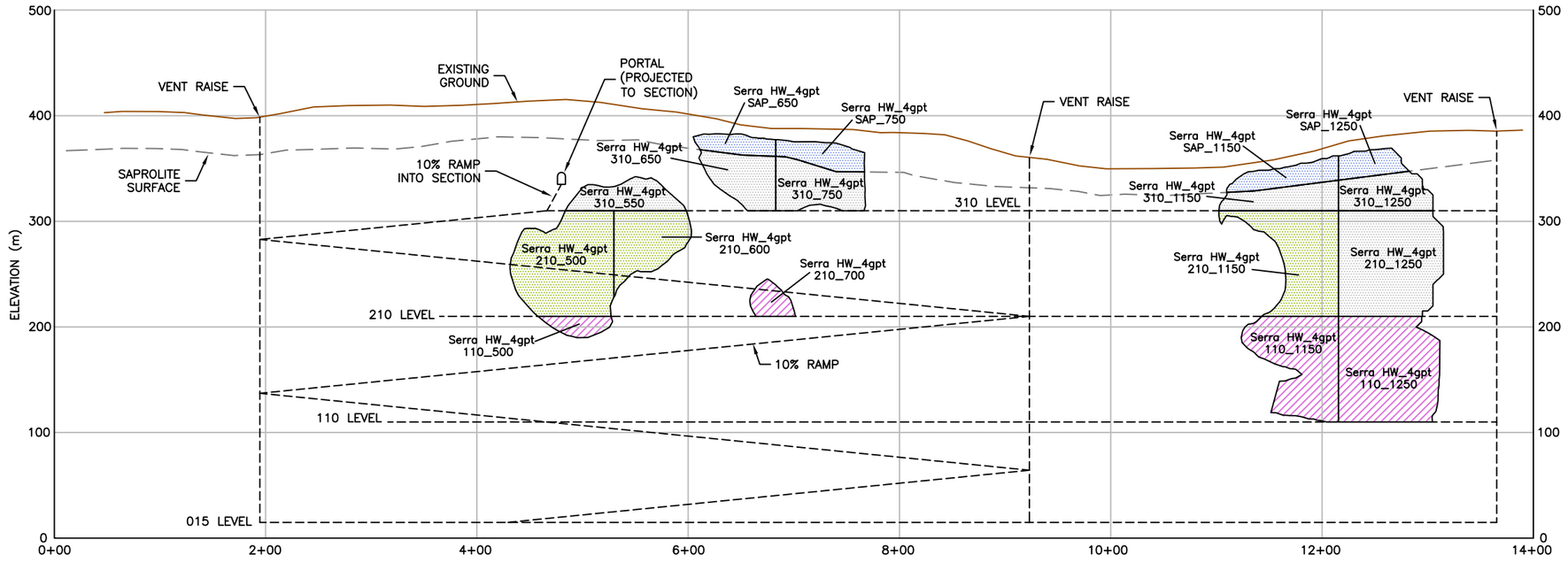

SERRA HANGING WALL SECTION

IN–SITU RESOURCES BY STOPE

Block	Stope	Cubic Meters	Tonnes	Grams Au	Grade g/tonne	Troy ounces
Serra HW_4gpt	110_1150	9,756	26,341	193,662	7.4	6,226
Serra HW_4gpt	110_1250	13,575	36,653	379,465	10.4	12,200
Serra HW_4gpt	110_500	1,847	4,986	26,748	5.4	860
Serra HW_4gpt	210_1150	9,511	25,679	260,157	10.1	8,364
Serra HW_4gpt	210_1250	14,129	38,147	554,226	14.5	17,819
Serra HW_4gpt	210_500	13,641	36,830	241,890	6.6	7,777
Serra HW_4gpt	210_600	5,823	15,722	138,807	8.8	4,463
Serra HW_4gpt	210_700	1,601	4,324	19,727	4.6	634
Serra HW_4gpt	310_1150	3,680	9,937	107,159	10.8	3,445
Serra HW_4gpt	310_1250	4,265	11,517	178,749	15.5	5,747
Serra HW_4gpt	310_550	3,754	10,137	75,690	7.5	2,433
Serra HW_4gpt	310_650	4,432	11,968	96,482	8.1	3,102
Serra HW_4gpt	310_750	5,187	14,006	104,198	7.4	3,350
Serra HW_4gpt	SAP_1150	2,699	7,287	79,320	10.9	2,550
Serra HW_4gpt	SAP_1250	2,032	5,488	78,870	14.4	2,536
Serra HW_4gpt	SAP_650	1,857	5,015	41,566	8.3	1,336
Serra HW_4gpt	SAP_750	2,434	6,573	54,197	8.2	1,742
Serra HW_4gpt	Total	100,225	270,607	2,630,913	9.7	84,586

STOPE DESCRIPTION = LEVEL_SECTION LOCATION

YEAR STOPE PRODUCTION BEGINS

- 2016
- 2017
- 2018
- 2019
- 2020
- 2021
- 2022
- 2023
- 2024

Rev	Description	By	Date
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REVISIONS

Designed by: GRE
Drawn by: DKS
Checked by: KJG
Approved by: KJG

Prepared by: GRE Global Resource Engineering

Prepared for: MAGELLAN MINERALS LTD.

Title: SERRA HANGING WALL STOPES 4 gpt Au CUTOFF

REVISION B

Project: CORINGA | Project No.: 12.1032 | DRAWING 06

Location: PARA, BRAZIL | Date: 11/2012

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717,500 E

716,500 E

9,163,000 N

9,162,500 N

9,162,000 N

MEIO-COME QUIETO RESOURCE BLOCK

ORE HAULAGE ROAD (NOTE 3)

DRIFT/ STATION LINE

0+00 1+00 2+00 3+00 4+00 5+00 6+00 7+00 8+00 9+00 10+00 11+00 12+00 13+00 14+00 15+00 16+00 17+00

VENT RAISE

VENT RAISE

VENT RAISE

PORTAL

PORTAL

CLEARED AREA

DDH07
DDH09
DDH10
DDH08
DH151
DH154
DH126
DH122
DH120
DH88
DH89
DH87

300 325 325 325 350 375 360

LEGEND

- CORE DRILLHOLE LOCATION AND PROJECTION (2007–2008)
- CORE DRILLHOLE LOCATION AND PROJECTION (2010–2011)
- MINERALIZED VEIN
- OLD ARTISANAL SURFACE WORKING (GARIMPO)
- EXISTING ACCESS ROAD

NOTES:

1) EXISTING GROUND TOPOGRAPHY PROVIDED BY MAGELLAN.

2) MINERALIZED VEINS SHOWN IN PLAN AT OUTCROP TO SURFACE.

3) EXISTING ROAD TO BE IMPROVED FOR ORE HAULAGE AT 10% GRADE MAXIMUM.

N E S W

60 0 30 60 120 240

SCALE IN METERS
CONTOUR INTERVAL: 5m

Rev	Description	By	Date
B	ISSUED FOR REPORT	GRE	11/2012
A	ISSUED FOR REVIEW	GRE	10/2012

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Prepared by: GRE Global Resource Engineering

Prepared for: MAGELLAN MINERALS LTD.

