Effective Date: 30 June 2010

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VALE S.A.

EXTERNAL AUDIT OF MINERAL RESERVES

VOLUME 2, SECTION 8

SOSSEGO MINE

Submitted to:
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REPORT

Project **Number:** 10-1117-0032 Phase 8000

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Executive Summary

Golder Associates S.A. representatives Dr Marcelo Godoy, Mr Ronald Turner, Mr Carlos Becerra and Mr José Bertini visited the site from 21 to 25 June 2010 to carry out an independent audit of the mineral reserves estimated by Vale for the Sossego Copper Project.

During the site visit they inspected mining operations, interviewed personnel and gathered information required to evaluate the appropriateness of the data and methodology used to estimate the mineral resources and mineral reserves. A list of people contacted for this study includes:

- Roberto Albuquerque e Silva – General Manager of Copper Mine Planning and Quality Control
- Juarez Lopes de Morais – Long Term Mining Planning Manager
- Benevides Aires Filho – Master Geologist
- Cassio Diedrich – Geostatictician
- Frederico Santana Castro – Long Term Planning Engineer
- Evandro Costa e Silva – Mineral Process Engineer
- Eugenio Oliveira – Manager of Quality Control and Port Operations
- Aline Salgado – Economic Evaluation
- Alan Araujo – Environmental Geologist

This study includes a review of technical reports, memoranda and supporting technical information obtained from Vale. Reports on previous internal and external technical reviews and audits were also made available to Golder (e.g., an independent reserve reconciliation review of the Sossego Mine by PAH carried out in February, 2008).

The mineral reserve estimates provided to Golder were expected to conform to the requirements of the Securities Exchange Commission's Industry Guide 7 and to Canadian National Instrument (NI) 43-101 using specific terminology from CIM (2004), No exceptions were found to these requirements.

The mineral reserve statement at June 30, 2010 for the Vale was audited by Golder. The mineral reserve audited by Golder was based on the mineral resource models and was prepared using costs, optimisation, mine design and scheduling practices that are appropriate. Golder accepts the procedure adopted to convert the mineral resource into a mineral reserve. The numbers are appropriate for the purpose of public reporting in that they provide an acceptable prediction of the available mineral reserves. The tonnes and grades are reported at an appropriate economic cut-off grade based on documented costs and prices.

The following table with the mineral reserve figures are provided at the appropriate level of precision for public reporting.

Estimated mineral reserve for the Sossego Mine Complex as at 30 June 2010

Mine	Ore Type	Proven			Probable			Proven and Probable		
		Tonnage (Mt)	Cu (%)	Au (g/t)	Tonnage (Mt)	Cu (%)	Au (g/t)	Tonnage (Mt)	Cu (%)	Au (g/t)
Sossego Complex	Sulphide Ore	100.55	0.97	0.27	39.58	0.88	0.23	140.13	0.95	0.26
	Mixed Ore	0.23	0.40	0.02	0.25	0.49	0.09	0.48	0.45	0.06
	Stockpile	18.8	0.42	0.27	-	-	-	18.8	0.42	0.27
	Total	**119.6**	**0.89**	**0.27**	**39.83**	**0.88**	**0.23**	**159.4**	**0.88**	**0.26**

Significant Opinions

- ***Golder considers the sample preparation and chemical analysis procedures to be of an appropriate standard for the purpose of mineral reserve estimation. The standard samples show acceptable accuracy and precision.***

- ***For the purposes of an in situ mineral resource estimate, the overall estimation approach adopted by Vale for total copper, gold and density is acceptable.***

- ***The slope regimes for the Siqueirinho and Sossego pits are modeled appropriately during pit optimization and the pit slopes are considered a low risk area for the Mineral Reserves.***

- ***The Sequeirinho open pit will be approximately 500 m deep at completion. This is a very deep open pit excavation and extra care will need to be taken in the mining operations to ensure stability of the final pit walls to allow for full extraction of the reserve.***

- ***The copper and gold prices used for pit optimisation are considered appropriate for the development of a mineral reserve estimates. In particular the values adopted meet generally accepted SEC guidelines which suggest using values that are less or equal the average price for the last 3 years.***

- ***The differences in terms of waste tonnage between the final pit design and the selected Whittle pit shell is considered excessive and should be reviewed in detail. The differences are probably due to a marginal phase incorporated in the mine design. In any case these differences should be properly documented with the appropriate explanations. There may be a considerable upside potential related to mine design optimisation.***

- ***The mining equipment fleet considered in LOM (Life of Mine) plan was reviewed and is considered suitable for purpose. The effectiveness of the mining fleet has been demonstrated over the last couple of years. The mine appears to be adequate areas available for waste dumping and tailings deposition that support the LOM plan and therefore the mineral reserve.***

- ***Reconciliation results for 2009 indicate conformance of planned versus realised production. The reconciliation process is considered to be of high standard. The process plant is clean, well-maintained and employs modern and appropriate process control. In general it gives the impression of a very efficient and well-designed operation. Process control uses modern instrumentation.***

- ***Vale holds all environmental permits required by Brazilian legislation to operate the Sossego mine. No fatal flaws regarding environmental aspects of the Sossego operation have been identified by Golder. The Sossego operation manages environmental responsibilities and liabilities appropriately.***

- ***Conversion of the mineral resource estimate to a mineral reserve is based on appropriate mine design and planning. In particular, dilution and mine recovery are supported by historical data. The tonnes and grades are reported at an appropriate economic cut-off grade. The mine has demonstrated sufficient economic viability to justify the conversion of mineral resources to mineral reserves.***

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FIGURES

8.0 SOSSEGO MINE

8.1 Location

The Sossego Mine Complex is located in the Carajás region in the Southeastern portion of Pará State in Brazil, 42 km west of Canaã do Carajás (Figure 8-1). Access to the Sossego Mine Complex is via bitumen road from the township of Parauapebas. Commercial airports at Carjás and Marabá provide additional logistical support.

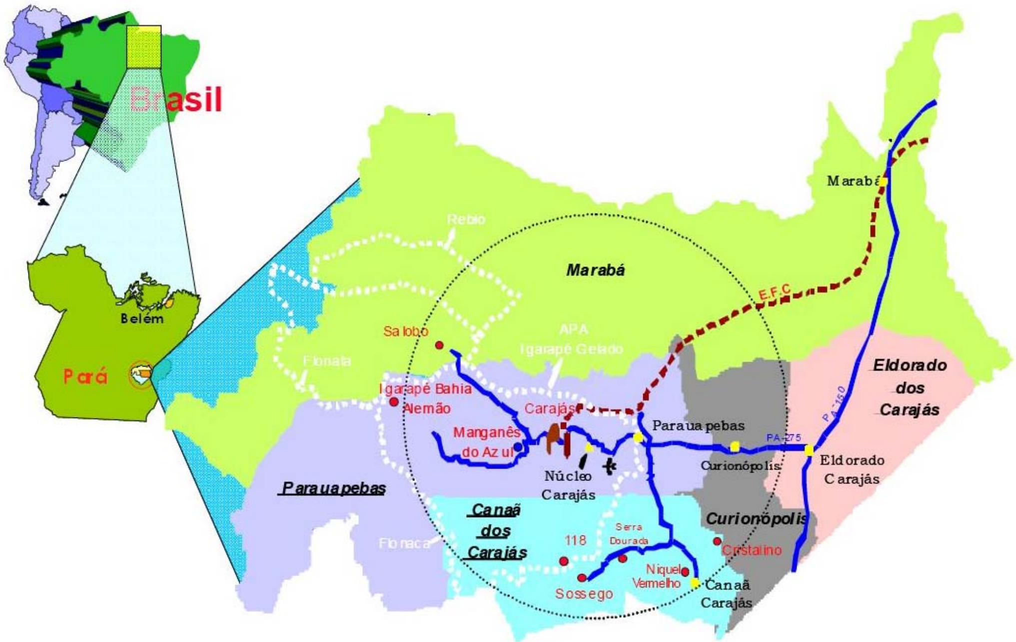


Figure 8-1: Sossego Mine complex location

8.2 Ownership

The Sossego Mine Complex tenement titles are 100% owned by Vale S.A.

8.3 Land Tenure and Mining Rights

The Sossego Mine Complex is located in two claims. The area named Sossego/Sequeirinho (DNPM 851.355/92) refers to Exploration Permit no. 340, dated March 10 1995, defined as an 8 sided polygon of 7,140 hectares. The area named Sequeirinho Sul (DNPM 851.148/92) refers to Exploration Permit no. 1957, dated September 05 1995, being a 4 sided polygon of 1,560 hectares.

Brazilian legislation separates the ownership of the surface from the underground. A mining company can operate a mine even if does not own the surface. In this case it is necessary to pay a royalty to the surface owner. The royalty is calculated as 50% of the CFEM (Compensation for Financial Exploitation of Mineral Resources) which is paid to the government. The mining concessions are updated every year on presentation by Vale of the annual report of mining production to the DNPM (Brazilian Mining Regulatory Agency).

8.4 Infrastructure

The Sossego mine will be supported by the towns of Parauapebas and Canaa dos Carajás, with the majority of employees and contractors coming from Canaa dos Carajás (40 km).

The area is well-served by railroads and highways. Regularly scheduled air service is available in Carajás and Marabá approximately 225 km from Sossego by highway. Most flights connect to the capital, Brasilia. Diesel fuel is tucked onsite and stored on a bulk storage facility.

Copper concentrate is shipped by truck (40 t) from the Sossego Mine to a rail loading facility near the city of Parauapebas, 85 km from the mine. From there it is transported 870 km by rail to the Ponta da Madeira Marine Port in Sao Luis from where the concentrate is shipped.

Waste Dump and Tailings Facility

The mine waste dumps utilize standard mining procedure of bottom up development. The flat topography surrounding the mining area results in large waste dump footprints to maintain a low height of the dump. The overall angle of the waste dump is 29°. This is obtained by dumping on 10 m lifts which have an angle of repose of 37°. A 10 m bench is left between each successive lift. The oxide and sulfide stockpiles are developed in the same manner.

There are four waste dumps at the Sossego complex with capacity of 159 Million m^3 to the South-east, 41 million m^3 to the South-west, 10.2 million m^3 to the North and 22.6 million m^3 to Sossego.

At the Sossego mine there appears to be adequate areas available for waste dumping and tailings deposition that support the LOM plan and therefore the Mineral Reserve.

Mineral Processing

The Sossego processing plant is located about 4 km west southwest of the primary crusher, which is south of the Sequeirinho deposit central axis. The Sossego pit is about 1.5 km east of the primary crusher.

The processing facilities consist of four main components: crushing, grinding, flotation, and concentrate dewatering. Figure 8-2 illustrates a simplified flowsheet of the plant. A comprehensive description of the processing facilities is provided in the technical report “Mineral Resource and Mineral Reserve Estimate for the Sossego Mining Complex” (Vale, 2010).

The design production capacity of the plant commissioned in March 2004 was 15 Mtpa. The production rate as defined in the Feasibility Study was suppose to start at 13.5 Mt in 2004 and gradually ramp up to reach full capacity in 2005. A slower ramp-up was experienced and the plant never reached its original design capacity. Since 2006 a series of improvements have been implemented in the plant and a new design capacity of 13 Mtpa has been commissioned in March 2010.

Power Supply

A power line of 87 km supplies the Sossego site. The power comes from Tocurui, a 3,800 MW generating station on the Tocantins River, 200 km north of Marabá.

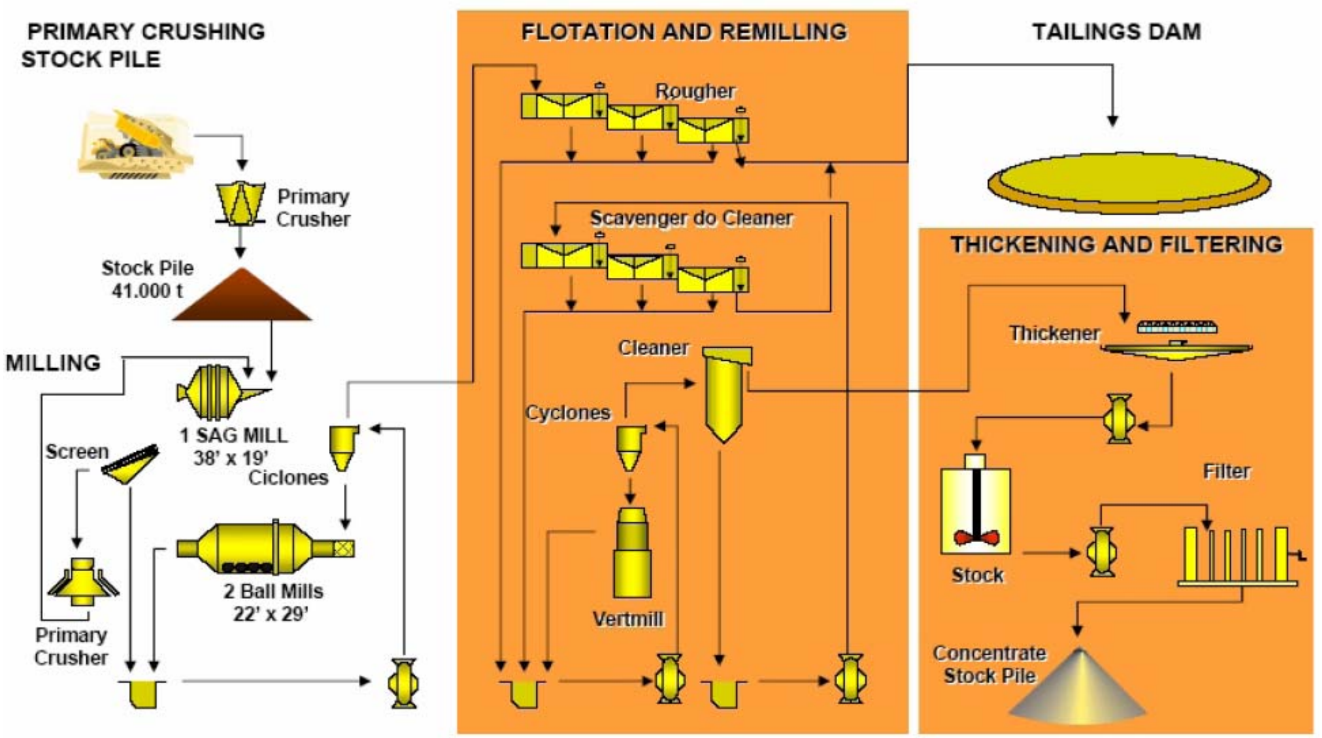


Figure 8-2: Simplified flow sheet of the Sossego plant

8.5 Production Process and Products

Sossego was the first copper project taken into operation by Vale and has been producing copper concentrate since 2004. In 2009 Vale produced approximately 394 kt of copper-gold concentrate with an average grade of 29.7% Copper and 7.5 g/t gold. The Sossego Mining Complex has two open pit mines named Sossego and Sequeirinho.

Standard truck and shovel open pit mining is developed on 16 m benches. Contract mining is used for pre-stripping and also for mining at the contact between oxide and sulfide materials. After drilling and blasting cable shovels and front end loaders are used for excavation and loading. The material is hauled mostly by 240 t trucks either to the waste dumps near the pit or to the primary crusher. Low grade ore (Cueq < 0.5%) is stockpiled near the crusher and north of the pit for later use. The material is crushed and transported by conveyor for 4 km to the concentrator. The ROM production in 2009 was approximately 15 Mt at 0.98% Cu which required the removal of 47 Mt of waste material.

The plant has nominal capacity to process 13 Mtpa and consists of primary crushing, milling, flotation and filtration. In 2009 the plant processed 12,659,763 dmt. The concentrate is transported by conventional trucks to Parauapebas using a well paved road (85 km) and from there by train to the São Luis seaport (870 km). The concentrate is shipped with 9% moisture.

Figure 8-3: Aerial view of the Siqueirinho Pit

Figure 8-4: Aerial view of the Sossego pit

8.6 Metal Recoveries

The metallurgical recoveries currently used in LOM plan and to estimate mineral reserves, consider an average recovery of approximately 89% from 2011 to 2021. Recovery drops to 86.7% in the last year which is consistent with current estimates for the processing of low grade stock piles. Figure 8-5 shows the actual recoveries achieved in 2008 and 2009 and the projected values used in the mine plan. The drop in recovery that can be seen in the graph is mostly related to a drop in grades.

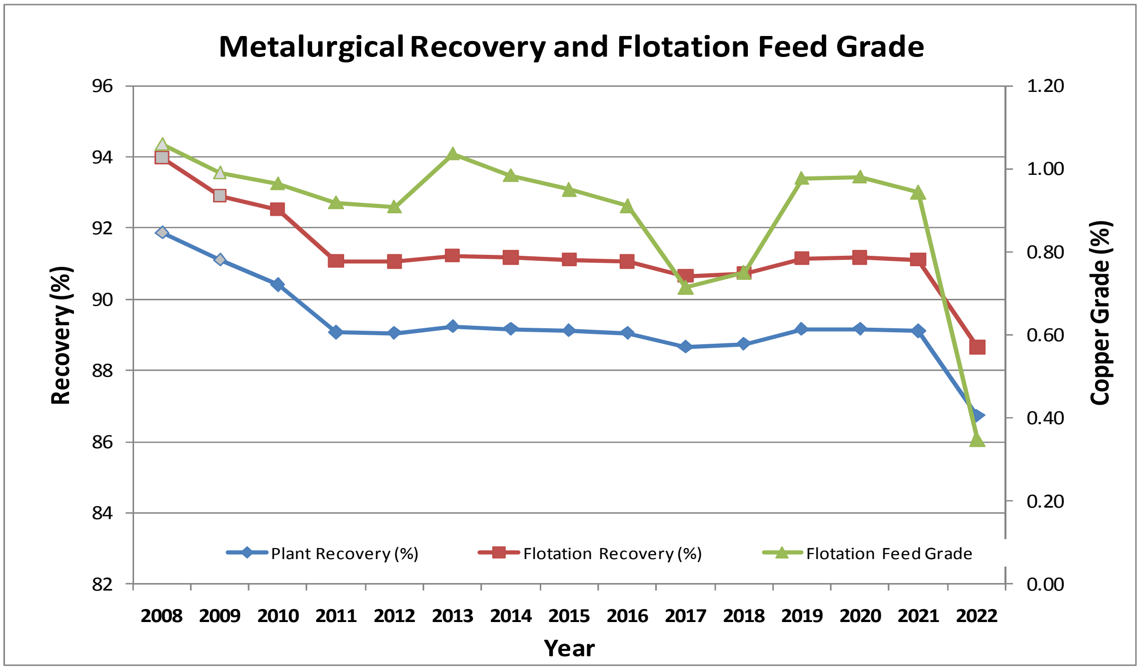


Figure 8-5: Historical and projected metallurgical recoveries

8.7 Market

Vale currently has a portfolio of about 10 direct clients (smelters) and two traders for the concentrate produced by the Sossego operation. The contracts that are in place will last from 5 to 10 years and there is no reason to believe that most of these contracts are not going to be renewed. All current forecasts of copper demand indicate that there will be a demand for the Sossego concentrate over the entire life of the mine.

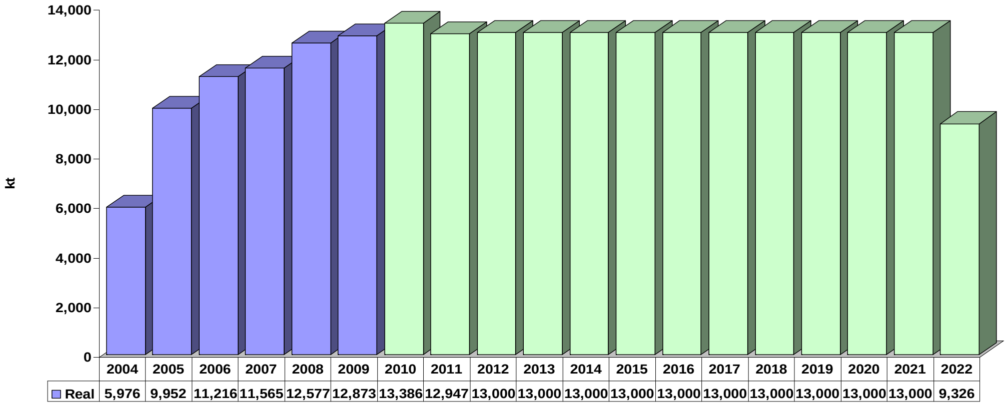
8.8 Historic Production

Pre-stripping started at the Sequeirinho Pit in 2002, due mostly to requirements of rock material to raise the tailing dam and other earthworks. The processing plant started production in March 2004 and in 2009 reached its highest production capacity to date. Figure 8-6 and Figure 8-7 show the historical production and planned production figures for the LOM.

In 2008 and 2009 the plant processed 12,348,506 dmt and 12,659,763 dmt respectively. The current LOM plan provides for the plant to process 13 Mtpa from 2012 to 2021. Figure 8-8 shows the historical production of copper concentrate from 2004 to 2009.

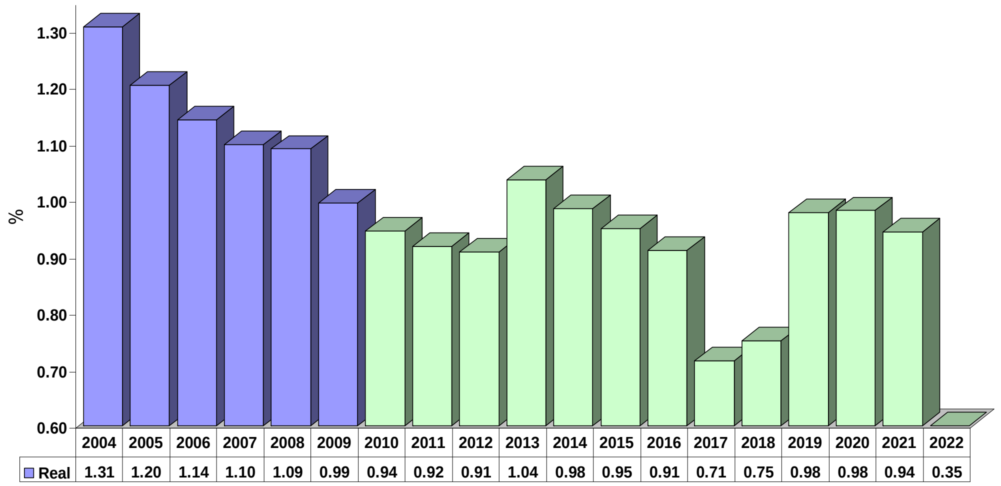
As part of the audit Golder personnel inspected the plant and reviewed the historical production information since start-up. The Sossego processing plant uses established technology which is well understood and documented. Based on production reporting, all processes appear to be working as designed and costs are reasonable. Production data is properly documented and reconciliations between the resource model and production data (grade control, plant) are periodically carried out.

The process plant is clean, appears to be well-maintained and employs modern and appropriate process control. In general it gives the impression of a very efficient and well-designed operation. Process control uses modern instrumentation. The plant is competently managed and Vale has the know-how that is required to operate it.



