NI43-101 Technical Report

Prefeasibility Study (PFS) for the Araguaia Nickel Project

Pará State, Brazil

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1 Summary

1.1 Introduction

This report is a National Instrument 43-101 (NI 43-101) Technical Report on the Pre-Feasibility Study (PFS) for the Araguaia Nickel Project (the Project), a mineral development project located on the eastern margin of the State of Pará, to the north of the town of Conceição do Araguaia in the country of Brazil. The Project is 100% owned by Horizonte Minerals Plc (HZM). This Technical Report has been prepared for HZM by Snowden Mining Industry Consultants Limited (Snowden). IGEO Mineração Inteligente Ltda (IGEO), KH Morgan and Associates (KHM) and Prime Resources (Pty) Ltd (Prime) have also contributed,the work by IGEO and KHM has been reviewed by Snowden and Prime.

This Technical Report has been prepared for HZM by or under the supervision of qualified persons within the meaning of NI 43-101 Standards of Disclosure for Mineral Projects in support of HZM's disclosure of scientific and technical information for the Project.

This study was conducted with the objective of evaluating the economic viability of the Project to produce ferronickel (Fe-Ni). This Technical Report summarises the geological, hydrological and engineering studies performed at a PFS level (± 25% accuracy) and used in the economic evaluation of the Project.

The engineering design solutions offered in this Technical Report are considered industry standard practices. The mining of nickel laterites is typically via open pit configurations which involve well-developed mining practices and earthmoving machine applications. This study considers the open pit configuration and builds upon this knowledge for the exploitation of nickel laterites to establish the production of ROM from seven open pits which supply a targeted 0.9 million tonnes per annum (Mtpa) of ore to a processing and smelter facility that uses the RKEF process with the product being sold at the mine gate. Initially, two production scenarios were considered:

- 2.7 Mtpa (contractor and owners mining and haulage)
- 0.9 Mtpa (contractor and owners mining and haulage)

The Base Case of 0.9 Mtpa was selected based as a consequence of HZM's desire to minimize the capital expenditure and overall capital intensity, and to optimize overall cash flow, payback, and the economics of the Project. Opportunity does exist to increase and expand production subject to further engineering and a potential to increase in reserve base. The Base Case for this study assumes an ore processing rate of 0.9 Mtpa after a two year initial ramp up period. A plant construction period of two years has been assumed and the pre-production capital construction costs for the plant have been divided 30%, in Year 1, and 70%, in Year 2. In addition, sustaining capital has been provided for over the life of the mine and process plant. To minimise capital, the Base Case also assumes contractor mining which includes ore haulage to the plant. Supply chain factors have also been considered for in-bound and out-bound logistics for key consumables such as coal for smelter requirements.

The economic analysis contained in this Technical Report is based on Probable Reserve estimates. All dollar values are in United States Dollares (USD).

1.2 Property location, accessibility and climate

The Project is centred approximately 07° 37'S and 49° 24'E (675000E/9120000N) (Figure 1.1) and extends marginally across the south-eastern border of Pará State with Tocantins State. This location is approximately 40 km north of Conceição do Araguaia (population of 46,206) and is accessed by a network of unsealed roads branching eastward from the unsealed Conceição do Araguaia– Floresta road (PA 449) that passes through the Project. The area has a close reticulated system of earth roads servicing numerous cattle properties.

The Project can be reached by air from São Paulo via Palmas, the capital of Tocantins State situated to the east of Rio Araguaia. From there it is a further 400 km drive on mainly sealed highways to the main Araguaia field office in Conceição de Araguaia. Local flights are supported by airports at Palmas (Tocantins State), and Redenção via Belém/Marabá.

In the Araguaia region, the rain season mostly extends from October to May with approximately 1,754 mm of average annual fall. An average temperature of around 25° C is maintained throughout the year. Average maximum of approximately 35° C occurs in August and September with an average minimum of approximately 20° C from June to August. Summer rain results at the southern edge of the equatorial Intertropical Convergence Zone (ITCZ). The seasonal shift of the ITCZ gives rise to a wet and dry season to the Araguaia region. Short periods of rain can occur in the winter dry season through northward extensions of the Polar Front. Rainfall average for Conceição do Araguaia is shown on Figure 1.2 and temperature on Figure 1.3.

Figure 1.1 Location map and local infrastructure

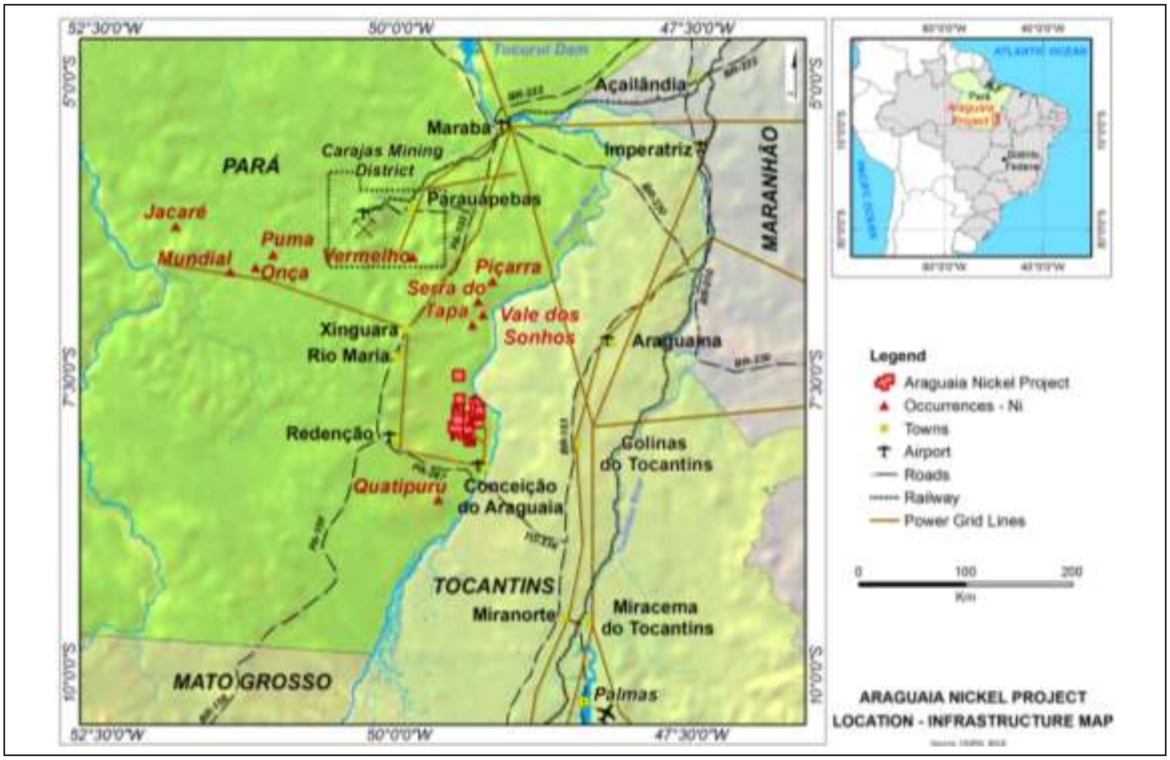


Figure 1.2 Average monthly rainfall, Conceição do Araguaia

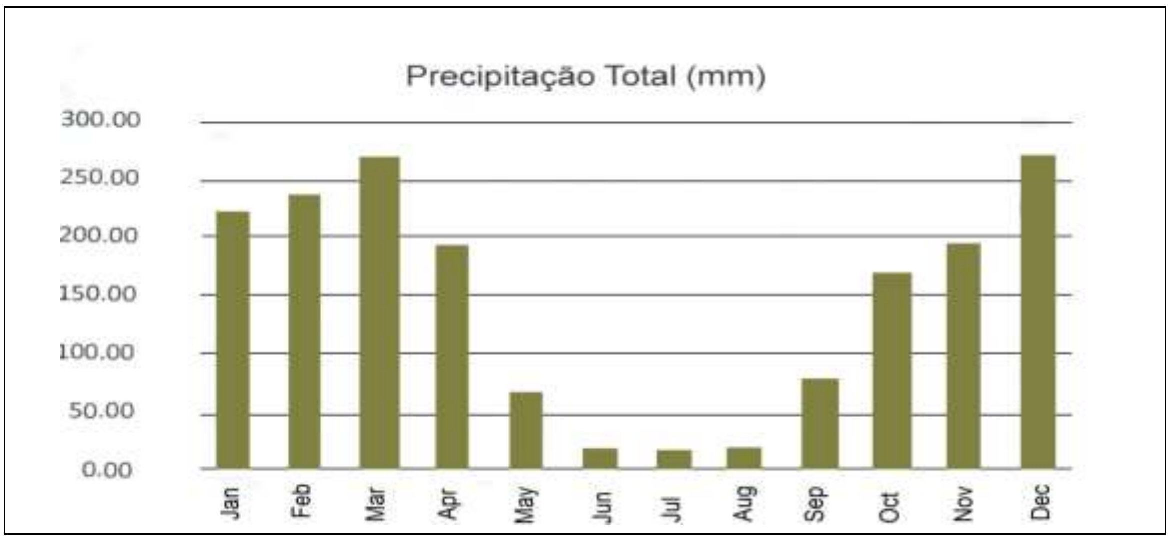
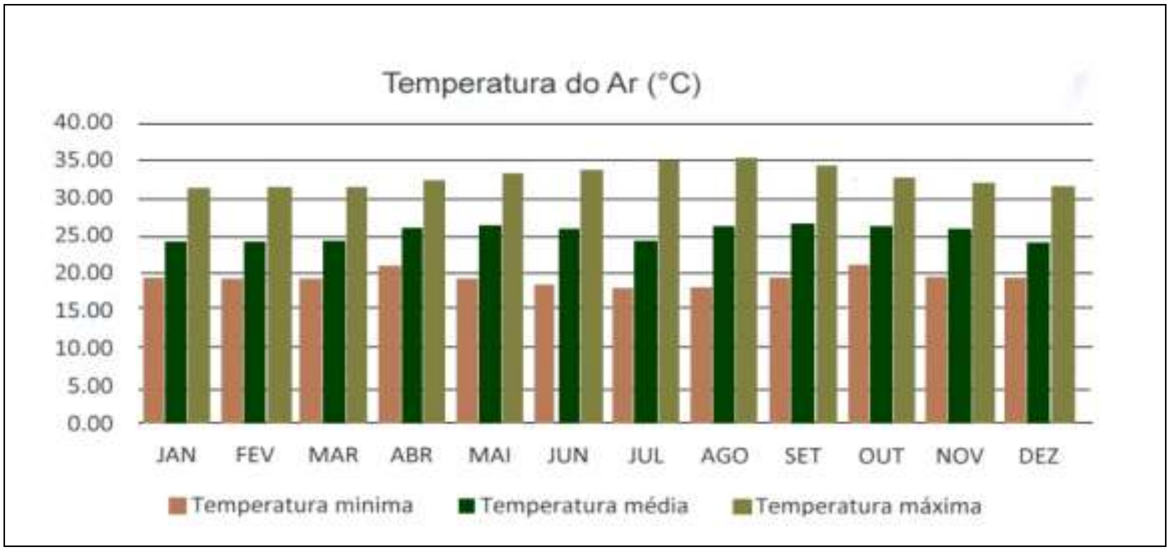


Figure 1.3 Monthly temperature, Conceição do Araguaia



1.3 Property ownership

The Project is comprised of 14 exploration licences in two non-contiguous blocks within the prominent Araguaia Nickel Araguaia Nickel Belt located in Pará state, Brazil. The licences have been acquired by HZM through application and acquisition. Lontra was a discovery by HZM and the Araguaia licences were acquired through the purchase of Teck Resources Limited's adjacent Araguaia project. The Floresta and Villa Oito licences were acquired via share purchase from Lara Exploration. All licences are now held under one Brazilian company 100% owned by HZM.

1.4 Property description

The Project covers an area of 118,557 Ha. Topography of the area, defined in the quadrangle 9,110,000N to 9,135,000N and 670,000E to 685,000E, is considered mildly undulating with various rounded hill features ranging in elevations of 217 m (AMSL) to 360 m (AMSL). Figure 1.4 provides a generalised view of the topography in the Project area.

More than 60 % of the Project is cleared of vegetation for open paddock cattle grazing. The area was never primary rainforest and is termed cerrado in Brazil.

Figure 1.4 Typical topography of project area

Source: KHM 2013

1.5 Geology and mineralisation

The deposits of the Project are typical examples of nickel laterites formed in a seasonally wet tropical climate on weathered and partially serpentinised peridotite. The nickel in such deposits is derived from altered olivine, pyroxene and serpentine that constitute the bulk of tectonically emplaced ultramafic oceanic crust and upper mantle rocks.

Laterisation of these serpentinised peridotite bodies occurred during the Tertiary period and the residual products have been preserved as laterite profiles over plateaus/amphitheatres, elevated terraces and ridges/spurs.

Supergene concentration of the nickel by leaching from the limonite zone and enrichment in the underlying saprolite zones is also common. The degree of the nickel concentration and the detailed type of regolith profile developed is determined by several factors including climate, topography, drainage, tectonism and the mineralogy and structures in the parent rock.

A nickel laterite deposit profile at the Project typically consists from surface to bedrock of:

A Soil Horizon – 0.6 m to 1.6 m average thickness

Ferricrete Horizon (including unconsolidated and cemented types, iron cap and pisolites) – 0.6 m to 4.3 m average thickness

Limonite Horizon (red and yellow types) – 7.5 m to 11.6 m average thickness

Transition Horizon (upper plastic, green and brown types depending on the ratio of nontronite, goethite and manganese minerals) – 3.2 m to 6.3 m average thickness

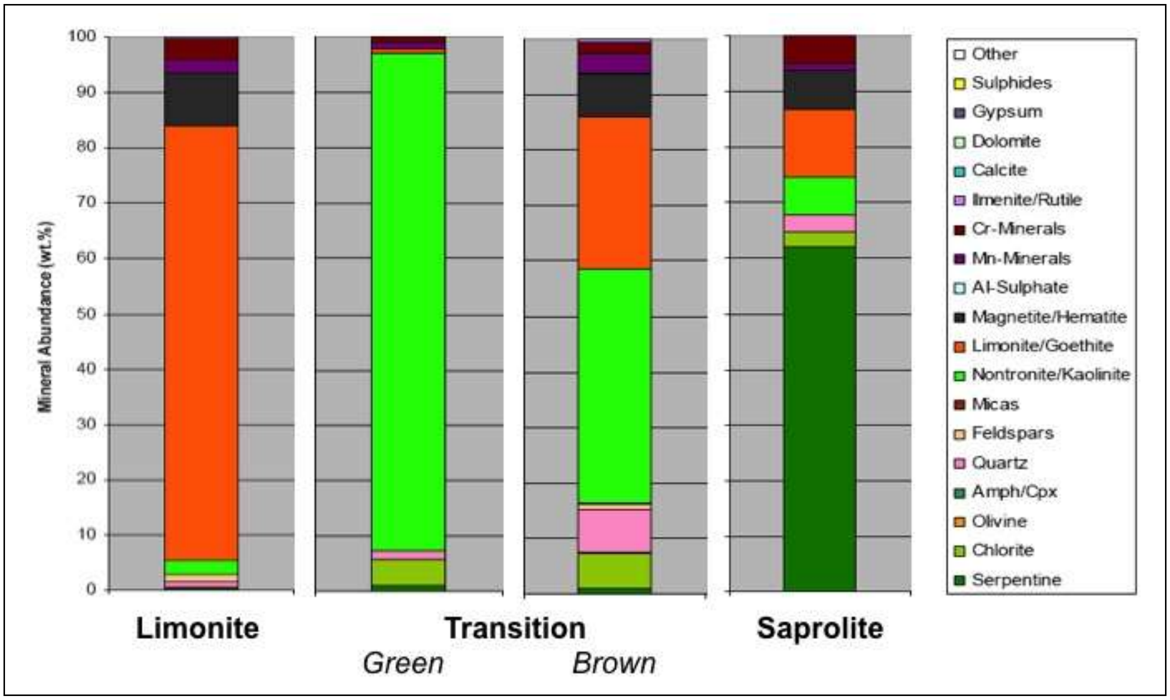
Saprolite Horizon (earthy, rocky and silicified types) – 5.2 m to 10 m average thickness.

The mineralogy of the main mineralised horizons is provided in Figure 1.5.

Eighteen nickel laterite targets are located in the Project, with Mineral Resources estimated for fourteen deposits. Of these, seven are sufficiently drilled and sampled to permit the declaration of Indicated Mineral Resources, and these are the basis for the PFS.

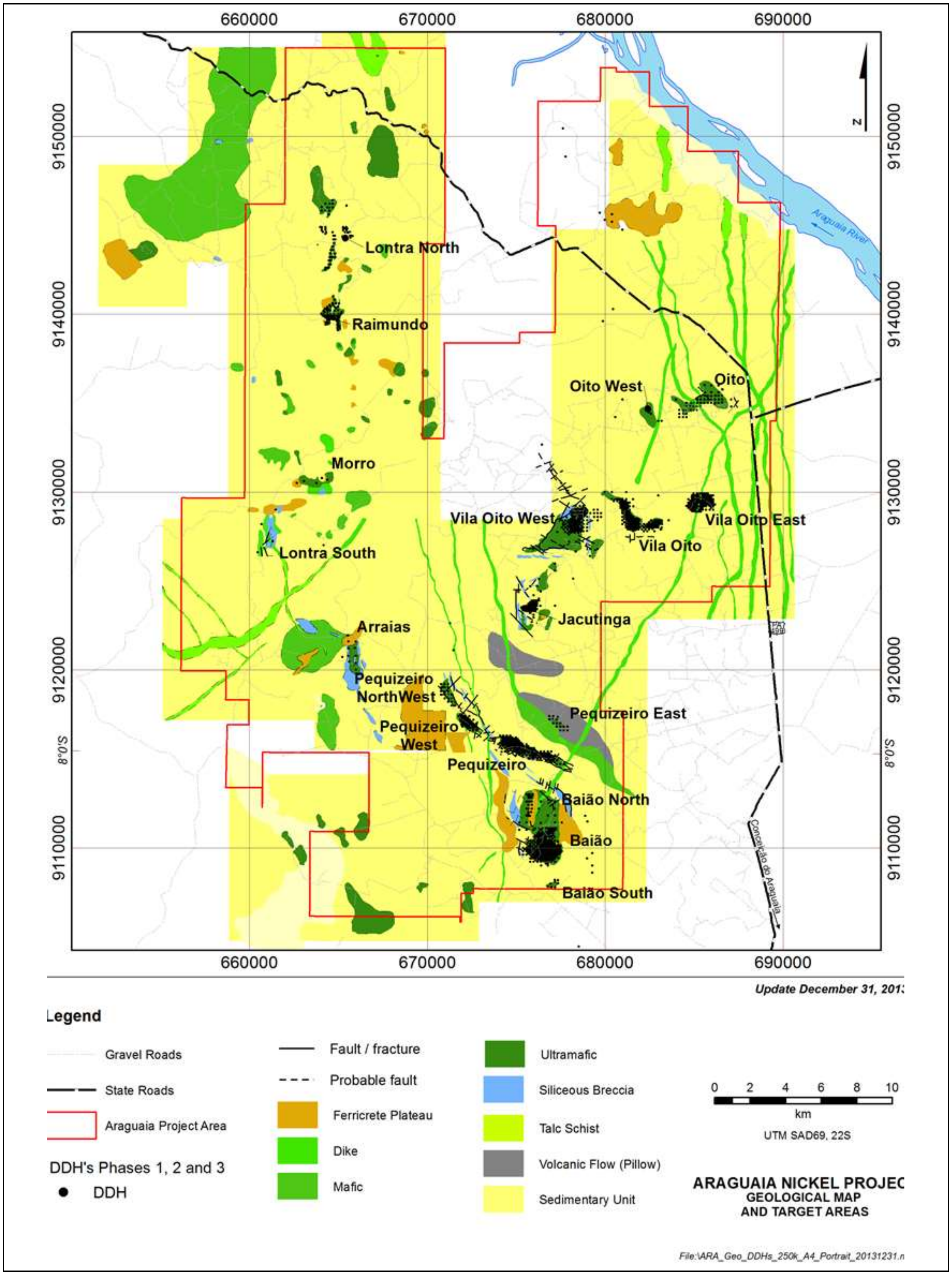
From North to South, the seven PFS deposits are: Vila Oito East (VOE); Vila Oito (VOI); Vila Oito West (VOW); Jacutinga (JAC); Pequizeiro West (PQW); Pequizeiro (PQZ), and Baião (BAI) identified in Figure 1.6.

Figure 1.5 Mineralogical distribution in the principal mineralised horizons



Source: SGS 2011

Figure 1.6 Simplified geology map



Source: HZM, 2013

1.6 Drilling

Diamond core drilling programmes undertaken by Teck and HZM totalling 1,120 drillholes for a total length of 29.1 km are the basis for Mineral Resource estimates for the PFS. Auger drilling and reverse circulation (RC) drilling has also been undertaken for exploration purposes however data from these has not been used for Mineral Resource estimates.

First pass irregular spaced exploratory RC drilling was undertaken by Teck between September and November 2006 to test nickel in soil geochemical, airborne geophysical anomalies and identified target areas.

Following positive results from the RC drill programmes, 400 m x 400 m spaced diamond drilling took place at the Baião, Pequizeiro, Jacutinga, Vila Oito West, Vila Oito, Vila Oito East and Oito targets between April and November 2007.

Where preliminary results from drill core were positive, 200 m x 200 m spaced diamond drilling was undertaken. In November 2008, having completed the diamond drilling over selected targets, for a total of 489 holes and 11,404 m, Teck ceased exploration on the project

In 2008 HZM initiated the first of three phases of a diamond drilling programme. In total 63 diamond drillholes were completed totalling 1,299.5 m to test the Northern and Raimundo Zone target anomalies. The programme consisted of 31 holes completed on the Northern anomaly; 31 holes completed on the Raimundo anomaly; and 1 exploratory hole on the Southern anomaly.

Within the programme vertical holes were drilled to 15-25 m in depth, ensuring that the saprock-fresh rock interface was intersected. Drillhole spacing was as follows: on 400 m spaced lines with 80 m hole centres (for geological sections and interpretation); on 200 m x 200 m centres (for resource potential identification); and on 100 m x 100 m centres (in the Raimundo high grade zone for definition of grade variation).

In October 2010 HZM commenced a Phase 2 drilling programme on the combined Teck Araguaia and HZM Lontra Licences. This phase of work comprised diamond core drilling to infill the previous drilling completed by Teck and HZM. From October 2010 to December 2011 HZM completed 539 drillholes for 13,261 m. To manage and support this programme more efficiently an exploration office was established in Conceição do Araguaia in September 2010.

The drilling programme was designed to reduce the 200 m x 200 m Teck drilling grid to a 100 m x 100 m grid on the Pequizeiro and Baião targets. In addition the drill spacing was reduced to 141m x 141m on the Pequizeiro West and Vila Oito East targets and to approximately this same spacing on the Lontra North and Raimundo targets. Seven scout holes were completed at Lontra South

At both Pequizeiro and Baião a set of 25 holes was drilled on a 25m x 25 m grid for geostatistical evaluation purposes. The resource drilling completed in September 2011 was specifically focused on converting the mineral resource estimate on the Pequizeiro and Baião targets to an Indicated resource category in accordance with the JORC Code.

From September 2012 to April 2013 HZM conducted a Phase 3 mineral resource drilling programme. This programme was designed to complete infill drilling on 100 m x 100 m grids on the Jacutinga, Vila Oito West, Vila Oito, Vila Oito East and Pequizeiro West targets in order to convert Inferred resources to Indicated resource categories. 321 holes (9,309 m) were completed including 35 holes (1,186 m) on Jacutinga, 84 holes (1,669 m) on Vila Oito West, 133 holes (4,228 m) on Vila Oito, 44 holes (1,509 m) on Vila Oito East and 25 holes (717 m) on Pequizeiro West.

Table 1.1 provides a summary of diamond core drilling for the PFS.

Table 1.1 Summary of resource delineation drilling by HZM and Teck for PFS

Target	№	Metres drilled
Vila Oito West (VOW)	143	3,096.5
Vila Oito (VOI)	182	5,573.4
Vila Oito East (VOE)	127	3,901.7
Jacutinga (JAC)	59	1,720.9
Pequizeiro (PQZ)	219	6,114.9
Pequizeiro West (PQW)	60	1,626.0
Baião (BAI)	330	7,098.0
Total	1,120	29,131.4

Sample methodology and approach employed for the HZM drilling data is undertaken and verified by Snowden through several site visits.