Title: MEIO-COME QUIETO UNDERGROUND MINE LAYOUT PLAN - 1

B REVISION

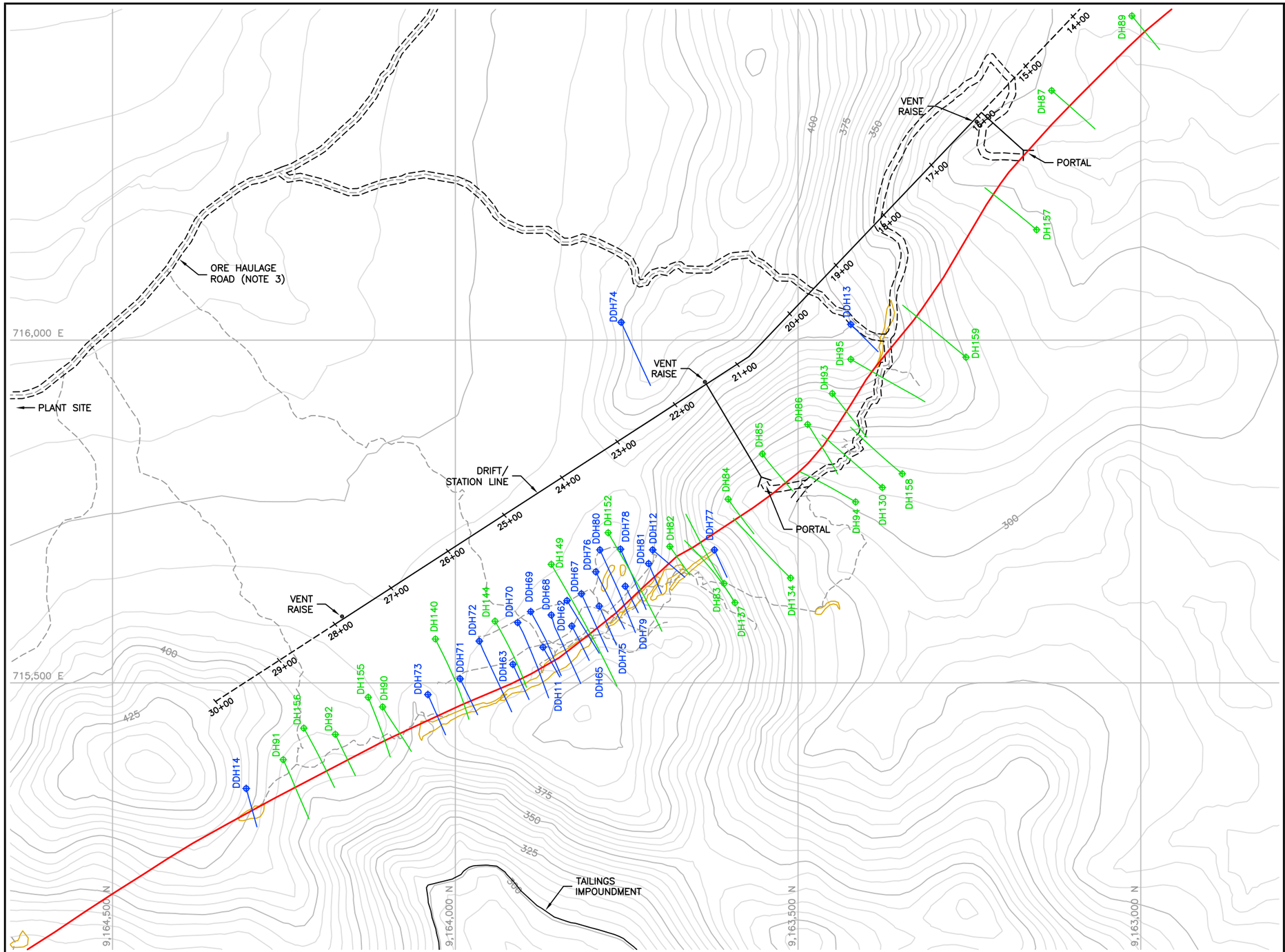
Project: CORINGA

Project No.: 12.1032

Location: PARA, BRAZIL

Date: 11/2012

DRAWING 07



LEGEND

- CORE DRILLHOLE LOCATION AND PROJECTION (2007–2008)
- CORE DRILLHOLE LOCATION AND PROJECTION (2010–2011)
- MINERALIZED VEIN
- OLD ARTISANAL SURFACE WORKING (GARIMPO)
- EXISTING ACCESS ROAD

NOTES:

1) EXISTING GROUND TOPOGRAPHY PROVIDED BY MAGELLAN.

2) MINERALIZED VEINS SHOWN IN PLAN AT OUTCROP TO SURFACE.

3) EXISTING ROAD TO BE IMPROVED FOR ORE HAULAGE AT 10% GRADE MAXIMUM.

N E S W

60 0 30 60 120 240

SCALE IN METERS
CONTOUR INTERVAL: 5m

Rev	Description	By	Date
B	ISSUED FOR REPORT	GRE	11/2012
A	ISSUED FOR REVIEW	GRE	10/2012

REFERENCE | REVISIONS

Designed by: GRE
Drawn by: DKS
Checked by: KJG
Approved by: KJG

Prepared by: GLOBAL RESOURCE ENGINEERING

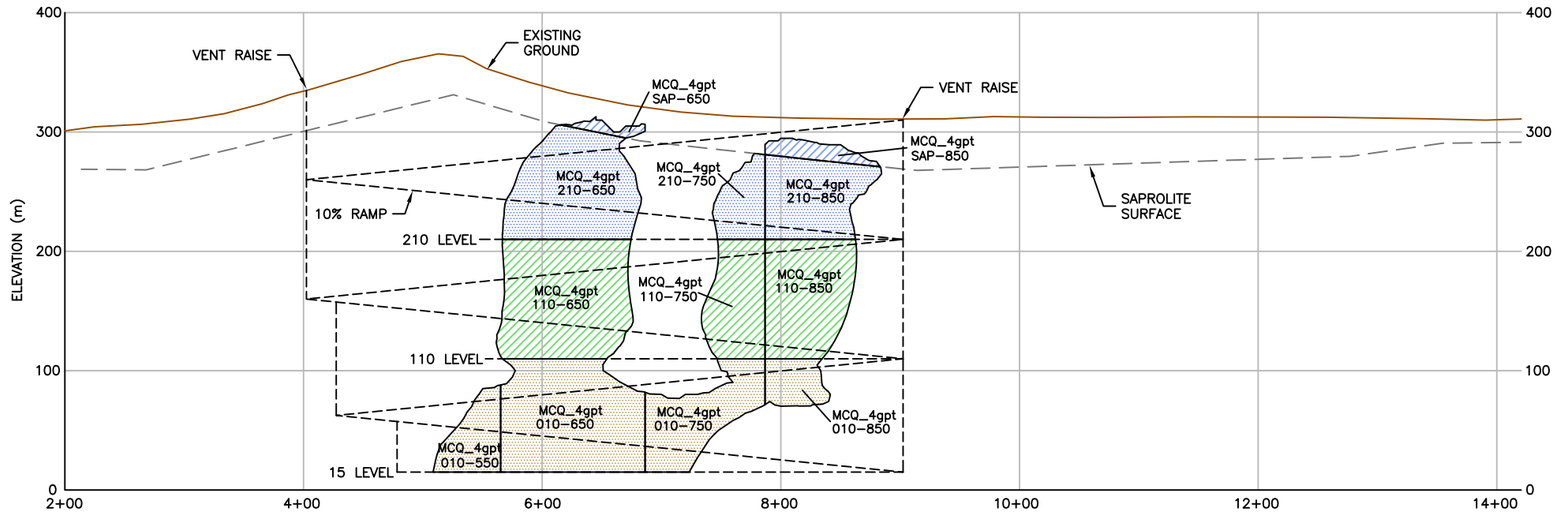
Prepared for: MAGELLAN MINERALS LTD.

