FORM 6-K

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

Report of Foreign Private Issuer

Pursuant to Rule 13a-16 or 15d-16
of the Securities Exchange Act of 1934

Date: March 30, 2010
Commission File Number 001-31528

IAMGOLD Corporation

(Translation of registrant's name into English)

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(Address of principal executive offices)

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

Form 20-F ☐ Form 40-F ☒

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

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Description of Exhibit

Exhibit	Description of Exhibit
99.1	Rosebel Technical Report March 29, 2010

Signatures

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

IAMGOLD CORPORATION

Date: March 30, 2010

By: /s/ LARRY E. PHILLIPS
Executive Vice-President, Corporate Affairs

Exhibit 99.1

IAMGOLD CORPORATION

Rosebel Mine, Suriname

NI 43-101 Technical Report

Report prepared by
Gabriel Voicu, PhD, P.Geo
Geology & Mine Exploration Superintendent
Rosebel Gold Mines N.V., Suriname

March 29 th , 2010

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SUMMARY

1. INTRODUCTION

This report on the Rosebel mine, located in the Brokopondo district of Suriname, South America, provides an updated technical report prepared according to the Canadian National Instrument 43-101 Standards of Disclosure for Mineral Projects, for the purpose of supporting certain public disclosures to be made by IAMGOLD Corporation.

The Rosebel Mine concession covers an area of 170 square kilometres in north central Suriname at a latitude of 55 º 25’ North and a longitude of 55 º 10’ West. The property lies in the district of Brokopondo, between the Suriname River to the east and the Saramacca River to the west, approximately 80 kilometres south of the capital city of Paramaribo.

The Rosebel concession is held by Rosebel Gold Mines N.V. (“RGM”), in which Iamgold Corporation holds an indirect 95% interest. RGM is also the holder of three exploration concessions referred to as Rights Of Exploration (“ROE”). ROE’s for the adjacent Headley’s Reef, Thunder Mountain and Triangle properties were in the past renewed in favour of Golden Star and thereafter assigned, conveyed and transferred to RGM simultaneously with the Rosebel mine concession.

Commercial production of gold at the Rosebel mine began in February 2004. The Rosebel mine has an excellent history of mineral reserve renewal. Since 2004, more than 350,000 meters of diamond drilling contributed to reserve expansion and development. The exploration will continue in 2010 (more than 80,000 meters of drilling) and supposedly during the following years.

The mining operations take place actually in four open pit gold deposits from a total of eight deposits defined on the Rosebel concession. A wealth of geological information has been gathered from production and exploration activities. The existing Rosebel databases contain all this information, while additional data is acquired every day.

This data is used for deposit modeling and in the calculations of ore and waste tonnage, grade distribution and resource and reserve estimates. The Rosebel block models are updated at least once a year, as a function of new information coming from the exploration and production drilling. In the active pits, the use of the production drilling data increases the quality and precision of modeling. However, due to the complexity of the Rosebel deposits, the tonnage and grade predicted by the block models still underestimates the production. This was the main reason for the introduction of adjustment factors between production and reserves in 2009.

The estimates presented in this report for Measured, Indicated & Inferred Resources are constrained in pit shells. In 2008, only Inferred Resources were presented in this way, with Measured and Indicated Resources stated without pit shell constraints.

The reserve data presented in this report has been estimated using a gold price of US$850/oz, while the resource data has been estimated using a gold price of US$1,000/oz.

Based on the review of the Rosebel mine for the purposes of this report, the author makes the following recommendations:

- Several tests using 3 metre and 5 metre composites should be performed to verify the grade distribution that best matches the mining production tonnage and grade.
- Several tests should be performed to compare the grade distribution derived from Inverse Distance and Ordinary Kriging modelling.
- Additional definition drilling is required to transfer existing Inferred Resources into the Measured or Indicated Resource categories.
- Most gold deposits at Rosebel are still open at depth and on strike. Exploration drilling is therefore required to identify potential extensions to known deposits and increase the resource base of the project.

- The Rosebel and FILAB laboratory performance must be improved. An external audit covering sample preparation and assay procedures should be considered.
- Standard operating procedures for drilling, sampling and block modelling should be created to ensure greater consistency of results.
- Bulk density data should derive from both the diamond drill core and working face sampling in the active pits.
- Reconciliations between Rosebel mine production and mineral reserves indicate that block models have consistently underestimated production grade and tonnage. The use of the adjustment factors between production and reserves in 2009 should be revised on an annual basis. Fine tuning the block models for some of the active pits could also improve reconciliations.

1.1. GEOLOGY AND MINERALIZATION

The Rosebel concession lies within the Lower Proterozoic Guiana Shield, which stretches from the Amazon River in Brazil to the Orinoco River in Venezuela. The Suriname portion of the shield consists of distinct belts of low-grade metamorphic rocks, separated by large areas underlain by granites and gneisses. Local remnants of the sub-horizontal Precambrian Roraima Formation unconformably overlie this basement, and younger Proterozoic and Permo-Triassic diabase dykes cut the sequence.

Two sedimentary sequences showing different facies characteristics are recognized on the property and were classified as the Armina and Rosebel Formations in previous reports. However, stratigraphic relationships on the property are poorly defined and the terms ‘deep -water sediments’ and ‘shallow-water sediments’ will be adopted in this report. The term Armina Formation is retained for sequences dominated by volcanic rocks. Further geological clarifications are expected in 2010 when the results of a detailed U-Pb zircon geochronology study are made available.

The geology of the property and the style of gold mineralization vary between the northern and southern limbs of a regional syncline, and also between the various deposits. In the north limb, the mineralized trend has a strike length of 12 kilometres, and hosts the Pay Caro – East Pay Caro, Koolhoven and "J" Zone deposits as well as the Spin and Mamakreek anomalies. The mineralized trend in the south limb has a strike length of 15 kilometres, and hosts the Mayo, Royal Hill, Roma and Rosebel deposits, as well as the Eriaan Hill and Monsanto Hill anomalies.

Gold deposits in the Rosebel camp are classified as Orogenic Gold Deposits and characterized by mineralized quartz veins emplaced during or shortly after the Trans-Amazonian Orogeny (between 2180-2000 Ma).

Most rock types in the Rosebel concession host quartz veins, some of which are gold bearing. Primary gold mineralization occurs in several different styles on the property. At Royal Hill, mineralization is hosted by at least three generations of veining that can be distinguished based on infill composition: quartz, quartz-carbonate-tourmaline and quartz-chlorite.

Veins are emplaced during and after major episodes of deformation and are generally restricted to lithological contacts, fold closures and sub-vertical shear corridors. Veins vary from a few centimetres up to four metres in thickness. Gold typically occurs in its native form as free grains, often precipitated close to the vein selvages at an early stage of hydrothermal activity or as intergrowths in pyrite crystals within veins and adjacent country rocks. At Royal Hill, a second generation of veining, represented by centimetric to metric-scale veins with a quartz-carbonate-tourmaline infill assemblage is also associated with native gold occurrences. These are overprinted by a locally developed millimetric to centimetric-scale conjugate quartz-chlorite +/- carbonate +/- gold veining system.

Wall rock alteration typically consists of 2 to 5% pyrite and pyrrhotite, with weak carbonate alteration around quartz-carbonate veins, and K-feldspar around quartz-carbonate-feldspar veins. Epidote and muscovite occur locally in the vicinity of the Brinks Granite. Alteration haloes range from 25 centimetres around thin veins to over 20 metres around major vein sets.

1.2. Resource And Reserve Estimation

1.2.1. Database

The database consists of 3,601 diamond drill holes, 4,170 auger drill holes, 893 Banka drill holes and 35,899 metres of trenches. The total number of samples is 416,365.

1.2.2. Modeling

Modeling work is done using the *GEMS version 6.1.4.2* software package. The main lithologies, structural elements, weathering profiles and ore zones of each deposit are modeled using 3D rings created on 25 metre, evenly spaced cross sections. The weathering profiles, saprolite, transition and fresh rock are determined from the geotechnical classifications and measurements taken on the core by the geotechnicians. The laterite model is designed from geological observations completed by geologists on the core.

Orezone modeling is strongly guided by the geological models of each project which provide lithological and structural constraints. Generally, ore zone envelopes are drawn from assays in the drilling database with a gold content higher than 0.3 g/t Au and include no more than three cumulative metres of assays of less than 0.3 g/t Au. Ore zones must be at least four metres thick in saprolite and at least five metres thick in transition and fresh rock, except for the Mayo deposit where a three metre minimum thickness is tolerated in some cases to accommodate thinner ore zones.

From the 3D rings drawn on the sections, surfaces and solids are built and validated. For some projects including blast holes (East Pay Caro, Pay Caro, Koolhoven, Royal Hill and Mayo), the ore solids are built from two sets of 3D rings. One set is designed on section with drill hole assays and a second set is designed on bench where blast holes assays can be more easily viewed. Both sets are then attached together to create a full 3D skeleton of each ore zone.

1.2.3. Statistical Analyses

Composites of five metres are generated from uncapped assay results of diamond, auger and Banka drill holes and channel sampling. Because blast holes generally correspond to one five metre long assay, it is not necessary to composite this particular set of data. All composites are constrained within the ore zone solids first and then within the lithology and weathering solid limits. Poor representative composites, i.e. ones including more than 2.5 metres of missing assays and/or the composites smaller than one metre, such as composites created at the end of a solid interval or at the bottom of a hole, are not taken into consideration for resource estimation.

Gold grade statistics from the set of composites are calculated in *Snowden Supervisor* software by deposits and by Rock Group. The two limits (High Grade Limit and High Grade Transition Limit) that are used in the treatment of high grade values during resource estimation are determined from those statistics (see Section 17.7 for details). The High Grade Limit corresponds to outliers observed in the histogram plots. The High Grade Transition Limit corresponds to inflexion points representing different grade domains on the curve of cumulative probability plots.

1.2.4. Block Modeling and Grade Interpolation

The block modeling estimation is done using the *GEMS version 6.1.4.2* software package. One block model is constructed for each deposit. After the completion of a drilling campaign, the block model is partially or completely updated.

In 2009, Royal Hill, Mayo, Koolhoven, Roma and J-Zone block models were updated. Mayo, Royal Hill and Roma were updated twice. For Royal Hill, the two updates correspond to the complete geological reinterpretation of the NW and SE pits. No changes were made for Pay Caro, East Pay Caro and Rosebel as the assay results were pending, but updates are scheduled for 2010 once the additional drilling information has been received.

Interpolations of grades in the block models are performed using the inverse distance cube method to the third power (ID^3) with anisotropic distances. The gold grade estimates are generated from five metres composites.

In all deposits, geological and mineralized contacts are considered as hard boundaries to avoid smearing gold grades from one mineralized zone to another or into waste. In line with this, a unique rock code is assigned to each block and composite when at least 50% is located inside the solid (ore zone, weathering and lithology solids). The resource estimates are done using a sample search approach. During the interpolation process, the rock code in each block is read from the rock code block model. Then the composites data set is scanned for composites that are associated to the same rock code and that are located within the limits of the search ellipse.

For Koolhoven, J-Zone, Pay Caro, East Pay Caro, Roma and Royal Hill SE, anisotropic search ellipses are used and set according to the orientation and dip of the mineralized zones. Spherical search ellipses are used for the grade interpolation in Mayo, Royal Hill NW and Rosebel.

The grade evaluations done with diamond drill hole (DDH) composites are performed in three different cumulative steps corresponding to three different levels of confidence (Measured, Indicated and Inferred resources). When blast hole (BH) assays are present in the project, a fourth step is added; two grade evaluations are performed separately for Measured resources with 1) BH composites and 2) with BH and DDH composites. These estimations overwrite a specific selection of blocks derived from the first three cumulative steps. In detail, grades calculated with a combination of composites from DDH and BH overwrite only blocks located inside the pit design down to 50 metres below the mined out surface; grades calculated with assays from BH overwrite only blocks located inside the pit design above the current mined out surface.

1.2.5. Classification

The mineral resource estimations for all projects are classified according to the Canadian Institute of Mining, Metallurgy and Petroleum, "CIM", Definition Standards for Mineral Resources and Reserves (December 11, 2005).

Measured

For measured resource classifications, the search ellipses have a circular radius of 50 metres, corresponding to the average diamond drilling grid spacing, and a small radius varying from 10 to 25 metres for ore zones in saprolite, transition and rock and 20 metres for laterite. The spherical ellipsoids have a radius of 50 metres. A minimum of five up to a maximum of twelve data points (five metre composites) from which no more than two composites originating from the same source of information (DDH or BH) are used to evaluate a grade in a block. Composites are thus taken from at least three different locations. The parameters defined above ensure that blocks classified as measured resources are estimated only where mineralization grade and continuity are highly reliable.

Indicated

The search ellipse's circular and small radii for laterite and ore zones in the indicated resource classification are respectively extended to 75 metres, 30 metres and 10-37.5 metres depending on the project area. In other projects using the spherical ellipsoids, the radius is lengthened to 75m. A minimum of three up to a maximum of twelve composites from which no more than two originating from the same source (DDH only) are used to evaluate a grade in a block. Composites are thus taken from at least two different locations. The parameters described above ensure that blocks classified as indicated resources are estimated only where mineralization grade and continuity are reasonably reliable.

The measured and indicated resources are used for pit optimization and pit design.

Inferred

The search ellipses in the inferred resources classification are extended to 150 metres in the azimuth and dip directions, to 30-75 metres for the small radius for ore zones and to 30-60 metres for laterite. The radius of the spherical ellipsoids is prolonged to 150 metres. A minimum of one up to a maximum of twelve composites are used to evaluate a grade in a block. The number of composites originating from an individual source of information (DDH only) is not limited.

The inferred resources are not used for pit optimization or pit design.

1.2.6. Calculation of Economic Cut-off

Pit optimization is performed by the Lerchs-Grossman algorithm using *Whittle Analyzer* software. This technique generates a series of optimal pits given the geological block models, operating costs, recoveries, geo-technical constraints and gold prices. The optimal pit chosen is normally one which maximizes the undiscounted or discounted cash flow at the given economic parameters, but pit shells can also be produced for a range of specific gold prices, if so required. The pit design process is iterative. After the theoretical optimal pit is obtained, additional mining constraints such as minimum mining widths and practical mining access ramps are included. The process is repeated until a stable design is obtained.

The mining and ore based costs, including electricity, used for the optimization are based on direct observations at Rosebel since January 2004 and are updated frequently. An incremental ore haulage cost was included in the optimization process because of the different ore haulage distance for each pit. This results in slightly different economic cut-off grades for each pit.

A portion of the Capital Expenditures was added to the milling (ore based) costs. For 2009 this amount was US$76.40 million and was factored using 6% discount rate, to $64.13 million, based on LOM plan of 92.55 million tonnes, this would contribute $0.69 per tonne to the ore based costs.

The economic modeling parameters are based on the operating costs determined since the start of operations at Rosebel. An adjustment to mining cost was included to represent increased haulage cost with increasing depth in pits.

A royalty payment of 2.25% of the gold produced is payable to the Surinamese government. This royalty is valid up to 425$/oz, above this, an additional royalty of 6.5% is applied. For example, for a gold price of $ 600, the total royalty payable is $24.88 per ounce, comprised of 2.25% x $600 or $13.50 per ounce and 6.5% x ($600 – $425) or $11.38 per ounce.

Metallurgical recovery of gold is based on five years of ore processing as follows: Soft Rock = 96.0%; Transition Rock = 94.0%; Hard Rock = 92.0%.

This reserve estimate is based on a gold price of US$850 per troy ounce and on an exchange rate of US$1.00 = 2.78 Suriname Dollars.

1.2.7. Mineral Resource Estimation

The official resource estimation for the Rosebel project as at December 31st 2009 is summarized on Table 1.1.

1.2.8. Mineral Reserve Estimation

A Mineral Reserve is the economically mineable part of a Measured or Indicated Mineral Resource. Rosebel Mineral Reserves as at December 31st, 2009 are listed on Table 1.2.

Rosebel Gold Mines N.V.
Measured and Indicated Resources - 31 December 2009 (inside pit shells)
$1000 / ounce including Adjustment Factors

Deposit	Cutoffs			Laterite			Saprolite			Transition			Rock			Total			Gold
	Laterite & Saprolite	Transition	Hard Rock	Tonnes (000)	Au (g/t)	Au (K oz)	Tonnes (000)	Au (g/t)	Au (K oz)	Tonnes (000)	Au (g/t)	Au (K oz)	Tonnes (000)	Au (g/t)	Au (K oz)	Tonnes (000)	Au (g/t)	Au (K oz)	Ounces contained
Koolhoven/Bigi	0.25	0.30	0.35	356	0.67	8	3,761	0.84	100	17,054	0.99	553	1,758	1.17	66	22,929	1.0	727	727,000
JZone	0.25	0.30	0.35	966	0.66	20	1,491	0.71	34	5,298	0.86	147	7,132	0.91	208	14,887	0.9	409	409,000
Pay Caro	0.24	0.28	0.33	599	0.62	12	752	0.63	15	4,054	0.79	103	29,345	1.04	981	34,750	1.0	1,111	1,111,000
East Pay Caro	0.25	0.29	0.34	573	0.70	13	818	0.89	23	2,797	0.94	84	13,579	1.05	458	17,767	1.0	579	579,000
Mayo	0.27	0.31	0.36	4,539	0.72	106	12,500	1.00	405	5,213	0.93	156	31,477	1.03	1041	53,729	1.0	1,708	1,708,000
Roma	0.26	0.30	0.35	748	0.84	20	1,382	0.83	37	924	1.15	34	2,576	1.09	90	5,630	1.0	181	181,000
Royal Hill	0.26	0.30	0.35	1,647	0.72	38	1,390	0.94	42	2,005	1.04	67	48,674	1.08	1687	53,717	1.1	1,834	1,834,000
Rosebel	0.27	0.32	0.37	3,097	0.71	71	5,216	0.90	151	3,316	0.86	92	7,758	1.14	284	19,388	1.0	597	597,000
TOTAL				12,525	0.72	288	27,312	0.92	808	40,661	0.94	1,235	142,299	1.05	4,815	222,797	1.0	7,147	7,147,000
Stockpiles																3,192	0.8	84	84,000
GRAND TOTAL	M+I Resources			12,525	0.7	288	27,312	0.9	808	40,661	0.9	1,235	142,299	1.1	4,815	225,989	1.0	7,231	7,231,000

Rosebel Gold Mines N.V.
Inferred Resources - 31 December 2009 (inside pit shell limit)
$1000 / ounce including Adjustment Factors

Deposit	Cutoffs			Laterite			Saprolite			Transition			Rock			Total			Gold
	Laterite & Saprolite	Transition	Hard Rock	Tonnes (000)	Au (g/t)	Au (K oz)	Tonnes (000)	Au (g/t)	Au (K oz)	Tonnes (000)	Au (g/t)	Au (K oz)	Tonnes (000)	Au (g/t)	Au (K oz)	Tonnes (000)	Au (g/t)	Au (K oz)	Ounces contained
Koolhoven/Bigi	0.25	0.30	0.35	128	0.92	4	120	1.29	3	169	1.73	10	5	1.29	0	422	1.4	18	18,000
JZone	0.25	0.30	0.35	488	0.54	8	179	0.49	3	99	1.17	4	182	1.30	8	948	0.7	23	23,000
Pay Caro	0.24	0.28	0.33	387	0.44	5	138	0.48	2	249	0.82	7	4,919	1.06	168	5,693	1.0	183	183,000
East Pay Caro	0.25	0.29	0.34	399	0.44	6	131	0.70	3	172	0.50	3	1,012	0.93	30	1,714	0.8	42	42,000
Mayo	0.27	0.31	0.36	543	0.49	9	661	0.57	12	156	0.59	3	6,842	1.30	286	8,203	1.2	310	310,000
Roma	0.26	0.30	0.35	872	0.61	17	67	0.70	2	12	0.89	0				951	0.6	19	19,000
Royal Hill	0.26	0.30	0.35	815	0.54	14	71	0.48	1	16	0.48	0	6,226	1.42	283	7,128	1.3	299	299,000
Rosebel	0.27	0.32	0.37	450	0.41	6	700	0.68	15	251	0.74	6	709	0.91	21	2,110	0.7	48	48,000
TOTAL				4,081	0.53	69	2,067	0.65	43	1,126	0.89	32	19,896	1.24	796	27,169	1.1	941	941,000

Table 1.1. 2009 Rosebel Mineral Resources.

Rosebel Gold Mines N.V.
Mineral Reserve Estimates - December 31, 2009
$850 / ounce including Adjustment Factors

Proven Mineral Reserves

Ore:

Deposit	Cutoffs			Laterite			Saprolite			Transition			Rock			Total		
	Laterite & Saprolite	Transition	Hard Rock	Tonnes (000)	Au (g/t)	Au (K oz)	Tonnes (000)	Au (g/t)	Au (K oz)	Tonnes (000)	Au (g/t)	Au (K oz)	Tonnes (000)	Au (g/t)	Au (K oz)	Tonnes (000)	Au (g/t)	Au (K oz)
Koolhoven/Bigi	0.30	0.35	0.40	183	0.76	4	2,395	0.92	70	9,277	1.05	320	175	1.30	7	12,030	1.0	401
JZone	0.30	0.35	0.40															
Pay Caro	0.28	0.33	0.39	216	0.68	5	209	0.83	6	1,918	0.90	55	10,579	1.11	380	12,923	1.1	445
East Pay Caro	0.29	0.34	0.39	283	0.66	6	368	0.68	8	1,334	0.71	31	4,864	0.77	121	6,849	0.7	165
Mayo	0.31	0.36	0.42	2,271	0.85	62	7,272	1.12	265	2,856	1.04	95	8,468	1.09	295	20,868	1.1	718
Roma	0.30	0.35	0.41															
Royal Hill	0.30	0.35	0.41	216	0.87	6	695	1.12	25	1,273	1.20	49	23,632	1.18	894	25,816	1.2	974
Rosebel	0.32	0.37	0.43															
TOTAL				3,171	0.82	83	10,939	1.1	374	16,658	1.0	549	47,719	1.1	1,697	78,486	1.1	2,703
Stockpiles																3,192	0.8	84
GRAND TOTAL	Proven			3,171	0.8	83	10,939	1.1	374	16,658	1.0	549	47,719	1.1	1,697	81,678	1.1	2,787

Probable Mineral Reserves

Ore:

Deposit	Cutoffs			Laterite			Saprolite			Transition			Rock			Total		
	Laterite & Saprolite	Transition	Hard Rock	Tonnes (000)	Au (g/t)	Au (K oz)	Tonnes (000)	Au (g/t)	Au (K oz)	Tonnes (000)	Au (g/t)	Au (K oz)	Tonnes (000)	Au (g/t)	Au (K oz)	Tonnes (000)	Au (g/t)	Au (K oz)
Koolhoven/Bigi	0.30	0.35	0.40	57	0.7	1	720	1.0	22	2,607	1.2	103	151	1.3	6	3,536	1.2	133
JZone	0.30	0.35	0.40	665	0.8	16	1,254	0.8	31	4,047	0.9	124	2,177	1.0	72	8,143	0.9	243
Pay Caro	0.28	0.33	0.39	106	0.6	2	277	0.7	6	583	1.0	18	5,869	1.3	245	6,834	1.2	272
East Pay Caro	0.29	0.34	0.39	182	1.0	6	373	1.2	14	1,232	1.3	51	4,881	1.4	222	6,668	1.4	293
Mayo	0.31	0.36	0.42	1,451	0.7	34	3,395	1.0	114	887	1.1	32	5,117	1.3	210	10,850	1.1	390
Roma	0.30	0.35	0.41	704	0.9	20	1,135	0.9	32	721	1.3	30	1,669	1.2	64	4,229	1.1	146
Royal Hill	0.30	0.35	0.41	319	0.7	8	301	1.0	9	288	0.9	8	8,753	1.2	325	9,661	1.1	350
Rosebel	0.32	0.37	0.43	2,349	0.8	60	4,489	1.0	141	2,665	0.9	80	5,048	1.2	200	14,552	1.0	480
TOTAL				5,833	0.8	147	11,945	1.0	370	13,031	1.1	446	33,665	1.2	1,343	64,474	1.1	2,307
Stockpiles																		
GRAND TOTAL	Probable			5,833	0.8	147	11,945	1.0	370	13,031	1.1	446	33,665	1.2	1,343	64,474	1.1	2,307

Table 1.2 . 2009 Rosebel Mineral Reserves.

2. INTRODUCTION AND TERMS OF REFERENCE

2.1. Introduction and Terms of Reference

This report is to comply with disclosure and reporting requirements set forth in the Toronto Stock Exchange manual, National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101), Companion Policy 43-101CP to NI 43-101 and Form 43-101F1 of NI 43-101.

The Rosebel project has been operating commercially since February 2004. The project is owned by Rosebel Gold Mines NV, a Surinamese company and subsidiary of IAMGOLD Corporation. This report is prepared for IAMGOLD and the purpose is to update our project status with the new reserves numbers as of December 2009.

2.2. Qualifications and Experience

Gabriel Voicu, professional geologist (OGQ #367) and author of this technical report has the appropriate relevant qualifications and experience to be considered as Qualified Person as defined in Canadian National Instrument 43-101 Standards of Disclosure for Mineral Projects.

His work experience includes more than fifteen years an exploration and mine geologist in open pit mines including acting as Geology and Mine Exploration Superintendent for Rosebel Gold Mines since 2007. The author is also an Associate Professor with the Atmospheric and Earth Sciences Department, Université du Québec à Montréal.

2.3. Principal Sources of Information

The source of information for this technical report is based on data obtained in the Feasibility Study published in August 2002 and Mineral Reserve Reports published between 2004 and 2009.

The information and data contained in this technical report come from:

- Cambior Inc. – Rosebel Gold Project Feasibility Study, August 2002.
- Rosebel Gold Mines N.V. – Mineral Reserve Report, January 2004
- Rosebel Gold Mines N.V. – Mineral Reserve Report, January 2005

- Rosebel Gold Mines N.V. – Mineral Reserve Report, July 2005
- Rosebel Gold Mines N.V. – Mineral Reserve Report, January 2006
- Rosebel Gold Mines N.V. – Mineral Reserve Report, January 2007
- Rosebel Gold Mines N.V. – Mineral Reserve Report, January 2008
- Rosebel Gold Mines N.V. – Mineral Reserve Report, January 2009
- Rosebel Gold Mines N.V. – Mineral Reserve Report, December 2009

These documents were prepared by or under the supervision of geologists and engineers who are Qualified Persons as defined in Canadian National Instrument 43-101. In this sense, the information should be considered reliable.

In addition, the following material stored on the IT Rosebel network has been used:

1- Gems 6.1.4 database containing the block models with different attributes

2- Drill hole, auger, banka database (Gems 6.1.4) containing collar location, down-hole survey, assay, geology and geotechnical data

3- Three-dimensional models of the interpreted ore zones, topography and lithology

4- Grade block models

5- Quality control data

6- Bulk density data

7- Cost parameters for calculation of economic cut-offs

8- Historical production and reserves

9- Description of the metallurgical process

The following Rosebel Gold Mines personnel participated in the preparation of this report:

Judith Saint-Laurent – Resource Geologist

Harold Brisson – Resource Geologist

Caroline Daoust – Geologic Database Supervisor

Bjarne Westin – Senior Special Project Geologist

Pascal Lehouiller – Senior Mine Geologist

Peter Pecek – Senior Mine Engineer

Ian Horne – Senior Mine Engineer

Yohann Bouchard – Engineering Superintendent

Aldo Crino – Exploration Manager Suriname

John Grignon – Mill Manager

3. RELIANCE ON OTHER EXPERTS

The author reviewed the Rosebel Gold Mines information regarding the current status of legal title and property agreements.

The author is not a qualified person with respect to environmental laws, regarding issues addressed in this report – Environmental Liabilities and relied upon Mr. Ross Gallinger, Senior Vice President, Health, Safety and Sustainability, IAMGOLD Corporation.

Based on his experience, the author considers that the presented information should be considered reliable.

4. PROPERTY DESCRIPTION AND LOCATION

The Rosebel Mine concession (468_02) covers an area of 170 square kilometres in north central Suriname at a latitude of 55 ° 25’ North and a longitude of 55 ° 10’ West. The property lies in the district of Brokopondo, between the Suriname River to the east and the Saramacca River to the west, approximately 80 kilometres south of the capital city of Paramaribo.

The rights to the Rosebel mine property were initially held through a Right of Exploration (“ROE”) granted by the Ministry of Natural Resources, valid and renewable for two-year periods. The ROE was renewed and extended for two years from February 25, 2002, in favour of Golden Star. On May 16, 2002, Golden Star assigned, conveyed and transferred its Rosebel ROE to Rosebel Gold Mines N.V. (“RGM”). Finally on December 16, 2002, RGM was granted a 25-year renewable Right of Exploitation for the Rosebel mine from the Government of Suriname, following the Government’s approval of the updated feasibility study and environmental impact assessment.

RGM is also the holder of three exploration concessions referred to as Rights of Exploration (“ROE”). ROE’s for the adjacent Headley’s Reef, Thunder Mountain and Triangle properties were in the past renewed in favour of Golden Star and thereafter assigned, conveyed and transferred to RGM simultaneously with the Rosebel Mine concession. Headley’s Reef and Thunder Mountain ROE’s have been renewed on July 3, 2008 for a period of three years. Triangle ROE is in the application process of its renewal for a period of two years. The total area of the three ROE’s is 62,355 hectares and entirely surrounds the Rosebel Mining concession (Fig. 4.1).

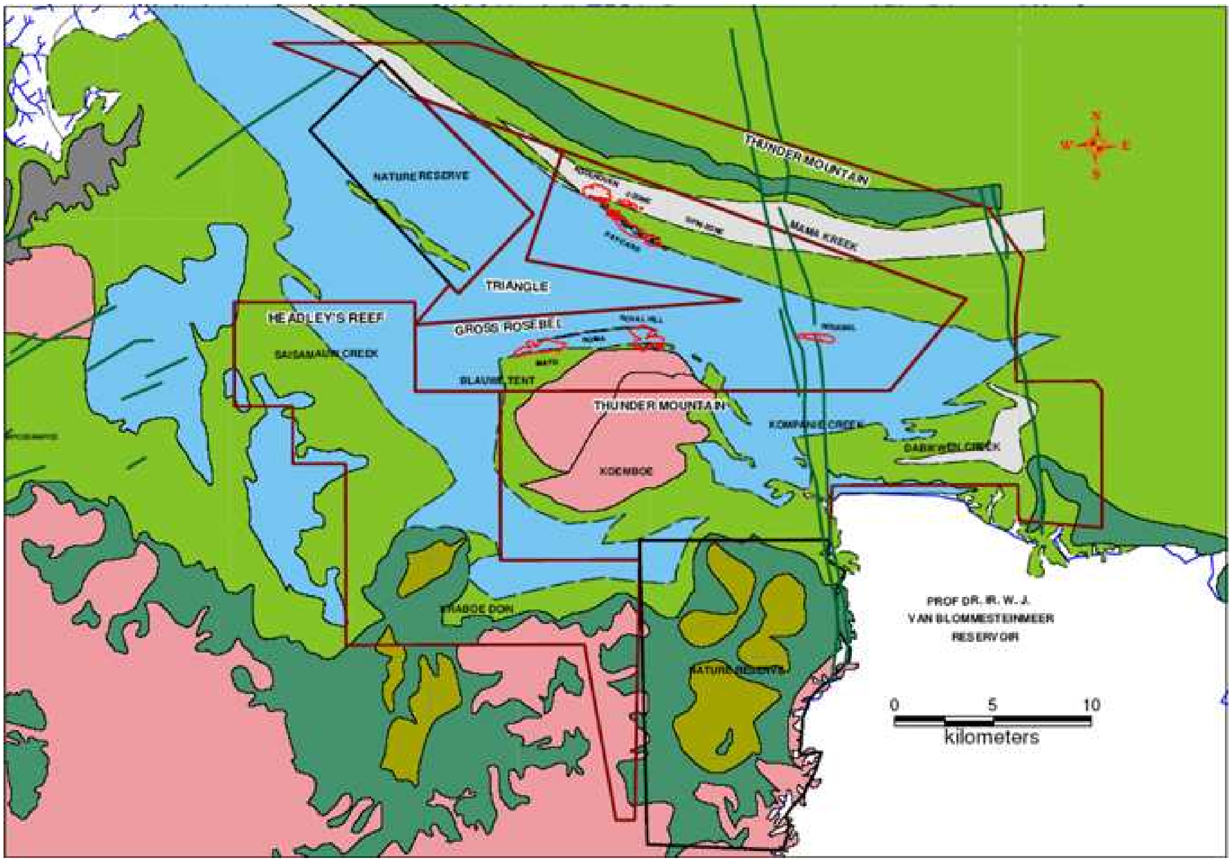


Figure 4.1. Rosebel mine concession and adjacent Rights of Exploration (ROE).

5. ACCESSIBILITY, CLIMATE, LOCAL RESOURCES AND INFRASTRUCTURE

5.1. ACCESSIBILITY

There are presently two access routes from Paramaribo to the Rosebel project. One route utilizes a 30 kilometre paved road which connects Paramaribo to Paranam. From Paranam, a recently paved road courses south, following the Afobaka road. From there, an unpaved road courses south and west to reach the property. The other route is a paved road which connects Paramaribo to the international airport at Zanderij. A gravel road connects Zanderij to the Afobaka road halfway between Paranam and Afobaka. The route then follows the Afobaka, Brownsweg and Nieuw KoffieKamp roads until reaching the property access road. Travel distance for both routes from Paramaribo is approximately 100 kilometres.

5.2. TOPOGRAPHY, VEGETATION AND CLIMATE

Rosebel is located in an area of small hills covered with tropical rain forest and separated by flat-lying savannah with a light cover of low trees, shrubs and grass. The maximum elevation reaches 150 meters above the sea level.

The climate of Suriname is classified as tropical: warm during the entire year with the mean temperature of the coldest month being higher than 20 degrees Celsius. The average monthly rainfall is greater than 60 millimetres in the driest month(s). Like much of Suriname, the Rosebel property is characterized by consistently warm temperatures and high humidity with little seasonal variation.

Suriname weather is dictated mainly by the northeast and southeast prevailing winds of the Inter-Tropical Convergence Zone (“ITC” zone, also known as the “Equatorial Trough”). The ITC zone passes over Suriname twice a year and results in four seasons:

- early February to late April, a short dry season;
- late April to mid-August, a long rainy season;
- mid-August to early December, a long dry season; and,

- early December to late February, a short rainy season.

Weather data has been regularly collected on the Rosebel property since 2003 using an automated weather station.

The average annual precipitation for Rosebel is estimated to be 2,251 millimetres per year, while the mean annual temperature is 25.0 degrees Celsius. The daily fluctuation in temperature in the interior of Suriname, including the Rosebel area, is approximately 10 to 12 degrees Celsius. The average monthly relative humidity at Rosebel ranges from 84.8 percent in February to 93.5 percent in June, with an annual average of 89 percent. This relative humidity trend results from rainfall and temperature changes.

The most common wind direction is from the east (approximately 20 percent of the time) followed by the southeast (approximately 9 to 14 percent of the time). The Rosebel site does not experience sustained strong winds, with hourly average wind speeds rarely exceeding 5.0 metres per second. The most common recorded wind speed ranges from 1.0 to 2.5 metres per second.

5.3. Local Resources and Infrastructure

The Rosebel area currently hosts the small village of Nieuw Koffie Kamp, located approximately two kilometres from the old exploration base camp and at about one kilometre from the Royal Hill pit. The village consists of approximately 500 permanent inhabitants belonging primarily to the maroon group, who are descendants of African slaves.

The economy of the village remains dependent on the Surinamese coastal economy. Principal activities include subsistence agriculture on relatively poor land, small-scale gold mining, forestry and trade.

The village, originally named Koffie Kamp, relocated to its present site in 1964 when the previous site was flooded during development of the Brokopondo hydroelectric project. Relations between project management and the villagers have occasionally been strained due primarily to illegal mining activities being carried out on the Rosebel property by the villagers and others.

Other than the road between Paramaribo and the mine site, the local infrastructure consists of site roads that include access from the main gate to the camp, pits, tailing area, the process plant area, and administration building area (Figure 5.1). These roads are mostly built with laterite and are typically 10 metres wide. Culverts are installed and ditching is done to provide adequate drainage.

An existing airstrip with an approximate length of 1.2 kilometres is used for emergency evacuation and gold shipment. The airstrip is located 6 kilometres from the administration building. A new airstrip is under construction.

The camp complex is located approximately 0.5 kilometres to the south of the process plant and truck shop/administration building. The camp complex includes a kitchen, recreation area, camp offices and different types of dormitories.

Surface rights in Suriname are distinct from mining rights and must be acquired separately. The holder of a Right of Exploration or a Right of Exploitation is entitled to use the surface to conduct mining operations including the construction of facilities required for such operations. RGM currently holds the surface rights to the Rosebel property area.

The Mineral Agreement (as defined in Section 19.5) authorizes the use of all available water sources on or in the vicinity of the Rosebel property area for project purposes, subject to applicable laws, provided that the quality and quantity of existing water sources are not materially diminished. There are no laws currently in force in Suriname governing water rights.

Electrical energy is purchased directly from the Surinamese government. Power is delivered from the Afobaka hydroelectric generating station. This plant is owned and operated by Suralco, a subsidiary of Alcoa.

RGM’s personnel consists of approximately 1,500 people, including national and expatriate personnel. Most national personnel comes from the villages surrounding the Rosebel property or from Suriname’s capital city of Paramaribo.

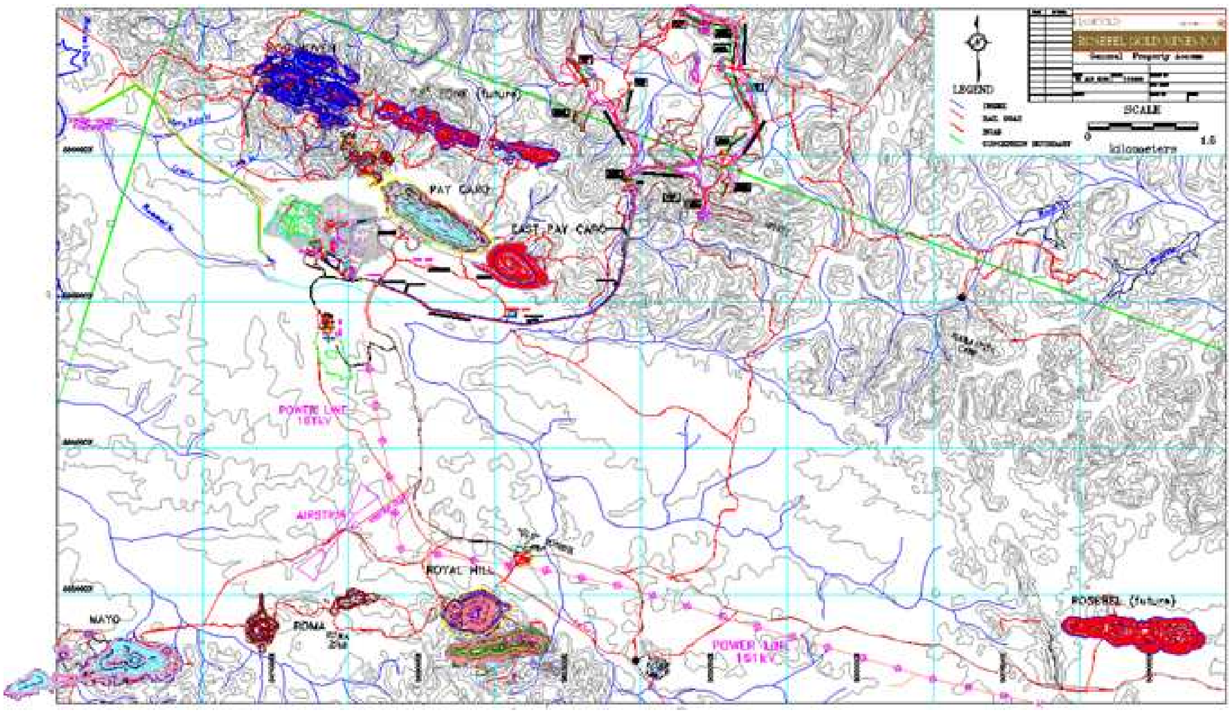


Figure 5.1 Rosebel mining concession

6. HISTORY

6.1. HISTORY OF PROJECT

Documentation on the history of gold production from the Rosebel district is fragmentary. Records were either not kept or, in most cases, have disappeared. Gold was first discovered in the area in 1879, with approximately 600 small scale miners reported to be working on the property. Since that time, approximately half of the recorded production of Suriname has been produced from the district.

Between 1885 and 1939, several large companies exploited alluvial material, surface deposits and veins. Various methods of mechanized mining were tried, including dredges, stamp mills and hydraulicing, with varying degrees of success. The larger companies eventually sub-leased concessions to miners, who continued exploitation using manual recovery methods. Some of the more prominent companies that operated in the area and for which records still exist, are listed below.

Guyana Gold Placer Company operated dredges in the Nieuw Foto and Groote Louis Kreeks of the Koolhoven area circa 1910. The company sub-leased some ground in the Koolhoven area to an American group, who underground-mined on a series of quartz veins up to 5 metres wide. Production was said to include a "nugget" of nearly 8 ounces.

De Jong Brothers owned the Royal Hill area, which was mined manually by adits, shafts and open cuts during the 1920's and early 1930's. Records indicate that from 1924 to 1933 the average output was 1,600 ounces of gold per year.

White Water Mines Ltd. acquired the Royal Hill area from De Jong in 1935. Widespread veins were mined by shafts and adits, with ore being carried to a central mill by narrow-gauge railway. Operations ceased at the start of the Second World War in 1939. Production records for this period have not been recovered.

Van Emden Gold Mines Ltd. operated three mines in the area in the 1930's: Mayo, Koolhoven and Donderbari. These were the best-planned, operated and capitalized operations to date, using large-size ball and stamp mills and extensive narrow-gauge railway systems.

The Suriname Government operated the deposit intermittently from 1950 to 1952, reporting some production from Royal Hill. Since that time, several companies or individuals have carried out resource estimations on the Rosebel gold deposits, to various levels of reliability.

Surplacer – 1976

In 1974, the present property was granted to Surplacer, a joint venture between Placer Development Ltd. of Vancouver and the Surinamese Government. The exploration program identified several kilometre-long gold anomalies, located along two major trends, one in the north and the other in the south of the area. Detailed follow-up work, involving 900 hand auger holes, 4 kilometres of bulldozer trenches and 43 reverse circulation drill holes, partially delineated surficial and near-surface gold mineralization: the Royal Hill, Mayo and Rosebel areas in the south and Pay Caro in the north. When Placer Development terminated the joint venture and left Suriname in 1977, the resource estimate indicated nearly 700,000 ounces of gold (Table 6.1). However, the estimated resources use other categories than the ones set out by NI43-101. “Probable resources” as stated by Surplacer may eventually be the equivalent of “indicated resources”.