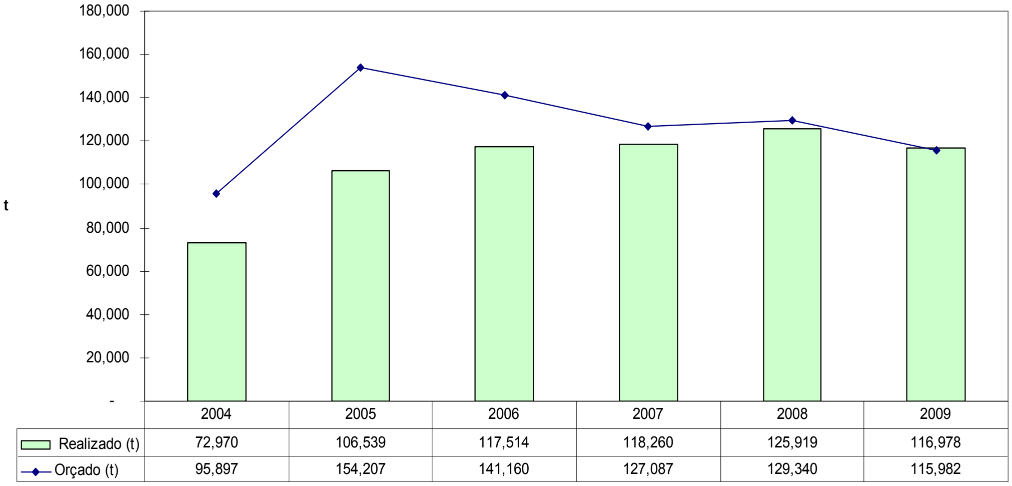
	2004	2005	2006	2007	2008	2009	2010	2011	2012	2013	2014	2015	2016	2017	2018	2019	2020	2021	2022
▨ Real	5,976	9,952	11,216	11,565	12,577	12,873	13,386	12,947	13,000	13,000	13,000	13,000	13,000	13,000	13,000	13,000	13,000	13,000	9,326

Figure 8-6: Historical and projected production – Ore tonnage (dmt)



	2004	2005	2006	2007	2008	2009	2010	2011	2012	2013	2014	2015	2016	2017	2018	2019	2020	2021	2022
▨ Real	1.31	1.20	1.14	1.10	1.09	0.99	0.94	0.92	0.91	1.04	0.98	0.95	0.91	0.71	0.75	0.98	0.98	0.94	0.35

Figure 8-7: Historical and projected production - Copper grades



	2004	2005	2006	2007	2008	2009
Realizado (t)	72,970	106,539	117,514	118,260	125,919	116,978
Orçado (t)	95,897	154,207	141,160	127,087	129,340	115,982

Figure 8-8: Historical production of copper concentrate

8.9 Geology and Mineral Deposits

Regional Geology

The Carajás Mining District is located in the south eastern portion of the Pará State, Brazil, and corresponds to a late Archean basin deformed into a sigmoid shape, trending west northwest - east southeast. This shape is defined by several major west-northwest – east-southeast lineaments such as the Carajás and Cigano left lateral fault zones. Also some northeast and northwest fault systems crosscut the region.

The Carajás volcano-sedimentary sequence is composed mainly of bimodal volcanics, chemical sediments, including the banded iron formations (BIF) that host the Carajás iron deposits, pyroclastic and clastic sediments. The basin is filled with a metasedimentary and metavolcanic sequence named the Itacaiubas Supergroup and granites and schists that form the Xingu Complex occur to the south.

The Carajás units are intruded by Archean intrusives, including the calc-alkaline Plaquê Suite (2.77 Billion years old), and the alkaline Sossego and Estrela granites (2.5 billion years old). These units have a strong correlation with copper–gold mineralization in Carajás. There are many generations of mafic bodies, including some post-mineral dykes. A Proterozoic Suite (1.88 Billion years old) of alkaline granites, in the central Carajás granite (Cigano and Pojuca), also intrude the Carajás Sequence. Several generations of young mafic dykes crosscut the entire sequence.

Local Geology

The Sossego deposit is associated with a 10 km regional structural shear zone striking northeast to southeast and dipping steeply to the south. A series of copper gold deposits are associated with this structure including Sequerinho, Sossego, Jatobá, Serra Dourada and Cristalino deposits (Figure 8-9).

The Sossego deposit has been divided into 4 main zones, from west to east areas: Pista, Sequerinho, Baiano and Sossego Hill of which, the Sequeirinho deposit is the most important in terms of size.

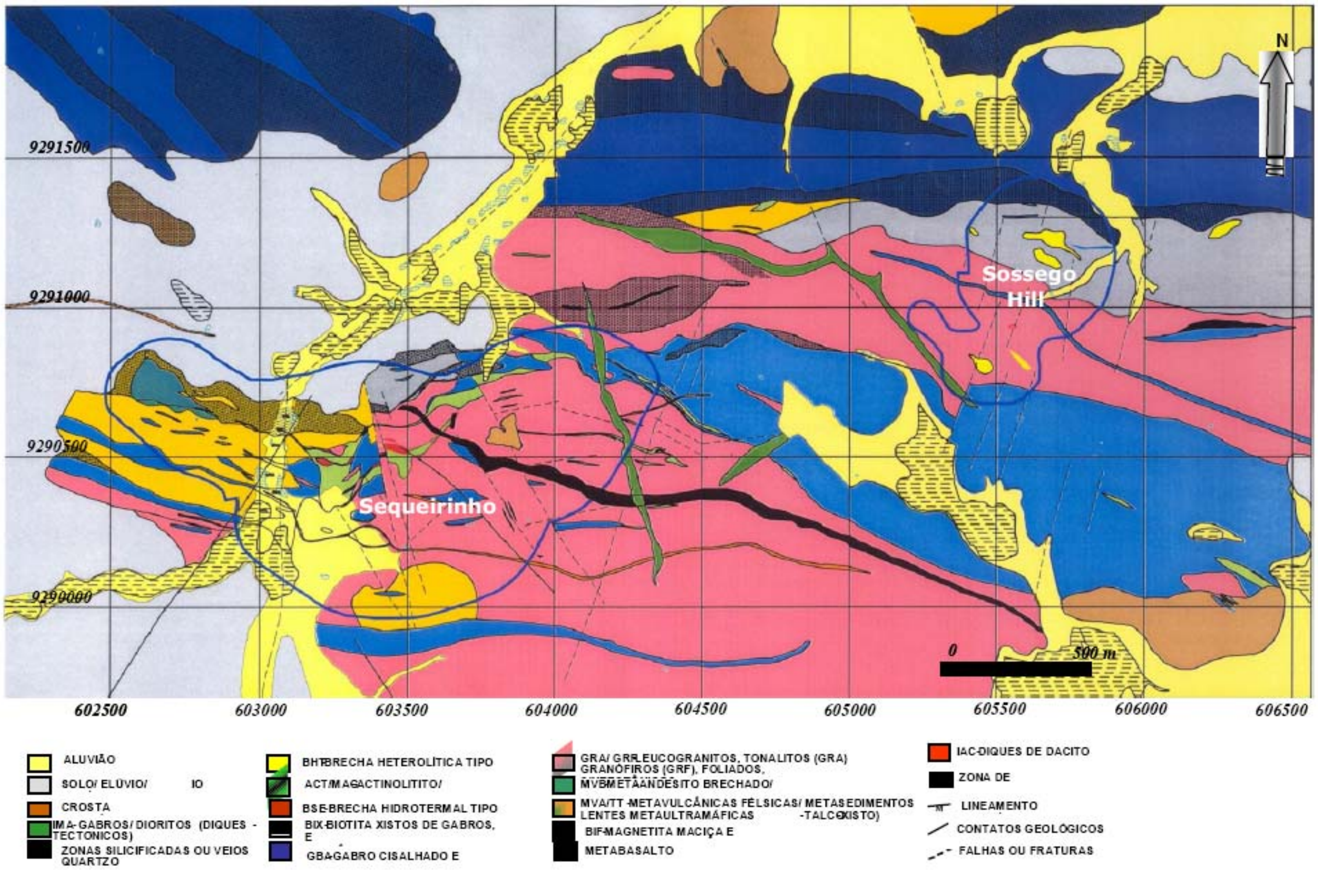


Figure 8-9: Local geology of Sossego mine, from Vale 2010 report

Sequeirinho

Sequerinho consists of a single and continuous zone of mineralization with a strike length of approximately 2000 m with weaker mineralization extending another 1000 m to the west. This deposit is situated about 700 m west-southwest of Sossego Hill and out crops as a prominent hill with approximately 40 m of relief over the general landscape.

The sulphide deposit has a sigmoidal shape, trending west-northwest on the extremities of the deposit, and northeast in the central portion. The western extremity is referred to as the Pista sector and the eastern portion is called the Baiano sector. Mineralization generally occurs as a series of sub-parallel bodies with aggregate thicknesses ranging from 25 m to nearly 300 m. Dips are generally steeper on the extremities, ranging from 70°S to sub-vertical. In the central portion of the deposit dips are locally as shallow as 45° to the southeast. The deposit is cut by a north-northwest trending diabase dike at the eastern transition of the Sequeirinho and Baiano portions of the deposit.

Mineralization at Sequeirinho is structurally controlled and cuts the felsic volcanic, granite and gabbro host rock units. The majority of the hanging wall is composed of a granitic/tonalitic rock with dikes or remnants of mafic rocks. The footwall is defined by a sharp contact that separates the mineralized zone from the altered biotite schists of the Carajás volcanic sequence (locally felsic volcanics). The footwall boundary can be roughly followed by a topographic break to the north of Sequeirinho and is characterized by an abrupt drop in the chalcopyrite-actinolite-magnetite content and an increase in the scapolite-biotite content.

The high-grade sulphide mineralization occurs in breccias that are usually richer in copper and gold near the footwall and hanging wall contacts of the ore body as well as in crosscutting zones. The breccias have a chalcopyrite matrix and clasts of magnetite, amphibole and other lithic fragments. The breccias zones are surrounded by a lower grade envelope consisting of irregularly distributed disseminated chalcopyrite mineralization. Copper and gold are well correlated. Near surface oxidation has produced pseudo-malachite and malachite.

Sossego Hill

The Sossego Hill deposit is part of a broad circular feature, about 100 m in diameter and standing about 10 m in elevation. The core of the circular structure is a medium to coarse-grained pinkish to dark gray granite cut locally by gabbro dikes. At the contact with the Carajás felsic volcanic rocks, the granite has a fine grained texture (granophyric texture) forming a granophyre with up to 200 m in width characterized by quartz-potash feldspar intergrown with blue quartz eyes. This granophyre usually has a transitional contact with the granite, indicating that the fine-grained texture is related to a border facies. This entire set is cross cut by two main chalcopyrite-rich breccia pipes, namely Sossego Hill and Curral.

Mineralization occurs in three zones; the roughly circular, 200 m diameter Sossego breccia in the northern part of the mineralized area; the elongated 80 m by 400 m Curral breccia to the south, and the Vein Zone, an irregularly veined zone between the Curral and Sossego breccias. The breccias at Sossego Hill and Curral are heterolithic, matrix supported with fragments of granitoids, mineralized fragments, granophyre, and gabbro in a hydrothermally altered matrix composed of potassium feldspar, calcite and varying proportions of chlorite, magnetite, apatite and chalcopyrite. The heterolithic breccias also occur within irregular hydraulic fractures of variable width (from a few centimetres to nearly 20 m) that often branch and terminate as nearly pure chalcopyrite veins. Stockwork and disseminated mineralization is also present. The contacts between high-grade mineralization and barren material are abrupt. Chalcopyrite is the main copper bearing mineral.

8.10 Exploration and Development Drilling

Golder was provided with the recovery data related to drill holes D-278 to D-410. Core recoveries average 98%. Drill holes are predominantly 47.6 mm (NQ) diameter and in some cases 36.5 mm (BQ) diameter in the deeper intervals. Due to the geological characteristic of the deposit the drill holes of Sequerinho are orientated to the north with a dip of 50° to 60°, while in Sossego the holes are oriented the the NNE with a dip of 50° to 60° (Figure 8-10).

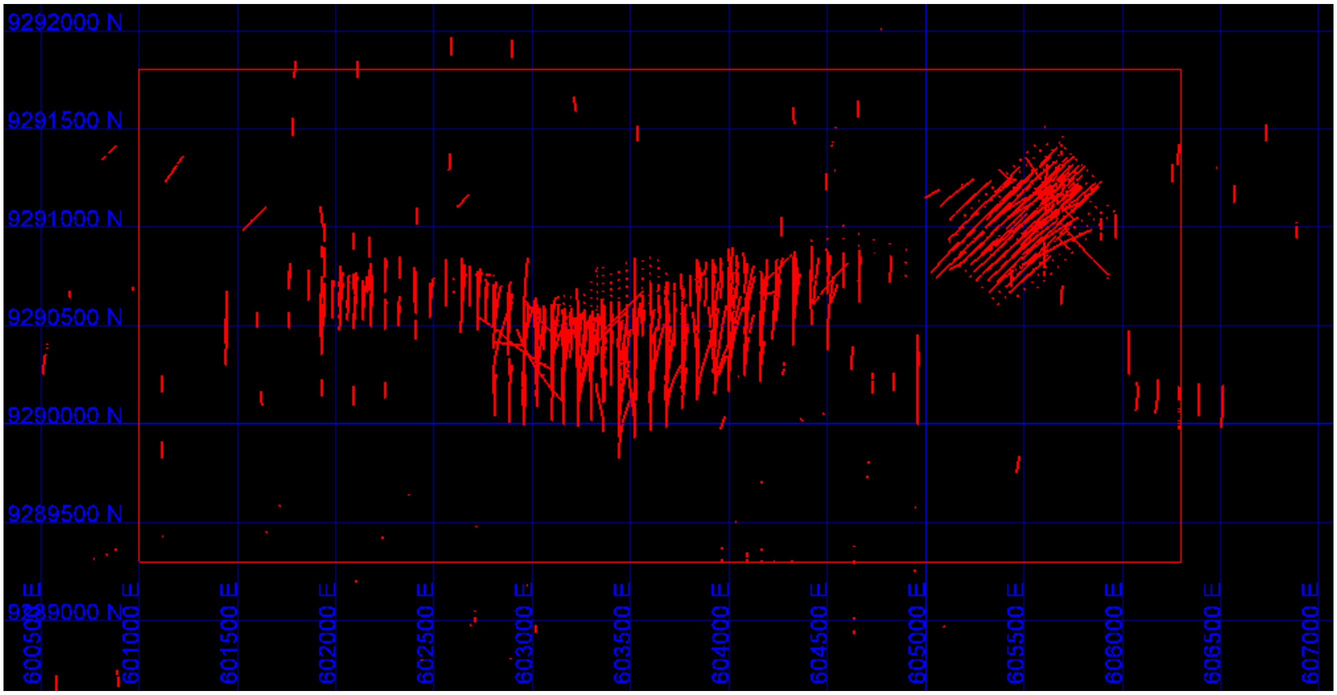


Figure 8-10: Distribution of drill holes for Sossego deposit

Table 8-1 summarizes the holes drilled at the Sossego deposit listing diamond and reverse circulation drill holes data separately.

Golder visited the DDH rig that was drilling the FD-440 drill hole to observe the current drilling practices. In general the practices were good and no important issues were observed (Figure 8-11A and Figure 8-11B).

Table 8-1: Summary of Sossego drill holes

Hole						Period	
From	**To**	**Type**	**Number**	**Sample Number**	**Length**	**From**	**To**
SOSD-001	SOSD-071	DDH	66	24,219.00	24,751.05	1997	1998
SOSD-072	SOSD-150	DDH	72	22,025.00	31,628.92	1998	1999
SOSD-151	SOSD-320	DDH	172	44,742.00	65,011.57	1999	2000
SOSD-321	SOSD-420	DDH	100	15,990.00	21,497.52	2000	2003
SOXD-001	SOXD-020	DDH	20	640.00	624.42	2002	2003
		Total	**430**	**107,616.00**	**143,513.48**		
SOSD-421	SOSD-427	DDH	7	1,699.00	1,834.60	2005	
SOSP-01	SOSP-70	RC	24	1,761.00	3,521.00	2001	2002
SOXP-001	SOXP-505	RC	399	13,307.00	13,106.00	2002	2003
	Grand Total		860	124,383	161,975.08		

Figure 8-11: A) Core piled up on rig platform waiting to be transported. B) Equipment use for core sampling

Topography

Collar coordinates were obtained using total station survey equipment, with a validation procedure that uses a visual review against topography.

The topographic survey method used at Sossego is considered appropriate.

Down hole Survey

Three different methods for down hole survey were used:

- DDI-Fotobor; measures were taken at 3 m intervals.
- Maxibor; measures were taken at 3 m intervals. The procedure takes measurements down and up the hole. Both measures are averaged to obtain the final value.
- Gyroscope; measures taken at 30 m intervals. The procedure takes measurements down and up the hole. Both measures are averaged to obtain the final value.

The QAQC criteria for down hole surveys considered a maximum of 2.0% of deviation of the total length between the expected and final coordinates at the end of the hole.

According to the Sossego report and due to operational difficulties, several holes do not have reliable down-hole surveys: SOSD-32 to SOSD-34, 37, SOSD-40 to SOSD-42, SOSD-45 to SOSD--48, SOSD-50 and SOSD-51. Deviations for these holes were estimated using averaged deviations from surrounding holes. The hole SOSD-166 was used for modeling but not for kriging due a high horizontal angle deviation.

Table 8-2: Summary of Sossego down hole survey measure method

	Drill holes			
Method	**Form**	**To**	**Total N°**	**Total %**
DDI	SOSD-001	SOSD-025	25	13%
DDI	SOSD-027	SOSD-028	2	
DDI	SOSD-072	SOSD-098	26	
GYROSCOPE	SOSD-099	SOSD-420	321	77%
MAXIBOR	SOSD-026		1	10%
MAXIBOR	SOSD-029	SOSD-071	42	

All the RC drill holes only have theoretical survey values as no down the hole surveys measurements were undertaken. Golder does not consider this to be a critical issue because these drill holes are mainly within the oxide zone and are not long.

The down hole surveying methodology is considered appropriate and no anomalies were observed in the procedures.

8.11 Deposit Sampling Methods and Data Management

Diamond drill hole core (DDH) makes up the majority sample type for geological modelling and mineral reserve estimation at Sossego. Blast holes (BH) are drilled but used only for short term planning.

Golder reviewed the sampling procedures for drilling used in the 2010 model. The sample lengths used at Sossego is generally of 1 m in mineralized zones, 4m in saprolite and 4 m for barren zones. The length of samples is modified according to the ore type, sector and/or weathering.

Logging

Logging is carried out by Vale geologists using internal log sheets for drill core logging. The geologist records the majority code for lithology, alteration, mineralization, and textural characteristic of the interval every 1 m, with 10 cm as the definition unit. Geological contacts are logged with higher precision. The logging is done on site in spacious and well lit facilities (Figure 8-12A).

An excellent geological wall display, showing examples of all the lithologies, alterations and mineralisation was available on site for geologist to consult during logging activities (Figure 8-12B).

Figure 8-12: A) Logging facilities on site. B) Geological wall display available for logging

Sampling

The DDH core is collected and placed in wooden boxes. The core is then transferred to the core shed where geological and geotechnical logging is carried out. Before logging, pictures are taken of all cores. After logging, the core is split in half using an electric saw. The position where the core is to be split is marked by Vale geologists.

DDH cores are sampled at 1m intervals in mineralized zones and 4m intervals for barren zones. The sample length is modified according to ore type, sector and/or weathering conditions. One half of the core is sent to the analytical laboratory for chemical analysis and the remaining half is stored on site.

Logging procedures are industry standard and Golder considers them as appropriate for this type of deposits.

Sample Storage

Facilities for drill core storage consist of a couple of warehouses special for this purpose and are located at the project site. Trays with cores rejects are stored in wooden trays and labelled with metallic plates. At the time of the Golder site visit, the core shed was being reconstructed due to damage caused by a storm. The work is expected to be complete by mid 2010 (Figure 8-13).

Figure 8-13: A) General view of core shed. B) Boxes with core stored

Pulps are stored in paper envelopes grouped in plastic bags, while the coarse rejects are stored in plastic bags. Both are organized in properly identifies boxes. All rejects are stored inside a purpose built warehouse. Figure 8-14 shows the conditions of store and the facilities available for drill core and rejects. The warehouse is well organized.

Figure 8-14: A) General view of pulp and coarse reject storage area. B) Plastic bag containing pulp rejects

The drill core and samples storage facilities are acceptable.

Sample Preparation and Analysis

The mechanical sample preparation and analysis of DDH core is conducted on site by a Vale laboratory. The sampling procedure is summarised as follows (Figure 8-15).

- Cutting: drill cores are cut in half using a diamantine saw, packed, labelled and sent for mechanical preparation. Samples are labelled with a bar code.
- Drying: samples are dried in an electric oven at 105°C.
- Primary crushing: the entire sample is crushed with a jaw crusher to approximately < 6.35 mm particle size.
- Secondary crushing: the entire sample is crushed with a Roll crusher to < 4 mm particle size.
- Splitting: the sample is passed through a Jones riffle splitter to obtain one quarter of the sample.
- Pulverization: the sample is pulverised until 95% of the sample passes < 0.105 mm.
- Sample is homogenized.
- Splitting: one sample of 250 g is collected with the rotary splitter. The remaining material is stored as pulp reject.
- The sample is packed in a paper envelope, labelled with a bar code, and sent to the chemical laboratory for assay.

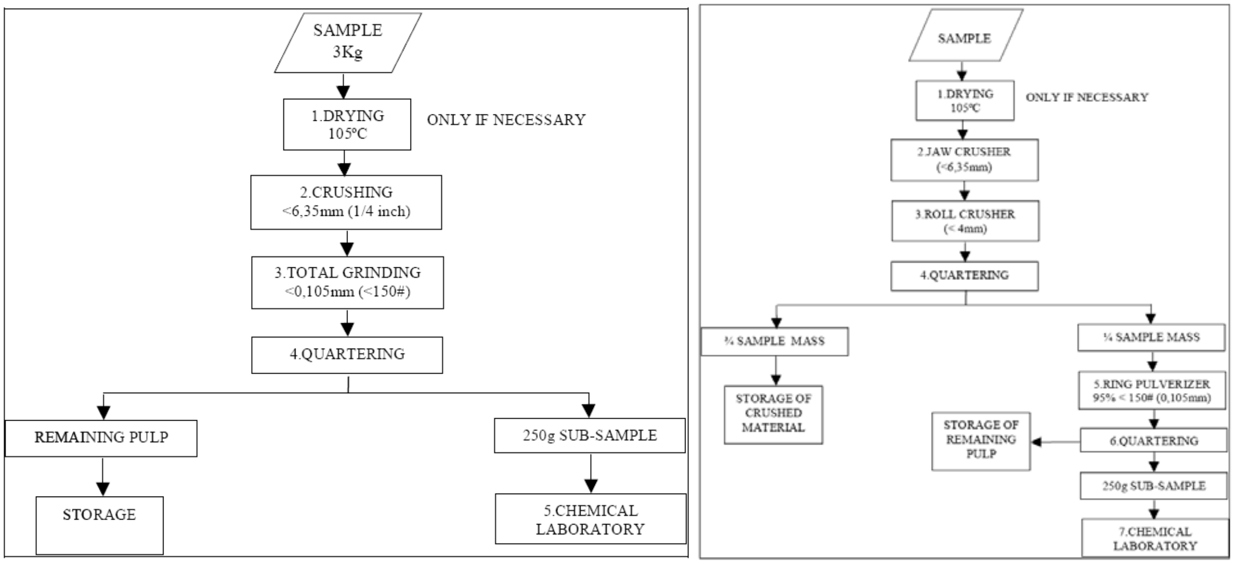


Figure 8-15: Mechanical sample preparation schema A) used by Nomos laboratory for drill holes SOSD-001 to SOSD-025. B) Schema used by Nomos laboratory for drill holes SOSD-21 to SOSD-420. From Vale internal (2010).

The chain of custody at the sample preparation facility is properly established. All steps from the reception of the sample to the transfer of the prepared sample to the chemical laboratory are registered using bar code readers.

Golder reviewed the laboratory used for mechanical sample preparation (Figure 8-16) and found them to be of a good standard.

Routine chemical analysis is by Atomic Absorption for copper, silver and fluorite. Gold is analysed by fire assay methods. According to Vale personnel, the laboratories used for sample analysis has changed over time. Table 8-3 summarizes the laboratories used for chemical analysis.