Title: MEIO-COME QUIETO UNDERGROUND MINE LAYOUT PLAN - 2

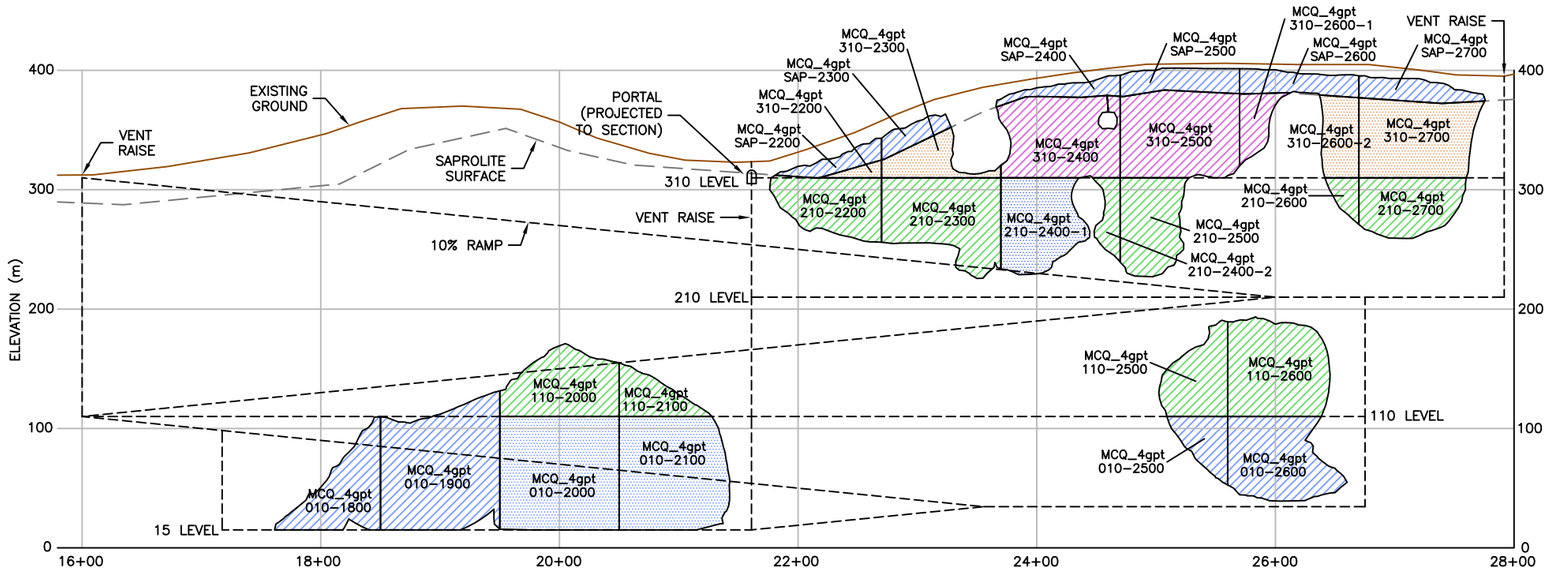
REVISION B

Project: CORINGA | Project No.: 12.1032
Location: PARA, BRAZIL | Date: 11/2012
DRAWING 08

LOCATION: D:_Clients\GRE\Coringa\PEA Study\November 2012\Dwg 08_MCQ Plan--2.dwg DATE: 11/26/2012 12:38 PM PLOT SCALE = 1:50.8 PLOTTED BY: DSHAW



MEIO-COME QUIETO SECTION



MEIO-COME QUIETO SECTION

IN–SITU RESOURCES BY STOPE

Block	Stope	Cubic Meters	Tonnes	Grams Au	Grade g/tonne	Troy ounces
MCQ_4gpt	010-1800	6,143	16,587	102,252	6.2	3,287
MCQ_4gpt	010-1900	14,657	39,573	236,564	6.0	7,606
MCQ_4gpt	010-2000	14,232	38,428	298,798	7.8	9,607
MCQ_4gpt	010-2100	12,537	33,850	227,311	6.7	7,308
MCQ_4gpt	010-2500	2,600	7,019	36,888	5.3	1,186
MCQ_4gpt	010-2600	8,204	22,152	119,674	5.4	3,848
MCQ_4gpt	010-550	4,098	11,065	48,305	4.4	1,553
MCQ_4gpt	010-650	19,292	52,089	240,997	4.6	7,748
MCQ_4gpt	010-750	10,648	28,750	133,886	4.7	4,305
MCQ_4gpt	010-850	3,355	9,060	82,814	9.1	2,663
MCQ_4gpt	110-2000	7,038	19,004	105,908	5.6	3,405
MCQ_4gpt	110-2100	2,949	7,963	39,480	5.0	1,269
MCQ_4gpt	110-2500	4,891	13,205	68,570	5.2	2,205
MCQ_4gpt	110-2600	9,762	26,357	139,183	5.3	4,475
MCQ_4gpt	110-650	20,680	55,836	283,016	5.1	9,099
MCQ_4gpt	110-750	8,682	23,441	123,978	5.3	3,986
MCQ_4gpt	110-850	10,408	28,101	186,765	6.6	6,005
MCQ_4gpt	210-2200	5,948	16,059	77,653	4.8	2,497
MCQ_4gpt	210-2300	9,559	25,809	140,475	5.4	4,516
MCQ_4gpt	210-2400-1	7,454	20,126	146,080	7.3	4,697
MCQ_4gpt	210-2400-2	2,025	5,467	29,367	5.4	944
MCQ_4gpt	210-2500	8,964	24,202	139,645	5.8	4,490
MCQ_4gpt	210-2600	1,016	2,744	13,345	4.9	429
MCQ_4gpt	210-2700	6,046	16,323	86,710	5.3	2,788
MCQ_4gpt	210-650	15,498	41,844	211,328	5.1	6,794
MCQ_4gpt	210-750	4,349	11,742	56,990	4.9	1,832
MCQ_4gpt	210-850	8,119	21,921	178,860	8.2	5,750
MCQ_4gpt	310-2200	594	1,605	8,380	5.2	269
MCQ_4gpt	310-2300	2,892	7,807	42,111	5.4	1,354
MCQ_4gpt	310-2400	9,570	25,839	214,894	8.3	6,909
MCQ_4gpt	310-2500	12,224	33,005	465,815	14.1	14,976
MCQ_4gpt	310-2600-1	2,090	5,644	43,291	7.7	1,392
MCQ_4gpt	310-2600-2	3,656	9,871	53,919	5.5	1,734
MCQ_4gpt	310-2700	13,560	36,612	212,217	5.8	6,823
MCQ_4gpt	SAP-2200	1,627	4,392	21,675	4.9	697
MCQ_4gpt	SAP-2400	1,508	4,072	29,120	7.2	936
MCQ_4gpt	SAP-2500	2,480	6,696	98,029	14.6	3,152
MCQ_4gpt	SAP-2600	2,682	7,241	48,076	6.6	1,546
MCQ_4gpt	SAP-2700	3,413	9,215	52,219	5.7	1,679
MCQ_4gpt	SAP-650	702	1,896	8,477	4.5	273
MCQ_4gpt	SAP-850	1,761	4,754	33,253	7.0	1,069
MCQ_4gpt	SAP-2300	1,488	4,019	21,036	5.2	676
MCQ_4gpt	**Total**	**289,400**	**781,380**	**4,907,354**	**6.3**	**157,775**