Half split core samples, taken by HZM, are crushed and pulverised at SGS laboratory in Goiania and the resultant pulps analysed at SGS laboratory in Belo Horizonte using tetraborate fusion X-Ray Fluorescence. Full QA/QC procedures are implemented, including the insertion of standards, duplicates and blanks by HZM and checked by a programme of umpire assays on sample pulps by ACME laboratory, Vancouver.

Snowden's Qualified Person concluded that the sampling and analytical procedures are acceptable and that the resulting records are suitable for use in Mineral Resource estimation.

1.7 Mineral resource

Mineral Resource estimates are reported in Table 1.2. At a cut-off grade of 0.95% Ni, a total of 72 dry Mt at a grade of 1.33% Ni is defined as Indicated Mineral Resource and a further 25 dry Mt at a grade of 1.21% Ni is defined as Inferred Mineral Resource.

Mineral Resources reported for the PFS deposits were prepared under the supervision of Mr. Andrew F. Ross.

Mineral Resources for other deposits in the Project area were prepared by Dr. Marc-Antoine Audet and were reported in Audet, M A, et al (2012). The other deposits are Pequizeiro NW, Oito Main, Lontra North and Raimundo for which Inferred Mineral Resources are reported. These other deposits were not considered in the PFS.

Table 1.2 Mineral Resources for Araguaia as at March 2014 by material type (0.95% Ni cut-off grade)

Araguaia	Category	Material type	Tonnage (kT)	Density (t/m3)	Contained Ni metal (t)	Ni (%)	Co (%)	Fe (%)	MgO (%)	SiO_2 (%)	Al_2O_3 (%)	Cr_2O_3 (%)
Sub-total	Indicated	Limonite	11,560	1.35	137,790	1.19	0.127	36.50	2.76	19.45	9.48	2.61
Sub-total	Indicated	Transition	24,110	1.19	346,920	1.44	0.060	19.87	11.36	41.19	5.05	1.38
Sub-total	Indicated	Saprolite	36,310	1.32	473,960	1.31	0.034	11.82	23.67	42.27	3.62	0.85
Sub-total	Inferred	Limonite	8,830	1.34	100,310	1.14	0.097	35.85	3.94	19.77	9.48	1.78
Sub-total	Inferred	Transition	9,340	1.28	122,040	1.31	0.053	20.34	13.94	37.80	5.31	1.20
Sub-total	Inferred	Saprolite	7,190	1.41	84,370	1.18	0.033	12.07	23.92	41.46	4.16	0.80
TOTAL	Indicated	All	71,980	1.28	958,660	1.33	0.058	18.48	16.19	38.25	5.04	1.31
TOTAL	Inferred	All	25,350	1.34	306,730	1.21	0.063	23.40	13.29	32.56	6.43	1.29

Note: Totals may not add due to rounding. Mineral Resources are inclusive of Mineral Reserves

The PFS estimates were prepared in the following steps:

- data preparation
- geological interpretation and horizon modelling
- establishment of block models and definitions, with a block size of 25 m x 25 m x 2 m
- compositing of assay intervals
- exploratory data analysis and variography
- Ordinary Kriging estimation method.
- model validation
- calculation of dry density
- classification of estimates with respect to JORC (2012) guidelines
- resource tabulation and resource reporting.

Table 1.3 and Table 1.4 list Snowden's assessment of the criteria that were considered when classifying the 2013 Araguaia PFS resource estimates in accordance with the JORC Code (2012 edition) guidelines.

The resource classification scheme adopted by Snowden was based on the following:

- Mineralisation was classified as Indicated where the drilling density was 100 mE by 100 mN (or less).
- Mineralisation delineated using a drilling density larger than 100 mE by 100 mN and up to about 150 m spacing was classified as Inferred.
- Mineralisation delineated using sparse spacings was not classified.

For the other deposits, not included in the PFS Mineral Resources were estimated by Dr. Marc-Antoine Audet using block estimation by Inverse Distance at the power of 2 (ID2) interpolation methodologies on 25 m × 25 m × 2 m blocks.

Three-dimensional models for these deposits were created using surveyed holes. The models integrate the concept of geological horizons (limonite, transition and saprolite) to create a 3D block model. For each deposit, a surface geological constraining envelope was generated using drillhole data as well as information from geological mapping.

There are no Mineral Resource estimates for other prospects (Morro, Southern, Oito West and Pequizeiro East) due to insufficient drill sample information.

Table 1.3 JORC Code (2012) Table 1 Section 1 – Sampling techniques and data

Item	Comments
Sampling techniques	The data used for resource estimation is based on the logging and sampling of diamond core drilling (100% of the sample data).
Drilling techniques	The drilling was completed using vertical core holes. Vertical drillholes are appropriate given the strike and dip of the mineralisation.
Drill sample recovery	HZM has required the drilling contractor to redrill where recoveries were less than 85 %, thus ensuring the recoveries in the provided database are adequate.
Logging	Almost all of the geological information has been obtained by the logging of drill samples, and supplemented by surface geological mapping and interpretation of geophysical surveys. Logging of drillhole samples was done with sufficient detail to meet the requirements of resource estimation and mining studies, and in accordance with HZM Standard Operating Procedures.
Sub-sampling	Cores were sampled at 1 m intervals. Half split core samples are crushed and pulverised at SGS laboratory in Goiania and the resultant pulps analysed at SGS laboratory in Belo Horizonte using tetraborate fusion X-Ray Fluorescence. Full QA/QC procedures are implemented, including the insertion of standards, duplicates and blanks.
Quality of assay data and laboratory tests	Snowden's analysis of the QAQC data (standards, blanks, duplicates) and assessment made by HZM did not identify any significant issues which could be material to the resource estimate.
Verification of sampling and assaying	Drilling from earlier phases was verified by independent Qualified Persons. In 2013, HZM dispatched 457 duplicate pulp samples to the ACME Laboratory in Canada for umpire check analyses. Original assay was completed by SGS in Brazil and the same analysis method (tetraborate fusion XRF) was applied at both laboratories. A reasonable level of accuracy has been demonstrated.
Location of data points	The project area is centred about the following co-ordinates: WGS 84 Latitude 07° 54' 9.0" South; UTM SAD 69 22S 9126200mN; and WGS 84 Longitude 49° 26' 1.8" West; UTM SAD 69 22S 672700mE. Collar locations were surveyed using a DGPS (precision +/- 10 cm) by Independent Licenced Surveyor. Elevation differences between drillhole collars and the supplied topography DTM were checked and eliminated by pressing the collars to the DTM.
	No downhole surveys were collected for the drilling. Mitigating this issue to a large extent is the fact that most of the drilling consists of shallow vertical core holes and the drill rig alignment is checked by HZM staff prior to drilling.
Data spacing and distribution	Drilling was completed along a set of oriented sections. The drillhole spacing is essentially 100 m apart with two small areas on areas PQZ and BAI drilled to 25 m spacing for variogram analysis. These drillhole spacings are sufficient to establish the degree of geological and grade continuity necessary to support the resource classifications that were applied.
	The drilling was composited downhole using a 1.0 m interval which corresponds to the dominant assay interval.
Orientation of data in relation to geological structure	The location and orientation of the Araguaia drilling (vertical) is appropriate given the geometry and orientation (horizontal) of the laterite mineralisation.
Sample security	All sampling and data collection is handled by HZM personnel and the drill core is subsequently transferred into core boxes. Drill core is stored in a secure facility in Conceição do Araguaia. Sample security procedures are provided in section 11.3. Pulp and crush rejects are returned after a 90 day period at SGS, pulp rejects are stored in cardboard boxes and crush rejects in large plastic boxes sequentially batch by batch also onsite.
Audits and reviews	The drilling database was reviewed by Snowden and sufficient cross-checks with assay certificates, drill core and logging, collar surveys were undertaken to confirm that the data is suitable for use in mineral resource estimation.

Table 1.4 JORC Code (2012) Table 1 Section 3 – Estimation and reporting of Mineral Resources

Item	Comments
Database integrity	The new Phase 3 drilling data was supplied to Snowden in Microsoft Excel spread sheets and then imported into an existing GEMS Project database by Snowden. Internal validation checks were made by Snowden and any discrepancies were corrected in consultation with HZM.
Geological interpretation	Snowden believes that the local geology is well understood as a result of work undertaken by HZM and Dr Marc-Antoine Audet in respect of chemical classification of rock types. The contacts between laterite horizons have been interpreted based on a combination of logging and geochemistry as described in Section 6. Alternative interpretations of the mineralisation are unlikely to significantly change the overall volume of the Horizons.
Dimensions	The Araguaia mineralisation estimated by Snowden consists of seven areas. Descriptions of the deposits are provided in Section 4.3. Maximum and average thickness of the laterite horizons are provided in Section 7.0.
Estimation and modelling techniques	Unfolded ordinary block kriging using hard boundary domains, with sub-celling to accurately reflect horizon contacts, was undertaken in Datamine software. The deposits have been estimated previously by Dr Marc-Antoine Audet in GEMS software using an unwrinkling approach.
Moisture	All tonnages have been estimated as dry tonnages.
Cut-off parameters	The nickel mineralisation was reported above a 0.95% nickel cut-off grade.
Mining factors and assumptions	It is assumed the deposits will be mined using open cut methods.
Metallurgical factors and assumptions	None. Metallurgical test work reported in Section 13 indicates there is a reasonable prospect for metal recovery using current technologies.
Environmental factors or assumptions	These are discussed in Section 20 “Environmental Studies, permitting, and social or community impact”.
Density	There are sufficient bulk density measurements (water displacement method) to relate major chemistry to density by linear regression. Block estimates of dry density were calculated from block grade estimates.
Classification	The resources have been classified based on continuity of both the geology and the nickel grade along with the drillhole spacing. Additionally the information summarised in this table has been used to support the resource classification categories of Indicated and Inferred.
Audits and reviews	The Snowden models compiled in 2013 have not been independently reviewed in detail but have been discussed with HZM’s Technical Advisor, Mr F R Billington.
Accuracy and confidence	The resource was classified by taking into consideration the confidence in the continuity of nickel grades and the confidence in the geological interpretation.

1.8 Mining

Seven mining pits were identified through a process of pit optimisation using costs, and process recoveries. All seven pits were designed through a standard process of pit optimisation, waste dump design and pit design.

The pit design used smoothed pit shells from the pit optimisation and altered for the removal of small satellite pits. This was deemed by Snowden to be appropriate for pits with no ramp requirements. It is likely that the actual pit floor will be dictated by operating conditions as they are mined, although the quantities mined from each will be similar to those calculated by Snowden in this study.

Feedback from all relevant stakeholders was used to determine a waste disposal concept for each pit, including both ex-pit and in-pit options. After calculating the volumes of each waste type waste dumps were designed to contain this material and minimise required haulage distances as much as possible

The Project was scheduled on the basis of panels. A total of 43 panels for the project were designed and scheduled. Within each panel, a number of “bins” are generated on the basis of rock type and nickel grade. The production schedule was completed in quarterly increments over the life of the project.

A number of processing constraints were applied to the schedule which included a 13 month processing feed quantity ramp up period, and specific process feed grade constraints throughout the life of the project:

- Fe grade between 15.0% and 16.5%
- Al_2O_3 grade between 4.0% and 5.5%
- SiO_2/MgO ratio between 2.2 and 2.6.

Each of the deposits is proposed to be mined with typical truck and excavator mining. Although the primary fleet requirement changes throughout the life of the project a typical configuration is 6 x 48 t operating weight (OW) excavators, 3 x 50 t OW front end loaders, 17 x 40 t rated payload (RP) articulated off-highway trucks and 2 x 30 t RP on-highway trucks for longer inter-pit haulage. This fleet is supported by the usual array of support and ancillary equipment.

Grade and mineralogy will be closely monitored in the mining process using close spaced grade control drilling ahead of mining.

1.9 Mineral reserve estimate

The estimation of Mineral Reserves used estimates of Indicated Mineral Resources for the Project as reported in Table 1.2.

A Mineral Reserve estimate of 21.2 kt (dry) at an average grade of 1.66% Ni was estimated. The detailed breakdown of the Mineral Reserve by deposit is presented in Table 1.5.

Table 1.5 March 2014 Mineral Reserve estimate

Class	Deposit	Ore Dry Mass (kt)	Ni (%)	Fe (%)	Al_2O_3 (%)	SiO_2/MgO
Probable	Baião	3,500	1.67	17.41	4.58	2.56
Probable	Pequizeiro	9,300	1.70	15.58	5.39	2.56
Probable	Pequizeiro West	380	1.57	20.38	4.63	4.29
Probable	Jacutinga	960	1.81	15.13	2.96	2.11
Probable	Vila Oito East	2,450	1.55	15.97	3.73	2.22
Probable	Vila Oito	3,580	1.63	14.61	3.63	2.05
Probable	Vila Oito West	1,020	1.59	19.35	4.25	3.32
	Total Probable	21,200	1.66	16.01	4.59	2.44
	Proven	NIL	N/A	N/A	N/A	N/A
	Total Proven and Probable	21,200	1.66	16.01	4.59	2.44

1.10 Metallurgical testwork

From late 2011 to 2013, HZM contracted various organisations and metallurgical laboratories to conduct metallurgical testwork on material from the Project. The testwork was aimed at providing design data for the preferred processing option, a typical Rotary Kiln Electric Furnace (RKEF) installation, designed to treat lateritic nickel ore containing between 1.4% and 1.9% Ni and producing a final refined Fe-Ni product containing between 18% and 22% Ni.

The main testwork included:

- Agglomeration testwork by Feeco International
- Laboratory based tests by Xstrata Process Support
- Slag chemistry testwork and simulations by Kingston Process Metallurgy
- Liquidus measurements of Fe-Ni slag by Kingston Process Metallurgy
- Evaluation by FL Smidth Inc.
- Screening study for ore smelting by Hatch Ltd.

The testwork was carried out with a view to establishing both the feasibility of applying the RKEF process to Araguaia ore, and subsequently, determining the main RKEF parameters. Based on this testwork, it was found that Araguaia ore is suited to processing by RKEF; a target Ni grade in Fe-Ni of 20% Ni was considered (range 18 to 22% Ni).

1.10.1 Prior testwork

During earlier stages of the Project hydrometallurgical testwork was conducted, with two approaches reviewed:

- Atmospheric tank leach
- Bottle roll tests to simulate heap leaching.

Sulphuric acid was selected for both types of leaching tests. Acid consumption was high and these options were not further pursued after the initial testwork was completed.

1.10.2 Agglomeration testwork – Feeco International

A number of agglomeration tests were run in the pilot rotary agglomerator at Feeco International using a 51% transition to 49% saprolite blend. The objectives of the tests were to observe the agglomeration behaviour of the as-received ore (nominally 38% moisture content). The three variables that were changed in the tests were:

- feed rate
- drum rotational speed
- with and without a liner and lifters inside the drum.

During the testwork it was visually observed that agglomerates were reasonably competent and that fines were generally contained within the agglomerates.

The drop test demonstrated good resilience of the agglomerated particles to breaking during the test. The round particles did however deform into flat disks during the tests.

Particle size distribution ranged from approximately 3 mm to 25 mm. Agglomerated particles did tend to break up during exposure to water in a flat tray, however, it was observed that the feed material tended to break up faster than the agglomerated particles.

The rotary action of the drum (which resembled the action of the kiln dryer) was generally able to produce balled or agglomerated material from as-received wet laterite ore over a range of feed rates and drum revolving speeds and independently of whether the liner was in place or not.

1.10.3 Sample characterisation and smelting tests – Xstrata Process Support (XPS)

XPS first performed a series of sample characterization tests which included moisture measurements (free and crystalline), particle size analyses and chemical assays of the ore and calcine samples. Subsequently a series of smelting tests were conducted at elevated temperature (1520 ^{0}C). At the end of each smelting test, the products, Fe-Ni alloy and slag, were collected and weighed. The alloy and slag samples were submitted for chemical analyses. As a part of the study, the potential recovery of Ni was also estimated based upon small scale batch tests.

Preliminary heat and mass balance calculations were also performed, estimating energy requirements of the process and were used to generate a simplified flow sheet. The flow sheet included steps such as feed preparation, drying, calcining and smelting.

The results of the study are summarized as follows:

- Particle size analyses indicated that due to the fineness of the ore material handling challenges are to be expected, particularly during wet weather.
- Blending of the feed will be necessary to achieve optimum Fe/Ni ratios in the ore feed. The feed preparation step can also be tailored towards the production of Fe-Ni alloy or matte as the final products.

- Preliminary batch smelting tests were conducted at 1520 ^{0}C on three ore samples, targeting an alloy grade of 30 wt% in each case. The tests were successful in producing this grade of alloy for two of the samples. Similarly, the series of smelting tests with specific ore blend ratios with varying Fe/Ni ratios were also successful, with the Fe-Ni grades obtained in the tests varying from 14% Ni to 53% Ni.
- Smelting tests were performed adding sulphur in order to generate matte as the final product. The liquidus temperature of the resulting matte phase was also predicted with thermo-chemical calculations.
- The possibility of utilizing the sensible heat and chemical reduction potential of the off-gas from the smelting step opens up a number of flow sheet options which could be explored.
- Extensive thermo-chemical modelling was performed predicting the liquidus temperatures of all possible phases. Based upon preliminary heat and mass balance calculations, a basic flow sheet was developed to treat the deposits.
- The basic flow sheet proposed can be further optimized to reduce the fuel and reductant consumptions. Opportunities for energy savings were also identified and could include technologies such as flash drying, fluidized bed calcination, hot calcine charging and DC smelting.
- The testwork demonstrated that the SiO_2/MgO ratio has a significant impact on the slag liquidus temperature.

On review of the XPS report, IGEO expressed concerns regarding some of the data quoted in the report which could impact on process design criteria such as energy requirements and hence the associated operating costs. These concerns were clarified with HZM prior to completion of the process design.

1.10.4 Slag chemistry testwork and simulations – Kingston Process Metallurgy (KPM)

Simulations conducted by KPM, using FACT, indicated that XPS results were in close agreement with the thermodynamic calculations. The coke consumption in the electric furnace was calculated to range between 3% and 5% of feed depending on the ore blend and also the level of pre-reduction of the calcine.

HZM also requested optimised conditions for smelting calcine produced by a blend of 13.7% limonite, 43.8% transition and 42.5% saprolite when producing a 25% Ni alloy in the electric furnace. FACT simulations were conducted to test these conditions.

KPM also performed a number of FACT$_{®}$ calculations regarding the slag chemistry aimed at:

- simulating the results of selected laboratory smelting tests carried out at XPS
- evaluating the smelting of a different laterite blend proposed by HZM.

Results indicated that:

- XPS experimental results are in close agreement with the thermodynamic calculations performed in this study.

The new blend proposed for the Araguaia laterites brings a challenge and an opportunity in the smelting of this ore by providing a low melting point slag, which will allow operating the electric furnace at lower temperatures. However, the higher superheat required to maintain the metal as liquid will impact on furnace design.

The alumina content of the slag (~7% wt) also favours a low slag liquidus temperature. The soluble Ni in slag should be under 0.05% Ni and the remainder would be in the form of entrained alloy droplets. Total losses are expected to be at nearly 0.15% when compared to similar operations.

KPM sees no major technical constraint in smelting this type of blend in an electric furnace, provided that a careful furnace design is taken into account.

1.10.5 Slag liquidus temperatures - KPM

KPM also measured slag liquidus temperatures for a Fe-Ni smelting process during a separate testwork project, using differential thermal analysis combined with thermo-gravimetry (TGA/DTA). Synthetic slag was used in a TGA/DTA unit under argon to ensure that the composition was accurate and to prevent oxidation of FeO. In all, six different slag compositions were studied to determine the influence of alumina (Al_2O_3) concentration and silica/magnesia (SiO_2/MgO) ratio on the liquidus temperatures.

The liquidus increased from 1,368 ^oC to 1,405 ^oC when the SiO_2/MgO ratio was decreased from 2.7 to 2.3. A significant decrease in liquidus (1,431 °C to 1,383 °C) was found when the alumina concentration was increased from 4.00 % to 7.36 %. The liquidus temperatures measured experimentally followed the same trend as the projected (FACT) values but were consistently lower by 10 ^oC to 20 ^oC.

1.10.6 Measurement of physical and chemical ore properties – FL Smidth (FLS)

A laboratory study was performed by FLS during 2012 to evaluate the physical and chemical properties of two nickel laterite ore blends.

- Mix 1: 51% Transition and 49% Saprolite
- Mix 2: 14% Limonite, 44% Transition and 42% Saprolite.

The samples were analysed to determine free moisture content, bulk density, angle of repose, particle size distribution, drying curves and particle degradation (tumble testing). From the testwork FLS concluded the following:

- The Araguaia ore is characterized by a very fine natural particle size. The fine particles demonstrate binding properties similar to clays when dried, thereby yielding relatively hard agglomerates resistant to significant degradation and dusting.
- The onset of particle sintering is 50°C to 100°C lower than many lateritic ores evaluated by FLS. This suggests a limited achievable calcine temperature of 800 to 825°C during rotary kiln processing, which will also limit the degree of iron pre-reduction that can be obtained. For equipment design purposes IGEO have considered target of 850^{o}C for the kiln discharge and 825^{o}C for calcine feed to the furnace.
- The use of 10% Ni and 60% Fe pre-reduction targets for commercial Fe-Ni line design were proposed. Pilot rotary kiln testing was recommended. For design purposes, IGEO have considered pre-reduction of 20% for Ni and 70% for Fe.

- Briquetting appears to be a viable option for producing an agglomerated feed suitable for kiln processing to yield a granule calcine with acceptable dusting rates.
- Rotary drum agglomeration demonstrated the production of agglomerates resistant to fines generation during tumbling.
- The results of this laboratory study suggest that the Araguaia ore is suited for rotary kiln processing in an RKEF system provided that proper agglomeration provisions are adapted and that lower calcine temperature and pre-reduction levels are considered in the electric furnace design.
- Larger scale pilot testing was recommended to confirm the conclusions of this study.

1.10.7 Smelting testwork review

Slag Characteristics – Key factors

- The combined impact of the ratio SiO_2/MgO, FeO and Al_2O_3 contents of the slag on the liquidus temperatures and the viscosity of the slags.
- The impact of impurities, particularly carbon and silicon, on the liquidus temperatures and viscosity of the metal.
- The impact of the issues in 1 and 2 above can be affected/controlled to varying degrees by controlling the degree of reduction of Fe effected during the smelting process
- Slag foaming can occur with viscous slags and silica and alumina play significant roles in this. Such foaming is likely to increase nickel losses into the slag.
- The degree of iron reduction dictates the grade of nickel to be achieved in the final metal product as well the silicon and carbon content of the metal.
- At the SiO_2/MgO ratio of 2.29 and the FeO and Al_2O_3 content anticipated in the Araguaia slags, the liquidus temperatures can be significantly depressed. However, minimum tapping temperatures of both metal and slag must be attained to ensure fluidity.

Metal Liquidus Temperature

- The XPS report considered the binary Fe-Ni system only and reported temperatures above 1,500^oC. KPM did not include the effects of residual C, Si and S on the alloy and reported a metal liquidus temperature of 1,479^oC.
- 50% iron reduction (with respect to Fe/Ni ratio) is required to get substantial amounts of C and Si into the metal. These elements have an important influence on the metal liquidus temperature. Reduction of over 50% would result in high C, Si and Cr in the metal and a liquidus temperature of 1,250 ^oC to 1,350^oC. A reduction of over 50% would result in a Fe-Ni grade of 15%.
- Introduction of sulphur into the metal could be used to reduce the metal liquidus temperature. This will require further treatment methods of the matte and was therefore not considered for this PFS phase of the Project.