Table 8-3: Summary of laboratories used for chemical analysis

Drill hole		
From	**TO**	**Laboratory**
SOSD-001	SOSD-006	Nomos, Rio de Janeiro
SOSD-07A	SOSD-077	Nomos, Parauapebas
SOSD-078	SOSD-320	CVRD, Belo Horizonte (GAMIK)
SOSD-321	SOSD-420	Lakefield Geosol, Belo Horizonte

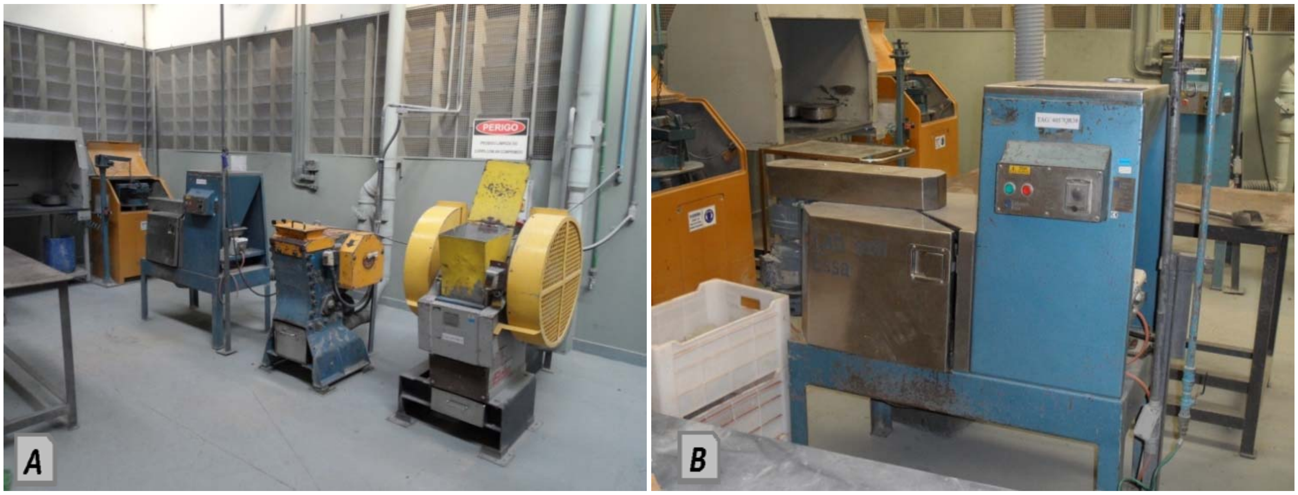


Figure 8-16: A) General view of laboratory mechanical sample preparation equipment. B) Rotary splitter

Golder considers the sample preparation and chemical analysis procedures to be of an appropriate standard for the purpose of mineral reserve estimation.

Quality Assurance and Quality Control (QAQC)

Quality Assurance (QA) is the system and set of procedures used to ensure that the sampling and assaying results are of high quality. Quality Control (QC) is the data used to prove that the results of sample preparation and chemical analysis are adequate.

Depending on the drilling campaign to be assay Lakefield, Bondar Clegg or GAMIK were considered as secondary laboratories.

Golder was provided with the QC data from Sossego for the drill holes SOSD-01 to SOSD-420. For this audit Golder carried out an independent review of the QA procedures and the results from the analysis of the QC data for both copper and gold assays

Procedures

According to the documentation provided by Sossego the QAQC program requires the insertion of one blank, and one standard for every 43 routine samples (one batch). Usually duplicates were sent to a secondary laboratory in a special batch at the end of the drilling campaign.

Golder was provided with two reports. The first one is “Analytical Quality Control for Project Sossego a Consolidated Report April/2001” and is related to the 2000 QAQC program. The second is "Relatório Técnico de Atividade Controle de Qualidade das Análises Químicas Campanha de Sondagem 2002/2003” that analyze in detail the results for the 2002-2003 drilling campaign. In this report is detailed the actions taken for anomalous samples.

On 2000 Sossego developed a study orientated to the validation of the analytic data exists at that moment. This study divided the data between Fase I (SOSD-01 to SOSD-77, laboratory NOMOS) and Fase II (SOSD-78 to SOSD410, laboratory GAMIK). The main results for this study indicate that for Fase I Total Copper (CuT) results of the primary laboratory (NOMOS) are 8% high, while for Au is acceptable. For Fase II for CuT < 4.55% the primary laboratory (GAMIK) is acceptable, while data CuT >4.55% is precise but slightly biased. Au is acceptable. All the check developed considered the use of pulp duplicates, and no test to review the sampling procedure was made.

Duplicates were not made simultaneously with the routine analysis. The duplicate considered for stage I 392 samples analyzed in Gamik as secondary laboratory. For stage II the QA for the 2002-2003 drill campaign considered two sets: 5% of samples Gamik were sent to a secondary laboratory, and 5% of samples >0.2% CuT were sent to 3 laboratories, GAMIK, Lakefield and Bondar Clegg. These samples were prepared from pulps rejects.

The implementation of a non on line QA procedure is not a good practice, QA must be done on line with the routine analysis. An adequate QA must consider the insertion by the client of pulp and coarse duplicates, field duplicates, blanks and standards. The control samples should by unidentifiable by the laboratory.

Standards

Standard samples composed of materials of “known” grade are used to validate the accuracy of other assay results when included in a batch of samples submitted to the laboratory for analysis. The standard samples are also known as Certified Reference Materials (CRMs).

Two sets of standards were used, in both cases considered material from Sossego. As shown in Table 8-4, nine standards have been used to assess the accuracy of data for Sossego. The standard deviation value is used as acceptance criteria. The standards expected values were obtained from assays carried out at Bondar Clegg, Gamik and Lakefield-Geosol laboratories. A total of 10 samples per standard were analysed by each laboratory. The results obtained were detailed in Vale (2003).

For the construction of the standards the use of the same laboratories considered for the routine analysis is not a good practice. It is recommended the use of different laboratories than the considered in the routine analysis.

Table 8-4: Copper and gold standard used at Sossego

Material	Standard	Cu (%)		Au (ppm)	
		Mean	Std. Error	Mean	Std. Error
Sulphides	P1G	4.600	0.1100	1.200	0.1400
	P2G	0.890	0.0059	0.300	0.0330
	P3G	0.420	0.0240	0.100	0.0072
	P4G	0.069	0.0005	0.020	0.0028
	P5G	2.700	0.0700	1.000	0.1300
Oxides	P11G	3.300	0.1100	0.900	0.0610
	P12G	1.370	0.0140	0.600	0.0850
	P13G	0.415	0.0015	0.100	0.0110
	P14G	0.135	0.0021	0.030	0.0032

The analysis of standards was carried out in conjunction with all drilling.

The precision is measured in terms of averaged HARD values. As shown in Table 8-5, average HARD values for copper are all below 10%, indicating an acceptable range of precision. This can also be observed in the cumulative HARD graphs, where, for all standards, 90% of the samples have HARD values below 10% indicating an acceptable precision.

Results for gold are shown in Table 8-6. HARD values are typically greater than 10% and some greater than 20%, indicating a precision for this element worse than for copper. The cumulative HARD plot for gold shows poor precision, although this results may be expected due to its mineralizing pattern (nugget or fines), and therefore the levels of precision observed may be acceptable.

Accuracy is measured in terms of averaged HRD values. The results are summarized in Table 8-5 and Table 8-6 for copper and gold respectively. They indicate a slight positive and negative bias for copper. Averaged HRD values fluctuate between -5% to 3%. Copper standard P1G for drill holes D51-100, and P2G for drill holes D101-278 shows a consistent positive bias whereby the samples are mostly above the 0% HRD line.

For gold, the results indicate a slight to evident bias. The averaged HRD values fluctuate between -26 to 66%. Gold standard P1G for drill holes D51-100, P2G for drill holes D51-100, and P2G for drill holes D279-420, show a consistent negative bias indicating the existence of underestimation of the gold values. On the other hand, standard P4G for drill holes D101-278, P4G for drill holes D279-420 and P14G for drill holes D101-278 indicate a consistent positive bias.

Table 8-5: Summary of HARD and HRD values obtained from standards analysis for copper

Standard	N° Samples	Avg_HARD (%)	Avg_HRD (%)	Bias (%)	Drill holes
P1G	16	2.66	2.57	5.39	D51-100
	123	3.05	-0.62	-0.28	D101-278
	20	4.96	-3.38	-5.33	D279-420
P2G	19	1.74	0.41	0.95	D51-100
	228	3.85	-0.35	2.22	D101-278
	18	7.27	-4.01	-3.56	D279-420
P3G	206	3.87	-0.54	1.42	D101-278
	18	4.59	-3.44	-5.42	D279-420
P4G	383	6.68	-0.45	18.44	D101-278
	118	7.43	1.72	67.04	D279-420
P5G	98	5.70	-2.37	-1.07	D101-278
	13	6.90	-4.70	-5.58	D279-420
P11G	11	3.68	-1.48	-2.29	D101-278
	2	1.47	1.47	3.03	D279-420
P12G	13	1.97	0.52	1.18	D101-278
P13G	19	12.19	-3.93	6.79	D101-278
P14G	45	9.32	-1.05	66.58	D101-278

Table 8-6: Summary of HARD and HRD values obtained from standard analysis for gold

Standard	N° Samples	Avg_HARD (%)	Avg_HRD (%)	Bias (%)	Drill holes
P1G	15	15.75	-15.75	-26.50	D51-100
	122	17.99	1.12	16.11	D101-278
	16	14.12	-3.33	2.52	D279-420
P2G	225	16.56	-3.16	9.19	D101-278
	18	27.56	-16.79	-11.39	D279-420
P3G	12	17.87	-5.91	-3.33	D51-100
	192	13.60	10.49	34.79	D101-278
	15	12.36	-5.98	-8.07	D279-420
P4G	32	66.45	66.45	776.56	D101-278
	27	23.07	15.22	81.48	D279-420

Standard	N° Samples	Avg_HARD (%)	Avg_HRD (%)	Bias (%)	Drill holes
P5G	97	17.10	5.23	26.10	D101-278
	10	14.49	-8.37	-11.95	D279-420
P11G	11	16.54	-5.49	-3.64	D101-278
	2	26.75	-26.75	-40.44	D279-420
P12G	12	26.55	0.15	0.09	D101-278
P13G	5	17.79	10.73	54.00	D51-100
	16	9.75	-2.20	-1.25	D101-278
P14G	11	51.12	38.40	696.97	D101-278

In some cases the lack of data makes the statistical analysis non representative. According to the QAQC database, 1350 standard samples were tested for copper representing approximately 1.15% of the sample population while for gold, 838 samples were assayed representing approximately 1.5% of sample population.

The standard samples show acceptable accuracy and precision. Some minor biases were identified, but these are not expected to materially impact on the quality and representativity of the data to support mineral resources.

Blanks

Blanks samples are materials with an expected grade of zero, and are used to detect contamination from sample preparation equipment, laboratory hardware, or reagents.

Golder was provided with the results for the blanks analysis for copper and gold for drill holes SOSD-101 to SOSD-420. The analysis indicates the existence of anomalous high values. This could have resulted from sample or laboratory contamination or the sample is not an appropriate material to be used as a blank. According to the analysis made by Sossego some of these anomalous values are related to mismatch situations.

According to Vale geologist when anomalous values of blanks occurs a re-analysis of the samples and standards was done.

Duplicates

A series of duplicates analysis has been carried out for copper and gold. They include internal duplicates assayed at the same laboratory and external duplicates assayed at secondary laboratory. The following analysis is related to external duplicates sent to a secondary laboratory. The following analysis is related to external duplicates.

During 2004 a set of 5% of copper samples assayed by Gamik (primary laboratory) were sent to Lakefield (secondary and external laboratory) in order to detect bias between those laboratories. These samples were prepared by Gamik from the 250 g aliquot prepared for analysis. The results show an acceptable precision and repeatability with averaged HARD values of approximately about 7%. No obvious bias was detected with average HARD value lower than1%.

In general, pulp duplicates for copper indicates acceptable precision showing averaged HARD values lower than 10%. No evident bias was detected during duplicates analysis. In case of gold, marginal to poor precision is observed, where averaged HARD values are above 10%. Slight negative bias was detected with averaged HARD values between -2 to -7 percent.

Table 8-7: Summary of duplicates analysis for copper and gold at Sossego

Element	N° Samples	Avg_HARD (%)	Avg_HRD (%)	Avg_Bias	Precision (at 83.4%)
Cu	731	7.09	-0.67	0.01%	25.4
Cu	560	1.14	-0.13	0%	4.6
Cu	1141	9.43	-1.72	0.01%	33.4
Au	282	10.22	-2.37	-0.01 ppm	30.7
Au	842	32.33	-6.78	0.01 ppm	83.4

The results for precision obtained for copper are acceptable while those obtained for gold are expected due to its mineralization pattern (nuggets or fines).

Database

Sossego uses the SQL database administrative platform for the centralisation of all the geological information. The data base has an appropriate security program, including backups and limited access, to secure the quality of data in case computational problems occur. One person is responsible for all data input and management of the database.

On July 22 2010, Golder was provided with the following files, which, according to Sossego personnel, contain the data used for the construction of the 2010 geological modelling:

- SSDATA.CSV – Cooper composited data;
- HEADER_AUDIT.csv – BHID data;
- SURVEY_AUDIT.csv – Survey data;
- ASSAYS_AUDIT.csv – Assays data;
- LITHOLOGY_AUDIT.csv – Geological data;
- MODELO_16_SOS.txt and MODELO_8_SOS.txt– Block model file;
- Vert_Min_Seq_Sos_2010.3DR – Vertical sections with interpreted lithology;
- Horz_Min_Seq_Sos_2010.3DR - Horizontal sections with interpreted lithology;
- 2008 topography surface;
- 2008 lithology wireframes;
- DENSITY_AUDIT.csv – Density samples.

All checks and analyses carried out on the database, geological models and grade estimation were based on this information. Table 8-8 summarises the information from the tables contained in the drill hole database.

Table 8-8: Summary of drill hole database

	Entire Data Base	
Table	**Drill holes**	**Length (m)**
Collar	1,216	194,695.90
Survey	1,172	153,033.59
Litho	1,049	192,865.15
Assays	924	134,797.00

All the information related to drill holes, such as analytical certificates, surveys, etc, was transferred to digital files. Paper copies are also available on site.

Database Validation

To ensure the drill hole data in all databases are coherent, Golder reviews and checks for the internal integrity of the database. The collar, survey, assay and geology tables were imported and processed with Golder´s Datacheck© software. The analysis detected several inconsistence in the drill holes related to hole IDs with prefix PX-**. The results of the review detected:

- 8 drill holes with span or missing intervals in the assay table;
- 23 drill holes where length on ASSAY table exceed the collar length;
- 292 drill holes with no information in the assays table;
- 4 drill holes with span or missing intervals in the lithology table;
- 167 drill holes with no information in the lithology table;
- 5 drill holes with large survey angle variation (>10°variation between two consecutive measurements);
- 4 drill holes with large survey azimuth variation (>10°variation between two consecutive measurements);
- 6 down hole survey measurements in 155 drill holes that are taken at intervals over the recommended 50 m;
- 3 drill holes have excessive dip deviations (> 10°variation between two consecutive measurements).

Density

Density samples for Sossego mine include approximately 60 000 samples collected across the entire deposit as shown in Figure 8-17. Measurements are taken in mineralized and barren units. Due to differences in porosity and permeability, differentiation is recorded between samples taken in saprolite and those collected in bedrock. Specific gravity in the saprolite zone is determined on competent core using a wax-coat, immersion technique.

The procedures for testing density are documented in respective procedure Vale (2010).

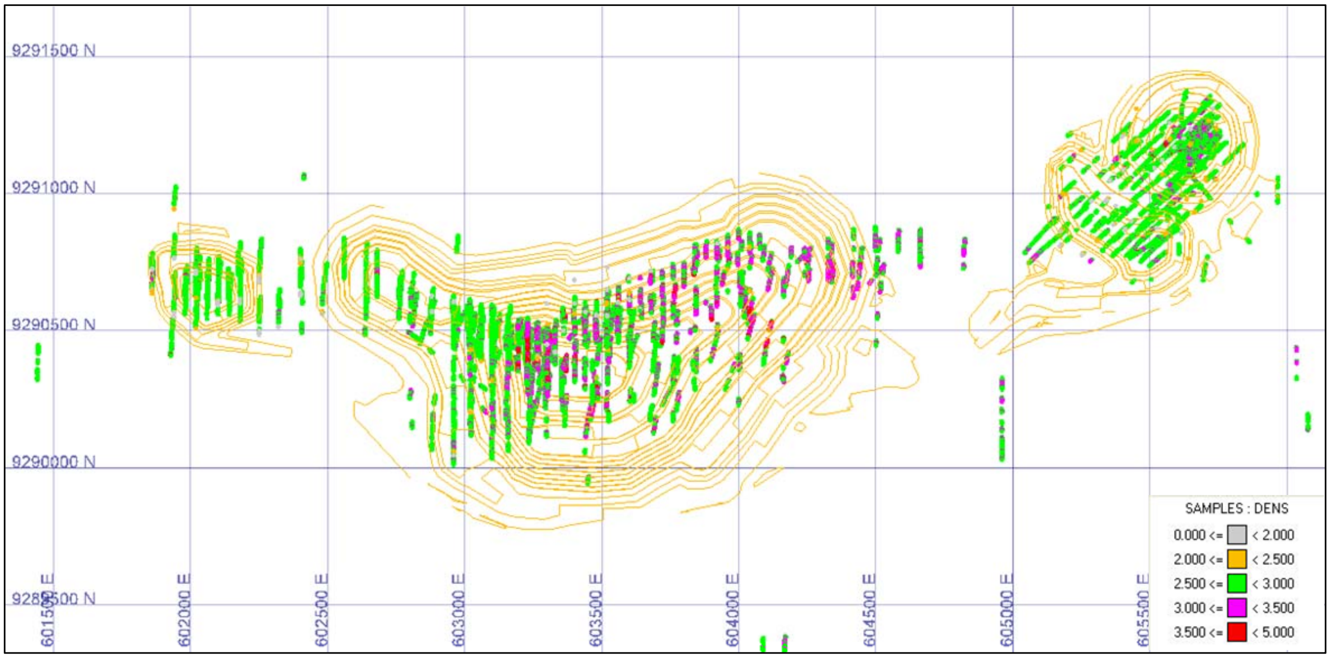


Figure 8-17: Location of density samples in Sossego mine

The methodology used to determine density of bedrock initially considered split samples, measuring the dry weight, and the weight immersed in water.

The samples are not dried prior to the procedure, but because of the lack of porosity and permeability, drying the sample will not significantly change the dry weight of the sample. Vale tested this at the Alemaõ operation and found that the maximum difference in weight for samples dried at 105°C for 14 hours was 0.15 percent, an insignificant difference.

Basic statistics on density were calculated for each lithology type, results are shown in Table 8-9. The results show clear differences in mean value between weathered rock (INT=2.38 g/cm^3) and fresh rock (INT greater than 2.64 g/cm^3).

Table 8-9: Basic statistics for density to each lithology type

Lithology	Code	No. Obs.	Minimum	Maximum	Mean	Std. Dev.	Variance	Coeff Var
none	-1	2	1.89	2.71	2.63	0.241	0.058	0.091
INT	0	1,876	1.15	3.82	2.38	0.460	0.212	0.193
GRA	1	12,845	1.71	4.40	2.76	0.139	0.019	0.050
GRF	2	6,450	1.92	4.11	2.79	0.132	0.018	0.047
BIX	3	3,904	2.28	4.28	2.81	0.110	0.012	0.039
GBA	4	8,157	2.00	4.44	2.97	0.140	0.020	0.047
ACT	5	7,598	1.65	4.43	3.09	0.254	0.065	0.082
BHT	6	2,844	2.15	4.20	2.95	0.174	0.030	0.059
BSO	7	382	2.47	4.13	3.24	0.281	0.079	0.087
BSE	8	1,951	2.05	4.36	3.28	0.256	0.066	0.078
TON	9	209	2.43	3.21	2.83	0.131	0.017	0.046
MVA	10	12,976	1.81	4.08	2.73	0.094	0.009	0.034
TTX	12	361	2.11	3.45	2.88	0.123	0.015	0.043
IMA	16	98	2.73	3.50	2.95	0.132	0.017	0.045
IAC	17	169	2.20	2.95	2.71	0.081	0.007	0.030
MAG	21	654	2.46	4.49	3.61	0.302	0.091	0.084
ZCS	22	5	2.62	2.68	2.64	0.021	0.000	0.008
CLX	23	3	2.62	2.74	2.68	0.049	0.002	0.018
	Total	60,484	1.15	4.49	2.86	0.250	0.062	0.087

Golder considers that the methodology applied for the determination of density is appropriate and that the results obtained and the amount of data complies with the standards of the mining industry to make an appropriate characterization of the density in the deposit.

Geological Modelling

The geological model consists of two main models, lithology and mineralisation (or ore grade shells). Both models were constructed as follows:

- Interpretation was undertaken on vertical NS and EW sections, based on a 2m composited data base with the majority lithology code, 8m was used as the minimum width to be considered.
- A set of solids were constructed based on the interpretation polygons. These solids were cut on 8m horizontal sections, coinciding with the top and bottom bench height.
- The polygons were revised and modified according to information considered as “final polygons”, extruded to 8m solids.
- These solids were used to generate a block model of 10m by 10m by 8m block size. In the case that one block contains two or more codes precedence was used to define the code. Gemcom was used as software for geological model construction.

The lithology units considered in the model is detailed in Table 8-10.

Table 8-10: Codes of units modeled in the lithology model

Abrv	Code	Lithology
GRA	1	Granite
GRF	2	Granofire
BIX	3	Biotite Schist
GBA	4	Gabbro
ACT	5	Actinolitite
Breccia	6	Breccia
MVA	10	Acid Volcanic
TTX	12	Talc Tremolite Schist
IMA	16	Mafic Intrusive
IAC	17	Acid Intrusive
MAG	21	Magnetite
SIL	24	Silicification

The mineralisation model is based on the lithology model and the knowledge of the geology. Two zones were interpreted to link geology with CuT grade distribution. The sulphur breccia with actinolite alteration event is represented with an envelope of 1% CuT, while the alteration halo corresponds to a disseminated or stockwork mineralization with 0.2% CuT. Anything outside the envelopes is not estimated and CuT grade is 0.

The block model was divided into 4 sectors based on geological features such as style of mineralisation (Figure 8-18). A surface representing the boundary between saprolite and sulphide was also interpreted.

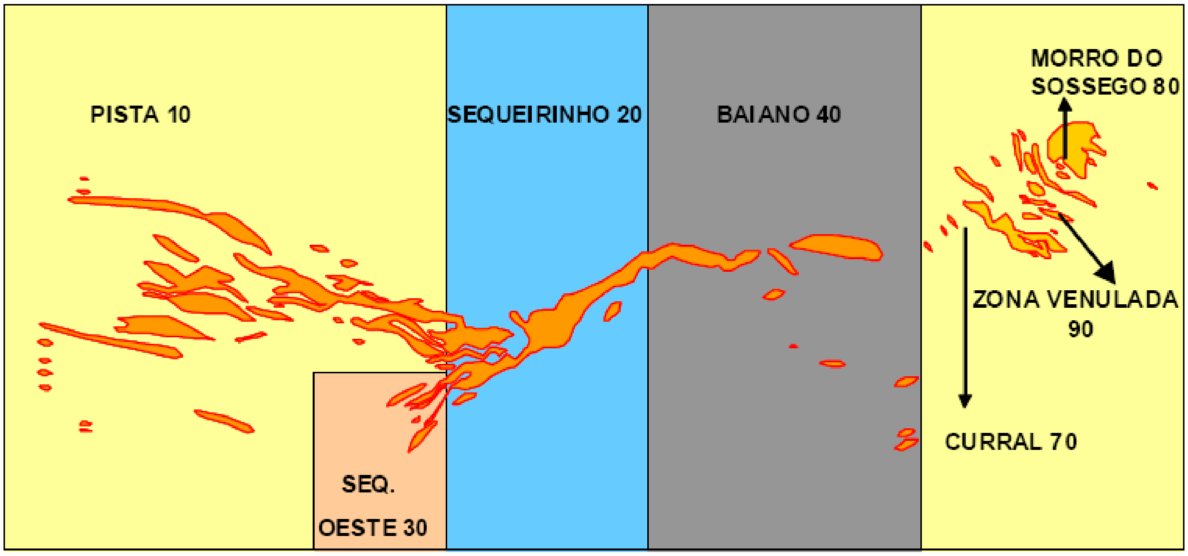


Figure 8-18: Identification for Sossego block model sectors

Visual Review

Golder visually reviewed the models using Vulcan, checking section by section, to evaluate how well the model honours the drill hole data and to check its geological coherence. The geological models were also evaluated against each other to check for inconsistencies. In addition, the geological codes assigned to the block model were compared to the wireframes solids.