STOPE DESCRIPTION = LEVEL_SECTION LOCATION

YEAR STOPE PRODUCTION BEGINS

- 2016
- 2017
- 2018
- 2019
- 2020
- 2021
- 2022
- 2023
- 2024

Rev	Description	By	Date
B	ISSUED FOR REPORT	GRE	11/2012
A	ISSUED FOR REVIEW	GRE	10/2012

REFERENCE

REVISIONS

Designed by: GRE
Drawn by: DKS
Checked by: KJG
Approved by: KJG

Prepared by: GLOBAL RESOURCE ENGINEERING

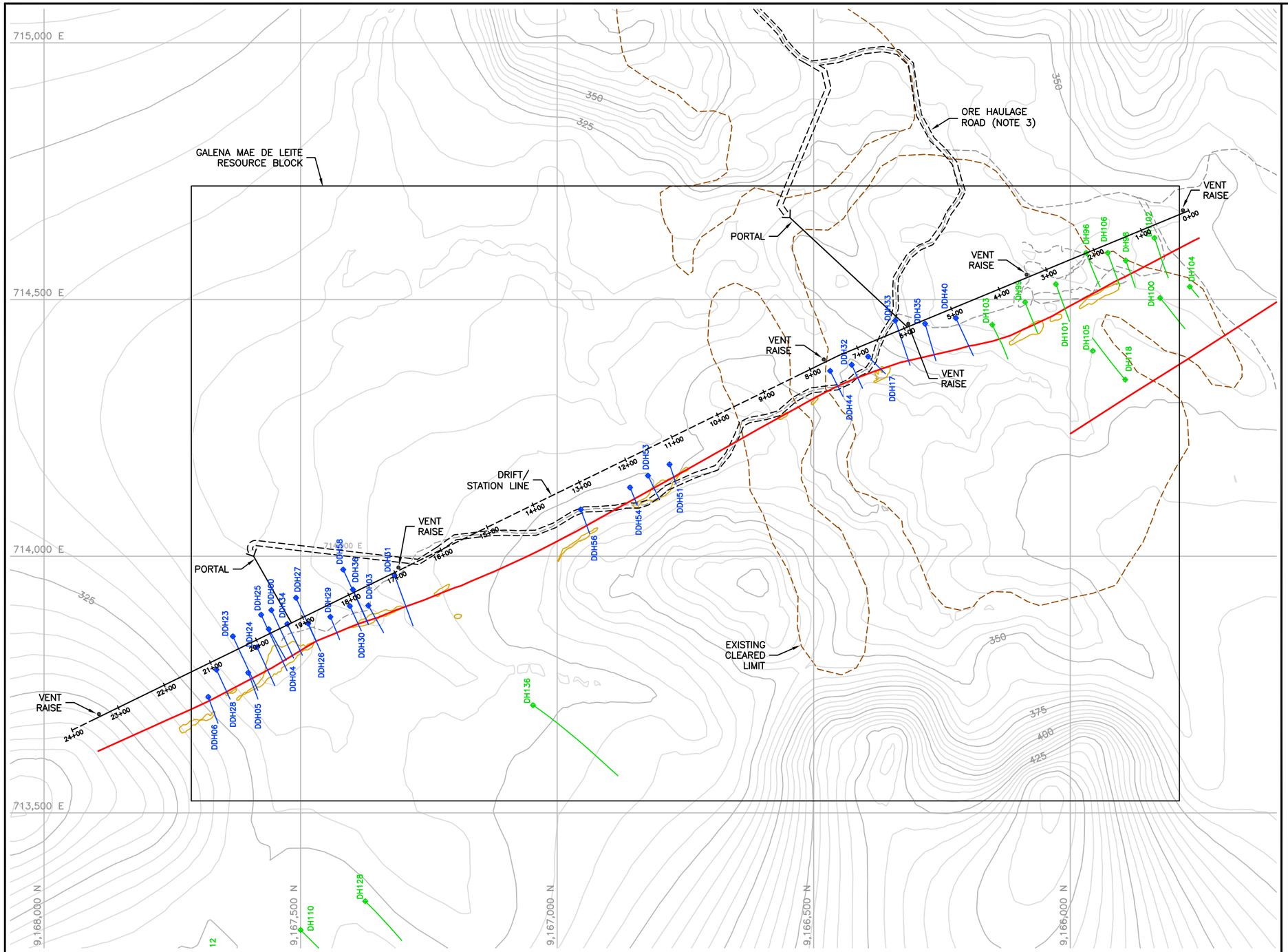
Prepared for: MAGELLAN MINERALS LTD.

Title: MEIO-COME QUIETO STOPES
4 gpt Au CUTOFF

REVISION: B

Project: CORINGA
Project No.: 12.1032
Location: PARA, BRAZIL
Date: 11/2012
DRAWING: 09

LOCATION: D:_Clients\GRE\Coringa\PEA Study\November 2012\Dwg 09_MCQ 4gpt Stope Section.dwg DATE: 11/26/2012 12:39 PM PLOT SCALE = 1:50.8 PLOTTED BY: DSHAW



LEGEND

- CORE DRILLHOLE LOCATION AND PROJECTION (2007–2008)
- CORE DRILLHOLE LOCATION AND PROJECTION (2010–2011)
- MINERALIZED VEIN
- OLD ARTISANAL SURFACE WORKING (GARIMPO)
- EXISTING ACCESS ROAD

NOTES:

1) EXISTING GROUND TOPOGRAPHY PROVIDED BY MAGELLAN.

2) MINERALIZED VEINS SHOWN IN PLAN AT OUTCROP TO SURFACE.

3) EXISTING ROAD TO BE IMPROVED FOR ORE HAULAGE AT 10% GRADE MAXIMUM.