DEPOSIT	Tonnage (000 t)	Grade (g/t Au)	Ounces Au (000 oz)
Royal Hill	5,421	2.4	413
Mayo	1,012	1.8	57
Rosebel	1,939	2.3	143
Pay Caro	530	2.2	38
Alluvium	472	2.5	37
Total	9,374	2.3	688

Table 6.1 . Probable laterite and saprolite-hosted resources (circa 1977).

Grassalco – 1984

On July 26, 1979 the Rosebel property was awarded to Grassalco, who carried out a new resource estimate using 1,500 hand auger holes and excluded the Placer data (Table 6.2). Grassalco was forced to abandon operations in the middle of 1985, due to an unstable political situation.

DEPOSIT	Tonnage (000 t)	Grade (g/t Au)	Ounces Au (000 oz)
Royal Hill	1,844	2.0	117
Mayo	628	1.7	35
Rosebel	567	2.2	39
Alluvium	472	2.5	37
Total	3,511	2.0	228

Table 6.2. Probable laterite and saprolite-hosted resources (circa 1979).

Smith – 1987

The last work performed on the Rosebel property prior to its acquisition by Golden Star was a 1987 Ph.D. thesis by I.H. Smith of the University College of Cardiff, Wales (Table 6.3). The thesis evaluated resources from several gold deposits in Suriname using a statistical analysis of published data.

DEPOSIT	Tonnage (000 t)	Grade (g/t Au)	Ounces Au (000 oz)
Royal Hill	2,029	1.33	85
Mayo	690	1.5	34
Rosebel	624	1.4	29
Total	3,343	1.4	148

Table 6.3 . “Mineable” and probable laterite-hosted resources (circa 1987).

6.2. OWNERSHIP

Golden Star acquired the Right of Exploration to the Rosebel property pursuant to a Preliminary Mineral Agreement between Golden Star, Grassalco and the Government of Suriname dated May 8, 1992. A Mineral Agreement between Golden Star, Grassalco and the Government of Suriname was signed on April 7, 1994 and replaced the 1992 agreement. In accordance with the 1994 Mineral Agreement, Golden Star was granted the Right of Exploration to the Rosebel property for five years.

Golden Star entered into an agreement with Cambior Inc. on June 7, 1994, granting Cambior the option to earn an undivided 50 percent of Golden Star’s interest in the 1994 Mineral Agreement pertaining to the Rosebel property. This agreement provided that Cambior could exercise its option by funding approximately $6.1 million in exploration and development expenditures on the Rosebel property by June 30, 1996.

A Feasibility Study and an Environmental Impact Statement were filed with the Government of Suriname in May 1997. Following additional drilling on the property, a revised Feasibility Study was submitted to the Government of Suriname in December 1997. Between 1998 and 2000, the Rosebel Project remained on care and maintenance.

In December 2000, a Pre-feasibility Study was delivered to the Ministry of Natural Resources that considered only the mining and processing of the soft rock and transitional ore portions of the Rosebel deposits. The project's estimated capital expenditure was reduced to $80 million from the $175 million contemplated in the original 1997 Feasibility Study. A second Feasibility Study was completed in August 2002.

On October 26, 2001, Golden Star agreed to sell its 50 percent interest in the Rosebel property to Cambior for a cash consideration of $8 million and a gold price participation right on future production. $5 million was paid at closing (May 2002) and the remainder was paid in three equal installments over a three year period. Under its gold price participation right, Golden Star would receive a quarterly payment of an amount equal to 10 percent of the excess, if any, of the average quarterly market price above $300 an ounce for gold production from Rosebel's soft and transitional rock portions and above $350 an ounce from Rosebel's hard rock portion, up to a maximum of 7 million ounces produced. In addition, Golden Star transferred its rights in the adjacent Headley's Reef and Thunder Mountain exploration properties to Rosebel Gold Mines N.V., a wholly owned subsidiary of Cambior Inc.

In 2004, Golden Star Resources sold the royalty interest in Rosebel production to Euro Ressources SA (formerly Guyanor Ressources SA).

On November 7, 2006, IAMGOLD Corporation acquired 100 percent of Cambior Inc. (the previous owner of Rosebel Gold Mines N.V.) through a court approved plan of arrangement.

In December 2008, IAMGOLD Corporation acquired 84.55 percent of the current share capital of Euro Ressources SA.

7. GEOLOGICAL SETTING

7.1. Regional Geology

The Rosebel concession lies within a Paleoproterozoic greenstone belt of the Guiana Shield that extends from the Amazon River in Brazil to the Orinoco River in Venezuela and covers an area of more than 900,000 square kilometres.

The Paleoproterozoic of the Guiana shield was cratonized during the Trans-Amazonian orogeny. The Proterozoic part of the shield becomes progressively younger towards the southwest; with TTG-greenstone belts in the north and Late Paleoproterozoic to Mesoproterozoic volcanic, intrusive and sedimentary rocks in the southernmost part separated by a central granitoid terrane (Fig. 7.1). The geological evolution of the Guiana Shield is divided in four distinct stages: formation of the Archean basement – Main Trans-Amazonian orogeny – Late Trans-Amazonian orogeny – subsequent Proterozoic and Paleozoic anorogenic events.

The Main Trans-Amazonian orogeny (D_1), constrained between 2.26Ga and 2.08Ga, was a crustal growth event that generated the TTG-greenstone belts found in the north of the shield. The Main Trans-Amazonian orogeny involved south-verging subduction during north-south convergence of the North Amazonian and the West African cratons; the result being the consumption of juvenile crust and consequent generation of greenstone-TTG belts and associated sedimentation. Progressively, the north-south convergence is inferred to have switched to NE-SW oblique convergence, generating regional sinistral strike-slip movement and syntectonic felsic plutonism (D_{2a}). Development of strike-slip structures led to the formation of pull-apart basins along the North Guiana Trough that were synchronously filled with detrital sediments. The Late Trans-Amazonian event (D_{2b}), extending from 2.07Ga to 1.93Ga, was associated with extreme crustal stretching of the Guiana Shield under continued sinistral transpression. Resultant mantle upwelling is interpreted to have generated the high-grade metamorphic (granulitic) rocks and charnockitic intrusives of the Bakhuis Horst in Western Suriname.

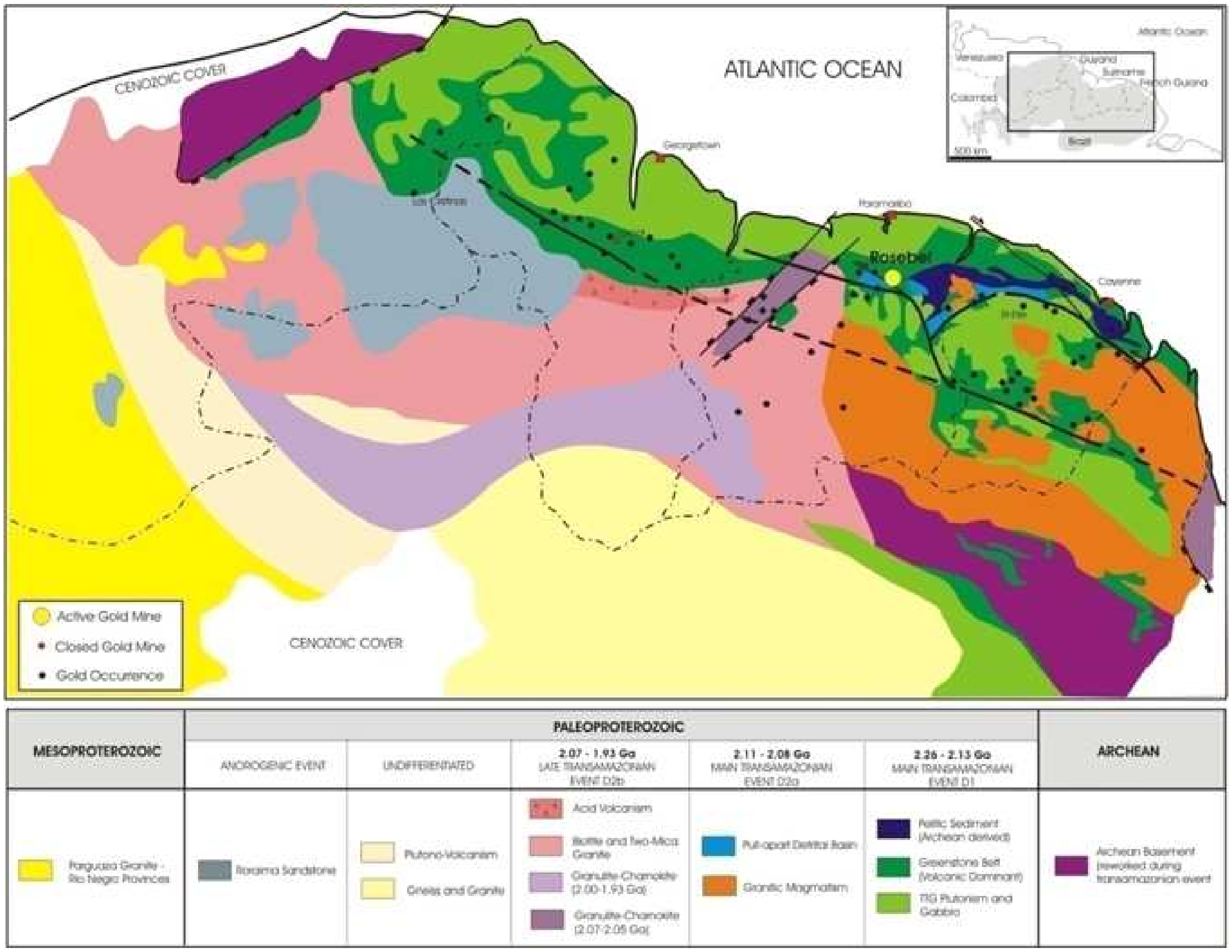


Figure 7.1. Simplified geological of the map Guiana Shield.

The general lithostratigraphic succession of the greenstone belts that host most of the gold mineralization in the Guiana Shield (including the Rosebel deposit) comprises: 1) a lower sequence of tholeiitic basalt overlain by, 2) a mainly calc-alkaline volcanic suite including felsic to mafic members and, 3) an upper sedimentary succession comprising greywacke, pelite, chert and conglomerate. In Suriname, sedimentary and volcanic units of the greenstone belts form the Marowijne Supergroup, which can be subdivided into two formations: the Paramaka and the Armina Formations. The Paramaka Formation mainly consists of volcanic rocks, whereas the Armina Formation is characterized by flysch sequences of greywacke and mudstone. The volcanic succession is associated spatially and temporally to tonalite-trondjhemite-granodiorite (TTG) plutonism.

The Marowijne Supergroup is unconformably overlain by an arenitic/conglomeratic sedimentary sequence that is locally termed the Rosebel Formation. This sedimentary sequence is interpreted to have been deposited in a series of intracontinental pull-apart basins that developed along major sinistral strike-slip structures during the later stages of the Trans-Amazonian orogeny. Granitic magmatism occurred synchronously with the formation of these basins in the Eastern part of the Guiana Shield (Suriname, French Guiana and Brazil).

The Western and Southern areas of Suriname are dominated by Late Trans-Amazonian rocks. These include granulitic and charnockitic rocks of the Bakhuis Horst, formed during a high-grade tectono-metamorphic and magmatic event. Younger, post-Trans-Amazonian rocks of Suriname include remnants of the sub-horizontal Middle Proterozoic Roraima Formation, younger Proterozoic alkali granites and the Permo-Triassic Apatoe dyke swarm, the latter coinciding with the initial stages of the Atlantic Ocean opening.

The entire Guiana Shield has undergone prolonged chemical weathering under a humid, tropical paleoclimate that may have started as far back as the Cretaceous period. Weathering has produced a laterite/saprolite profile to depths of up to 100 metres below surface. The thick cover of rain forest vegetation has protected the soil from erosion, and the thin soil profile is generally preserved. The chemical effects of the deep weathering include leaching of mobile constituents (alkali and alkali earths), partial leaching of SiO_2 and Al_2O_3, formation of stable secondary minerals (clays, Fe-Ti and Al-oxides), mobilization and partial precipitation of Fe and Mn and the concentration of resistant minerals (zircon, magnetite, quartz).

7.2. Local Geology

The Rosebel deposits are hosted in the Marowijne Supergroup volcano-sedimentary sequence and overlying Rosebel Formation. Five distinct lithological associations have been identified: 1) felsic to mafic volcanic rocks, 2) deep-water sediments, 3) shallow-water sediments, 4) felsic intrusions and, 5) late diabase dykes. Economic gold mineralization has been recognized in the sedimentary and volcanic associations. The felsic intrusions contain minor gold occurrences and the dykes are totally barren.

Regional metamorphism is lower-greenschist to greenschist facies. The main regional fabric varies from east-west in the southern part of the property to WNW-ESE in the north and follows the regional tectonic grain as illustrated on Figure 7.2. Two phases of deformation are recognized on the property. The poorly preserved first phase (D_1) is characterized by steeply plunging, tight to isoclinal folds occurring only in the volcanic and deep-water sediments. The second phase (D_2) affected all rock units except the diabase dykes and is defined by open to close folding, development of the regional foliation and faulting. The introduction of gold mineralization is interpreted to have occurred late in the second deformation phase; gold-bearing shear veins within faults and adjacent mineralized tension veins are typically not overprinted by younger deformation textures.

Gold mineralization occurs in three different regional trends on the Rosebel property; North, Central and South (Figs. 7.3, 7.4 and 7.5). The northern trend follows a WNW-ESE orientation and includes the J-Zone – Koolhoven pits and the Pay Caro – Pay Caro East pits which respectively occur along the northern and southern contacts of a volcanic sequence. The east-west striking central trend contains the Rosebel pit and the similarly orientated southern trend hosts the Mayo, Roma and Royal Hill pits.

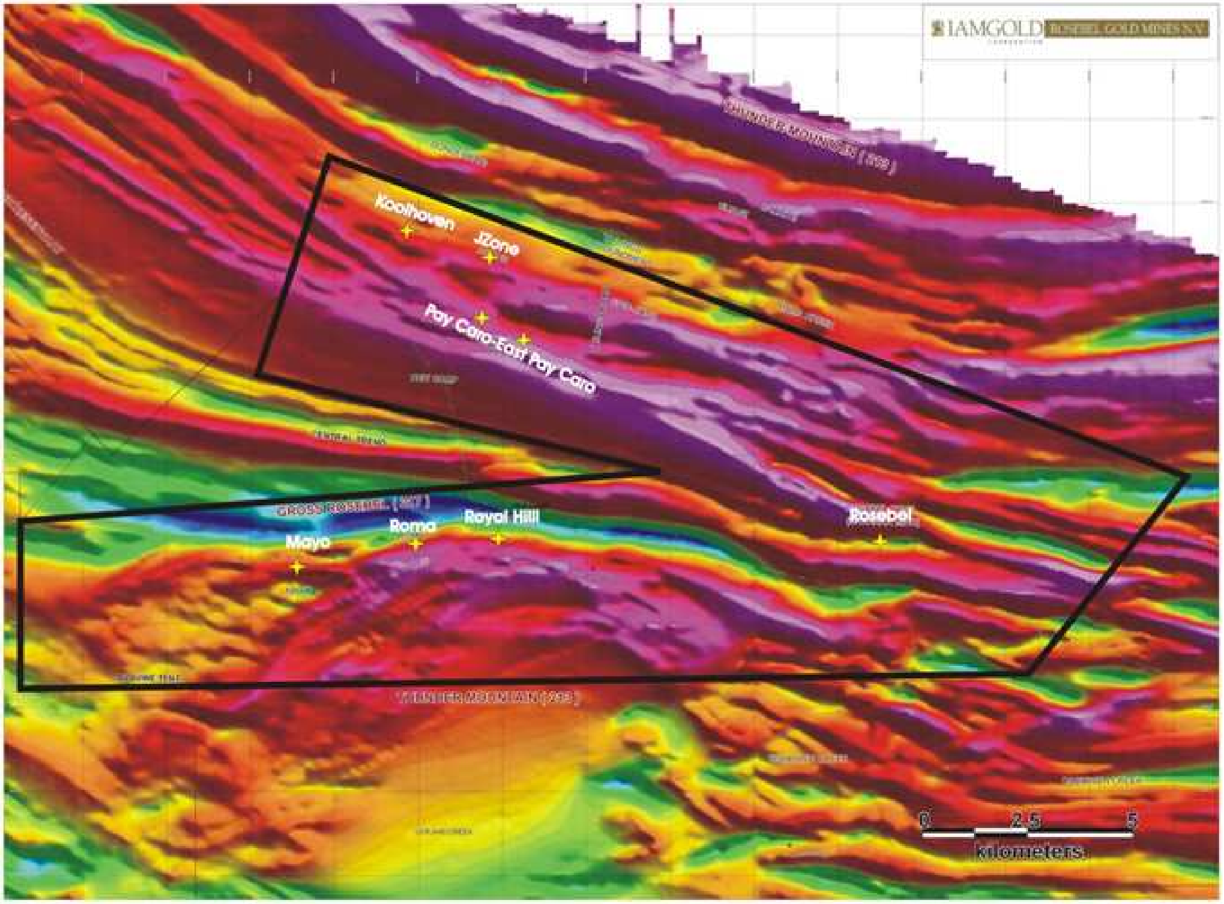


Figure 7.2. Structural fabric of the Rosebel property highlighted in the aeromagnetic data.

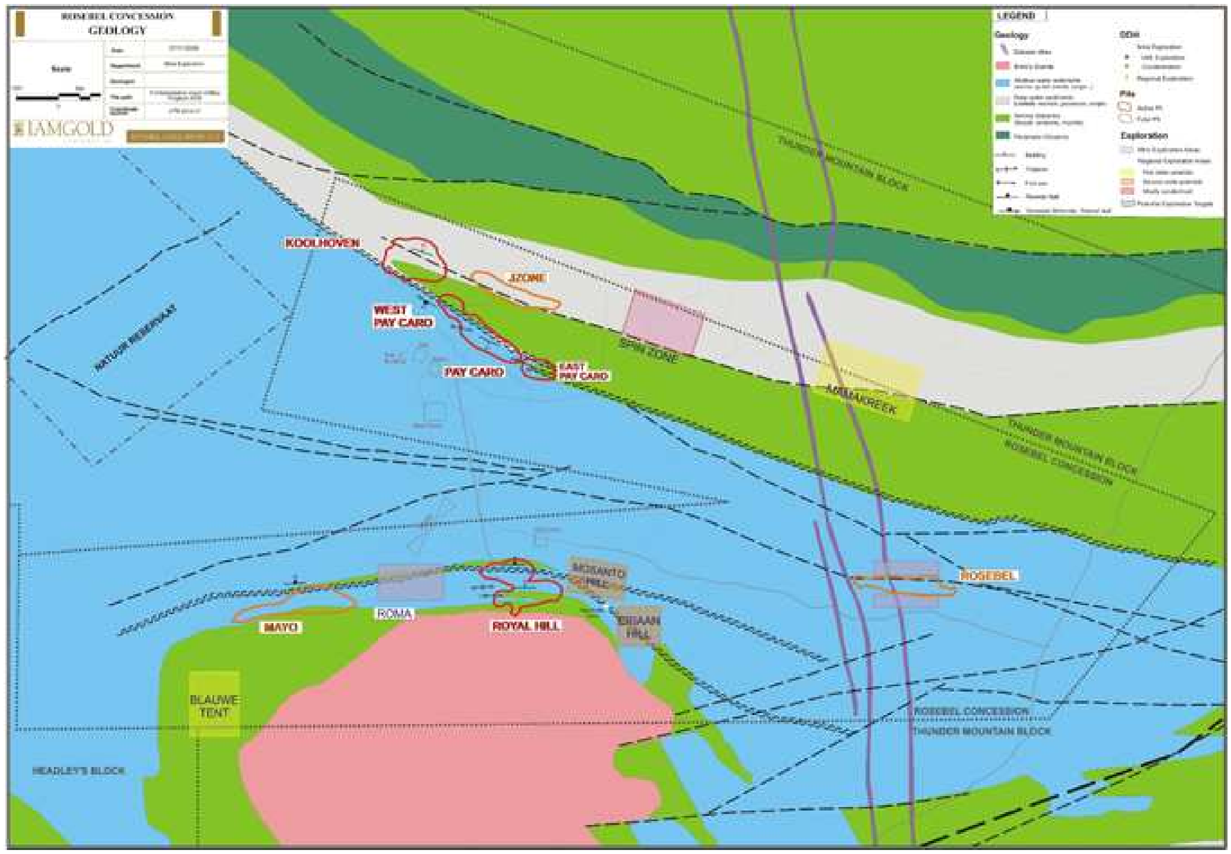


Figure 7.3. Geological map of the Rosebel property.

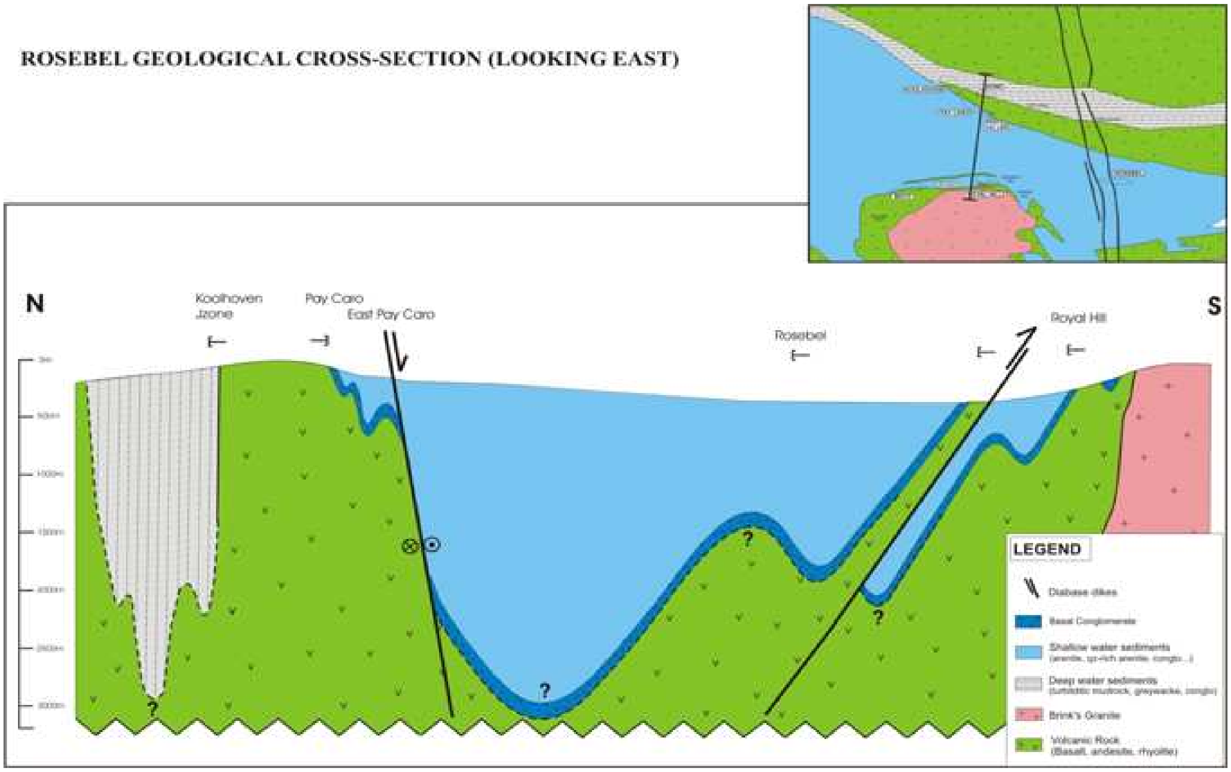


Figure 7.4. North-south cross-sectional interpretation of the Rosebel property.

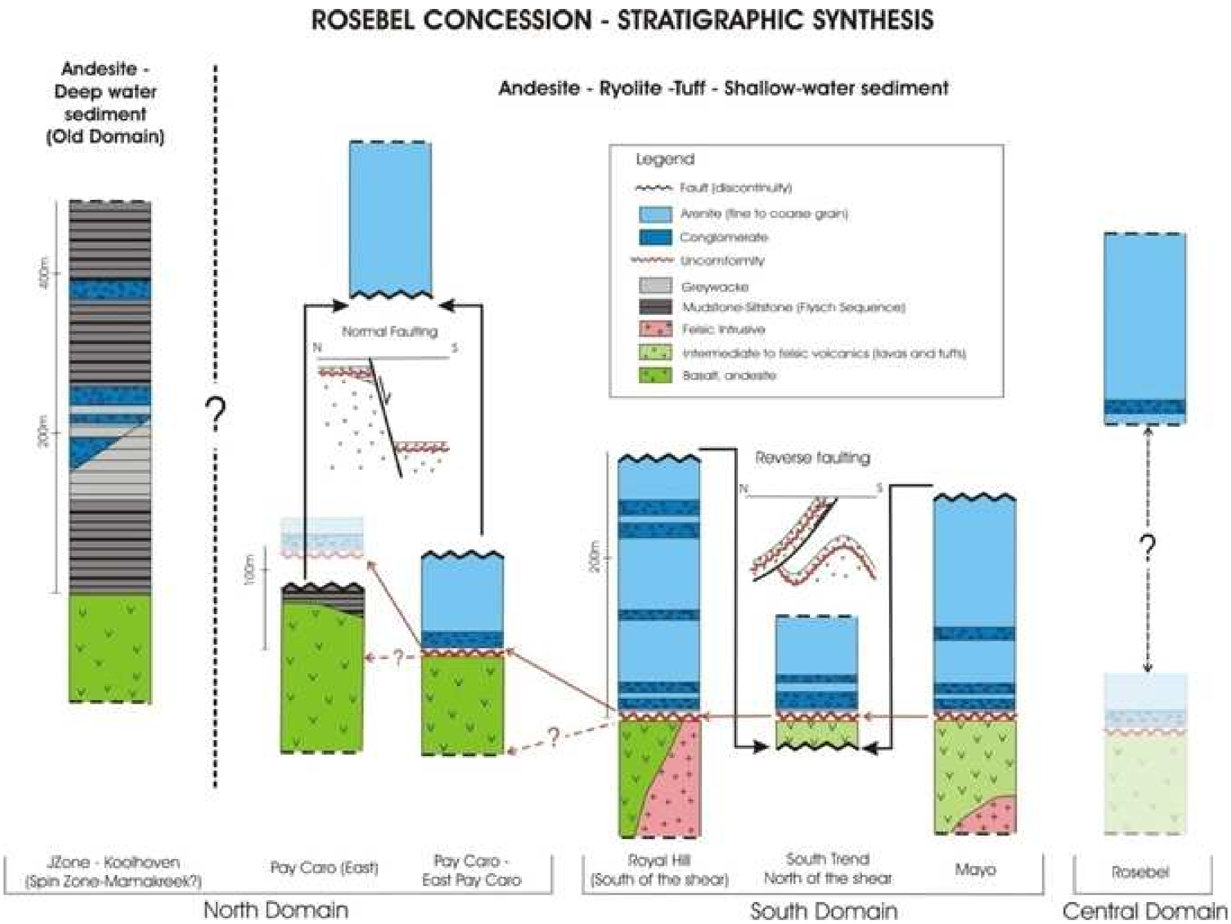


Figure 7.5. Stratigraphic synthesis of the Rosebel property.

8. MINERALIZATION

The Rosebel mineralization systems are classified as Orogenic Gold Deposits and are characterized by mineralized quartz veins emplaced during or shortly after the Trans-Amazonian Orogeny (between 2180-2000 Ma).

8.1. Remobilized Gold

A significant part of the lateritic cover is enriched in gold that has been remobilized from the underlying saprolite and rock by groundwater fluctuations. Gold usually occurs as coarse grains of free gold attached to goethite or hematite, or within cubic voids left by the weathering of pyrite crystals. Nuggets up to 8 millimetres in diameter have been found lying on the surface after heavy rain. Extensive small scale mining over the years has created a significant amount of tailings and “dumps” that contain gold. Gold-bearing alluvial material occurs in every pit currently in production (Royal Hill, Pay Caro, Mayo, and Koolhoven).

8.2. Vein-style gold mineralization

In general, vein mineralogy comprises quartz, carbonate, feldspar, tourmaline and pyrite, although the proportion of principal minerals and the nature of secondary minerals differ between the northern and southern mineralized trends.

In the southern trend, the typical infill mineral assemblage includes quartz (qtz) + carbonate (cb; calcite) + tourmaline (to) ± chlorite (chl) ± sericite (sr) ± pyrrhotite (po) ± pyrite (py) with accessory sphalerite, plagioclase and magnetite. Although tourmaline is not present in all veins, it can constitute more than 50% of vein mineralogy and appears to be more abundant in veins near the Brinks intrusion. Alteration selvedges mainly comprise chlorite, carbonate (mostly calcite), sericite, pyrrhotite, with tourmaline and pyrite occurring locally. Like tourmaline, pyrrhotite appears to be more abundant in the Royal Hill area than in the Mayo area.

In the northern trend, near the main shear, vein mineralogy is characterized by qtz + cb (calcite + ankerite) + feldspar (fds) + chl ± sr ± py assemblages. Tourmaline can be present but is rare. Away from the shear, vein mineralogy is restricted to a qtz + cb ± chl ± sr ± py assemblage. Alteration selvedges in the Pay Caro deposit are zoned from a central, pervasive chl + sr + fds + hematite (hem) + cb + py assemblage to a marginal chl + cb + sr ± py assemblage that only partially replaces primary mineralogy. The nature of the host rock also plays an important role in the type of alteration; volcanic rocks are mostly altered to chlorite and calcite, whereas sedimentary rocks are altered to sericite and chlorite depending on sediment maturity.

Mineralization is hosted in spatially and temporally related shear and tension vein arrays. The association of these two vein systems in the Rosebel deposits is typical of orogenic gold systems where tension veins develop in extensional fractures that have accommodated deformation. Tension veins are more important in terms of contained gold, although shear veins can carry significant grades (e.g. Pay Caro Deposit) and are thought to be a fundamental control on hydrothermal fluid circulation. The detailed geometry of tension and shear vein arrays differ for each deposit within the Rosebel property. The disparity in veining style is particularly evident between the northern and southern mineralized trends (Figs. 8.1 and 8.2).

In the southern trend, shear veins occur inside and parallel to a regional through-going shear zone and are generally confined to the footwall of this structure. Shear veins are characterized by banded textures, with quartz typically alternating with tourmaline near vein margins. Vein thickness varies from a few decimetres to more than three metres and individual veins can be continuous throughout a pit. Shear veins are generally orientated parallel to bedding. In the Royal Hill area where host rocks have been folded, shear veins have been emplaced parallel to the northern limb of an anticline that dips around 60° towards the north. In Mayo, shear veins parallel the relatively shallow (40°) northern dip of the host rocks. The most economically important shear veins are emplaced along contacts between volcanic and sedimentary rocks or along conglomeratic units within the arenite sequence.

Shear veins in the northern trend are sub-vertical to steeply south dipping and are oriented WNW-ESE. Vein density and textures vary depending on the structural style of the host structure(s). In Pay Caro, shear veins are developed within a broad shear zone that preserves evidence for normal movement. Veins can be several metres thick, typically have brecciated textures and are associated with a strong deformation fabric and an intense alteration halo developed in adjacent host rocks. In the Koolhoven and J-Zone pits where shears are more discrete, shear veins are usually centimetric to decimetric, can be banded or brecciated and are characterized by poorly developed deformation fabrics and weak alteration haloes.

North-south tension veins dipping between 60° and 75° to the west are present throughout the property. Their timing relative to other vein sets has not been well established in the southern trend, but clear cross-cutting relationships observed in the Koolhoven pit indicate that they are slightly earlier than a stacked north-dipping vein array. Analysis of gold content demonstrates a grade distribution comparable to all other vein sets, and so, these veins probably originate from the same mineralizing fluids. Unfortunately, the distribution of this vein set is not well constrained due to the predominant north-south to NNE-SSE drilling orientation. Further pit mapping should provide clarification.

Two additional sets of tension veins are spatially associated with shear veins in the southern trend; flat veins and north-dipping veins. Flat veins are present in the northernmost part of the Royal Hill deposit and are developed within or adjacent to conglomerate units in the hinge of an anticline. The north-dipping veins occur as stacks in the immediate footwall or hangingwall of shear veins. In Royal Hill, they dip around 30° to 45° to the north, but are mostly sub-horizontal to shallow north-dipping in the Mayo pit where host structures are less steep.

Two sets of tension veins have also been identified in the northern trend: east-west, sub-vertical veins and stacks of north-dipping veins. East-west veins are sub-vertical to steeply north dipping and only occur in the Pay Caro deposit near the main shear zone.

North-dipping tension vein occur proximal to shear veins and form WNW-ESE trending stacks dipping around 45° to the NNE. They are thought to be related to the latest normal movements undergone by the host shear zone (Fig. 8.3).

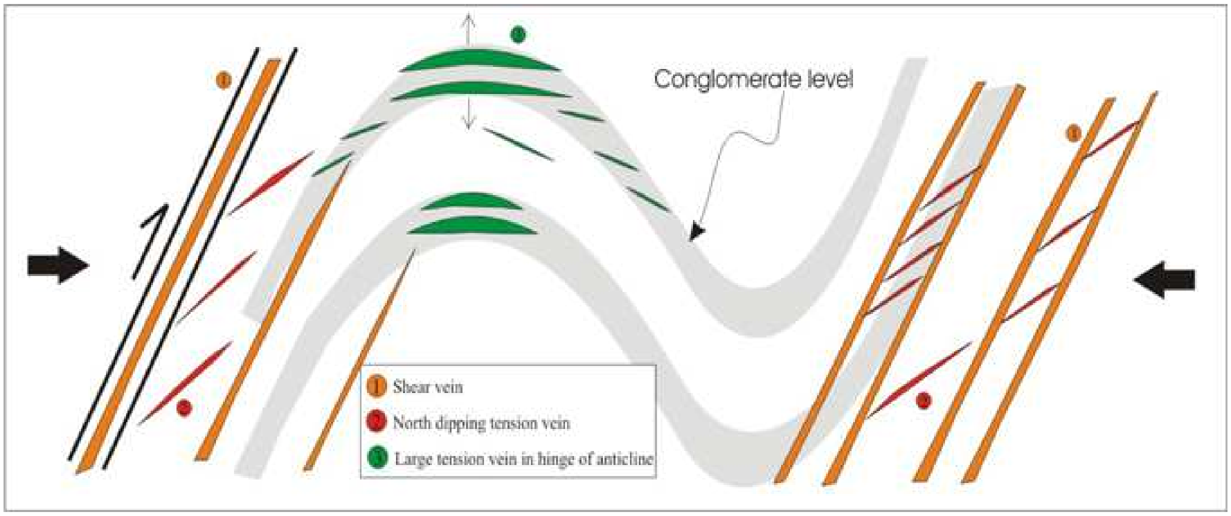


Figure 8.1. Schematic representation of vein systems in the southern mineralized trend.

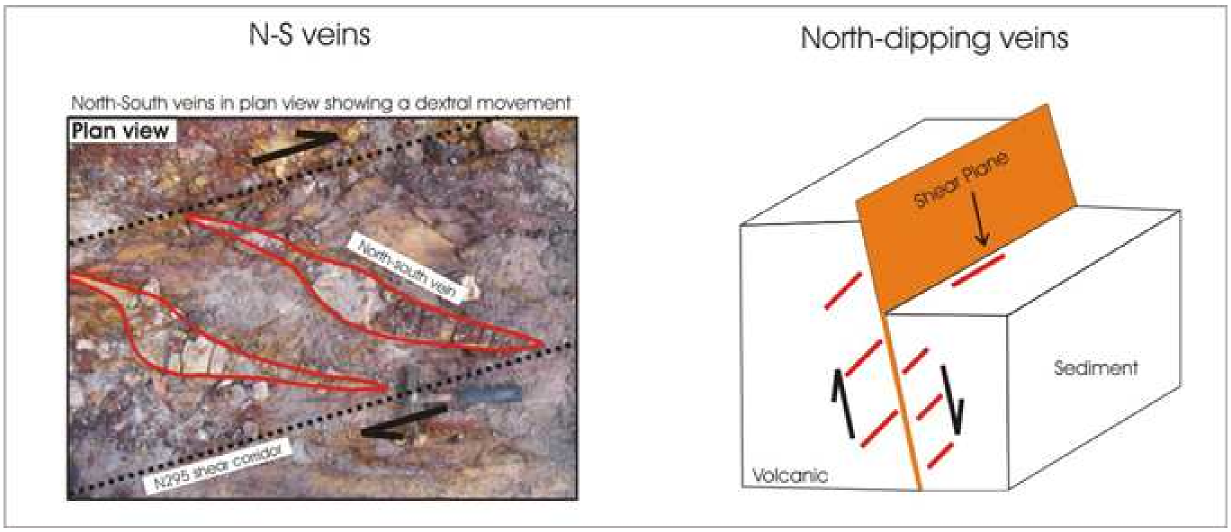


Figure 8.2. Northern mineralized trend – two episodes of mineralisation.

An important characteristic of all mineralized vein types is the lack of subsequent deformation, indicating that gold deposition is likely to have occurred late in the Trans-Amazonian orogenic history. Overprinting relationships demonstrate that veining took place after folding and has commonly utilized pre-existing structures, extensional fractures or rock heterogeneities. Consequently, anticlinal hinges, lithological contacts and conglomerate units have provided good structural traps for mineralizing fluids. In fact, seven of the eight known deposits are located near lithological contacts.

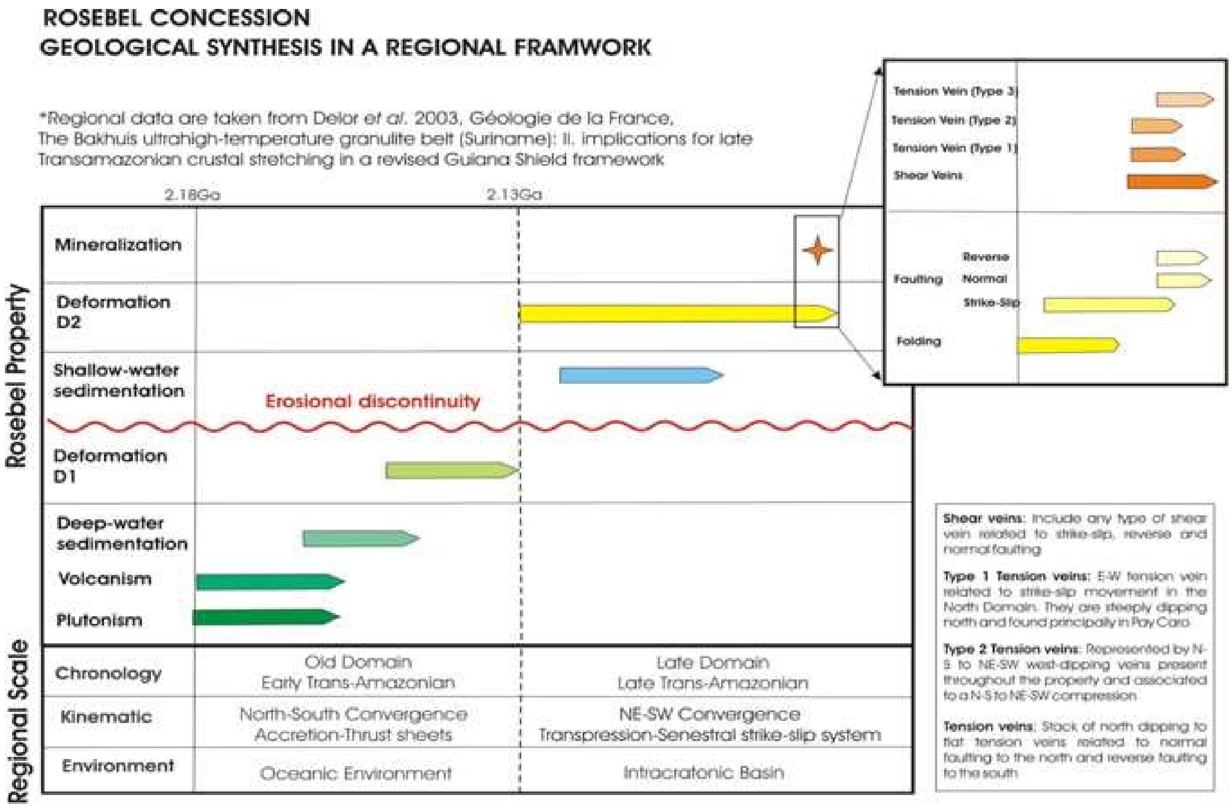


Figure 8.3. Synthesis of the Rosebel geological history.

9. DEPOSIT GEOLOGY

The geology of the property and the style of gold mineralization vary between the northern and southern limbs of a regional syncline and also between the various deposits. In the northern limb, the mineralized trend has a strike length of 12 kilometres, and hosts the Pay Caro – East Pay Caro, Koolhoven and "J" Zone deposits as well as the Spin and Mamakreek anomalies. The mineralized trend in the southern limb has a strike length of 15 kilometres and hosts the Mayo, Roma, Royal Hill and Rosebel deposits, as well as the Eriaan Hill and Monsanto Hill anomalies. A description of each of the main deposits is given below.

9.1. MAYO

The Mayo deposit (Fig. 9.1) lies in the western extremity of the southern trend, and has undergone the least deformation of all Rosebel deposits. The base of the local stratigraphy is marked by a fine-grained, generally featureless felsic volcanic unit (flow breccia textures have been observed locally) at the southern edge of the current Mayo pit design. To the north, this volcanic unit is unconformably overlain by the shallow-water sedimentary association which is represented by two basal conglomerate units and a thick overlying arenitic sequence. The stratigraphy is not complicated by folding and the average attitude of bedding is 40° g 350°. The host sequence is truncated to the north by the principal shear zone of the southern trend which is orientated parallel to stratigraphy and has emplaced volcaniclastic rocks (mainly felsic tuff and lapilli tuff) on the top of the sedimentary sequence. This reverse structure is practically barren in the Mayo area, with mineralization mainly hosted in shear and tension veins in the immediate footwall.

The structural architecture of the deposit can be modelled as a south-verging thrust fault system, with low-angle reverse faults and fractures developed along contacts of the basal conglomerate units. These structures are thought to have been the main conduits for hydrothermal fluids during deformation, with gold deposition accompanying the development of moderately north-dipping quartz ± carbonate shear vein arrays and gently north-dipping to horizontal tension veins. A north-south striking, steeply dipping vein set also occurs in the pit, but its extent and distribution is poorly constrained.