The mineralized model shows reasonably good correlation between drill hole information and the block model. Figure 8-19 shows a typical cross section of the mineralisation model.

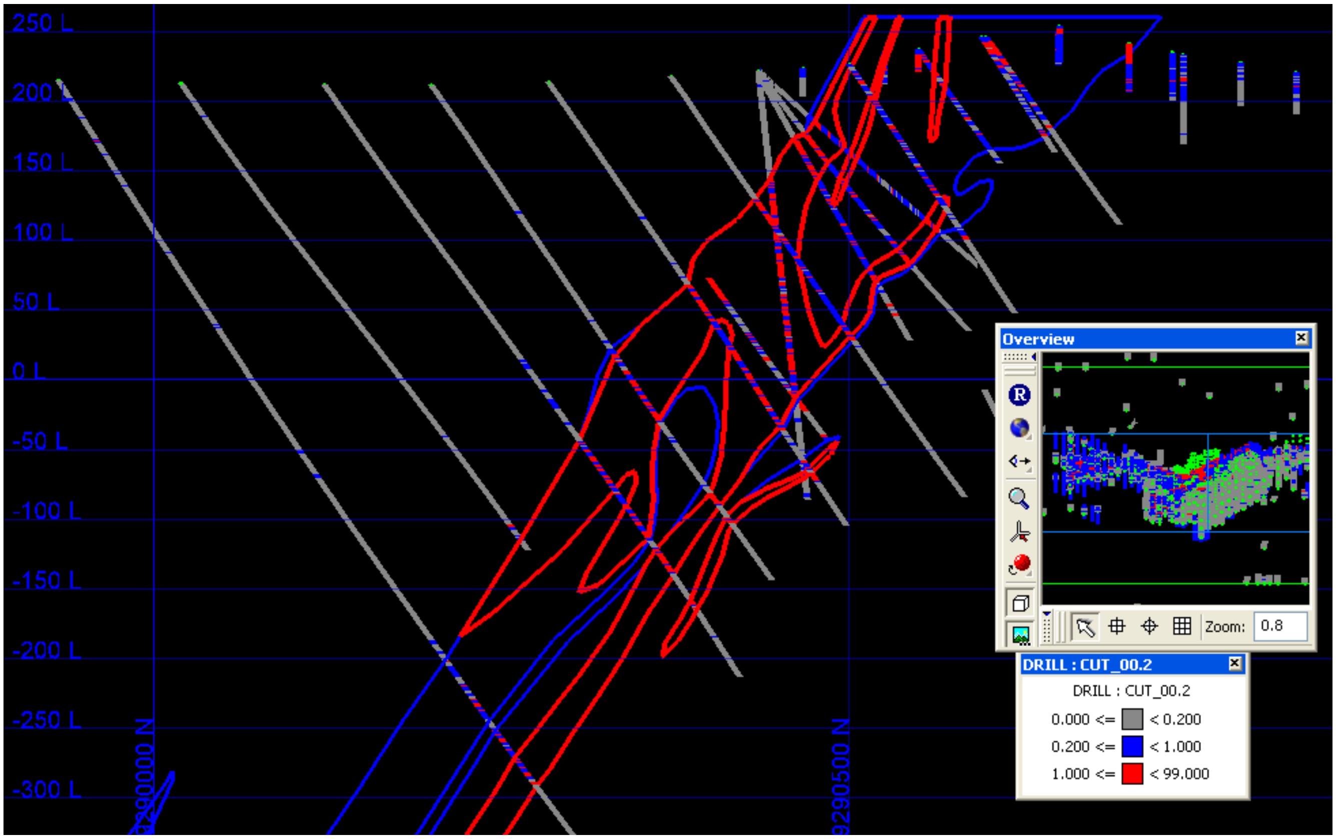


Figure 8-19: Mineralisation model, section 603.520 E

When both models are compared, the relation between the BX-ACT unit with the >1% CuT envelope is clear.

A visual review of the geological solids and block model detected local inconsistencies mainly related to geological continuity of interpreted solids.

An interpreted body of breccia mineralisation exists in the south of the model that does not fit with any of the lithologies that supposedly control the distribution of the mineralisation.

Golder reviewed the surface representing the boundary between fresh and oxide rock and detected the existence of inconsistencies. Some drill holes are not considered in the interpretation and, in some cases, the exact point were the change occurs is not honoured (Figure 8-20).

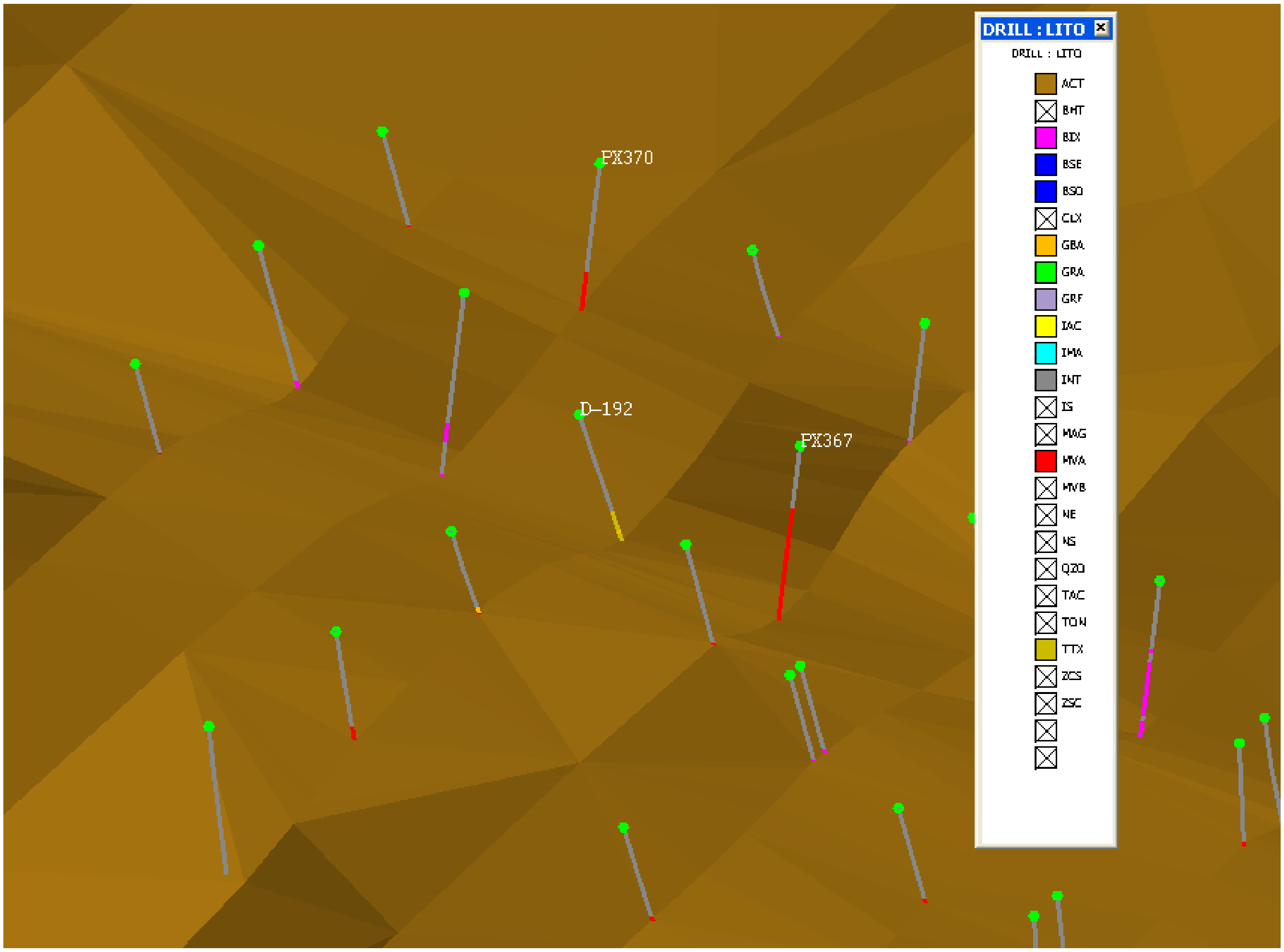


Figure 8-20: Surface that represents the limit fresh - non fresh rock

Back-flagging Analysis

Statistical analysis was carried out to compare the correlation between the original data (drill hole logs) and the interpreted data (block model/geological model). In order to validate the geological models, drill hole composites were back-flagged with the block model. Sossego provided Golder with geological information in the block model/geological model with 10x10x8 m block size and 2 m run length composited data.

The results show that the majority of the modelled units have good correspondence, with values above 80%, and in some cases above 90%. The units with lower values, like TTX or IAC, have little representation in the data.

Unit 1006 (breccias) shows values around 70% and because the importance it has in the grade distribution, some efforts can be made to improve that value. It must also be considered that the majority of the intervals not correctly modelled correspond to isolated intervals that are difficult to interpret with geological continuity.

In the geological model, the ACT lithology is commonly interpreted as part of the BX and it is reflected in the number of composites of ACT modeled as BX. The criteria used for this assumption is not clear.

Back-flagging analysis of the lithology model between the drill holes and block model shows no obvious anomalies and acceptable consistency exists between the drill hole data and the block model.

8.12 Mineral Resource Estimation

Documentation Review

The mineral resource estimation procedure at Sossego is described in the report “Mineral Resource and Mineral Reserve Estimate – 2010 Technical Report” (file *Geo_SOS_2010_AUDITORIA.pdf*, dated June 17 2010) provided by Vale. Additionally, scripts, parameter files and output files from the different stages of the process were provided as documentation of the resource estimation process.

The documentation provided is sufficient to reproduce the mineral resource estimate. However, explanations and justifications for the decisions involved in the Sossego mineral resource estimation process for all variables should be properly documented in the report (the gold and density estimation processes are not thoroughly detailed in the report).

Estimation Databases

Grade estimation at Sossego was carried out by Vale in Gemcom® for total copper (Cu), gold (Au) and density (De). The drill holes used to construct the Sossego geological and resource model (provided by Vale in the Microsoft Access® database file *GD_AVAL_SOS_2010_REV2.mdb*) were composited to 2 m using the Run Length tool. Composites were flagged against the block model for lithology, alteration and mineral zones.

The 2 m composites database used to build the Sossego resource model (provided by Vale in file *EXPORT_COMP_2.txt*) was imported to Vulcan® to file *sossossego.com.isis*. The database was used in the complete resource estimation process, including domain definition, variography and grade estimation.

The composites database used in the estimation of the resource model (constructed from a subset of the drill hole database) is composed of 931 drill holes with a total length of 180443 m. Of the total length, there are 176596 m (98%) with valid total copper grades, 176395 m (98%) with valid gold grades and 61234 m (34%) of valid density measurements.

Sample Sources

Diamond drill hole (DDH) and reverse circulation (RC) composites were used to build the resource model. Combining samples from different drilling methods for grade estimation purposes is not recommended, unless this has been clearly validated by appropriate checks, since this practice may introduce bias into the estimation. The proportion of RC composites compared to DDH composites is approximately 16%.

A comparison of the sample database with the composite database shows a difference in the number of drill holes present. There are less drill holes in the composites database than the original sample database, indicating a selection of drill holes were used. However, the selection criteria for the drill holes that are actually used in the estimation process could not be found in the Vale documentation or scripts.

A comparison of the grade distributions between the two sample sources was carried out to evaluate the differences. The comparison shows that, in general, total copper grades are similar between the two sample sources for values below 1.5% Cu (representing more than 90% of the distribution). For values above 1.5% Cu the correlation becomes erratic. First RC composites show higher grades than DDH composites for grades below 2.5% Cu and then the relationship reverses. Considering that the correlation between the different sample sources for the upper 10% of the distribution is not consistently biased, and that for 90% of the distribution the grades do not show differences, combining DDH and RC data for total copper estimation is acceptable.

The distribution comparison for gold shows that, in general, copper grades of RC samples are higher than the values obtained for DDH samples. The inclusion of RC samples could add a positive bias to the gold estimates.

Compositing

The drill hole database, which contains assays every 1 m, was composited in Gemcom® to 2 m using the Run Length tool. The compositing process considered breaks in the presence of non-assayed intervals or in ore/waste contacts.

Length Analysis

No length restrictions were applied to composites in the estimation process. Golder carried out a complete analysis of length versus grade distribution, to ensure that no measurable bias is being introduced by including samples with lengths that varied significantly from the composite length.

The compositing procedure produces length values that range from 0.01 m to 2 m. It is common practice to reject the use of composites shorter than 50% of the composite size; hence, the analysis was performed breaking the composite database with a length threshold of 1 m.

As seen in Table 8-11 the mean total copper and gold grades for composites less than 1 m long are considerably lower than the mean grades for composites with lengths between 1 m and 2 m. A smaller difference can be observed for density.

Table 8-11: Composite basic statistics by length set

Variable	Set	No. Obs.	Minimum	Maximum	Mean	Stand. Dev.	Variance	Coeff Var
Cu (%)	Length>=1	88,386	0.005	50.050	0.362	0.938	0.880	2.592
	Length<1	202	0.005	3.070	0.086	0.305	0.093	3.550
Au (ppm)	Length>=1	88,281	0.005	19.080	0.094	0.358	0.128	3.809
	Length<1	198	0.005	1.450	0.025	0.123	0.015	4.874
Density (g/cm^3)	Length>=1	30,622	1.240	4.490	2.855	0.250	0.062	0.088
	Length<1	28	2.610	3.750	2.838	0.241	0.058	0.085

In light of the statistical analysis results, the length distributions were compared for the three variables. Overlain cumulative probability plots were produced for total copper, gold and density respectively.

In line with the statistical analysis results, the copper and gold distributions show that grades are consistently lower for lengths below 1 m, suggesting the need for length restrictions in the estimation process due to the potential negative bias introduced. Even though the differences between the density distributions and statistics are not significant, it would be adequate and consistent to apply the same length restrictions as for grade variables. Implementing this restriction would mean that less than 0.25% of the total available data would be discarded for estimation purposes.

A way to control this problem is incorporating sample length as a weighting variable during the ordinary kriging process. In other words, multiplying the kriging weight by the sample length, then normalise the modified weights to sum to one. The drawback of this methodology is that re-normalisation can significantly re-distribute the kriging weights, departing from those originally devised from the kriging matrix which reflects the variogram.

Exploratory Data Analysis

Exploratory data analysis aims to find similarities in the distributions between the different sample populations and determine possible groupings of geological attributes into estimation domains.

The statistical appropriateness of the domain definitions was reviewed using different statistical and geostatistical tools. To perform the analysis, basic statistics, scatter plots diagram of mean and standard deviation and cumulative probability plots were produced to evaluate the definition of estimation domains adopted for resource estimation at Sossego. All the statistical analyses were performed using the composite database after the corresponding restrictions were applied as in the estimation process.

Estimation Domains

Estimation domains for the three variables were based on the units defined for copper grade interpolation. They are based on a combination of structural information, lithology, alteration and mineralization.

The structural data was used to divide the deposit into sectors, which represent similar mineralization orientations and styles. Table 8-12 shows the structural zones defined for Sossego, which are shown in plan view in Figure 8-18.

Table 8-12: Sector definition - structural trends

Sector	Sector code	Azimuth	Dip
Pista	10	110°	75°
Sequeirinho SW	20	60°	55°
Sequeirinho	30	60°	55°
Baiano	40	85°	70°
Curral	70	125°	90°
Morro do Sossego	80	140°	0°
Vein zone	90	140°	90°

The structural sectors are combined with mineralization, alteration and lithology, to define two major subsectors: a low grade disseminated zone and a high grade breccia zone. Green and red oxides, subdivided into saprolite and transition zones were also modelled; however, they are not part of the mineral resources / ore reserves and hence were not audited. The final sulphide estimation domains are shown in Table 8-13. In practice, the disseminated zones for the Pista, Sequeirinho and Baiano sectors are treated as a single estimation domain for all variables.

Table 8-13: Estimation domain definition

Sector	Mineralization	Code	Sample codes used		
			Cu	Au	De
Pista	Disseminated	15	15+25+35+45		
	Breccia	16	16+26+36+46	16+26+36	16
Sequeirinho SW	Disseminated	25	15+25+35+45		
	Breccia	26	16+26+36+46	16+26+36	26+36
Sequeirinho	Disseminated	35	15+25+35+45		
	Breccia	36	16+26+36+46	16+26+36	26+36
Baiano	Disseminated	45	15+25+35+45		
	Breccia	46	16+26+36+46	46	46
Curral	Disseminated	75	75	75	75
	Breccia	76	76	76	76
Morro do Sossego	Disseminated	85	85	85	85
	Breccia	86	86	86	86
Vein zone	Disseminated	95	95	95	95

Additionally for gold grade estimation, indicator kriging was performed to define waste blocks. If the resulting proportion of waste for breccia or disseminated material within a block was greater than 41%, the block was defined as waste and a gold grade of 0.005 ppm was assigned (kriging of grades was not performed).

Total Copper

Table 8-14 summarizes the total copper basic statistics obtained for each estimation domain. Composites were weighted by length. In general the estimation domain definition is consistent with the statistical features. Disseminated domains for the Pista, Sequeirinho and Baiano sectors are adequately grouped, and most of the independent domains prove to be statistically independent.

The statistical analysis suggests that domains 36 and 46 could be considered as independent domains. However, the cumulative probability plots indicate that the distributions for domains 16, 26 and 36 are very similar. The distribution for domain 46 shows lower grades than the rest of the group, but considering the low number of composites in the unit, the grouping is considered adequate.

The statistical analysis also suggests the grouping of domains 75 – 85 and 76 – 86. Even though the cumulative probability plot analysis indicates that the breccia and disseminated domains have very similar distributions, the spatial location of the two units indicates that domain 95 (vein zone) lies between the domains, which implies a change in the geological setting and structural trends for the domains. Also, the orientation of the mineralization continuity is different between domains. Due to geological and structural considerations, these domains are considered adequately defined as independent estimation units.

The definition of estimation domains for total copper is considered adequate for Sossego.

Table 8-14: Summary statistics for copper composites (weighted by length)

Domain	No. Obs.	Minimum	Maximum	Mean (Cu %)	Stand. Dev.	Variance	Coeff Var
15	4,619	0.005	5.870	0.421	0.446	0.199	1.061
16	1,113	0.010	13.530	1.990	1.572	2.472	0.790
25	4,959	0.005	9.010	0.510	0.527	0.278	1.033
26	3,368	0.020	14.740	2.029	1.621	2.629	0.799
35	186	0.005	2.550	0.489	0.342	0.117	0.699
36	64	0.080	7.000	1.815	1.282	1.642	0.706
45	998	0.005	6.340	0.450	0.506	0.256	1.124
46	256	0.040	13.810	1.632	1.674	2.804	1.026
75	787	0.005	12.440	0.833	1.448	2.096	1.738
76	195	0.030	10.080	1.418	1.571	2.470	1.108
85	1,954	0.005	19.800	0.844	1.645	2.706	1.948
86	1,660	0.009	21.430	1.720	2.548	6.490	1.482
95	1,032	0.005	15.950	0.994	1.771	3.137	1.781

Gold

The statistical analysis (summarized in Table 8-15) shows a good correlation between the disseminated domains with low gold grades and the breccias with high gold grades. The Pista, Sequeirinho and Baiano disseminated sectors show a group of very low mean grades; however, the grouping of codes 15, 25, 35 and 45 is not found to be adequate.

The mean vs. standard deviation plot along with the distribution analysis shows that the potential groupings for disseminated domains would be 15 – 45 and 25 – 35. Considering the spatial location of domain 45, along with the low amount of composites available, it would be adequate to group it with 25 and 35, but unit 15 should be considered as an independent estimation domain.

Also, the Morro do Sossego domains 75, 85 and 95 do not seem to be independent units for gold grades. Their statistic features and distributions suggest that they should be grouped for estimation purposes.

Breccia domain 46 is estimated independently, which is considered adequate since it presents the lowest mean value and is spatially apart from domain 16. Grouping domains 16, 26 and 36 is considered adequate as well as considering domains 76 and 86 as independent.

The definition of estimation domains for gold grades is generally considered adequate for Sossego. Domain 15 could be estimated independently and domains 75, 85 and 95 could be merged into a single estimation unit.

Table 8-15: Summary statistics for gold composites (weighted by length)

Domain	No. Obs.	Minimum	Maximum	Mean (Au ppm)	Stand. Dev.	Variance	Coeff Var
15	4,613	0.005	4.960	0.091	0.167	0.028	1.843
16	1,089	0.005	9.110	0.497	0.814	0.662	1.636
25	4,945	0.005	4.470	0.143	0.237	0.056	1.664
26	3,364	0.005	9.770	0.586	0.810	0.657	1.383
35	186	0.005	1.750	0.133	0.205	0.042	1.533
36	64	0.005	7.330	0.538	1.118	1.251	2.079
45	998	0.005	1.800	0.111	0.163	0.027	1.467
46	256	0.005	13.630	0.489	1.098	1.206	2.245
75	787	0.005	7.190	0.224	0.545	0.297	2.437
76	195	0.005	9.280	0.537	1.220	1.490	2.271
85	1,955	0.005	5.690	0.230	0.528	0.279	2.297
86	1,660	0.005	8.200	0.558	0.919	0.844	1.647
95	1,032	0.005	11.160	0.239	0.665	0.442	2.776

Density

The domains defined for copper do not seem to control the density grades. As seen in the summary statistics included in Table 8-16 and the mean vs. standard deviation plot, the used groupings and independent estimation domains do not seem be statistical consistent.

Analyzing the statistics along with the density distributions Golder concludes the following:

- The disseminated units for Pista, Sequeirinho and Baiano should not be estimated as a single domain. Considering the low amount of composites available for domain 35 and the location of the domain, it could be grouped with unit 15.
- Breccia domains 26 and 36 should not be grouped. Domains 16 – 36 and 26 – 46 could be grouped respectively to allow for a larger number of samples to be available for the estimation of domains with a low number of composites.
- For the Morro do Sossego sector, domains 76 – 86 and 75 – 95 could be grouped respectively for estimation purposes.

The definition of estimation domains for density should be reviewed.

Table 8-16: Summary statistics for density composites (weighted by length)

Domain	No. Obs.	Minimum	Maximum	Mean (Au ppm)	Stand. Dev.	Variance	Coeff Var
15	3,413	2.030	3.660	2.759	0.116	0.014	0.042
16	1,007	2.180	4.030	2.923	0.275	0.075	0.094
25	3,979	2.270	4.490	2.958	0.241	0.058	0.081
26	2,700	2.600	4.290	3.124	0.248	0.062	0.079
35	141	2.590	3.080	2.814	0.118	0.014	0.042
36	52	2.700	3.240	2.922	0.128	0.016	0.044
45	793	2.460	4.040	3.034	0.154	0.024	0.051
46	175	2.770	3.900	3.160	0.208	0.043	0.066
75	545	2.360	3.450	2.777	0.113	0.013	0.041
76	151	2.660	3.380	2.951	0.144	0.021	0.049
85	1,600	2.390	3.850	2.828	0.143	0.021	0.051
86	1,536	2.510	3.840	2.975	0.185	0.034	0.062
95	701	2.480	3.790	2.781	0.128	0.016	0.046

Total Copper Contact Analysis

Golder carried out an independent grade contact analysis for the current estimation domains for total copper. The analysis showed sudden grade changes between all disseminated – breccia contacts, supporting the use of hard boundaries for block grade estimation.