N E S W

80 0 40 80 160 320

SCALE IN METERS
CONTOUR INTERVAL: 5m

Rev	Description	By	Date
B	ISSUED FOR REPORT	GRE	11/2012
A	ISSUED FOR REVIEW	GRE	10/2012

REFERENCE

REVISIONS

Designed by: GRE
Drawn by: DKS
Checked by: KJG
Approved by: KJG

Prepared by: GRE Global Resource Engineering

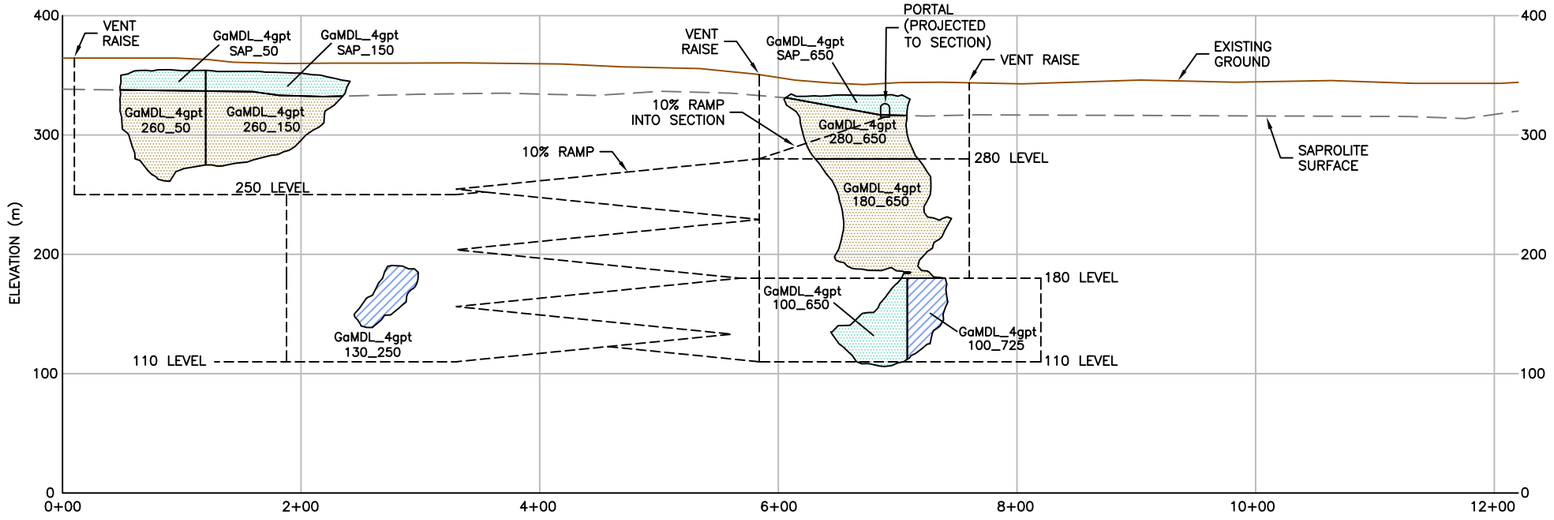
Prepared for: MAGELLAN MINERALS LTD.

Title: GALENA MAE DE LEITE UNDERGROUND MINE LAYOUT PLAN

B REVISION

Project: CORINGA
Project No.: 12.1032
Location: PARA, BRAZIL
Date: 11/2012
DRAWING 10

LOCATION: D:_Clients\GRE\Coringo\PEA Study\November 2012\Dwg 10_GMDL Plan.dwg DATE: 11/26/2012 12:40 PM PLOT SCALE = 1:50.8 PLOTTED BY: DSHAW



GALENA MAE DE LEITE SECTION

IN–SITU RESOURCES BY STOPE

Block	Stope	Cubic Meters	Tonnes	Grams Au	Grade g/tonne	Troy ounces
GaMDL_4gpt	100_650	4,047	10,927	45,655	4.2	1468
GaMDL_4gpt	100_725	2,559	6,908	28,907	4.2	929
GaMDL_4gpt	130_250	2,135	5,764	24,206	4.2	778
GaMDL_4gpt	180_650	11,223	30,303	203,396	6.7	6539
GaMDL_4gpt	200_1800	5,703	15,398	93,597	6.1	3009
GaMDL_4gpt	200_1850	8,971	24,221	191,110	7.9	6144
GaMDL_4gpt	200_1950	9,913	26,766	270,832	10.1	8707
GaMDL_4gpt	260_150	7,614	20,557	109,475	5.3	3520
GaMDL_4gpt	260_50	6,654	17,966	90,925	5.1	2923
GaMDL_4gpt	280_650	5,647	15,247	140,347	9.2	4512
GaMDL_4gpt	SAP_150	3,000	8,100	45,294	5.6	1456
GaMDL_4gpt	SAP_1800	969	2,617	14,853	5.7	478
GaMDL_4gpt	SAP_1850	4,549	12,281	138,352	11.3	4448
GaMDL_4gpt	SAP_1950	4,841	13,071	139,834	10.7	4496
GaMDL_4gpt	SAP_50	1,757	4,745	24,979	5.3	803
GaMDL_4gpt	SAP_650	1,545	4,171	31,231	7.5	1004
GaMDL_4gpt	**Total**	**81,126**	**219,041**	**1,592,993**	**7.3**	**51216**

STOPE DESCRIPTION = LEVEL_SECTION LOCATION

GALENA MAE DE LEITE SECTION

YEAR STOPE PRODUCTION BEGINS

- 2016
- 2017
- 2018
- 2019
- 2020
- 2021
- 2022
- 2023
- 2024

Rev	Description	By	Date
B	ISSUED FOR REPORT	GRE	11/2012
A	ISSUED FOR REVIEW	GRE	10/2012

REFERENCE

REVISIONS

Designed by: GRE
Drawn by: DKS
Checked by: KJG
Approved by: KJG

Prepared by: GRE Global Resource Engineering

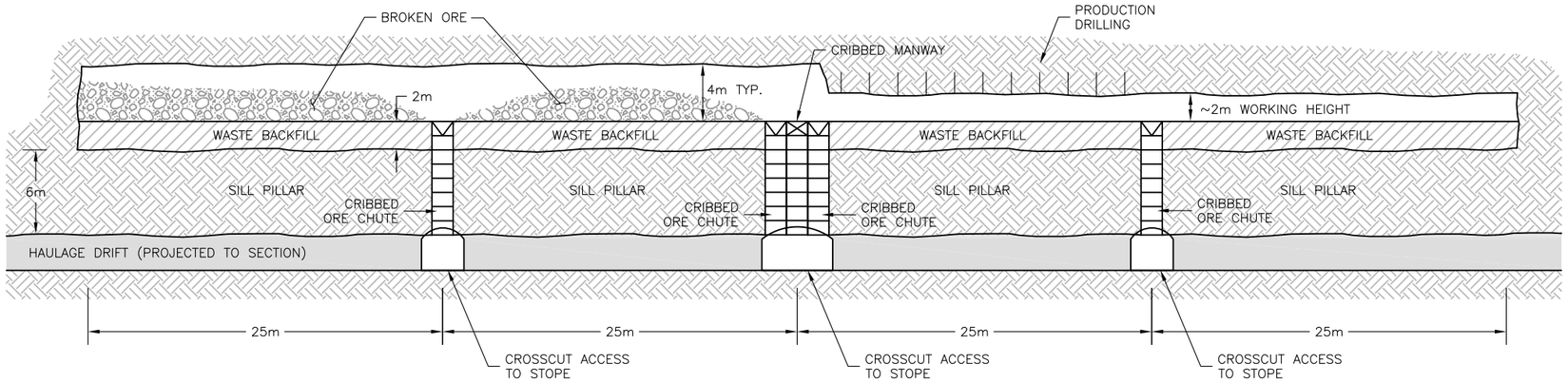
Prepared for: MAGELLAN MINERALS LTD.