Sulphide mineralogy in the ore zones comprises 2 to 7 percent pyrite, with traces of pyrrhotite. A series of late-stage north-south striking faults with horizontal displacements of up to 20 metres locally offset the host sequence.

The mineralized zone has been tested by drilling for a strike length of 3,000 metres and to a depth of 350 metres below surface. Significant deep drilling intercepts in the basal conglomerate unit and underlying volcanic sequence indicate potential depth extensions to the deposit. Eastwards toward Roma, the near-surface mineralization appears to pinch out, although interesting grades intersected at depth require follow-up.

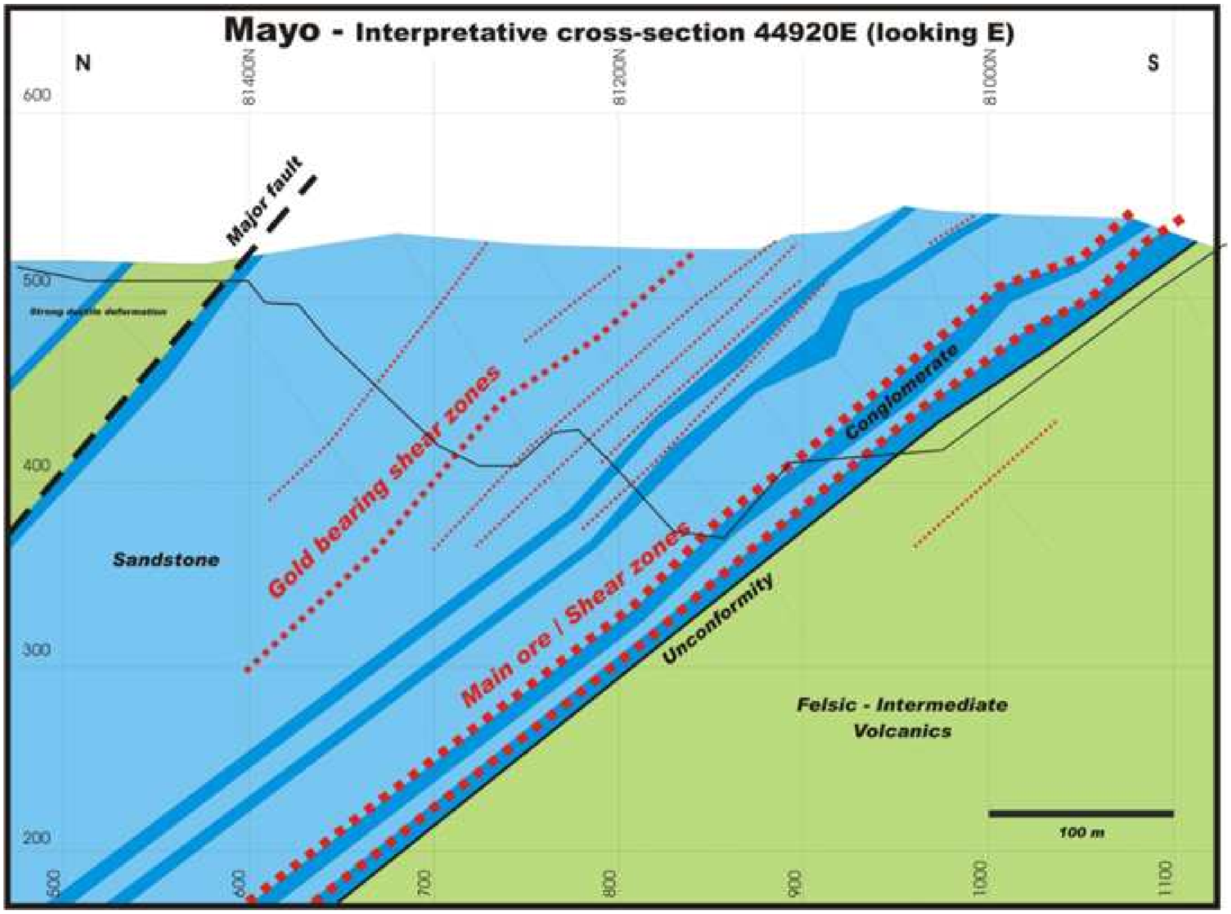


Figure 9.1 . Mayo Deposit – Interpretative cross-section.

9.2. Roma

In 2008, the Roma exploration project was transferred from Regional Exploration to the Mine Exploration Department. Resource delineation drilling commenced in February 2008, with block modelling and an initial resource estimate completed the following year.

The Roma deposit (Fig. 9.2) is located on strike between the Royal Hill and Mayo deposits and is similarly truncated to the north by a major south-verging thrust fault. The basal volcanic sequence differs from that described for Mayo, consisting mainly of felsic tuff and lapilli tuff. Host rocks are deformed by large-wavelength, open folds with an east-west striking axial planar foliation. The deposit is separated from the Royal Hill deposit by a late NNW-striking and steeply west-dipping fault and is itself cross-cut by two late faults oriented NNE and dipping between 50° and 75° to the east.

The deposit consists of two distinct pits (East and West). Gold mineralization in the East pit occurs in centimetric to decimetric, north-dipping tension veins that are hosted within the basal conglomerate unit in an anticlinal structure. Mineralization in the West pit is hosted in a thick pile of cross-bedded arenite that contains metre-thick conglomerate units on the northern flank of the anticline. Veins in the Western pit are typically thicker than in the East, have a greater density and locally higher grades. Shear veins strike east-west to SSW-NNW and dip 45° toward the north while tension vein sets are sub-horizontal to gently north-dipping. Mineralization is associated with small amounts of pyrite, generally less than 1%-3%.

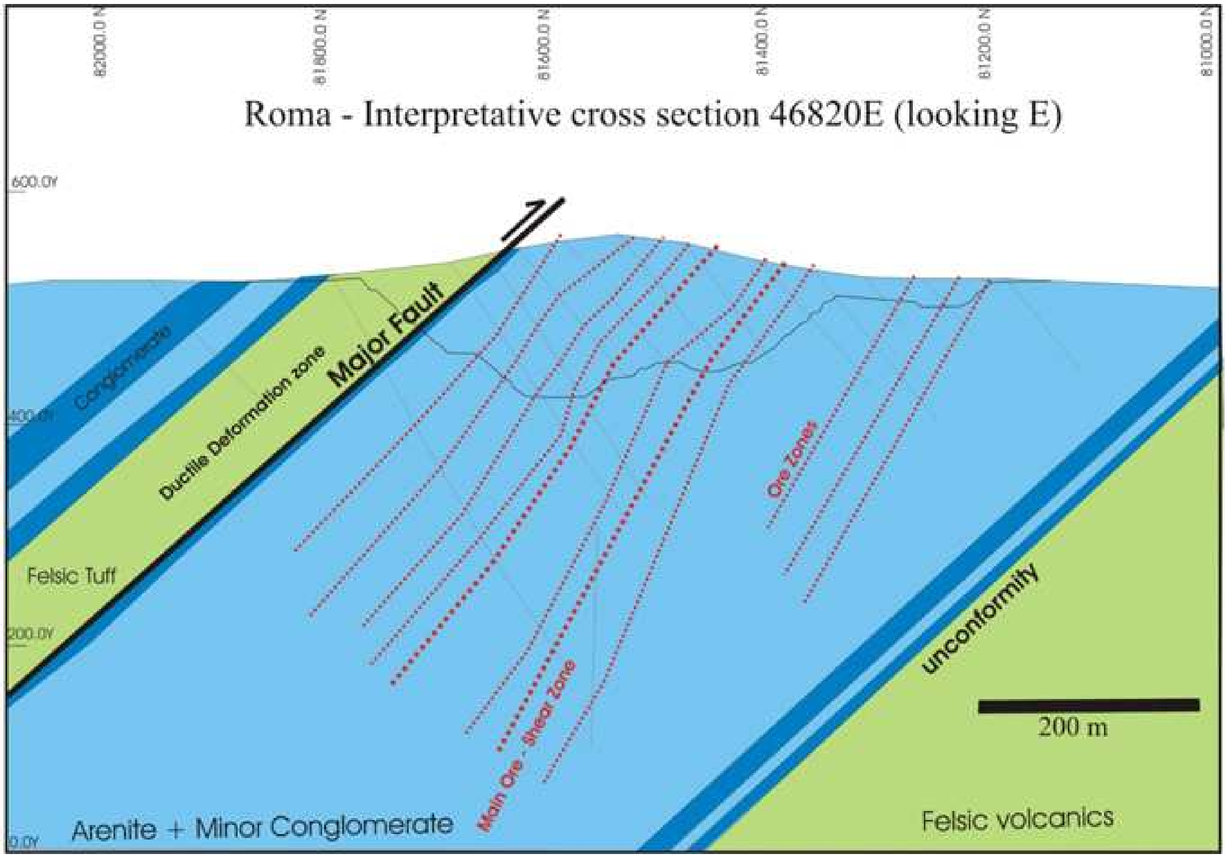


Figure 9.2. Roma Deposit – Interpretative cross-section.

9.3. ROYAL HILL

The Royal Hill deposit (Fig. 9.3) is located immediately south of the old exploration camp and is the easternmost deposit identified within the southern trend (although other gold occurrences such as Eriaan Hill and Monsanto Hill lie further to the east). The area was exploited by several companies during the late 19th and early 20th centuries and presently contains the largest gold reserve on the property. The deposit comprises two distinct pits that are centred on anticlinal structures and separated by the intervening and near gold-barren syncline. Host lithology and structure differ between the two pits.

The Royal Hill area exposes a similar stratigraphic sequence (volcanic-conglomerate-arenite) to that observed in the Mayo area, but with small variations. Firstly, the pit is bordered to the south by a tonalitic intrusion (Brinks Granite), and secondly, the basal volcanic sequence is characterized by andesitic to tholeiitic basalt compositions. In addition, the Royal Hill area is folded by a succession of closed anticlines and synclines that plunge gently (10°-20°) towards the west.

The basal volcanic part of the stratigraphic sequence occurs only in SE pit where it underlies two conglomerate units and an upper succession of immature arenite that contains a number of discrete conglomerate intervals. Gold mineralization mainly occurs in the hangingwall of the volcanic-sedimentary contact, on the east-west striking and steeply-dipping (60°) northern limb of the southernmost anticline. Mineralization is associated with north-dipping (60°), east-west striking shear veins and stacks of gently north-dipping tension veins. The abundance of shear veins increases towards the volcanic-sedimentary contact.

The NW pit exposes a shallowly west-plunging anticline as indicated by S_0/S_1 angular relationships measured in the pit. Host rocks are dominated by sedimentary rocks, although volcanics have been intercepted in the hinge of the anticline at a depth of 460 metres below surface. In the center of the pit, gold mineralization is associated with tension veins emplaced in the hinge of the anticline near or within conglomerate units. These veins are generally sub-horizontal and parallel bedding. The NW pit is bounded to the north by the 55-60° north-dipping main shear zone of the southern trend. Like the Mayo area, this shear zone juxtaposes a thin horizon of highly sheared felsic tuff against the main host sequence. However, at Royal Hill this structure is mineralized. Gold is hosted in both shear veins and stacks of north-dipping tension veins in the footwall of the shear zone. North-south veins can also carry significant gold grades in both pits, but their relative economic importance is not yet well established.

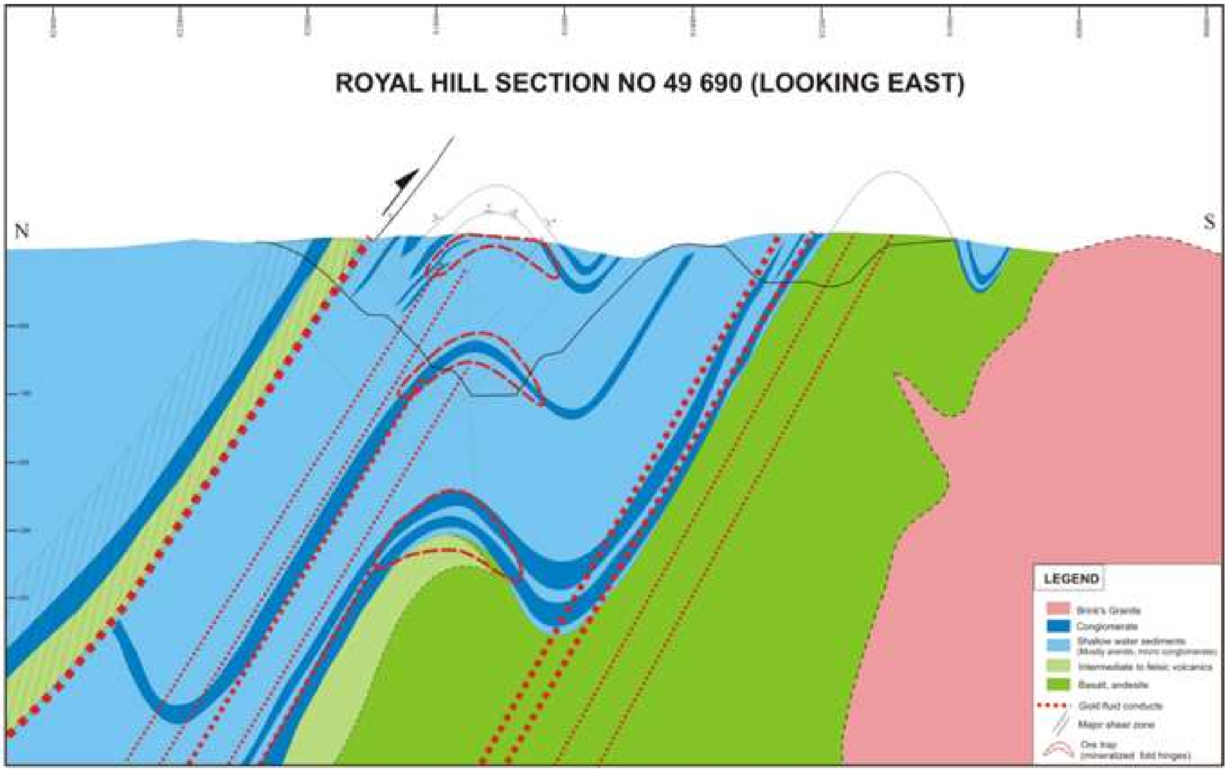


Figure 9.3. Royal Hill Deposit – Interpretative cross-section.

9.4. ROSEBEL

The Rosebel deposit (Fig. 9.4), located approximately 13 kilometres east of the mill, is hosted by a steeply dipping and broadly northwards fining succession of conglomerate, arenite and siltstone in the central mineralized trend. The southern part of the pit exposes an interval of conglomerate within a coarse-grained arenite sequence that progressively grades to finer-grained arenite and siltstone, suggesting a general northward polarity. The stratigraphy strikes 80° and is sub-vertical to steeply north or south dipping. The deposit is intruded by three sub-vertical, north-south striking diabase dykes that are devoid of any mineralization.

The Rosebel deposit area is characterized by relatively low gold grades distributed across a broad chlorite – sericite alteration zone that is locally intensified along numerous shear zones. Gold mineralization is associated with north-dipping quartz ± carbonate tension vein arrays and 1 to 3 percent pyrite alteration. The vein sets are localized in shear corridors developed at contacts between sandstone and siltstone units. Diamond drilling performed between 2005 and 2009 intersected economic gold mineralization down to 200 metres below surface and the continuity of mineralization can now be traced for over two kilometres along strike. The deposit remains open on strike and at depth.

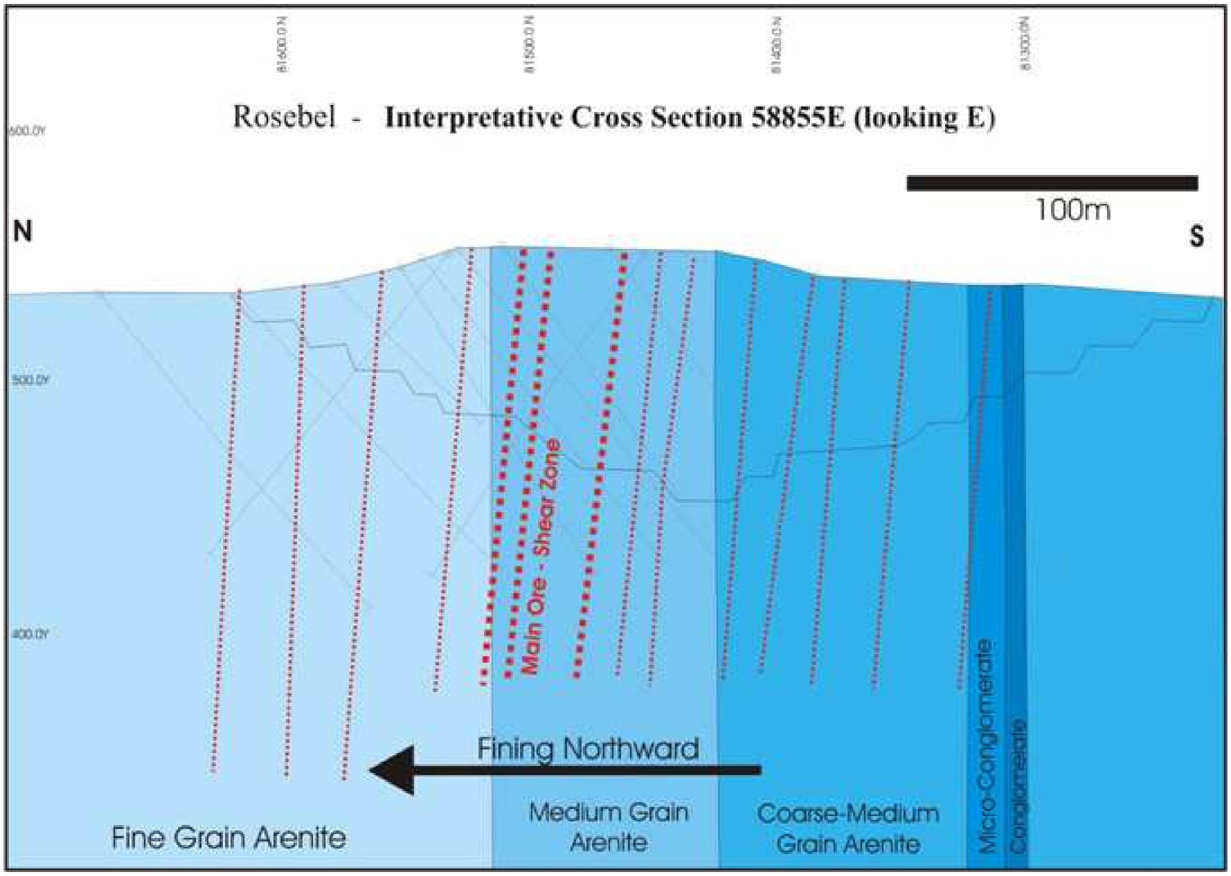


Figure 9.4. Rosebel Deposit – Interpretative cross-section.

9.5. Pay Caro – East Pay Caro

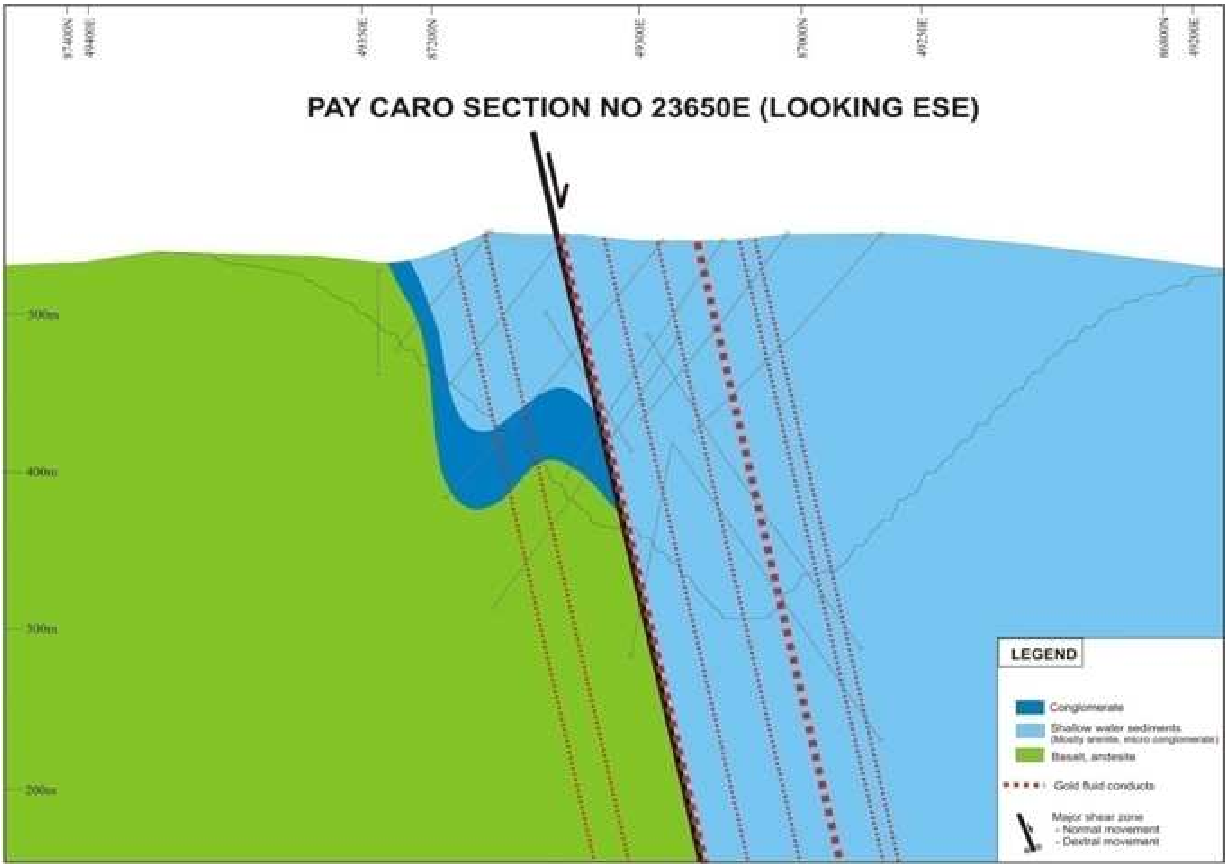
The Pay Caro and East Pay Caro deposits (Fig. 9.5) are located along the strongly deformed southern structural domain of the northern trend. The general polarity is to the south, except where folds are encountered. The stratigraphic succession comprises of a basal intermediate volcanic pile, predominantly andesitic flows (locally intercalated with pillow lavas) and minor volcaniclastics, which are overlain by a conglomeratic unit and a fine-grained arenite sequence with local intercalations of siltstone and mudstone. The lower part of the succession contains micro-conglomerates and greywacke while the upper part appears more mature, being dominated by well-sorted fine-grained arenite. The contact between the volcanic pile and the overlying sedimentary succession is conformable, except where the volcanics rocks have been structurally juxtaposed with the upper part of sedimentary sequence. The sequence is deformed by z-type parasitic folds that plunge between 30° and 50° to the WNW. The East Pay Caro pit is centred on a parasitic fold with a wavelength of approximately 200 metres.

The principal fault zone in Pay Caro is characterized by a brecciated texture, shear veins reaching several metres in thickness and pervasive alteration of host rocks with total replacement of primary mineralogy. Tension veins can be subdivided into three main sets based on orientation: 1) north-south striking, west-dipping veins, 2) east-west striking, sub-vertical to north-dipping veins, and 3) stacks of “en echelon” WNW-striking, moderately (30°-50°) north-dipping veins. All vein sets carry similar gold grades. North-south orientated veins occur throughout Rosebel property but are not well documented outside the pits owing to the prevailing north-south drill direction. East-west striking tension veins are restricted to Pay Caro and mostly occur in the central zone. The deposit contains three pockets of high grade material with an average plunge of 45° toward N285°. This orientation appears to represent the intersection of the east-west veining system with the principal shear.

Sulphide mineralogy consists of 2 to 5 percent pyrite, with traces of chalcopyrite. The delineated strike length of the zone is 2,200 metres, with mineralization tested to a depth of nearly 600 metres below surface. The zone remains open to the west towards the Koolhoven deposit, although mineralization in West Pay Caro is limited to narrow and generally sub-vertical to steeply south-dipping zones composed of steeply north-dipping ladder veins striking N280° to N290°. Mineralization is also open at depth.

Gold mineralization in East Pay Caro is primarily hosted in tension veins propagated along the contact between the volcanic rocks and the basal conglomerate, mostly within the closure of the parasitic anticline. Mineralization is associated with 2 to 5 percent sulphide (mainly pyrite with traces of chalcopyrite and pyrrhotite), extends for over 825 metres along strike and has been delineated to a depth of 280 metres below surface.

East Pay Caro is open at depth to the west, along the volcanic-sediment contact and in the west-plunging antiform hinge



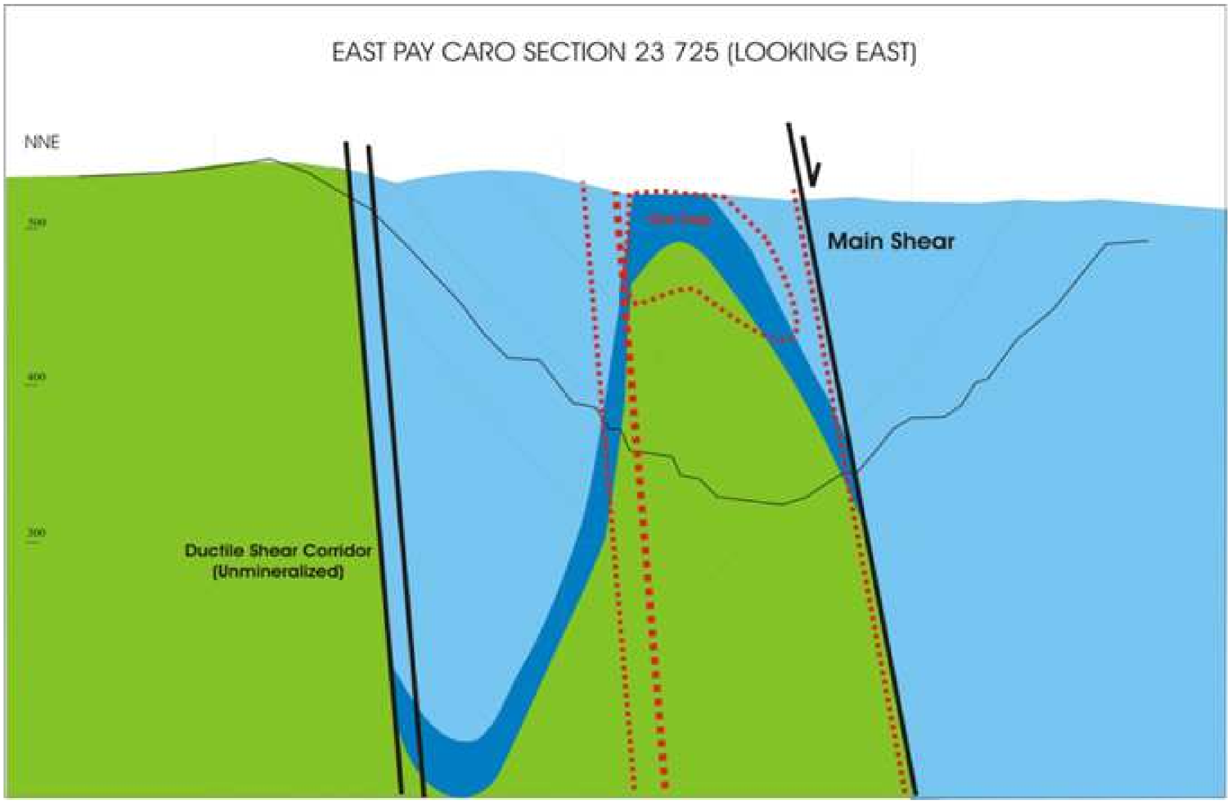


Figure 9.5 . Pay Caro and East Pay Caro Deposit – Interpretative cross-sections.

9.6. KOOLHOVEN

The Koolhoven deposit (Fig. 9.6) is located in the northern structural domain of the northern trend. The local stratigraphy comprises intermediate to felsic volcanics and volcaniclastics in the extreme south and an overlying sedimentary sequence to the north. The latter consists of alternating turbiditic mudstone and greywacke, with conglomeratic lenses identified in the northern part of the deposit. The sequence has been deformed by tight to isoclinal folding, with wavelengths of several hundred metres and fold axes plunging 70°-90° to the west. These fold structures, even though coaxial to the main phase of folding observed throughout the property, may represent an older deformation event that has affected the Paramaka Formation and deep-water sedimentary succession only. The shallow-water sedimentary succession would therefore postdate this early deformation.

Koolhoven gold mineralization is associated with quartz-carbonate veins and wall-rock alteration comprising 2 to 5 percent disseminated pyrite. The gold bearing quartz vein system is fundamentally controlled by sets of discrete, WNW-striking, vertical to steeply south dipping shear zones. The mineralization is concentrated along three principal zones (North, Central and South) where the most significant shear zone occurs in the middle of the central zone. The shear zones are occupied by lenticular, centimetric to decimetric shear veins and associated with two sets of tension veins. The first set of tension veins strike WSW-ESE and dip moderately towards the north and are the most important in terms of contained gold. Vein thickness and density decrease with distance from shear zones. The second set of tension veins generally strike N-S and are dip steeply towards the west. Higher competency units (like conglomerates) appear to have been more favourable for vein development. Drilling has tested the mineralization over a 1,700 metre strike length and to a depth of 400 metres below surface. The ore zones remain open at depth and along strike to the northwest.

9.7. J-Zone

The J-Zone deposit (Fig. 9.6) is located approximately one kilometre ESE of Koolhoven in the same structural domain. Like Koolhoven, J-Zone is hosted by a sequence of turbiditic greywacke, laminated siltstone/mudstone and subordinate polymictic conglomerate (deep-water sedimentary succession). A conglomeratic sequence forms the northern margin of the mineralized corridor, while the southern margin corresponds to the underlying mafic–intermediate volcanic sequence. Several small scale folds have been observed in the host sedimentary sequence, but the majority of sedimentary structures indicate a general northerly facing direction.

Gold mineralization is primarily associated with quartz-carbonate veining. Wall rock alteration is dominated by carbonate-pyrite, with a sericite-albite assemblage developed proximal to veins. Pyrite mineralization is weakly anomalous in gold and characterized by disseminated, millimetric crystals that may have plurimetric extension halos. Quartz veins are related to discrete, sub-vertical to steeply south dipping shear zones are generally a few centimetres to a few decimetres in thickness and generally devoid of sulphides. Three main auriferous quartz vein types have been identified: 1) moderately north dipping tension veins (predominant type), 2) NNE-striking sub-vertical tension veins and 3) texturally complex shear-breccia veins that fill shear discontinuities.

Drilling has delineated mineralization over a 2,600 metre strike length and to a depth of 230 metres below surface. The J-Zone resource may be expanded with additional drill testing on strike to the northwest, at depth and in the eastern part of the deposit

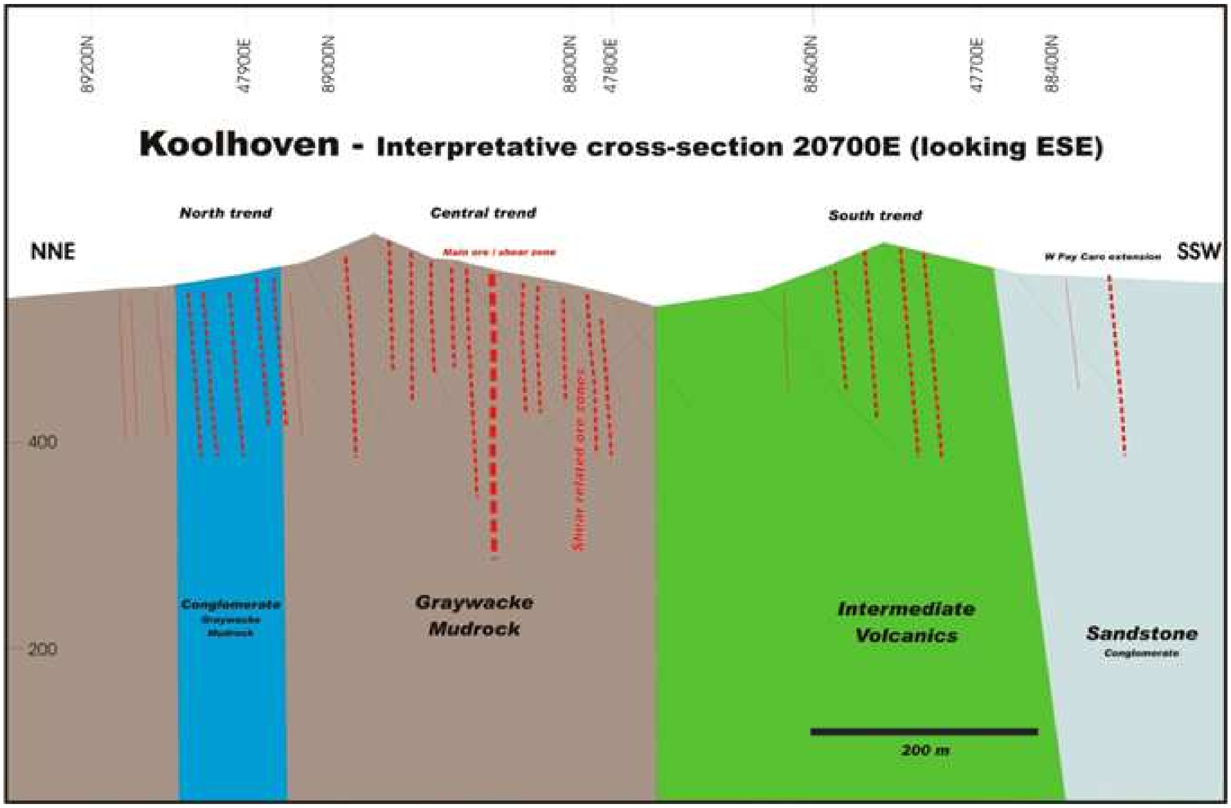


Figure 9.6. Koolhoven (and J-Zone) Deposits – Interpretative cross-section.

10. EXPLORATION

Exploration at Rosebel is divided between near-mine exploration (reserve development) and regional exploration. Near-mine exploration is carried out in the vicinity of the known deposits for the purposes of resource and reserve delineation using diamond drilling. Regional exploration is carried out using various methods; including soil, stream sediment and auger sampling, ground and airborne geophysics, trenching and diamond drilling.

Of all the activities performed through the various stages of exploration, adequate sample collection and preparation techniques remain the most important steps for the mineral inventory. Detailed procedures were implemented for sample collection and a summary for all sample types is presented below.

10.1. Soil Sampling

Soil sampling grids are initially laid out on lines spaced either 200 metres or 400 metres apart and samples are collected at 50-metre intervals along each line using a 5-centimetre auger drill. Holes are drilled to a depth of 1 metre or 2 meters and all material collected is sent for assay (in 1 meter intervals). Infill sampling over resulting soil anomalies reduces the grid spacing to 25 metres by 50 metres, although smaller grids are used over narrow anomalies as required. Maps of gold values are drafted, with contours typically at 0.50 and 1.00 g/t Au. This type of work is used to guide further exploration and data is not included in reserve estimation.

10.2. Deep Augering

Soil anomalies are further investigated using 10-centimetre auger drill holes, initially on a 50 metre by 50 metre grid, which is reduced to 25 metres by 25 metres where better definition is required. The rods can be either manually or mechanically driven. Holes are drilled to a target depth of 10 metres, although the average depth is 6 metres with many holes encountering shallow quartz veins, duricrust or harder ground. Samples are collected in one metre intervals and sent for assay. Holes that reach saprolite can

provide lithological information for geological mapping. Deep auger hole locations are surveyed and assay results are included in reserve estimation.

10.3. BANKA DRILLING

This is a specialized manual auger drill of 15 centimetres diameter, developed for sampling wet, unconsolidated deposits, especially alluvial material. As the hole deepens, steel casing is pushed down the hole to support the walls and avoid contamination. Samples are collected in one metre lengths, dried and quartered, with one quarter being sent for assay. Banka hole locations are surveyed and assay results are included in reserve estimation.

10.4. TRENCHING

Trenches are typically excavated at intervals of 100 metres along the strike of auger anomalies. They are initially opened by bulldozer to a depth of 2 to 3 metres. If significant mineralization is exposed, the trenches are deepened using a mechanical excavator to depths of up to 6 metres depending on ground hardness. Geological mapping is carried out to identify rock types and mineralized zones. Channel samples of 1 to 3 metres in length are collected from one trench wall. Channels are vertical in laterite layers and perpendicular to veins in saprolite. Auger holes (10 centimetre diameter) are drilled in the bottom of the trenches at 15 metre spacings to a target depth of 10 metres; each metre of material is collected for assay. Trench channel sample and auger hole locations are surveyed and assay results are included in reserve estimation.

10.5. DIAMOND DRILLING

Intensive diamond drilling programs were carried out on the property between 1992 and 1997. Between 1998 and 2000, the Rosebel project was placed on care and maintenance and no additional drilling was undertaken. Drilling resumed in 2002 with the objective of sterilizing the waste dump at Pay Caro. Additional geotechnical drilling was completed at the mill site and tailings pond. Exploration and definition drilling resumed in 2004 and since then, a total of 356,725 metres of diamond drilling has been carried out on the Rosebel property. The mineral reserve estimates prepared between January 2004 and January 2007 have been reported by the previous RGM owner, Cambior Inc., while the reserve estimates prepared between January 2008 and December 2009 have been reported by the actual owner, Iamgold Corporation. All reserve estimates, historical and actual, were prepared under the supervision of a qualified person as defined in Canadian National Instrument 43-101 Standards.

Exploration has been extremely successful in delineating new reserves and resources (Table 10.1). None of the Rosebel deposits have been completely closed off, and there is good potential to delineate new reserves and resources in the future.

	January 1st 2004 Mineral Reserve			January 1st 2005 Mineral Reserve			January 1st 2006 Mineral Reserve		
Deposit	Total			Total			Total		
	Tonnes (000)	Au (g/t)	Au (K oz)	Tonnes (000)	Au (g/t)	Au (K oz)	Tonnes (000)	Au (g/t)	Au (K oz)
Koolhoven	4,478	1.45	209	4,789	1.39	214	5,144	1.33	220
Pay Caro	16,742	1.81	975	17,266	1.70	943	17,223	1.52	844
East Pay Caro	6,026	1.39	269	5,761	1.22	225	10,892	1.13	397
Mayo	5,379	1.56	270	5,916	1.47	280	8,065	1.32	343
Royal Hill	11,926	1.41	539	14,232	1.29	591	22,014	1.42	1,006
Rosebel	2,615	1.42	119	2,898	1.34	125	7,972	1.33	340
J-Zone									
TOTAL	47,165	1.57	2,382	50,862	1.45	2,379	71,309	1.37	3,150

	January 1st 2007 Mineral Reserve			January 1st 2008 Mineral Reserve			January 1st 2009 Mineral Reserve		
Deposit	Total			Total			Total		
	Tonnes (000)	Au (g/t)	Au (K oz)	Tonnes (000)	Au (g/t)	Au (K oz)	Tonnes (000)	Au (g/t)	Au (K oz)
Koolhoven	15,796	1.09	553	11,159	1.15	412	14,256	0.99	452
Pay Caro	23,014	1.24	917	16,726	1.39	749	16,385	1.18	622
East Pay Caro	12,183	1.11	436	9,505	1.13	344	10,915	1.09	384
Mayo	15,630	1.17	588	13,887	1.24	552	22,055	1.15	813
Royal Hill	24,066	1.24	962	19,508	1.36	850	18,169	1.36	793
Rosebel	8,687	1.18	329	7,234	1.26	292	10,819	1.13	393
J-Zone				5,207	1.00	168	6,920	0.92	205
TOTAL	99,376	1.18	3,785	83,226	1.26	3,366	99,518	1.14	3,662

Deposit	December 31st 2009 Mineral Reserve		
	Total		
	Tonnes (000)	Au (g/t)	Au (K oz)
Koolhoven	15,566	1.07	535
Pay Caro	19,757	1.13	717
East Pay Caro	13,517	1.05	458
Mayo	31,718	1.09	1,107
Roma	4,229	1.08	146
Royal Hill	35,477	1.16	1,324
Rosebel	14,552	1.03	480
J-Zone	8,143	0.93	243
TOTAL	142,960	1.09	5,010

Table 10.1. Historical Rosebel mineral reserves. Stockpiles are not included.

The surveys and investigations for reserve estimates, as well as the interpretation of the exploration information have been carried out by the RGM personnel. Exploration drilling was carried out by a contractor, Major Drilling, a Canadian company based in Moncton, New Brunswick – Canada, under the supervision of the RGM's personnel.

11 . DRILLING

Diamond drilling programs carried out on the Rosebel property between 1992 and 1997 were initially performed by the internal drilling team of Golden Star using a BBs-18A portable diamond drill and subsequently, Longyear 38 and Longyear 44 diamond drilling rigs. Production was 12.5 metres per shift, for two twelve-hour shifts per day, with an average core recovery of between 80 and 90 percent. In 1995, the Golden Star drilling team was sold to Major Drilling and the drilling contract put out to tender. The contract was then awarded to Forage Orbit and all drilling activities were then carried out using a combination of Longyear 38 and Acker hydraulic rigs, with up to four rigs operating at one time. Diamond drilling production gradually increased to 35 metres per shift at an average recovery rate of 90 percent

Between 1998 and 2000, the Rosebel Project was placed on care and maintenance and no additional drilling was undertaken. Drilling resumed in 2002 with the objective of sterilizing the waste dump at Pay Caro. Additional geotechnical drilling was completed at the mill site and tailings pond.

Exploration and definition diamond drilling resumed in 2004. Diamond drilling campaigns totalled 33,803, 54,854, 64,553, 52,914, 64,758 and 85,843 metres in 2004, 2005, 2006, 2007, 2008 and 2009 respectively. Since 2004, Major Drilling has been the sole drilling contractor on the Rosebel property. They use UDR-200D track mounted rigs. Production is generally 50 metres per shift at an average recovery rate greater than 90 percent. Until 2009, there were generally two to three drill rigs running twenty four hours per day on two shifts. A forth drill rig running twelve hours per day was added in mid-2009. It is expected that four drill rigs will be running continuously in 2010.

Holes are drilled using HQ size wireline equipment, usually reducing to NQ size in transitional to hard rock. Core recovery in saprolite and transition material is improved by using polymer additives combined with high concentrations of bentonite. Core is packed in corrugated plastic boxes at the drill site, prior to being transported to the core shack.