Smooth grade transitions at contacts were observed between disseminated or breccia domains which share samples for estimation purposes, supporting the soft boundaries implied in the domain grouping. The use of soft boundaries improves the robustness of the estimation at the contacts between domains.

The contact analysis for total copper grades supports the definition of boundaries used in the grade estimation process at Sossego.

Gold Contact Analysis

The independent gold contact analysis showed hard boundaries between disseminated and breccia domains, as expected. In general, the nature of the contacts is consistent with the definition of gold estimation domains: domains that share samples present smooth grade transitions and independent domains show abrupt transitions at contacts.

However, some exceptions were found which agree with the domain definitions suggested in Section 10.

The contact analysis between domains 15 and 25 suggests a hard boundary. This result is in line with the idea of defining domain as an independent estimation unit. Also, the contact between domains 85 and 95 show a soft transition of grades (the peak observed at approximately 15 m was calculated using only 8 pairs), supporting the suggestion to merge these units into a single estimation domain.

The contact analysis study showed that, in general, boundaries have adequately been applied for gold estimation domains. However, the results are concordant with the questioned gold domains and should be updated if the definition of these units is modified.

Density Contact Analysis

The contact analysis for density values resulted in very variable profiles, considering the lower amount of data available compared to the other two variables. This precludes an accurate analysis and further assessment should be carried out when more density measurements are available. For the profiles that do provide meaningful results, the boundaries between disseminated and breccia domains generally show sudden changes in density when approaching the contact, suggesting hard boundaries.

Some discrepancies were found, where a soft boundary can be observed for approximately 20 m from the contact, suggesting that composites from both domains could be shared for estimation purposes.

The density contact analysis study showed that the currently applied boundaries need to be reviewed when more density data is available. It should also be analyzed in accordance with the potential new estimation domains produced by the review of the current units.

Spatial Correlation and Variography

Variography calculation and modelling was completed in Gemcom® by Vale. Golder checked the Sossego variography by performing independent correlogram calculations of all estimation domains for total copper, gold and density. The correlogram models used in the validation process were based on the models described in the estimation parameter files for each element. The process involved the following steps:

- Calculating experimental correlograms with Golder software;
- Checking the interpretation of the nugget effect by means of down-the-hole (DTH) correlograms; and
- Assessing the fit of the correlogram models to the independently calculated experimental correlograms in the main directions of continuity.

Parameters such as lag distance, angle tolerance and bandwidth were reproduced in the experimental correlogram independent calculations.

Total Copper

The down-the-hole correlogram analysis, produced to check the interpretation of the nugget effect for total copper, indicates that in general the modelled values are acceptable. The comments of the obtained results are the following:

- The nugget effects fitted for domains 16 and 85 are considered adequate.
- The DTH correlograms for domains 15 and 25-36 indicate that the nugget effects fitted are acceptable; however the value used could be slightly higher to improve fit to the experimental values.

- The DTH correlograms for domains 45-46 and 86 indicate that the nugget effects adopted are acceptable; however the value used could be slightly lower to improve fit to the experimental values.
- The nugget effect fitted for domain 75-76 is considered to be underestimated. The adopted value should be higher to fit the experimental values. The directional correlograms do not support the nugget effect election for domain 75-76.
- The nugget effect fitted for domain 95 is considered to be overestimated. Its value should be lowered to fit the experimental values. The directional correlograms do not support the nugget effect election for domain 95.

The correlogram models used for total copper estimation are summarized in Table 8-17. The models were fitted to the experimental correlograms obtained independently by Golder. The majority of the models are acceptable and subtle changes could be applied in some cases to improve the fit to the experimental correlograms. The following comments resulted from the model checks:

- The correlogram models for domains 45-46 and 75-76 adequately fit the experimental correlograms.
- The model used in the minor direction of continuity for domains 15 and 16 could be shortened to improve fit; however, the models are considered acceptable.
- The model used in the major direction of continuity for domains 85 and 95 hould be shortened to fit the experimental data. Considering that the kriging search ellipsoids are based on correlogram ranges, overestimating ranges would have a direct impact on the estimation results.
- The models used in the major and semi-major directions of continuity for domain 25-36 could be slightly shortened to improve fit; however, the model is considered acceptable.

In general, the correlogram models fitted for total copper are considered acceptable. Only a few modifications are needed to improve the model fit to the experimental correlogram and to avoid impacting the kriging plans with artificially large search ranges.

Table 8-17: Total copper correlogram model summary

Cu Domain	Direction (Az/Dip)	Nugget Effect	First Structure			Second Structure		
			Sill Cont.	Type	Range	Sill Cont.	Type	Range
15	200/-75	0.30	0.50	sph	25.0	0.20	sph	80.0
	110/0				45.0			100.0
	200/15				50.0			90.0
16	200/-75	0.30	0.50	sph	90.0	0.20	sph	130.0
	110/0				90.0			100.0
	200/15				30.0			40.0
25 - 36	150/-55	0.10	0.60	sph	35.0	0.30	sph	200.0
	60/0				35.0			100.0
	150/35				25.0			90.0
45 - 46	175/-70	0.30	0.55	sph	45.0	0.15	sph	149.8
	85/0				30.0			79.8
	175/20				30.0			60.1
75 - 76	125/0	0.30	0.50	sph	20.0	0.20	sph	80.0
	35/0				10.0			40.0
	125/90				15.0			60.0
85	140/90	0.50	0.30	sph	10.0	0.20	sph	100.0
	140/0				5.0			50.0
	50/0				5.0			25.0
86	140/90	0.50	0.30	sph	10.0	0.20	sph	110.0
	140/0				30.0			50.0
	50/0				10.0			40.0
95	140/0	0.50	0.30	sph	5.0	0.20	sph	50.0
	50/0				5.0			25.0
	140/90				10.0			100.0

Gold

The nugget effect analysis for the gold correlograms indicates the following:

- The modelled nugget for domain 15-16 is considered adequate.
- Even though the DTH correlograms indicate that the nugget effect has been underestimated for domains 25-36 and 45-46, the directional correlograms support its interpretation. The values modelled for these domains are considered acceptable.
- Nugget effects have been underestimated for domains 75-76, 85-86 and 95. The directional correlograms support the observed underestimation.

The correlogram models used for gold estimation are summarized in Table 8-18. All models are found acceptable and subtle changes could be applied in some cases to improve the fit to the experimental correlograms. The following comments resulted from the model checks:

- The correlogram ranges used for the major directions on domains 15-16, 25-36 and 75-76 could be shortened to improve the fit to the experimental values. These models are considered acceptable.
- The correlogram ranges used for the major and semi-major directions on domain 45-46 could be shortened to improve the fit to the experimental values. The model is considered acceptable.
- The correlogram models fitted for domains 85-86 and 95 are considered adequate.

The correlograms adopted for gold are considered acceptable. Modifications are suggested to improve the model fit to the experimental correlograms.

Table 8-18: Gold correlogram models summary

Au Domain	Direction (Az/Dip)	Nugget Effect	First Structure			Second Structure		
			Sill Cont.	Type	Range	Sill Cont.	Type	Range
15 - 16	200/-75	0.50	0.20	sph	30.0	0.30	sph	200.0
	110/0				30.0			160.0
	200/15				10.0			80.0
25 - 36	150/-55	0.30	0.50	sph	30.0	0.20	sph	200.0
	60/0				30.0			100.0
	150/35				10.0			90.0
45 - 46	175/-70	0.30	0.50	sph	33.0	0.20	sph	180.9
	85/0				29.7			128.9
	175/20				15.0			114.1
75 - 76	125/90	0.30	0.50	sph	30.0	0.20	sph	60.0
	125/0				10.0			90.0
	35/0				10.0			20.0
85 - 86	140/90	0.30	0.50	sph	10.0	0.20	sph	60.0
	140/0				10.0			35.0
	50/0				10.0			60.0
95	140/0	0.30	0.50	sph	10.0	0.20	sph	35.0
	50/0				10.0			60.0
	140/90				10.0			60.0

Density

In general, the nugget effect analysis carried out for density correlograms shows that most of the interpretations are adequate. The only suggestions are focused on the Morro do Sossego sector, as follows:

- The DTH correlogram for domain 85-86 indicates that the nugget effect fitted is acceptable; however the value used could be slightly higher to improve fit to the experimental values.
- The nugget effects fitted for domains 75-76 and 95 are considered to be underestimated.

The correlogram models used for density estimation are summarized in Table 8-19. Some models are acceptable, with only subtle changes needed to improve the fit to the experimental correlograms. Other domains need to reduce ranges in order to adequately fit the experimental correlograms. The following comments resulted from the model checks:

- The models adopted for domains 75-76 and 95 are considered to adequately fit the experimental correlograms.
- The correlogram ranges used for the major and semi-major directions on domain 85-86 could be shortened to improve the fit to the experimental correlograms. The model is considered acceptable.
- The model used for domains 15-16, 25-36 and 45-46 should be shortened to fit the experimental data. Considering that the kriging search ellipsoids are based on correlogram ranges, overestimating ranges would have a direct impact on the density results.

Table 8-19: Density correlogram models summary

Dens Domain	Direction (Az/Dip)	Nugget Effect	First Structure			Second Structure		
			Sill Cont.	Type	Range	Sill Cont.	Type	Range
15 - 16	110/0	0.10	0.90	sph	140.0	-	-	-
	200/-75				200.0			-
	200/15				120.0			-
25 - 36	150/-55	0.10	0.45	sph	50.0	0.45	sph	400.0
	60/0				40.0			200.0
	150/35				20.0			150.0
45 - 46	175/-70	0.10	0.60	sph	70.0	0.30	sph	300.0
	85/0				20.0			200.0
	175/20				10.0			150.0
75 - 76	125/90	0.20	0.80	sph	75.0	-	-	-
	125/0				80.0			-
	35/0				60.0			-
85 - 86	140/90	0.20	0.50	sph	10.0	0.30	sph	110.0
	140/0				10.0			40.0
	50/0				10.0			80.0

Dens Domain	Direction (Az/Dip)	Nugget Effect	First Structure			Second Structure		
			Sill Cont.	Type	Range	Sill Cont.	Type	Range
95	140/90	0.20	0.50	sph	10.0	0.30	sph	110.0
	140/0				10.0			40.0
	50/0				10.0			60.0

The correlogram models used for density need some modifications to improve the model fit to the experimental correlograms and to avoid impacting the kriging plans with artificially large search ranges and to adequately interpret the nugget effect.

Block Modelling and Grade Estimation

Block Model Definition

The Sossego resource model was generated by Vale in Gemcom®. The resource model was estimated using two regular block models with block sizes of 10m by 10 m by 8 m (for elevations above 200 m) and a maximum of 10 m by 10 m by 16 m (for elevations below 199 m) and then re-blocked to the parent block size of 10 by 10 by 16 m for reporting purposes. The use of the first block model with a smaller block height is supported by the need to adequately model the geometry of the oxide mineral zone. Considering that the limit between both models is defined by elevation, sulphide blocks are present in both models.

Block models were provided by Vale in ASCII files (*MODELO_8_SOS.txt* and *MODELO_16_SOS.txt*) and imported to Vulcan® for further analyses. The Vulcan® models correspond to files *SOSSEGO_08_JB.bmf* and *SOSSEGO_16_JB.bmf* respectively. The definitions for the two blocks models are detailed in Table 8-20 and Table 8-21.

Table 8-20: Sossego 10 m by 10 m by 8 m block model definition

Orientation	**Bearing**	**Dip**	**Plunge**
	90	0	0
Origin	East	North	Elevation
	601 000	9 289 300	120
Parent block size	10	10	8
Parent number of blocks	530	250	20

Table 8-21: Sossego 10 m by 10 m by 16 m block model definition

Orientation	**Bearing**	**Dip**	**Plunge**
	90	0	0
Origin	East	North	Elevation
	601 000	9 289 300	-600
Parent block size	10	10	16
Parent number of blocks	530	250	50

For the purposes of an in situ mineral reserve estimate, the block model cell sizes and composite lengths are acceptable to achieve a reasonable local estimation quality and controlled smoothing effect.

Grade Estimation

In general, estimation for all variables was performed using a three (3) pass ordinary kriging approach by estimation domain. Additional passes were performed for some domains to allow the estimation of all blocks. A block discretization of 5 by 5 by 8 was adopted for the 16 m block height model and 5 by 5 by 4 for the 8 m height model.

For the purposes of an in situ mineral reserve estimate, the overall estimation approach adopted by Vale for total copper, gold and density is acceptable.

Estimation Parameters

The search parameters used for total copper grade, gold grade and density are detailed in Table 8-22, Table 8-23 and Table 8-24 respectively. The kriging plan developed for each estimation domain consists of a consistent sample configuration scheme, with the search radii increasing with the estimation pass.

The sample configuration for the three passes is as follows:

- Minimum number of samples : 12
- Maximum number of samples : 24
- Maximum samples per drill hole : 8

In order to allow all blocks to be estimated, additional estimation passes were performed for some estimation domains:

- Total copper and gold
    - Domain 15 : 3 additional passes
    - Domain 35 : 1 additional pass.
- Density
    - Domain 15 : 2 additional passes
    - Domains 25, 35, 45 and 46 : 1 additional pass.

Some of these additional passes decreased the minimum number of samples to 3. These were pass 6 – domain 15 and pass 4 – domain 35 for total copper and gold; pass 5 – domain 15 and pass 4 – domain 25 for density. All the additional passes increased the search radii used in the third pass. The distances used are detailed in Table 8-25, Table 8-26 and Table 8-27 for total copper, gold and density respectively.

Table 8-22: Total copper search radii by estimation pass

Domain	Search Orientation	Pass 1			Pass 2			Pass 3		
	(az/dip)	Major / Semi-major/ Minor Search Radius			Major / Semi-major/ Minor Search Radius			Major / Semi-major/ Minor Search Radius		
15	200/-75	160	160	150	240	240	225	320	320	300
16	200/-75	260	260	80	390	390	120	800	800	800
25 - 36	150/-55	240	140	120	360	210	180	600	600	600
45 - 46	175/-70	180	100	80	270	150	120	700	700	700
75 - 76	125/0	60	50	120	90	75	180	1200	1200	1200
85	140/90	120	60	50	180	90	75	600	600	600
86	140/90	280	120	100	420	180	150	600	600	600
95	140/90	112	28	56	168	42	84	600	600	600

Table 8-23: Gold search radii by estimation pass

Domain	Search Orientation	Pass 1			Pass 2			Pass 3		
	(az/dip)	Major / Semi-major/ Minor Search Radius			Major / Semi-major/ Minor Search Radius			Major / Semi-major/ Minor Search Radius		
15	200/-75	400	320	160	600	480	240	800	640	320
16	200/-75	160	160	60	240	240	90	800	800	800
25 - 36	150/-55	240	140	120	360	210	180	600	600	600
45 - 46	175/-70	180	100	80	270	150	120	700	700	700
75 - 76	125/0	60	50	120	90	75	180	1200	1200	1200
85 - 86	140/90	120	60	50	180	90	75	600	600	600
95	140/90	112	28	56	168	42	84	600	600	600

Table 8-24: Density search radii by estimation pass

Domain	Search Orientation	Pass 1			Pass 2			Pass 3		
	(az/dip)	Major / Semi-major/ Minor Search Radius			Major / Semi-major/ Minor Search Radius			Major / Semi-major/ Minor Search Radius		
15	200/-75	400	280	240	600	420	360	800	560	480
16	200/-75	400	280	240	600	420	360	800	800	800
25 - 35	150/-55	400	200	150	600	300	225	700	700	700
26 - 36	150/-55	800	400	300	1200	600	450	400	400	400
45 - 46	175/-70	600	400	300	900	600	450	600	600	600
75 - 76	125/0	150	120	160	225	180	240	1200	1200	1200
85 - 86	140/90	220	160	80	330	240	120	600	600	600
95	140/90	220	120	80	330	180	120	600	600	600

Table 8-25: Total copper search radii for additional estimation passes

Domain	Pass	Search Orientation	Major / Semi-major/ Minor Search Radius		
		(az/dip)			
15	4	200/-75	400	400	375
15	5	200/-75	600	600	600
15	6	150/-55	1000	1000	1000
35	4	150/-55	1000	600	600

Table 8-26: Gold search radii for additional estimation passes

Domain	Pass	Search Orientation	Major / Semi-major/ Minor Search Radius		
		(az/dip)			
15	4	200/-75	1000	800	400
15	5	200/-75	1200	960	480
15	6	150/-55	1500	1500	1500
35	4	150/-55	1000	1000	1000

Table 8-27: Density search radii for additional estimation passes

Domain	Pass	Search Orientation	Major / Semi-major/ Minor Search Radius		
		(az/dip)			
15	4	200/-75	1000	700	600
15	5	200/-75	1000	1000	1000
25	4	150/-55	800	800	800
35	4	150/-55	1000	1000	1000
45	4	175/-70	600	600	600
46	4	175/-70	1000	1000	1000

The resource model report indicates that the first search radii set is obtained using two times the range value for 95% of the correlogram model sill, and the second and third sets are twice and three times the range of the first respectively. Comparing the search radii with the correlogram models showed that this is not the case: the anisotropy of the search ellipsoid does not match the correlogram ranges. Also, the second pass ellipsoids are defined as 1.5 times the first pass ellipsoid and the third pass is mostly isotropic, except for domain 15 for the three variables.

Search Distances and Anisotropies

Search distances are too large for the first and second estimation passes. It would be good practice to use 95% of the sill to define the ellipsoids of the first pass and to adequately weight the radii for the second and third passes (i.e., 2 and 3 times), maintaining the correlogram anisotropy for all estimation passes. The search distances for fourth and greater passes are acceptable, but no blocks estimated in these passes should be classified as measured or indicated mineral resource.

The current search ellipsoids combined with the minimum and maximum sample parameters could produce highly smoothed results. Also, large search radii combined with the use of high yield restrictions with no low yield restrictions could result in artificial smearing of low grades, underestimating the global means in certain estimation domains.

Minimum/Maximum Samples

As stated in the search ellipsoid analysis, the minimum and maximum number of samples are considered high. This could produce highly smoothed estimated results, especially for 10 m by 10 m by 8 m blocks. Considering that the goal of the first estimation two passes is to obtain smaller volumes but more robust results, both parameters should be lowered for the first two passes.

Octants

The use of an Octant based search is useful as it can ensure good spatial coverage and influence from different directions. Currently, no octant restrictions are applied in the kriging plans. Not applying octant restrictions could lead to an interpolation of blocks by extrapolating samples from a single hemisphere. Such blocks should not be classified as measured or indicated according to the international standards for reporting.

Maximum Samples per Drill Holes

The use of a maximum number of samples per drill hole in conjunction with a higher minimum number of composites is a good practice since it forces the use of more than two drill holes, ensuring that the samples used are not biased towards a single hole.

The current configuration ensures that samples from at least 2 drill holes are used, avoiding using samples from a single drill hole to estimate a block. This practice is considered adequate.

High Grade Restrictions

It is good practice to restrict the influence of anomalous high values that deviate from the grade cumulative distribution of each unit.

High yield restrictions have been applied to all domains for the three variables. The thresholds used and the influence volumes are summarized in Table 8-28 for all variables.

The influence volume has been defined using approximately 50% of the total range of the correlogram model. Considering the variability of the Sossego deposit, using ranges that would smear high values for up to 40 blocks is not optimal. The chosen volumes should preferably restrict the influence of high values to a vicinity of approximately 4 or 5 blocks in the main anisotropy direction. The radii used in the secondary directions should be picked so that the anisotropy of the domain is honoured.

Table 8-28: High yield treatment definition

Domain	Hy-value Cu	Major / Semi-major/ Minor Search Radius			Hy-value Au	Major / Semi-major/ Minor Search Radius			Hy-value Density	Major / Semi-major/ Minor Search Radius		
15	1.38	40	50	20	0.37	100	80	40	3.04	100	70	60
16	6.34	65	50	20	2.21	40	40	15	3.56	100	70	60
25	1.71	100	50	45	0.58	60	35	30	3.57	100	50	40
26	5.85	100	50	45	2.44	60	35	30	3.70	200	100	75
35	1.40	100	50	45	0.71	60	35	30	3.06	100	50	40
36	5.84	100	50	45	5.29	60	35	30	3.17	200	100	75
45	1.39	75	40	30	0.47	45	25	20	3.37	150	100	75
46	6.19	75	40	30	1.76	45	25	20	3.69	150	100	75
75	4.70	30	20	40	1.62	15	12	30	3.08	38	30	40
76	5.43	30	20	40	3.25	15	12	30	2.90	29	23	30
85	5.91	50	13	25	1.51	30	15	12	3.19	55	40	20
86	9.77	70	30	25	2.97	30	15	12	3.35	40	30	15
95	7.97	50	13	25	1.59	28	7	14	3.12	55	30	20

Independent checks using the cumulative probability plots for each variable by estimation domain were produced to validate the thresholds applied for high yield restrictions. Typically, the choice of the threshold values respond to breaks in the upper tail of the distribution or a fixed percentage of the population (e.g., 95%). The results of the analysis indicate that a fixed percentage of approximately 97% of the population was the chosen tool for the definition. However, the criteria to define the high yield thresholds should be properly documented.

The analysis for total copper indicates that the thresholds used for domains 15, 16, 25, 26, 45 and 85 are low. The values adopted were not picked due to a break in the distribution

The high yield threshold check for gold grades shows a similar behaviour. Domains 15, 16, 25, 26, 45, 75, 85 and 95 use lower values than the distribution. The threshold picked for domain 36 could be lower.

The distribution analysis for density indicates that only high yield values adopted for domains 35, 36 75 and 95 are considered acceptable. The rest of the thresholds are considered too low, except for domain 46, which is considered high.

Block Model Validation

To check the validity of the Sossego resource model, an independent validation of the block model was carried out to assess the kriging performance and conformance to the input data. A series of comprehensive checks were performed including:

- Comparison between composites and block model statistics;
- Visual validation of estimated grades versus composite grades;
- Swath plots comparing block grades against composite grades; and
- Reconciliation analysis based on production data and short term models.

In general the validations performed in this audit, only determine whether the estimation has performed as expected. Acceptable validation results do not necessarily mean the model is correct or derived from the right estimation approach. It only means the resource model is a reasonable representation of the data used and the estimation method applied.

Statistical Comparison

Comparing global statistics between composites and estimated values is a useful way to perform a global check of the estimation results. The block model should show conformance to the input data, and close agreement should be observed with the data mean grades. The mean grade conformance is tabulated by the following criteria:

- Values close to 0% represent good conformance. Negative values indicate that the model estimates are conservative with respect to the data, and positive values show overestimation of the global average in the estimates. In general, differences lower than 5% are desirable, differences greater than 10 % require attention.

Initially, to achieve a representative sample distribution by element and domain, an independent series of declustering weights were produced using cell declustering, alternative kriging plans, inverse distance squared and other methods. The block model statistics consider the volume size of the sub cell block model as a weighting factor. The results showed that inverse distance squared driven weights were the best declustering method for the deposit, hence the statistical comparison was performed using these declustering weigths.

The global statistical analysis for total copper indicated that block estimates of domains 76, 86, 95, 120 and 124 acceptably honour the global means of the declustered composites. The results for domains 75 and 85 show a relatively high underestimation of global means. Even though the disseminated domains correspond to low total copper units and that the observed underestimation imply a somewhat conservative model, the estimation for these units should be reviewed to obtain an acceptable reproduction of the global means.