1.11 Process design and recovery

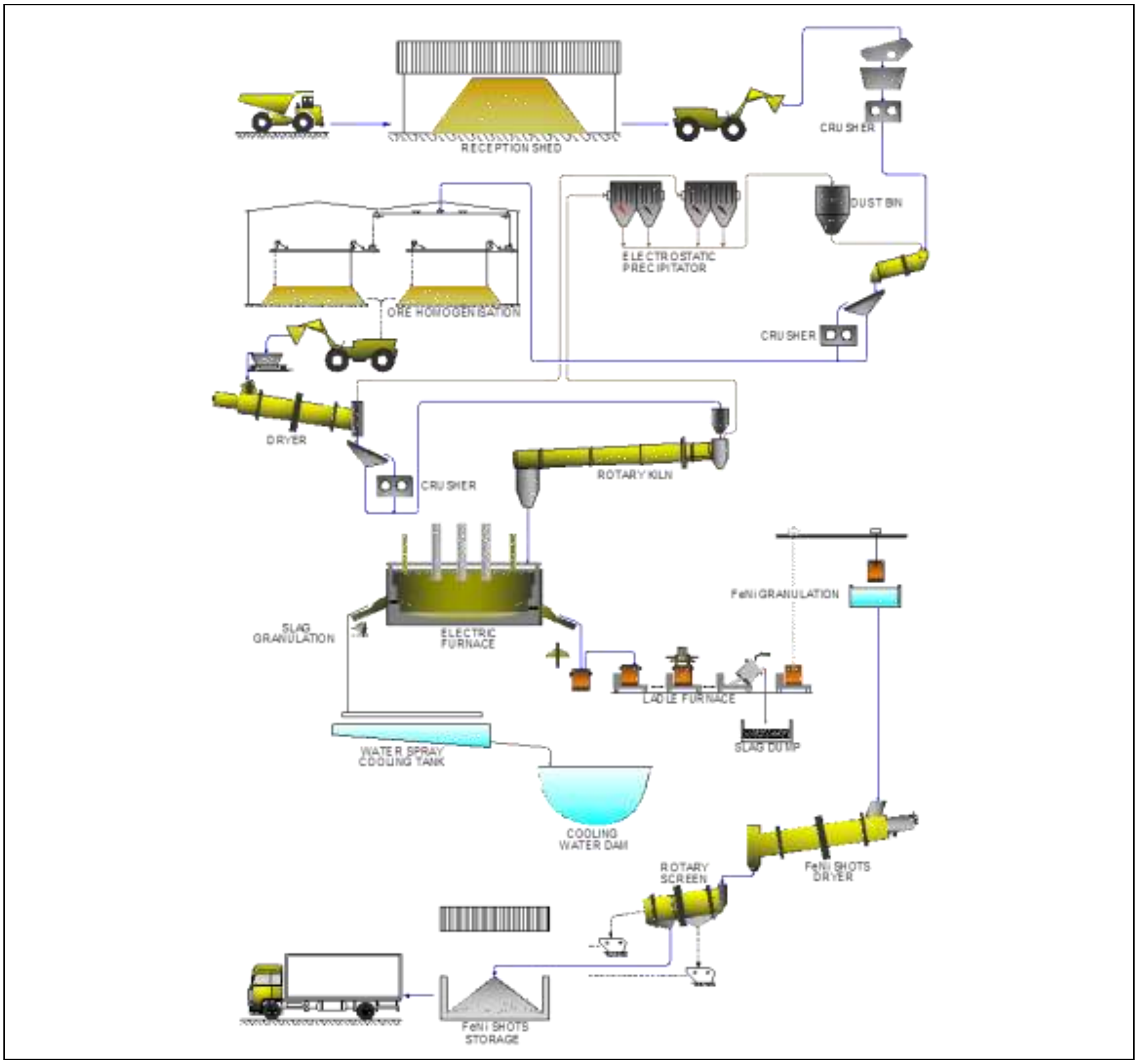
1.11.1 Process selection

As part of the metallurgical testwork, both pyrometallurgical and hydrometallurgical test work was undertaken. This testwork subsequently confirmed the preferred suitability of the conventional RKEF process for the treatment of the Araguaia ore to produce Fe-Ni and this process was adopted for the PFS by HZM.

1.11.2 Process description

Although two throughput options were considered during the prefeasibility study, the preferred Base Case is a single line RKEF installation for 0.9 Mtpa (dry) ore, producing approximately 15,000 tpa nickel as Fe-Ni. The overall process flow block diagram shown in Figure 1.7 summarises the various processing stages of the proposed project.

Figure 1.7 The Project process flow block diagram



The initial process stages encompass ore preparation, where the ore is sized to match the subsequent metallurgical process requirements. Kiln dust is recycled to the process before the secondary crushing stage. The ore is then homogenized, partially dried and fed to the kiln with the addition of a reductant material. In the kiln, the ore is completely dried and calcined to remove chemically combined moisture, and partially pre-reduced. Calcine is transferred into an electric furnace for the separation of the metal and slag at high temperatures. The metal is conveyed in ladles to the refining stage. The refining oxidized slag is granulated with water, while the reducing slag is transported molten and disposed of in a specific site. The final Fe-Ni product is granulated with water, screened, dried and stockpiled prior to dispatch to the market. The process design criteria (PDC) are outlined in Table 1.6.

Table 1.6 Summarised key process design criteria

Item	Unit	Value
Plant throughput	Mtpa (dry)	0.9
Moisture content	%	30
Coal consumption in rotary kiln	dry kg/dry t ore	76.4
Coal consumption in dryer	kg/wet t ore	13.6
Design Ore grade	%Ni	1.8
Overall Ni recovery	%Ni	93
Ni production at design grade	tpa	15,067
RKEF combined availability	%	90
Furnace power	MW	50
Furnace energy consumption	kWh/ t calcine	511
Ni grade in metal	%Ni	20.0
Refinery		Ladle furnace
Final product		Granulated Fe-Ni

1.12 Project infrastructure

The Project will comprise the following infrastructure:

- Slag dump
- Port facilitation
- Rail
- Roads (access, trunk and ancillary)
- Process facilities
- Ancillary buildings (administration, workshops, ablutions, etc.)
- Power distribution system
- Cooling water dam
- Coal storage facility
- Water acquisition, storage and distribution systems
- Fuel storage
- Communications
- Fire systems.

The Project is supported by existing infrastructure including a network of Federal highways and roads. In addition, the existing port city of São Luís provides the primary supply chain facility for in-bound and out-bound logistics for bulk material handling of coal and potentially Fe-Ni product.

Further details are found in Section 18.

1.13 Execution plan

A two year construction period has been estimated.

A ramp up schedule is shown in Table 1.7. Based on this tonnage in the first year of production will be approximately 50% of the design capacity and will be increased gradually over the first 10 months of production to 100% of the nominal design tonnage of 0.9 Mtpa.

Table 1.7 Proposed process plant ramp-up

ARAGUAIA NICKEL PROJECT				
Preliminary Process Plant Ramp-up for Mining Studies - Base Case				
Year	Month	One Process Line - 0.900M dmt ore per year		
		Tonnage % of Nominal	Ni Recovery Actual %	Ni Produced % of Ni Input
1	1	50	45	22.5
	2	55	65	35.8
	3	60	75	45.0
	4	69	78	53.8
	5	76	81	61.6
	6	83	83	68.9
	7	87	85	74.0
	8	90	87	78.3
	9	93	88	81.8
	10	95	90	85.5
	11	100	90	90.0
	12	100	93	93.0
2	13	100	93	93.0
	14	100	93	93.0
	15	100	93	93.0
	16	100	93	93.0
	17	100	93	93.0
	18	100	93	93.0
	19	100	93	93.0
	20	100	93	93.0
	21	100	93	93.0
	22	100	93	93.0
	23	100	93	93.0
	24	100	93	93.0

Notes: 1) Nominal plant feed 900 ktpa dry ore
2) Ni production = % of Ni input in ROM ore

1.14 Capital cost

All $ used in this Technical Report are USD.

The capital costs are discussed in detail in Section 21.

The Base Case for this study assumes an ore processing rate of 0.9 Mtpa after an initial ramp up period. The ore processing methodology is the pyro-metallurgical conversion of a nickel bearing laterite ore into a Fe-Ni product using the RKEF process that will be sold at the mine gate.

The pre-production capital costs in t have been allocated 30% in year 1 and 70% in year 2 of the 2 years of construction. Deposits will be required on the high value long lead items in year 1 and the balance will be required in year 2.

The pre-production capital costs are shown in Table 1.8 and the production capital costs are shown in Table 1.9 .

Table 1.8 Pre-production capital costs

Item	$ million
Plant direct	376.088
Plant indirect	38.206
Owners costs	18.313
Infrastructure	56.034
Slag storage facility	5.242
Social	6.000
Mining	5.000
Contingency at 15%	76.092
First fills and spares	1.200
Total pre-production capital costs	582.176

Table 1.9 Production capital costs

Item	$ million
Mining & plant Sustaining	43.313
Closure (2 Years)	20.000
Total Production Capex	63.313
Salvage	1.400

1.15 Operating cost

The operating costs, royalties and taxation are shown in Table 1.10 below and are discussed in detail in Section 21.

The Base Case assumes a contractor for mining and ore haulage to the plant.

Table 1.10 LOM operating costs

Item	$ million	$/tonne
Mining (contractor)	552.998	26.08
Processing	2,641.667	124.57
Off-site overheads	99.000	4.67
Total operating costs	3,293.665	155.32

1.16 Royalties (CFEM)

The calculation for royalties is discussed in detail in Section 21.

The calculation of the Compensation for Exploitation of Mineral Resources (CFEM) is carried out using the accumulated cost of production as a deduction up to the point the ore has no physico-chemical modification.

In the case of a pyro metallurgical project such as the Araguaia project the calculation of the CFEM encompasses the following costs: mining, stockpiling, crushing, coal preparation, administration, maintenance and some environmental costs up to and including calcining. The addition of all these costs gives a value that will be multiplied by 2%. The CFEM payable is shown in Table 1.11.

Table 1.11 Royalties

Item	$ million
Royalty	43.301

1.17 Taxation

The taxation regime in Brazil is discussed in detail in Section 21.

The taxation regime uses a taxation rate of 15.25% of the taxable income for the initial 10 years of production after which the rate is increased to 34% of the taxable income. The taxable income is calculated after deducting all operating expenses and depreciation of capital items.

Depreciation is calculated on a straight line method over 10 years. For this project, an initial value of $15 million was allowed for previous expenditure not deducted. The taxation payable is shown in Table 1.12 below.

Table 1.12 Taxation

Item	$ million
Taxation	668.051

1.18 Economic evaluation

Several cases were studied to identify the financial implications to the Project. From the results of these studies the Base Case was chosen to present in detail for this report. The basis of the studies and the reasons for choosing the Base Case are discussed in more detail in Section 22.

Snowden prepared an economic cashflow and financial analysis model based on inputs from mining and processing schedules as well as capital and operating cost estimates including royalties for the Base Case. The model was prepared from mining schedules estimated on a quarterly basis for the first 4 years of production and then annually for the remaining project life. All inputs are consolidated annually in this report.

The following Table 1.13 and Table 1.14 provide the project headline results before and after taxation (Table 1.15). After a review of a consensus opinion and historical prices, the Ni price used is $19,000 per tonne and the Fe price is $150 per tonne. Both are flat for the life of the project. Base Case KPI before taxation provides a summary of the Project KPI's as shown in Table 1.15.

Table 1.13 Base Case economic model headline results before taxation

Item	Unit	Value
Net Cashflow	$M	2,433.933
NVP_8	$M	730.067
IRR	%	22
Production payback period	years	4.1

Table 1.14 Base Case economic model headline results after taxation

Item	Unit	Value
Net Cashflow	$M	1,765.882
NVP_8	$M	519.233
IRR	%	20
Production year payback	years	4.4

Table 1.15 Base Case KPI before taxation

Item	Unit	Value
Value of product sold	$/t ore	302.58
Cash cost	$/t ore	157.36
Total cost	$/t ore	187.80
Production year payback	year	4.1
Prorata cash cost	$/lb Ni	4.48
Prorata cash cost	$/t Ni	9,883
Prorata total cost	$/lb Ni	5.35
Prorata total cost	$/t Ni	11,795
Brooke Hunt methodology C1 cost	$/lb Ni	4.16
Brooke Hunt methodology C1 cost	$/t Ni	9,166

1.19 Summary of the project risks

A full risk assessment was conducted as part of the PFS.

A detailed examination of the actions associated with each of the risks and opportunities identified suggests that there are five common themes prevailing, as are described below:

1. The *management and technical competency* theme relates to the technical knowledge, ability and willingness of potential employees (management and workforce) to commission and maintain a complex nickel plant with high tolerances in terms of throughput, control and instrumentation and maintenance.

2. The *complexity theme* relates to the mineability and treatability of nickel laterites which is inherent.

3. The *data theme* covers timely data acquisition that is sufficiently well scoped that it covers all later needs. The PFS study has identified a number of data and information requirements which need to be satisfied before commencing a feasibility study.

4. The *water theme* relates to the mine being in a part of the world exposed to seasonal variations of heavy rainfall and drought which poses subsequent design requirements in terms of controlling moisture within excavated material (ore or waste), productivity aspects relating to excavation, together with potential inundation.

5. The *tenure and licencing theme* relates delays of permitting caused by delays in approval of various permits and key documentation. There is an inherent need to ensure advanced planning of key documentation for permits and permissions and close engagement by HZM with the relevant authorities.

1.20 Conclusions and recommendations

1.20.1 Conclusions

The Project has been investigated at a PFS level and this Technical Report provides a summary of the results and findings from each major area of investigation and study. This Technical report has drawn from a PFS engineering study in support of this Technical Report which have been detailed as Applicable Documents. Each section of this Technical Report describes in more detail the results of the various investigations and studies along with principal findings and appropriate discussions of significant risks that may have been identified during the PFS as well as conclusions and recommendations for further study.

Based on the accumulative findings from the various technical areas of the PFS, the economic analysis performed shows the Project to have merit and be worthy of additional detailed investigations.

The next step for the Project will be further engineering and geoscientific evaluation to advance the Project to feasibility study level.

A more detailed discussion of conclusions, relating to the PFS, is provided in Section 25 of this report.

1.20.2 Recommendations

Snowden recommends that HZM takes the project into a feasibility study to increase engineering definition and therefore improve the cost and economic accuracy for the Project.

Recommendations have been made for subsequent detailed metallurgical testwork for the feasibility study stage of the project. Such testwork is designed to provide additional confirmatory technical data and efficiency factors for the process design.

The mining of nickel laterites is well understood and has unique requirements and challenges. It is recommended that subsequent studies consider refining productivity and operational flexibility in earthmoving machinery to maximise the reserve potential.

Recommendations have been made from the social and environmental perspective including a listing of required permits and studies to enable the Project to proceed.

All recommendations are detailed in Section 26 of this report.

2 Introduction

2.1 Overview

This Technical Report has been prepared by Snowden Mining Industry Consultants (Snowden) for Horizonte Minerals Plc (HZM) in compliance with the disclosure requirements of the Canadian National Instrument 43-101 (NI 43-101). The trigger for preparation of this report is the 25 March 2014 press release by HZM disclosing an updated mineral resource and mineral reserve for the project, and the results of a pre-feasibility study.

Unless otherwise stated, information and data contained in this report or used in its preparation has been provided by HZM.

The Qualified Persons for preparation of the report are Andrew Ross, who conducted as site in November 2012, Anthony Finch, and Peter Theron who visited the project site in July 2013. Harald Muller did not conduct a site visit.

The responsibilities of each author are provided in Table 2.1.

Table 2.1 Responsibilities of each co-author

Author	Responsible for section/s
Andrew Ross	6-12,14, parts (1-3)
Anthony Finch	15, 16, 19, 21-27, parts (1-3)
Harald Muller	13, 17, 18, parts (1-3)
Peter Theron	4, 5, 20, parts (1-3)

Unless otherwise stated, all currencies are expressed in US dollars ($).

2.2 Sources of Information

2.2.1 Applicable Documents

Snowden has based this prefeasibility study (PFS) on the engineering and geological detail provided in the following studies that were undertaken in 2013.

Snowden

- Mining Engineering study for the Araguaia Nickel project, 900 Ktpa Scenario (Base Case) and 2.7 Mtpa Scenario (Option 1) – December 2013.
- Araguaia Nickel Project – Phase 3 - Mineral resource estimate – November 2013.
- Araguaia PFS Geotechnical Report, Plant Site and Slag Dump site ground investigation – December 2013
- Araguaia PFS Geotechnical Report - Araguaia Project Geotechnical Report – December 2013
- Araguaia Risk Analysis Report – December 2013

IGEO

- Araguaia Nickel Project Mineral Processing and Smelter Design – December 2013

KH Morgan and Associates

- Hydrology – Hydrogeology, Araguaia Nickel Project – December 2013

Prime

- Slag dump Design for the Araguaia Nickel project – December 2013
- Environmental Study for the Araguaia Nickel project – December 2013

2.2.2 References

All references are listed in Section 28.

3 Reliance on other experts

This Technical Report has been prepared by Snowden on behalf of HZM. The information, conclusions, opinions, and estimates contained herein are based on:

- Information available to Snowden at the time of preparing this Technical Report including previous Technical Reports prepared on the Project and associated licences within the project
- Assumptions, conditions, and qualifications as set forth in this Technical Report; and
- Data, reports, and other information supplied by HZM and other third party sources.

The qualified persons have not carried out any independent exploration work, drilled any holes or carried out any sampling and assaying on the Project, other than examining/verifying mineralisation and rock conditions in drill cores.

For the purposes of this report, Snowden has relied on ownership information provided by HZM. Snowden has not researched property title or mineral rights for the Araguaia Nickel Project and expresses no opinion as to the ownership status of the property. The description of the Property, and ownership thereof, is set out in Section 4 in this technical report, is provided for general information purposes only.

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

Information sources have included detail from the following organisations:

- CRU strategies (CRU), London, United Kingdom – Section 22.

- Ernst and Young (E&Y), Brazil – Sections 21 and Section 22.

4 Property description and location

4.1 Introduction

The Project is comprised of 14 exploration licences in two non-contiguous blocks within the prominent Araguaia Nickel Belt located in Pará state, Brazil. The licences have been acquired by HZM through application and acquisition. Lontra was a discovery by HZM and the Araguaia licences were acquired through the purchase of Teck's (TSX: TCK) adjacent Araguaia Project. The Floresta and Vila Oito licences were acquired via share purchase from Lara Exploration (TSX-V: LARA). All licences are now held under one Brazilian company 100% owned by HZM.

4.2 Location

The property is centred approximately 45 km northwest of the town of Conceição de Araguaia (population of 46,206) and approximately 25 km west of the north-south trending Araguaia River (Figure 4.1).

Initial historic discovery of nickel laterite in the region was at the Quatipuru deposit, located approximately 75 km southwest of the project area. Xstrata's Serra do Tapa and Vale dos Sonhos nickel laterite deposits, that form part of the Araguaia Nickel Belt, are located approximately 80 km north of the Project area (Figure 4.1).

The project area is centred about the following co-ordinates, for a SAD 69 Datum:

- LAT 07° 54' 7.48" S / LON 49° 26' 0.26" W
- UTM 22S 9126200 N / 672700 E

The Carajás mineral province (Mining District), situated approximately 200 km northwest of the Project, is host to a number of other prominent iron ore, nickel laterite and IOCG deposits. Carajás is the main centre of mining activity in the Pará State (Figure 4.1).

4.3 Licences and tenure

4.3.1 Mining legislation overview

The main sources of mining legislation in Brazil are the Federal Constitution and the Mining Code (Decree-law No. 227 of 28 February 1967). The Mining Code defines and classifies deposits and mines, sets requirements and conditions for obtaining authorizations, concessions, licences and permits, the rights and duties of holders of exploration licences and mining concessions. There are two main legal regimes under the Mining Code regulating exploration and mining, i.e. the "authorization" for exploration and the "concession" for mining.

Exploration, which is defined by the Mining Code as the work required to locate and define a deposit and determination of the economic feasibility thereof, can be carried out through an authorization from the Federal Government. The exploration authorization is granted through a licence issued by the Director General of the Departmento Nacional de Produção Nacional or DNPM as it is commonly referred to. DNPM is the federal agency in charge of implementing the country's exploration and mining, fostering the mining industry, granting and managing exploration and mining titles and monitoring the activities of exploration and mining companies.

Exploration licences may be for areas up to 10,000 hectares and be granted for a period of up to three years depending on the substance being sought. Nickel qualifies for up to the maximum area and 3 years. The term (3 yr) can be renewed once, at the discretion of the DNPM, upon its review of an interim Partial Exploration Report (Relatorio Parcial de Pesquisa - RPP) from the licence holder regarding exploration conducted to date which justifies further exploration.

Prior to the termination of the exploration licence, be it the initial three (3) year period or in the case of renewal its second three year period the holder must submit a Final Exploration Report (RFP) on the results of the work to DNPM. DNPM may then decide to: (i) approve the report, when it shows the existence of a resource which can be both technically and financially developed; (ii) dismiss the report, when the exploration work undertaken was insufficient or due to technical deficiencies in the report; (iii) file the report, when it has been proved that there was no deposit which may be both technically and/or financially developed; or (iv) postpone a decision on the report in the event the existence of a resource has been demonstrated, but for technical and/or financial reasons development of the property is not feasible at the time.

Item (iv) above, the decision to postpone a decision on the Final Exploration Report is referred to as Sobrestamento. With this decision DNPM will fix a time period in which the interested party will be required to submit a new technical - financial feasibility study of the deposit. This is normally a three year period (decree -portaria 21/97). The penalty for not meeting the deadline will be the archiving of the RFP and liberation of the area. If the new study does not demonstrate technical-financial feasibility DNPM may grant the interested party an extension to the time limits or declare the area free and available for claiming if they believe there are third parties who could feasibly mine the deposit. If the new study demonstrates technical-economic feasibility the Final Exploration Report (RFP) will be approved and the holder of the licence will have one year to apply for a mining concession.

4.3.2 Licencing details

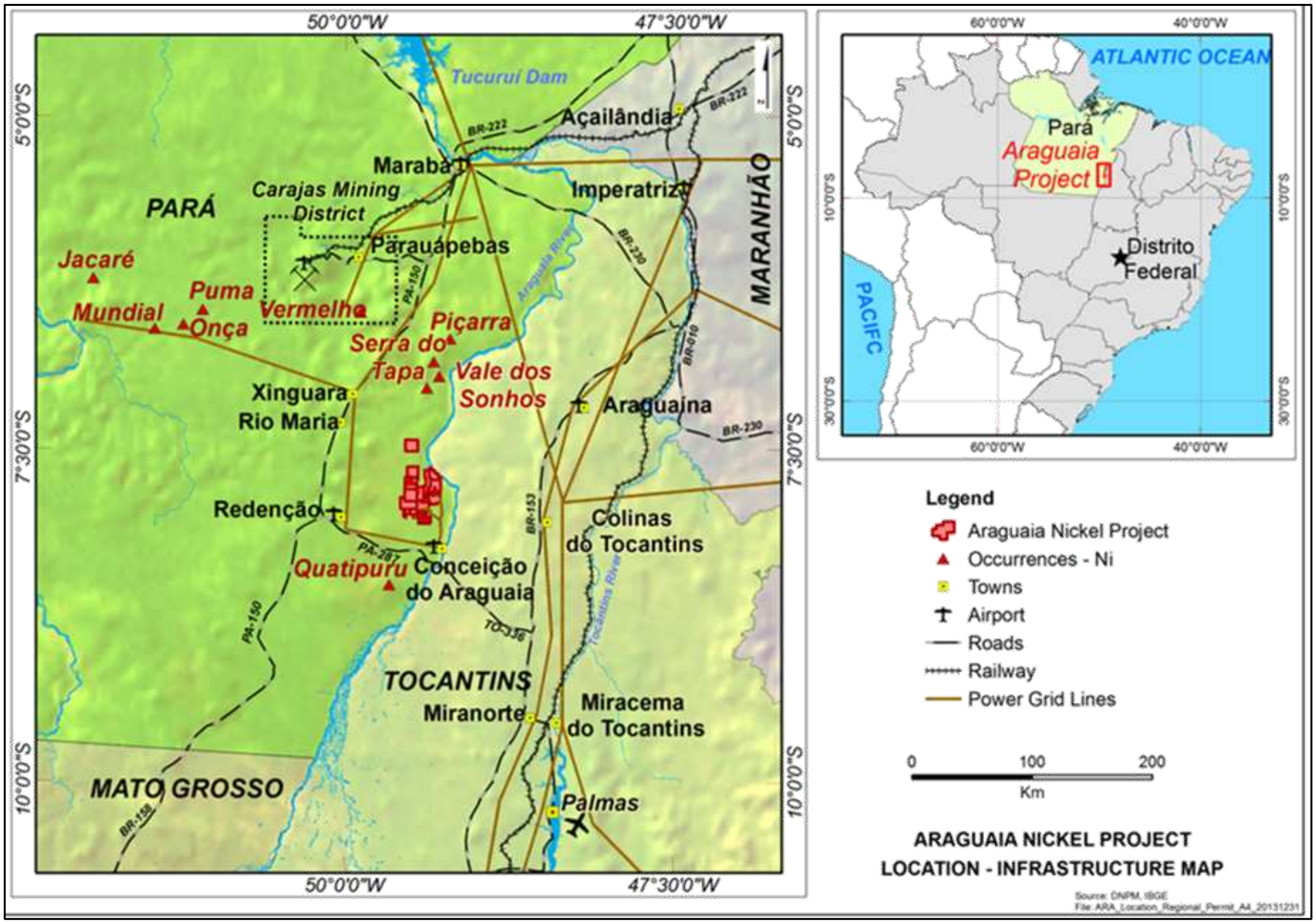
The Project is wholly owned by HZM through its Brazilian subsidiaries Araguaia Niquel Mineração Ltda and HZM do Brasil Mineração Ltda. It comprises fourteen exploration licences encompassing an area of approximately 1130 km^2 that extends approximately 70 km in a north-south direction, and 35 km in an east-west direction.