Before June 2005, drill holes were surveyed at down-hole intervals of approximately 50 metres using Tropari down-hole survey equipment and hydrofluoric acid tests. These methods were replaced by a more modern Flex-IT singleshot / multishot instrument, which can also provide magnetometric data down the length of the hole.

Core recovery is generally greater than 90 percent. Drill holes with unacceptably low recovery in mineralized zones (minimum of 75-80%, or 65 % over short intervals) are re-drilled until an acceptable representative sample is obtained.

At the core shack, the core is washed to remove the drilling fluids and expose the structures in the soft saprolite material. Geotechnical logging is carried out to record core recovery, RQD, rock hardness and fracture density. The core is then logged in detail and sample intervals usually 1 to 1.5 metres wide are marked by the geologist. After this is finished, the core is photographed and the geological technicians split the core for sampling.

Most drill holes are sampled continuously from top to bottom of the hole. Some infill holes completed between 1996 and 1997 were sampled only through the mineralized zones. Condemnation holes for the waste dumps were sampled only where potential signs of mineralization such as alteration or veining were logged. The soft laterite and saprolite material is split in half using a spackle knife, while the harder transition and fresh rock intervals are split in half using a hydraulic/impact core splitter or cut in half with a diamond saw. Half of the core is placed in a plastic bag with a unique sample number and sent to the laboratory for assaying while the second half is kept in the core racks for reference and/or further testing. On occasion, whole core samples from infill diamond drill holes are sent to the laboratory to enable comparative analysis of analytical accuracy.

11.1. DRILLING STATISTICS

Drilling statistics for each of the Rosebel deposits and near-mine exploration targets as of December 31st 2009 are summarized in Table 11.1.

Deposit or Zone	Diamond Drill		
	Holes	m	Samples
Koolhoven/Bigi	565	79,497	56,706
"J" Zone/Noutoe	264	40,126	28,622
Pay Caro	582	89,278	65,229
East Pay Caro	198	30,811	23,629
Mayo	575	77,551	58,385
Roma	200	31,601	23,386
Royal Hill	858	113,461	84,803
Rosebel	249	33,805	25,659
Spin/Tailings Pond	48	5,585	2,989
Mama Kreek	62	9,005	6,570
TOTAL	**3,601**	**510,720**	**375,978**

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Table 11.1. Total metres drilled by deposit as at December 31st 2009.

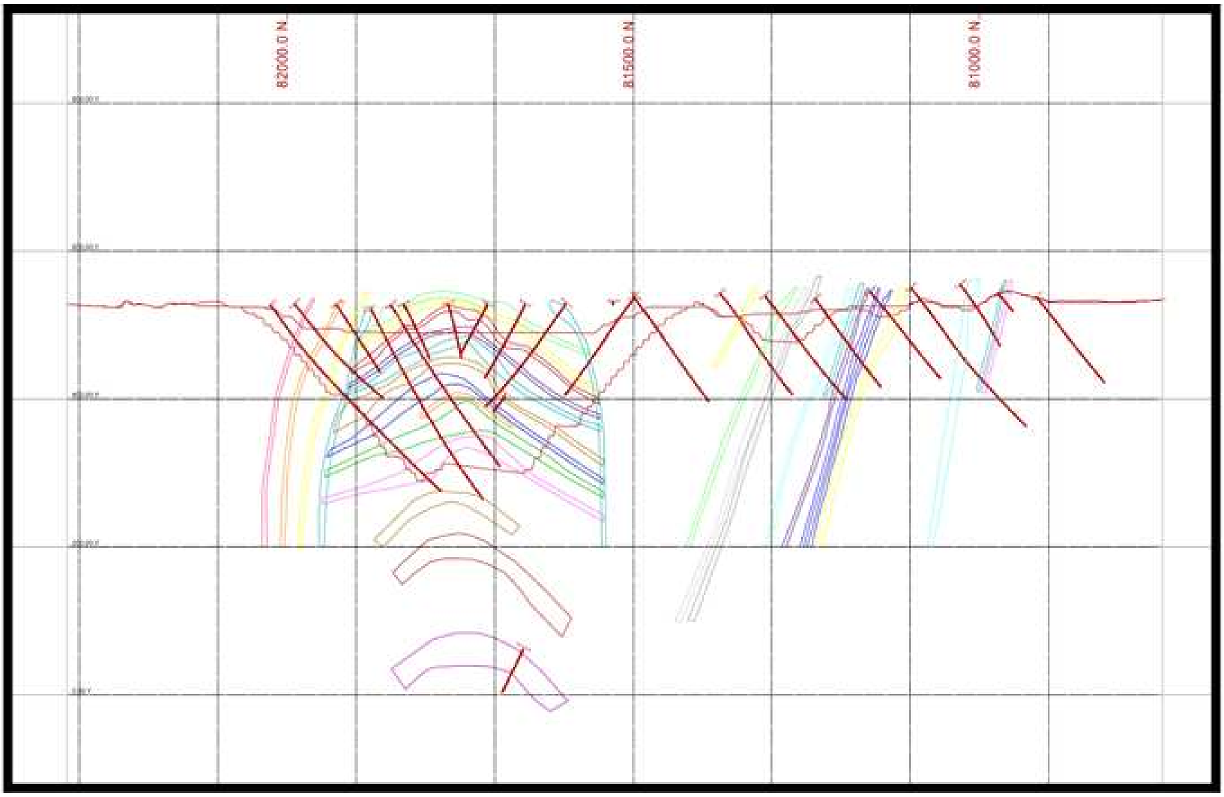


Figure 11.1. Vertical section at Royal Hill highlighting the typical diamond drilling pattern (49790 E, influence of 12.5 metres on both sides).

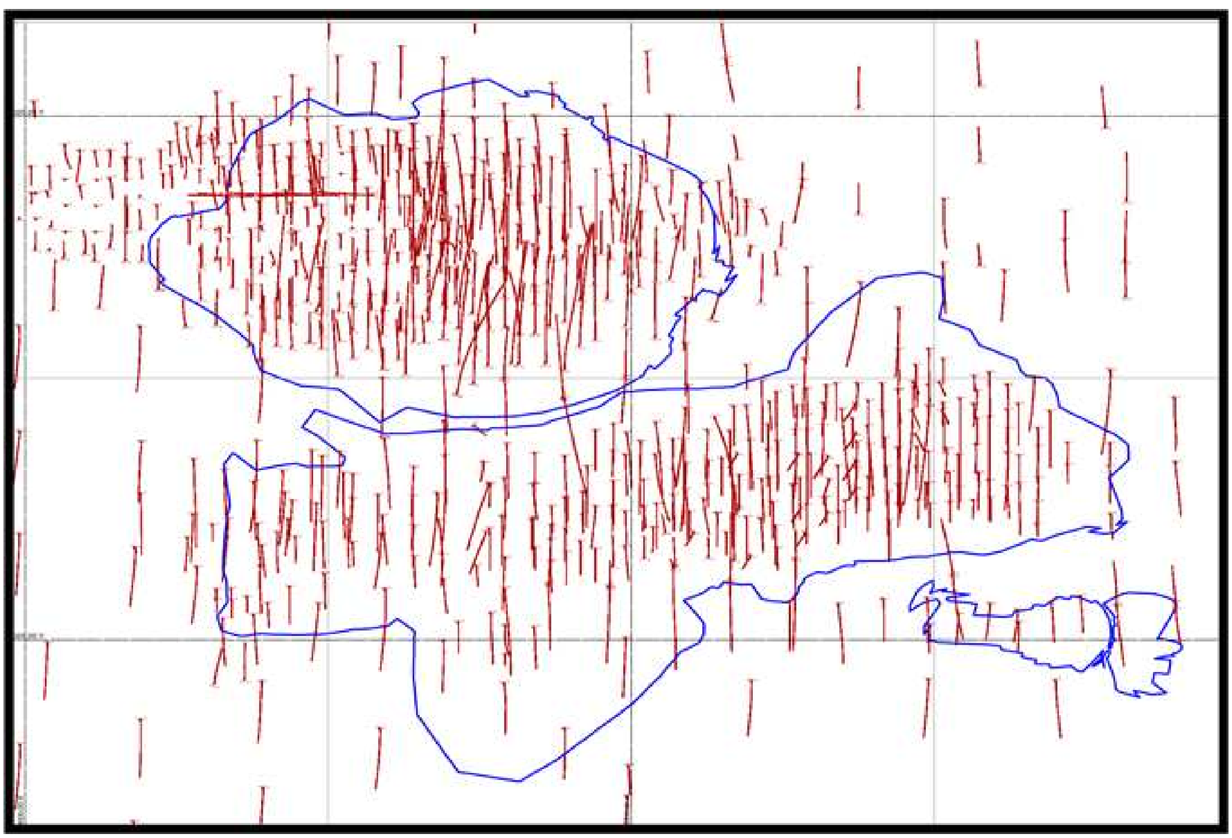


Figure 11.2 . Plan view of the diamond drilling grid at Royal Hill.

Figures 11.1 and 11.2 illustrate typical views of diamond drill patterns completed at Rosebel, together with the interpreted mineralized ore zones. The initial drilling pattern is usually completed on 100 metre x 100 metre centres, while the definition (infill) drilling pattern is reduced to 50 metres x 50 metres. In areas where mineralized zones have a more complicated geometry, 25 metre x 25 metre drilling patterns are employed.

Most holes are drilled using NQ and HQ core size with a diameter of 1.87 inches (4.76 cm) and 2.5 inches (6.35 cm) respectively. Exceptions are a number of infill definition drill holes that used a PQ core size (diameter of 3.35 inches or 8.50 cm) to provide better recovery and sample representativity in specific ore zones.

11.2. Methodology and Planning

In the Rosebel database, each drill hole has a unique identification that is linked to the project in which it was drilled and also to the sequence in which it was completed (e.g. KH-001 to KH-522 for Koolhoven as at December 31, 2009). Exploration, condemnation and resource definition holes are not differentiated.

The Mine Exploration Department typically designs drill hole programs directly onto the relevant vertical sections in GEMS, in the appropriate project. When created, the planned drill holes are identified by a prefix, stating the project and the year and a suffix that represents the order in which it was planned (e.g. KH08-001). There is no link between the suffix and the order in which the drill hole has to be drilled (programs are prioritized by the senior geologist and/or the geology superintendent).

Information such as drill hole azimuth, dip, planned hole length and special comments are noted in the appropriate areas in GEMS’ GEOddhPLAN workspace. Most holes are planned with azimuths that parallel the project’s associated vertical sections (0 / 18° or 180 / 198°) and the dip usually ranges between -46 and -65°. When the planned drill hole is completed, its name is changed to reflect the overall sequence in which it was drilled (e.g. KH08-010 to KH-058) and is imported into the workspace GEOddhOfficl.

Prior to drilling, a copy of the planned holes is printed and a meeting held between the senior geologist and the field technicians to establish an execution plan (access roads, holes spotting, pad building and drill moves). Once the access roads and drill pads have been constructed, the surveying team is sent to measure the location of the planned drill holes. Each drill hole collar is marked by the surveyors and the azimuth, dip and planned length is added by the field technician. Front sights pickets are subsequently installed on the drill pad.

Under supervision, the contractor sets the diamond drill onto the collar and the field technician aligns the drill with the help of the front sights. The inclination of the drill is fixed using a protractor. All drill holes are surveyed in the first 15 metres using the single shot function of the Flex-IT to ensure that the planned orientation and dip of the hole is respected. The hole is stopped and a new hole is collared a few metres away if the deviation from the planned azimuth and/or dip is too great.

Upon completion, drill holes are surveyed for orientation and dip every three metres from the bottom to the surface with the Flex-IT multishot/singleshot instrument. Survey data for the interval inside the magnetic casing (generally less than 50 metres) is estimated based on the last measurement taken beneath the casing and the initial singleshot measurement taken near the surface.

Core is placed in corrugated plastic boxes at the drill site and transported to the core shack by a geological technician before being laid on the logging tables.

Upon completion, drill holes are identified with a wooden post at the collar location. The sump containing the cutting rejects generated by the drilling process is closed and the pad is levelled with a dozer.

Orientation tests are not conducted. Previous methods were tested but lateritic, saprolitic and soft transition material gave poor results. Core recovery typically exceeds 90% except for lateritic and saprolitic profiles that are washed away more easily.

12. SAMPLING METHOD AND APPROACH

Sampling of Rosebel gold mineralization for the purposes of resource estimation is based mostly on diamond drill core (Fig. 12.1), but also on trenches, auger and banka drilling. Upon arrival at the core shack, drill core is washed to remove drilling fluids and residues. Geotechnical logging is then carried out to record core recovery, RQD, rock hardness and fracture density. Prior to logging, the geologist generates a unique sequence of bar coded sampling tags using a WASP printer. The core is then logged in detail for lithology, alteration, structure and veining. While logging, the geologist indicates sample intervals using the coded sample tags. The logging is performed by the junior and exploration geologists under the supervision of the core shack leader (exploration geologist) and the senior geologist. Drill holes are photographed prior to sampling.

The entire drill hole is typically sampled. Exceptions include selected drill holes from infill or condemnation programs where only potentially mineralized intervals are sampled. Samples intervals vary from 30 centimetres to a maximum of 1.5 metres depending on the style and intensity of quartz veining and/or alteration. Sample intervals do not cross lithological contacts or intervals of core loss.

The soft laterite and saprolite material is split in half using a spackle knife, while the harder transition and fresh rock intervals are split in half using a hydraulic/impact core splitter or cut in half with a diamond saw. Half of the core is placed in a plastic bag with a unique sample number and sent to the laboratory for assaying while the second half is kept in the core racks for reference and/or further testing. On occasion, whole core samples from infill diamond drill holes are sent to the laboratory to enable comparative analysis of analytical accuracy.

The sample quality is affected by core recovery. However, at RGM the core recovery is very good, generally more than 90%. The diamond drill holes that have recovery less than 85% are redrilled.

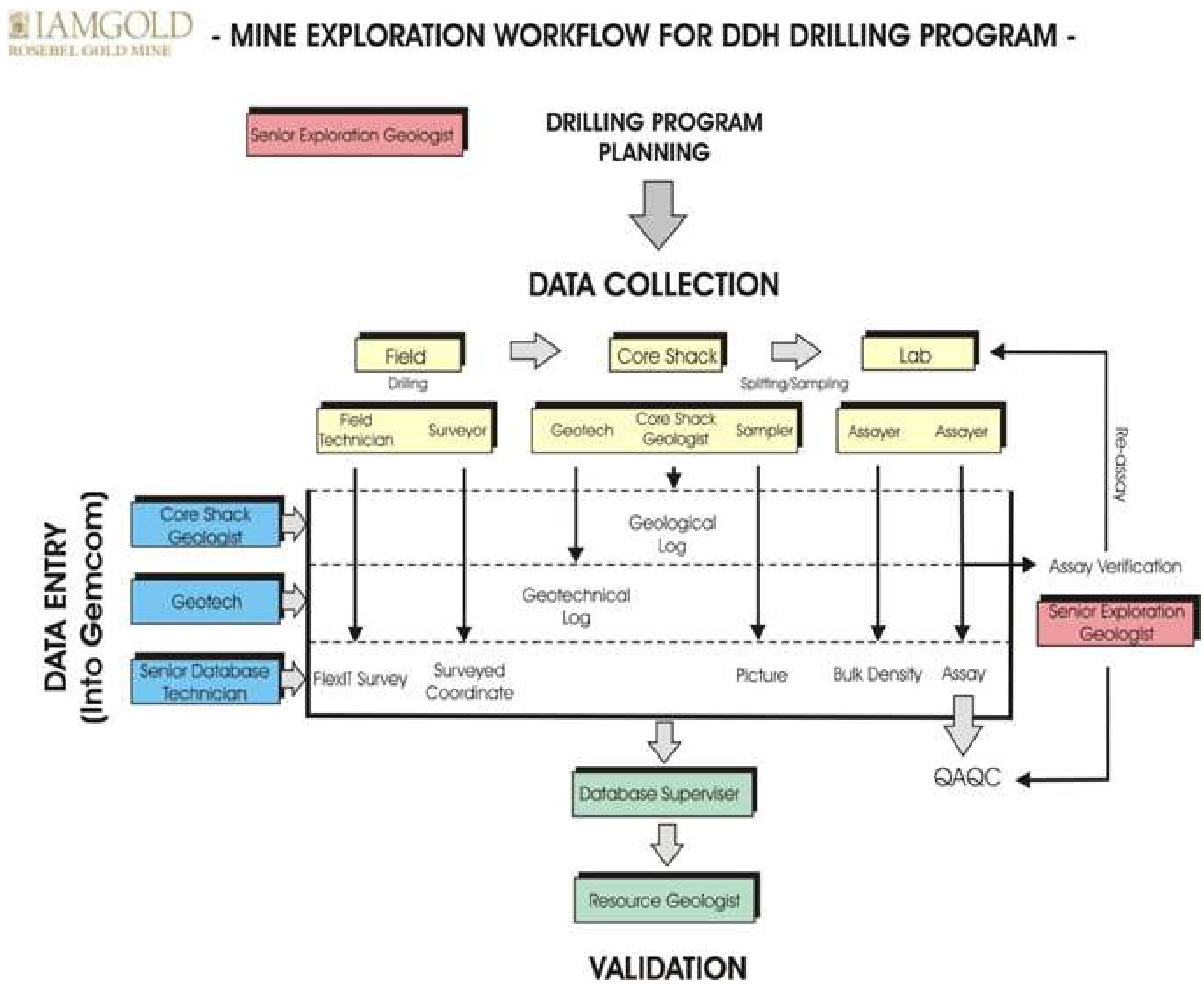


Figure 12.1. Workflow for diamond drilling programs.

13. SAMPLE PREPARATION, ANALYSIS AND SECURITY

13.1. Sample Security

Drill samples are placed in identified plastic bags and delivered to the mine laboratory with a submittal sheet. Since the opening of the mine, the assaying of mine exploration and production drill samples has been done exclusively on site by the RGM personnel. The laboratory is located within the mill complex which is fenced and has a security guard posted at the entrance at all times. Samples are validated against the submittal sheet, registered and stored as soon as they arrive at the laboratory.

In 2009, it was recognized that the budgeted 90,000 metre exploration drilling campaign, in conjunction with an increased production drilling schedule, would exceed the capacity of the RGM laboratory. It was therefore decided to submit a proportion of the mine exploration drill core samples to the FILAB laboratory in Paramaribo. Neither the RGM laboratory nor the FILAB laboratory is certified. RGM intends to commence the process of certifying the RGM laboratory in 2010 (Standards Association to be determined).

The samples sent to FILAB in 2009 were sourced from the following projects: Royal Hill drilling Phase II, J-Zone, Rosebel, Tailings Pond - Spin Zone, and Roma drilling Phase II. Samples are transported to Paramaribo by a contractor. The core shack supervising geologist validates each sample batch using the submittal sheet.

Official written procedures are made available to ensure consistency of sample preparation and fire assay, for both exploration drill holes and production blasthole samples. Prior to 2008, all assay results were manually recorded by a laboratory technician. A Laboratory Information Management System ("LIMS") provided by AssayNet was implemented in 2008 to automate data collection, tracking and management. All assay data is now collected by the LIMS and exported as CSV files to a folder that is accessed by the geologists.

13.2. Sample Preparation

Samples from each drill hole are submitted to the laboratory as a unique batch. At the laboratory, samples are placed in large pans and dried in an oven for approximately three hours at 150°C. Cooled samples are crushed; initially with a Bico-Badger crusher to -6 mesh (samples with a grain-size of greater than 5 cm are pre-crushed) and then with a Bico UA Pulveriser to ≈75% passing -8 mesh. One in every 25 samples is screened for percentage passing -8 mesh.

Samples are riffle-split to produce a representative ≈600g cut of the original sample. The remaining sample material is placed into a plastic bag (reject) and kept by the laboratory for use in the QAQC protocol (see Section 14). The 600g sample is pulverized using Bico UA pulverizers to ≈85% passing -170 mesh (pulp). One of every 22 samples is screened for percentage passing 170 mesh and results are recorded on a fire assay worksheet. A sand wash is used once between each sample, and twice where samples containing visible gold have been identified. The pulp is then homogenized and a 30g aliquot removed for fire assay (Fig. 13.1).

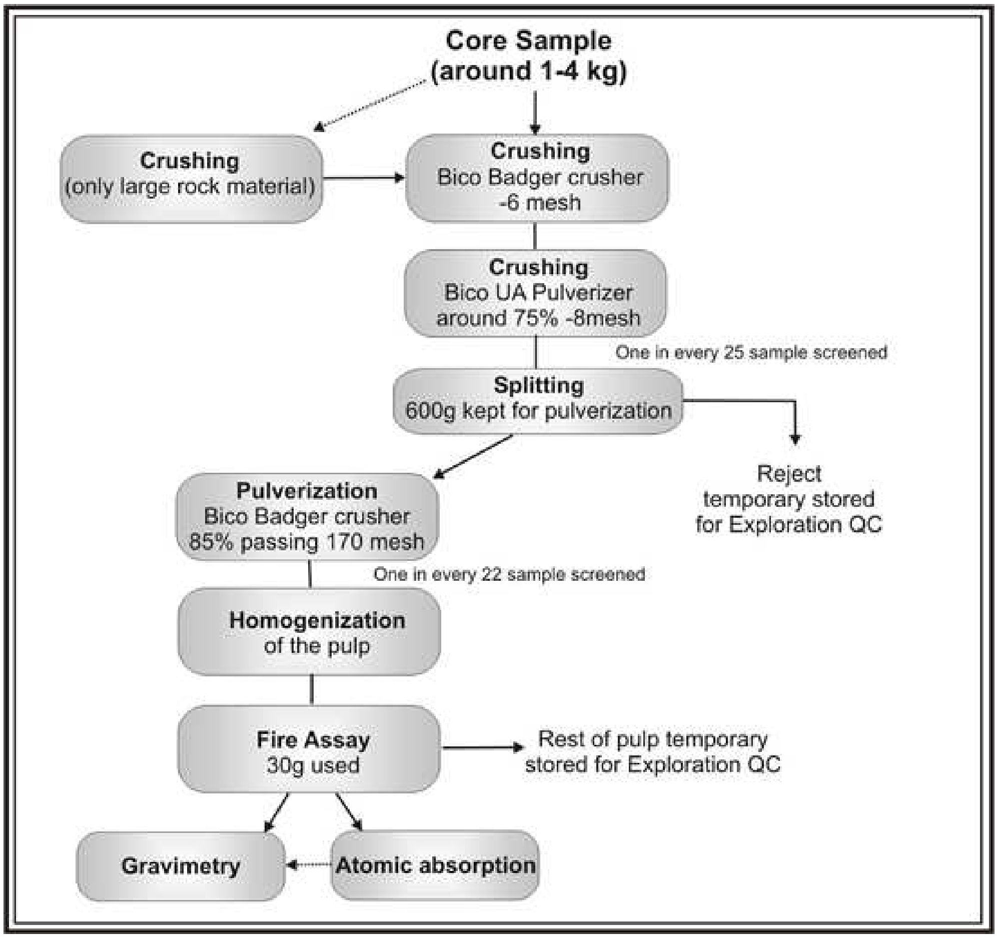


Figure 13.1. Exploration sample preparation flow sheet.

13.3. ANALYSIS

The 30g aliquot is mixed with the appropriate flux and a silver nitrate solution. Fusion of the sample occurs in a furnace after 50 minutes at 1010°C. When cooled, the lead containing the gold is separated, placed in a pre-fired cupel and positioned in the furnace at around 950°C. When the lead volatilizes, the remaining gold-silver prill is collected for atomic absorption finish.

Most gold analyses have been made by atomic absorption. This technique makes use of absorption spectrometry to assess the concentration of an analyte in a sample. In short, the electrons of the atoms in the atomizer are instantaneously promoted to higher orbitals by absorbing a set quantity of energy (i.e. light of a given wavelength). This amount of energy (or wavelength) is specific to a particular electron transition in a given element, and in general, each wavelength corresponds to only one element. This gives the technique its elemental selectivity. As the quantity of energy (the power) put into the flame is known, and the remaining quantity can be measured at the detector, it is therefore possible to calculate how many of these transitions took place and thus get a signal that is proportional to the concentration of the element being measured (Fig. 13.2).

At the RGM laboratory, the gold is not analyzed directly but rather in solution. The sample is placed in test tubes and digested in HNO_3. HCL and distilled water are added and silver chloride is formed. When all the silver has settled, the solution is read by atomic absorption. In cases of high gold concentrations, the sample may be analyzed by the gravimetric method of directly weighing the gold. A computer is connected to the spectrometer and records all the collected assays.

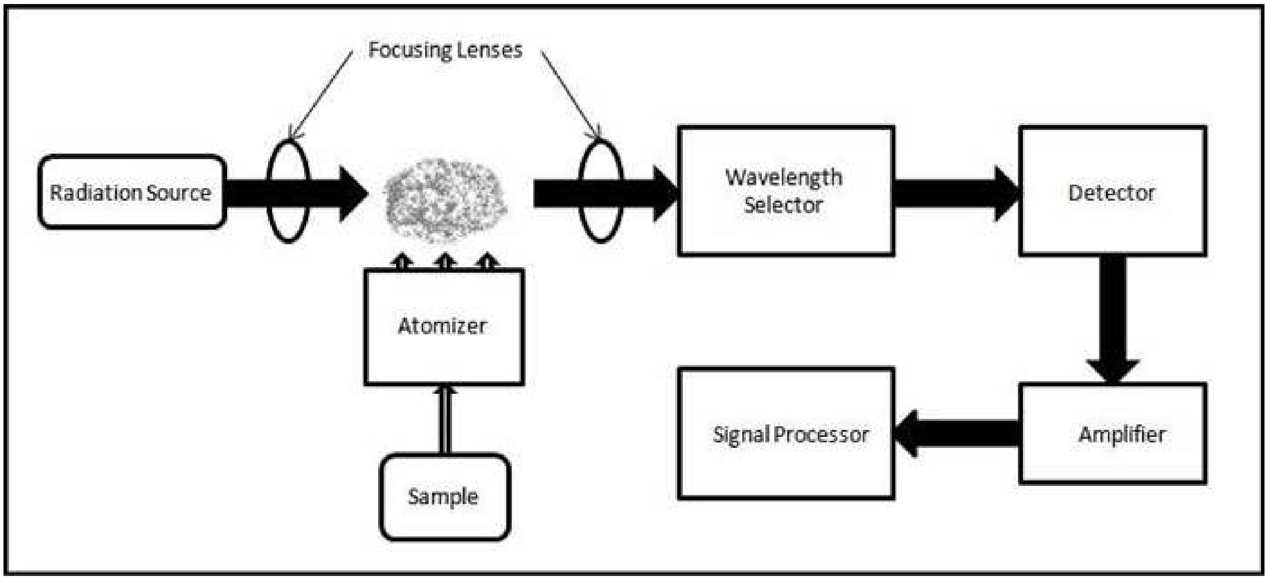


Figure 13.2. Schematic representation of the atomic absorption (AA) method.

14. DATA VERIFICATION

Sample control is maintained through documentation by means of shipment invoicing by batch with signature approval. This documentation is always important for chain-of-custody requirements. Training requirements for Analyst and Sample Preparation Technicians have been documented through our ongoing training program, which include equipment preventative maintenance, operation and calibration. Internal and external checks are carried out routinely in order to maintain the highest possible standard controls. The RGM and FILAB laboratory quality control is made both internally and by the client (production and exploration).

The QP of this report has verified the data referred to here in and believes that this data can be relied upon for reserve estimates.

14.1. LABORATORY QUALITY CONTROL PROCEDURES – MINE EXPLORATION

The various steps of the mine exploration QA/QC protocol are illustrated on Figure 14.1.

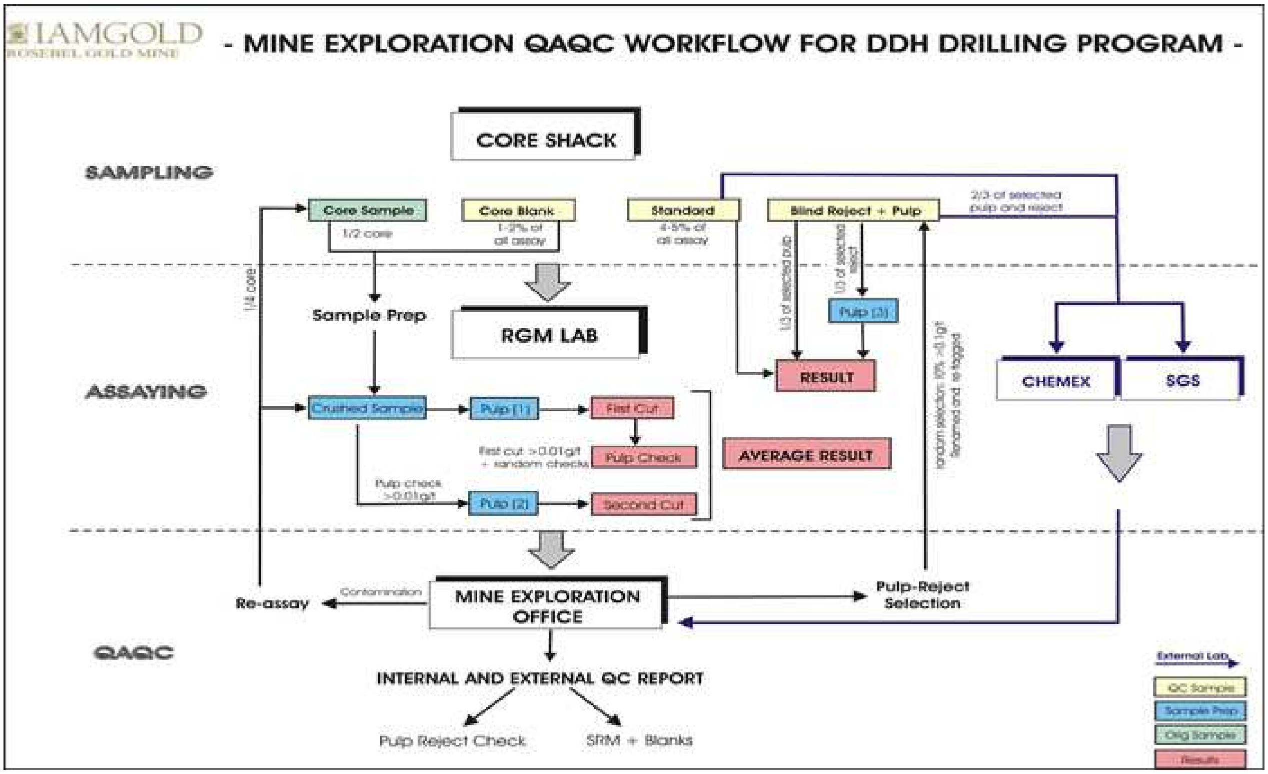


Figure 14.1 . Mine Exploration QA/QC Workflow Sheet – Drill Core Samples.

In 2009, it was recognized that the budgeted 90,000 metre exploration drilling campaign, in conjunction with an increased production drilling schedule, would exceed the capacity of the RGM laboratory. It was therefore decided to submit a proportion of the mine exploration drill core samples to the FILAB laboratory in Paramaribo. The samples sent to FILAB in 2009 were sourced from the following projects: Royal Hill drilling Phase II, J-Zone, Rosebel, Tailings Pond - Spin Zone, and Roma drilling Phase II.

A total of 20.9% and 5.0% of all sample pulps returned from the RGM and FILAB laboratories respectively were re-assayed as an internal check for pulverizing, pulp rolling and fire assay. In addition, all coarse rejects and pulps from samples grading at or above 0.2g/t Au were re-assayed at each laboratory as another internal check procedure, to maintain proper crushing sizing, sample reduction through splitting and homogeneity between results. Re-assay of coarse rejects from RGM and FILAB laboratories accounts for 8.7% and 9.3% of the 2009 drilling database respectively. In house prepared pulp standards were run with each fire assay fusion in order to ensure proper fluxing and to prevent accidental mix ups. These standards are also a control for Atomic Absorption standardization. Instrument standardization and pulp standard values are confirmed bimonthly with CANMET traceable **MA-2c** pulp standards.

The comparison of the 2009 original assay results with the pulp (for samples grading over 0.2 g/t Au) and coarse reject re-assays show that both the RGM and FILAB laboratories have been able to reproduce similar results for the vast majority of submitted samples (Table 14.1).

2009					
Number of Samples	**Test done on**	**Check Samples**	**% of database**	**RGM lab Original Assay Average (g Au/t)**	**RGM lab Second Assay Average (g Au/t)**
43,246	Pulps Coarse rejects	9,019 3,748	20.9% 8.7%	0.94 1.44	0.93 1.45
Number of Samples	**Test done on**	**Check Samples**	**% of database**	**FILAB lab Original Assay Average (g Au/t)**	**FILAB lab Second Assay Average (g Au/t)**
21,227	Pulps Coarse rejects	1,056 1,966	5.0% 9.3%	0.12 1.69	0.12 1.70

Table 14.1. Internal Quality Control - drill core samples.

A total of eighteen different Rocklabs Standard Reference Materials ranging from 0.20 to 14.90 g/t Au were randomly inserted into the half-core sample sequence in order to investigate on the accuracy of the laboratories. For the RGM Laboratory, the results indicate that the grade coming out of the laboratory is only 1% lower than the grade suggested by the nominal values of the 1,264 standards used in 2009. However, the standard deviation of the laboratory is 25%, which means that results returned from the laboratory on the standardized materials varies within a +/- 25.0% margin. For FILAB, the average grade is 2% higher than the nominal values, while the standard deviation is 12 % (grade within a margin of +/- 12.0 %).

Analytical blanks were submitted anonymously and placed within the sample sequence immediately after potentially mineralized half-core samples. The results proved that significant contamination did not occur at either laboratory, with 90% of the analytical blanks for RGM and 95% for FILAB passing the test of grading less than 0.1 g/t Au.

The main conclusion to be drawn from the 2009 QA/QC results is that the assaying equipment is properly calibrated, and that the manipulation procedures are adequate.

14.1.1. Analytical Blanks and Standards

Since 2005, the RGM laboratory's performance has been monitored through the use of analytical blanks and Standard Reference Materials ("SRM"). The same control materials were used to assess FILAB's performance in 2009. The results are presented on Tables 14.2a and 14.2b using a new template based on the LIMS QC module.

Blank samples are prepared from barren core usually taken from condemnation drill holes. As some variability is expected in the gold content of these blanks, they are considered to be contaminated only if the assay value is above 0.1 g/t Au. A sand wash is used once between each sample in sample preparation (crushing and pulverizing), and twice where samples containing visible gold have been identified. Blanks are typically inserted into the sample sequence immediately following a visible gold occurrence or promising-looking vein. In 2009, 90% of the blind blanks submitted to the RGM Laboratory returned gold values below 0.1 g/t Au, with an average grade of 0.08 g/t Au. For FILAB, 95% of the blanks passed with an average grade of 0.05 g/t Au. Consequently, the laboratories do not appear to have major sample contamination problems.

The LIMS QC module was used to investigate the performance of the SRM. This software provides a detailed data presentation to help to isolate and address problems. Thorough conclusions can be drawn in order to improve the quality and reliability of the laboratories.

RGM Laboratory

A total of 50 (4%) of the 1,264 SRM sent to the laboratory in 2009 were classified as outliers. These outliers represent gross errors, usually relating to mislabelling of SRM sample bags, sample inversion during manipulation in the laboratory or solution spilling while performing assays. A closer analysis reveals that a few specific SRM contributed a disproportionate number of outliers.

The RGM laboratory average grade shows a very slight bias (1%) when compared to the SRM nominal values. The 2009 laboratory performance therefore qualifies as accurate. However, it appears that the RGM laboratory is experiencing accuracy problems for some of the specific SRM, both oxide and sulphide (Table 14.2).

YTD - 2009	Blind to all - RGM Lab													
RGM Lab	W	Sul F	Sul G	OxG	Sul H	OxH	OxJ	OxL	OxM	Sul M	OxN	OxO	OxP	Sul N
Total	95	102	1	30	104	5	1	94	4	95	108	102	76	89
Total (without outlier)	92	100	1	27	101	5	1	89	4	92	106	98	71	84
Number outlier	3	2	0	3	3	0	0	5	0	3	2	4	5	5
Lab average	1.29	0.57		0.21	1.28	0.42		1.26	0.68	0.81	2.27	14.52	0.20	2.54
Standard deviation	0.12	0.09		0.03	0.17	0.04		0.10	0.14	0.09	0.36	2.31	0.03	0.21
% of Outlier	3	2	0	10	3	0	0	5	0	3	2	4	7	6
Lab average (%)	94	100	98	114	103	103	89	98	111	97	101	96	106	96
Standard Deviation (%)	14	43		28	43	10		16	22	24	48	23	28	18

YTD - 2009	Blind to all - RGM Lab			
RGM Lab	Sul P	OxQ	OxR	Sul O
Total	94	83	89	92
Total (without outlier)	87	81	86	89
Number outlier	7	2	3	3
Lab average	5.57	0.79	3.41	3.25
Standard deviation	0.77	0.06	0.30	0.35
% of Outlier	7	2	3	3
Lab average (%)	90	100	94	92
Standard Deviation (%)	23	13	17	16

1264	Total standard
1214	Total standard without outliers
50	Total number of outliers

4%	% outliers
99%	RGM Lab average (%) vs STD "real" value
25%	Standard deviation (%) of RGM Lab

BLANKS		
2009	**Failed**	**88**
	Passed	**760**
	Total	**848**
	%Passing	**90%**
	Lab Avrg	**0.08**

Table 14.2. Internal Quality Control – RGM blind standards – drill core samples.

An average standard deviation of 25% for 2009 indicates a lack of precision. Calculated from RGM laboratory's own assays results, this overall value reveals problems concerning laboratory performance as it relates to the random errors that occur in the assaying process. It appears that the RGM Laboratory is not able to reproduce its own results within an acceptable confidence interval that normally should not exceed 5-10%. The fact that no specific SRM or grade range is highlighted by the analysis suggests either manipulation errors or non-optimal laboratory procedures. The lack of qualified personnel and overload of the existing crew may have resulted in diversions from protocols and procedures while performing the assays. The general performance of the RGM Laboratory is not in critical state but it obviously needs to be improved.

To improve RGM laboratory performance, a periodical QC assessment is transmitted to senior assayers and laboratory supervisors on a regular basis, allowing them to evaluate potential problems and adjust the laboratory workflow accordingly.

FILAB Laboratory

The percentage of recorded outliers is 5%, with a total of 30 outliers out of the 629 SRM sent to the laboratory in 2009. Mishandling is suspected as being the reason for the high percentage.

The FILAB average grade is shows a slight bias when compared to the SRM nominal values. Laboratory performance is therefore considered accurate (Table 14.3).

An average standard deviation of 12% calculated from FILAB laboratory's own assay results indicates a rather poor performance in terms of reproducibility, although is mainly due to two problematic SRM.

FILAB overall performance on the SRM in 2009 qualifies as accurate but lacking precision. The success rate of blank samples is well within accepted limits and reveals no contamination problems throughout the sample preparation process.

YTD - 2009	Blind to all - FILAB Laboratory										
FILAB Laboratory	W	Sul F	Sul H	OxL	Sul M	OxN	OxO	OxP	Sul N	Sul P	OxQ
Total	54	37	49	43	54	49	48	31	48	44	55
Total (without outlier)	51	36	47	41	53	47	42	25	47	43	52
Number outlier	3	1	2	2	1	2	6	6	1	1	3
Lab average	1.37	0.62	1.35	1.32	0.83	2.40	14.91	0.20	2.54	5.90	0.82
Standard deviation	0.04	0.05	0.05	0.03	0.02	0.05	0.16	0.01	0.11	0.12	0.02
% of Outlier	6	3	4	5	2	4	13	19	2	2	5
Lab average (%)	99	113	103	103	100	101	100	103	95	99	102
Standard Deviation (%)	4	55	4	3	3	3	2	5	8	10	4

YTD - 2009		
FILAB Laboratory	OxR	Sul O
Total	55	62
Total (without outlier)	54	61
Number outlier	1	1
Lab average	3.61	3.61
Standard deviation	0.11	0.06
% of Outlier	2	2
Lab average (%)	107	103
Standard Deviation (%)	42	8

629	Total standard
599	Total standard without outliers
30	Total number of outliers

5%	% outliers
102%	FILAB Lab average (%) vs STD "real" value
12%	Standard deviation (%) of FILAB

BLANKS		
2009	**Failed**	**23**
	Passed	**444**
	Total	**467**
	%Passing	**95%**
	Lab Avrg	**0.05**

Table 14.3. Internal Quality Control – FILAB blind standards – drill core samples.

14.1.2. Reject and Pulp Re-assays

When results are received from the laboratories, two random 10% selections are made amongst samples grading 0.1 g/t Au or higher for QA/QC purposes: one from pulverized samples and the other from coarse rejects. Rejects are re-tagged and submitted to RGM or FILAB laboratories. Randomly selected pulps are re-bagged and re-numbered, then are separated into 3 groups: the first group of pulps is re-submitted to the RGM Laboratory or FILAB for internal control purposes, while the second and third groups are sent to ALS Chemex and SGS Laboratories respectively for external control.

RGM Laboratory

In 2009, 900 coarse reject samples were sent for internal checks at the RGM Laboratory, with 29.8% plotting within the 20% deviation limit. For the pulp material, only 25% of the 228 samples sent to the RGM Laboratory for internal checks plot within the accepted 10% deviation limit (Fig. 14.2).