Title: GALENA MAE DE LEITE STOPES
4 gpt Au CUTOFF

B REVISION

Project: CORINGA
Project No.: 12.1032
Location: PARA, BRAZIL
Date: 11/2012
DRAWING 11

LOCATION: D:_Clients\GRE\Coringa\PEA Study\November 2012\Dwg 11_GMDL 4gpt Stope Section.dwg DATE: 11/26/2012 12:41 PM PLOT SCALE = 1:50.8 PLOTTED BY: DSHAW



TYPICAL STOPE SECTION

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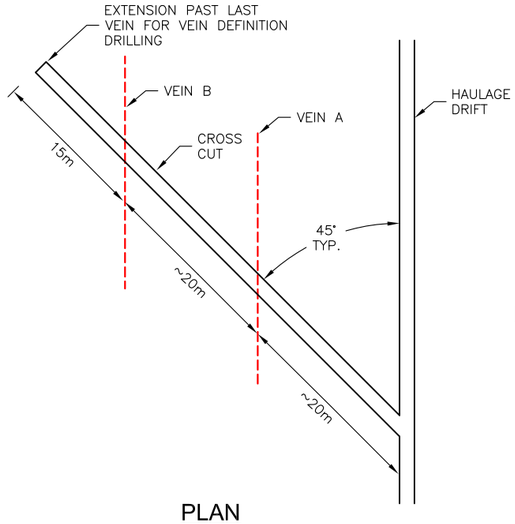
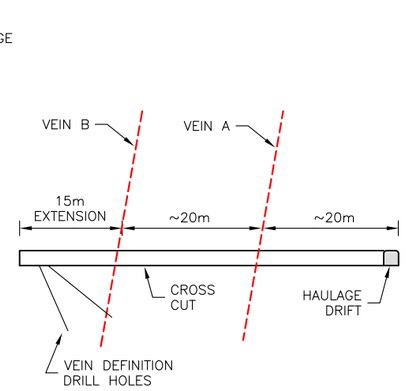


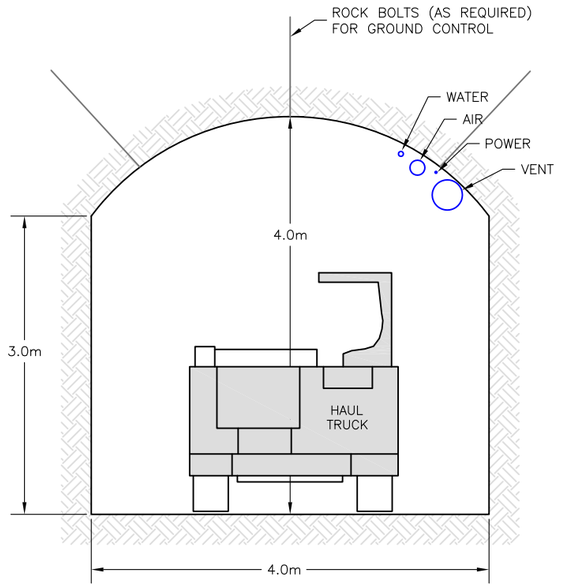
PLAN



SECTION

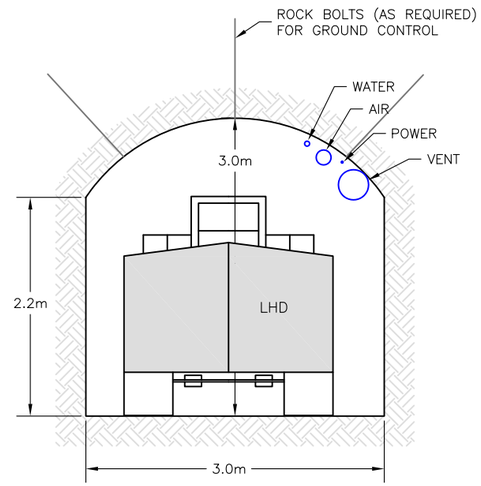
TYPICAL CROSSCUT DETAIL

NOT TO SCALE



TYPICAL RAMP SECTION

NOT TO SCALE



TYPICAL CROSSCUT SECTION

NOT TO SCALE

Rev	Description	By	Date
B	ISSUED FOR REPORT	GRE	11/2012
A	ISSUED FOR REVIEW	GRE	10/2012

REFERENCE — REVISIONS

Designed by: GRE
Drawn by: DKS
Checked by: KJG
Approved by: KJG

Prepared by: GRE Global Resource Engineering

Prepared for: MAGELLAN MINERALS LTD.