As part of the transaction that took place in August 2010 to acquire the Teck Araguaia licences, HZM took 100% control of the Lontra exploration licences, previously held in partnership with a number of Brazilian entities.

In July 2011 the licences held by Pan Brazilian Mineração Ltda and Curionópolis Mineração Ltda. were transferred to HMZ in an agreement with Lara Exploration Limited.

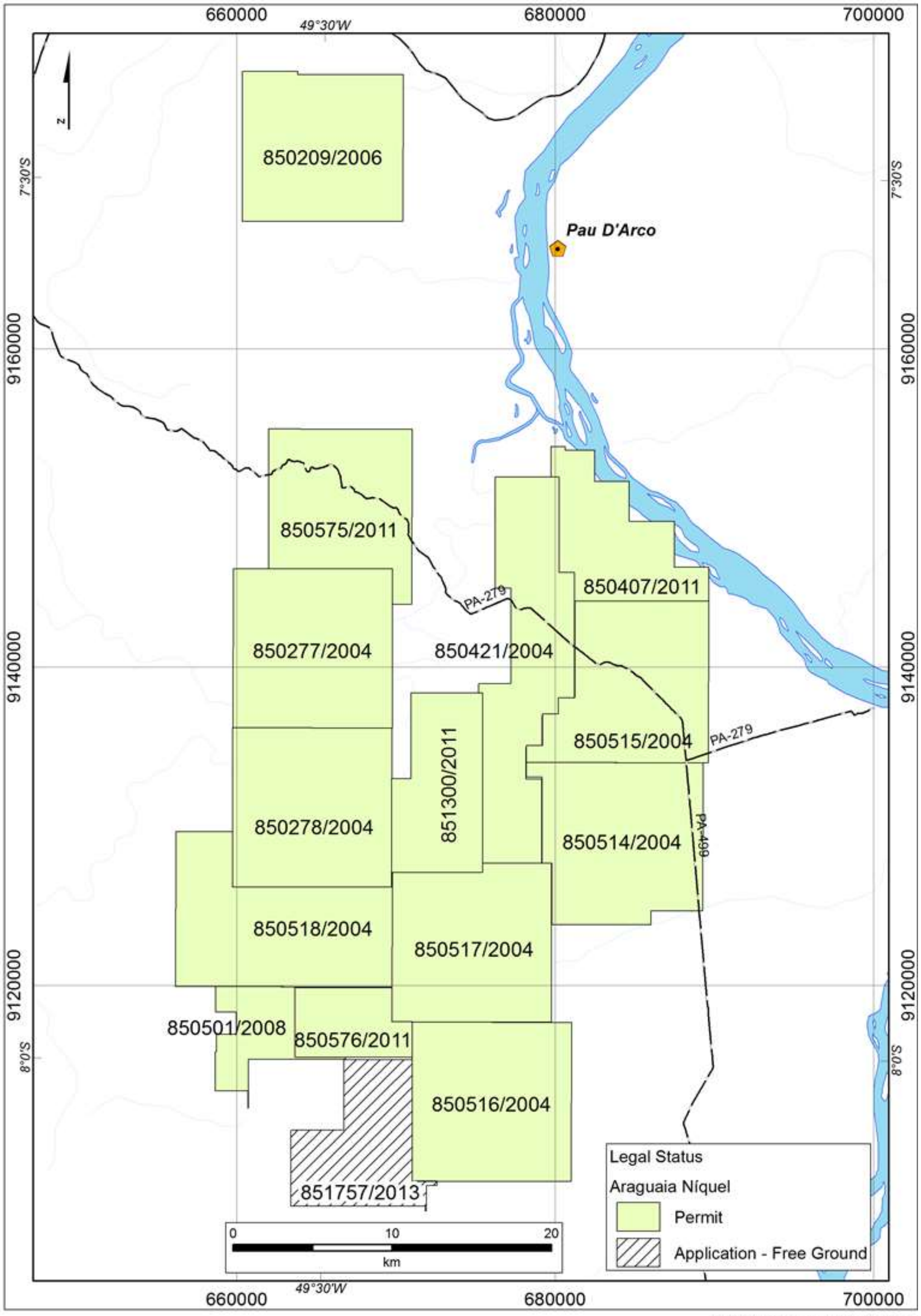
Tenement details are presented in Figure 4.2 and Table 4.1 below.

Figure 4.1 Project Location - infrastructure and other nickel laterite deposits



Source: DNPM, IBGE

Figure 4.2 Project Licence Map



Source: HZM, 2013

Table 4.1 Permit summary

Process Number	Permit / Request Holder	Area (ha)	Phase	Permit Publication Date	Deadline	Project	Comment
850.209/2006	Araguaia Níquel Mineração Ltda	9371.47	1st	21-06-11	21-06-14	Araguaia	
850.277/2004	Araguaia Níquel Mineração Ltda	10000.00	2nd	02-03-06	03-05-13	Araguaia	Suspension (Sobrestamento) - published 09-05-13
850.278/2004	Araguaia Níquel Mineração Ltda	10000.00	2nd	02-03-06	03-05-13	Araguaia	Suspension (Sobrestamento) - published 09-05-13
850.407/2011	Araguaia Níquel Mineração Ltda	5567.00	1st	21-06-11	21-06-14	Araguaia	
850.421/2004	Araguaia Níquel Mineração Ltda	9593.00	2nd	23-06-06	04-08-14	Araguaia	
850.501/2008	Araguaia Níquel Mineração Ltda	2560.00	2nd	16-09-09	16-09-12	Araguaia	Negative exploration report filed- awaiting DNPM decision
850.514/2004	Araguaia Níquel Mineração Ltda	9861.00	2nd	17-02-05	09-06-12	Araguaia	Suspension (Sobrestamento) - published 26-02-13
850.515/2004	Araguaia Níquel Mineração Ltda	9361.00	2nd	17-02-05	09-06-12	Araguaia	Suspension (Sobrestamento) - published 26-02-13
850.516/2004	Araguaia Níquel Mineração Ltda	10000.00	2nd	17-02-05	09-06-12	Araguaia	Suspension (Sobrestamento) - published 26-02-13
850.517/2004	Araguaia Níquel Mineração Ltda	9657.00	2nd	17-02-05	09-06-12	Araguaia	Suspension (Sobrestamento) - published 26-02-13
850.518/2004	Araguaia Níquel Mineração Ltda	9985.00	2nd	12-08-05	12-04-14	Araguaia	
850.575/2011	Araguaia Níquel Mineração Ltda	8174.00	1st	21-06-11	21-06-14	Araguaia	
850.576/2011	Araguaia Níquel Mineração Ltda	2945.00	1st	21-06-11	21-06-14	Araguaia	
851.300/2011	Araguaia Níquel Mineração Ltda	5791.80	1st	11-12-13	11-12-16	Araguaia	
851.757/2013	Araguaia Níquel Mineração Ltda	5690.00	Applica -tion			Araguaia	filed on 17-10-13 (replaces 850682/2007)

In Table 4.1, it is noted that seven of the licences show 2013 deadline dates in the sixth column. Though these dates are correct the licences are in good standing as the Final Exploration Reports in these cases were filed prior to the deadline date and are under evaluation by DNPM. Brazilian law protects the processes while they are being examined, a decision is reached and the decision published in the government gazette. In the comments column note that six of the seven licences are shown to be in suspension or sobrestamento which was mentioned in the opening overview on the mining code. The licences with sobrestamento in Table 4.1 will remain in good standing for the three year period granted, or until such time as the revised technical financial study is filed with DNPM.

HZM has not acquired any surface land rights for the Project but the company has agreements in place with the principle landowners for surface access rights covering the main deposits. Under the Brazilian Mining Law there is a compulsory purchase mechanism for surface land rights over mining projects in the event that suitable terms cannot be agreed between the landowner and Company. HZM currently has good working relationships with the principal landowners

4.4 Agreements and encumbrances

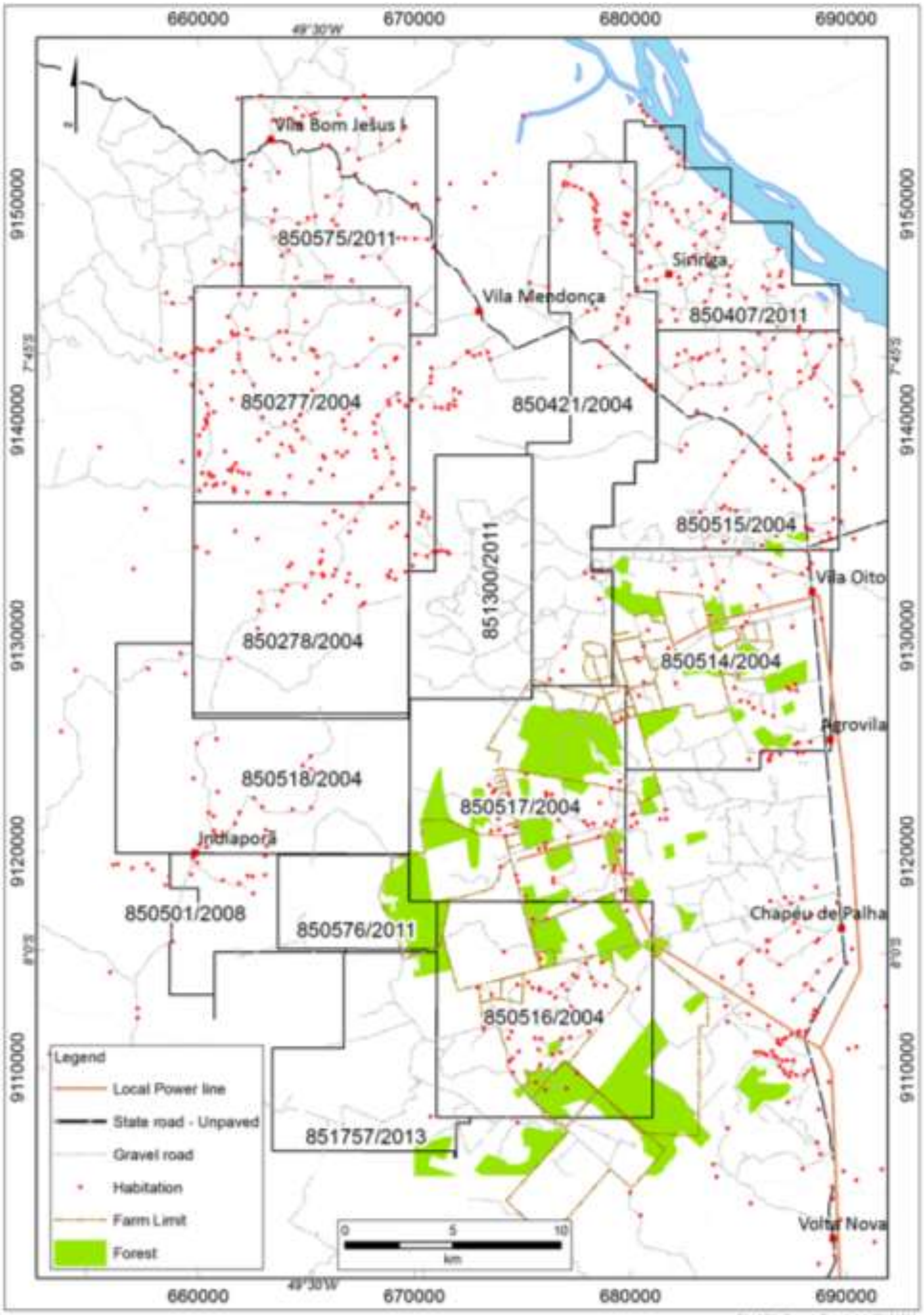
Agreements are in place with local farm landholders that allows access to land and conduct exploration with the minimum of disturbance.

4.5 Environmental obligations

HZM has obtained the necessary permits and authorizations to conduct exploration. An exploration operation licence or permit is required and issued by the state environmental agency. A water permit (Outorga) is required for drilling. When drill spacing in forested areas at less than 100 m spacing a Vegetation Removal Authorisation (ASV) permit is required. This requires a vegetation inventory of the forested areas that will be drilled, so that future recovery needs are quantifiable. HZM is renewing the Operation Licence (“LO”), and water permits and is in the process of obtaining an ASV for the next stage of drilling with drillhole spacing at 50 m or less. Figure 4.3 shows the principal forested areas within the Project.

The area is not subject to any environmental or native title reserves.

Figure 4.3 Map showing forested areas within the project licences



Source: HZM, 2013

In general, to develop a mining project, the Project must undergo a three-stage environmental licencing process. The State environmental authority is in charge of licencing a mining project, as opposed to the Federal environmental authority.

A Preliminary License ("LP") must be obtained at the planning stage of the mining project. An Environment Impact Assessment ("EIA-RIMA"), which is the equivalent of a Social and Environmental Impact Assessment ("SEIA"), and a plan for the restoration of degraded areas must be prepared at this stage. Public hearings are called to present the EIA-RIMA to the communities and authorities. Following the public hearing the State Environmental Council may or may not approve the issue the LP. The LP usually imposes conditions that the mining company must comply too. By granting the LP the environmental authority acknowledges that the project is environmentally acceptable. At this stage the environmental authority will also set the amount of the environmental compensation, which is a minimum of 0.5% of the projected development investment.

HZM has completed the collection and analysis of all of the baseline social and environmental data specified by the state environmental agency. This analysis together with the PFS and its characterization the envisioned mine project, provides the base on which the SEIA can be prepared. The ANP SEIA has concluded the 1st part of the SEIA which is the presentation of the socio-economic and environmental setting that the mine development will occur as well as the characterization of the mine project from installation through closure. Part 2, the impact assessment has been started.

The second stage of the environmental licensing process is the Installation Licence ("LI") stage. During this stage the mining company must produce an Environmental Control Plan ("PCA"), among other documents and submit it to the environmental authorities. Once the PCA is approved, the LI is granted, usually under certain conditions. The mining company may start construction of the mine, plant and infrastructure. A mining concession can only be granted by the Minister of Mines once the mining company has obtained the LI.

The last stage of the environmental licensing process is the one related to the LO. The LO is granted once the environmental authorities are satisfied that the development and construction were completed in accordance with all the conditions of the LI and that the PCA is correctly implemented. The LO authorizes a mining company to mine, process and sell (as well as other ancillary activities that may be described in the licence), from an environmental viewpoint.

5 Accessibility, climate, local resources, infrastructure and physiography

5.1 Access

The Project is centred approximately 07° 37'S and 49° 24'E or 675000E - 9120000N (Figure 5.1) and extends towards the south eastern border of Pará State with Tocantins State. Access to the site is via the Conceição do Araguaia – Floresta road (PA 449) which passes through the Project area. The Project area is supported by a system of dirt roads which have been developed to service existing cattle properties.

The Project can be reached by local flights from airports at Palmas (Tocantins State), and Redenção or via Belém/Marabá. There is an airport at Conceição do Araguaia, where HZM have their main field office but currently no scheduled flights are available.

5.2 Proximity to population centres and transport

Population density within the Project area is sparse and comprises solely of isolated farms. The Project is approximately 45 km north of the town and municipality of Conceição do Araguaia which has a population of 46,206 (source: Instituto Brasileiro de Geografia e Estatística - IBGE) which hosts commercial and municipal services required to support the town population including hospitals, hotels, restaurants, food markets and other such amenities.

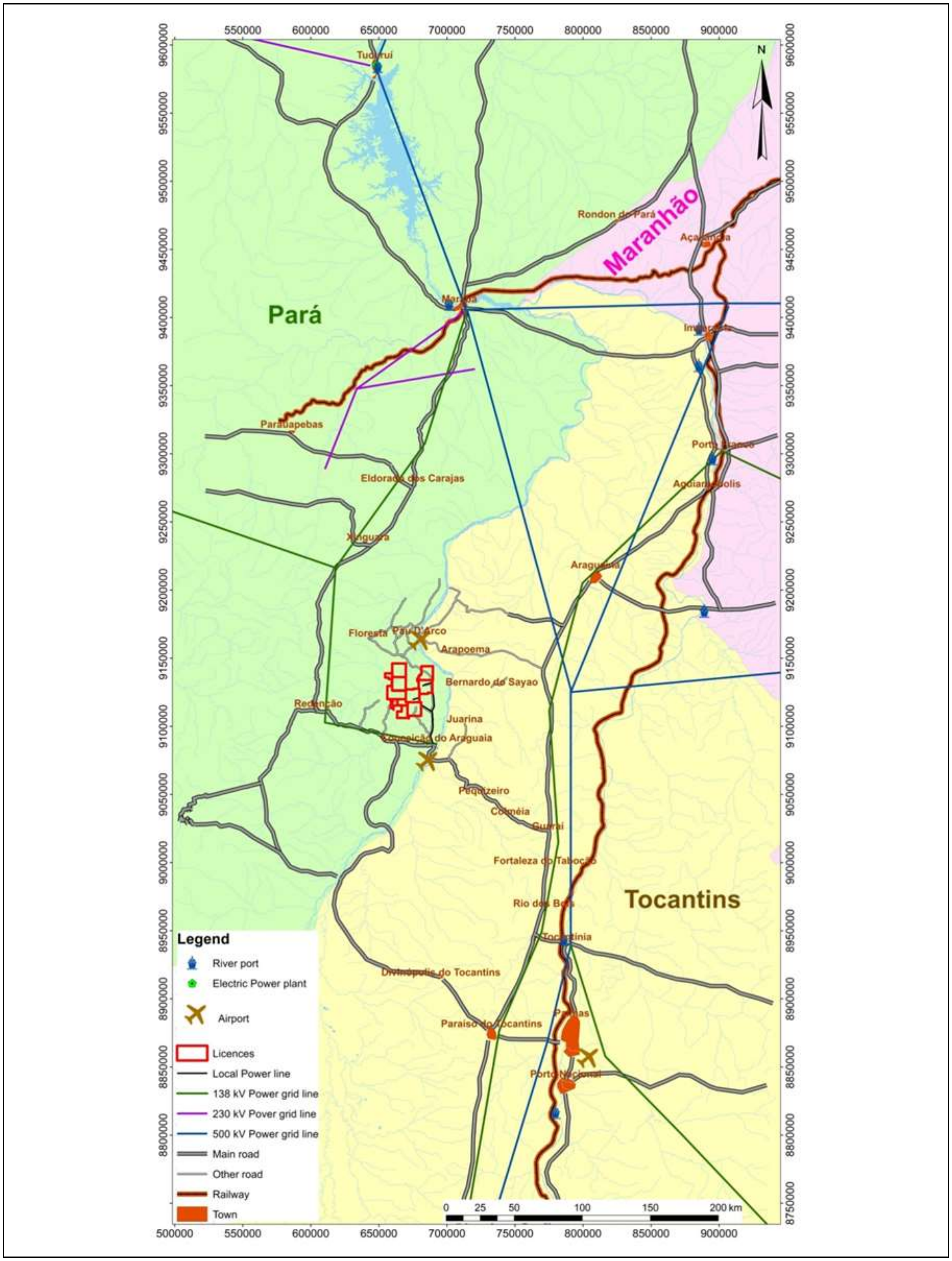
The town and municipality of Redenção is 110 km west of the Project area. Redenção is considered the nearest business centre and supports a population of 79,010 (Source: IBGE) with additional amenities required to support a larger population and business centre.

The city of Goiania is approximately 1,200 km to the South and is the traditional centre business centre for most activities in the region. Goiania is 170 km southwest from Brasilia which is the federal capital of Brazil.

Transport in the region is typically vehicular traffic which includes road haulage for supply of goods to and from the region. In addition, there is an existing rail system which was predominately developed for bulk material handling to support the iron ore industry together with other agricultural and commercial goods.

The Araguaia River is being developed as a water transport route with locks currently under construction at the Tucuruí dam allowing barging between Marabá and the sea port of Barcarena. The locks have been commissioned (late 2012) and are functional. However, the locks are barely used since the work necessary to make the Tocantins River, above the dam, safe for transportation has not been completed.

Figure 5.1 Araguaia Project location and regional infrastructure



Source: Audet, M A, et al 2012

5.3 Climate and length of operating season

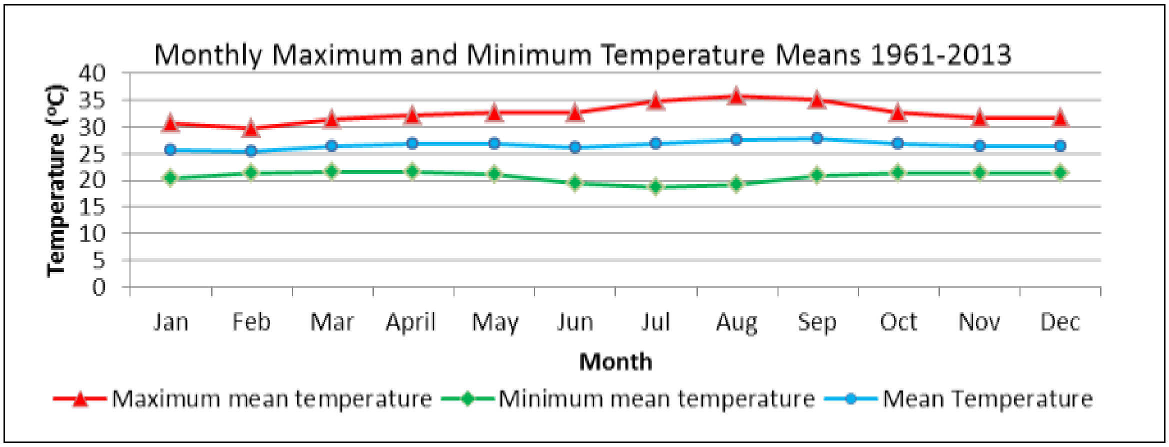
According to the Köppen classification, the climate in the municipality of Conceição do Araguaia is equatorial super-humid *Am* type, very near the transition boundary for *Aw.* The average annual temperature is 26.3 º C (Figure 5.2), with relative high humidity fluctuations between very rainy and very dry seasons going from 90% to 52%. The dry season from June to October is followed by heavy rain from November to May, with an annual rainfall of around 2,000 mm.

According the National Meteorological Institute (INMET) the lowest temperature registered in Conceição do Araguaia was 9.2º C on 3 June, 1964 and the highest temperature of 41.3ºC was registered on 15 September, 2010. The highest accumulated rainfall in 24 hours, 164.6 mm, was registered on 11 December, 2000.

The graph presented on Figure 5.2 represents the maximum and minimum mean temperature, by month, from 1961 through 2013, a 53 year period.

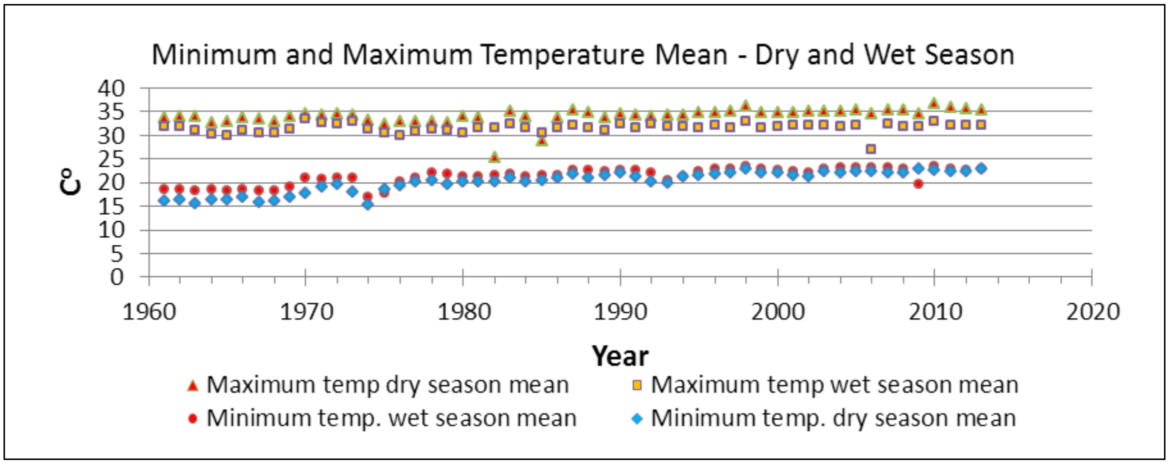
Figure 5.3 shows the maximum and minimum mean temperature, by year, for the dry and wet season for the same period.