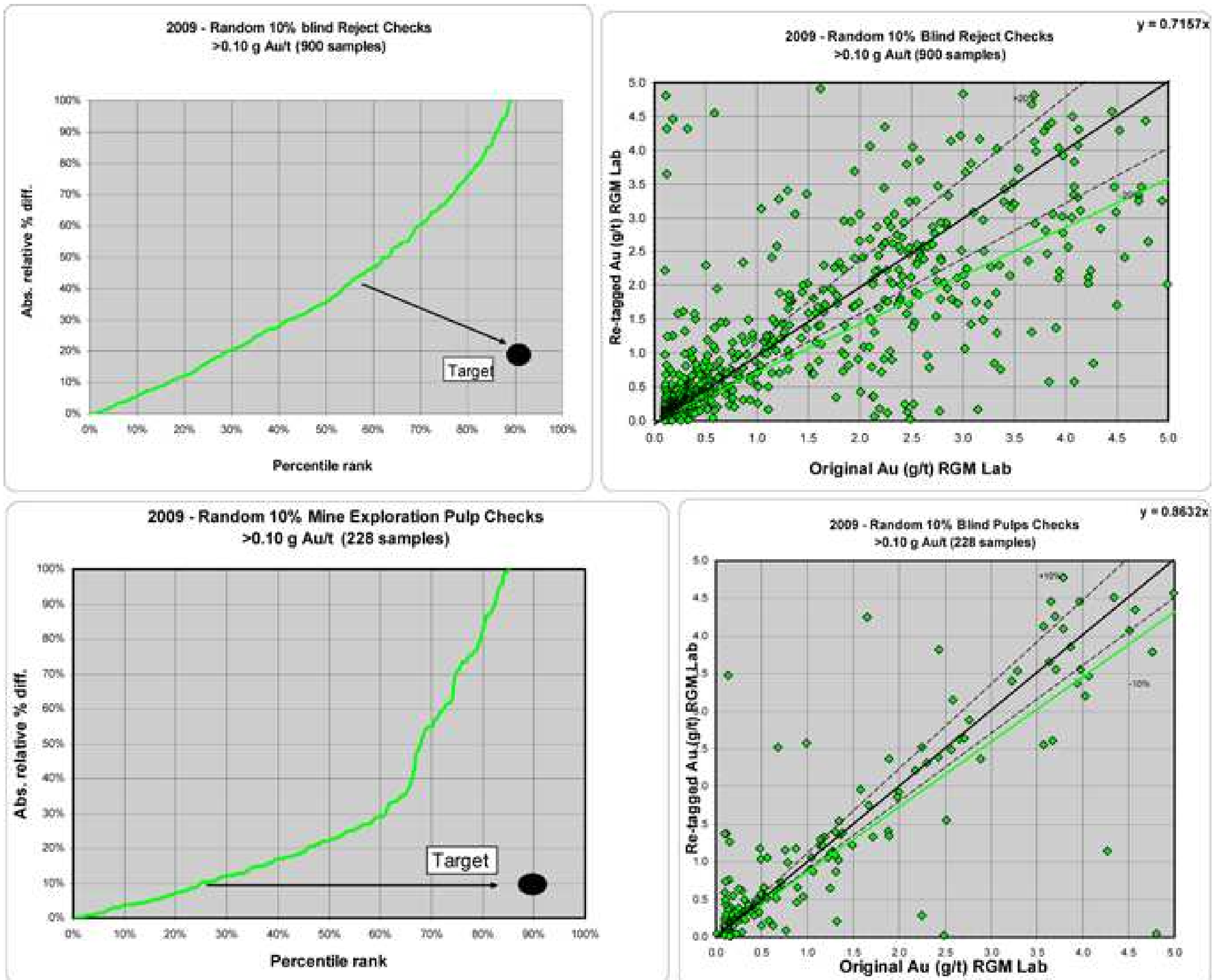


Figure 14.2. Internal assay checks on pulps and coarse rejects – RGM laboratory.

Results indicate a serious problem with the pulp material which seems to behave like a reject. The low rate of success for the coarse rejects is largely expected, especially when considering the nugget effect. That the behaviour of the pulp material is no different is problematic. The sample preparation processes, especially sample homogenization, require immediate review in order to ensure reliable results for mine exploration samples.

The 2009 results for the coarse rejects and pulp sample checks reveal a very significant bias for the rejects as well as a slight bias for the pulp samples, with the average trend line indicating an RGM laboratory average grade that underestimates both checked rejects and pulps (Fig. 14.2).

FILAB Laboratory

The re-assay procedures described for the RGM laboratory are likewise used for FILAB. The rate of success for blind check samples that are re-tagged and sent to FILAB is illustrated on Figure 14.3.

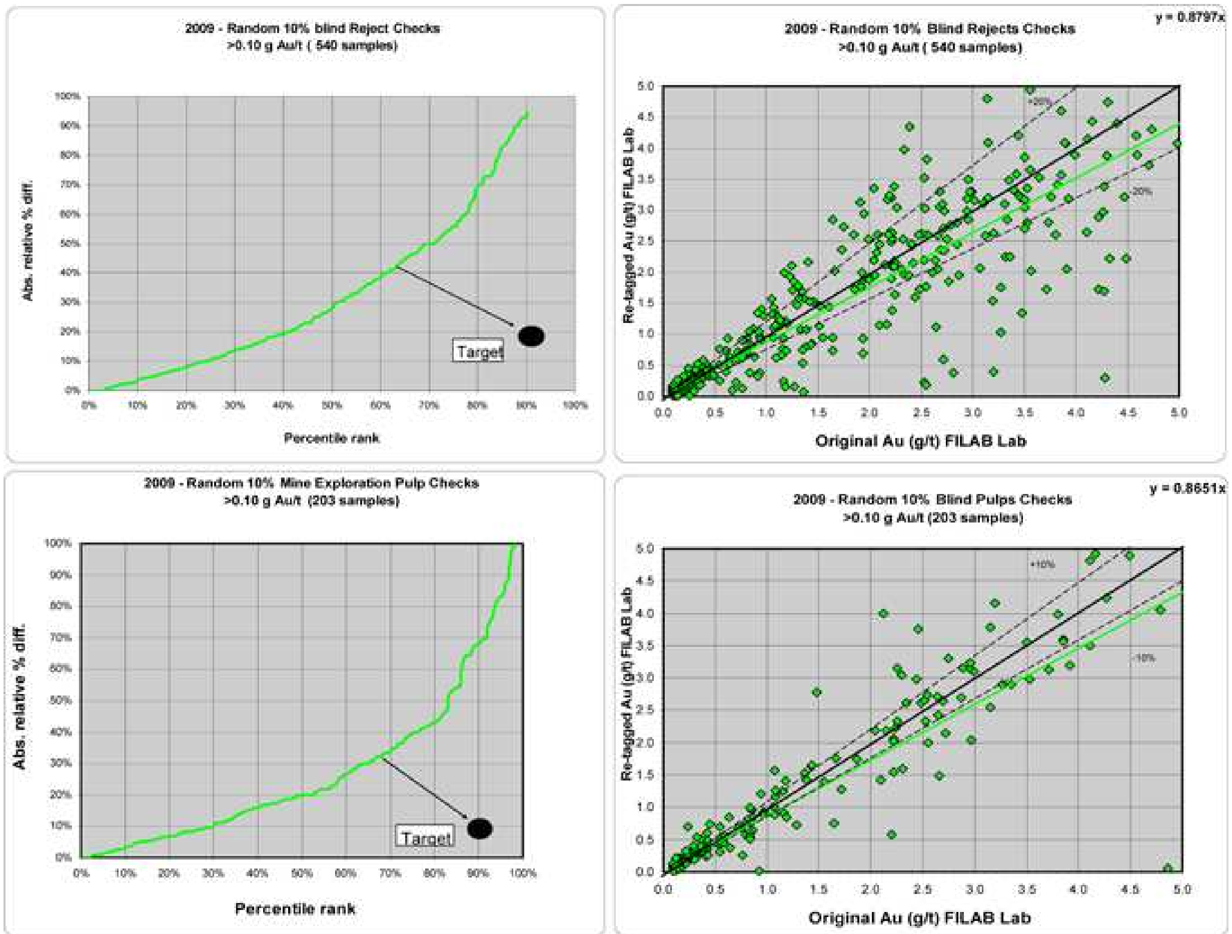


Figure 14.3. Internal assay checks on pulps and coarse rejects – FILAB laboratory.

In 2009, 540 coarse reject samples were sent for internal checks at the FILAB, with 41.5% plotting within the 20% deviation limit. For the pulp material, approximately 30% of the 203 samples sent to the laboratory as internal blind checks plot within the accepted 10% deviation limit. Based on the coarse reject results, FILAB shows an undervaluation bias of 12% compared to the 28.4% determined for the RGM laboratory.

For the pulps, FILAB appears to underestimate grade by 13.5% compared to the 13.7% estimated for the RGM laboratory.

It appears that FILAB is confronted with a similar problem to that discussed for the RGM laboratory, with pulp material behaving like a reject. The sample homogenization process is suspected to be at fault. The reproducibility of the rejects and pulp checks, even if not meeting RGM targets, is at least better than the RGM laboratory results.

14.1.3. Verification by external laboratories

The quality of the assay results generated by the FILAB and RGM laboratories is also assessed using external laboratories. Duplicate pulp samples are randomly selected, re-bagged and re-numbered, and then separated into three groups. The first group of pulps is re-submitted to the RGM Laboratory or FILAB as is the case, while the second and the third groups are sent to ALS Chemex and SGS laboratories respectively. Pulp samples are re-bagged as received, i.e. no splitting of the sample is performed.

Assay results from both external laboratories had not been received at the time of writing. The last batch of 2009 samples was shipped at the end of the drilling campaign in December, and results from the first half of the year are still pending. The quality assessment for 2009 will therefore be presented in next year's report. The following discussion on external QC data is based on 2008 data.

Analytical Standards

The same analytical SRM used for internal checks are sent to SGS and ALS Chemex laboratories, inserted within the pulp check sequence after every five samples.

For the ALS Chemex laboratory, a total of 49 SRM were assayed (Table 14.4). No outliers were recorded and laboratory average grade was within 1% of the SRM nominal values, demonstrating a good accuracy of results. The average standard deviation is within a 5% range indicating good precision.

For SGS Laboratories, the percentage of outliers was 2%, with only one outlier from the 55 SRM sent to the laboratory (Table 14.4). The laboratory shows good accuracy, with average grade within 2% of the nominal SRM values, and good precision with an average 6% standard deviation. The laboratory experienced minor problems with a few specific SRM that reported lower accuracy and precision of results.

YTD - 2008	Blind to all - ALS Chemex										
ALS Chemex Lab	Sul1	Sul2	Sul3	Ox11	Ox1	Sul4	Ox2	Ox3	Ox5	Ox6	Sul5
Total	4	4	2	4	3	3	3	7	3	4	4
Total (without outlier)	4	4	2	4	3	3	3	7	3	4	4
Number outlier	0	0	0	0	0	0	0	0	0	0	0
Lab average	6.01	0.50	4.23	1.04	0.19	1.36	0.40	1.89	1.27	0.58	0.83
Standard deviation	0.09	0.11	0.02	0.03	0.03	0.02	0.02	0.06	0.03	0.07	0.02
% of Outlier	0	0	0	0	0	0	0	0	0	0	0
Lab average (%)	102	84	105	102	93	103	96	101	99	95	100
Standard Deviation (%)	1	18	1	3	15	1	5	3	2	12	2

YTD - 2008	Blind to all - ALS Chemex				
ALS Chemex Lab	W	Ox7	Sul6	Ox8	Ox9
Total	2	1	1	1	3
Total (without outlier)	2	1	1	1	3
Number outlier	0	0	0	0	0
Lab average	1.41	14.70	2.67	0.79	3.59
Standard deviation	0.00				0.03
% of Outlier	0	0	0	0	0
Lab average (%)	102	99	101	98	100
Standard Deviation (%)	0				1

49	Total standard
49	Total standard without outliers
0	Total number of outliers
0%	% outliers
99%	ALS Chemex average (%) vs SRM real value
5%	Standard deviation (%) of ALS Chemex Lab

YTD - 2008	Blind to all - SGS Lab										
SGS Laboratories	W	Sul3	Sul4	Sul5	Sul6	Ox2	Ox3	Sul7	Ox4	Ox5	Ox7
Total	4	1	5	5	7	8	5	3	4	3	1
Total (without outlier)	1	1	5	5	7	7	5	3	4	3	1
Number outlier	3	0	0	0	0	1	0	0	0	0	0
Lab average	1.38	0.80	5.83	0.47	2.79	0.89	0.19	1.30	0.41	1.75	1.30
Standard deviation	0.07		0.29	0.23	1.93	0.36	0.01	0.06	0.05	0.15	
% of Outlier	75	0	0	0	0	13	0	0	0	0	0
Lab average (%)	100	96	99	94	99	100	93	99	100	94	101
Standard Deviation (%)	5		5	3	6	3	4	5	11	8	

YTD - 2008	Blind to all - SGS Lab				
SGS Laboratories	Ox8	Sul8	Ox9	Ox10	Sul9
Total	7	2	2	1	1
Total (without outlier)	7	2	2	1	1
Number outlier	0	0	0	0	0
Lab average	0.60	0.82	0.20	0.80	2.64
Standard deviation	0.04	0.09	0.01		
% of Outlier	0	0	0	0	0
Lab average (%)	98	98	98	99	100
Standard Deviation (%)	6	11	3		

55	Total standard
54	Total standard without outliers
1	Total number of outliers
2%	% outliers
98%	SGS Lab average (%) vs STM real value
6%	Standard deviation (%) of SGS Lab

Table 14.4 . External QC – ALS Chemex & SGS laboratory blind standards.

Reject and Pulp Re-assays

For the assay checks, only pulps are sent to the external laboratories to avoid the "nugget effect" and the potential segregation that coarse rejects may sustain during transport. For ALS Chemex, only 28.3% of the total 297 pulp checks assayed within a 10% deviation limit relative to the RGM laboratory original values, with an overvaluation of 6% on the grade reproducibility (Fig. 14.4). This rate of success is similar to that observed for the internal pulp checks at the RGM laboratory. The comparison also shows a slight bias (linear regression slope of 1.06), suggesting that the RGM laboratory slightly underestimates the grade. This finding is consistent with the observations revealed by the SRM results returned from the RGM laboratory.

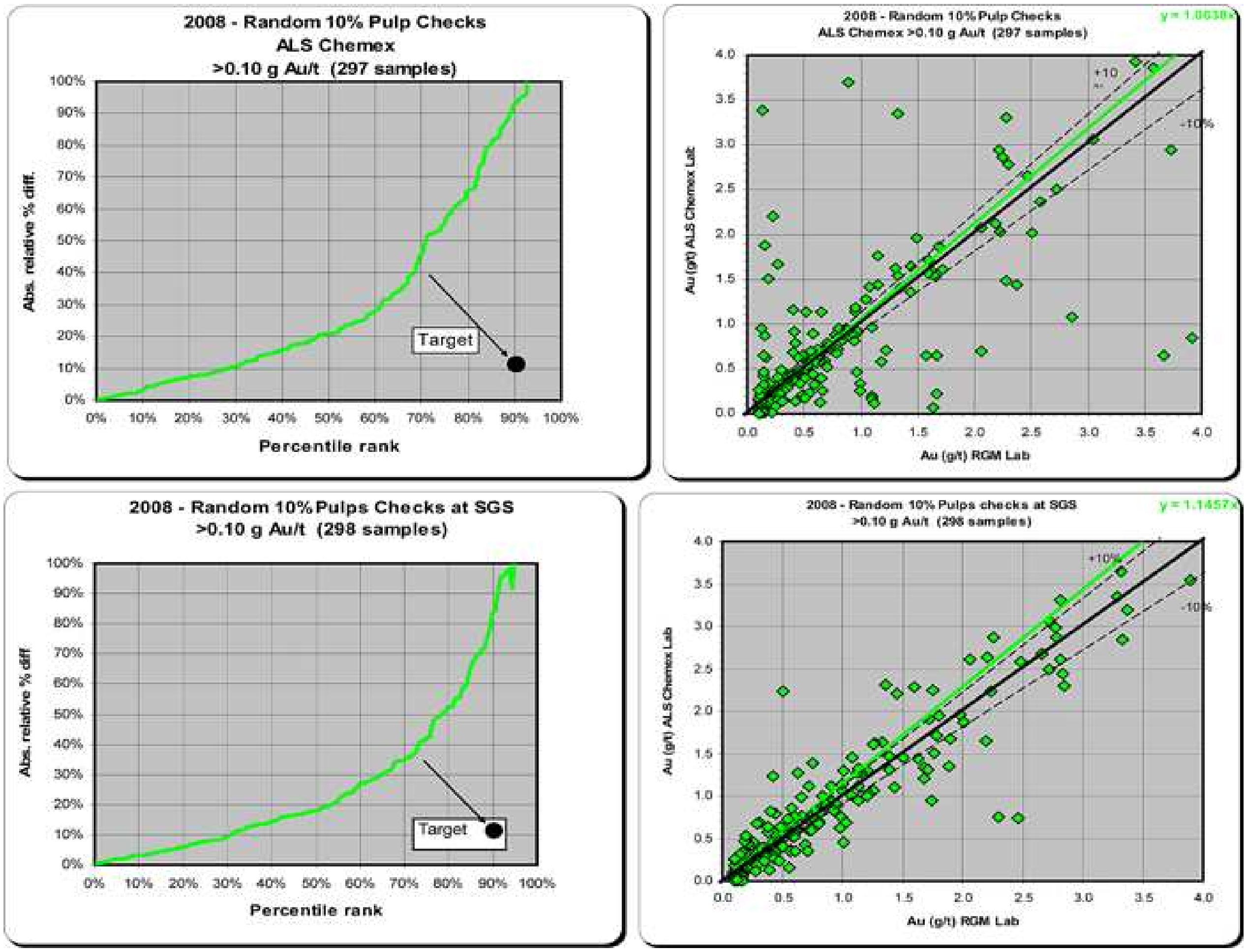


Figure 14.4. External assay checks on pulps – ALS Chemex & SGS laboratories.

A total of 298 pulp checks were sent to SGS laboratories in 2008. Only 30.8% of the assay results plot within a 10% deviation limit when compared to the original RGM laboratory values, with an overvaluation of 14% on the grade reproducibility (Fig. 14.4). Again, these results are similar to the internal checks performed at the RGM Laboratory. The comparison between SGS and RGM laboratory results reveals a bias (linear regression slope of 1.14) that suggests an underestimation of the grades by the RGM laboratory. This is in keeping with its performance on SRM in 2008.

The pulp check results from both external laboratories indicate that the RGM laboratory was slightly underestimating the grade in 2008, which is in agreement with the RGM laboratory performance on SRM for the same period. In addition, the rate of success for pulp checks plotting within the 10% deviation limit (28.3% for ALS Chemex and 30.8% for SGS) is no better than the performance of internal pulp checks at the RGM laboratory in 2008 (31%). These results point to a serious issue in the sample preparation process (pulp homogenization) at the RGM laboratory.

14.2. Laboratory Quality Control Procedures – Production Department

Assay data management and Quality Control (QC) reporting are performed through the Laboratory Information Management System ("LIMS") in the RGM laboratory. This system stores all metadata associated with assaying in a single database from which we are able to track the assaying process from reception of the sample to the approval of results. All samples are identified by a unique barcode tag at each step of the assaying process to reduce the potential for mishandling errors.

QC is ensured by processing internal method blanks, internal standard reference materials, pulp duplicates, coarse rejects duplicates and blind standard reference materials. Each fire assay tray comprises 21 blasthole samples, 1 internal method blank, 1 internal standard reference material and 1 duplicate sample. The duplicate is randomly chosen from the blasthole samples in the same tray, and is alternatively derived from the pulp or coarse reject of the sample. Moreover, 4 blind samples of standard reference material are randomly inserted into the process every day to provide additional information on calibration.

The coarse rejects from blastholes are randomly re-assayed to maintain proper crush sizing and sample reduction through splitting. Only those coarse reject re-assays returning grades above 0.1 g/t Au are considered as check samples for quality control (2,092 assays, representing 0.7% of all samples in 2009). Additionally, 5,678 (2.0%) of the pulps re-assayed in 2009 returned grades over 0.1 g/t Au to serve as checks for pulverizing, pulp rolling, and fire assaying. In-house prepared pulp standards and Rocklabs commercial standards are run with the fire assay fusion to ensure proper fluxing and prevent accidental mix ups. These are also a control for Atomic Absorption Standardization.

A comparison of the 2009 original assay results with the pulp check assays shows that the RGM laboratory has been able to reproduce similar results for the majority of blasthole samples (Table 14.5), with a slight bias towards underestimation of the duplicate. The average grade returned by the coarse reject check assays is slightly higher than the original assay, indicating a minor positive bias in the sample splitting process.

Laboratory accuracy is monitored by evaluating the performance of analytical standards. Outlier values are determined using the Grubb test, a statistical method that considers the performance trend of the laboratory. The outlier values are excluded from the average and standard deviation calculations. 2009 results reveal an overall average grade 1% above the theoretical value of the standards and a 5% standard deviation (Table 14.6). Only 1% of the samples returned outlier values. Analytical blanks generally tested negative for gold, indicating no major contamination issues.

2009					
Number of Samples	Test done on	Check Samples	% of database	RGM lab Original Assay Average (g Au/t)	RGM lab Second Assay Average (g Au/t)
287,828	Pulp	5,678	2.0%	1.06	1.03
	Coarse rejects	2,092	0.7%	0.83	0.87

Table 14.5. Internal Quality Control - blasthole samples.

	OxK48	OxH55	OxL63	SF30	SH 35
Total	82	2954	4994	1983	2731
Total (without outlier)	82	2934	4948	1966	2673
Number outlier	0	20	46	17	58
% of Outlier	0	1	1	1	2
Lab average (%)	101	105	100	101	100
Standard Deviation (%)	4	6	3	7	5

12744	Total standard
12603	Total standard without outliers
141	Total number of outliers
1%	% outliers
101%	RGM Lab average (%) vs standard "real" value
5%	RGM Lab Standard deviation (%)

	BLANKS
Total passed	14134
Total failed (> or = 0.1)	31
% Passing	99.8%
Lab average	0.012

Table 14.6 . Analysis of internal blanks and standards.

The variability in both diamond drill and blasthole assay results prompted the inclusion of blind standard materials in blasthole sample batches. The same type of SRM used for the diamond drilling program was purchased in ready-to-use 30g pouches. Four blind SRM samples were randomly inserted into fire assay trays per day. Results show more variability than revealed by internal standards (Table 14.7). A total of 7% of the SRM assays were recorded as outliers. The standard deviation calculated for each standard type varies between 6% and 20% (averaging 10%) while the overall average grade is 1% above the theoretical value of the standards.

2009	OxA	OxB	OxC	OxD	OxE	SulA	SulB
Total	75	72	73	74	69	84	50
Total (without outlier)	71	67	71	65	68	76	46
Number outlier	4	5	2	9	1	8	4
% of Outlier	5	7	3	12	1	10	8
Lab average (%)	102	102	98	100	98	103	99
Standard Deviation (%)	10	7	10	6	21	8	9

2009	SulC	HiSilA	SulD	SulE	OxF	OxG	OxH
Total	68	92	64	82	60	34	32
Total (without outlier)	61	88	58	76	55	33	27
Number outlier	7	4	6	6	5	1	5
% of Outlier	10	4	9	7	8	3	16
Lab average (%)	101	98	99	98	101	110	109
Standard Deviation (%)	9	8	8	6	8	14	20

929	Total standard
862	Total standard without outliers
67	Total number of outliers
7%	% outliers
101%	RGM Lab average (%) vs standard "real" value
10%	Standard deviation (%) of RGM Lab

Table 14.7. Internal Quality Control – blind standards – blasthole samples.

14.3. Summary of QA/QC Analysis

The QA/QC program at Rosebel Gold Mine is thorough and includes: 1) internal checks made by the laboratory (RGM and FILAB), 2) external checks made by the Mine Exploration and Geology (production) Departments and 3) an external laboratory check (Chemex and SGS) conducted by Mine Exploration. Control samples include randomly selected pulp duplicates and reject duplicates, standard reference materials and blanks.

In 2009, the RGM and FILAB laboratory performance is considered to have been reasonably accurate but lacking in precision. The laboratory has difficulties in reproducing its own assays within an acceptable level of confidence. Adjustments must be made immediately in order to improve next year's performance. Sample preparation processes, especially sample homogenization, require immediate review in order to ensure reliable results. The RGM laboratory and FILAB were informed of the QA/QC results and have agreed to establish additional tests and implement various process improvements in 2010 to provide higher quality results.

14.4. Database Verification

Prior to 2007, all data was stored in a Jet 4.0 database. In 2007, data was migrated into a SQL server 2000 database. This is a relational database management system in which all projects are separated into different databases. SQL resides on a computer server under the responsibility of the IT department and all users are connected by the network to any data stored in the databases. Only the database administrator and IT personnel are allowed to work directly on the station which hosts the SQL server (by remote connection). A database maintenance plan ensures that a backup of each database is made on a daily basis to prevent from permanent data loss. Moreover, SQL server allows the database administrator to set different permission levels for users, as a function of their profile group (geology, planning, engineering) or individually.

The database stores the exploration data in a structured series of related tables where a header table "lies over" related secondary tables. For exploration drill hole data, this header table contains the hole identification number (hole-id) and its related information. Secondary tables store data related to different domains such as geological information, assays and drill hole surveys. Users typically access the database via *GEMCOM version 6.1.4.2* , the software that is used by Geology, Exploration, Engineering, Planning and Surveying. In *GEMCOM* , each database relates to a specific project. These projects comprise various workspaces that contain both the primary and secondary database tables. Restricted permissions prevent most of the users from modifying the structure of those workspaces, which at the same time would modify the structure of the database itself. Table 14.8 illustrates the detailed workspace structure used for all projects.

Exploration data (geology, survey and assay results) is initially entered by a technician into a Microsoft Access database. This allows rapid data manipulation and retrieval which facilitates the import into *GEMCOM* official drill holes workspaces. *GEMCOM* provides a validation tool during imports which includes a cross-check for overlapping and missing intervals. Additionally, the database administrator personally validates every import to verify that all data has been correctly imported and that no data is missing. Following QA/QC analysis and database validation, the data is considered suitable for mineral resource calculation.

TABLE	FIELD
LITHOLOGY	HOLE-ID
	FROM
	TO
	ROCK TYPE
	ROCK CODE
	COMMENTS
	EXTRA #1
	EXTRA #2
	NOTE_PRIM
	NOTE_STRUC
	NOTE_DEFOR
	NOTE_ALTER
	NOTE_MINER
	NOTE_VEIN
STRUCTURE	HOLE-ID
	FROM
	TO
	FEATURE
	ANGLE CA
	COMMENTS
	AZIMUTH
	DIP
	S0/S1
DENSITY	HOLE-ID
	FROM
	TO
	SAMPLE_NO
	MATERIAL
	LITHOLOGY
	HARDNESS
	WEIGHT_AIR
	WEIGHT_DRY
	W_IMMERSED
	MOISTURE%
	SG_DRY
	ROCK CODE
	COMMENTS
VEINS	HOLE-ID
	FROM
	TO
	VN_CM
	QZ%
	CARB%
	TO%
	ACC_MIN
	ACC%
	ACC_MIN2
	ACC%2
	CA
	VG
	NOTES
	AZIMUTH
	DIP
	TYPE
	S1/VN
	TEXTURE
WEATHERING	HOLE-ID
	FROM
	TO
	ROCK_CODE
	EXTRA #1
FORM_GEOL	HOLE-ID
	FROM
	TO
	LENGTH
	ROCK CODE
	ROCK TYPE
	EXTRA #1
	COMMENT
MONTHLY	HOLE-ID
	FROM
	TO
	LENGTH
	ROCKCODE
	AVG AU G/T
	COMP-ID
STEEL	HOLE-ID
	FROM
	TO
	INTERVAL
	COMMENT
PRIM-TEXT	HOLE-ID
	FROM
	TO
	TEXTURE1
	TEXTURE2
	GRAIN_SIZE
	COMMENTS
DEFORM	HOLE-ID
	FROM
	TO
	INT_DEF
	TYPE
	COMMENT
MINERAL	HOLE-ID
	FROM
	TO
	PY
	PO
	CPY
	GN
	SPH
	AU
	TEXTURE
	COMMENTS
PICTURE	HOLE-ID
	FROM
	TO
	LINK-FILE

Table 14.8. Detailed workspace structure for official project workspaces.

15. ADJACENT PROPERTIES

Other prospective areas surrounding the Rosebel mining concession include the Thunder Mountain, Headley's Reef and Triangle Rights of Exploration, which are all held by Rosebel Gold Mines ("RGM"; Fig. 15.1). RGM is engaged in a multi-year exploration effort on these exploration concessions. Numerous large soil geochemical and geophysical surveys are on-going to generate drill targets within reasonable economic distance from the Company's operation.

15.1. THUNDER MOUNTAIN RIGHT OF EXPLORATION

The Thunder Mountain concession covers a V shaped area that is contiguous with the northern, eastern and southern boundaries of the Rosebel mining concession. Exploration targets therefore occur within a similar geological setting, and in part, cover the continuity of the Rosebel mineralized trends. Targets are prioritized based on regional panning results, evidence of past mining work (artisanal mining (porknocker) areas or old mine sites), historic exploration results (e.g. Golden Star), results of recent geochemical and geophysical surveys, recent drilling and subsequent interpretation.

Exploration activities completed by RGM since 2004 have included extensive geochemical sampling and ground geophysical surveys. The latter comprised ground magnetic and Induced Polarization surveys in the northern part of the concession. An airborne magnetic survey flown by Golden Star Resources in 1990 covering the whole concession has been reprocessed. Detailed geological mapping has also been carried out over outcrops found along cut lines cut or exposed in porknocker areas.

Systematic auger surveys and MMI geochemistry were used to assess a number of priority areas selected for their favourable geological setting and location relative to the known mineralized trends that extend from the Rosebel mining concession. These priority areas are: Mamakreek, Kompanie Creek, Bergendal and Koemboe Creek.

Mamakreek is on the same mineralized trend as the Koolhoven and J-Zone deposits. In 2009, 35 diamond drill holes totalling 4,850 metres and 25 RC drill holes totalling 1,675 metres were completed between March and November. Drilling tested known near-surface mineralization, structural targets, geochemical anomalies and anomalous geophysical responses (magnetic and IP). The 2009 exploration program demonstrated the potential for Mamakreek to add a small resource to the Rosebel inventory. Other geochemical anomalies to the south-east of the Mamakreek zone will be assessed early in 2010.

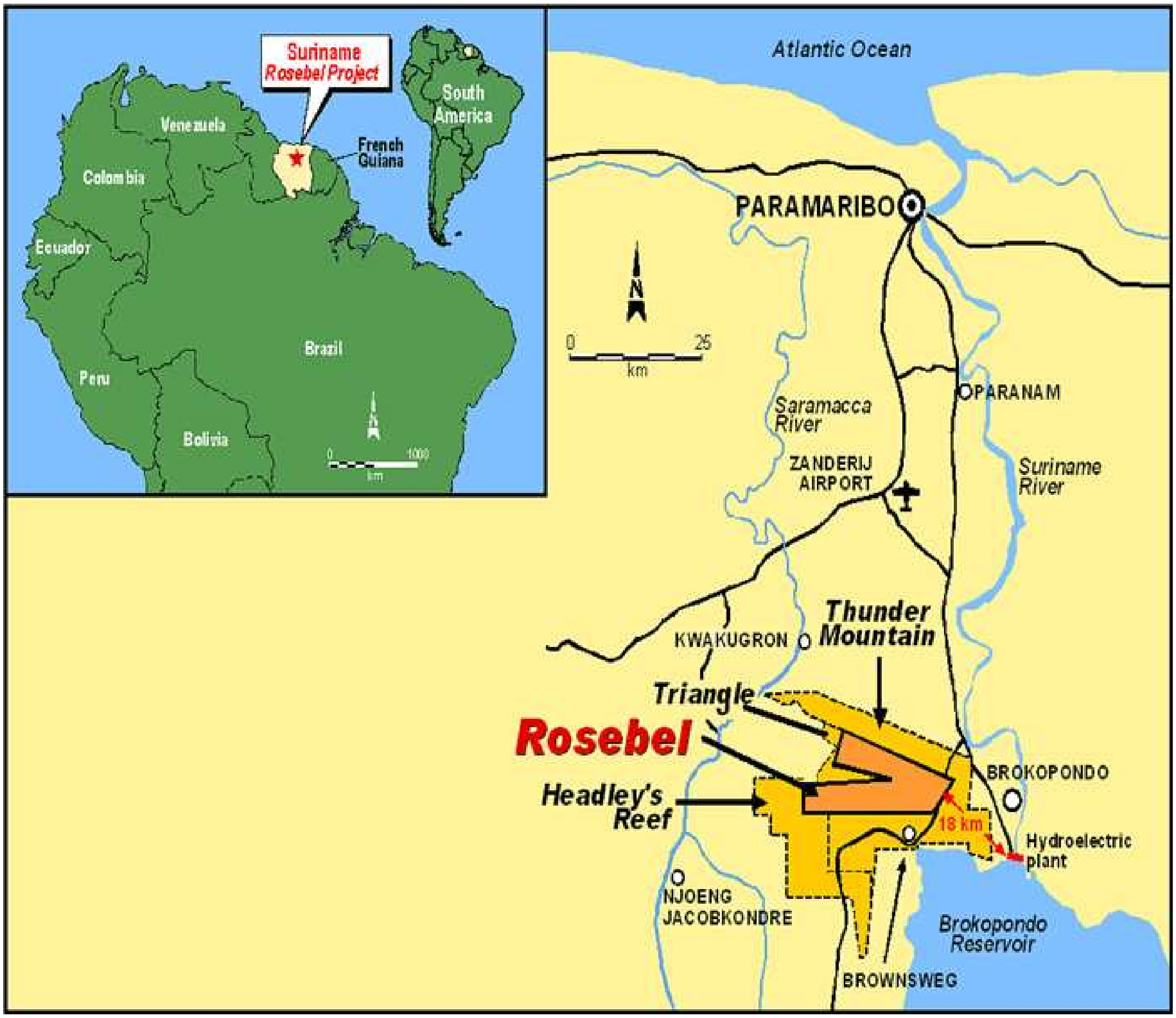


Figure 15.1. Location of the Thunder Mountain, Headley's Reef, and Triangle Rights of Exploration

15.2. HEADLEY'S REEF RIGHT OF EXPLORATION

The Headley's Reef concession lies to the southwest of the Rosebel mining concession and adjoins the Thunder Mountain concession to the east. The northern part of the

concession covers the western extension of the mineralized trend that hosts the Royal Hill, Roma and Mayo deposits and is underlain by a similar geology to the Rosebel mining concession. The southern part is underlain, at least in part, by volcanic and sedimentary units of the older Paramaka Formation and granitic-gneissic units of the younger Saramacca Complex.

Exploration work carried out by RGM since 2004 has involved geochemical auger sampling and geological mapping over outcrops found along cut lines cut or exposed in porknocker areas. The Golden Star airborne geophysical survey carried out in 1990 covers the entire concession.

The Kraboe Doin area is currently the highest priority target on the concession. It was initially defined by anomalous stream sediment sampling results and lies close to a significant area of small-scale mining activity located at the common boundary of the Headley's Reef and Thunder Mountain concessions. Systematic deep auger sampling and detailed geological mapping have defined several targets for diamond drilling in 2010. The southern extension of the Blauwe Tent trend (from the Rosebel mining concession) and the Koemboe Creek area (extending from the Thunder Mountain concession) are other priority areas where deep auger sampling and mapping have been carried out. Several anomalies have been delineated and further work is planned for 2010, including diamond drilling around the southern margin of the Brinks Granite.

15.3. TRIANGLE RIGHT OF EXPLORATION

Due to its position and limited surface area (4,225 ha), the Triangle concession has always been incorporated into exploration programs designed for the Rosebel mining concession. 2009 exploration work has expanded the line cutting and auger sampling coverage to cover the possible extension of the central mineralized trend which hosts the Rosebel deposit. Project-scale geologic mapping has been carried out to better define geological controls on mineralization and has contributed to the selection of several target areas for follow-up.

The area is mostly underlain by the shallow-water sedimentary succession, with occasional "windows" exposing the deep-water sedimentary succession. The area is covered by a thick layer of alluvial white sands and gravels. The 2008 RC drilling program identified favourable geology and alteration but returned only weak gold grades. MMI geochemical surveys completed in northern and southern parts of the concession highlighted several gold anomalies 5km northwest of the Mayo pit. These were tested by deep auger sampling at the end of 2009 with results expected by the end of the 1st quarter of 2010.

16. MINERAL PROCESSING AND METALLURGICAL TESTING

The metallurgical process is conventional grinding followed by leach, carbon in leach (CIL) with a gravity circuit installation for the recovery of gravity recoverable gold. The process was developed to accommodate varying ratios of soft rock, transition and hard rock ores. The process used at Rosebel Gold Mines was developed through various pilot plant programs and through additional initiatives by mill personnel to improve the process further to commissioning.

Further process optimization continues to target constraints and opportunities to further increase plant capacity and performance. Control strategies have been implemented to better control plant operational parameters with net benefits in process stabilization / automation. Ore characterization is currently in process which will establish mill work indexes and leach characteristics, grinding capacity and leach kinetic definition will be further defined resulting in increased plant capacity and recovery rates. Process optimization has also been extended to gravity / leach CIL through process audits currently underway, expected results will target improved recoveries.

A reference flow sheet is shown on Figure 16.1. To summarize the process, run of mine ore reports to the mill in one of two forms: hard rock which is crushed to minus 6” and soft rock reporting directly to feeders. Blended ore is fed to a SAG mill followed by two ball mills in parallel with cyclone classification and overflow reporting to thickener then leach / CIL, with approximately 20% of the cyclone underflow reporting to gravity. Carbon from CIL is processed through stripping then electro-winning, where gravity gold is recovered using cones, a Knelson concentrator and Diester table. Products from both circuits are melted using an induction furnace at the on-site refinery.

Since 2004, there have been two expansions targeted at increasing the throughput of higher ratio hard rock feed while maintaining recovery rates. Process throughput has increased from a 2004 daily average of 14,400 tpd to current production levels that exceed 35,000 tpd.

16.1. PROCESS PLANT

16.1.1. Comminution

Primary crushing is carried out using an Allis / Superior 54 x 74 gyratory crusher; product sizing is set at minus 6”, with an active hard rock stockpile maintained at approximately 50,000 tonnes. Soft rock feed is maintained through two grizzlies using one D9 dozer and two

Caterpillar 345 backhoes, with an active stockpile maintained at approximately 200,000 tonnes. Hard rock/soft rock blending to the SAG feed conveyor is maintained by two

Svedala 1.8m x 6.9m apron feeders respectively; typical feed ratio is 400 tph hard rock, 1,200 tph soft rock.

The SAG mill is a 7,500HP 30’ x 13’ Allis operating in an open circuit feeding two Simplicity 2.4m x 4.8m vibratory screens. Screen oversize is processed through a Metso 3’ cone crusher with 1/4” product returning to SAG. Two 4,500HP 16.5’ x 27’ Allis ball mills operating in parallel closed circuit receive SAG discharge screen undersize.

Classification is maintained through two banks of eight 26” cyclones; cyclone overflow discharge parameters are 80% passing 200 mesh at 35% solids.

Reagent additions including lime and cyanide are initiated at the SAG mill feed chute/discharge pump box with a secondary addition point at leach feed.

16.1.2. Gravity

Approximately 20% of the cyclone underflow reports to two OSNA 1.2m x 4.9m vibratory screens fitted with 4 mesh panels. Oversize reports to ball mill discharge pump boxes, undersize reports to four banks of Reichert cones at a rate of 180tph. Cone concentrate reports to a 30” Knelson concentrator via a magnetic drum separator. Knelson concentrate is upgraded to 80% purity on a Deister table prior to smelting. Gravity recovery represents up to 20% of gold production.

16.1.3. Thickening / Leach / CIL

Two Delcor DLS 2000 trash screens configured with 24 mesh screen cloths precede thickening. Oversize reports to an OSNA vibratory screen for wood removal, undersize reports to gravity tails. Trash screen undersize reports directly to a Westech 53m x 2.9m high rate thickener with underflow pumping maintained with two $^{12}/_{10}$ pumps.

Leach/CIL are configured as two parallel circuits, with one leach followed by six CIL tanks each. Residence capacity is 23 hours at 35,000tpd / 48% solids. Total carbon inventory is maintained at 540 tonnes. Further optimization initiatives to improve recovery rates will be completed by mid 2010 and will include four additional leach tanks in two parallel circuits consisting of two leach tanks followed by seven CIL tanks each for a retention time increase of 8 hours.

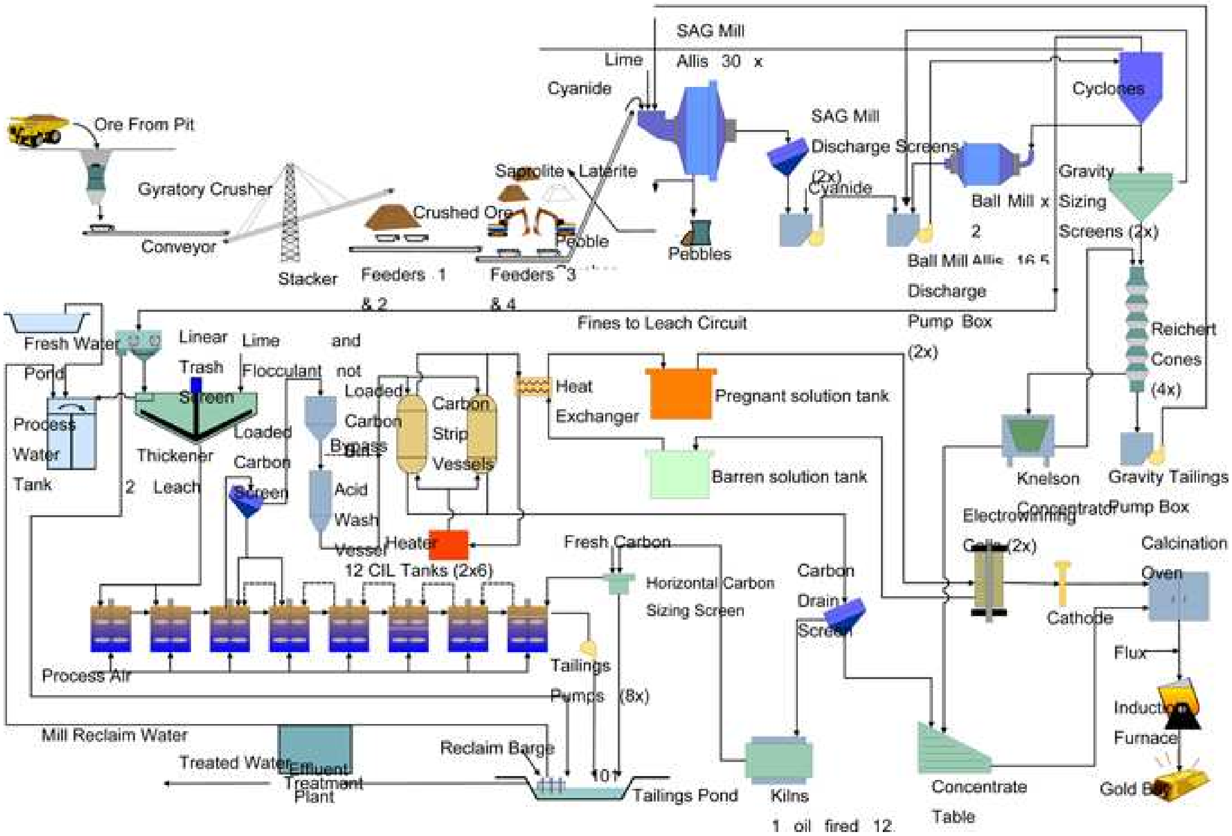
16.1.4. Stripping and Reactivation

Stripping of loaded carbon is maintained with two 10 tonne vessels at a rate of 3.0 strips per day at 85% efficiency, with strip solution reporting to two electro-winning cells running in parallel at 85% efficiency. 100% of loaded carbon is acid washed prior to stripping. Carbon regeneration is accommodated with two kilns; a 12 tonne oil fired kiln and a 4 tonne electric kiln.