As seen in the statistical comparison for gold global means, most estimation domains show differences below 10%, which is considered acceptable. The results for domains 46 and 76 show a high underestimation of global means.

The global means analysis for density indicates that the results are adequate for all domains.

Visual Validation

The visual validation performed for total copper indicates that generally the block estimates adequately honour the composite grades. It is possible to follow grade continuity supported by composite data.

However, the effects of large search ellipsoids and high minimum and maximum number of samples can be observed in the block estimates. The visual analysis suggests smoothed results. The grade continuity is reproduced acceptably but the high variability observed for the composite grades is absorbed by the smoothed results.

Also, large volumes of grades are being estimated in depth without composites available to support continuities. This result highlights the need to shorten the search ranges and to use a more restrictive high yield restriction influence volume. An example is shown in Figure 8-21.

The recommendations produced concerning search parameters and volumes of influence for high yield restrictions are confirmed by the visual validation for total copper.

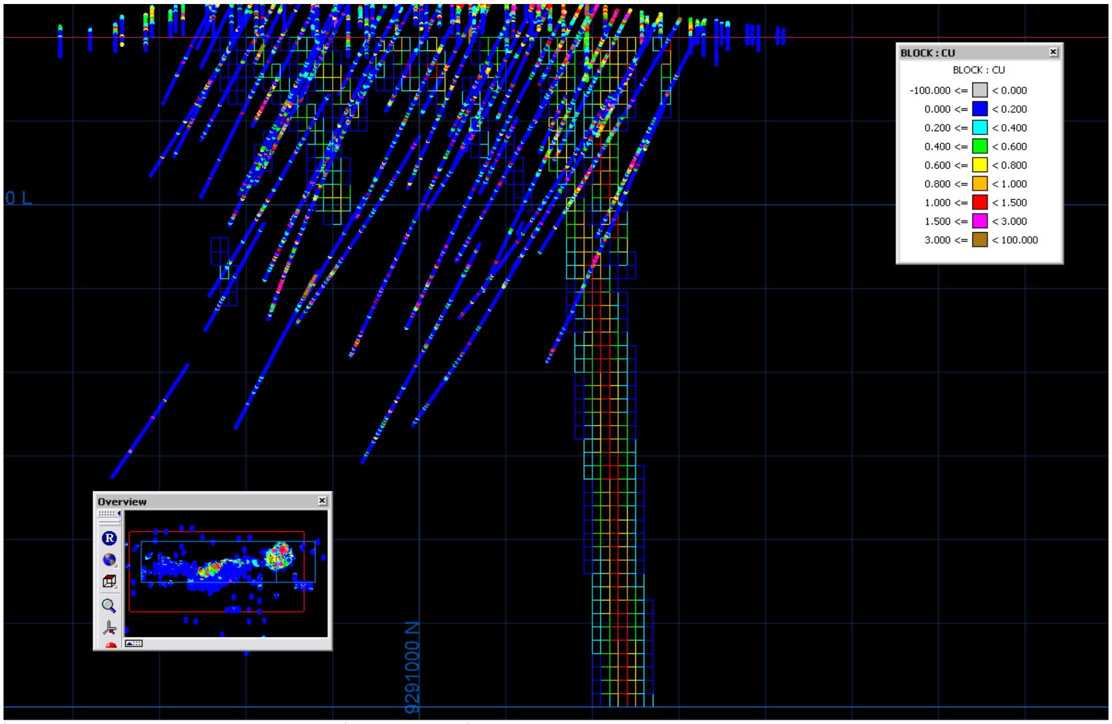


Figure 8-21: Section view @ E 605,469 ±100m: total copper

The visual validation for total copper indicates that the results are acceptable. However, potential modifications to the kriging plans should be assessed to improve the results.

The results for the visual validation for gold estimates produce similar results to the total copper analysis. In general, the low and high grade zones are acceptably reproduced. The high gold volumes are well controlled, but the results are somewhat smoothed considering the intrinsic high variability of gold grades. Special attention was paid to domain 15, considering the recommendation provided in section 8.12, to use it as an independent estimation unit. Figure 8-22 shows a plan view comparing blocks and composites coded 15. The high grade volume at the center of the image is supported but only two high grade composites. When displaying all the composites that are actually available to estimate the domain (Figure 8-23), a series of higher grades appear to the south and east of the high grade volume which are probably contributing to create it.

The gold analysis produces similar results to total copper: large volumes of blocks are being estimated in depth without composites available to support continuities. This result highlights the need to shorten the search ranges for the gold kriging plans.

In general, the gold estimates acceptably restrain the high grade smearing showing controlled volumes around high grade composites. The grade continuity is also adequately honoured. However, the analysis highlights the need to use domain 15 independently for estimation purposes and kriging plans should be reviewed to avoid unwanted smoothing and large volumes estimated at depth without composite support.

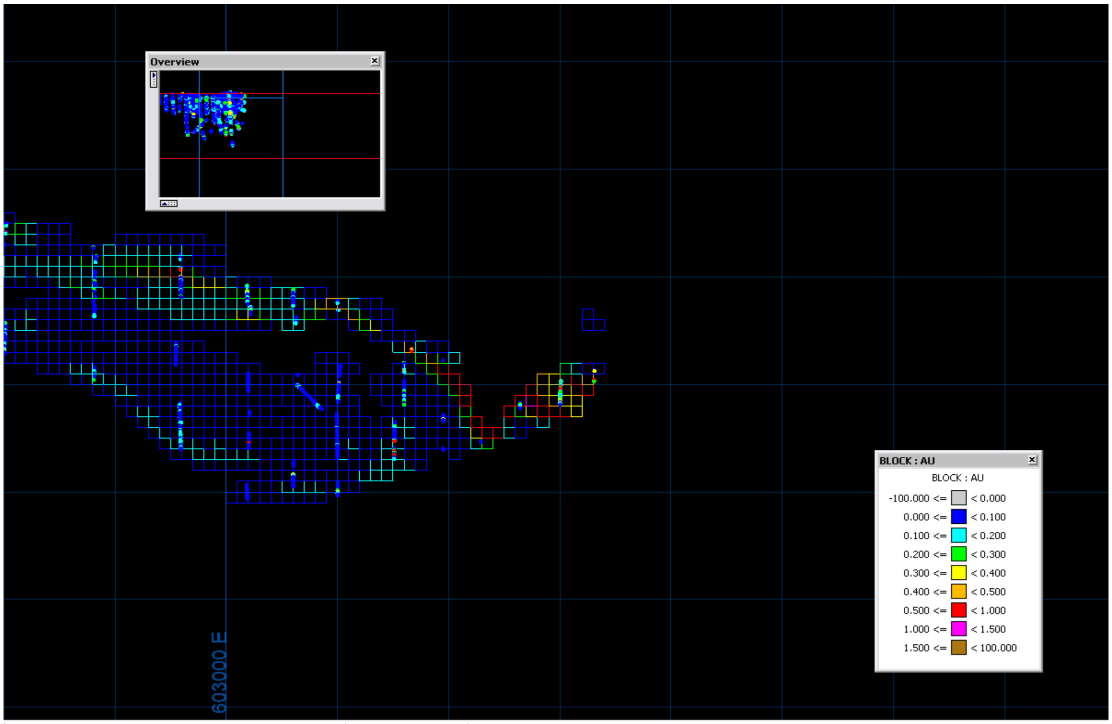


Figure 8-22: Plan view @ 150 m ±30m: gold domain 15 (blocks and composites)

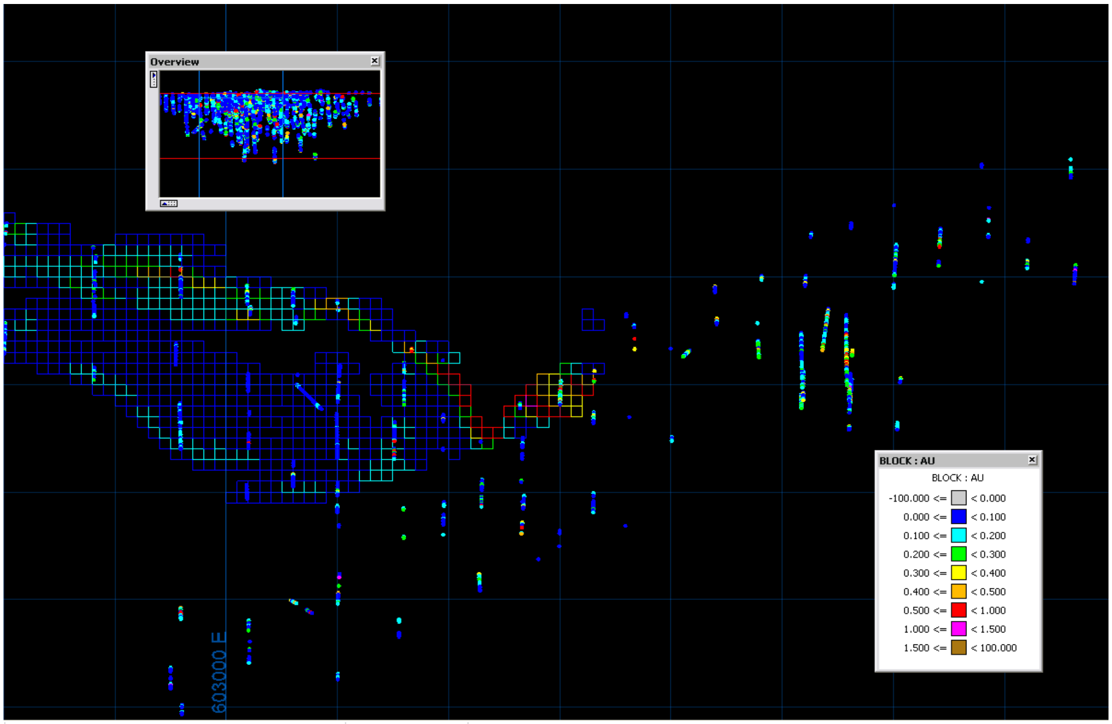


Figure 8-23: Plan view @ 150 m ±30m: gold domain 15 blocks and composites actually used for estimation

The density estimates compare well to the composite values globally. The smoothing observed for the other two variables is also present for density, which was expected since the kriging plans use large search radii, the same sample configuration and the number of available composites is lower compared to total copper or gold.

The visual validation for density indicates that the results are acceptable. However, potential modifications to the kriging plans should be assessed to improve the results.

Swath Plots

In many circumstances, global statistical comparisons between composite and estimated block grades can be potentially misleading because they are influenced by extrapolation effects, erratic drill hole coverage and geometry of estimation domains. To further evaluate the robustness of the conformance of the block grades to the sample grades, in a semi-local approach, swath plots were produced for Sossego.

Swath plot generation involves averaging both the blocks and samples in panels of 40 m (easting) by 40 m (northing) by 32 m (RL), then averaging of the panel averages into Easting, Northing and RL swaths to allow trend plots of block vs. composite values to be constructed. The use of panel averages also allows for the generation of scatter plots and Q-Q plots of panel grades to provide an indication of conditional bias and degree of smoothing. On these plots, two distributions that are very similar would plot over the 45 degrees line. Significant deviations indicate potential for over smoothing and bias.

It is important to note whereas the global statistics show the impact of the combined interpolation and the extrapolations. The swath plots can only demonstrate impact of interpolations, for this reason, any issue arising from the swath plots should be addressed since it occurs near the samples.

In general, the swath plots for total copper show good conformance between block grades and composite grades. The local block model grades acceptably honour the composites, although most domains result in smoothed results due to the large search radii and the high minimum and maximum number of composites chosen for the kriging plans. The result for total copper domain 85 shows that the local grades of the block model are underestimated when compared to the composite grades. This result is in line with the global statistical analysis for the domain.

Most of the gold domains show a good reproduction of local grades for block estimates, although smoothing can be observed in all swath plots. The results for domains 15, 25 and 45 show that the block estimates have higher local means than the composites. A slight underestimation of local mean is observed for domain 86.

Density swath plots show good local mean reproduction for all estimation domains.

Reconciliation

Using production data, Vale produces short term models for reconciliation purposes. Two reconciliation models are created: a short term model which is created monthly, with a block size of 10 m by 10 m by 10 m; and a “super short term” block model which is created every time new production data is available, with a block size of 5 m by 5 m by 16 m.

The resource model was compared to the production data and both short term models as a reconciliation exercise. Swath plots were used in the comparison.

Also, a Q-Q Plot was produced to compare production and composite data paired to 20 m. The plot shows that the blast hole total copper grade data is consistently higher than the composites, so the comparison between the resource model and the estimated short term models should be affected by this.

The results show good correspondence for grade trends. As expected, the blast holes show a slightly higher grade than the block estimates and block grade smoothing can also be observed.

The comparison shows that the mineral reserve estimates are acceptable compared to production data.

The comparison between the resource model and both short term models shows the effect of the higher grades obtained for the blast hole data compared to the composites. The resource model is used as “composite data” in the swath plots. The grade trends are well reproduced by the resource model, but its grades are lower than the short term models. These results imply that the resource model is conservative, especially for high grades.

Resource Classification

The criteria used to categorize the mineral resource estimate are based on a series of parameters related to an independent kriging run. This was estimated with shorter search distances (defined by estimating the range at 95% of the total correlogram sill) than the first estimation pass for total copper. The parameters considered are the total copper correlogram ranges, anisotropic distances to the nearest composite and the number of composites used to estimate a block. The anisotropic distance is calculated relative to the East direction in Gemcom®. A distinction is made between blocks that are interpolated (inside the drill grid) and blocks whose grades are extrapolated from a single drill hole. The categorization scheme is the following:

Measured mineral resources:

- Anisotropic distance to nearest sample < 1/2 range Cu correlogram;
- and, number of composites >= 2/3 of maximum;
- and, estimated in the first pass (roughly equal to the correlogram range);
- or, nearest composite distance <= 20 m (only for Sequeirinho).

Indicated mineral resources:

- Anisotropic distance to nearest sample < range Cu correlogram;
- and, number of composites >= 1/2 of maximum;
- and, estimated in the second pass (roughly equal to twice the correlogram range);
- or, anisotropic distance < ½ range Cu correlogram (extrapolation);
- or, estimated in the first pass (Sequeirinho and Sossego). This is equivalent to at least 2 holes within the range of the correlogram.

Inferred mineral resources:

- All the other blocks inside the envelopes.

The categorization scheme is shown in Figure 8-24.

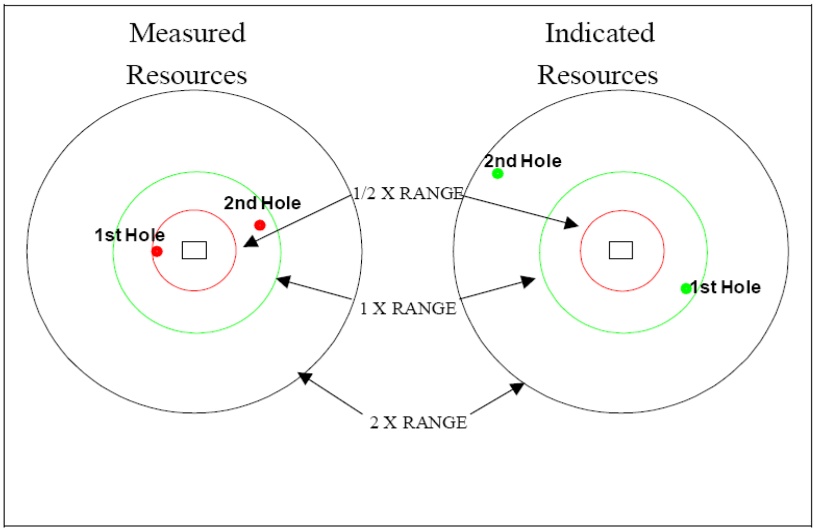


Figure 8-24: Categorization scheme for the Sossego resource model

The Sossego categorization does not consider smoothing. Isolated blocks can be found surrounded by blocks of a different category (Figure 8-25), which is known as the salt and pepper effect. Smoothing the categorization results is good practice, obtaining more continuous and coherent categorized volumes. In practice, smoothing should be performed with care to maintain the proportions of the different classes in the final result.

The categorization visual analysis also showed areas in which measured mineral resource is in contact with inferred material (Figure 8-27). The correct result would be for blocks to grade from measured to indicated to inferred mineral resource, so no contact should exist between measured and inferred mineral resource. This issue should be addressed.

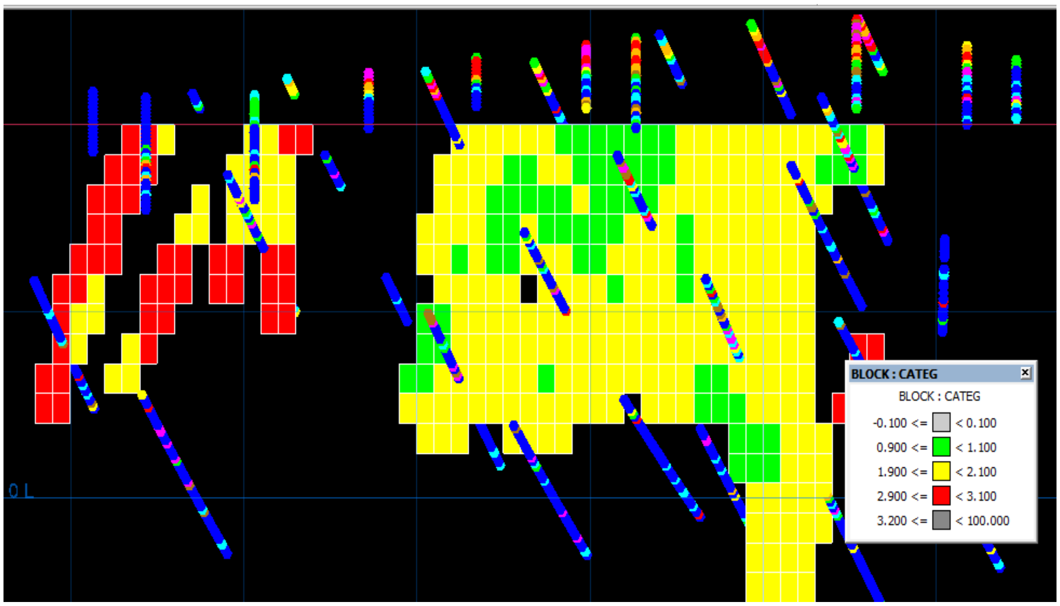


Figure 8-25: Sossego categorization: salt and pepper effect

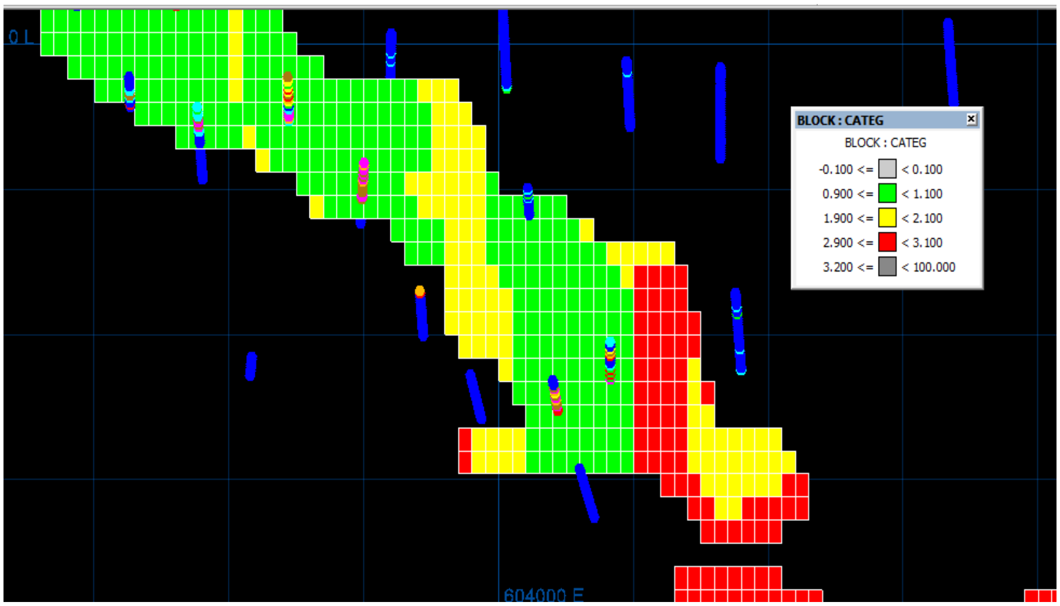


Figure 8-26: Sossego categorization: measured - inferred material contact

Another check performed on the block categorization indicated that blocks estimated in the third or fourth passes have been classified as measured or indicated mineral resources. Considering the large search ellipsoids used for the third and fourth kriging passes, no blocks estimated under these scenario should be categorized as measured or indicated mineral resource.

No blocks estimated in the third or fourth kriging passes should be categorized as measured or indicated mineral resources.

8.13 Mineral Reserve Estimation

Mining Model

Vale constructed a global long term resource model in Gemcom Software that forms the basis of the publicly reported resources and reserves. Individual models were created for the oxide and sulphide resources for kriging purposes, which were subsequently merged into a single resource model.

The Resource block model has a regular block size of 10 m east-west (X), 10 m north-south (Y) and 8 m vertically (Z) is used. Each block is assigned a majority code to identify lithology domains.

For open pit optimisation and mine planning purposes the resource model was regularised to 10 m by 10 m by 16 m. This block size was selected to approximate the smallest mining unit (SMU) based on the proposed mining fleet. In addition the geology codes were simplified to reflect material types defined based on an economic criteria. The material types used in the mining model are presented in Table 8-29.

Table 8-29: Definition of material types used in the mining model

Code	Definition
1	Oxidised red zone
2	Oxidised green zone
3	Transition between oxide and sulphide ore
4	Sulphide ore

The use of the Mining Material Types is a reasonable approach to creating a suitable model for the development of mine plans and mineral reserves.

Approach to Mine Planning

The main assumptions adopted for the estimation of Mineral reserves were:

- Estimates of capital and operating costs based on current costs;
- Estimated metallurgical recoveries at the concentrator based on historical performance;
- Geotechnical parameters revised in 2007;
- Metal price and exchange rate based on corporate forecast from May 2009.
- Only measured and indicated sulphide mineral resources were considered for pit optimization

Whittle Four-X was used for pit optimization. The optimization generates a set of incremental nested pit shells from which the ultimate pit shell is selected. The pushbacks or phases are then designed guided by groupings of incremental shells that are constrained by tonnage, grades, stripping ratio and/or other operational requirements.

Once the ultimate pit has been defined Gemcom is used to develop a detailed design of the pit limits and pushbacks. Cut-off grade optimization and production scheduling is completed with a non-commercial software by NCL Consultores. The long term prices adopted for copper and gold were US$1.98/lb and US$700/oz respectively.

Then, the normal mine planning routines of dump design, haul profile and equipment scheduling were carried out. For haul profiles analysis, the PLAN-INT software was used to calculate and optimize the haul cycles. The data is then used for economic evaluation.

The various plans and presentations provided by Vale demonstrate that this work is being carried out to a high standard. The long and medium term planning processes are sound and appear to be competently carried out by the engineers.

Short Term Planning

The short term planning team is responsible for designing the extraction polygons or ore blocks for drilling and blasting. The definition of such polygons needs to take into consideration multiple constraints:

- Production targets
- Equipment size
- Long and medium term physical advances (phases)
- Ore loss and dilution
- Availability of working areas
- Slope constraints
- Safety

These polygons are forwarded to the survey department who marks them in the pit. The major computational tasks are volume/tonnage calculation, reporting and drill and blast design. Gemcom software is used for this purpose. The drill-and-blast team then marks the blast holes in the field and the survey crew loads them into the dispatch system. The driller executes the drilling pattern as indicated on the dispatch screen. The survey team then picks up the final locations of the blast holes and loads them into the database.