Figure 5.2 Maximum and minimum mean temperature by month - 1961-2013



(Source: INMET)

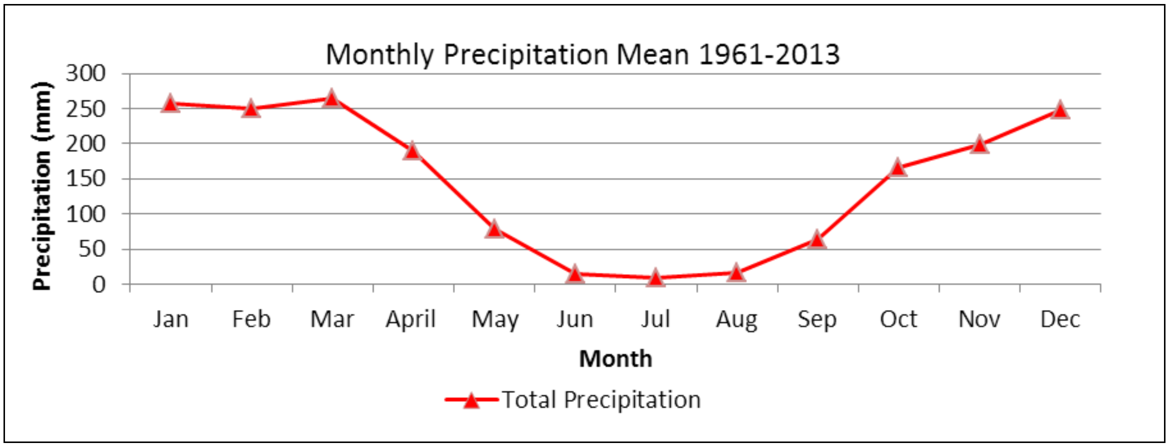
Figure 5.3 Maximum and minimum mean temperature for dry and wet season



(Source: INMET)

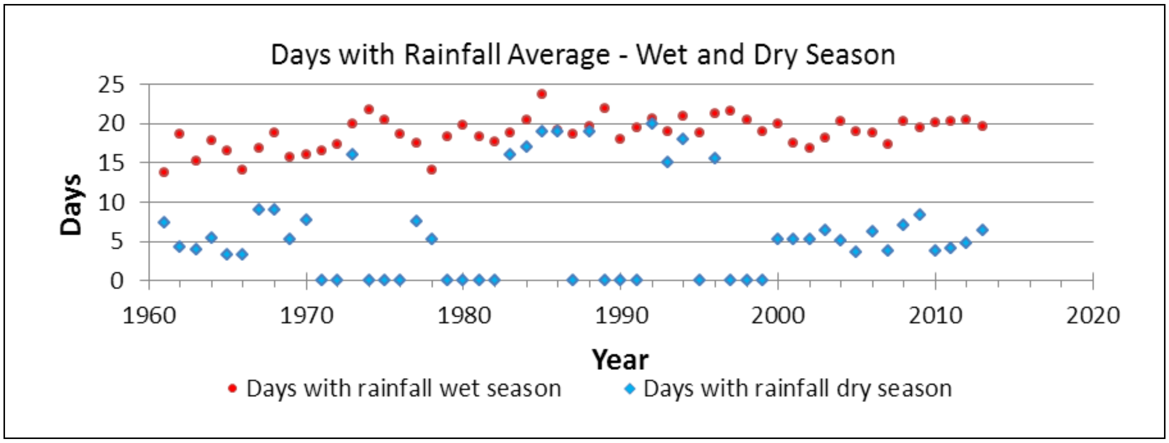
The graph in Figure 5.4 represents the average monthly precipitation for the period of 1961 through 2013, and Figure 5.5 the average number of days with rainfall for the wet and dry season. Figure 5.6 shows the total precipitation for the dry and wet seasons for this period.

Figure 5.4 Monthly precipitation mean by month - 1961-2013



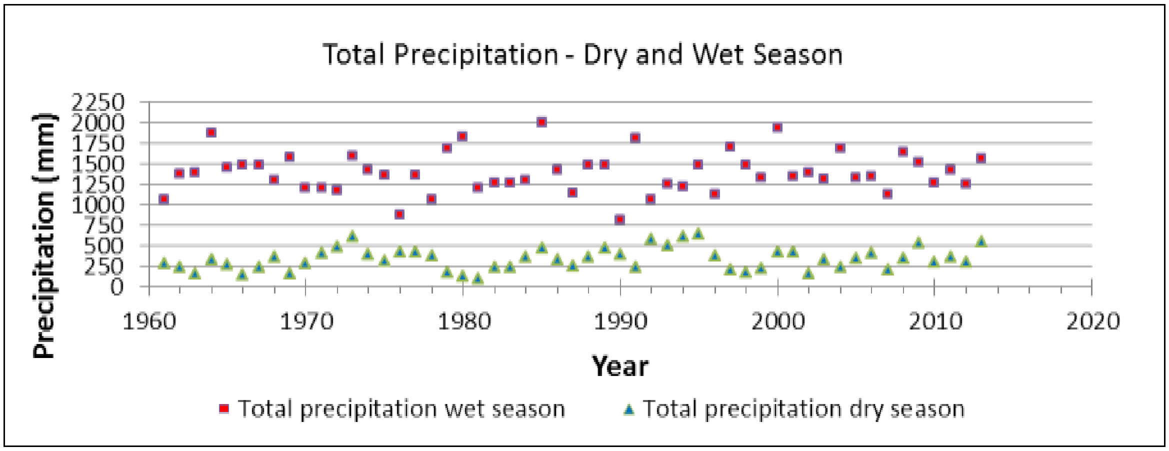
(Source: INMET)

Figure 5.5 Days with rainfall average - wet and dry season



(Source: INMET)

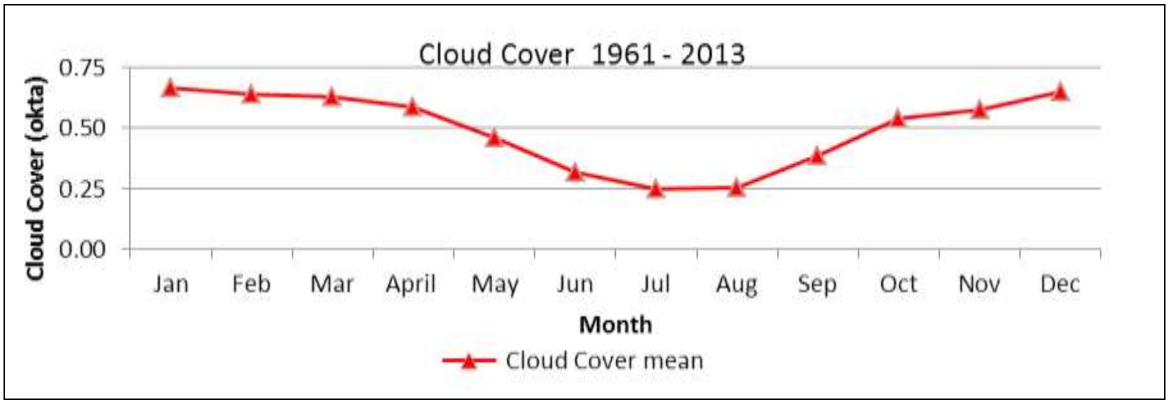
Figure 5.6 Total precipitation wet and dry season



(Source: INMET)

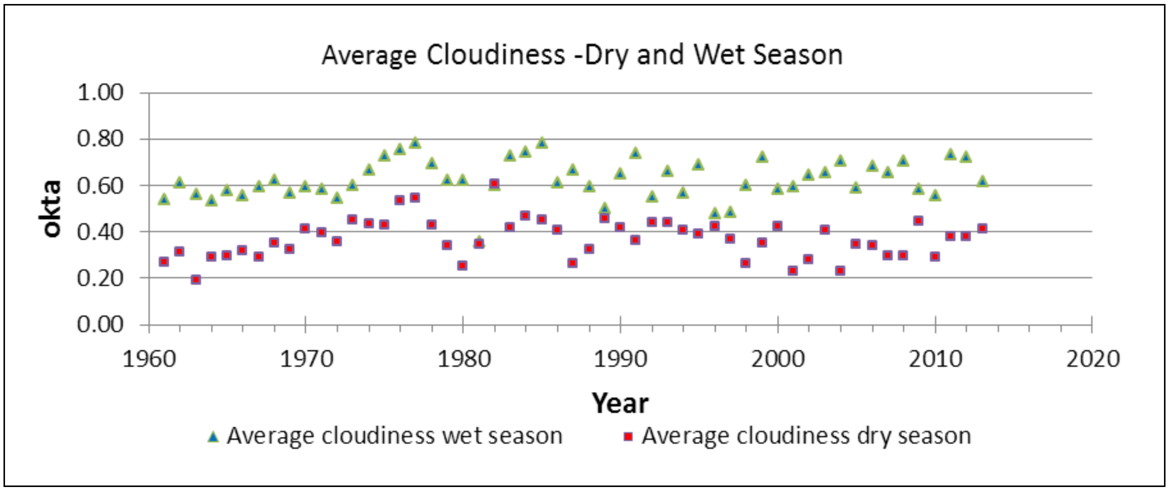
Figure 5.7 represents the average cloudiness, by month, for the period 1961 through 2013, and Figure 5.8 the average cloudiness, by year, for the dry and wet seasons.

Figure 5.7 Average cloudiness between 1961 and 2013



(Source: INMET)

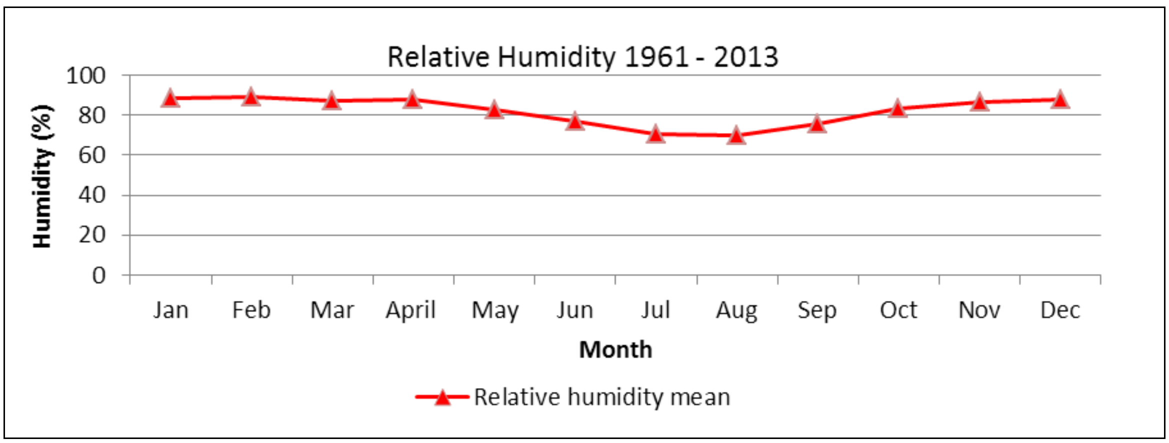
Figure 5.8 Average cloudiness for wet and dry season



(Source: INMET)

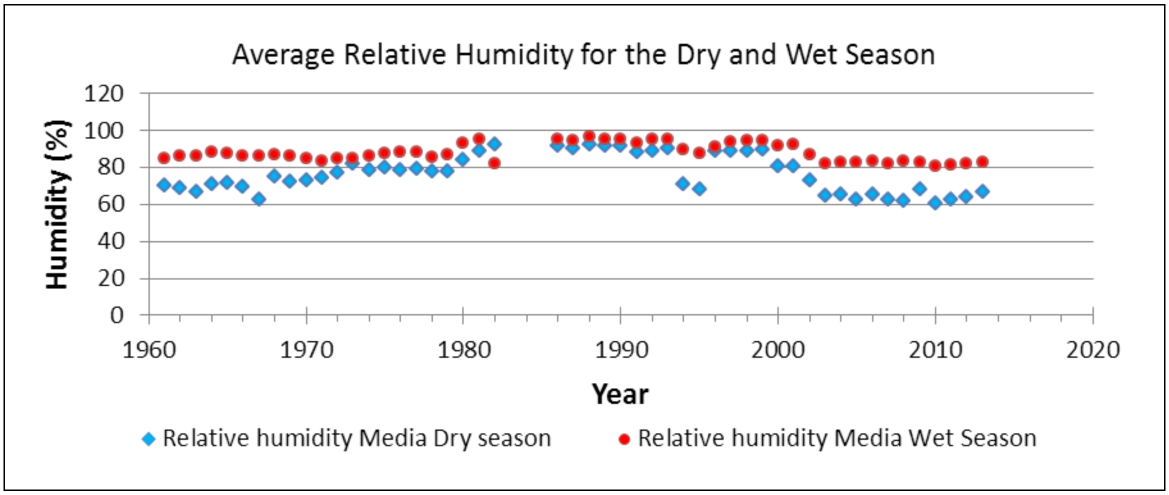
Figure 5.9represents the average relative humidity, by month, for the period 1961 through 2013, and Figure 5.10 the mean relative humidity, by year, for the dry and wet seasons.

Figure 5.9 Average relative humidity by month 1961-2013



(Source: INMET)

Figure 5.10 Average relative humidity for wet and dry season



(Source: INMET)

Consideration has been given in the PFS for the wetter periods that may affect mining and processing productivity.

Mining productivity has been based upon 362 mine production days with six being accounted for poor conditions due to rain.

Generally, it is accepted that the plant will be operating 24/7 for 365 days per year, but availability is different for different parts of the plant due to expected equipment maintenance and other scheduled and unscheduled downtime:

- Ore receipt / crushing and screen / homogenising - 75% availability
- Reclaiming from crushed ore stockpile to dryer – 85% availability
- Drying and screening – 85% availability
- Calcining (rotary kiln) – 90% availability
- Smelting – 90% availability
- Refining – 95% availability
- Metal granulation – 95% availability.

Experience from other mining operations in the vicinity has been considered.

5.4 Surface rights

As of the date of this report, HZM has not acquired any surface land rights for the Project but the HZM has agreements in place with the principal landowners for surface access rights covering the area with the deposits and proposed plant site considered in this PFS. Under the Brazilian Mining Law, there is a compulsory purchase mechanism for surface land rights for mining projects in the event that suitable terms cannot be agreed between the landowner and Company. However, HZM has good working relationships with the principal landowners.

5.5 Infrastructure

5.5.1 Power

The area is well-serviced with a 500 kV transmission line currently linking the Tucurui power generation plant, which has a generation capacity of 8,300 MW, to the national grid at Marabá, Imperatriz and Colinas. Colinas, will also be link to the Serra da Mesa power generation plant, with 1,300 MW generation capacity, and to the national grid in approximately 5 years. It will also link to Belo Monte with an average generation capacity of 4,500 MW and the transmission line which is currently approximately 7,000 MW capacity will expand to 11,000 MW.

Power for the Project will be via a grid connection at the Colinas substation. Potential opportunity exists for a connection to the Belo Monte – Paráuapebas – Miracema transmission line and shorten the distance to the Project site to approximately 35 km.

5.5.2 Roads

The main roads that connect the Amazonian region to the various waterways, ports, and train terminals throughout the country are the BR -153 (Belém- Brasília) and the BR-251. The BR-153 is a highway of approximate 4,335 km, connecting Rio Grande do Sul, in the south, to Pará in the north. Through the states of Goiás and Tocantins, the highway is paved and has good trafficability, except for a small stretch near Xambioá (TO), where the track is uneven. The BR-215 is a highway connecting Bahia to southern Mato Grosso state. This highway connects to BR-153 at Rialma (GO). In Goiás, the paved sections are discontinuous, with several areas occurring as graded natural bed.

The Project site is located some 150 km away from main BR-153 highway but is also supported by an existing road system. The Project area is supported by an existing infrastructure of dirt tracks used for access by local farms.

5.5.3 Rail

Rail infrastructure consists of a network which is owned by VALEC-Engineering, Construction and Railways SA (Valec) which is a public company controlled by the Ministry of Transport.

The North – South railway (FNS) extends from Anápolis in the state of Goias to Açailandia in the state of Maranhão (see Figure 5.1). In Açailandia it is linked to the Carajás railway, which is used by Vale to transport iron ore to the port of Itaqui, in São Luis.

Valec has granted the concession for exploration and operation of the North-South (FNS) railway to Vale for 30 years, adding an additional 1220 km, which passes approximately 180 km from the project site, to the original concession of approximately 570 km from Açailandia to – São Luis / Itaqui.

Currently, the railway is used for transportation of pig iron, fertilizers, fuel and other goods as well as passengers between the towns of Paráuapebas and São Luis. There are three scheduled trains per week, along the distance of approximately 1,000 km, which takes approximately 15 hours. However, delays of several hours are not uncommon.

5.5.4 Ports

Regionally, the Project is supported by existing port facilities at the port of Itaqui, in São Luis. This facility is well served by a railway and road infrastructure and is 1150 km from the Project and is currently in use by Vale who use this location to support their Carajás operation.

Other port facilities exist but Itaqui provides the key location for in-bound and out-bound logistics for imports of bulk consumables, such as coal and potentially export of Fe-Ni product.

5.5.5 Water

The provision of water for the project is described in Section 18.10.2. Water sources include river water, a closed-system for the plant and recycling effort from mine dewatering, water flows from waste and slag dumps and other catchments from facilities around the site. There are known periods of drought during the summer months but this PFS considers a number of engineering solutions which would be elaborated further in subsequent studies. The hydrological characteristic of the area suggests that the provision of water does not pose a problem for this operation with an appropriate design solution.

It should be noted that less than half of the dwellings in Conceição do Araguaia have access to running water. The main water supply sources are wells or springs located on the properties. Other sources include rain water stored in cisterns, water trucks, dams or streams.

The Project does provide an opportunity for Conceição do Araguaia to improve clean water availability as a sustainable solution.

5.5.6 Mining personnel

It is anticipated that mining personnel for the Project would be sourced from the population of Conceição do Araguaia and the general locale. The region already supports a mining community which provides an opportunity for the Project to attract a skilled workforce from operators through to technical and managerial staff. It is envisaged that additional skilled employees would be sourced throughout Brazil and internationally as the operational tolerances for processing and smelting will require a requisite high skill-set.

5.5.7 Slag and waste dumps

The Project requires a slag dump to facilitate waste from the smelting process and a number of locations have been considered as part of the PFS. The total volume of slag produced over the 25 years is estimated at 9.93 Mm^3. This will be accommodated in a waste dump as part of a waste management programme. A number of locations have been assessed for their viability. The slag dump will be lined with a clay liner which preliminary geotechnical testing indicates as suitable after engineering and compaction. High clay content material is readily available on site. This is described in further detail in Section 18.0.

Waste rock from mining, (defined as nickel grade less than 0.8%), is planned to be disposed in waste dumps and back into the mined out pits. Each mining area has a planned waste dump and sites have been identified. This is described in further detail in Section 16.4.4.

5.5.8 Process plant sites

Four locations were considered for the process plant and smelter locations. A site has been selected which was established in terms of required utility and the best economics. These are described in further detail in Section 18.

5.5.9 Communications

Cellular phone coverage is available in Conceição do Araguaia and the immediate surrounds but is intermittent to not available in the Project area. Internet connections are available at the HZM field office and in Conceição do Araguaia supported by local internet service providers.

5.6 Physiography

The Project area is characterised by undulating hill systems with elevated plateaus separated by shallow valleys with relief being typically in the region of 100 m. The highest elevation of the Project area is 360 m (AMSL) and the lowest elevation is 217 m. The topography across the Project area is considered reasonable level with gentle gradients with a downward slope across the project area, from West to East, towards the Araguaia River.

The original area is known as cerrado and has never been considered part of the Amazon rainforest. Some of the plateaus are used for cash crops such as pineapple plantations with the lower lying areas predominantly used for cattle ranching.

Views of typical relief, vegetation and land use are presented in the following photographs Figure 5.11, Figure 5.12, and Figure 5.13 below.

Figure 5.11 View of general Project area looking north-northwest

Source: Audet, M A, et al 2012

Figure 5.12 View to the south-east over Pequizeiro (main zone)

Source: Audet, M A, et al 2012

Showing large ferricrete plain surrounded by valleys (fault zones), contact to sediments to the west and east of the plain, photo taken from elevated position (silicified zone), semi-dense forest covering the centre zone of Pequizeiro (main).

Figure 5.13 View over the north part of Pequizeiro (main zone)

Source: Audet, M A, et al 2012

Showing view to the west, semi dense forest at the border of mineralised zone, showing position of 3 drill rigs at the end of dry season (Dec 2010)

6 History

6.1 Prior Ownership

The history of the mineral tenements that now comprise HZM's Project are summarised as follows:

- HZM commenced exploration by way of regional stream sediment sampling in 2006. This resulted in the discovery of seven nickel targets and resulted in the awarding of three exploration licences in 2007, held 100% by HZM. Two contiguous licence areas then held by a private Brazilian company (LGA Mineração e Siderugia) were acquired in a partnership agreement with HZM in 2007. Collectively the five mineral tenements were known as the Lontra project covering 22,556 hectares. The Lontra project included Northern Target, Raimundo Target, Southern and Morro Target (Figure 6.1).
- In July 2010 HZM entered into an agreement with Teck Resources Limited (Teck) to acquire Teck Cominco Brasil S.A. which owned 100% of Teck's Araguaia project. The merged Lontra and Araguaia tenements comprised eleven licences and licence applications covering 73,000 hectares and eight significant mineralised zones. Around the same time HZM acquired the remaining 50% interest in the Lontra project not then held by HZM.
- In July 2011, in an agreement with Lara Exploration Ltd (Lara), HZM acquired 100% of the licences containing the Vila Oito West and Floresta discoveries.

Subsequent to the acquisition of the Teck Araguaia and Lara tenements, in excess of 15 targets are identified within HZM Project (Figure 6.1).

6.2 General description of exploration work undertaken by previous owners or operators

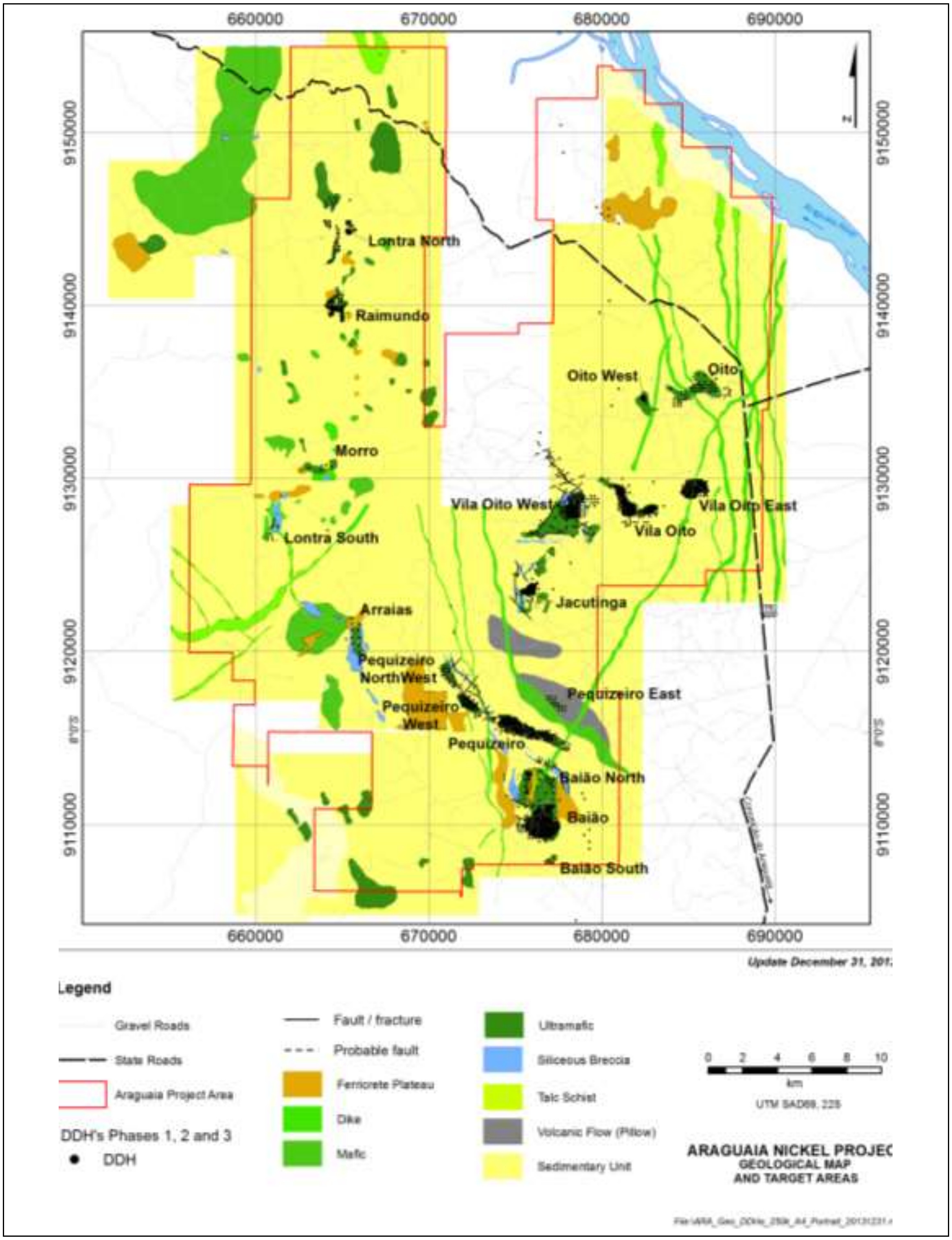
The following has been summarised from Audet, M A, et al (2012).