16.2. CONCLUSION

Increasing mill production capacity has presented many challenges to both the milling and mining groups. However, substantial gains have been realized for both groups following a number of process optimizations. Continued efforts will focus on maximizing production throughput in addition to increasing recovery rates.

Figure 16.1. Rosebel Gold Mines N.V. – Process Flowsheet



17. MINERAL RESOURCE AND MINERAL RESERVE ESTIMATES

17.1. DATABASE

A validation is conducted by the resource geologist in the drilling database. The information needed to perform resource and reserve estimation is transferred into a separate workspace that is assigned specifically for the purposes of resource work. Table 17.1 details the elements included in the resource workspace. For operating pits, the blast holes are included in the calculation of resources.

Deposit or Zone	Diamond Drill			Auger Drill			Banka Drill			Trenching			Total	2009
	Holes	m	Samples	Holes	m	Samples	Holes	m	Samples	Channel sampling	m	Samples	Samples	Samples
Koolhoven/Bigi	565	79,497	56,706	703	3,953	4,009	0	0	0	2,205	10,122	4,941	65,656	9,375
"J" Zone/Noutoe	264	40,126	28,622	0	0	0	0	0	0	843	3,124	1,669	30,291	8,131
Pay Caro	582	89,278	65,229	435	2,233	2,267	0	0	0	1,757	7,446	3,350	70,846	5,626
East Pay Caro	198	30,811	23,629	135	580	600	0	0	0	697	3,070	1,464	25,693	0
Mayo	575	77,551	58,385	711	5,750	5,758	291	781	933	464	4,579	1,719	66,795	17,129
Roma	200	31,601	23,386	737	2,361	600	0	0	0	0	0	0	24,186	9,928
Royal Hill	858	113,461	84,803	663	5,914	5,923	375	1,237	1,376	966	4,662	4,697	96,799	6,522
Rosebel	249	33,805	25,659	375	2,214	2,215	227	827	936	265	1,116	1,120	29,930	4,177
Spin/Tailings Pond	48	5,585	2,989	254	1,878		0	0	0	2,376	979		3,681	2,372
Mama Kreek	4	438		157	1,249		0	0	0	1,539	801		2,488	0
TOTAL	**3,543**	**502,153**	**369,408**	**4,170**	**26,132**	**21,572**	**893**	**2,845**	**3,245**	**11,112**	**35,899**	**18,960**	**416,365**	**63,260**

Note : *(1) Data from GEOddhOffici working database*
(2) Contains the old and the new data added by Regional Exploration in 2005-2007
(3) Total Samples column includes 2009 Samples
(4) Washout samples and No Samples are not included

Table 17.1. Summary of database used for December 2009 resource calculations.

17.2. MODELING

Modeling work is done using the *GEMS version 6.1.4.2* software package. The main lithologies, structural elements, weathering profiles and ore zone models of each deposit are constructed using 3D rings created on 25 metre, evenly spaced cross sections.

The weathering profiles, saprolite, transition and rock, are determined from the geotechnical classifications and measurements taking on the core by the geotechnicians. The laterite model is designed from the geological observations completed by geologists on the core.

Ore zone modeling is strongly guided by the geological models of each project which provide lithological and structural constraints. Generally, ore zones envelopes are drawn from assays in the drilling database with a gold content higher than 0.3 g/t Au and include no more than three cumulative metres of assays of less than 0.3 g/t Au. Ore zones must be at least four metres thick in saprolite and at least five metres thick in transition and fresh rock, except for the Mayo deposit where a three metre minimum thickness is tolerated in some cases to accommodate thinner ore zones.

From the 3D rings drawn on the sections, surfaces and solids are built and validated. For some projects including blast holes (East Pay Caro, Pay Caro, Koolhoven, Royal Hill, and Mayo), the ore solids are built from two sets of 3D rings. One set is designed on section with drill hole assays and a second set is designed on bench where blast holes assays can be more easily viewed. Both sets are then attached together to create a full 3D skeleton of each ore zone.

17.3. Statistical Analysis

17.3.1. Compositing

Gold grade statistics from the set of composites are calculated in *Snowden Supervisor* software by deposits and by rock group (Table 17.2). The two limits (*High Grade Limit* and *High Grade Transition Limit*) that are used in the treatment of high grade results during resource estimation are determined from those statistics (see section 17.7 for details). The *High Grade Limit* corresponds to outliers observed in histogram plots. The *High Grade Transition Limit* corresponds to inflexion points representing different grade domains on the curve of cumulative probability plots.

Deposit	Ore Zone	Rock Code	Composites Length	HIGH GRADE LIMIT Capping on Ellipsoid	Number of Composites	Au grade (g/t)		Coefficient of Variation
						Avg	Std. Dev.	
Koolhoven	Laterite - BH	5	5m, ≥1m	10	10,198	0.65	2.62	4.04
	Mineralized Zones - BH	8297-8369	5m, ≥1m	10	44,090	0.67	2.72	4.06
	Laterite - BH & DDH	5	3m, ≥0.6m	10	14,415	0.65	2.36	3.65
	Mineralized Zones - BH & DDH	8297-8369	3m, ≥0.6m	10	53,788	0.65	2.54	3.93
	Laterite - DDH	5	3m, ≥0.6m	10	4,217	0.64	1.54	2.43
	Mineralized Zones - DDH	8297-8369	3m, ≥0.6m	10	9,698	0.53	1.40	2.65
JZone	Laterite - DDH	5	3m, ≥0.6m	12	1,590	0.52	1.45	2.80
	Mineralized Zones - DDH	8297-8367	3m, ≥0.6m	12	4,985	0.56	1.84	3.31
Pay Caro	Laterite - BH	5	5m, ≥1m	6	13,294	0.54	1.22	2.24
	Mineralized Zones - BH	8013-8069	5m, ≥1m	6	123,729	1.03	2.09	2.03
	Host Rocks - BH	8106; 8203; 8204; 8303	5m, ≥1m	6	191,007	0.27	1.02	3.83
	Laterite - BH & DDH	5	5m, ≥1m	20	15,334	0.57	1.31	2.31
	Mineralized Zones - BH & DDH	8013-8069	5m, ≥1m	20	129,902	1.02	2.09	2.04
	Laterite - DDH	5	5m, ≥1m	20	2,040	0.72	1.80	2.49
	Mineralized Zones - DDH	8013-8069	5m, ≥1m	20	6,173	0.93	2.09	2.25
East Pay Caro	Laterite - BH & DDH	5	5m, ≥1m	11	4,568	1.12	1.86	1.66
	Mineralized Zones - BH & DDH	8059-8088	5m, ≥1m	12	16,056	1.24	1.82	1.47
	Host Rocks - BH	8106; 8203; 8204; 8301; 8303	5m, ≥1m	4	12,101	0.04	12.86	336.53
	Laterite - DDH	5	5m, ≥1m	10	965	1.11	2.09	1.88
	Mineralized Zones - DDH	8059-8088	5m, ≥1m	12	2,529	0.93	1.99	2.14
Mayo	Laterite - DDH	5	5m, ≥1m	10	1,945	0.60	1.33	2.22
	Mineralized Zones - DDH	8067-8099	5m, ≥1m	8	5,007	0.76	4.50	5.95
Roma	Laterite - DDH	5	5m, ≥1m	3	160	0.32	0.62	1.94
	Mineralized Zones - DDH	8038-8066	5m, ≥1m	6	960	0.64	2.10	3.28
Royal Hill	Alluvials & Laterite - BH	4-5	5m, ≥1m	6	64,510	0.85	2.77	3.25
	Mineralized Zones - BH	8025-8096	5m, ≥1m	6	126,941	0.88	4.15	4.74
	Host Rock - BH	8106; 8203; 8301; 8303	5m, ≥1m	6	282,943	0.35	1.90	5.47
	Alluvials & Laterite - BH & DDH	4-5	3m, ≥0.6m	8	68,498	0.88	7.47	8.46
	Mineralized Zones NW Pit- BH & DDH	8070-8096	3m, ≥0.6m	12	83,516	0.93	4.37	4.72
	Mineralized Zones SE Pit - BH & DDH	8025-8069	3m, ≥0.6m	10	53,693	0.81	3.91	4.85
	Alluvials & Laterite - DDH	4-5	3m, ≥0.6m	8	3,988	1.38	28.87	20.91
	Mineralized Zones NW Pit - DDH	8070-8096	3m, ≥0.6m	12	5,564	0.99	2.84	2.86
	Mineralized Zones SE Pit- DDH	8025-8069	3m, ≥0.6m	10	4,694	0.87	6.41	6.40
Rosebel	Laterite - DDH	5	5m, ≥1m	8	939	0.63	1.38	2.20
	Mineralized Zones - DDH	8053-8075	5m, ≥1m	10	2,348	0.69	1.57	2.29
	Host Rock - DDH	8203	5m, ≥1m	1	3,859	0.05	0.19	3.83

Notes:
(1) DDH: Diamond Drill Holes; BH: Blastholes
(2) Assays >= 0.00 g/t Au are used to define the statistical parameters
(3) High Grade Limit applies for all composites categories
(4) Applies also to Rock Code series of 7000 and 9000

Table 17.2. Composite statistics – December 2009 block models.

17.4. Bulk Density Data

In-situ bulk density samples are taken for all 2009 diamond drill holes in each weathering zone (laterite, saprolite, transition and rock), and depending on the deposit, in specific lithology units. The density is calculated by the RGM laboratory using the wax method. Samples are dried in an oven for about 12 hours at 150°C. The sample is then cleaned and weighed on a balance. This represents the dry weight. To calculate the specific gravity, the sample is covered in paraffin wax and weighed again. Finally, the waxed sample is weighed in water. Given that the density of paraffin wax is 0.9, the calculation for bulk specific gravity is:

$$\text{Bulk Specific Gravity} = \frac{A}{[\,D - E - ((D - A)/F)\,]}$$

The bulk specific gravity values used in the reserve calculations are an average of all measurements taken for the given weathering zone and/or lithology (Table 17.3).

Deposit	Laterite	Saprolite			Transition			Hard Rock		
		Sediments	Volcanics	Dyke	Sediments	Volcanics	Dyke	Sediments	Volcanics	Dyke
Koolhoven/Bigi	1.73	1.73	1.73	-	2.30	2.30	-	2.70	2.70	-
J Zone	1.70	1.70	-	-	2.30	-	-	2.80	-	-
Pay Caro	1.80	1.80	1.80	-	2.30	2.30	-	2.70	2.70	-
East Pay Caro	1.80	1.90	1.60	-	2.30	2.20	-	2.70	2.80	-
Mayo	1.73	1.70	1.70	-	2.30	2.30	-	2.74	2.74	-
Roma	1.70	1.70	-	-	2.20	-	-	2.70	-	-
Royal Hill	1.71	1.70	1.70	-	2.35	2.35	-	2.78	2.78	-
Rosebel	1.69	1.85	-	1.90	2.30	-	2.40	2.70	-	2.85

Table 17.3. Rosebel average bulk density data.

17.5. Block Modeling

The block modeling estimation is done using the *GEMS version 6.1.4.2* software package. One block model is constructed for each project. After the completion of a drilling campaign, the block model is partially or completely updated.

In 2009, Royal Hill, Mayo, Koolhoven, Roma and J-Zone block models were updated (Table 17.4). Mayo, Royal Hill and Roma were updated twice. For Royal Hill, the two updates correspond to the complete geological reinterpretation of the NW and SE pits. No changes were made for Pay Caro, East Pay Caro and Rosebel as the assay results were pending, but updates are scheduled for 2010 once the additional drilling information has been received.

Block Model	Drilling and Gold Assaying			Geology and Data Updating		Block Modeling		
	Starting Date	Termination Date	Gold Analysis - Termination date	Ore Zone, Weathering and Lithology Modelization	Assays and Composites Validation	Block Model Settings	Interpolation	Validation and Approval
Roma - Update I	Drilling Phase II - 2008 6-Oct-08	10-Nov-08	11-Jan-09					January 2009 RESDec08Upd
Koolhoven - Update I	Drilling Phase I - 2009 8-Feb-09	9-Mar-09	25-Mar-09					May 2009 RESApr09Upd2
Royal Hill - Update I and II	Drilling Phase I & II - 2009 5-Feb-09	16-Apr-09	27-May-09					July 2009 RESJune09Upd September 2009 RESAug09Upd
Pay Caro - Update I	Drilling Phase I, II and III - 2009 23-Feb-09	17-Nov-09	30-Dec-09					
Mayo - Update I	Drilling Phase I - 2009 11-Mar-09	16-May-09	8-Jun-09					July 2009 RESJul09
Roma - Update II	Drilling Phase I - 2009 30-Apr-09	9-Jun-09	12-Jul-09					August 2009 BM0809RESUpd
J Zone - Update I	Drilling Phase I - 2009 7-Jun-09	21-Jul-09	25-Sep-09					October 2009 RESDec09ffi
Koolhoven - Update II	Drilling Phase II - 2009 11-Jul-09	16-Aug-09	21-Sep-09					October 2009 RESOct09Upd
Rosebel - Update I	Drilling Phase II - 2009 21-Jul-09	31-Oct-09	29-Nov-09					
Mayo - Update II	Drilling Phase II - 2009 11-Aug-09	26-Sep-09	30-Nov-09					December 2009 ResDec09Offi
Roma - Update III	Drilling Phase II- 2009 24-Sep-09	9-Oct-09	In progress					

Table 17.4. Block modeling performed in 2009.

17.6. Grade Estimation Methodology

Interpolations of grades in the block models are performed using the inverse distance cube method to the third power (ID^3) with anisotropic distances. The gold grade estimates are generated from five metre composites.

In all deposits, geological and mineralized contacts are considered as hard boundaries to avoid smearing gold grades from one mineralized zone to another or into waste. In line with this, a unique rock code is assigned to each block and composite when at least 50% is located inside the solid (ore zone, weathering and lithology solids). The resource estimates are done using a sample search approach. During the interpolation process, the rock code in each block is read from the rock code block model. Then the composites data set is scanned for composites that are associated to the same rock code and that are located within the limits of the search ellipse. The parameters used to estimate gold grade in the block models are listed in Table 17.5.

For Koolhoven, J-Zone, Pay Caro, East Pay Caro, Roma and Royal Hill SE, anisotropic search ellipses are used and set according to the orientation and dip of the mineralized zones. Spherical search ellipses are used for the grade interpolation in Mayo, Royal Hill NW and Rosebel.

The grade evaluations done with diamond drill hole (DDH) composites are performed in three different cumulative steps corresponding to three different level of confidence (Measured, Indicated and Inferred resources; see Section 17.8 for details). When blast hole (BH) assays are present in the project, a fourth step is added; two grade evaluations are performed separately for Measured resources with 1) BH composites and 2) with BH and DDH composites. These estimations overwrite a specific selection of blocks derived from the first three cumulative steps. In detail, grades calculated with a combination of composites from DDH and BH overwrite only blocks located inside the pit design down to 50 metres below the mined out surface; grades calculated with assays from BH overwrite only blocks located inside the pit design above the current mined out surface.

17.7. TREATMENT OF HIGH GRADE VALUES

Two limits defined from the composite statistics (see Section 17.3.1) are used to constrain high grade values during the interpolation process: 1) *High Grade Limit* and *2) High Grade Transition Limit.* The detailed limit parameters applied on each search ellipse and sphere are shown in Table 17.5.

1) When the search ellipse reaches a composite with a grade higher than the *High Grade Limit* , its value is reduced to the *High Grade Limit* and then incorporated in the resource calculation.

2) When a composite grade exceeds the *High Grade Transition Limit* during the interpolation, the search ellipse axial ranges are cut in half. This way, high grades are confined to a more restricted search volume.

17.8. RESOURCE CLASSIFICATION

The Mineral Resources estimations for all projects are classified according to the Canadian Institute of Mining, Metallurgy and Petroleum, "CIM", Definition Standards for Mineral Resources and Reserves (December 11, 2005). Detailed parameters used in the calculation of each resource category are presented in Table 17.5.

Measured

For measured resource classifications, the search ellipses have a circular radius of 50 metres corresponding to the average diamond drilling grid spacing and a small radius varying from 10 to 25 metres for ore zones in saprolite, transition and rock and 20 metres for laterite. The spherical ellipsoids have a radius of 50 metres. A minimum of five up to a maximum of twelve data points (five metres composites) from which no more than two composites originating from the same source of information (DDH or BH) are used to evaluate a grade in a block. Composites are thus taken from at least three different locations. The parameters defined above ensure that blocks classified as measured resources are estimated only where mineralization grade and continuity are highly reliable.

Indicated

The search ellipse's circular and small radii for laterite and ore zones in the indicated resource classification are respectively extended to 75 metres, 30 metres and 10-37.5 metres depending on the project area. In other projects using the spherical ellipsoids, the radius is lengthened to 75m. A minimum of three up to a maximum of twelve composites from which no more than two originating from the same source (DDH only) are used to evaluate a grade in a block. Composites are thus taken from at least two different locations. The parameters described above ensure that blocks classified as indicated resources are estimated only where mineralization grade and continuity are reasonably reliable.

The measured and indicated resources are used for pit optimization and pit design.

Inferred

The search ellipses in the inferred resources classification are extended to 150 metres in the azimuth and dip directions, to 30-75 metres for the small radius for ore zones and to 30-60 metres for laterite. The radius of the spherical ellipsoids is prolonged to 150 metres. A minimum of one up to a maximum of twelve composites are used to evaluate a grade in a block. The number of composites originating from an individual source of information (DDH only) is not limited.

Inferred resources are not used for reserve estimation or mine optimization, but they represent excellent targets to increase the reserve base of the Rosebel project.

	Category	Blocks			Composites						Rock Code			Search Ellipse Settings										
		Calculation Method	Blocks Updated	Block Selection in Interpolation	Source	Point Area Wrk and Source Name	Number used Min	Number used Max	Max per hole	High Grade Limit	Profile Name	Description	Target Rock Code	Profile Name	Rotation Z	Rotation X	Rotation Z	Range (m) X	Range (m) Y	Range (m) Z	High Grade Transition Limit	High Grade Range (m) X	High Grade Range (m) Y	High Grade Range (m) Z
KOOLHOVEN	Measured	ID^2 Anisotropic	All blocks	All Blocks	DDH 3 meters composites	RESPA1309 BM1309-3Msth-50	7	18	3	10	3MSMEALA	Laterite	5	EL-LA-PV	0	0	0	50	50	50	6	25	25	10
											3MSMEA11	Ore Zones Group 1	8297, 8299, 8300, 8302, 8303, 8305, 8306, 8308, 8310, 8311, 8314, 8320 to 8327, 8330 to 8332	1-90-PV8	0	0	0	50	25	50	6	25	20	25
													8298, 8301, 8304, 8312	1-90-PV4	0	0	0	50	25	50	4	25	20	25
													8317, 8318	1-90-PV8	0	10	0	50	25	50	6	25	20	25
													8328, 8329, 8333	1-85-PV8	0	5	0	50	25	50	6	25	20	25
											3MSMEA12	Ore ZonesGroup 2	8336, 8346, 8347, 8365 to 8367	1-90-PV8	0	0	0	50	25	50	6	25	20	25
													8334, 8335, 8337, 8338	1-85-PV8	0	5	0	50	25	50	6	25	20	25
													8340, 8341, 8343 to 8345, 8368, 8369	1-850PV8	0	-5	0	50	25	50	6	25	20	25
													8342	1-850PV4	0	-5	0	50	25	50	4	25	20	25
	Indicated	ID^2 Anisotropic	All blocks not evaluated from the measured interpolation / Zero grade blocks	All Blocks	DDH 3 meters composites	RESPA1309 BM1309-3Msth-50	4	18	3	10	3MSINDLA	Laterite	5	EL-LA-P8	0	0	0	75	75	30	6	37.5	37.5	15
											3MSIND11	Ore Zones Group 1	8297, 8299, 8300, 8302, 8303, 8305, 8306, 8308, 8310, 8311, 8314, 8320 to 8327, 8330 to 8332	1-90-P88	0	0	0	75	37.5	75	6	25	25	25
													8298, 8301, 8304, 8312	1-90-P84	0	5	0	75	37.5	75	6	37.5	30	37.5
													8317, 8318	1-90-P88	0	10	0	75	37.5	75	6	37.5	30	37.5
													8328, 8329, 8333	1-85-P88	0	0	0	75	37.5	75	4	37.5	30	37.5
											3MSIND12	Ore Zones Group 2	8336, 8346, 8347, 8365 to 8367	1-90-P88	0	0	0	75	37.5	75	6	25	25	25
													8334, 8335, 8337, 8338	1-85-P88	0	5	0	75	37.5	75	6	37.5	30	37.5
													8340, 8341, 8343 to 8345, 8368, 8369	1-850P88	0	-5	0	75	37.5	75	6	37.5	30	37.5
													8342	1-850P84	0	-5	0	75	37.5	75	4	37.5	30	37.5
	Inferred	ID^2 Anisotropic	All blocks not evaluated from the measured and indicated interpolations / Zero grade blocks	All Blocks	DDH 3 meters composites	RESPA1309 BM1309-3Msth-50	1	18	—	10	3MSINFLA	Laterite	5	EL-LA-P5	0	0	0	150	150	60	6	75	75	30
											3MSINF11	Ore Zones Group 1	8297, 8299, 8300, 8302, 8303, 8305, 8306, 8308, 8310, 8311, 8314, 8320 to 8327, 8330 to 8332	1-90-P58	0	0	0	150	75	150	6	75	60	75
													8298, 8301, 8304, 8312	1-90-P54	0	5	0	150	75	150	6	75	60	75
													8317, 8318	1-90-P58	0	10	0	150	75	150	6	75	60	75
													8328, 8329, 8333	1-85-P58	0	5	0	150	75	150	4	75	60	75
											3MSINF12	Ore Zones Group 2	8336, 8346, 8347, 8365 to 8367	1-90-P58	0	0	0	150	75	150	6	75	60	75
													8334, 8335, 8337, 8338	1-85-P58	0	5	0	150	75	150	6	75	60	75
													8340, 8341, 8343 to 8345, 8368, 8369	1-850P58	0	-5	0	150	75	150	6	75	60	75
													8342	1-850P54	0	-5	0	150	75	150	4	75	60	75
	Measured	ID^2 Anisotropic	Blocks down to 10 meters below mined out surface (Status 15Sep09) inside pit design (Official 31Jan09)	All Blocks	DDH 3 meters composites & Blast hole 5m assays	RESPA1309 BM1309-3Msth-BH	7	18	3	10	3MSMEA21	Ore Zones Group 1	8297, 8299, 8300, 8302, 8303, 8305, 8306, 8308, 8310, 8311, 8314, 8320 to 8327, 8330 to 8332	1-90-PV8	0	0	0	50	25	50	6	25	20	25
													8317, 8318	1-90-PV8	0	10	0	50	25	50	6	25	20	25
													8328, 8329, 8333	1-85-PV8	0	5	0	50	25	50	6	25	20	25
													8298, 8301, 8304, 8312	1-90-PV4	0	0	0	50	25	50	4	25	20	25
											3MSMEA22	Ore Zones Group 2	8336, 8346, 8347, 8365 to 8367	1-90-PV8	0	0	0	50	25	50	6	25	20	25
													8334, 8335, 8337, 8338	1-85-PV8	0	5	0	50	25	50	6	25	20	25
													8340, 8341, 8343 to 8345, 8349, 8368, 8369	1-850PV8	0	-5	0	50	25	50	6	25	20	25
													8342	1-850PV4	0	-5	0	50	25	50	4	25	20	25
	Measured	ID^2 Anisotropic	Blocks above mined out surface (Status 15Sep09) inside the pit design (Official 01Jan09)	All Blocks	Blast holes 5m assays	RESPA1309 BM1309-BHassays	2	12	1	10	3-MEA-1	Ore Zones Group 1	8297 to 8300, 8308, 8310 to 8314, 8317, 8318, 8320 to 8333	3_BH	0	0	0	15	6	15	6	5	2	5
											3-MEA-2	Ore Zones Group 2	8334 to 8338, 8340 to 8347, 8349, 8365 to 8369, 7203, 9203, 8303, 8109	3_BH	0	0	0	15	6	15	6	5	2	5
J ZONE	Measured	ID^2 Anisotropic	All blocks	All blocks	DDH 3 meters composites	RESPA BM1309-3mCompJZ2	6	12	2	7	3M-MEALA	Laterite	5	ELL-LAPV	No Rotation			50	50	50	5	25	25	25
											3M-MEA1	Ore Zone	8297 to 8301, 8303, 8304, 8312, 8365, 8600 to 8614	ELL-PV	No Rotation			50	25	50	5	25	20	25
	Indicated	ID^2 Anisotropic	All blocks not evaluated from the measured interpolation / Zero grade blocks	All blocks	DDH 3 meters composites	RESPA BM1309-3mCompJZ2	3	12	2	7	3M-INDLA	Laterite	5	ELL-LAPB	No Rotation			75	75	75	5	37.5	37.5	37.5
											3M-IND1	Ore Zone	8297 to 8301, 8303, 8304, 8312, 8365, 8600 to 8614	ELL-PB	No Rotation			75	37.5	75	5	37.5	30	37.5
	Inferred	ID^2 Anisotropic	All blocks not evaluated from the measured and indicated interpolations / Zero grade blocks	All blocks	DDH 3 meters composites	RESPA BM1309-3mCompJZ2	1	12	-	7	3M-INFLA	Laterite	5	ELL-LAPS	No Rotation			150	150	150	5	75	75	75
											3M-INF1	Ore Zone	8297 to 8301, 8303, 8304, 8312, 8365, 8600 to 8614	ELL-PS	No Rotation			150	75	150	5	75	60	75

		Blocks			Composites						Rock Code			Search Ellipse Settings										
	Category	Calculation Method	Blocks Updated	Block Selection in Interpolation	Source	Point Area Wrk and Source Name	Number used		Max per hole	High Grade Limit	Profile Name	Description	Target Rock Code	Profile Name	Rotation			Range (m)			High Grade Transition Limit	High Grade Range (m)		
							Min	Max							Z	X	Z	X	Y	Z		X	Y	Z
PAY CARD	Measured	ID² Anisotropic	All blocks	All Blocks	DDH 3 meters composites	RESPAO408 DDH BM0408 CompIM	6	12	3	20	M_LAT	Laterite	5	LAT-PV	0	0	0	50	50	20	20	25	25	10
											ME-OZ-1	PC Ore Zones	8014, 8017 to 8042	Various	Various			50	50	10	10	25	25	5
											ME-OZ-2	PC Ore Zones	8040 to 8049	Various	Various			50	50	10	10	25	25	5
											ME-OZ-3	WPC Ore Zones	8013, 8015, 8016, 8051 to 8059	Various	Various			50	50	10	10	25	25	5
	Indicated	ID² Anisotropic	All blocks not evaluated from the measured interpolation / Zero grade blocks	All Blocks	DDH 3 meters composites	RESPAO408 DDH BM0408 CompIM	3	12	3	20	IND-LAT	Laterite	5	LAT-PB	0	0	0	75	75	30	20	37.5	37.5	15
											ID-OZ-1	PC Ore Zones	8014, 8017 to 8042	Various	Various			75	75	30		37.5	37.5	15
											ID-OZ-2	PC Ore Zones	8040 to 8049	Various	Various			75	75	30		37.5	37.5	15
											ID-OZ-3	WPC Ore Zones	8013, 8015, 8016, 8051 to 8059	Various	Various			75	75	30		37.5	37.5	15
	Inferred	ID² Anisotropic	All blocks not evaluated from the measured and indicated interpolations / Zero grade blocks	All Blocks	DDH 3 meters composites	RESPAO408 DDH BM0408 CompIM	1	12	-	20	INF-LAT	Laterite	5	LAT-PS	0	0	0	100	100	50	20	50	50	25
											IF-OZ-1	PC Ore Zones	8014, 8017 to 8042	Various	Various			150	150	30		75	75	15
											IF-OZ-2	PC Ore Zones	8040 to 8049	Various	Various			150	150	30		75	75	15
											IF-OZ-3	WPC Ore Zones	8013, 8015, 8016, 8051 to 8059	Various	Various			150	150	30		75	75	15
	Measured	ID² Anisotropic	Blocks down to 50 meters below mined out surface inside pit design	All Blocks	DDH & BH 5 meters composites	RESPAO408 BH DDH BM0408 CompIM	5	12	3	20	MD-OZ-1	PC Ore Zones	8014, 8017 to 8042	Various	Various			50	50	10	10	25	25	5
											MD-OZ-2	PC Ore Zones	8040 to 8049	Various	Various			50	50	10		25	25	5
											MD-OZ-3	WPC Ore Zones	8013, 8015, 8016, 8051 to 8059	Various	Various			50	50	10		25	25	5
	Measured	ID² Anisotropic	Blocks above mined out surface inside the pit design	All Blocks	BH 5 meters composites	RESPAO408 BH BM0408 CompIM	2	12	1	10	MEAS-BH1	PC and WPC Ore Zones	8013 to 8042	BH_IN	0	0	0	10	5	2.5	5	5	5	2.5
											MEAS-BH2	PC and WPC Ore Zones	8040 to 8049, 8051 to 8059											
											MEAS-BH3	Host Rock	7203, 8203, 8202, 8204, 8106, 8503											
EAST PAY CARD	Measured	ID² Anisotropic	All blocks	All Blocks	DDH 3 meters composites	RESPAO507 DDH only	5	12	2	10	BPDDLAPV	Laterite	5	JLAT-PV	0	0	0	50	50	20	6	25	25	10
										12	BPDDORPV	Ore Zone	8059 to 8060, 8082 to 8087, 8069 to 8079, 8083 to 8089	JVA-1PV	Various			50	50	10	6	25	25	5
											BPDDORPV	Ore Zone	8061, 8068, 8080, 8081	JVA-2PV	0	0	0	50	50	50	6	25	25	25
	Indicated	ID² Anisotropic	All blocks not evaluated from the measured interpolation / Zero grade blocks	All Blocks	DDH 3 meters composites	RESPAO507 DDH only	3	12	2	10	BPDDLAPB	Laterite	5	JLAT-PB	0	0	0	75	75	30	6	37.5	37.5	15
										12	BPDDORPB	Ore Zone	8059 to 8060, 8082 to 8087, 8069 to 8079, 8083 to 8089	JVA-1PB	Various			75	75	20	6	37.5	37.5	10
											BPDDORPB	Ore Zone	8061, 8068, 8080, 8081	JVA-2PB	0	0	0	75	75	75	6	37.5	37.5	37.5
	Inferred	ID² Anisotropic	All blocks not evaluated from the measured and indicated interpolations / Zero grade blocks	All Blocks	DDH 3 meters composites	RESPAO507 DDH only	7	12	-	10	BPDDLAPS	Laterite	5	JLAT-PS	0	0	0	150	150	30	6	60	60	15
										12	BPDDORPS	Ore Zone	8059 to 8060, 8082 to 8087, 8069 to 8079, 8083 to 8089	JVA-1PS	Various			150	150	20	6	75	75	10
											BPDDORPS	Ore Zone	8061, 8068, 8080, 8081	JVA-2PS	0	0	0	150	150	150	6	75	75	75
	Measured	ID² Anisotropic	Blocks down to 50 meters below mined out surface inside pit design	All Blocks	DDH & BH 5 meters composites	RESPAO507 BH+DDH	5	12	2	11	BPBDLAPV	Laterite	5	JLAT-PV	0	0	0	50	50	20	6	25	25	10
											BPBDORPV	Ore Zone	8059 to 8060, 8082 to 8087, 8069 to 8079, 8083 to 8089	JVA-1PV	Various			50	50	10	6	25	25	5
											BPBDORPV	Ore Zone	8061, 8068, 8080, 8081	JVA-2PV	0	0	0	50	50	50	6	25	25	25
	Indicated	ID² Anisotropic	Blocks down to 50 meters below mined out surface inside pit design	All Blocks	DDH & BH 5 meters composites	RESPAO507 BH+DDH	3	12	2	11	BPBDLAPB	Laterite	5	JLAT-PB	0	0	0	75	75	30	6	37.5	37.5	15
										10	BPBDORPB	Ore Zone	8059 to 8060, 8082 to 8087, 8069 to 8079, 8083 to 8089	JVA-1PB	Various			75	75	20	6	37.5	37.5	10
											BPBDORPB	Ore Zone	8061, 8068, 8080, 8081	JVA-2PB	0	0	0	75	75	75	6	37.5	37.5	37.5
	Inferred	ID² Anisotropic	Blocks down to 50 meters below mined out surface inside pit design	All Blocks	DDH & BH 5 meters composites	RESPAO507 BH+DDH	7	12	-	10	BPBDORPS	Ore Zone	8059 to 8060, 8082 to 8087, 8069 to 8079, 8083 to 8089	JVA-1PS	Various			150	150	20	6	75	75	10
											BPBDORPS	Ore Zone	8061, 8068, 8080, 8081	JVA-2PS	0	0	0	150	150	150	6	75	75	75
MAYO	Measured	ID² Anisotropic	All blocks	All Blocks	DDH 3 meters composites	RESPA BM1109-5mCompMM	6	15	2	10	H-M-5	Laterite	5	LAT-PV5	0	0	0	50	50	20	5	25	25	10
										8	H-M	Ore Zones	8067 to 8074, 8077 to 8096, 8099	SPH-PV5	0	0	0	50	50	50	5	25	25	25
	Indicated	ID² Anisotropic	All blocks not evaluated from the measured interpolation / Zero grade blocks	All Blocks	DDH 3 meters composites	RESPA BM1109-5mCompMM	3	12	2	10	H-Mt-5	Laterite	5	LAT-PB5	0	0	0	75	75	30	5	37.5	37.5	15
										8	H-Mt	Ore Zones	8067 to 8074, 8077 to 8096, 8099	SPH-PB5	0	0	0	75	75	75	5	37.5	37.5	37.5
	Inferred	ID² Anisotropic	All blocks not evaluated from the measured and indicated interpolations / Zero grade blocks	All Blocks	DDH 3 meters composites	RESPA BM1109-5mCompMM	1	12	—	10	H-Mf-5	Laterite	5	LAT-PS5	0	0	0	150	150	50	5	75	75	30
										8	H-Mf	Ore Zones	8067 to 8074, 8077 to 8096, 8099	SPH-PS5	0	0	0	150	150	150	5	75	75	75

		Blocks			Composites						Rock Code			Search Ellipse Settings										
	Category	Calculation Method	Blocks Updated	Block Selection in Interpolation	Source	Point Area Wrk and Source Name	Number used Min	Number used Max	Max per hole	High Grade Limit	Profile Name	Description	Target Rock Code	Profile Name	Rotation Z	Rotation X	Rotation Z	Range (m) X	Range (m) Y	Range (m) Z	High Grade Transition Limit	High Grade Range (m) X	High Grade Range (m) Y	High Grade Range (m) Z
ROMA	Measured	ID3 Anisotropic	All blocks	All blocks	DDH 3 meters composites	RESPA0809 RM Composi	8	12	2	3	RMEA_LA	Laterite	5	LA-MEA	0	0	0	50	50	50	2	25	25	25
										6	RMEA_OZ1	Ore Zones East	8030, 8035, 8040, 8045	ME_E_SPH	0	0	0	50	50	50	3	25	25	25
											RMEA_OZ1	Ore Zones Central	8053, 8054, 8057-8060, 8064	ME_C_ELL	0	75	0	50	25	50	3	25	25	25
											RMEA_OZ1	Ore Zones West	8041, 8042, 8044, 8045, 8047, 8049-8052, 8061, 8065	ME_W_ELL	10	35	0	50	25	50	4	25	25	25
											RMEA_OZ1	Ore Zones West	8048	M_W_8048	10	35	0	50	25	50	4	15	15	15
	Indicated	ID3 Anisotropic	All blocks not evaluated from the measured interpolation / Zero grade blocks	All blocks	DDH 3 meters composites	RESPA0809 RM Composi	8	12	2	3	RIND_LA	Laterite	5	LA-IND	0	0	0	75	75	75	2	37.5	37.5	37.5
										6	RIND_OZ1	Ore Zones East	8030, 8035, 8040, 8045	ID_E_SPH	0	0	0	75	75	75	3	37.5	37.5	37.5
											RIND_OZ1	Ore Zones Central	8053, 8054, 8057-8060, 8064	ID_C_ELL	0	75	0	75	37.5	75	3	37.5	30	37.5
											RIND_OZ1	Ore Zones West	8041, 8042, 8044, 8045, 8047, 8049-8052, 8061, 8065	ID_W_ELL	10	35	0	75	37.5	75	4	37.5	30	37.5
											RIND_OZ1	Ore Zones West	8048	ID_W8048	10	35	0	75	37.5	75	4	22.5	15	22.5
	Inferred	ID3 Anisotropic	All blocks not evaluated from the measured and indicated interpolations / Zero grade blocks	All Blocks	Zero Grade Blocks in selected rock codes	RESPA0809 RM Composi	1	12	—	3	RINF_LA	Laterite	5	LA-INF	0	0	0	150	150	150	2	75	75	75
										6	RINF_OZ1	Ore Zones East	8030, 8035, 8040, 8045	IF_E_SPH	0	0	0	150	150	150	3	75	75	75
											RINF_OZ1	Ore Zones Central	8053, 8054, 8057-8060, 8064	IF_C_ELL	0	75	0	150	75	150	3	75	37.5	75
											RINF_OZ1	Ore Zones West	8041, 8042, 8044, 8045, 8047, 8049-8052, 8061, 8065	IF_W_ELL	10	35	0	150	75	150	4	75	37.5	75
											RINF_OZ1	Ore Zones West	8048	IF_W8048	10	35	0	150	75	150	4	45	30	45
ROYAL HILL	Measured	ID2 - Anisotropic	All Blocks	All Blocks	DDH 3 meters composites	RESPA0809 Comp 3m DDH	7	14	3	5	1-LA-PV3	Laterite	5	LAT-PV	0	0	0	60	60	30	6	25	25	10
										12	1-NW-PV3	Ore Zones NW Pit	8070 to 8090	SPHNW-PV	0	0	0	60	60	60	6	33	33	33
										16	1-S1-PV3	Ore Zones SE Pit	8025 to 8039	ELLSE-PV	0	-75	0	60	60	25	6	33	33	17
											1-S2-PV3	Ore Zones SE Pit	8041 to 8050	ELLSE-PV	0	-75	0	60	60	25	6	33	33	17
											1-S3-PV3	Ore Zones SE Pit	8058 to 8065	ELLSE-PV	0	-75	0	60	60	25	6	33	33	17
	Indicated	ID2 - Anisotropic	All blocks not evaluated from the measured interpolation / Zero grade blocks	All Blocks	DDH 3 meters composites	RESPA0809 Comp 3m DDH	4	16	3	5	1-LA-PB3	Laterite	5	LAT-PB	0	0	0	75	75	30	6	38	38	16
										12	1-NW-PB3	Ore Zones NW Pit	8070 to 8090	SPHNW-PB	0	0	0	75	75	75	6	50	50	50
										16	1-S1-PB3	Ore Zones SE Pit	8025 to 8039	ELLSE-PB	0	-75	0	75	75	37.5	6	50	50	25
											1-S2-PB3	Ore Zones SE Pit	8041 to 8050	ELLSE-PB	0	-75	0	75	75	37.5	6	50	50	25
											1-S3-PB3	Ore Zones SE Pit	8058 to 8065	ELLSE-PB	0	-75	0	75	75	37.5	6	50	50	25
	Inferred	ID2 - Anisotropic	All blocks not evaluated from the measured and indicated interpolations / Zero grade blocks	All Blocks	DDH 3 meters composites	RESPA0809 Comp 3m DDH	1	16	—	5	1-LA-PS3	Laterite	5	LAT-PS	0	0	0	150	150	60	6	75	75	30
										12	1-NW-PS3	Ore Zones NW Pit	8070 to 8090	SPHNW-PS	0	0	0	150	150	150	6	100	100	100
										16	1-S1-PS3	Ore Zones SE Pit	8025 to 8039	ELLSE-PS	0	-75	0	150	150	75	6	100	100	50
											1-S2-PS3	Ore Zones SE Pit	8041 to 8050	ELLSE-PS	0	-75	0	150	150	75	6	100	100	50
											1-S3-PS3	Ore Zones SE Pit	8058 to 8065	ELLSE-PS	0	-75	0	150	150	75	6	100	100	50
	Measured	ID2 - Anisotropic	Blocks under 15 meters below the 01Aug09 mined out surface and within the 01Jul09 Official Pit Design $700	All Blocks	DDH 3 meters composites & BH 5 meters assays	RESPA0809 Comp 3m DDH BH	7	16	3	16	2-OZ-1	Ore Zones SE Pit	8025 to 8039	Vario_2	0	-75	0	50	50	25	6	33	33	17
											2-OZ-2	Ore Zones SE Pit	8058 to 8065	Vario_2	0	-75	0	50	50	25	6	33	33	17
										12	2-OZ-3	Ore Zones NW Pit	8070 to 8090	Vario_3	0	0	0	50	50	50	6	33	33	33
										5	2-OZ-4	Laterite	5	LAT-PV	0	0	0	50	50	30	6	25	25	10
	Measured	ID2 - Anisotropic	Blocks above the 01Aug09 mined out surface and within the 01Jul09 Official Pit Design $700	All Blocks	BH 5 meters assays	RESPA0809 Comp BH	2	12	—	6	3-BHIN-1	Laterite & Ore Zones NW Pit	5, 8070 to 8090	BH-IN	0	0	0	5	5	2.5	6	5	5	2.5
											3-BHIN-2	Ore Zones SE Pit	8058 to 8069	BH-IN	0	0	0	10	5	2.5	6	5	5	2.5
											3-BHIN-3	Ore Zones SE Pit	8070 to 8090	BH-IN	0	0	0	10	5	2.5	6	5	5	2.5
											3-BHIN-4	Host Rock	T203, 8005, 8303, 8106, 8301	BH-IN	0	0	0	10	5	2.5	6	5	5	2.5
ROSEBEL	Measured	ID2 Anisotropic	All blocks	All Blocks	DDH 3 meters composites	RESPA0809 Comp3m_DDH	8	12	2	5	LAT-MEA	Laterite	5	LAT-PV	0	0	0	50	50	20	3	25	25	10
										10	O-SP-MEA	Ore Zones	8000 to 8070, 8075	50SPH-PV	0	0	0	50	50	50	3	25	25	25
	Indicated	ID2 Anisotropic	All blocks not evaluated from the measured interpolation / Zero grade blocks	All Blocks	DDH 3 meters composites	RESPA0809 Comp3m_DDH	3	12	2	5	LAT-IND	Laterite	5	LAT-PB	0	0	0	75	75	30	3	37.5	37.5	15
										10	O-SP-IND	Ore Zones	8000 to 8070, 8075	75SPH-PB	0	0	0	75	75	75	3	37.5	37.5	37.5
										1	HOST-IND	Host Rock	T203, 5005, 8203	HSTN-PB	0	-90	0	60	90	5	0.4	25	25	5
	Inferred	ID2 Anisotropic	All blocks not evaluated from the measured and indicated interpolations / Zero grade blocks	All Blocks	DDH 3 meters composites	RESPA0809 Comp3m_DDH	1	12	—	5	LAT-INF	Laterite	5	LAT-PB	0	0	0	150	150	60	3	75	75	30
										10	O-SP-INF	Ore Zones	8000 to 8070, 8075	150SPH-PS	0	0	0	150	150	150	3	150	150	150

Table 17.5. Detailed Block Model Parameters

17.9. Economic Parameters

17.9.1. Optimization Parameters

Pit optimization is performed by the Lerchs-Grossman algorithm using *Whittle Analyzer* software. This technique generates a series of optimal pits given the geological block models, operating costs, recoveries, geo-technical constraints and gold prices. The optimal pit chosen is normally one which maximizes the undiscounted or discounted cash flow at the given economic parameters, but pit shells can also be produced for a range of specific gold prices, if so required. The pit design process is iterative. After the theoretical optimal pit is obtained, additional mining constraints such as minimum mining widths and practical mining access ramps are included. The process is repeated until a stable design is obtained.