Monthly plans are developed by the short term team to provide tactical direction for the operation. Sossego personnel indicated the intention of having a rolling three months plan to anticipate medium term infrastructure requirements. The mine geology team is responsible for the construction of the short term grade model. Currently reconciliations between the short and long term models are reported on a yearly basis.

Golder supports the plan to have the short term model reconciled against the long term model in a monthly basis and then reported on a quarterly basis.

Geotechnical Parameters

A procedure is currently in place that includes periodic inspections on slopes for the open pits, waste dumps stockpile and tailings dam. The objectives of these inspections are to verify stability conditions, drain systems and ongoing workings.

Golder Associates Brazil participated in the definition of pit slopes at the project development level between 1999 and 2001. In January 2005 international consultant Mr. Peter Stacey made a reviewed the site conditions and recommended a program of data collection to support a review of the slope design criteria and proposed the use of controlled blasting to improve excavation control. Since then a monitoring plan has been in place together with a controlled blasting program. There is currently a contract in place with a local geotechnical consulting firm to update the geological and geomechanical models using current structural information gathered mainly from bench mapping. Figure 8-27 shows the maximum general slopes angles used for the pit optimization.

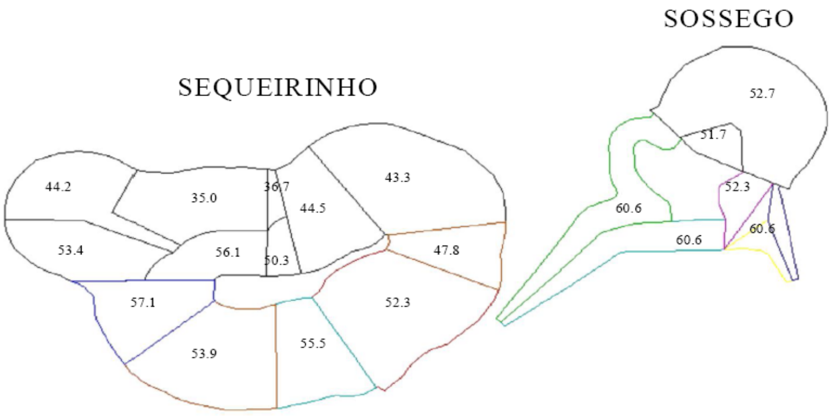


Figure 8-27: Maximum slope angles considered for pit optimization

Figure 8-28: Pit slope at the Siqueirinho Pit

The slope regimes for the Siqueirinho and Sossego pits are modeled appropriately during pit optimization and the pit slopes are considered a low risk area for the Mineral Reserves.

The Sequeirinho open pit will be approximately 500 m deep at completion. This is a very deep open pit excavation and extra care will need to be taken in the mining operations to ensure stability of the final pit walls to allow full extraction of the reserve.

Pit Optimisation

The most important aspect of mineral reserves estimation is to have available a suitable pit outline. For Sossego this was achieved by carrying out a Whittle optimization.

Metal Prices and Selling Costs

The prices and selling costs used in the optimization are presented in table Table 8-30. These values appear reasonable and were properly applied in the optimization process.

Table 8-30: Metal prices and selling costs used for pit optimisation

Item	Value
PrCu (US$/lb)	1.98
CvCu (US$/lb)	0.3812
PrAu (US$/oz)	700
CvAu (US$/oz)	0.01346

The Figure 8-29 shows the historical behaviour of the copper price over the last 10 years and the price of metal adopted for Vale for pit optimisation and economic evaluation. Figure 8-30 shows the same graph but for the price of gold.

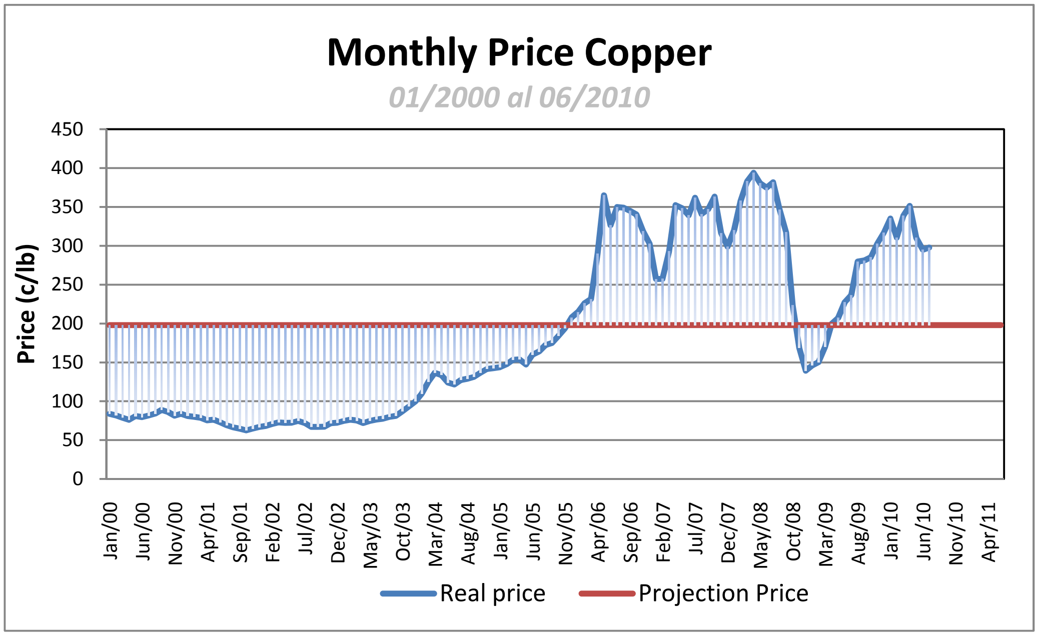


Figure 8-29: Historical behaviour of the copper price in the international market and long term price used for optimization

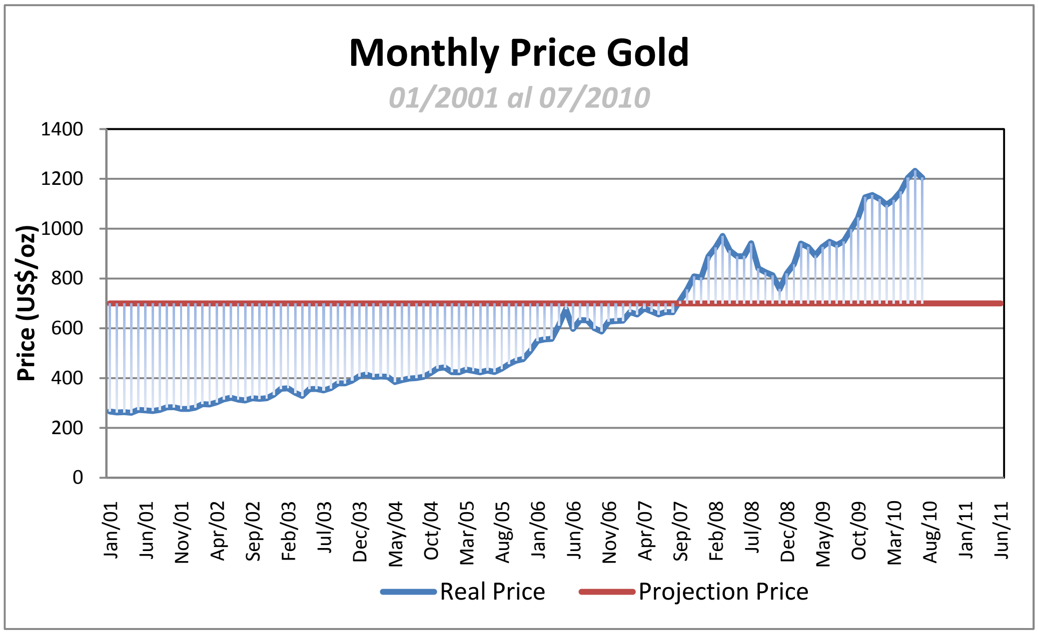


Figure 8-30: Historical behaviour of the gold price in the international market and long term price used for optimization

The copper and gold prices used for pit optimisation are considered appropriate for the development of a mineral reserve. In particular the values adopted meet generally accepted SEC guidelines which suggest using values that are less or equal the average price for the last 3 years.

Mining Cost

A base mining cost of 2.29 US$/t was used by Vale for pit optimisation. In addition a Mining Cost Adjustment Factor (MCAF) was applied to increase the cost with depth. For benches below the elevation 250, the increase is 0.0438 US$/t. Thus, the function of Whittle that satisfies this condition is

MCAF = R(IZ,1+(53-IZ)*0.04308/2.29,53,1.0)

Golder checked that the mining cost and adjustment factors have been properly transcribed into the Whittle parameter files. No problems or inconsistencies have been found between the parameter files and the documentation provided by Vale.

Processing Cost and Recoveries

The process cost used in the optimization includes the general costs and the administration costs. The process is affected by metallurgical recovery and flotation recovery and its values can be seen in Table 8-31.

Table 8-31: Processing cost and recoveries used for pit optimization

Processing Cost (US$/t)	10.48
Copper recovery (%)	(90.9997156-0.363825/(%Cu)*0.9616)
Gold recovery (%)	77.845

In the Figure 8-31 it is possible to appreciate the variation of the recoveries of gold and copper. In the case of copper, the highest grade present inside the block model is 8.3% which amounts a maximum recovery of 90.96%, for gold, the highest grade inside the block model is 2.59 g/t but a constant recovery of 77.85% has been adopted.

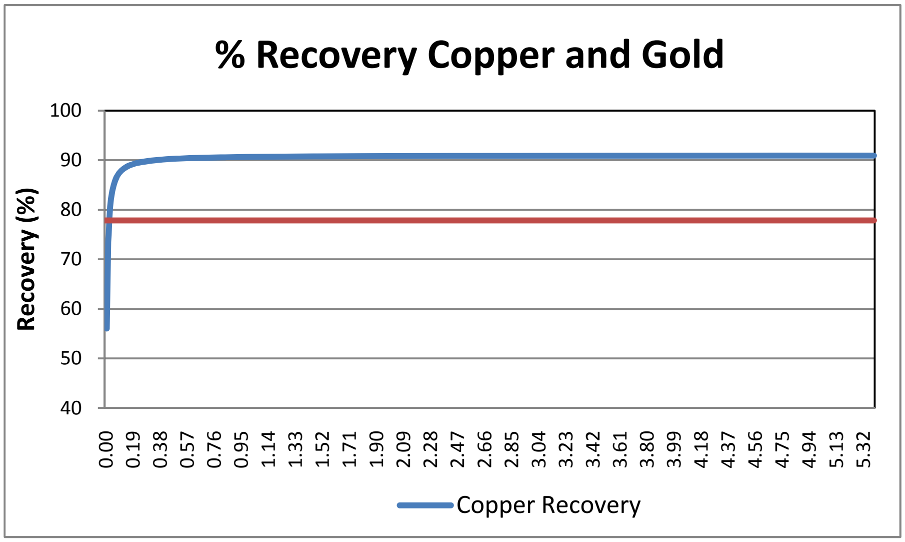


Figure 8-31: Copper and Golde flotation recovery

Slope Angles

The slope angles were properly applied into the Whittle model. Table 8-32 shows the slope angles defined according to the rock mass parameters.

Table 8-32: Slope angle by lithology

Lithologhy	**Granite**	**Biotite Schist**	
		Azimuth	Azimuth
		≈285° to ≈75°	≈240° to ≈285° and ≈75° to ≈120°
Bench Slope	75°	65°	60°
Inter-ramp slope angle	61°	56°	54°
Global slope	55°	52°	50°

The slope angles applied to the pit optimisation comply with geotechnical recommendations.

Optimization Results

The results were provided in the form of Whittle files. Golder independently carried out the optimisation in Whittle using the files provided and the results presented by Vale were exactly reproduced.

Figure 8-32 presents a pit by pit graph showing the undiscounted & discounted cash flow for the best case. It also includes the tonnage of ore and waste for each incremental pit shell. The revenue factor used in the optimization corresponded to prices from 0.89 US$/lb to 2.87 US$/lb.

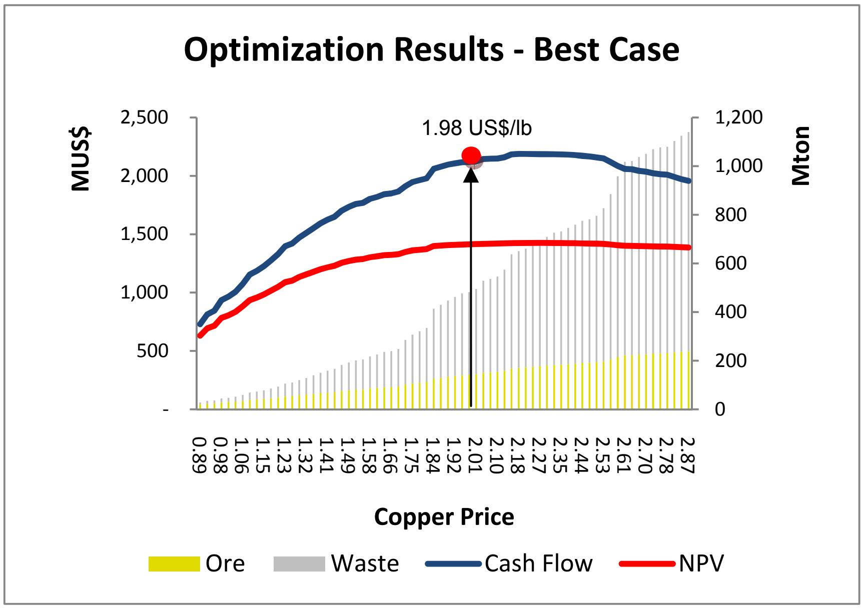


Figure 8-32: Optimization results: undiscounted cash flow and NPV

Selection Criteria

The final pit selected was the number 39 which corresponds to a copper price of 1.98 US$/lb. This pit includes a total of 141.2 Mt measured and indicated mineral resources with average grade of 0.97% copper and 0.27 g/t of gold. Table 8-33 presents the optimisation results for pit 39.

Table 8-33: Whittle results for the pit shell selected as the ultimate pit limit

Pit	Price (Cu)	REM	Total	Sulphide mineral			Contained Mineral			Custo Total	Total Cost	Incremental Cost	NPV	UCF
	US$/lb		Mt	Mt	%CuT	gpt Au	MtCu	MOz Au	Kt CuEq	MUS$	US$/lbEq	US$/lbEq	MUS$	MUS$
39	1.98	2.42	482.3	141.2	0.97	0.27	1,372.5	1205.7	1.565.8	3,759.8	1.09	1.57	1,412.8	2,119.1

Mine Design

After the ultimate pit limit has been selected it is necessary to make the final pit design and also the operational design for the pushbacks. Table 8-34 shows the tonnages of optimized and designed final pits. The results of the comparison show a large increase in the amount of waste incorporated into the operational design.

Table 8-34: Comparison between Whittle final pit and operational design

Item	Operational Pit	Whittle Pit	Increase
Total Material	630,428,817	482,269,231	31%
Waste	485,975,270	341,085,030	42%
Ore	144,453,547	141,184,201	2%
REM	3.36	2.42	39%

A visual comparison between the pit design and the Whittle shell is presented in Figure 8-33, Figure 8-34 and Figure 8-35 for Sossego, Pista and Siqueirinho respectively. These figures shows that the design follows in gross terms the optimisation outline but some differences may be noticed that are probably due to a marginal phase incorporated into the final pit.

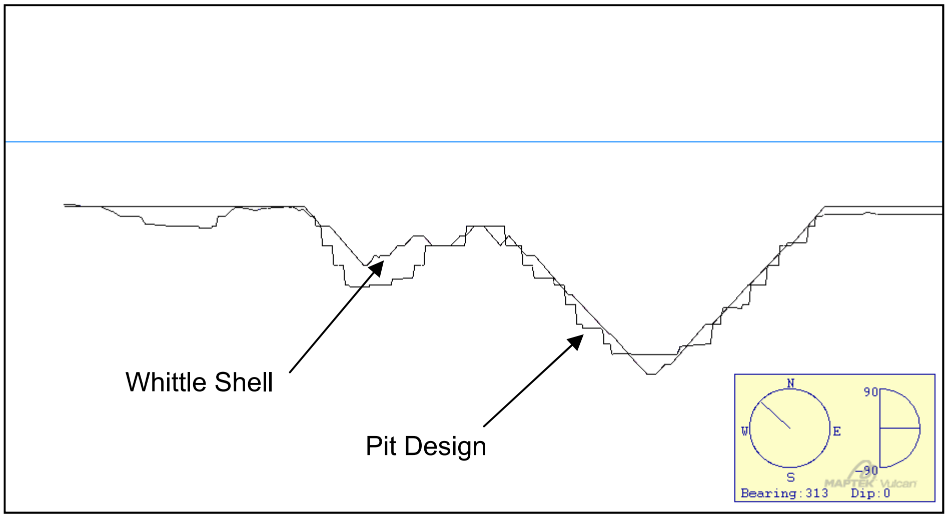


Figure 8-33: Comparison between pit design and Whittle shell - Sossego Pit

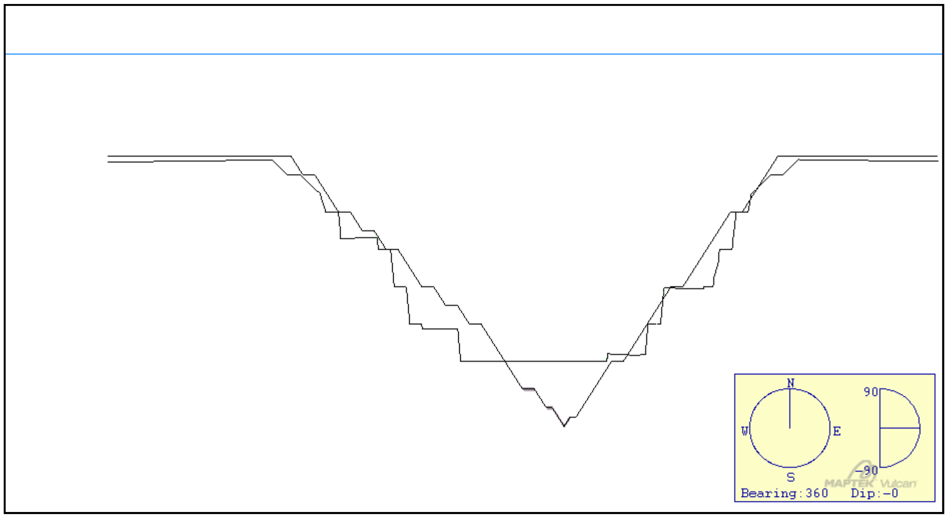


Figure 8-34: Comparison between pit design and Whittle shell - Pista Pit

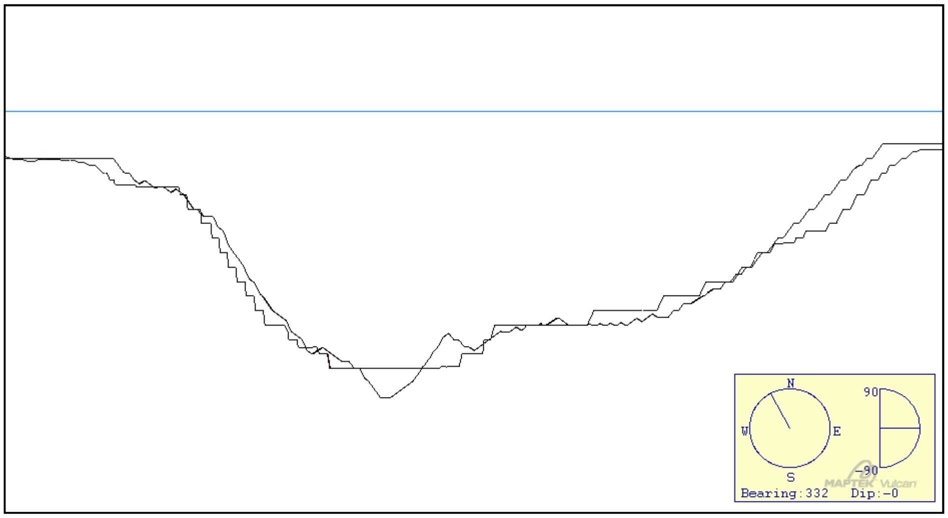


Figure 8-35: Comparison between pit design and Whittle shell - Siqueirinho Pit

The differences in terms of waste tonnage between the final pit design and the selected Whittle pit shell is considered excessive and should be reviewed in detail. The differences are probably due to a marginal phase incorporated in the mine design. In any case such level of differences should be properly documented with the appropriate explanations. There may be a considerable upside potential related to mine design optimisation.

The geotechnical recommendations have been properly applied in the mine design carried out by Vale.

Mining Equipment

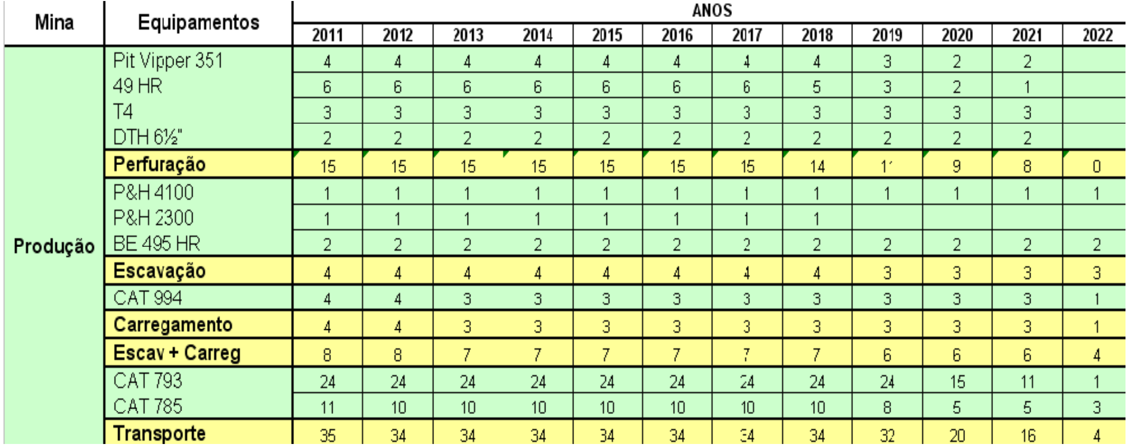
The mining operation at the Sossego Complex is composed by drilling, blasting, loading and hauling. The mining fleet also includes all additional support equipments necessary to maintain the working areas, mine access, waste dumps and stockpiles. Table 8-35 shows a summary of the equipment presently used at the Sossego Complex.

Table 8-35: Current mining equipment

Equipment Types	Quantity	Capacity	Average Work Hours	Productivity (t/h)
Loading				
BE 495 HR	2	73 jd^3	7,685	3413
P & H 4100 XPB	1	72 jd^3	3,667	3817
P & H 2300 XPA	1	18 jd^3 or 28 jd^3	3,481	1700
Cat 994 D	4	19 jd^3	18,448	1250
Hauling				
Cat 793 C	24	240 t	121,473	428
Cat 785 D	11	150 t	610,080	285
Drilling				
Pit Vipper 351	3	12 1/4"	11,417	1923
Sandivic Titton - Contracted	1	6 1/2"	7,389	-
Atlas Copco T4 BH - Vale	3	9 7/8"	11,558	-
BE 49 HR	6	12 1/4"	17,708	2810
Auxiliary				
Dozer Cat D11N	8	850 hp	36,849	
Dozer Cat D10 N	4	580 hp	22,639	
Wheel Dozer 988 B	2	475 hp	11,776	
Motor Grade CAT 16H	7	275 hp	30,000	
Hydraulic Excavator CAT 330 B	1	220 hp	4,618	
Hydraulic Excavator Cat 3545 B	2	321 hp	9,925	
Dozer CAT 854 G	3	800 hp	17,645	

Table 8-36 presents the fleet requirements from 2010 to 2022. It is noted that the fleet is basically constant until 2018. After this there is a gradual reduction at the end of the mine's life.