6.2.1 Introduction

Exploration work for nickel laterite deposits historically consisted of geologic mapping, soil geochemical sampling (both surficial grab samples and/or shallow auger drilling); with subsequent reverse circulation drilling (RC) and diamond core (DDH).

The initial phases of exploration and discovery of nickel laterite mineralisation in the Araguaia Project area were reported by Lara (Barry 2006). Subsequent to that report, exploration activities was carried out by Xstrata (formerly Falconbridge) until early 2007, Lara in 2007 and by Teck from September 2007 until November 2008.

Figure 6.1 HMZ Project combined target map



Source: HMZ, 2013

Xstrata, Lara and Teck all carried out regional geological mapping at various degrees of detail based on photo-interpretation and geophysical image interpretation complimented by field reconnaissance. The outcrop in the region is generally poor, because of the widespread laterite hard cap cover and generally deep tropical weathering. The magnetic images proved to be particularly useful in this terrain as they readily outline the magnetic ultramafic bodies and late cross-cutting mafic dyke structures that show a strong contrast with the regional non-magnetic phyllite sequences.

The Xstrata Airborne Magnetics and Radiometric Survey covered the Vila Oito and Floresta Blocks. This produced a high resolution, or high-density survey, with flight lines flown at 100 m above the ground and with east-west orientated flight lines at 500 m spacing along the trend. The survey was flown in several stages (June 2004, February and June 2005) by Prospectors Ltd. In addition Xstrata flew an area of 440 km^2 in the Vila Oito Block using helicopter-mounted Versatile Time Domain Electromagnetic (VTEM) survey system.

The data for the Vila Oito Block was made available to Lara when Xstrata pulled out of the Joint Venture. Lara was able to purchase the data for the Floresta Block and other areas of interest held in the region in early 2007. This data was an essential part of the database used by Lara to carry out an in-house evaluation and target selection for priority nickel targets across some 300,000 ha of claims controlled by Lara in 2007. This data was made available to Teck on entering into a Joint Venture agreement with Lara and was also an integral component for their nickel laterite target definition programme.

6.2.2 Lontra

The Lontra area had previously been claimed for phosphate and then iron, although to HZM's knowledge no exploration was undertaken. While ultramafic bodies are known in the Araguaia Belt the existing regional geologic maps had indicated that the Lontra area was underlain by packages of fine to coarse-grained clastic sediments.

6.2.3 Teck

Nickel exploration across the Teck Araguaia licence areas dates back to the 1970's. During the period work conducted by CVRD (Docegeo) and VALE led to the discovery of a small ultramafic intrusive hosted nickel laterite deposit at Serra do Quatipuru (DNPM 850514/2004) (Figure 6.1).

In the 1990's Rio Tinto Desenvolvimento Mineral (RTDM) conducted exploration for magmatic nickel mineralisation associated with ultramafic rocks in the region of Couto Magalhaes (DNPM 850514/2004). Results of this work are unknown.

From 2006 until 2008 Teck Resources completed 5 main stages of exploration, including geological sampling, airborne geophysical surveys, and drilling.

Auger and reverse circulation drilling

Teck completed 46 shallow auger drillholes for initial exploration purposes. Bottom-of-hole auger samples were typically less than 1 metre due to limitations with auger penetration at depth (Bennell, 2010).

First pass irregular spaced exploratory reverse circulation (RC) drilling was undertaken by Teck in 2006 to test nickel-in-soil geochemical and airborne geophysical anomalies in identified target areas.

A total of 69 RC holes were drilled for 1,996 m testing 5 target areas at Baião, Pequizeiro, Jacutinga, Vila Oito West and Vila Oito (DNPM 850.514/2004). Positive drill results were returned for each target tested.

One metre bulk RC samples were collected in marked plastic bags from the cyclone and transported to a RC receiving area on site. Bulk samples were chipped, with chipped 1.0 m intervals being stored in compartmentalised RC wood boxes similar to core boxes for logging and future reference.

At the RC receiving area 1.0 m samples were laid out on plastic sheets to sun-dry. Once dry samples were put through a Jones riffle splitter where 50% of the sample was spilt for dispatch to the laboratory for preparation and analysis.

The remainder of the 1.0 m bulk sample was stored at the RC receiving facility on site or other Teck storage facility elsewhere. At present almost 100% of the Teck rejects, pulps and bulk samples, are stored in the HZM facility at Conceição do Araguaia.

Data from these drillholes have not been used in mineral resource estimates reported in this Technical Report.

Diamond drilling

Following positive results from the RC drill programmes, 400 m x 400 m spaced diamond drilling took place at the Baião, Pequizeiro, Jacutinga, Vila Oito West and Vila Oito targets in 2007.

Where preliminary results from drill core were positive, 200 m x 200 m spaced diamond drilling was undertaken. In November 2008, having completed the diamond drilling over selected targets, for a total of 489 holes and 11,404 m, Teck ceased exploration on the project.

Teck diamond drillholes are included in the data subsequently provided by HZM to Snowden for mineral resource estimation. The average thicknesses of mineralised intercepts calculated at a 1.0% nickel cut-off-grade for the four main sectors range from 5.12 m to 7.55 m, with maximum thicknesses varying from 13.08 m to 21.30 m.

Teck drill core handling and processing involved similar steps to that described for HZM (sections 10 and 11).

After sampling, remaining half drill core was retained and stored in the core box for future reference with sample intervals marked on the core box with the use of metal tags (Bennell, 2010).

In total, some 18,712 individual samples were taken and sent for preparation and analysis from the Teck drillholes comprising of 15,841 from DDH’s and 470 from RC drillholes (numbers include quality control standards and blanks). The remaining 2,401 samples are believed to be from surface sampling.

Geological logging

Drill core was photographed and logged prior to sampling. Evidence suggests core was dry photographed only.

Drill core and RC geological logging intervals were determined by lithology rather than set intervals and recorded using hardcopy graphical logging sheets to capture pertinent geological information for each deposit including lithology, facies and texture.

Geological information recorded as hand written sheets was then transferred to Microsoft® Excel spreadsheets direct to an AcQuire database.

For geotechnical logging Teck recorded core recovery, RQD and expansion.

Drill cores were routinely measured for magnetic susceptibility, using a Terraplus Inc. KT-9 digital magnetic susceptibility meter. Magnetic susceptibility was measured for all core at 20 cm intervals. This information was stored in the database for use in geological logging and further deposit analysis and interpretation.

Topographic survey

In 2006 Teck commissioned Prospectors Aerolevantamentos e Sistemas Ltda to undertake geophysical surveys across the Araguaia project area and as part of this survey a digital 10m topographical coverage of the project area was also acquired. Data for the surveys were recorded using an RMS DGR 33A data acquisition system, a Magnavox/Leica MX 9212 twelve channel GPS receiver mounted on a twin engine Piper Navajo/Chieftain PA31-350.

Teck drillholes were positioned with handheld GPS and surveyed using DGPS.

No downhole surveys were conducted due to the short, vertical nature of the drillholes.

6.2.4 Lara

Between 2006 and 2008, joint venture (JV) work with Falconbridge Ltd and, later Teck, resulted in the discovery of nickel laterite mineralisation at Vila Oito, between the Teck and Lontra discoveries, and at Floresta to the north.

In 2009 Lara reported that exploration programs on their Araguaia Nickel project conducted since 2006 have identified bodies of nickel laterite mineralisation in the southern part of the Vila Oito licence block and in the southwest of the Floresta licence block, with 64 diamond drillholes and 55 auger drillholes completed to date (Lara 2009).

Teck completed significant exploration immediately to the south and east of Vila Oito and in January 2009 presented a conceptual grade and tonnage estimate for the combined targets in their properties together with the Vila Oito target of Lara (now called Vila Oito West by HZM).

6.3 Historical mineral resource, mineral reserve estimates and production

There are no historical mineral resource, mineral reserve estimates and production to be reported for deposits within the HZM Project.

7 Geological setting and mineralisation

7.1 Geological setting

7.1.1 Regional geology

The project lies within the Neoproterozoic Araguaia Fold Belt. This belt is a large north to south trending orogenic zone along the contact of the Amazon Craton to the west and the São Francisco Craton to the east (Figure 7.1). The Belt is 1,000 km long and 150 km wide and its evolution is believed to be contemporary with the Brazilian thermal event at the Neoproterozoic boundary.

The belt comprises metamorphosed and deformed marine-clastic sediments of the Tocantins Group and can be split into two halves based on the degree of metamorphism present. The more highly metamorphosed Estrondo Formation comprises the eastern half of the belt while the western half displaying lower levels of metamorphism is termed the Couto de Magalhães Formation.

The Estrondo Formation is dominated by greenschist to amphibolite facies grade metamorphosed sediments with occasional banded iron formations, carbonates and exposures of Achaean basement. Proterozoic granites intrude the eastern belt.

The Couto de Magalhães Formation contains weakly metamorphosed, marine pelites with local carbonate, iron-rich, and mafic to ultramafic bodies.

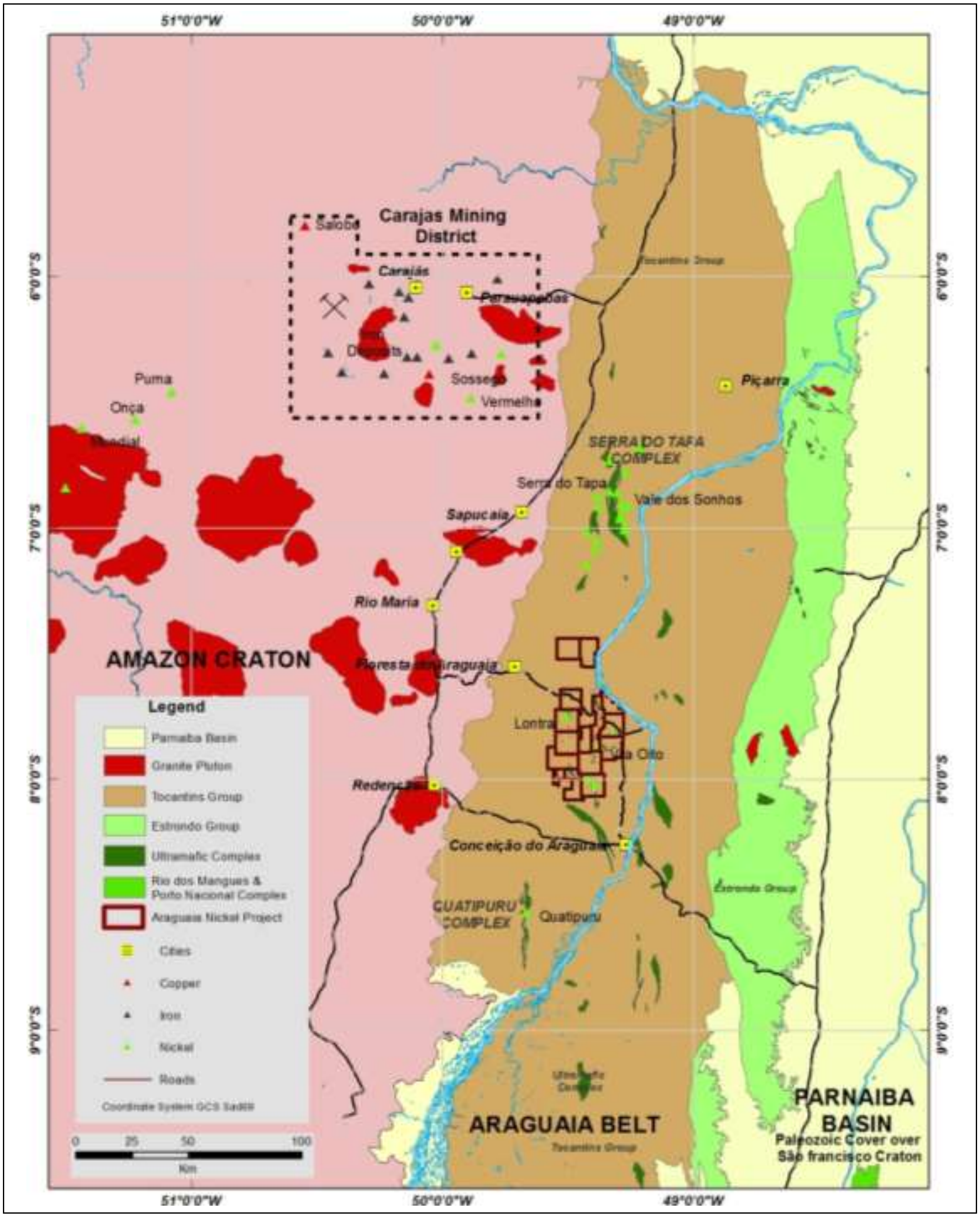
7.1.2 Project geology

The local geology has largely been interpreted from airborne geophysical survey data, soil sampling data and mapping by HZM and previous owners of the tenements. Various types of metasediments cover the vast majority of the licence area. Large plateau areas, varying in size from a few hundred square metres to several square kilometres, and generally capped with a hard iron rich duricrust that is occasionally silicified are frequently developed over mafic and ultramafic bodies. These bodies and numerous northwest-southeast to north-south trending lineaments have been identified from magnetic data and outcrop. These bodies are often bounded by a siliceous breccia. Bodies of pillow lava and other volcanic material also exist. The area is cut by numerous mafic dykes.

A distinctive lateritic sequence is developed over ultramafic and mafic rocks within the Project area and the same sequence can be recognised at each of the target sites though the thickness and extent of each facies and the complete sequence itself may vary from location to location. The sequence can be split into 6 main facies types: soil, ferricrete, limonite, transition, saprolite and fresh rock as well as numerous sub-facies.

The interpreted project geology is shown in Figure 7.1 and discussed further in Section 7.4.

Figure 7.1 Regional geolocical map



Source: HZM 2013

7.2 Lithologies and mineralisation

7.2.1 Physical criteria

The lithological facies of the laterite profile are described as follows:

Soil Horizon

A dark brown layer rich in humus material constitutes the uppermost soil layer. This layer comprises occasional ironstones as well as organic material derived from the breakdown of plants and the networks of fine plant roots. The chemical composition of this layer is characterized by low Ni and MgO. The soil material forms a thin horizon that generally averages less than 1.0 m to 1.6 m thickness and is absent in many places (Table 7.1).

Table 7.1 Soil thickness statistics derived from Project drill logs

Area[1]	Minimum (m)	Maximum (m)	Average thickness (m)
VOW	0.00	5.05	1.21
VOI	0.00	4.97	1.34
VOE	0.00	3.48	1.14
JAC	0.00	5.69	1.61
PQZ	0.00	7.00	0.59
PQW	0.00	2.50	0.63
BAI	0.00	5.61	0.74

Ferricrete Horizon

This facies comprises a hard, cohesive, red to yellow brown material, high in hematite/goethite and often containing magnetite with occasional chromite. Ferricrete is present as both an unconsolidated horizon with ubiquitous haematitic pisolites (code PF) and a cemented goethite rich horizon containing distinctive worm burrows (code LF). Ferricrete is present in virtually all locations with thickness varying from absent to approximately fifteen metres; commonly two to three metre thick horizons are developed (Table 7.2 and Table 7.3).

[1] *VOW (Vila Oito West); VOI (Vila Oito); VOE (Vila Oito East); JAC (Jacutinga); PQZ (Pequizeiro); PQW (Pequizeiro West); BAI (Baião)*

Table 7.2 Pisolithic ferricrete thickness statistics derived from Project drill logs

Area	Minimum (m)	Maximum (m)	Average thickness (m)
VOW	0.00	5.30	1.21
VOI	0.00	10.49	2.80
VOE	0.00	7.35	1.70
JAC	0.00	3.02	0.95
PQZ	0.00	9.59	2.44
PQW	0.00	5.00	1.87
BAI	0.00	11.30	2.33

Table 7.3 Cemented ferricrete thickness statistics derived from Project drill logs

Area	Minimum (m)	Maximum (m)	Average thickness (m)
VOW	0.00	6.87	1.10
VOI	0.00	8.21	1.81
VOE	0.00	14.44	1.83
JAC	0.00	3.70	0.64
PQZ	0.00	12.80	1.38
PQW	0.00	9.49	4.32
BAI	0.00	14.06	3.92

Limonite Horizon

The limonite layer follows immediately below the soil or the ferricrete layer, and consists of deeply weathered material. The upper part of the limonite, sometimes called Red Limonite, is a red-brown or more often, chocolate-brown clayey material with little internal structure although layering has been observed. The material consists entirely of fine-grained minerals of silt to clay fractions, predominantly hydrated iron oxides.

The lower part of the limonite, sometimes called Yellow Limonite, is yellow-brown to orange coloured and generally has a more compact appearance than the red limonite. The yellow limonite rarely contains coarse fragments of weathered material. Both Red and Yellow Limonite maybe well developed; alternatively only one sub-type may be present or occasionally neither.

Transitional Horizon

Three sub-facies are recognised:

- The Upper Transition facies (UT) is a dark red to brown red, cohesive, soft, plastic, and fictile material, with fine granulation. It is differentiated from Red Limonite by the presence of manganese oxide (up to 2%), whitish gibbsite pockets (up to 5%), and incipient texture. UT can contain up to 15% of disseminated green serpentine, which increases the nickel content in this horizon. Manganese oxide also considerably increases the cobalt and nickel content.

- The Green Transition facies (GT) predominately hosts nontronite/kaolinite minerals (approx. 85-90%) and approximately 5% Mn minerals, and is characterised by the association of green material and brown clayish material. The green material represents from 30 to 70% of the GT facies and the clayish material occurs as laminations or disseminations. The nickel content is associated with the relative presence of these materials and, usually, the greener the material the higher the content. GT is usually compact and plastic. It can also occur in a friable form, but without losing its plastic property. The clayish material is usually brown, but it can be orange/brown or reddish brown, depending on the amount of associated goethite and/or hematite. Chlorite, vermiculite, manganese oxide (asbolane), and talc can also occur disseminated (> 1%). Free silica can be present in the form of millimetre sized veins or pockets.
- The Brown Transition facies (BT) consists of approximately 40% nontronite/kaolinite, 30% Mn minerals and a portion of approximately 20% Limonite/Goethite, the latter responsible for the brownish, clayey fraction. BT is the most common transition facies and is formed by granules of millimetre or centimetre size of green to light green nontronite immersed in a brown to reddish brown clayey matrix. The material is compact and granular and presents incipient texture. The clayey matrix can represent up to 30% of material and can also occur as laminations. The nontronite granules can form cohesive aggregates, but the hardness is usually low. Manganese oxide and chlorite can sometimes occur.

Saprolite Horizon

Three sub-facies are recognised:

- **Earthy Saprolite (SAP)** is pervasively altered rock composed of hydrated Fe-oxides, serpentine and clays. Minor amounts of quartz, olivine and chromite are present. Primary rock textures have been obliterated with some visible relics; trace amounts of clay/serpentine pseudomorphs after pyroxene or olivine. The horizon is usually reddish to brownish.
- **Rocky Saprolite (SROC**) is hard saprolite and is competent dark green to greyish rock of weathered peridotite and with moderate saprolite alteration, occurring mostly along fractures. Primary olivine and orthopyroxene exhibit patchy replacement by fine-grained hydrated iron oxides and amorphous silica. Granular textures are well preserved and the material consists of cores of angular fresh rock (20–50%) with successive rims of increasingly altered material. Silica boxwork is rarely seen in the hard saprolite but a bright green garnierite staining can often be seen on fracture planes.
- **Silicified Saprolite (SIS)** is a saprolitic material with high silica content. The hardness and colour of the material vary according to the silicification intensity, but the material usually presents moderate to high hardness and whitish brown to reddish brown colour. Sometimes, the texture is still preserved and the presence of free silica is common.

Bedrock

Bedrock has a dark green to dark brown colour and consists of massive to fractured, varyingly serpentinised peridotite, whose interface with the weathered profile can be highly irregular and undulating. Bedrock is commonly exposed along rivers and creeks and in major landslides.

7.2.2 Chemical criteria

A facies distinction by chemical composition was devised by consulting geologist Dr. Marc-Antoine Audet in 2011-12, based on factor analysis of the initial Teck data and updated thereafter using the entire core assay database. The discrimination is made using mainly Fe, MgO, SiO_2, Al_2O_3 and Ni grades (Table 7.4).

Typical Limonite-facies laterite contains 0.78% Ni, 0.11% Co, 2.4% Cr_2O_3, less than 2% MgO, 36.5% Fe and 19.7% SiO_2. The underlying Transition material typically has 1.20% Ni, 0.05% Co, 11.7% MgO, 18.3% Fe and 44.3% SiO_2. The underlying Earthy - Rocky Saprolite typically has 1.29 – 0.92% Ni, 0.04 – 0.03% Co, 18.3 – 27.0% MgO, 14.8 – 9.7% Fe and 41.8 – 42.2 % SiO_2.

Table 7.4 Average composition per facies based on Teck and HZM diamond drilling to 2013

Facies	N° assays	Ni%	Co%	Fe%	MgO%	SiO_2%	Al_2O_3%	Cr_2O_3%
Weathered Peridotite								
Soil	3,640	0.153	0.033	29.63	0.19	25.00	17.30	1.75
Ferricrete	384	0.367	0.098	47.70	0.22	9.69	9.36	2.15
Limonite	6,698	0.775	0.113	36.47	1.80	19.72	11.09	2.39
Transition	4,557	1.196	0.051	18.29	11.65	44.25	4.89	1.21
Earthy Saprolite	1,653	1.293	0.041	14.79	18.26	41.97	4.44	1.00
Rocky Saprolite	6,643	0.918	0.025	9.96	26.96	42.19	3.52	0.70
Silicified Saprolite	512	0.413	0.023	8.34	7.14	70.94	3.20	0.48
Bedrock	5,443	0.302	0.012	6.15	34.19	41.34	1.68	0.44
Other Protore								
Sediment	4,744	0.073	0.014	9.22	2.18	58.71	14.91	0.19
Quartz vein	97	0.047	0.010	2.52	0.95	93.53	0.58	0.12
CaO Rich	65	0.107	0.010	4.44	18.85	22.56	1.26	0.26
Dike Al-rich	379	0.135	0.010	5.36	3.62	61.06	15.85	0.07
Diorite	1,136	0.175	0.015	12.31	4.97	46.75	16.15	0.20

7.2.3 Loss-on-Ignition (LOI) statistics

The assay database provides LOI results for each sample. LOI average statistics for each Horizon in selected deposits are presented in Table 7.5.