The economic parameters used for soft rock, transition and hard rock material in the optimization process are shown in Table 17.6. The mining and ore based costs, including electricity, used for the optimization are based on direct observations at Rosebel since January 2004 and are updated frequently. An incremental ore haulage cost was included in the optimization process because of the different ore haulage distance for each pit. This results in slightly different economic cut-off grades for each pit (Table 17.6).

A portion of the Capital Expenditures was added to the milling (ore based) costs. For 2009 this amount was US$76.4 million and was factored using 6% discount rate, to $64.1 million. Based on LOM plan of 92.55 million tonnes, this would contribute $0.69 per tonne to the ore based costs.

Lerchs-Grossman Optimization Parameters December 2009				
Description Mining Costs	Laterite $/t	Saprolite $/t	Transition $/t	Rock $/t
Drilling	0.06	0.06	0.10	0.12
Blasting	0.03	0.03	0.15	0.26
Loading	0.09	0.09	0.09	0.09
Hauling	0.30	0.30	0.30	0.32
Maintenance, Major Support	0.43	0.43	0.43	0.43
Minor Support & Admin.	0.37	0.37	0.37	0.37
Total Ore Based Costs (Pay Caro)	**1.274**	**1.274**	**1.426**	**1.574**
Description Other Costs	**Laterite $/t**	**Saprolite $/t**	**Transitio $/t**	**Rock $/t**
Milling	2.75	2.75	3.09	3.53
Electricity	1.49	1.49	2.18	2.91
General Services	2.10	2.10	2.10	2.10
$64.13 mill CAPEX	0.69	0.69	0.69	0.69
Total	**7.03**	**7.03**	**8.06**	**9.23**
Mill Recovery	92.0% - 96.0%			
Suriname Royalty	2.25% up to $425, then +6.5%			
Au Price	850$ /oz			
Transportation & refining	2.50$ /oz			

Incremental Ore Haulage Cost and Cut Off Grade					
Deposit	Additional Haulage Distance Km	Incremental Ore Haul Cost $/t	Cut Off Grade Saprolite Au g/t	Cut Off Grade Transition Au g/t	Cut Off Grade Rock Au g/t
Pay Caro	-	0.00	0.28	0.33	0.39
East Pay Caro	2.50	0.10	0.29	0.34	0.39
Koolhoven	3.00	0.33	0.30	0.35	0.40
Royal Hill	5.35	0.45	0.30	0.35	0.41
Mayo	9.80	0.69	0.31	0.36	0.42
Rosebel	15.0	0.92	0.32	0.37	0.43
Roma	7.7	0.45	0.30	0.35	0.41
J - Zone	3.50	0.33	0.30	0.35	0.40

Table 17.6. Economic parameters used in the optimization process.

The economic modeling parameters are based on the operating costs determined since the start of operations at Rosebel. An adjustment to mining cost was included to represent increased haulage cost with increasing depth.

A royalty payment of 2.25% of the gold produced is payable to the Surinamese government. This royalty is valid up to 425$/oz, above this, an additional royalty of 6.5% is applied. For example, for a gold price of 600$, the total royalty payable is $24.88 per ounce, comprised of 2.25% x 600$ or $13.5 per ounce and 6.5% x (600$ - 425$) or $11.38 per ounce.

Metallurgical recovery of gold is based on five years of ore processing as follows: Soft Rock = 96.0%; Transition Rock = 94.0%; Hard Rock = 92.0%.

This reserve estimate is based on a gold price of US$850 per troy ounce.

This reserve estimate is also based on an exchange rate of US$1.00 = 2.795 Suriname Dollars.

17.9.2. Ore Recovery

Mining recovery is accounted for in the block model. No additional factors are applied for mining recovery. Experience and observations over the duration of operations at Rosebel suggest that there is no reason to challenge this assumption so far. Indeed, more ore has been mined from Royal Hill, Pay Caro, East Pay Caro, Koolhoven and Mayo than would be expected from the reserve models.

17.9.3. Estimation Results

There was an increase in the December 31 st 2009 reserve ounces in all of the RGM deposits, due mainly to additional exploration drilling, the use of an $850 USD/oz price for gold in the pit optimization process and improved geological models.

Pay Caro (Main Pit)

The Pay Caro and East Pay Caro deposits are located adjacent to the mill site. RGM has mined continually at Pay Caro since October 2003. During 2006, the internal Phase 1 was completed and a larger Phase1A pushback was started. This second pushback was depleted at the end of the 2009 and the stripping of the final (Ultimate Phase 3) has begun. Minor modifications to the pit design were made during 2009, focusing mostly on pit wall stability issues as well as adjusting for some pit operational issues such as dewatering.

The current Pay Caro pit design is known as the Phase 3 push back and it encircles the mined out Phase 1 A. During 2009, the design of Phase 3 has been refined, primarily to reduce the strip ratio, but in some areas, more conservative flatter walls have been introduced to address operational and geotechnical issues. Pay Caro Phase 3 is one of the largest and deepest pits at Rosebel and was mined at a high strip ratio in the early years because the majority of the ore occurs at depth. For this reason it is being delayed in the current Life of Mine Schedule (“LOM”), and other pits such as Mayo and Rosebel have been moved forward. There is no mining activity planned for Pay Caro between 2010 and 2012 in the latest version of the LOM.

Immediately west of Pay Caro is an area known as West Pay Caro, and reserves for West Pay Caro are included with Pay Caro Phase 3 (Ultimate Pit) reserves. The West Pay Caro Pit will be mined at the end of the current LOM due to its slightly higher stripping ratio relative to the other pits.

In the most recent pit design, the access to the bottom of the Phase 3 (Ultimate) pit is provided by two 25m wide ramps that narrow to 15m near the very bottom. One ramp exits in the south-centre of the pit and provides access to the ore haulage to mill and the south dumps (Fig. 17.1). The second ramp exits the pit to the north and will be primarily used for waste haulage to the North Waste Rock Storage.

Further revisions to the pit design are expected to follow from updates to the geological model and improvements in the logistics of ore and waste haulage and slope design criteria. It is important to maintain a corridor between the Pay Caro and West Pay Caro pits to allow for the flow of the Grotte Louis Creek and the proposed haulage road access to the future J-Zone Pit. The smaller West Pay Caro area is planned to be mined independently of the main Phase 3 pit.

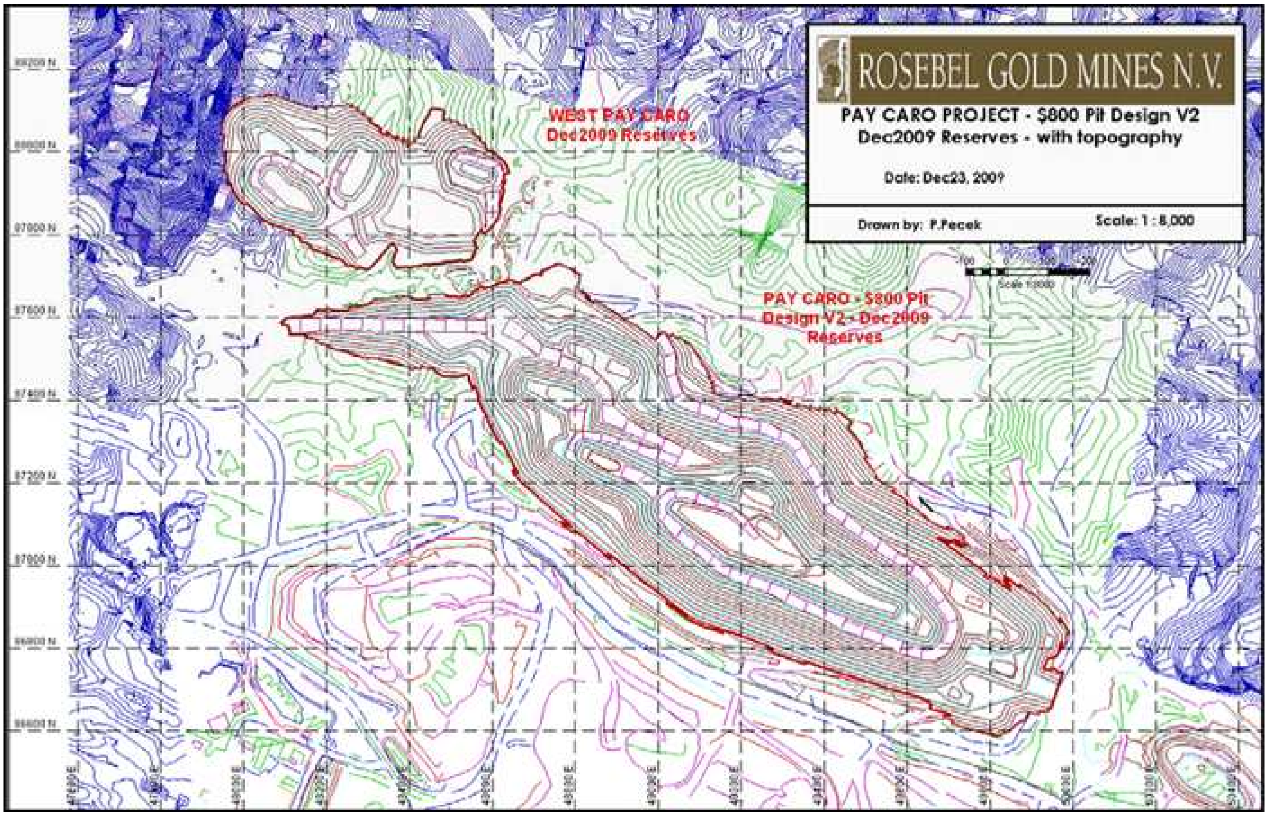


Figure 17.1. Pay Caro pit design (Main Pit and West Pay Caro).

East Pay Caro

The East Pay Caro ("EPC") pit is much smaller than Pay Caro and was the second mining area to be developed by RGM in 2004. A new pit was designed following the $800/oz gold price optimization and is shown in Figure 17.2. The pit design parameters are similar to the Pay Caro pit. The EPC pit design includes a very conservative two to one slope above the 500m bench. This may be subject to modifications in the future, especially in the east end of the pit. A smaller internal Phase 2 (not shown) will be mined in 2010 and 2011.

The pit base is planned to be at 325m elevation. Access is via a 25m wide ramp which leads from the pit rim to the pit bottom. The ramp starts along the south wall and continues in a counter-clockwise direction. The ramp narrows and the gradient increases near the bottom. A small waste rock storage dump was designed to the north of EPC, mainly to prevent the collection of water in a valley directly above the pit. This waste dump could be increased to include waste removed from the J-Zone Pit located further to the north. An additional ramp to mine more waste rock from EPC may be included in the design of the north wall.

Koolhoven

The Koolhoven deposit is located northwest of the Pay Caro deposits. The topography is more pronounced than at Pay Caro, ranging from 525 to 650m elevations. Mining began at Koolhoven in the fourth quarter of 2007.

Most of the ore at Koolhoven is close to surface and primarily occurs in soft and transition rocks. The pit design consists of four main pits, as well as several smaller pits to exploit near-surface ore zones (Fig. 17.3). The pits are relatively shallow with pit bottom elevations at 410m, 420m, 440m and 455m. Access to the eastern pit is from the 510m elevation and the pit will be developed to the 450m elevation. The western pit also has access at an elevation of 510m, with a pit bottom at the 445m elevation. The central pit has a pit bottom at the 420m elevation. Because the pits are situated on hillsides, the maximum wall height is 190 metres. The ramp access takes advantage of the topography by entering the pits at low points. There is very little hard rock in Koolhoven. The slope in hard rock is designed at 50 degrees inter-ramp.

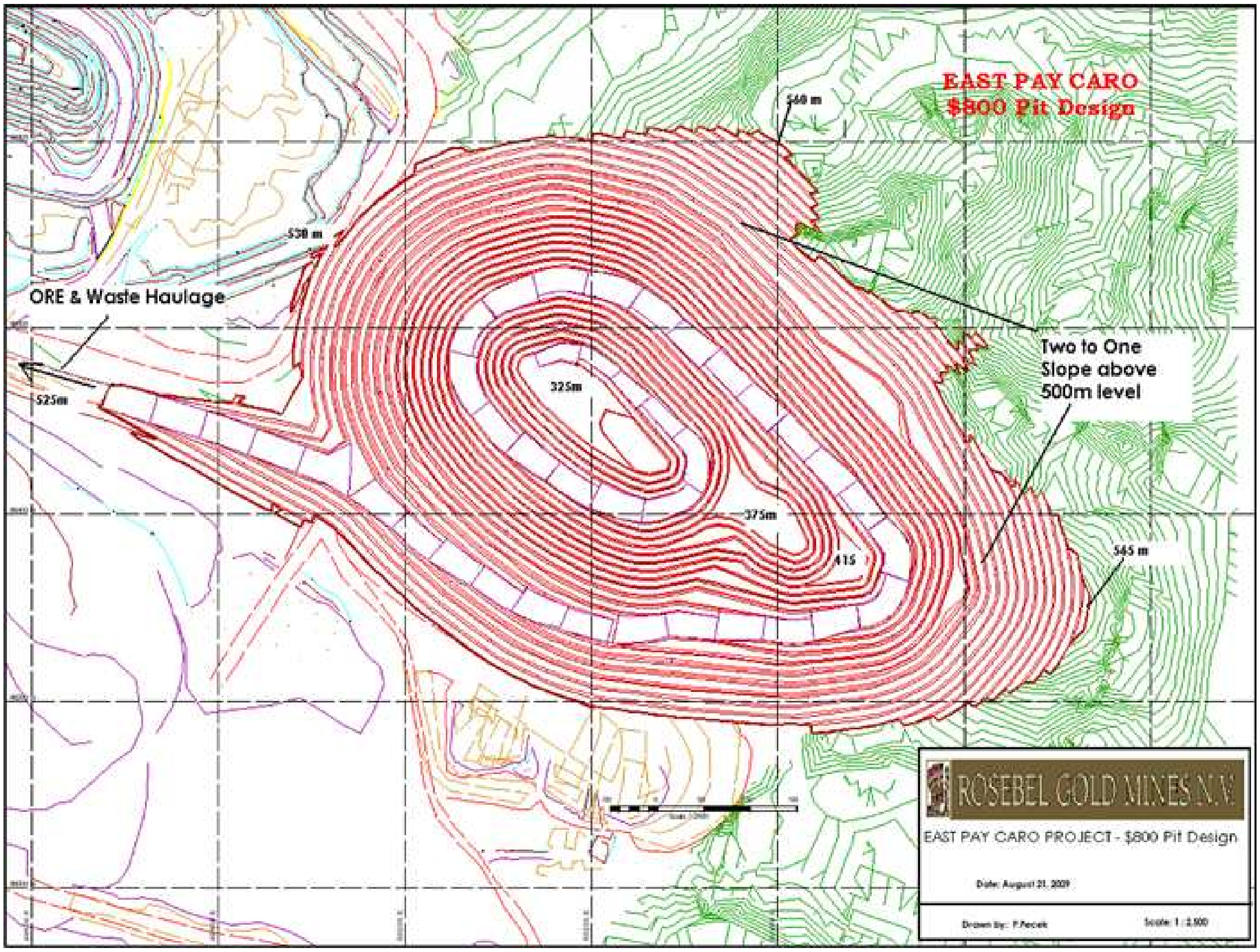


Figure 17.2. East Pay Caro pit design.

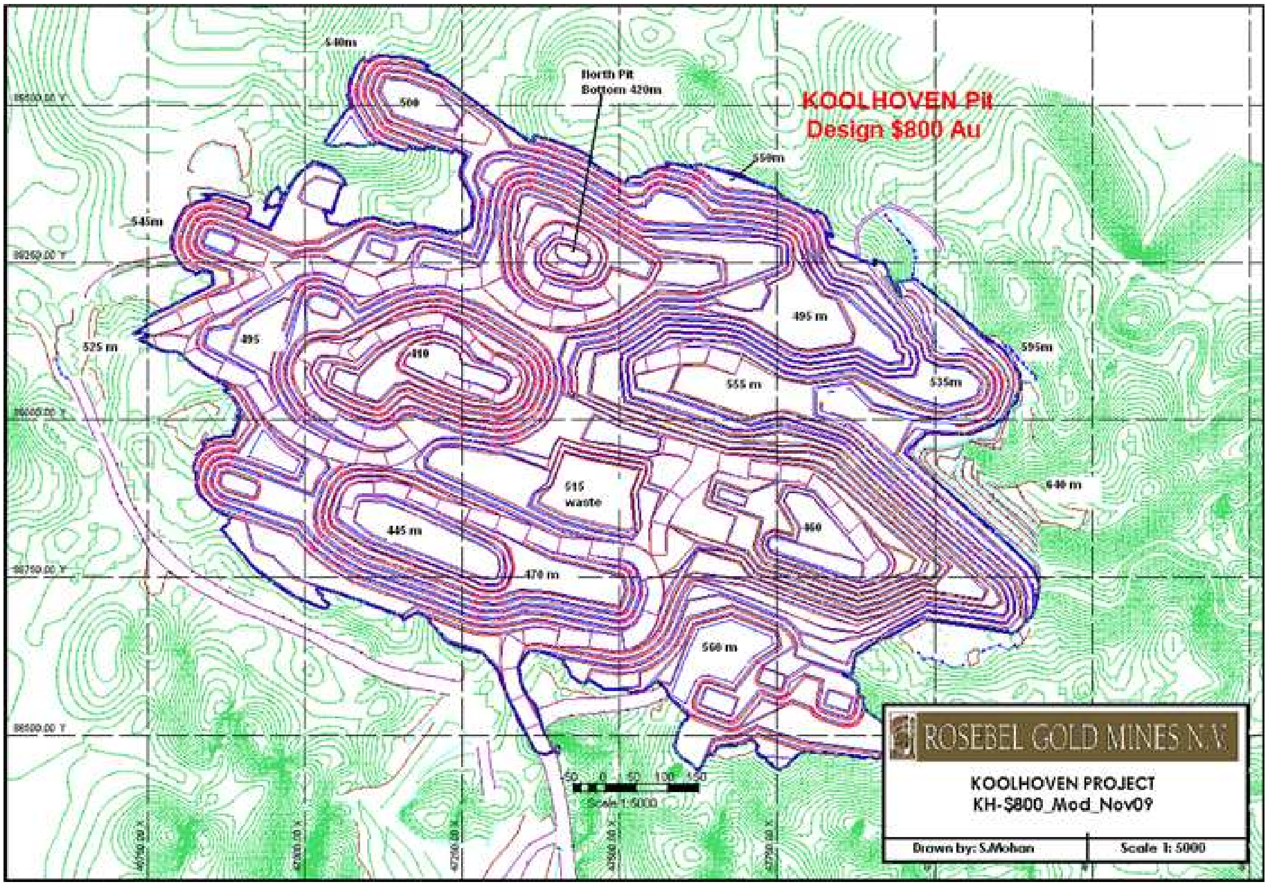


Figure 17.3. Koolhoven pit design.

Royal Hill

The Royal Hill deposit is located on the southern mineralized trend. Topography locally ranges between 520 and 570m. Two pits have been designed for Royal Hill, the NW (northwest) and SE (southeast) pits. RGM began mining the NW pit in August 2004 and had progressed to the 460m elevation at the end of 2009. Mining began in the SE pit during the second quarter of 2005 and had reached the 505m bench by the end of December 2009.

For the optimization exercise, wall slopes of 26 degrees were specified for soft rock, allowing a two to one slope in the pit design where so required. A 40 degrees inter-ramp slope angle was used for the relatively shallow transition rock zones. For hard rock, 45 degrees was specified in the optimization to allow for a spiral ramp in the NW pit with 50 degree inter-ramp slopes. Information from additional in-fill drilling during 2009, as well as improved financial criteria and a more conservative approach to some geotechnical issues justified the re-design of the two Royal Hill pits (Fig. 17.4).

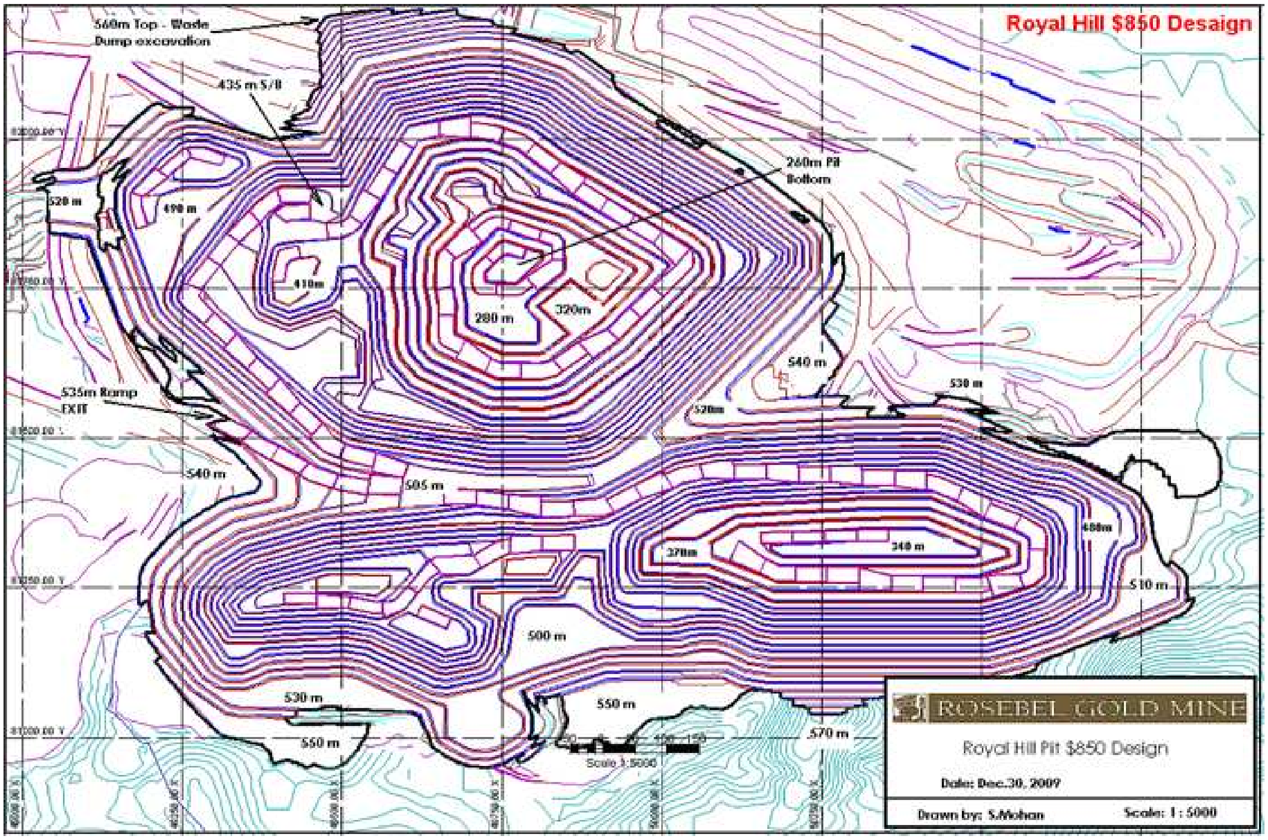


Figure 17.4 . Royal Hill pit design.

Mayo

The Mayo deposit is located on the southern mineralized trend. Elevation ranges from a low of 510m in the north to a high of 550m directly over the deposit. The material in the pits is approximately 50% soft rock and 50% hard rock at the lower benches. The current pit design is based on an optimized shell at $800/oz gold price with the Pit Reserves being reported at the $850/oz gold price grade cut-off. The new pit design is shown in Figure 17.5.

More slots have been included in the new design, replacing the circular ramps of the 2008 pit design. The main pit bottom is at 320m and the pit crest is at an elevation of between 530 and 540m. Two small satellite pits are planned to be mined to the north of the main pit. There is substantial increase of hard rock in the larger and deeper pit compared to 2008. The increase in ounces comes predominantly from the hard rock. The main waste rock storage area will be located to the north of the pit. The geometry of the pit will also allow for a substantial amount of in-pit, back-filling resulting in substantially shorter waste cycles.

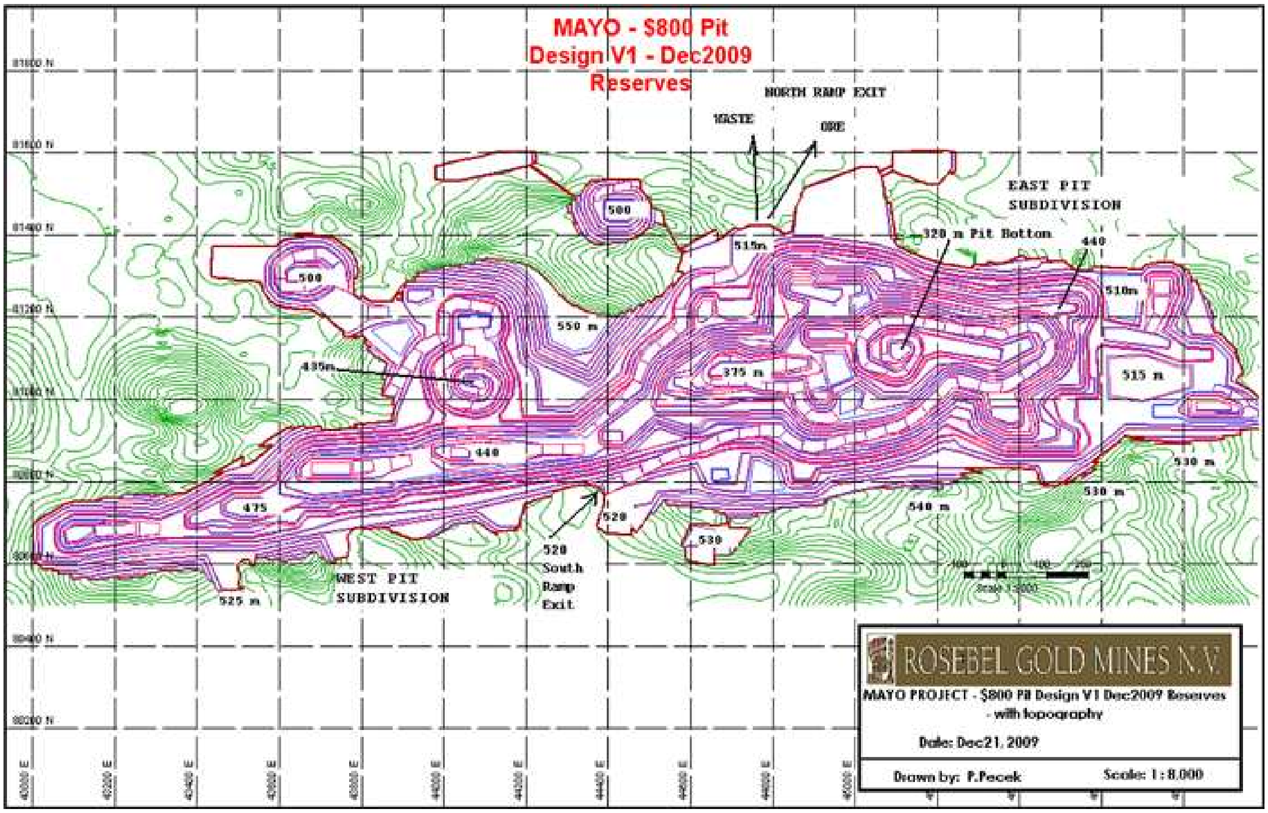


Figure 17.5. Mayo pit design.

Rosebel

The Rosebel deposit is located on the central mineralized trend, about 9 kilometres east of Royal Hill. RGM has not yet commenced mining at Rosebel. Topography ranges from 530m adjacent to the savannah to a high of 575m. The pit has a planned strike length of 2,100m (Fig. 17.6).

A combination of 25 metre wide ramps for the majority of the mining, and 15m wide ramps for the lower elevations is proposed. There are two ramps planned, each with access on the north side of the pit. The eastern ramp provides access to the ore in the eastern end of the pit which has a planned final elevation of 390m. The second ramp allows access to the western end of the pit that has a final elevation of 485m.

A small Phase 1 starter pit is planned for Rosebel and will be included in the LOM Schedule.

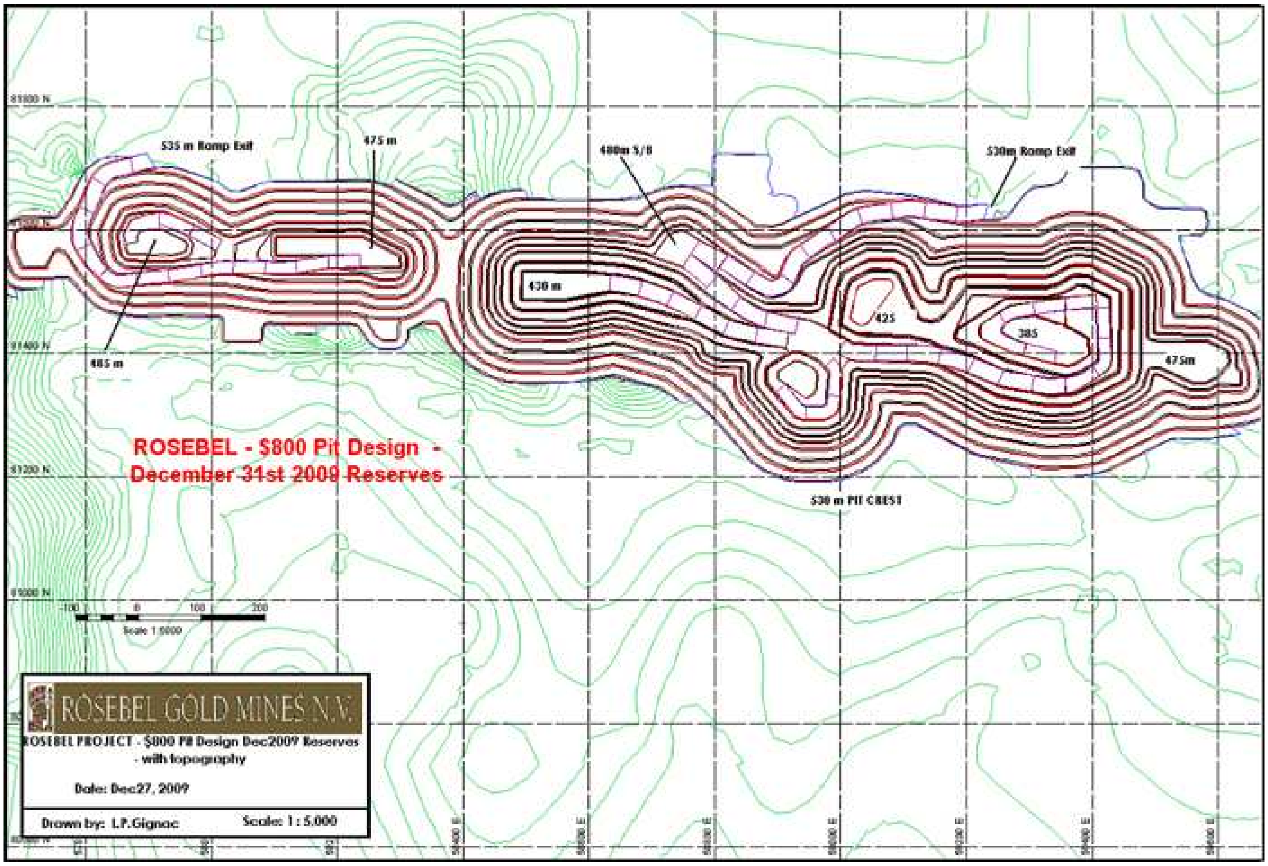


Figure 17.6. Rosebel pit design.

J-Zone

J-Zone is located to the north of the Pay Caro Pit. Two pits with a total strike length of approximately 2,500m are planned to be mined: the West Pit and East Pit (Fig. 17.7). The West Pit ranges in elevation from 420m at the pit bottom to 570m at highest pit crest for a total pit depth of 100m. The East Pit has the pit bottom at 470m with the pit crest at an elevation of 600m.

The East pit consists of a single ramp providing access to the bottom of the pit at an elevation of 470m. The access for the ramp is on the western side of the pit, in between the two pits. The West pit has two ramps. One ramp is located on the south-eastern side of the pit and provides access to the deepest part of the pit at an elevation of 420m. The second ramp provides access to the western side of the pit at a maximum elevation of 460m.

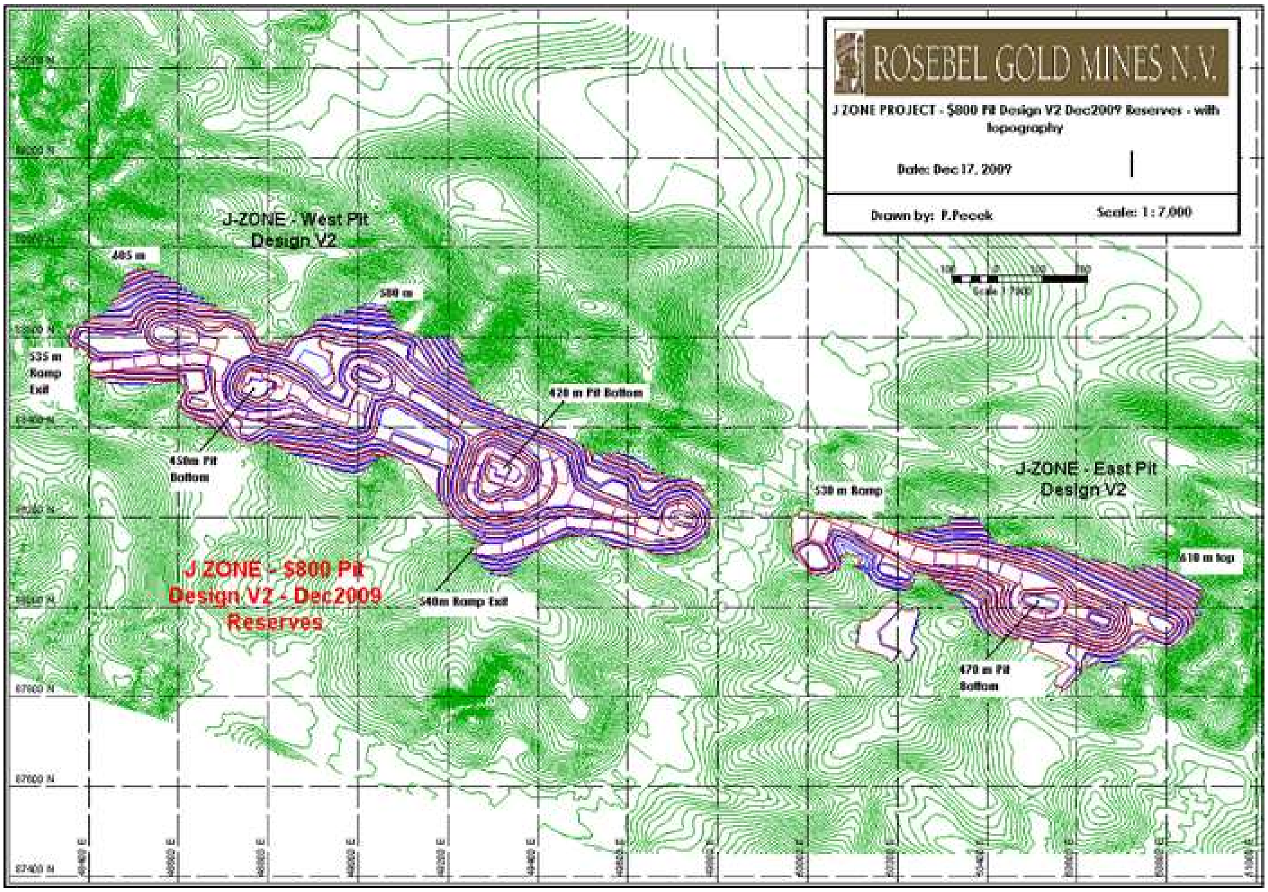


Figure 17.7. J-Zone East and West pit designs.

Roma

Roma is a new open pit design added to the RGM reserves this year. It is located between the Royal Hill and Mayo deposits on the southern mineralized trend. The topography is relatively flat, with elevations not rising more than 20m above the 515m elevation of the surrounding savannah. Two pits are proposed to be mined, a larger one to the west and a smaller sized excavation to the east (Fig. 17.8).

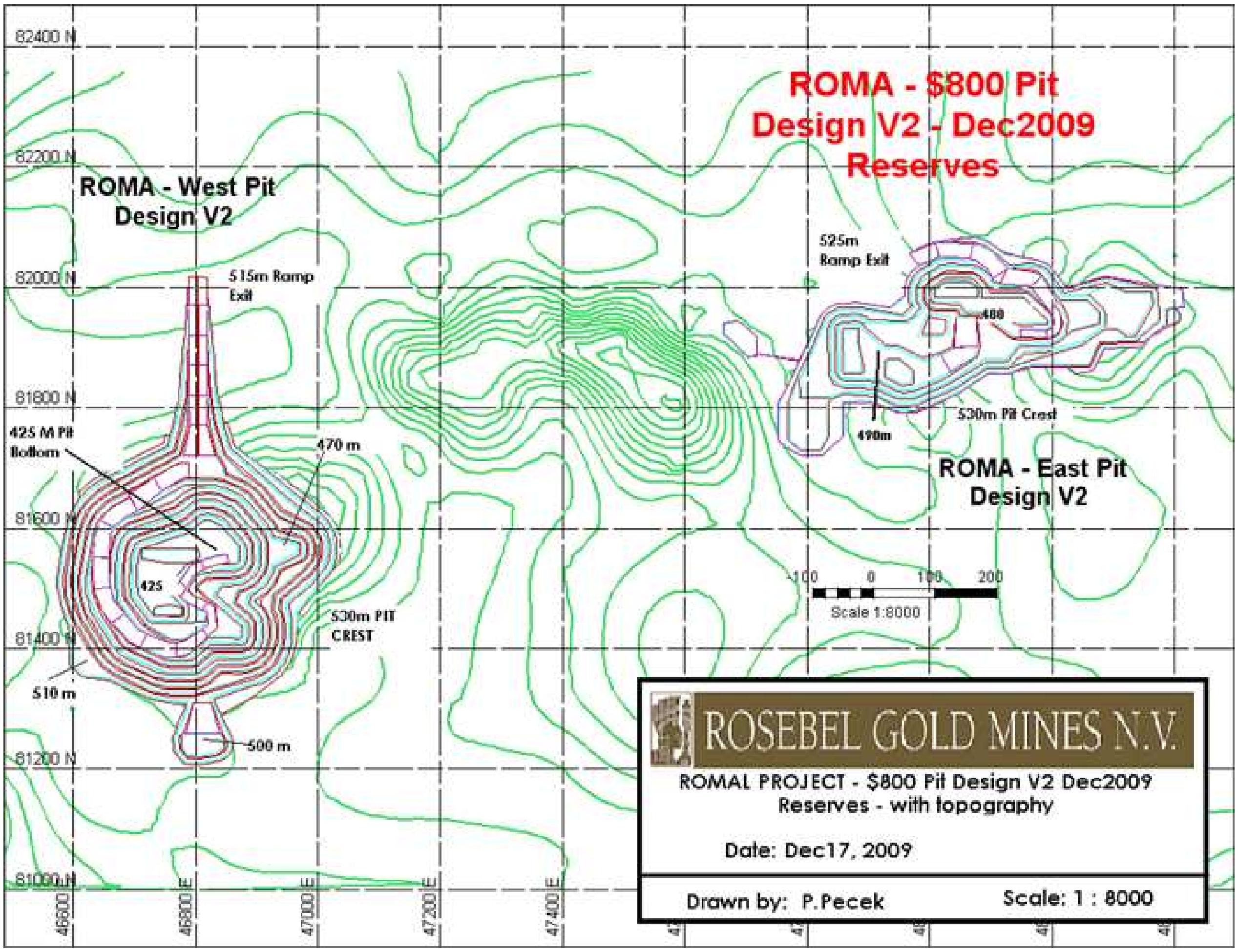


Figure 17.8 . Roma East and West pit designs.

Summary

After the Whittle pit optimization routines were completed to yield a set of theoretical pit shells, the pits were designed using *GEMCOM* software taking into account the fact that practical access would be required using ramps and geotechnical pit wall angles. Additional mining areas require slightly different cut-offs to account for variable haulage distances.

Measured resources are transferred to proven reserves and indicated resources to probable reserves for the Pay Caro, East Pay Caro, Royal Hill, Koolhoven and Mayo deposits as mining proceeds. The ore continuity and gold grade estimates were confirmed by mining activities. In addition, the measured resources from pits not mined yet are transferred to probable reserves. Stockpile inventory is also classified as proven reserves.

17.10. Mineral Resources

The 2009 Rosebel mineral resource estimation is summarized on Tables 17.7 and 17.8.

For 2009 mineral resources and reserves, it was decided to take in account the historical reconciliations between ore production and mining reserves by using adjustment factors.

Chapter 18 details how the adjustment factors have been calculated.

Table 7.12 shows the adjustment factors by pit and by material type.

The adjustment factors will be re-evaluated every year and readjusted to take in account the latest reconciliation between ore production, mineral reserves and mill results.