Title: TYPICAL STOPE SECTIONS AND DETAILS

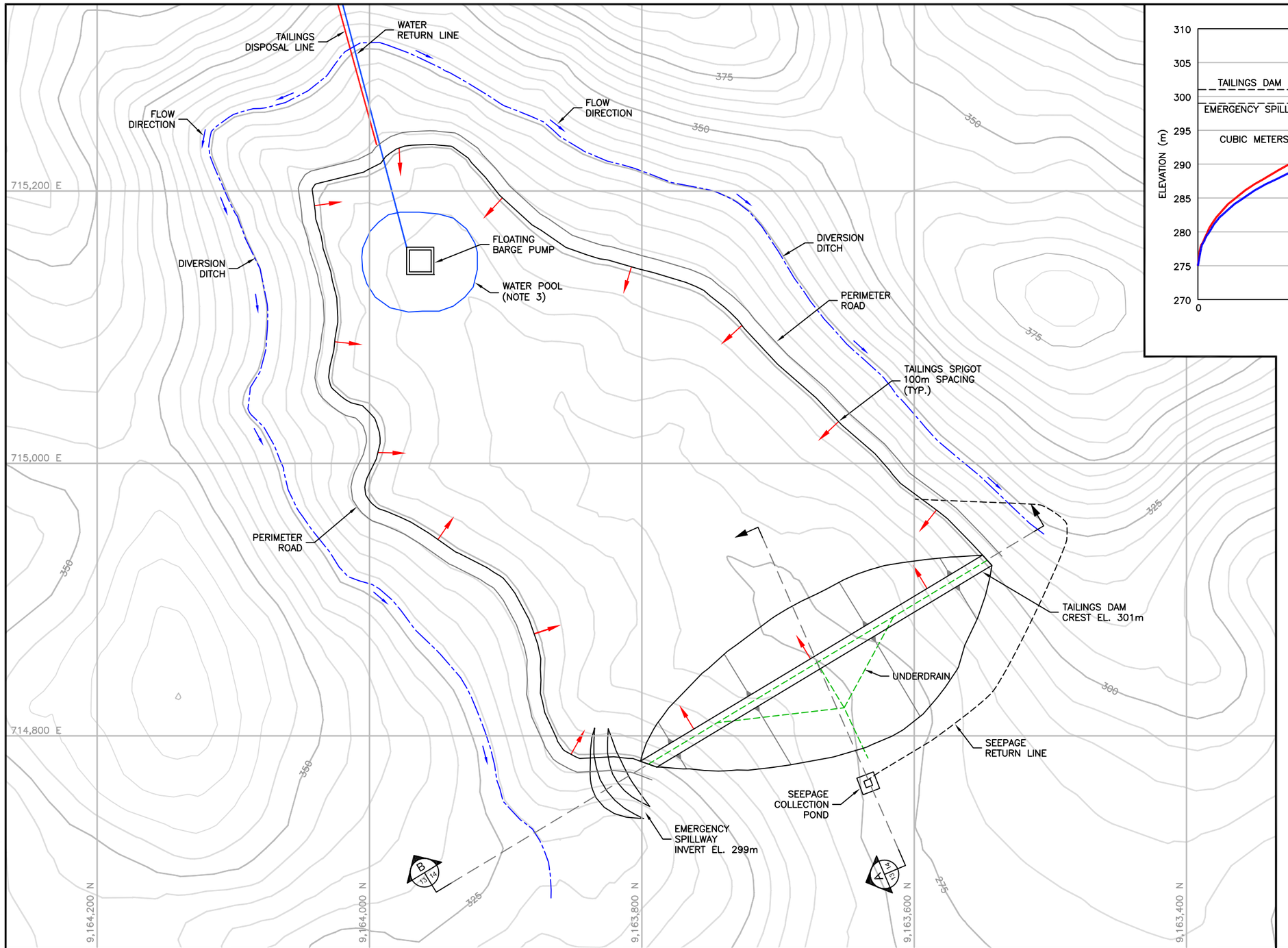
REVISION: B

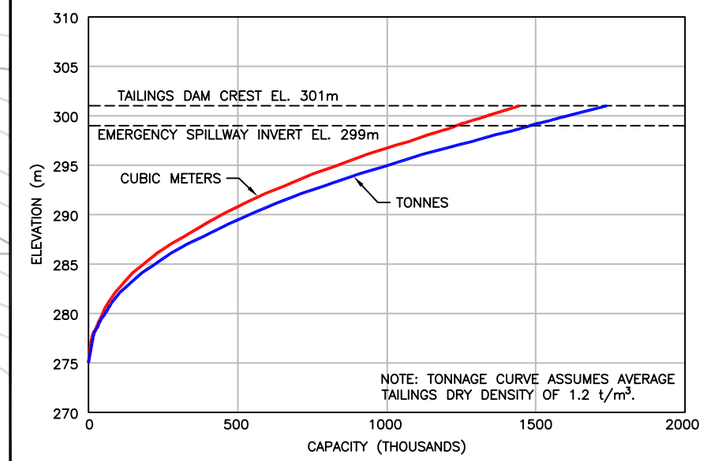
Project: CORINGA | Project No.: 12.1032

Location: PARA, BRAZIL | Date: 11/2012

DRAWING 12

LOCATION: D:_Clients\GRE\Coringa\PEA Study\November 2012\Dwg 12_Stope Details.dwg DATE: 11/26/2012 12:41 PM PLOT SCALE = 1:50.8 PLOTTED BY: DSHAW



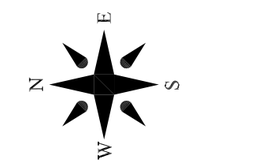


NOTES:

1) 1.2 t/m³ OVERALL DENSITY ASSUMED FOR THIS DESIGN.

2) CAPACITY CURVE ASSUMED FLAT LYING TAILINGS SURFACE FOR DESIGN WITH 100 YR, 24 HR DESIGN STORM + 2m MIN. DRY FREEBOARD ABOVE WATER POOL.

3) WATER POOL MAINTAINED IN EAST END OF IMPOUNDMENT AWAY FROM DAM FACE.



30 0 15 30 60 120

SCALE IN METERS
CONTOUR INTERVAL: 5m

Rev	Description	By	Date
B	ISSUED FOR REPORT	GRE	11/2012
A	ISSUED FOR REVIEW	GRE	10/2012

REFERENCE

REVISIONS

Designed by: GRE
Drawn by: DKS
Checked by: KJG
Approved by: KJG

Prepared by: GLOBAL RESOURCE ENGINEERING

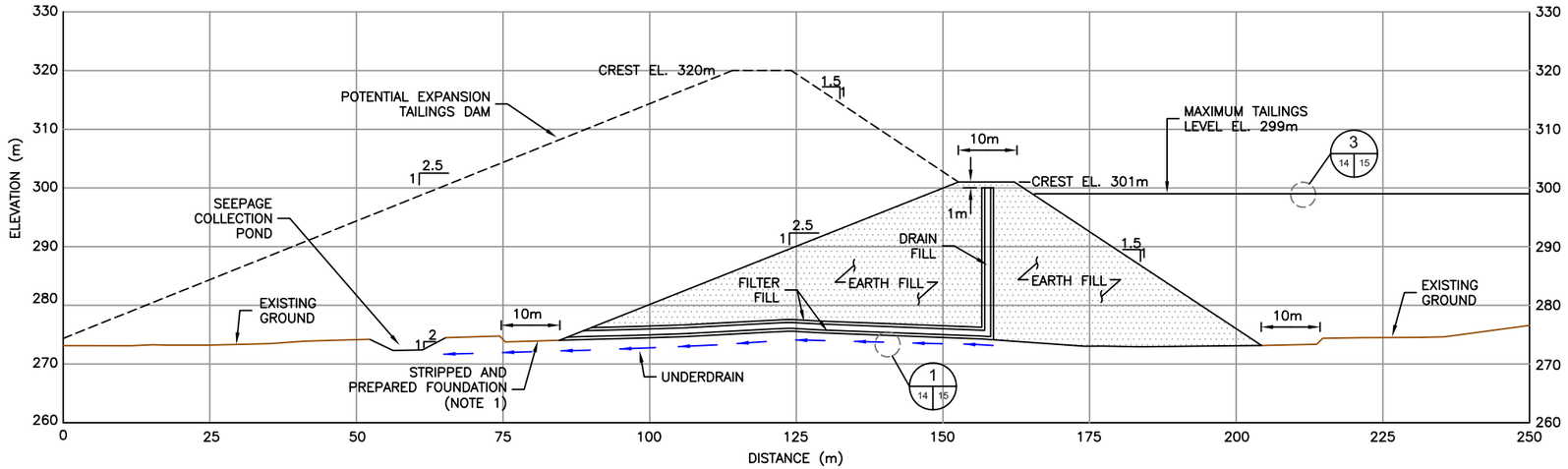
Prepared for: MAGELLAN MINERALS LTD.