Table 7.5 Average LOI statistics by area and horizon

Area	Horizon 100 (LOI %)	Horizon 200 (LOI %)	Horizon 300 (LOI %)
VOW	9.40	8.65	9.89
VOI	10.12	8.45	10.41
VOE	9.41	9.28	10.71
JAC	8.46	8.88	9.40
PQZ	10.51	10.74	19.43
PQW	10.13	9.45	10.02
BAI	10.39	9.28	10.06

7.2.4 Facies distribution

The deposits at the Araguaia project are heterogeneous as far as lateritic facies distribution is concerned (Table 7.6). The average thickness for the limonite facies ranges from 7.5 m to 11.6 m, while maximum thicknesses vary from 23.9 m to 45 m. The saprolite horizon shows similar average variations while the total thickness is highly variable. The transition horizon is thinner than limonite or saprolite and generally less continuous laterally, which is shown by the large variation in maximum thickness observed from one deposit to another.

Table 7.6 Maximum and average thickness of laterite horizons (including 2013 data)

Sector	Area	Drillholes (№)	Thickness (m)					
			Limonite		Transition		Saprolite	
			Max.	Avg.	Max.	Avg.	Max.	Avg.
South	Baião Baião South	361	23.90	7.46	17.83	4.12	28.11	6.74
Pequizeiro	Pequizeiro Pequizeiro East Pequizeiro West Pequizeiro NW	265	44.99	8.97	33.63	5.86	36.94	9.57
Centre	Vila Oito East Vila Oito Vila Oito West Jacutinga	444	32.10	8.71	48.65	5.72	54.05	10.04
North	Oito Oito West	58	31.30	11.56	25.20	6.25	37.86	9.87
Lontra	North Raimundo	144	30.55	10.38	10.25	3.15	23.70	5.16

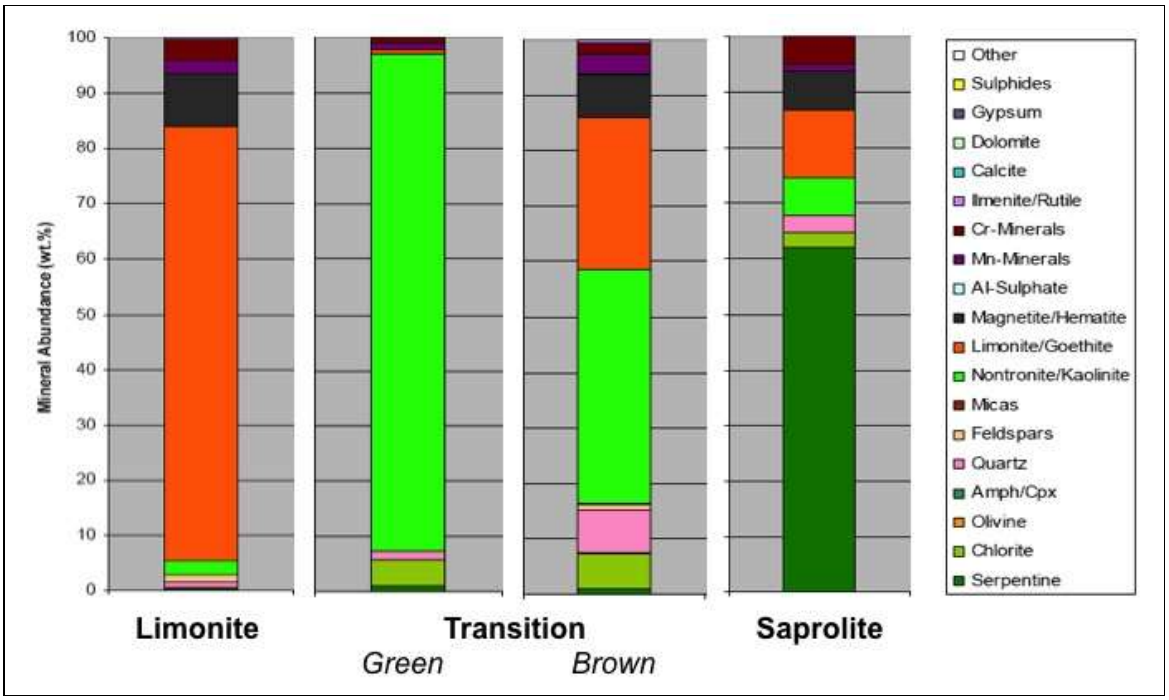
7.3 Mineralogical studies

In December 2010 SGS Mineral Services, Lakefield, Ontario, Canada (SGS), was contracted to undertake a high definition mineralogical study on four samples selected from the remaining half core drilled in the Teck exploration programmes on the Baião and Pequizeiro targets. One sample of each of the mineralised Yellow Limonite, Green Transition, Brown Transition and Saprolite facies was selected.

Techniques employed included QEMSCAN technology (Quantitative Evaluation of Materials by Scanning Electron Microscopy), X-ray diffraction (XRD), optical microscopy and electron microprobe (EMP) analyses. The main purpose of this test programme was to identify the mineral assemblage and modal abundance of the various nickel bearing horizons, as well as to determine the overall nickel deportment amongst the samples.

The results of this work were reported in July 2011 (SGS Mineral Services, 2011). The mineralogical distribution is shown in Figure 7.2 and the deportment of nickel in the principal mineralised facies is summarised in Figure 7.3.

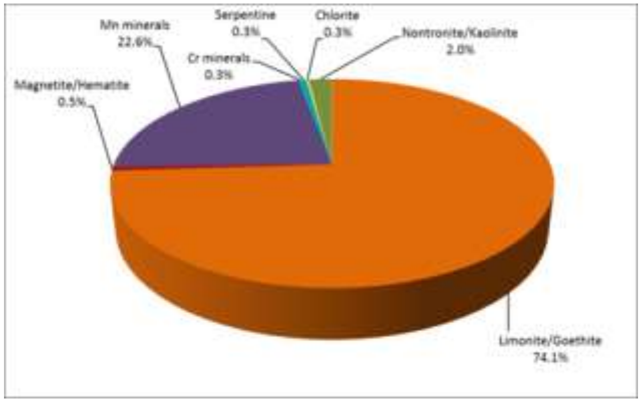
Figure 7.2 Mineralogical distribution in the principal mineralised facies



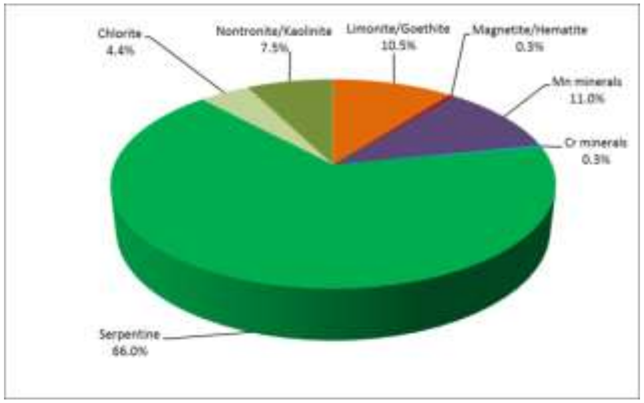
Source: SGS 2011

In December 2012 SGS was contracted to undertake an additional high definition mineralogical study on a sample split of the Transition plus Saprolite blend (HM_51T_49S) used in the metallurgical test work completed at FLSmidth. The techniques employed were identical to those used in the earlier work and included QEMSCAN, XRD, optical microscopy and EMP analyses (SGS 2013).

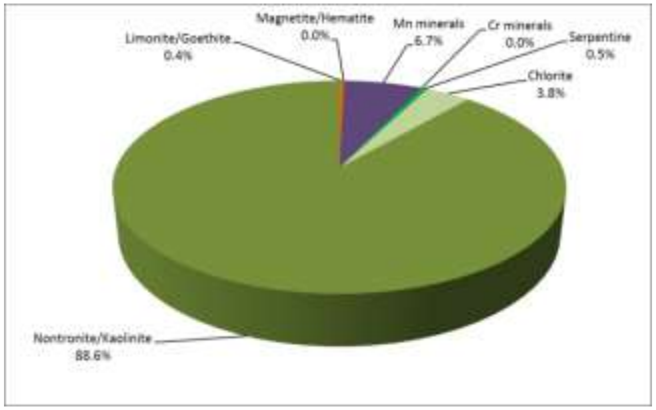
Figure 7.3 Deportment of nickel in principal mineral facies (SGS 2011)



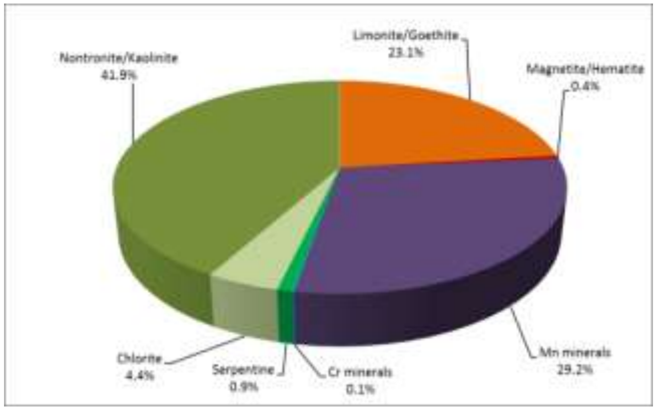
Limonite



Saprolite



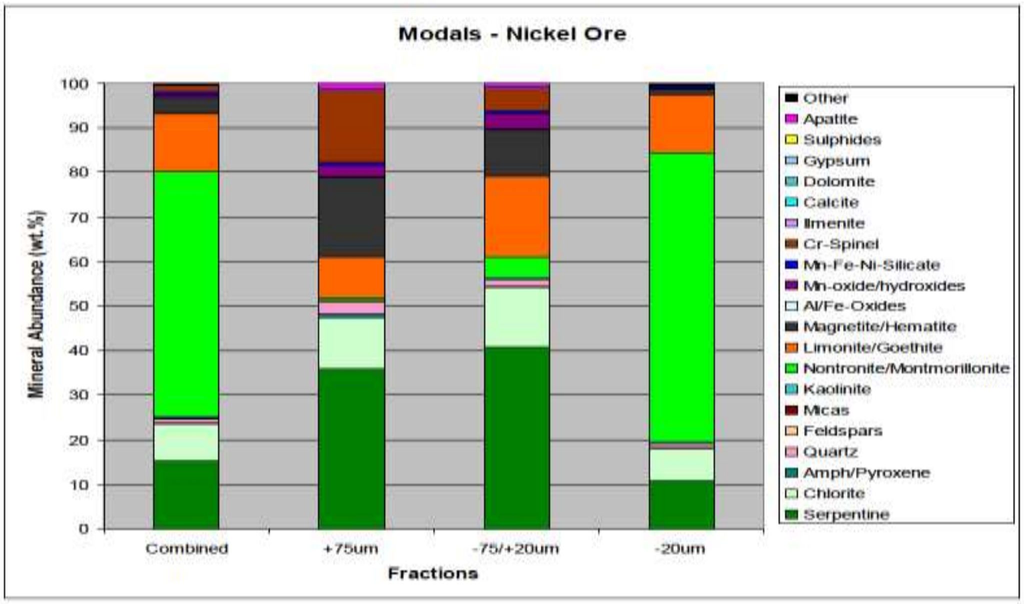
Transition – green



Transition - brown

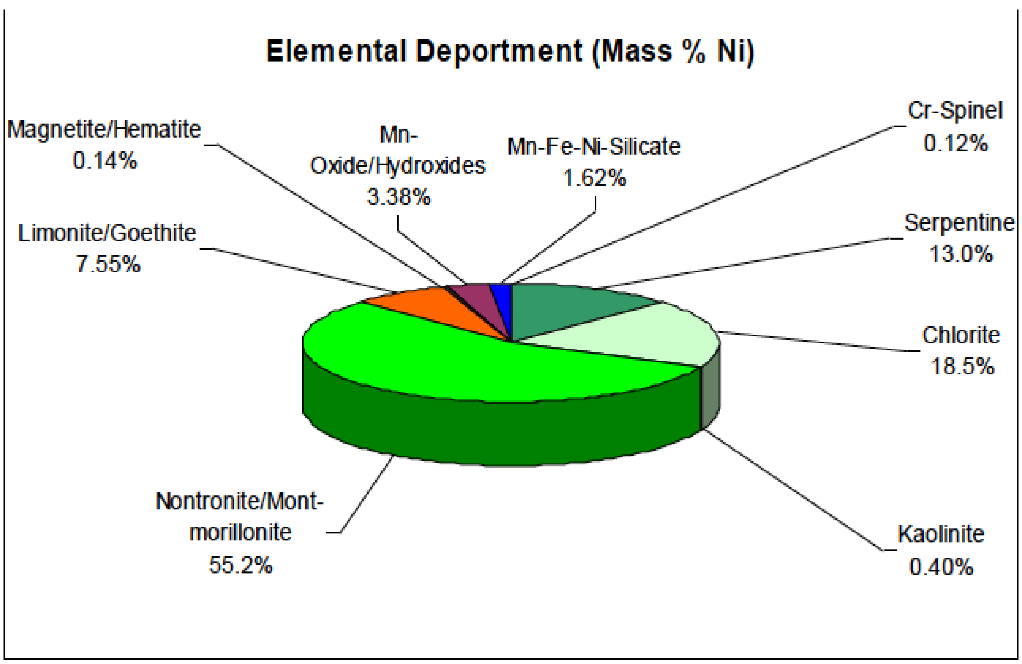
The mineralogical distribution by size fraction is shown in Figure 7.4 and the deportment of nickel in the principal mineral species is summarised in Figure 7.5.

Figure 7.4 Mineralogical distribution by size fraction in Blended Sample (HM_51T_49S)



Source: SGS 2013

Figure 7.5 Deportment of nickel in the principal mineral species in Blended Sample (HM_51T_49S)



Source: SGS 2013

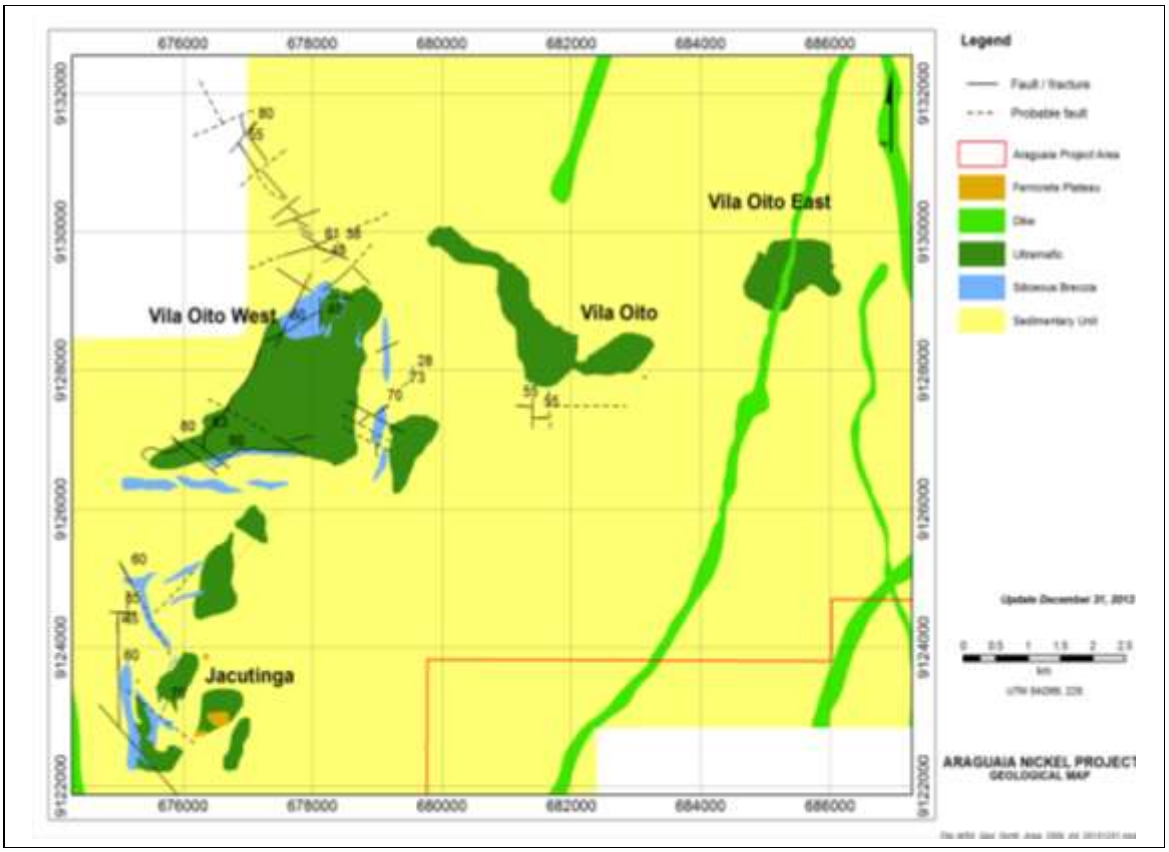
7.4 Deposit geology

The following has been excerpted from Audet, M A, et al (2012) and updated where appropriate.

7.4.1 Centre (JAC, VOW, VOI, VOE)

The Centre Sector: Jacutinga (JAC); Vila Oito West (VOW); Vila Oito (VOI); and Vila Oito East (VOE), covers an area of approximately 10 km^2 (Figure 7.6). Since VOI and VOE deposits are located in the flat areas with no discernible outcrop, only JAC and VOW have been surface mapped though the connection between these ultramafic bodies is unconfirmed.

Figure 7.6 Bedrock structural geology map of Centre Sector



Source: HZM 2013

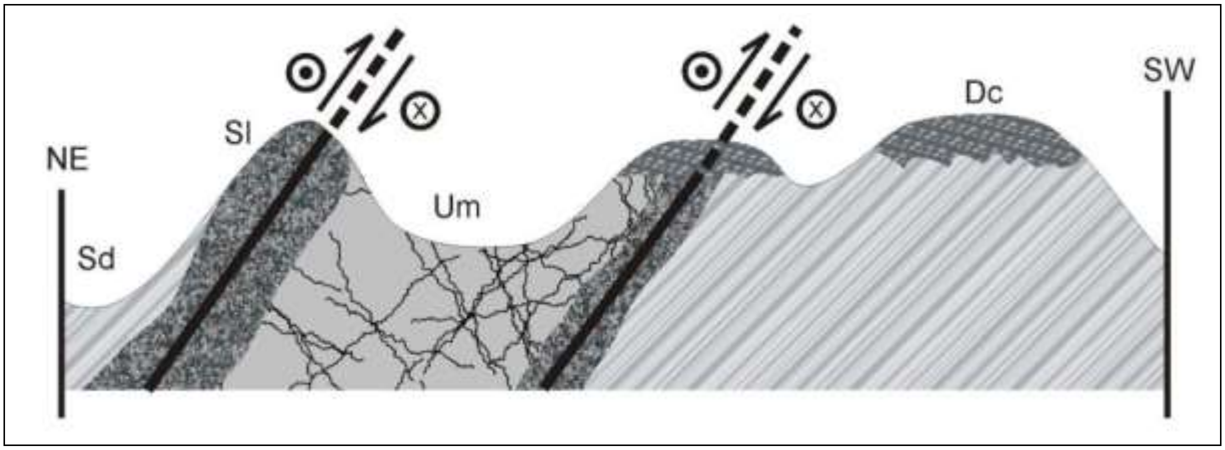
Northwest-southeast, northeast-southwest and north-south trending silica filled fault zones are found within the JAC area. A minor east-west trending steep fault cross-cuts the north-south trending fault and a northwest-southeast trending silica filled fault is the contact between ultramafic and sedimentary rocks. The conical hill is composed of both ultramafic and sedimentary rock, while the larger triangle shaped hill is mostly composed of massive silica and silicified sedimentary rocks.

In VOW, silica filled fault zones represent the sedimentary and ultramafic rock boundaries and trend northeast-southwest and east-west along the northwest and southern limits of the target respectively. Both are dislocated by northwest-southeast trending steep cross faults. Shear indicators indicate dextral movement.

A ridge, located north of VOW, exposes an oblique thrust fault zone filled by massive silica. The fault zone trends northwest-southeast and dips to the northeast at 55° to 80°. Sedimentary rocks occur on both sides of this zone, however ultramafic rocks outcrop within the fault zone covering an area of 150 x 1,000 m. A 3 km displacement separates this ultramafic outcrop from the closest similar body. Duricrust covers both sediment and ultramafic rocks in the far north-eastern area (Figure 7.7). Northeast-southwest trending, steep faults dislocate the silica ridges. Some north-south trending faults located in the southern rim dislocate northeast-southwest trending cross structures.

An east-west trending silica filled fault zone marks the only topographic high around VOI. This 200 m long hill is dislocated by a north-south trending cross fault in the middle and terminates with some north-south fractures in the eastern rim.

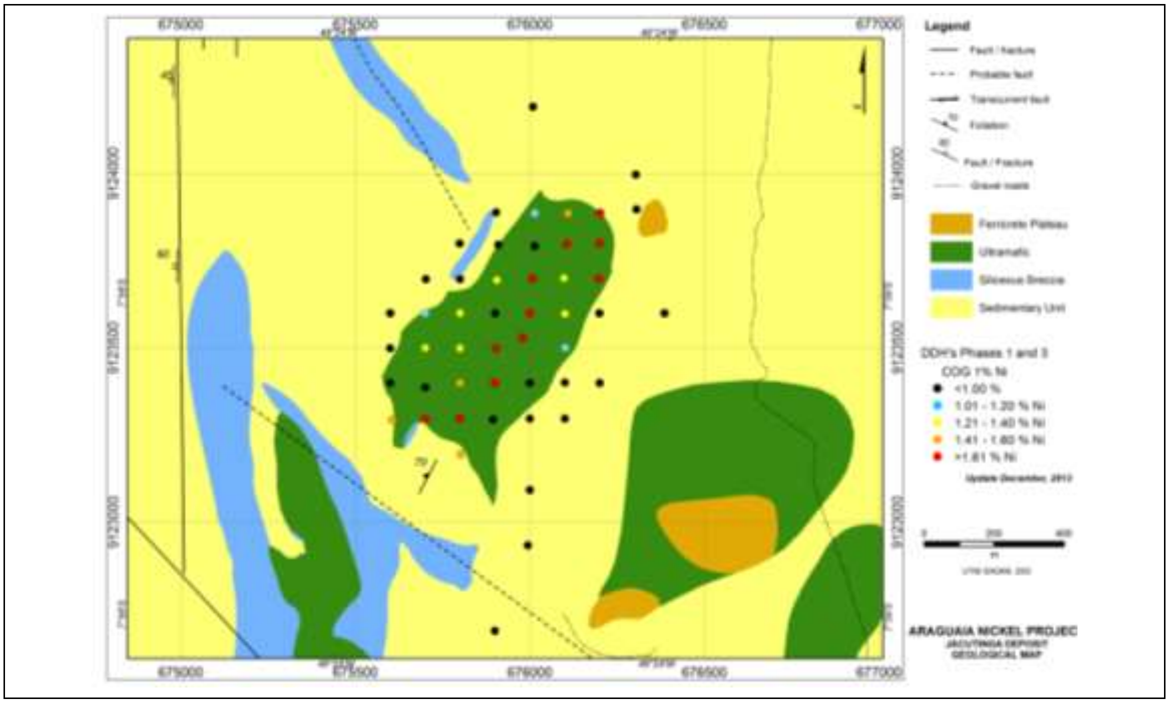
Figure 7.7 Outcrop of ultramafic rock within sinistral oblique thrust fault zone



Note: Sd: Sedimentary rock; Sl: Silica; Um: Ultramafic rock; Dc: Duricrust (not to scale)
Source: Audet, M A, et al 2012

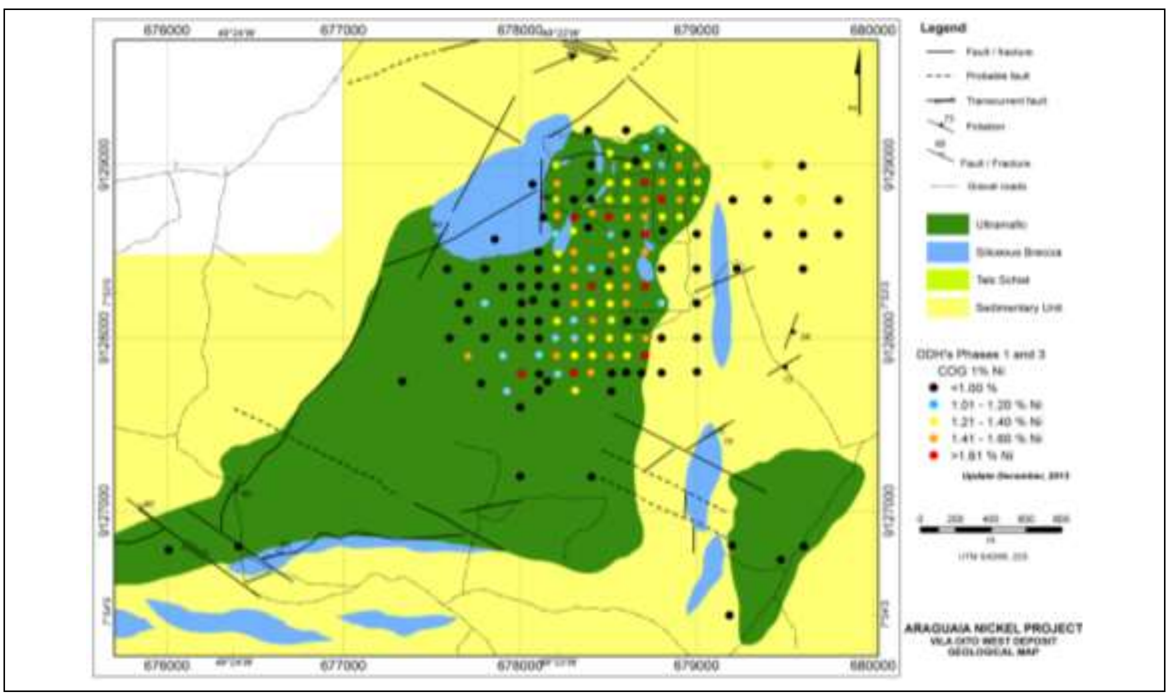
HZM provided Snowden with updated surface geology maps for each of the deposits, following the completion of the 2012-13 drilling programme (Figure 7.8, Figure 7.9, Figure 7.10, and Figure 7.11). These maps and drillhole results were used by Snowden to guide the construction of 3D wireframe interpretations of the limonite, transition and saprolite horizon contacts.