Deposit	Laterite			Saprolite			Transition			Rock		
	Tonnes (000)	Au (g/t)	Au (K oz)	Tonnes (000)	Au (g/t)	Au (K oz)	Tonnes (000)	Au (g/t)	Au (K oz)	Tonnes (000)	Au (g/t)	Au (K oz)
Koolhoven/Bigi	0.81	1.11	0.89	0.81	1.11	0.89	1.59	1.12	1.81	1.00	1.00	1.00
JZone	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00
Pay Caro	0.86	1.01	0.86	0.86	1.01	0.86	1.14	0.99	1.12	1.10	0.98	1.08
East Pay Caro	1.04	1.08	1.13	1.04	1.08	1.13	1.00	1.00	1.00	1.00	1.00	1.00
Mayo	1.18	1.09	1.30	1.18	1.09	1.30	1.00	1.00	1.00	1.00	1.00	1.00
Roma	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00
Royal Hill	1.14	1.10	1.26	1.14	1.10	1.26	0.82	1.05	0.86	1.21	0.94	1.14
Rosebel	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00

Table 17.7. Adjustment factors by pit and material type.

The 2009 Rosebel mineral resource estimation is summarized on Tables 17.8 and 17.9. The measured and indicated resources inside the pit shells at a gold price of US$1,000/oz indicates a significant upside potential related to the gold price.

Rosebel Gold Mines N.V.
Measured and Indicated Resources - 31 December 2009 (inside pit shells)
$1000 / ounce including Adjustment Factors

Deposit	Cutoffs			Laterite			Saprolite			Transition			Rock			Total			Gold	%
	Laterite & Saprolite	Transition	Hard Rock	Tonnes (000)	Au (g/t)	Au (K oz)	Tonnes (000)	Au (g/t)	Au (K oz)	Tonnes (000)	Au (g/t)	Au (K oz)	Tonnes (000)	Au (g/t)	Au (K oz)	Tonnes (000)	Au (g/t)	Au (K oz)	Ounces contained	Au (K oz)
Koolhoven/Bigi	0.25	0.30	0.35	356	0.67	8	3,761	0.84	100	17,054	0.99	553	1,758	1.17	66	22,929	1.0	727	727,000	10%
JZone	0.25	0.30	0.35	966	0.66	20	1,491	0.71	34	5,298	0.86	147	7,132	0.91	208	14,887	0.9	409	409,000	6%
Pay Caro	0.24	0.28	0.33	599	0.62	12	752	0.63	15	4,054	0.79	103	29,345	1.04	981	34,750	1.0	1,111	1,111,000	16%
East Pay Caro	0.25	0.29	0.34	573	0.70	13	818	0.89	23	2,797	0.94	84	13,579	1.05	458	17,767	1.0	579	579,000	8%
Mayo	0.27	0.31	0.36	4,539	0.72	106	12,500	1.00	405	5,213	0.93	156	31,477	1.03	1041	53,729	1.0	1,708	1,708,000	24%
Roma	0.26	0.30	0.35	748	0.84	20	1,382	0.83	37	924	1.15	34	2,576	1.09	90	5,630	1.0	181	181,000	3%
Royal Hill	0.26	0.30	0.35	1,647	0.72	38	1,390	0.94	42	2,005	1.04	67	48,674	1.08	1687	53,717	1.1	1,834	1,834,000	26%
Rosebel	0.27	0.32	0.37	3,097	0.71	71	5,216	0.90	151	3,316	0.86	92	7,758	1.14	284	19,388	1.0	597	597,000	8%
TOTAL				12,525	0.72	288	27,312	0.92	808	40,661	0.94	1,235	142,299	1.05	4,815	222,797	1.0	7,147	7,147,000	100%
Stockpiles																3,192	0.8	84	84,000	
GRAND TOTAL	M+I Resources			12,525	0.7	288	27,312	0.9	808	40,661	0.9	1,235	142,299	1.1	4,815	225,989	1.0	7,231	7,231,000	

Rosebel Gold Mines N.V.
Inferred Resources - 31 December 2009 (inside pit shell limit)
$1000 / ounce including Adjustment Factors

Deposit	Cutoffs			Laterite			Saprolite			Transition			Rock			Total			Gold	%
	Laterite & Saprolite	Transition	Hard Rock	Tonnes (000)	Au (g/t)	Au (K oz)	Tonnes (000)	Au (g/t)	Au (K oz)	Tonnes (000)	Au (g/t)	Au (K oz)	Tonnes (000)	Au (g/t)	Au (K oz)	Tonnes (000)	Au (g/t)	Au (K oz)	Ounces contained	Au (K oz)
Koolhoven/Bigi	0.25	0.30	0.35	128	0.92	4	120	1.29	5	169	1.73	10	5	1.29	0	422	1.4	18	18,000	2%
JZone	0.25	0.30	0.35	488	0.54	8	179	0.49	3	99	1.17	4	182	1.30	8	948	0.7	23	23,000	2%
Pay Caro	0.24	0.28	0.33	387	0.44	5	138	0.48	2	249	0.82	7	4,919	1.06	168	5,693	1.0	183	183,000	19%
East Pay Caro	0.25	0.29	0.34	399	0.44	6	131	0.70	3	172	0.50	3	1,012	0.93	30	1,714	0.8	42	42,000	4%
Mayo	0.27	0.31	0.36	543	0.49	9	661	0.57	12	156	0.59	3	6,842	1.30	286	8,203	1.2	310	310,000	33%
Roma	0.26	0.30	0.35	872	0.61	17	67	0.70	2	12	0.89	0				951	0.6	19	19,000	2%
Royal Hill	0.26	0.30	0.35	815	0.54	14	71	0.48	1	16	0.48	0	6,226	1.42	283	7,128	1.3	299	299,000	32%
Rosebel	0.27	0.32	0.37	450	0.41	6	700	0.68	15	251	0.74	6	709	0.91	21	2,110	0.7	48	48,000	5%
TOTAL				4,081	0.53	69	2,067	0.65	43	1,126	0.89	32	19,896	1.24	796	27,169	1.1	941	941,000	100%

Table 17.8. 2009 Rosebel Mineral Resources.

Measured Resources - 31 December 2009 (inside pit shells)
$1000 / ounce including Adjustment Factors

Deposit	Cutoffs			Laterite			Saprolite			Transition			Rock			Total			Gold	%
	Laterite & Saprolite	Transition	Hard Rock	Tonnes (000)	Au (g/t)	Au (K oz)	Tonnes (000)	Au (g/t)	Au (K oz)	Tonnes (000)	Au (g/t)	Au (K oz)	Tonnes (000)	Au (g/t)	Au (K oz)	Tonnes (000)	Au (g/t)	Au (K oz)	Ounces contained	Au (K oz)
Koolhoven/Rigi	0.25	0.30	0.35	244	0.66	5	2,758	0.85	74	12,982	0.96	408	874	1.15	32	16,859	1.0	520	520,000	14%
JZone	0.25	0.30	0.35	477	0.71	11	471	0.68	10	2,887	0.90	83	4,145	0.88	117	7,980	0.9	221	221,000	6%
Pay Caro	0.24	0.28	0.33	344	0.67	7	288	0.71	7	2,472	0.81	64	12,951	1.05	436	16,055	1.0	514	514,000	13%
East Pay Caro	0.25	0.29	0.34	336	0.61	7	396	0.65	8	1,437	0.69	32	5,890	0.75	143	8,059	0.7	189	189,000	5%
Mayo	0.27	0.31	0.36	2,575	0.79	66	8,141	1.05	277	3,648	0.94	110	13,286	0.96	412	27,650	1.0	865	865,000	23%
Roma	0.26	0.30	0.35	155	0.54	3	528	0.78	13	450	1.13	16	1,565	1.07	54	2,698	1.0	86	86,000	2%
Royal Hill	0.26	0.30	0.35	519	0.81	14	854	1.03	29	1,498	1.09	52	28,465	1.10	1,004	31,335	1.1	1,099	1,099,000	29%
Rosebel	0.27	0.32	0.37	1,834	0.81	48	3,036	0.96	94	2,014	0.81	53	3,752	1.08	131	10,637	1.0	325	325,000	9%
TOTAL				6,483	0.77	160	16,472	0.97	512	27,389	0.93	819	70,928	1.02	2,329	121,272	1.0	3,820	3,820,000	100%
Stockpiles																3,192	0.8	84	84,000	
GRAND TOTAL	M Resources			6,483	0.8	160	16,472	1.0	512	27,389	0.9	819	70,928	1.0	2,329	124,464	1.0	3,904	3,904,000	

Rosebel Gold Mines N.V.
Indicated Resources - 31 December 2009 (inside pit shell)
$1000 / ounce including Adjustment Factors

Deposit	Cutoffs			Laterite			Saprolite			Transition			Rock			Total			Gold	%
	Laterite & Saprolite	Transition	Hard Rock	Tonnes (000)	Au (g/t)	Au (K oz)	Tonnes (000)	Au (g/t)	Au (K oz)	Tonnes (000)	Au (g/t)	Au (K oz)	Tonnes (000)	Au (g/t)	Au (K oz)	Tonnes (000)	Au (g/t)	Au (K oz)	Ounces contained	Au (K oz)
Koolhoven/Rigi	0.25	0.30	0.35	112	0.70	2	1,003	0.81	26	4,072	1.09	145	884	1.20	34	6,071	1.1	208	208,000	6%
JZone	0.25	0.30	0.35	489	0.61	10	1,020	0.72	24	2,411	0.82	63	2,987	0.95	91	6,908	0.8	188	188,000	6%
Pay Caro	0.24	0.28	0.33	256	0.56	5	463	0.57	9	1,581	0.77	39	16,394	1.03	544	18,695	1.0	596	596,000	18%
East Pay Caro	0.25	0.29	0.34	237	0.82	6	423	1.11	15	1,359	1.20	52	7,689	1.28	315	9,709	1.2	389	389,000	12%
Mayo	0.27	0.31	0.36	1,964	0.63	40	4,359	0.91	128	1,565	0.90	45	18,191	1.07	629	26,078	1.0	843	843,000	25%
Roma	0.26	0.30	0.35	593	0.92	17	854	0.86	24	474	1.17	18	1,011	1.13	37	2,932	1.0	96	96,000	3%
Royal Hill	0.26	0.30	0.35	1,128	0.68	25	536	0.78	14	508	0.87	14	20,210	1.06	683	22,382	1.0	735	735,000	22%
Rosebel	0.27	0.32	0.37	1,263	0.56	23	2,180	0.81	57	1,302	0.93	39	4,005	1.19	153	8,751	1.0	272	272,000	8%
TOTAL				6,042	0.66	128	10,839	0.85	296	13,272	0.98	416	71,371	1.08	2,486	101,525	1.0	3,327	3,327,000	100%

Rosebel Gold Mines N.V.
Inferred Resources - 31 December 2009 (inside pit shell limit)
$1000 / ounce including Adjustment Factors

Deposit	Cutoffs			Laterite			Saprolite			Transition			Rock			Total			Gold	%
	Laterite & Saprolite	Transition	Hard Rock	Tonnes (000)	Au (g/t)	Au (K oz)	Tonnes (000)	Au (g/t)	Au (K oz)	Tonnes (000)	Au (g/t)	Au (K oz)	Tonnes (000)	Au (g/t)	Au (K oz)	Tonnes (000)	Au (g/t)	Au (K oz)	Ounces contained	Au (K oz)
Koolhoven/Bigi	0.25	0.30	0.35	128	0.92	4	120	1.29	5	169	1.73	10	5	1.29	0	422	1.4	18	18,000	2%
JZone	0.25	0.30	0.35	488	0.54	8	179	0.49	3	99	1.17	4	182	1.30	8	948	0.7	23	23,000	2%
Pay Caro	0.24	0.28	0.33	387	0.44	5	138	0.48	2	249	0.82	7	4,919	1.06	168	5,693	1.0	183	183,000	19%
East Pay Caro	0.25	0.29	0.34	399	0.44	6	131	0.70	3	172	0.50	3	1,012	0.93	30	1,714	0.8	42	42,000	4%
Mayo	0.27	0.31	0.36	543	0.49	9	661	0.57	12	156	0.59	3	6,842	1.30	286	8,203	1.2	310	310,000	33%
Roma	0.26	0.30	0.35	872	0.61	17	67	0.70	2	12	0.89	0				951	0.6	19	19,000	2%
Royal Hill	0.26	0.30	0.35	815	0.54	14	71	0.48	1	16	0.48	0	6,226	1.42	283	7,128	1.3	299	299,000	32%
Rosebel	0.27	0.32	0.37	450	0.41	6	700	0.68	15	251	0.74	6	709	0.91	21	2,110	0.7	48	48,000	5%
TOTAL				4,081	0.53	69	2,067	0.65	43	1,126	0.89	32	19,896	1.24	796	27,169	1.1	941	941,000	100%

Table 17.9. 2009 Measured, Indicated and Inferred Resources.

17.11. Mineral Reserves

The 2009 Rosebel mineral reserve estimation is summarized on Tables 17.10 and 17.11. The mineral reserves are included in mineral resources presented in Chapter 17.10.

The mineral reserves have been estimated by the qualified person of this report, Gabriel Voicu. Mr. Voicu is the Geology & Mine Exploration Superintendent of RGM, which is a subsidiary of Iamgold Corporation.

Rosebel Gold Mines N.V.
Mineral Reserve Estimates - December 31, 2009
$850 / ounce including Adjustment Factors

Proven Mineral Reserves

Ore:

Deposit	Cutoffs			Laterite			Saprolite			Transition			Rock			Total		
	Laterite & Saprolite	Transition	Hard Rock	Tonnes (000)	Au (g/t)	Au (K oz)	Tonnes (000)	Au (g/t)	Au (K oz)	Tonnes (000)	Au (g/t)	Au (K oz)	Tonnes (000)	Au (g/t)	Au (K oz)	Tonnes (000)	Au (g/t)	Au (K oz)
Koolhoven/Bigi	0.30	0.35	0.40	183	0.76	4	2,395	0.92	70	9,277	1.05	320	175	1.30	7	12,030	1.0	401
JZone	0.30	0.35	0.40															
Pay Caro	0.28	0.33	0.39	216	0.68	5	209	0.83	6	1,918	0.90	55	10,579	1.11	380	12,923	1.1	445
East Pay Caro	0.29	0.34	0.39	283	0.66	6	368	0.68	8	1,334	0.71	31	4,864	0.77	121	6,849	0.7	165
Mayo	0.31	0.36	0.42	2,271	0.85	62	7,272	1.12	265	2,856	1.04	95	8,468	1.09	295	20,868	1.1	718
Roma	0.30	0.35	0.41															
Royal Hill	0.30	0.35	0.41	216	0.87	6	695	1.12	25	1,273	1.20	49	23,632	1.18	894	25,816	1.2	974
Rosebel	0.32	0.37	0.43															
TOTAL				3,171	0.82	83	10,939	1.1	374	16,658	1.0	549	47,719	1.1	1,697	78,486	1.1	2,703
Stockpiles																3,192	0.8	84
GRAND TOTAL	**Proven**			3,171	0.8	83	10,939	1.1	374	16,658	1.0	549	47,719	1.1	1,697	81,678	1.1	2,787

Probable Mineral Reserves

Ore:

Deposit	Cutoffs			Laterite			Saprolite			Transition			Rock			Total		
	Laterite & Saprolite	Transition	Hard Rock	Tonnes (000)	Au (g/t)	Au (K oz)	Tonnes (000)	Au (g/t)	Au (K oz)	Tonnes (000)	Au (g/t)	Au (K oz)	Tonnes (000)	Au (g/t)	Au (K oz)	Tonnes (000)	Au (g/t)	Au (K oz)
Koolhoven/Bigi	0.30	0.35	0.40	57	0.7	1	720	1.0	22	2,607	1.2	103	151	1.3	6	3,536	1.2	133
JZone	0.30	0.35	0.40	665	0.8	16	1,254	0.8	31	4,047	0.9	124	2,177	1.0	72	8,143	0.9	243
Pay Caro	0.28	0.33	0.39	106	0.6	2	277	0.7	6	583	1.0	18	5,869	1.3	245	6,834	1.2	272
East Pay Caro	0.29	0.34	0.39	182	1.0	6	373	1.2	14	1,232	1.3	51	4,881	1.4	222	6,668	1.4	293
Mayo	0.31	0.36	0.42	1,451	0.7	34	3,395	1.0	114	887	1.1	32	5,117	1.3	210	10,850	1.1	390
Roma	0.30	0.35	0.41	704	0.9	20	1,135	0.9	32	721	1.3	30	1,669	1.2	64	4,229	1.1	146
Royal Hill	0.30	0.35	0.41	319	0.7	8	301	1.0	9	288	0.9	8	8,753	1.2	325	9,661	1.1	350
Rosebel	0.32	0.37	0.43	2,349	0.8	60	4,489	1.0	141	2,665	0.9	80	5,048	1.2	200	14,552	1.0	480
TOTAL				5,833	0.8	147	11,945	1.0	370	13,031	1.1	446	33,665	1.2	1,343	64,474	1.1	2,307
Stockpiles																		
GRAND TOTAL	**Probable**			5,833	0.8	147	11,945	1.0	370	13,031	1.1	446	33,665	1.2	1,343	64,474	1.1	2,307

Table 17.10. 2009 Rosebel Proven and Probable Mineral Reserves.

Rosebel Gold Mines N.V.
Proven and Probable Mineral Reserve Estimates - December 31, 2009
$850 / ounce including Adjustment Factors

Deposit	Cutoffs			Laterite			Saprolite			Transition			Rock			Total		
	Laterite & Saprolite	Transition	Hard Rock	Tonnes (000)	Au (g/t)	Au (K oz)	Tonnes (000)	Au (g/t)	Au (K oz)	Tonnes (000)	Au (g/t)	Au (K oz)	Tonnes (000)	Au (g/t)	Au (K oz)	Tonnes (000)	Au (g/t)	Au (K oz)
Koolhoven/Bigi	0.30	0.35	0.40	240	0.8	6	3,115	0.9	92	11,884	1.1	423	326	1.3	14	15,566	1.1	535
JZone	0.30	0.35	0.40	665	0.8	16	1,254	0.8	31	4,047	0.9	124	2,177	1.0	72	8,143	0.9	243
Pay Caro	0.28	0.33	0.39	322	0.7	7	486	0.7	11	2,501	0.9	74	16,448	1.2	625	19,757	1.1	717
East Pay Caro	0.29	0.34	0.39	466	0.8	12	741	0.9	22	2,566	1.0	81	9,745	1.1	342	13,517	1.1	458
Mayo	0.31	0.36	0.42	3,722	0.8	96	10,667	1.1	379	3,743	1.1	127	13,585	1.2	505	31,718	1.1	1,107
Roma	0.30	0.35	0.41	704	0.9	20	1,135	0.9	32	721	1.3	30	1,669	1.2	64	4,229	1.1	146
Royal Hill	0.30	0.35	0.41	535	0.8	14	996	1.1	35	1,561	1.1	57	32,385	1.2	1,219	35,477	1.2	1,324
Rosebel	0.32	0.37	0.43	2,349	0.8	60	4,489	1.0	141	2,665	0.9	80	5,048	1.2	200	14,552	1.0	480
TOTAL				9,003	0.8	231	22,884	1.0	743	29,689	1.0	996	81,384	1.2	3,040	142,960	1.1	5,010
Stockpiles																3,192	0.8	84
GRAND TOTAL	**Proven & Probable**			9,003	0.8	231	22,884	1.0	743	29,689	1.0	996	81,384	1.2	3,040	146,152	1.1	5,094

Deposit	Laterite Ore Tonnes (000)	Laterite Waste Tonnes (000)	Saprolite Ore Tonnes (000)	Saprolite Waste Tonnes (000)	Trans. Ore Tonnes (000)	Trans. Waste Tonnes (000)	Rock Ore Tonnes (000)	Rock Waste Tonnes (000)	Total Ore Tonnes (000)	Total Waste Tonnes (000)	Total Ore+Waste Tonnes (000)	Strip Ratio
Koolhoven/Bigi	240	530	3,115	15,891	11,884	30,890	326	1,408	15,566	48,719	64,285	3.1
JZone	665	700	1,254	6,627	4,047	16,258	2,177	6,039	8,143	29,624	37,767	3.6
Pay Caro	322	808	486	5,715	2,501	14,570	16,448	55,893	19,757	76,986	96,744	3.9
East Pay Caro	466	905	741	5,549	2,566	17,053	9,745	26,576	13,517	50,083	63,601	3.7
Mayo	3,722	3,075	10,667	39,439	3,743	17,574	13,585	52,036	31,718	112,124	143,842	3.5
Roma	704	1,372	1,135	9,351	721	3,588	1,669	4,926	4,229	19,238	23,468	4.5
Royal Hill	535	849	996	17,613	1,561	22,818	32,385	118,842	35,477	160,121	195,598	4.5
Rosebel	2,349	2,009	4,489	23,352	2,665	12,233	5,048	15,762	14,552	53,356	67,908	3.7
TOTAL	9,003	10,248	22,884	123,538	29,689	134,984	81,384	281,483	142,960	550,252	693,212	3.8

Table 17.11. 2009 Rosebel Proven + Probable Mineral Reserves.

18. RECONCILIATION BETWEEN RESERVES AND PRODUCTION

For 2009, mine production was higher than the mineral reserves by 12% in tonnes and 15% in grade for a total of 28% in ounces (Table 18.1). The mineral reserves have systematically understated ore mining production since the start of the Rosebel operation in 2003. For the mine life, the comparison shows a gain of 12% in tonnes and 7% in grade for a total gain of 19% in ounces (Table 18.1).

The main reason for ore mining production reconciling systematically higher than the mining reserves relates to the orientation of diamond drilling during exploration. Each of the Rosebel deposits is characterized by multiple and variably orientated mineralized vein systems. The predominant drilling orientation does not allow for optimal intersection of all these vein systems. Consequently, modeling and reserve estimation typically underestimate at least one vein system.

For the December 2009 mineral reserves and resources, it was decided to take into account the historical reconciliations between ore production and mining reserves by using adjustment factors. The adjustment factors have been calculated by extracting the difference between the mill feed grade and mill grade (which is 2% for the mine life, see Table 18.2) from the 19% difference in ounces between ore production and ore reserves. The average adjustment factor of 17% in ounces was split by deposits using several criteria. The adjustment factors have been used only in the pits with mining history (Pay Caro, East Pay Caro, Royal Hill, Koolhoven, and Mayo), only for the material types (laterite, saprolite, transition or hard rock) that have been already mined in the active pit and only if the mined tonnage of a certain material type such as saprolite, transition or fresh rock was over 1 M tonnes. In addition, as 2009 will be the first time that adjustment factors will be applied to the mining reserves and resources, only 65% of the calculated adjustment factors by pit and by material type have been used to adjust the mineral reserves.

Pit	YTD 2009			Mine Life		
	Mine Production	Mining Reserves	Variance %	Mine Production	Mining Reserves	Variance %
Pay Caro (Dec08 BM)						
Tonnage (t)	1,952,820	1,868,908	4%	18,860,525	17,440,824	8%
Grade (g Au/t)	1.23	1.07	15%	1.45	1.48	-2%
Ounces diff.			**20%**			**6%**
East Pay Caro (Dec08 BM)						
Tonnage (t)	81,260	91,316	-11%	2,931,176	2,665,240	10%
Grade (g Au/t)	0.80	1.07	-26%	1.50	1.37	10%
Ounces diff.			**-34%**			**21%**
Royal Hill (Dec08 BM)						
Tonnage (t)	3,596,614	2,995,905	20%	19,307,293	16,492,811	17%
Grade (g Au/t)	1.27	1.15	10%	1.38	1.23	12%
Ounces diff.			**32%**			**31%**
Koolhoven (Dec08 BM)						
Tonnage (t)	3,614,252	3,655,150	-1%	5,819,841	5,746,113	1%
Grade (g Au/t)	1.08	0.92	18%	1.07	0.92	17%
Ounces diff.			**17%**			**18%**
Mayo (Dec08 BM)						
Tonnage (t)	3,130,267	2,452,482	28%	3,130,267	2,452,482	28%
Grade (g Au/t)	1.32	1.17	13%	1.32	1.17	13%
Ounces diff.			**45%**			**45%**
Total						
Tonnage (t)	12,375,213	11,063,761	12%	50,049,102	44,797,470	12%
Grade (g Au/t)	1.22	1.06	15%	1.38	1.29	7%
Ounces diff.			**28%**			**19%**

Table 18.1. Reconciliation between ore production and mineral reserves for 2009 and mine life.

	Mine		Reserves		Mill Feed		Milling		Stockpiles (end of year)	
	Tonnes	Grade (Au g/t)	Tonnes	Grade (Au g/t)			Tonnes	Grade (Au g/t)	Tonnes	Grade (Au g/t)
2003	454,524	1.28	541,692	1.42					454,524	1.28
2004	6,661,264	1.73	5,650,604	1.61	5,059,784	1.91	5,067,021	1.84	664,129	1.39
2005	6,861,672	1.57	5,974,235	1.55	7,210,197	1.56	7,196,385	1.56	1,736,119	1.11
2006	7,049,633	1.40	5,711,023	1.43	7,722,086	1.40	7,708,773	1.30	1,076,840	0.93
2007	7,764,177	1.27	7,840,900	1.15	7,492,111	1.29	7,505,281	1.24	1,335,279	0.86
2008	8,882,623	1.26	8,015,256	1.22	8,367,569	1.28	8,308,195	1.35	1,909,703	0.89
2009	12,375,213	1.22	11,063,761	1.06	11,113,626	1.25	11,093,074	1.25	3,192,382	0.81
Total	**50,049,106**	**1.38**	**44,797,471**	**1.29**	**46,965,372**	**1.40**	**46,878,729**	**1.38**		

Comparison	Tonnes	Grade
Mine/Reserves	11.7%	6.9%
Mill Feed/Milling	0.0%	2%

Table 18.2. Reconciliation between mining production, mineral reserves and milling.

19. ADDITIONAL REQUIREMENTS FOR TECHNICAL REPORTS ON DEVELOPMENT PROPERTIES AND PRODUCTION PROPERTIES

19.1 MINING METHOD

Mining operations employ conventional open pit methods using a 5-meter bench interval and CAT 777 haul trucks with seven CAT 5130 (in both shovel and backhoe configurations) and two CAT 993K wheel-loaders. A fleet of ten production drills create a 3.9m x 4.5m grid pattern for sampling using 6-inch diameter holes. Holes in hard rock and transition rock are loaded with explosives for blasting. Ore blocks are surveyed after drilling, blasted (where necessary) and then separated from the waste zones for delivery to the mill.

19.2 MINE OPERATIONS

Mining took place in 6 pits in 2009, namely Pay Caro, East Pay Caro, Royal Hill North-West, Royal Hill South-East, Koolhoven and Mayo.

At Pay Caro during 2009, phase 1A was completed and the push back of phase 3 began. Pay Caro pit operated in 2009 with two Caterpillar 5130 hydraulic shovels until December when one shovel was moved to East Pay Caro. The material mined was a mix of saprolite, transition and rock. This pit, at a depth of 165 meters, is the deepest of the active pits at Rosebel. The pit is constantly monitored for wall stability with an array of 24 prisms and more than 100 drainage holes. Since the completion of phase 1A, we plan to keep the water level below the elevation of the rock/transition interface. The dewatering layout remains unchanged; the water is evacuated to a sedimentation pond close to the administration building to allow the deposition of total solids in suspension.

In December 2009, we began phase 2 in East Pay Caro with one Caterpillar 5130 shovel transferred from Pay Caro. The material mined was a mix of saprolite and transition. No dewatering was required.

At Royal Hill, mining takes place in two distinct pits namely the North-West and the South-East. The North-West pit operated in 2009 with one Caterpillar 5130 hydraulic shovel. It is a 70 meter deep rock pit and is deep enough to require dewatering. Water is sent to a sedimentation pond on the north side of the pit prior to release into the environment. As required by ISO 14001 standard procedures, every sedimentation pond at Rosebel is tested periodically. The South-East pit operated in 2009 with two Caterpillar 5130 hydraulic shovels. The South-East pit is 50 meters deep and does not currently require dewatering.

The production at Koolhoven began at the end of 2007. The Koolhoven pit operated in 2009 with two Caterpillar 5130 hydraulic shovels. The material mined was a mix of saprolite and transition. Because the production consists of excavating hill tops, Koolhoven is a free drainage pit. The excavation needed to change the course of Foto creek has began and drainage should be fully operational in the first quarter of 2010.

In January 2009, Mayo became the fifth pit to be mined at Rosebel. One hydraulic shovel from the Royal Hill South-East pit was moved into Mayo to begin production. The material mined is essentially saprolite. Because the production consists of excavating hill tops, Mayo is a free drainage pit.

The yearly milling rate was budgeted at 10.0 Mt for 2009. The following table lists the production levels planned for the next five years.

Year	Mining production (Mt)	Milling Production (Mt)
2010	57.0	11.5
2011	57.0	11.5
2012	57.0	11.5
2013	57.0	11.5
2014	57.0	11.5

Table 19. 1 Five Year Production Planning

The mine was budgeted $19.6 million for capital spending in 2009. This includes $4.1 million for sustaining capital, $3.2 million to bring the tailings dam to the 547m elevation, $9.1 million for replacement mining equipment and $3.2 million for new mining equipment.

19.3 Recoverability

Further process optimization continues to target constraints and opportunities to further increase plant capacity and performance. Control strategies have been implemented to better control plant operational parameters with net benefits in process stabilization / automation. Ore characterization is currently in process which will establish mill work indexes and ore leach characteristics, grinding capacity and leach kinetic definition will be further defined resulting in increased plant capacity and grinding rates. Process optimization has also been extended to gravity / leach CIL through process audits currently underway, expected results will target improved recoveries.

Rosebel Gold Mines, which has an Environmental Management System (EMS), is ISO 14001-2004 certified. Each year SGS International Certification Services Canada Inc. (SGS) conducts a maintenance audit on the EMS and every three years a re-registration audit is conducted for re-certifying the EMS. Each year the EMS is successfully implemented by meeting the requirements of the ISO 14001 management standard.

19.4 Environmental Considerations

Progressive reclamation activities have been undertaken by RGM, and are described in the RGM mine closure plan. Reclamation activities have been mainly focused on Royal Hill, Pay Caro and in 2009 on the Koolhoven waste rock dump. In 2009 a nursery was set up and a pilot project was started with the Clitoria Fairchildiana.

A hydroseeder has been purchased to simplify reclamation activities. As of September 30th, 2009 the total amount of land disturbed, as a result of Rosebel Mine's activities, is estimated at 1,580 ha. The total amount of land estimated to be disturbed, as a result of Rosebel Mine's activities (including opening of future pits), is estimated to be approximately 2,435 ha for the whole life of mine.

19.5 TAXES

The relationship between IMG-Qc (as successor to Cambior), a wholly-owned subsidiary of the Company, RGM and the Republic of Suriname is governed by a mineral agreement executed on April 7th, 1994, as amended and supplemented by an agreement dated March 13th, 2003 (the “Mineral Agreement”). The Mineral Agreement provides, in particular, for the Republic of Suriname holding a 5% carried participation in the share capital of RGM (the Class A shares) and 2 million redeemable shares (the Class B shares). The capital structure of RGM had provided for the redemption of Class B shares in accordance with the Commercial Code of Suriname prior to distribution of dividends to its shareholders. In October 2009, the remaining Class B shared held by the Republic of Suriname were redeemed such that as at December 31, 2009, the Republic of Suriname only held 50 Class A shares and IMG QC held 950 Class A shares and 7,999,000 Class B shares in RGM. Through a unanimous resolution of the shareholders of RGM passed in December 2009, only holders of Class A shares are now entitled to receive dividends as and when declared by RGM.

The Mineral Agreement outlines various business conditions, including the right to export gold, to hold funds in foreign bank accounts, to access local currency at market rates and to import goods, with few exceptions, on a duty-free basis. The Mineral Agreement provides for an income tax rate being the lesser of the statutory rate in effect (currently 36%) and 45%, an international dispute resolution mechanism and a debt-to-equity capital structure of 4 to 1.

On December 16, 2002, RGM was granted a 25-year renewable Right of Exploitation for the Rosebel mine by the Government of Suriname, following the Government’s approval of the updated feasibility study and environmental impact assessment.

Production from the Rosebel mine is subject to a fixed royalty of 2% of production, paid in-kind, and, a price participation royalty of 6.5% on the amount exceeding a market price of $425 per ounce of gold, when applicable, payable to the Government of Suriname, and a fixed royalty of 0.25% of production payable in kind to a foundation established by RGM, Grassalco, an entity owned by the Government of Suriname, and the Government of Suriname to promote the local development of natural resources. The Suriname Environmental and Mining Foundation's board is composed of two representatives from RGM, two from Grassalco and one from the Government of Suriname.

19.6 Capital and Operating Cost Estimates

The table bellow shows the operating and capital cost estimates for the next 5 years of mining. The operating costs are based on the 5 year mine plan developed last September 2009. The capital expenditures listed below have been evaluated with the objective to sustain a mining rate of 57 million tonnes per year.

Operating Cost

Items		2010	2011	2012	2013	2014
Mining	$/t Mine	1.53	1.58	1.6	1.58	1.58
Processing	$/t milled	3.11	3.18	3.29	3.34	3.53
G&A	$/t milled	2.35	2.35	2.35	2.35	2.35
Power	$/t milled	2.11	2.01	2.30	2.32	2.37

Capital Cost

Items		Total	2010	2011	2012	2013	2014
Process Plant (Initial + Sustaining)							
Leach Tank	($ x 000)	**16,263**	16,263				
Sustaining Capital	($ x 000)	**5,205**		2,205	1,000	1,000	1,000
Owners Costs (G&A during constr)	($ x 000)	**7,500**	1,500	1,500	1,500	1,500	1,500
Capitalized exploration	($ x 000)	**12,516**	12,516				
Tailings Storage (Initial + Expansions	($ x 000)	**35,180**	3,180	5,000	7,000	10,000	10,000
Mine							
Mine Exploration	($ x 000)	**38,452**	8,224	8,226	8,002	6,200	7,800
777 Cat Truck	($ x 000)	**14,400**	2,400	6,000	3,600	2,400	
Shovel	($ x 000)	**8,000**		4,000			4,000
Dozers	($ x 000)	**3,200**	1,600			1,600	
Graders	($ x 000)	**1,600**		1,600			
Truck Shop	($ x 000)	**4,000**	4,000				
Dewatering	($ x 000)	**1,000**			1,000		
Sustaining Capital	($ x 000)	**4,000**			4,000		
Working Capital	($ x 000)	**0**					
Total Capital Cost	**($ x 000)**	**151,316**	**49,683**	**28,531**	**26,102**	**22,700**	**24,300**

19.7 PAYBACK

Rosebel (RGM) entered into a credit facility agreement with Cambior Inc. Following the acquisition of Cambior by Iamgold Corporation, on November 8, 2006, all rights and obligations of Cambior under the agreement were assumed by Iamgold-Quebec (IMG-Q). Following were the terms and conditions of the agreement:

(i) IMG-Q's committed to make advances under the Credit Facility Agreement up to a maximum principal amount of $120,000,000;

(ii) The loan would mature eight years following the commencement of commercial production (i.e. February 2004);

(iii) RGM was required to make repayments based on a minimum of 80% of its net cash flow or on demand in the event of a material adverse change to RGM's financial condition or financial forecast and projections, and prior to any payment of dividends; and,

(iv) RGM was required to pay interest at US prime plus 3% on the amount drawn under the credit facility.

Under the Mineral Agreement and the Credit Facility Agreement, RGM was required to maintain a debt to equity ratio of 4:1.

The loan under the Credit Facility Agreement was repaid completely in 2009.

19.8 MINE LIFE

At the actual rate of mining, the life of the Rosebel mine is 12 years (until 2022). However, the exploration potential is still high and it is expected that new mineral reserves will be added in the future.

19.9 ECONOMIC CASH FLOW STATEMENT

		Units	Summary	2010	2011	2012	2013	2020	2021
Mining									
Ore Milled		kt	145,128	11,500	11,500	11,500	11,500	11,030	11,000
Grade (Au)		gpt	1.10	1.17	1.13	1.11	1.13	1.12	1.39
Contained (Au)		oz	5,127,779	432,588	418,307	408,703	416,558	396,317	493,288
Gold Price		$/oz	906.92	$1,065	$1,065	$1,000	$940	$850	$850
Gross Revenue		($ x 000)	4,332,392	$430,646	$420,056	$385,953	$367,455	$311,936	$386,130
Operating Cost			Factor						
Mining		$/t Mine	1.0	1.47	1.47	1.47	1.51	1.58	1.58
Processing		$/t milled	1.0	3.07	3.07	3.11	3.15	3.25	3.25
G&A		$/t milled	1.0	2.35	2.35	2.35	2.30	2.05	1.95
Power		$/t milled	1.0	2.11	2.01	2.30	2.32	2.61	2.86
Total Operating Cost		($ x 000)	2,422,092	177,212	173,485	178,606	178,593	199,154	165,082
Capital Cost									
Total Capital Cost		($ x 000)	191,216	49,683	28,531	26,102	22,700	4,400	1,000
Pre-Tax Cash Flow		($ x 000)	1,719,084	203,751	218,040	181,245	166,162	108,383	220,048
Cumulative Pre-Tax Cash Flow Model		($ x 000)		203,751	421,791	603,036	769,198	1,359,467	1,579,515
Tax Analysis									
IMG After Tax Cash Flow		($ x 000)	1,097,757	125,288	142,049	117,538	103,792	68,572	136,504
Cumulative Cash Flow Model		($ x 000)		125,288	267,337	384,875	488,667	857,040	993,544

Financial Analysis	Discount	Pre-Tax	After-Tax
Net Present Value, $ Million	5%	1,282	816
	8%	1,102	631
	10%	1,005	577
	12%	922	531
	15%	820	473

20. INTERPRETATION AND CONCLUSIONS

The Rosebel project is a large-scale, large-tonnage and low-grade mine operation. It consists of eight separate gold deposits, four on the northern mineralized trend (Pay Caro, East Pay Caro, Koolhoven and "J" Zone), one on the central trend (Rosebel) and three on the southern trend (Royal Hill, Roma and Mayo). Five of these were in production in 2009 (Pay Caro (until November), East Pay Caro (since November), Royal Hill, Mayo, and Koolhoven).

Since the commercial operation started in February 2004, a wealth of geological information has been gathered from production and exploration activities. The existing Rosebel databases contain all this information, while additional data is acquired every day.

This data is used for deposit modeling and in the calculations of ore and waste tonnage, grade distribution and resource and reserve estimates. The Rosebel block models are updated at least once a year, as a function of new information coming from the exploration and production drilling. In the active pits, the use of the production drilling data increases the quality and precision of modeling. However, due to the complexity of the Rosebel deposits, the tonnage and grade predicted by the block models still underestimates the production. This was the main reason for the introduction of adjustment factors between production and reserves in 2009.

The estimates presented in this report for Measured, Indicated & Inferred Resources are constrained in pit shells. In 2008, only Inferred Resources were presented in this way, with Measured and Indicated Resources stated without pit shell constraints.

The reserve data presented in this report has been estimated using a gold price of US$850/oz, while the resource data has been estimated using a gold price of US$1,000/oz.

RECOMMENDATIONS

Based on the review of the Rosebel mine for the purposes of this report, the author makes the following recommendations:

- Several tests using 3 metre and 5 metre composites should be performed to verify the grade distribution that best matches the mining production tonnage and grade.
- Several tests should be performed to compare the grade distribution derived from Inverse Distance and Ordinary Kriging modelling.
- Additional definition drilling is required to transfer existing Inferred Resources into the Measured or Indicated Resource categories.
- Most gold deposits at Rosebel are still open at depth and on strike. Exploration drilling is therefore required to identify potential extensions to known deposits and increase the resource base of the project.
- The Rosebel and FILAB laboratory performance must be improved. An external audit covering sample preparation and assay procedures should be considered.
- Standard operating procedures for drilling, sampling and block modelling should be created to ensure greater consistency of results.
- Bulk density data should derive from both the diamond drill core and working face sampling in the active pits.
- Reconciliations between Rosebel mine production and mineral reserves indicate that block models have consistently underestimated production grade and tonnage. The use of the adjustment factors between production and reserves in 2009 should be revised on an annual basis. Fine tuning the block models for some of the active pits could also improve reconciliations.

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21. SIGNATURE PAGE

This report titled Rosebel Mine, Suriname: NI 43-101 Technical Report dated March 29th, 2010 was prepared by Gabriel Voicu, PhD, P. Geo (OGQ#367) Geology & Mine Superintendent Rosebel Gold Mines N.V.

Dated in Suriname, South America
March 29, 2010

Gabriel Voicu, PhD, P.Geo
Geology & Mine Superintendent
Rosebel Gold Mines N.V

22. CERTIFICATE OF QUALIFIED PERSON

As the author of the “Rosebel Mine, Suriname - NI43-101 Technical Report”, dated March 29, 2010, on a mineral property of IAMGOLD CORPORATION (Rosebel Gold Mines) in the Brokopondo District, Suriname, South America, I, Gabriel Voicu, Professional Geologist of the Province of Quebec, Canada, do hereby certify that:

- I reside at 3240 Chèvremont, in the city of Ile Bizard, province of Québec, Canada, H9C 1V7;
- I am a registered member of Québec Order of Geologists (Ordre des Géologues du Québec), # 367, a Fellow of the Society of Economic Geologists # 835295 and a Member of Geological Association of Canada # F6331;
- I graduated from the University of Bucharest, Romania, in 1983 with a MScA degree in Geological Engineering and from Université du Québec à Montréal in 1999 with a PhD degree in Mineral Resources;
- I have practiced my profession in mine geology, exploration and scientific research over the last 25 years;
- I have been working for Cambior\IAMGOLD since 1997 as senior geologist and geology and mine exploration superintendent;
- As I am an employee of Rosebel Gold Mines N.V., I am at Rosebel most week days;
- I have been the Geology & Mine Exploration Superintendent of Rosebel Gold Mines N.V. since 2007 and I have been working at the Rosebel mine since January 2004;

- I have read National Instrument 43-101 – Standards of Disclosure for Mineral Projects and this report has been prepared in compliance therewith;
- I am responsible for the entirely of this technical report;
- I am a “Qualified Person” for the purposes of National Instrument 43-101 – Standards of Disclosure for Mineral Projects;
- I am not independent of IAMGOLD Corporation as I am a full-time employee of Rosebel Gold Mines N.V. (IAMGOLD Surinamese Subsidiary);
- To the best of my knowledge, this technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Prepared at Rosebel Mine site, Brokopondo District, Suriname, this 29 th day of March 2010,

Signed
Gabriel Voicu, PhD, P.Geo (OGQ #367)
Geology & Mine Exploration Superintendent
Rosebel Gold Mines N.V