Table 8-36: Current mining equipment



Mina	Equipamentos	ANOS											
		2011	2012	2013	2014	2015	2016	2017	2018	2019	2020	2021	2022
Produção	Pit Vipper 351	4	4	4	4	4	4	4	4	3	2	2	
	49 HR	6	6	6	6	6	6	6	5	3	2	1	
	T4	3	3	3	3	3	3	3	3	3	3	3	
	DTH 6½"	2	2	2	2	2	2	2	2	2	2	2	
	Perfuração	15	15	15	15	15	15	15	14	11	9	8	0
	P&H 4100	1	1	1	1	1	1	1	1	1	1	1	1
	P&H 2300	1	1	1	1	1	1	1	1				
	BE 495 HR	2	2	2	2	2	2	2	2	2	2	2	2
	Escavação	4	4	4	4	4	4	4	4	3	3	3	3
	CAT 994	4	4	3	3	3	3	3	3	3	3	3	1
	Carregamento	4	4	3	3	3	3	3	3	3	3	3	1
	Escav + Carreg	8	8	7	7	7	7	7	7	6	6	6	4
	CAT 793	24	24	24	24	24	24	24	24	24	15	11	1
	CAT 785	11	10	10	10	10	10	10	10	8	5	5	3
	Transporte	35	34	34	34	34	34	34	34	32	20	16	4

The availability, utilization and productivity of the mining equipments have been estimated based on recent and historical information and although they are considered slightly below industry standard values, the numbers are considered acceptable for this type of hard rock mining.

The mining equipment fleet considered in LOM plan was reviewed and is considered suitable for purpose. The effectiveness of the mining fleet has been demonstrated over the last couple of years.

Mining Dilution and Ore Loss

A mine dilution factor of 5% was considered in the optimization process. Mining dilution factors are not applied for planning purposes i.e. estimated in situ tonnes and grades are directly applied to the mine plan.

The transformation of the original resource model into the mining model includes two types of dilutions:

1. The proportional model of waste and ore is used to calculate the final block grade; and
2. At the outer boarders of the units when a grade bearing unit is in contact with a barren unit additional dilution is applied to the copper grades.

Current reconciliation results indicate a good conformance of the long term model with production data. This indicates that the level of smoothing imposed by the grade estimation process and post-processing already accounts for some mine dilution that is being incurred by the operation.

Sossego personnel have indicated that a program is being initiated to assess dilution as part of the reconciliation process. Golder endorses this initiative.

Cut-off Grade

For the purpose of estimating the mineral reserves an optimized cut-off of 0.33% Cu was adopted. The following equation was used in the estimation of the marginal cut-off grade:

$$\text{Cut-Off} = CTP/\{(PrCu-CvCu) * RcCu * (100-PT) * 22.0462 * 100\}$$

Where:

CTP= Processing, G&A and Rehandling Cost (US$/ton)
PrCu = Copper Price (US$/lb)
CvCu = Selling Cost (US$/lb)
RcCu = Metallurgical recovery
PT = Lost of concentrate by transport

This equation defines the cut-off grade necessary to pay for the processing and administration costs, including 0.4 US$/t rehandling cost. Table 8-37 shows the values used for the cut-off grade calculation.

Table 8-37: Values used for cut-off grade calculation

Item	Value
PrCu (US$/lb)	1.98
CvCu (US$/lb)	0.3812
RcCu (%)	90.5
PT (%)	0.5
CTP (US$/t)	10.48
Cut Off (%)	**0.33**

The average recovery was obtained from the following equation:

$$RcCu = f(Cu) = (90.9997156 - 0.363825/(\%\ Cu) * 0.9616)$$

Where Cu is the average grade of the block model, Cu = 0.675%.

The cut-off grade of 0.33% of copper was used to estimate the LOM plan and was applied over the equivalent copper grade. The estimation was based on costs, prices and metallurgical recoveries. The general expression for the equivalent copper grade (Cu_Eq) is the following:

$$Cu\_Eq = Cu + \left[Au * \frac{(\Pr Au - CvAu) * RCAu * RFAu}{(\Pr Cu - CvCu) * RCCu * RFCu} * \frac{31.103}{22.0462} \right]$$

Where:

Cu = Copper grade (%)

Au = Gold grade (g/t)

PrAu = Gold selling price (US$/oz)

CvAu = Gold selling cost (US$/oz)

RCAu= Gold flotation recovery (%)

RFAu= Gold smelting recovery (%)

PrCu = Copper selling price (US$/lb)

CvCu = Copper selling cost (US$/lb)

RCCu= Copper flotation recovery (%)

RFCu= Copper smelting recovery (%)

31.103 = Conversion factor: oz to grams

22.0462 = Conversion factor: ton to lb

The cut-off grade strategy used was based on the algorithm of K. Lane 1964, so as to maximize the net present value of the exploitation, respecting the limitations of movements, processing capacity and the demand for concentrate. Table 8-38 presents the cut-off grade strategy by period. The mineralization between the internal and the optimized cut-offs is stockpiled for later processing

Table 8-38: Variable cut-off grade strategy for the LOM plan

Period	**Cut Off**		
	SEQ	SOS	PISTA
2011	0.33	0.33	0.33
2012	0.40	0.40	0.40
2013	0.40	0.40	0.40
2014	0.40	0.40	0.40
2015	0.40	0.33	0.40
2016	0.33	0.33	0.33
2017	0.33	0.33	0.33
2018	0.33	0.33	0.33
2019	0.33	0.33	0.33
2020	0.33	0.33	0.33
2021	0.33	0.33	0.33
2022	0.33	0.33	0.33

8.14 Reported Mineral Reserves

Golder Associates S.A. (Golder) has audited the mineral reserve estimates produced by Vale personnel for the Sossego mining complex and has found that the figures provided are appropriate for public reporting under international standards such as the United States Securities and Exchange Comission (SEC) Industry Guide 7 and the current NI 43-101. Table 8-39 presents the Sossego reserve at the appropriate level of precision for public reporting.

Table 8-39: Estimated Mineral Reserve as at 30 June 2010

Mine	Mineralization Type	Proven			Probable			Proven and Probable		
		Tonnage (Mt)	Cu (%)	Au (g/t)	Tonnage (Mt)	Cu (%)	Au (g/t)	Tonnage (Mt)	Cu (%)	Au (g/t)
Sequeirinho	Sulphide	88.64	0.99	0.28	26.72	0.92	0.24	115.35	0.97	0.27
	Mixed Ore	0.15	0.40	0.03	0.16	0.40	0.08	0.30	0.40	0.05
	Total	**88.78**	**0.99**	**0.28**	**26.87**	**0.92**	**0.24**	**115.66**	**0.97**	**0.27**
Pista Sector	Sulphide	7.56	0.80	0.10	1.25	0.75	0.09	8.81	0.79	0.10
	Mixed	0.08	0.40	0.01	0.09	0.64	0.09	0.17	0.53	0.05
	Total	**7.64**	**0.80**	**0.10**	**1.34**	**0.74**	**0.09**	**8.97**	**0.79**	**0.10**
Sossego	Sulphide	4.35	1.03	0.33	11.62	0.80	0.24	15.97	0.86	0.26
	Mixed	0.00	1.12	0.17	0.01	0.76	0.27	0.01	0.78	0.26
	Total	**4.35**	**1.03**	**0.33**	**11.63**	**0.80**	**0.24**	**15.98**	**0.86**	**0.26**
Stockpile	**Total**	**18.8**	**0.42**	**0.27**	**-**	**-**	**-**	**18.8**	**0.42**	**0.27**
Sossego Complex	**Total**	**119.6**	**0.89**	**0.27**	**39.83**	**0.88**	**0.23**	**159.4**	**0.88**	**0.26**

Golder accepts the procedure adopted by Vale to convert mineral resources into mineral reserves. The numbers have been checked and are appropriate for the purpose of public reporting in that the mineral reserves provide an acceptable prediction of the available material expected from mining.

Conversion of the mineral resource estimate to a mineral reserve is based on appropriate mine design and planning. In particular, dilution and mine recovery are supported by historical data. The tonnes and grades are reported at an appropriate economic cut-off grade.

8.15 Reconciliation and Reserve Audits

Previous Audits

Golder participated on earlier studies of the Sossego project in the areas of tailings and geotechnical engineering. In 2004 Golder carried out an Audit of the Sossego mineral reserve as part of the Corporate Vale audits to comply with SEC. The Ore Evaluation Services team of Golder Associates has not had prior involvement in resource estimation and mine planning related to the Sossego mineral reserves.

Auditing of the reported mineral resources and mineral reserves of the Vale's properties began in 1998 in support of the filing of an F-3 Form with the United States Securities and Exchange Commission (SEC) as a requirement of the initial listing and public offering of Vale shares on the New York Stock Exchange.

In 1999 an audit was carried out with pre-feasibility study purposes, the external auditor was the U.S. based company Mineral Resources Development, Inc. (MRDI), . MRDI was acquired by AMEC in May 2000.

Feasibility studies during 2001 and 2002 were audited by KVAERNER and AMEC Plc respectively, mainly with the purpose to validate resource model updates. In 2003 Snowden group completed an audit with a complete database and resource model validation.

Golder Associates was engaged to provide an audit of mineral reserve estimates to Sossego Copper Mine, as of December 31, 2004. AMEC Plc. completed one mineral reserve audit for the Sossego Mine in early 2006. These reports were prepared in support of Vale's 20F Annual Report filing with the United States Securities and Exchange Commission (SEC).

Pincock, Allen & Holt (PAH) was retained to provide mineral reserve statement for Sossego Mine in February 2008, based on a review of Amec's 2005 report and mineral reserve reconciliation. This report was prepared in support of Vale's 20F Annual Report filing with the United States Securities and Exchange Commission (SEC).

Reconciliation

The reconciliation for the Sossego mine is done using mining call factors and includes reconciling the following sources of information: Long Term Model (LTM); Short Term Model (STM); Production (PRD); Total ore Sent to Crusher (TSC) and Processed Ore (PO). The control factors are obtained using the same base, where tones and metal content are expressed in dry basis. The factors are calculated as follow:

- F1 = STM / LTM
- F2 = PRD / STM
- F3 = PO / TSC

The following figure shows the mining call factors of Sossego Mining Complex, considering the topographic surface at 31 December 2009. The blue bar measures the tonnage, the red bar the copper grades and green bar the copper content. The red control line is accepted by Vale's Internal Reserve Committee and the blue control line is accepted by DIOC.

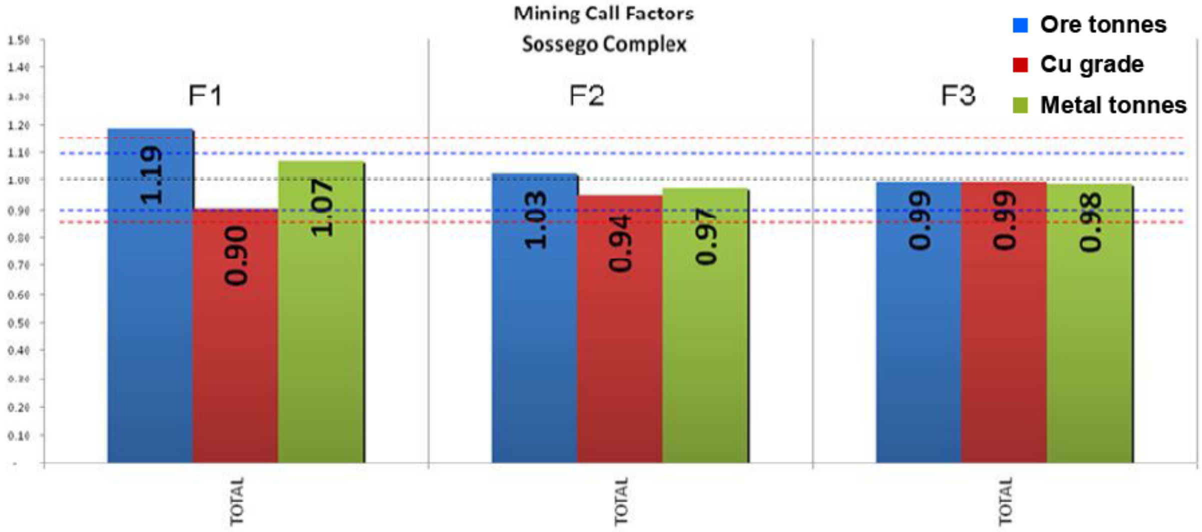


Figure 8-36: Reconciliation results for 2009

Reconciliation results for 2009 indicate good conformance of planned versus realised production. The reconciliation process is considered to be of high standard.

Grade Control

Grade control at the Sossego mine is based on blast holes data. The locations of these holes are obtained with high precision GPS by the surveyors. The blast hole sample is taken with a probe of approximately 40 cm in length and 8 cm in diameter. The probe is inserted approximately 4 times in each quadrant of the cuttings to obtain an approximately 4 kg sample. The sub-drill is not scraped off the cuttings before the sample is taken, but is included in the sample. The cuttings are placed in a plastic box and homogenized by shaking the box vigorously. The homogenized cuttings are then run through a Jones splitter and separated into two samples, bagged and numbered. The final sample is approximately 2 kg. In this manner, duplicate samples are created in the field.

Not all the blast holes are sampled. A geologist is responsible for determining which samples are taken. The samples are sent to the laboratory in the process plant for assaying. It takes approximately 8 hours for copper assays, 24 hours for the gold assays and 36 hours for the additional elements (Ag, Pt, Cl, and F). The assay results are forwarded to the Geology Department where they are placed into the ore control database.

Geology has responsibility for creating mieralized polygons to export to the Dispatch system. These polygons also cover the area where the material was thrown from the free-face blasting.

Production Control

The dispatch system is used to control activities of all mine equipment. Low precision GPS is used for trucks and support equipment. High precision GPS is used for drill rigs and shovels. There is no physical marking of ore and waste at the bench as the shovel operators have online access to the dig limits on screen.

Statistics regarding production, delays and availabilities are maintained in the Dispatch database. The system is run with one dispatcher per shift located at the processing plant control room. The system seems to be well operated and maintained.

Production of the haulage fleet comes from the VIMS system of the Caterpillar trucks. This records the loads on the truck and transmits the information to the Dispatch server. If the scale on the truck does not work, a representative load factor from past historical data is used.

Daily reports are generated for management regarding production statistics.

8.16 Environmental

Golder personnel interviewed Sossego's environmental team during the site visit in reference to the environmental aspects of the mine.

Vale holds all environmental permits required by Brazilian legislation to operate the Sossego mine. No fatal flaws regarding environmental aspects of the Sossego operation have been identified by Golder.

The Sossego operation manages environmental responsibilities and liabilities appropriately.

8.17 Community and Government Affairs

No Issues have been identified in relation to local communities and government affairs that represent a risk to the Mineral Reserves for the Sossego mining complex. The local communities and both the state and federal governments are supportive of mining activities in the region.

Vale has an ongoing community relations program in place that is related with the Sossego mining complex. No major conflicts are currently registered with either neighbours or other third parties.

The closest major cities are Canaa dos Carajás and Parauapebas (85 Km), the latter is the fastest growing town in Brazil due to the strong presence of mining.

8.18 Operating Costs

Both pit optimization and economic analysis use current operating costs as a long term estimates. Consumable costs are based on current contracts for materials including delivery costs. Operating costs estimated by Vale for the Sossego operation are presented in Table 8-40.

Table 8-40: Operating costs

Copper Price	US$2.73/lb
Mining Cost	3.33 USD/t
Processing Cost	8.74 USD/t
Logistics	37 USD/t conc.

8.19 Capital Costs

A total Capex of US$ 600M has been estimated for equipment replacement and other sustaining capital costs for the life of mine. This cost has been properly accounted for in the economic evaluation.

Golder has reviewed current cost information from the Sossego operation. The methodology used for the estimation of long term operating and capital costs meet industry standards.

8.20 Taxation

In Brazil, there are seven different taxes, duties and Royalties that are levied by the Federal, Provincial or Municipal entities. The legal taxation on the cash flow is the CFEM (Corporate Income Tax) and the CSLL (Social Contribution on Corporate Profits) that is also an income tax. The CFEM is variable for each mineral commodity and for the Nickel is 2% over the gross revenue (less the transportation costs). Furthermore, taxes are applied differently depending on whether the product is intended for the internal or external markets. In the case of Onça Puma the assumption is that 90% of the nickel product will be sold on external markets, with the balance sold on internal markets.

The income tax is calculated based on the gross profits (revenues less operating costs and depreciation) and is fixed in 25%. The CSLL is fixed in 9%. The combined rates for all levies equates to approximately 15% for internal product and between 15 and 20% for external products

8.21 Economic Evaluation of Mineral Reserves

Golder consultants were not provided with a copy of the Sossego discounted cash flow (DCF) spreadsheet model; however, Golder was permitted to review and audit the DCF model on secure Vale computers to gain an understanding of the model, to assess its correctness and to test project sensitivities to key input variables.

Key Assumptions

A summary of the key parameters used in the economic analysis for the Sossego operation is presented in External Audit of Mineral Reserves, Volume 1, Consolidated Report, Key Assumptions.

Sossego Cash Flow Evaluation

The cash flow forecast is based on the June 2010 update of the 2009 Model, including mineral reserve depletion year-to-date; a review of pit optimisation; and increased operating costs, but which also reflects the following assumptions:

- The financial calculations are based on an after tax discount rate.
- Taxes are calculated per the discussion in Section 8:20 of this report. Tax holidays, deferrals, and recoveries are included in the economic model, where applicable and appropriate
- All costs and prices are in un-escalated “real” dollar terms

- The operating costs include both fixed and variable cash mining costs, based on the mine plans, and milling and delivery variable cash costs and are assumed to be based on the 2010 actual costs to the end of June
- Closure cash costs are included as annual capital instalments, with a lump sum payment following the completion of the life of mine plan
- Unit cost assumptions are based on a defined metal throughput for the 2009 Plan (not reviewed by Golder)
- Future unit cost assumptions assume similar metal production
- Capital costs include forecast expenditures for all relevant departments
- Production is based on the Sossego mineral reserves only; no external feeds or concentrates have been included in this economic analysis
- Mill recoveries for copper are based on a mill model, with factors updated to match the production plan (not reviewed by Golder)
- Revenue is calculated from the recoverable metal and the long term forecast of metal prices and exchange rate, based on SEC reporting requirements (three-year moving average prices). Revenue from the sale of a copper concentrate is included, based on the contained metal, accountability factors and the long term forecast for metals prices and exchange rates.

Sensitivity analysis

Golder was permitted to review and audit the DCF model on secure Vale computers to gain an understanding of the model and to assess its correctness and to test project sensitivities to key input variables.

It was observed that the model contained construction costs, reclamation and closure costs, detailed federal and provincial tax sheets, sustaining capital allowances, and the correct schedule. The base case cost and price assumptions have been updated since the release of the 2009 Financial Model, and these changes are reflected within the Financial Model.

Base case cash flows were observed for individual years using the three-year moving average price assumption scenario. Using the DCF spreadsheet, significant changes were made to price and cost assumptions to test the robustness of project economics. As the models were not made available to Golder, detailed sensitivity analysis was not possible; however, the cases tested involved making +/-20% changes, in five percentage point increments, to nickel price, capital expenditure, operating costs and foreign exchange. Furthermore, Golder tested the effect of changes in discount rate between 6% and 10%, in increments of half a percentage point.

The results are presented in Figure 8-37.

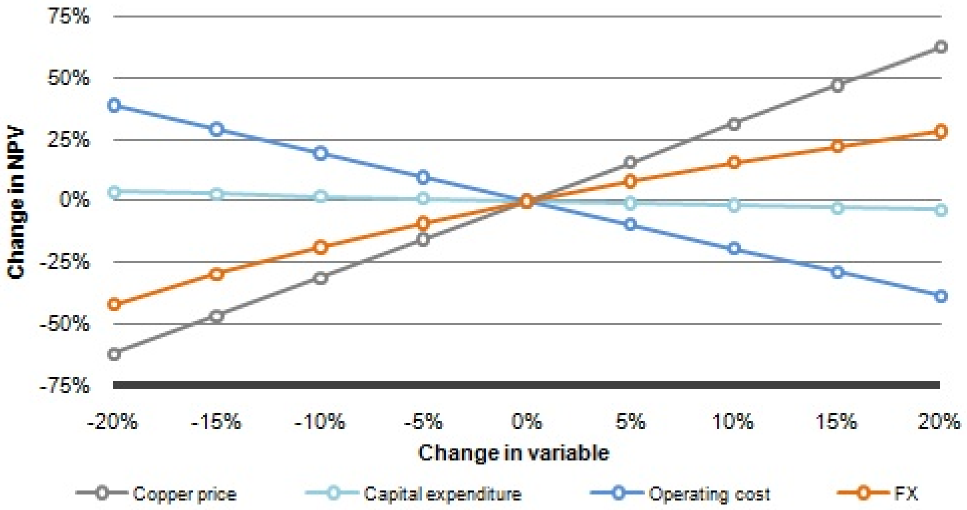


Figure 8-37: Sossego sensitivity analysis

The NPV was most highly sensitive to copper price, with other variables having a lesser, though still significant effect on the NPV. Copper price in considered a highly significant value driver. The NPV was least sensitive to capital expenditure.

Conclusions and recommendations

In both cost and pricing assumptions scenarios used (Vale and three-year moving average), positive project economics support conversion of mineral resources to mineral reserves. Under sensitivity analysis, in all cases tested the NPV remained positive, suggesting robust project economics.

8.22 Mine life

The LOM plan is the input to a cash flow analysis demonstrating the economic viability of the operation and therefore the Mineral Reserve. The LOM plan for the Sossego mining complex gives a life of mine of 12 years based entirely on Mineral Reserves.

Table 8-41 presents the life of mine plan for the Sossego Mine covering the period of March 2010 to December 2021.

Table 8-41: Life Of Mine plan for the Sossego Mine - March 2010 to December 2021

	2010	2011	2012	2013	2014	2015	2016	2017	2018	2019	2020	2021	2022
Total Movement (ktpa)	59,290	70,051	70,000	70,000	70,000	70,000	70,000	70,000	60,000	40,000	25,000	20,000	9,500
Waste Removal (ktpa)	42,162	50,424	49,927	48,160	47,644	48,522	50,880	56,029	44,729	22,014	8,746	5,738	-
Ore Production (ktpa)	17,128	19,627	20,073	21,840	22,356	21,478	19,120	13,971	15,271	17,986	16,254	14,263	9,500
OMs (ktpa)	5,172	7,164	6,349	6,233	6,638	6,301	6,120	5,359	6,700	4,986	3,800	5,425	9,500
Stripping Ratio	3.53	4.05	4.07	3.72	3.48	3.20	3.91	6.51	5.22	1.69	0.70	0.65	0.00
Plant Feed Mt (dmt)	9.88	12.95	13.00	13.00	13.00	13.00	13.00	13.00	13.00	13.00	13.00	13.00	13.00
Feed Grade (%Cu)	0.97	0.92	0.91	1.04	0.98	0.95	0.91	0.71	0.75	0.98	0.98	0.94	0.35
Feed Grade (%Au)	0.22	0.24	0.27	0.31	0.26	0.23	0.06	0.04	0.05	0.11	0.16	0.18	0.09
Concentrate (tbs)	296,756	360,658	358,292	409,318	388,665	374,856	359,069	280,421	294,779	385,779	387,423	372,223	96,467
Concentrate Cu Grade (%)	29.71	29.70	29.70	29.70	29.70	29.70	29.70	29.70	29.70	29.70	29.70	29.70	29.70
Concentrate Au Grade (g/t)	5.9	7.7	8.1	8.1	7.6	7.3	4.8	5.4	5.4	5.3	6.1	6.5	8.0
Recovery	90.4	89.1	89.1	89.3	89.2	89.1	89.1	88.7	88.8	89.2	89.2	89.1	86.7

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