Title: TAILINGS IMPOUNDMENT PLAN

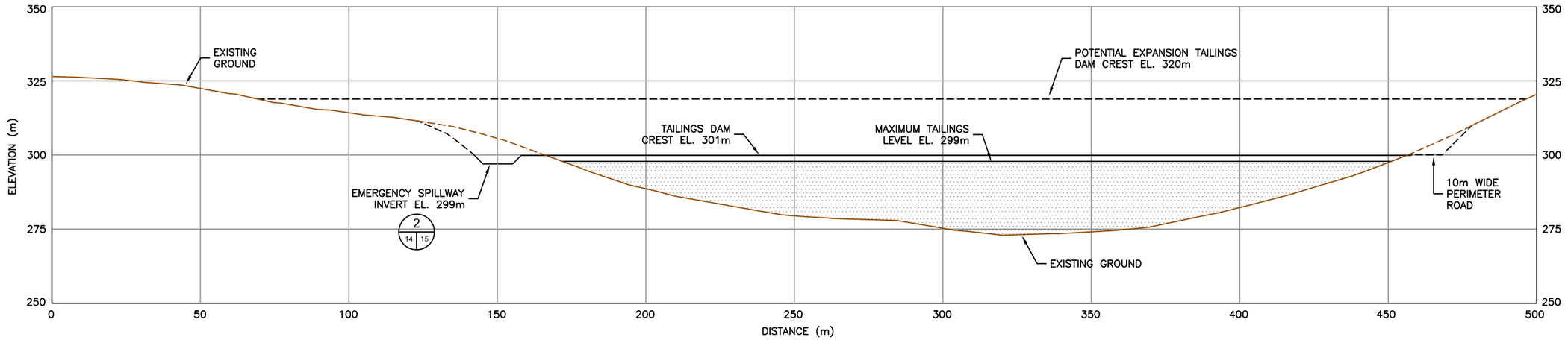
B REVISION

Project: CORINGA
Project No.: 12.1032
Location: PARA, BRAZIL
Date: 11/2012
DRAWING 13

LOCATION: D:_Clients\GRE\Coringa\PEA Study\November 2012\Dwg 13_TSF Plan.dwg DATE: 11/26/2012 12:42 PM PLOT SCALE = 1:50.8 PLOTTED BY: DSHAW



TAILINGS DAM SECTION A (13 | 14)



TAILINGS DAM SECTION B (13 | 14)

NOTES:

1) 1m DEEP CUT ASSUMED WITHIN 10m PERIMETER OF EMBANKMENT FILL.

Rev	Description	By	Date
B	ISSUED FOR REPORT	GRE	11/2012
A	ISSUED FOR REVIEW	GRE	10/2012

REFERENCE | REVISIONS

Designed by: GRE
Drawn by: DKS
Checked by: KJG
Approved by: KJG

Prepared by: GRE Global Resource Engineering

Prepared for: MAGELLAN MINERALS LTD.

Title: TAILINGS DAM SECTIONS

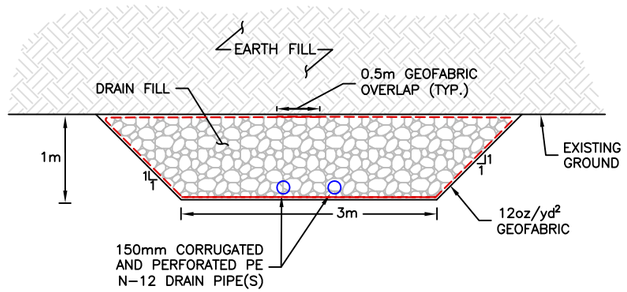
REVISION: B

Project: CORINGA | Project No.: 12.1032

Location: PARA, BRAZIL | Date: 11/2012

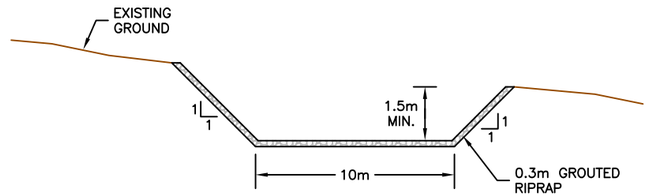
DRAWING 14

LOCATION: D:_Clients\GRE\Coringa\PEA Study\November 2012\Dwg 14_TSF Sections.dwg DATE: 11/26/2012 12:43 PM PLOT SCALE = 1:50.8 PLOTTED BY: DSHAW



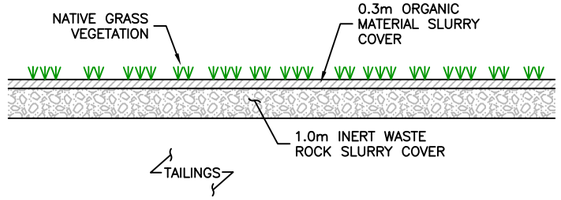
DAM UNDERDRAIN DETAIL 1 (14 | 15)

NOT TO SCALE



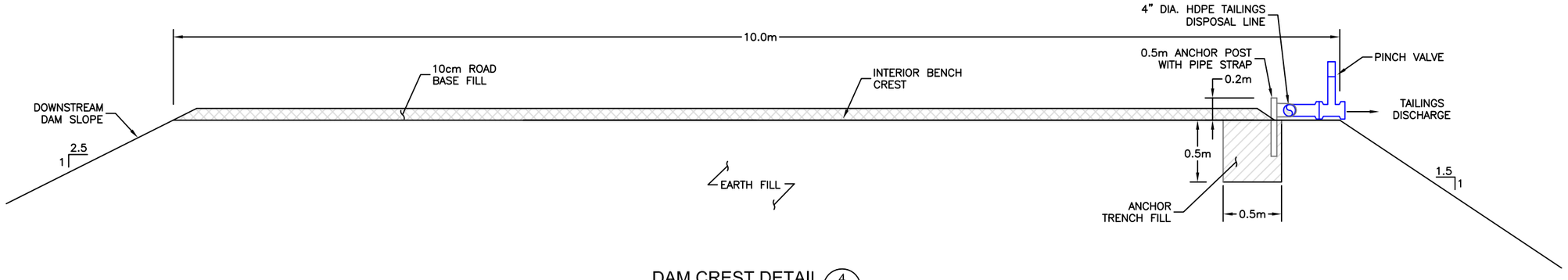
EMERGENCY SPILLWAY DETAIL 2 (14 | 15)

NOT TO SCALE



CONCEPTUAL CLOSURE COVER DETAIL 3 (14 | 15)

NOT TO SCALE



DAM CREST DETAIL 4 (14 | 15)

NOT TO SCALE

REFERENCE	Rev	Description	By	Date
	B	ISSUED FOR REPORT	GRE	11/2012
	A	ISSUED FOR REVIEW	GRE	10/2012

REVISIONS

Designed by: GRE
Drawn by: DKS
Checked by: KJG
Approved by: KJG

Prepared by: GRE Global Resource Engineering

Prepared for: Magellan Minerals Ltd.

Title: TAILINGS DAM DETAILS

B REVISION

Project: CORINGA | Project No.: 12.1032

Location: PARA, BRAZIL | Date: 11/2012

DRAWING 15

LOCATION: D:_Clients\GRE\Coringa\PEA Study\November 2012\Dwg 15_TSF Details.dwg DATE: 11/26/2012 12:44 PM PLOT SCALE = 1:50.8 PLOTTED BY: DSHAW