Figure 7.8 Updated bedrock JAC geology map after 2012-13 drill programme



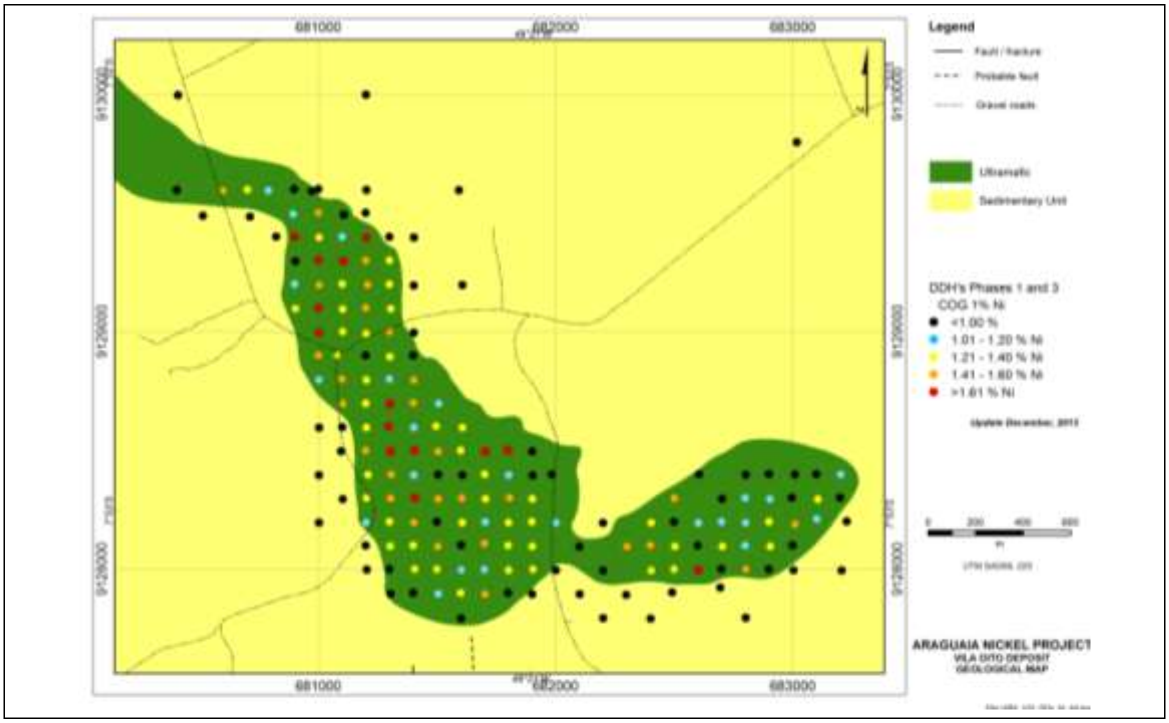
Source: HZM 2013

Figure 7.9 Updated bedrock VOW geology map after 2012-13 drill programme



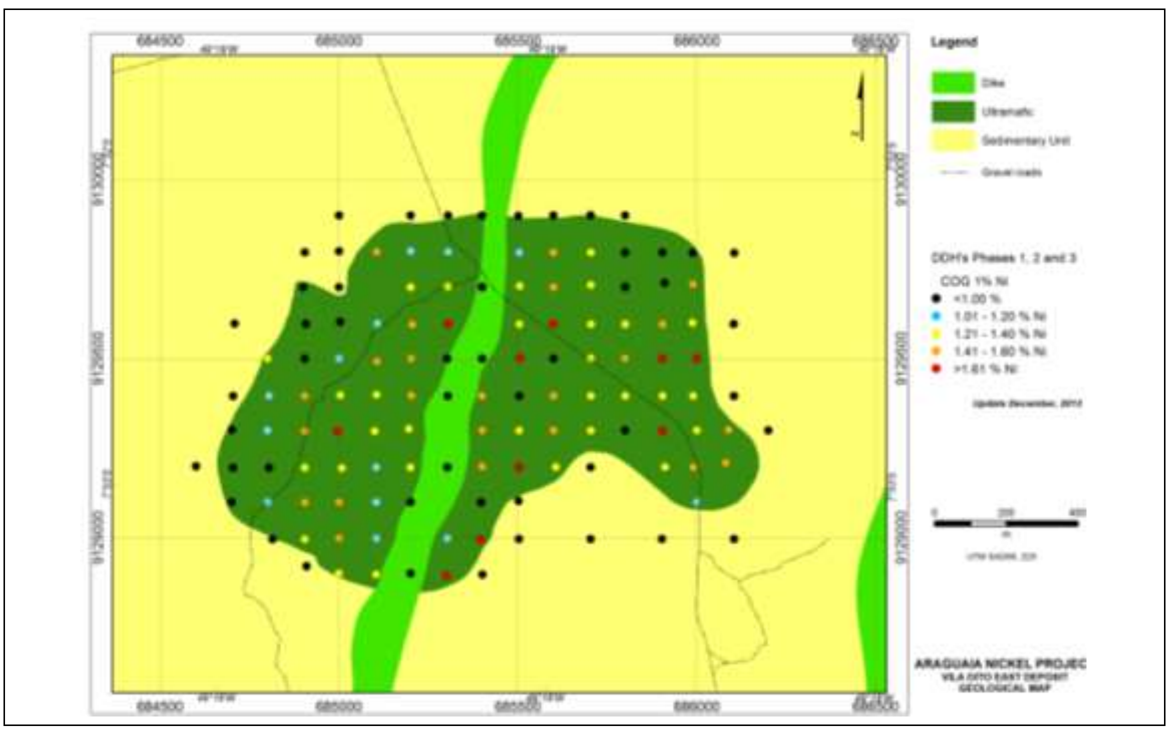
Source: HZM 2013

Figure 7.10 Updated bedrock VOI geology map after 2012-13 drill programme



Source: HZM 2013

Figure 7.11 Updated bedrock VOE geology map after 2012-13 drill programme



Source: HZM 2013

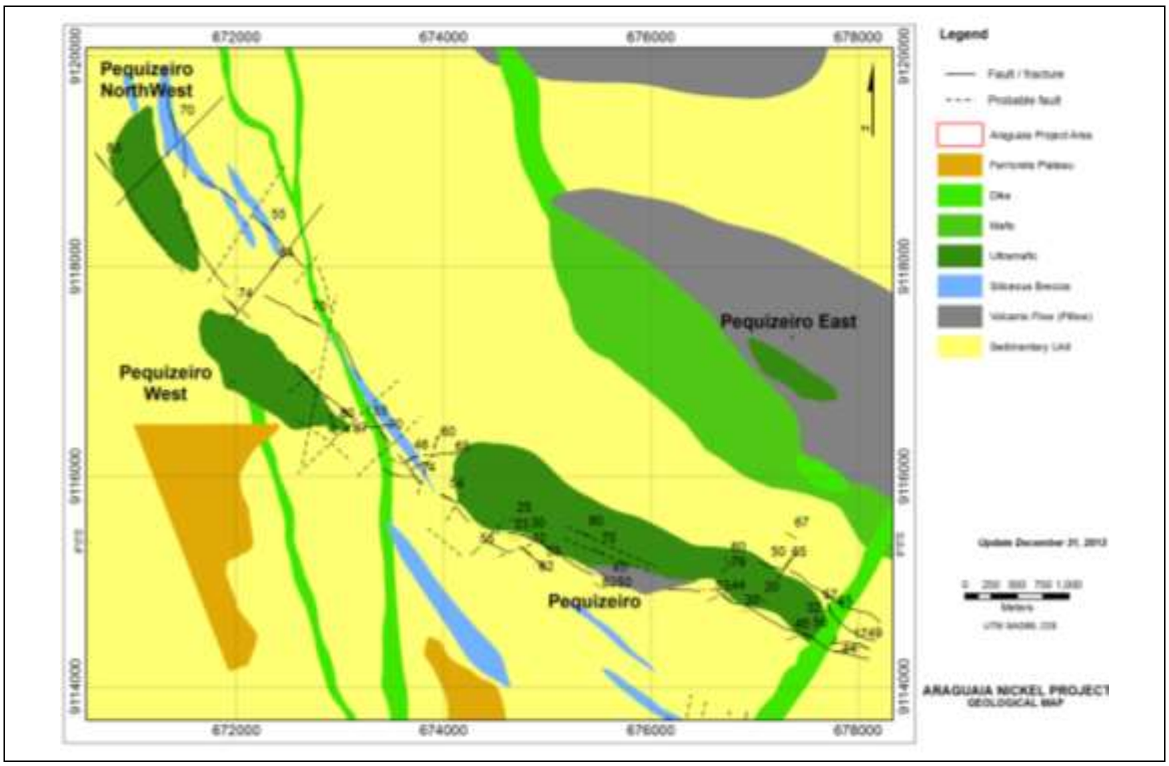
7.4.2 Pequizeiro (PQW, PQZ)

Three northwest trending mineralised bodies cover an area of approximately 3 km^2 in this sector. The deposits are enclosed by steeply dipping fault zones along northeastern and southwestern margins leading to their elongate outlines (Figure 7.12). The three main areas show the same style of mineralisation and characteristics and are therefore interpreted as one body of mineralisation that has subsequently been partitioned as a result of the major northwest trending fault system and later stage cross-faults trending northeast.

Sedimentary rocks show intense folding and silicification at the ultramafic contact where massive silica is absent (Figure 7.13).

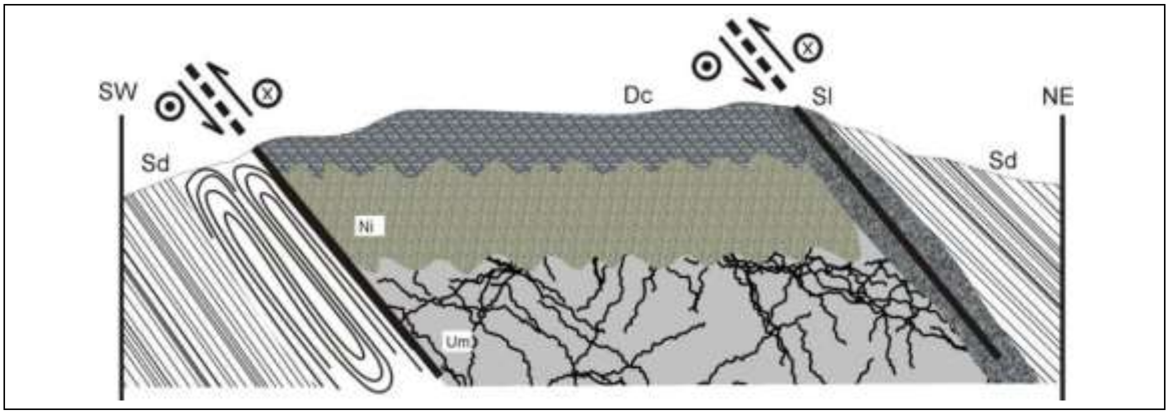
A large prominent hill between Pequizeiro and Pequizeiro West demarks the intersection of three fault zones. The hill is composed of massive silica with pervasive iron oxide within the fault zones.

Figure 7.12 Bedrock structural geology map of Pequizeiro Sector



Source: HZM 2013

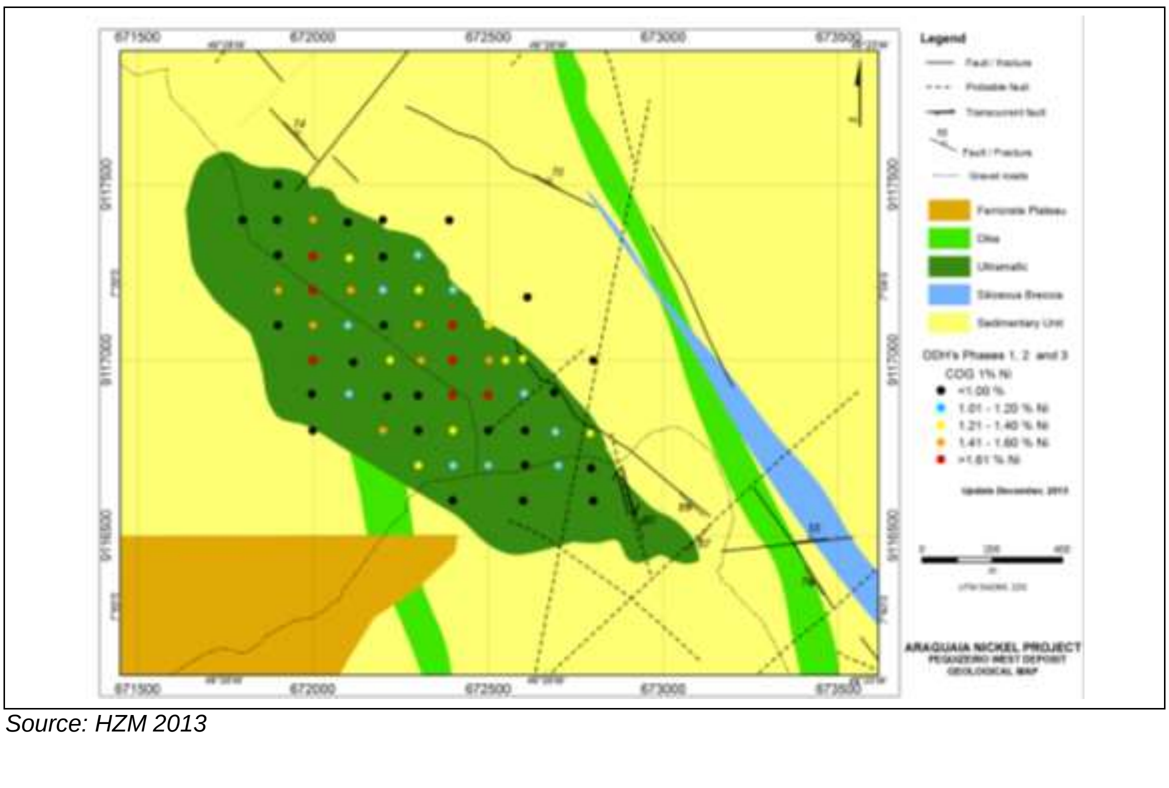
Figure 7.13 Silica and folded sediments along the margins of Pequizeiro



Note: Sd: Sedimentary rock; Sl: Silica; Um: Ultramafic rock; Dc: Duricrust; Ni: Nickeliferous zone (not to scale)
Source: Audet, M A, et al 2012

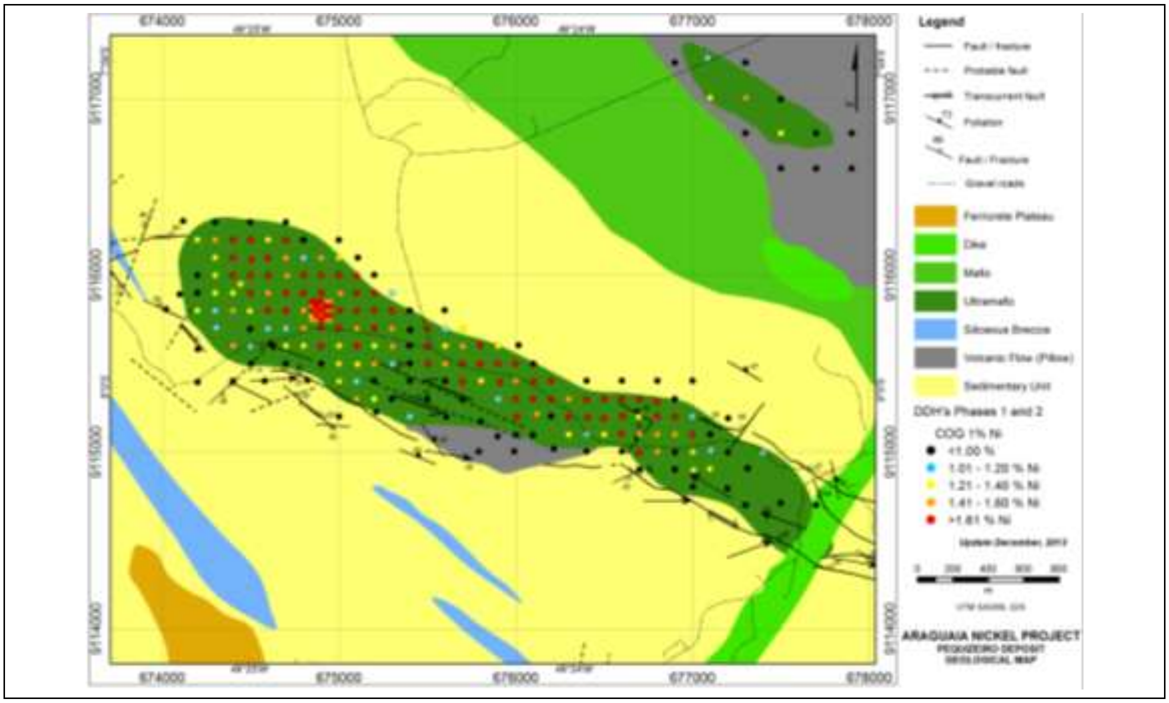
HZM provided Snowden with an updated surface geology map for PQW, following the completion of the 2012-13 drilling programme (Figure 7.14). This map and drillhole results were used by Snowden to guide the construction of 3D wireframe interpretations of the limonite, transition and saprolite horizon contacts. Snowden elected to use the existing horizon wireframes for PQZ, subject to review (Figure 7.15).

Figure 7.14 Updated bedrock PQW geology map after 2012-13 drill programme



Source: HZM 2013

Figure 7.15 Updated bedrock PQZ geology map



Source: HZM 2013

7.4.3 South (Baião or BAI)

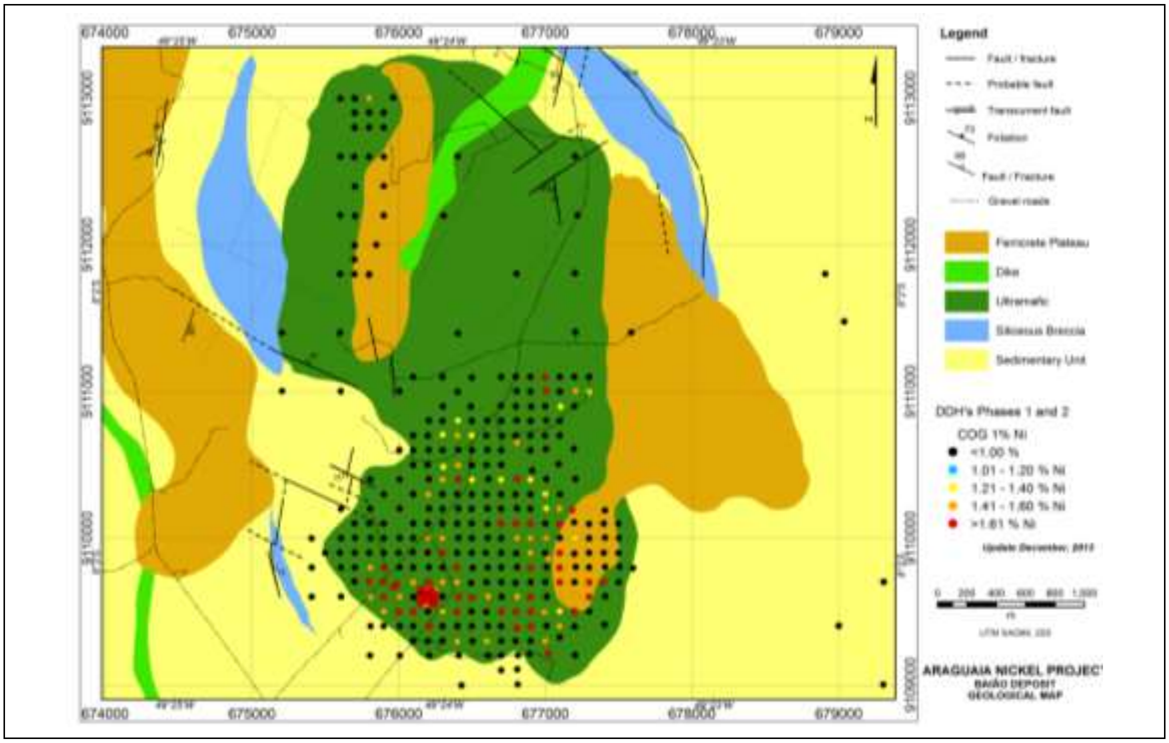
The South sector consists of three separate ultramafic bodies, the largest being Baião and covers an area of around 8 km^2 (Figure 7.16). No structures were identified from field mapping in the sector due to the flat topography, lack of exposure in the southern part and dense forest cover in the east.

A steeply dipping, silica filled fault zone is located along a north-northwest to south-southeast trending ridge in the northeastern part of Baião and reaches up to a width of 250 m. The direction of the fault zone changes to west-northwest to east-southeast further north of the ridge. A 200 m wide fresh ultramafic outcrop is located within the rupture zone that is constrained between two steep northeast-southwest trending cross-cutting faults (Figure 7.17).

The western part of Baião is limited by a zone of massive silica and silicified sedimentary rock, almost 1 km wide. Steeply dipping structures trend north-northeast to south-southwest and northeast-southwest trending. These major trends are dislocated by cross-cutting faults. Contrary to the other cross faults in Pequizeiro and northeast of Baião, these are northwest-southeast trending and filled by silica instead of iron-oxide.

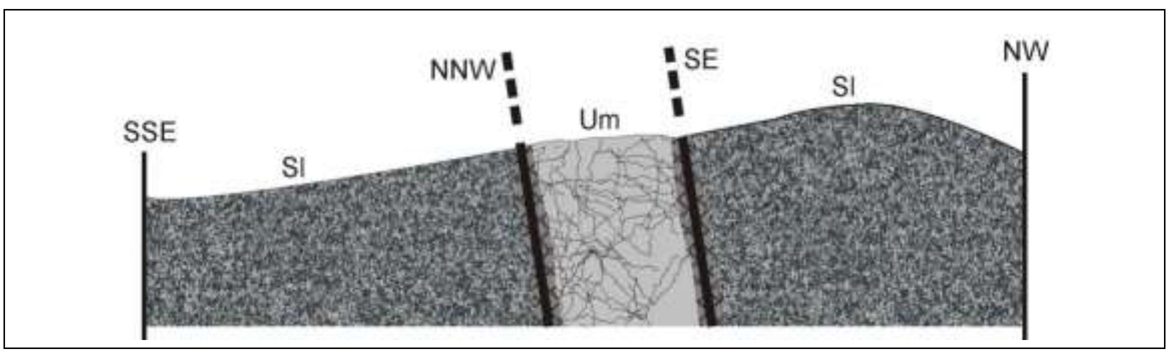
Snowden elected to use the existing horizon wireframes for Baião, subject to review.

Figure 7.16 Updated bedrock BAI geology map



Source: HZM 2013

Figure 7.17 Ultramafic unit within the rupture zone and silica ridge

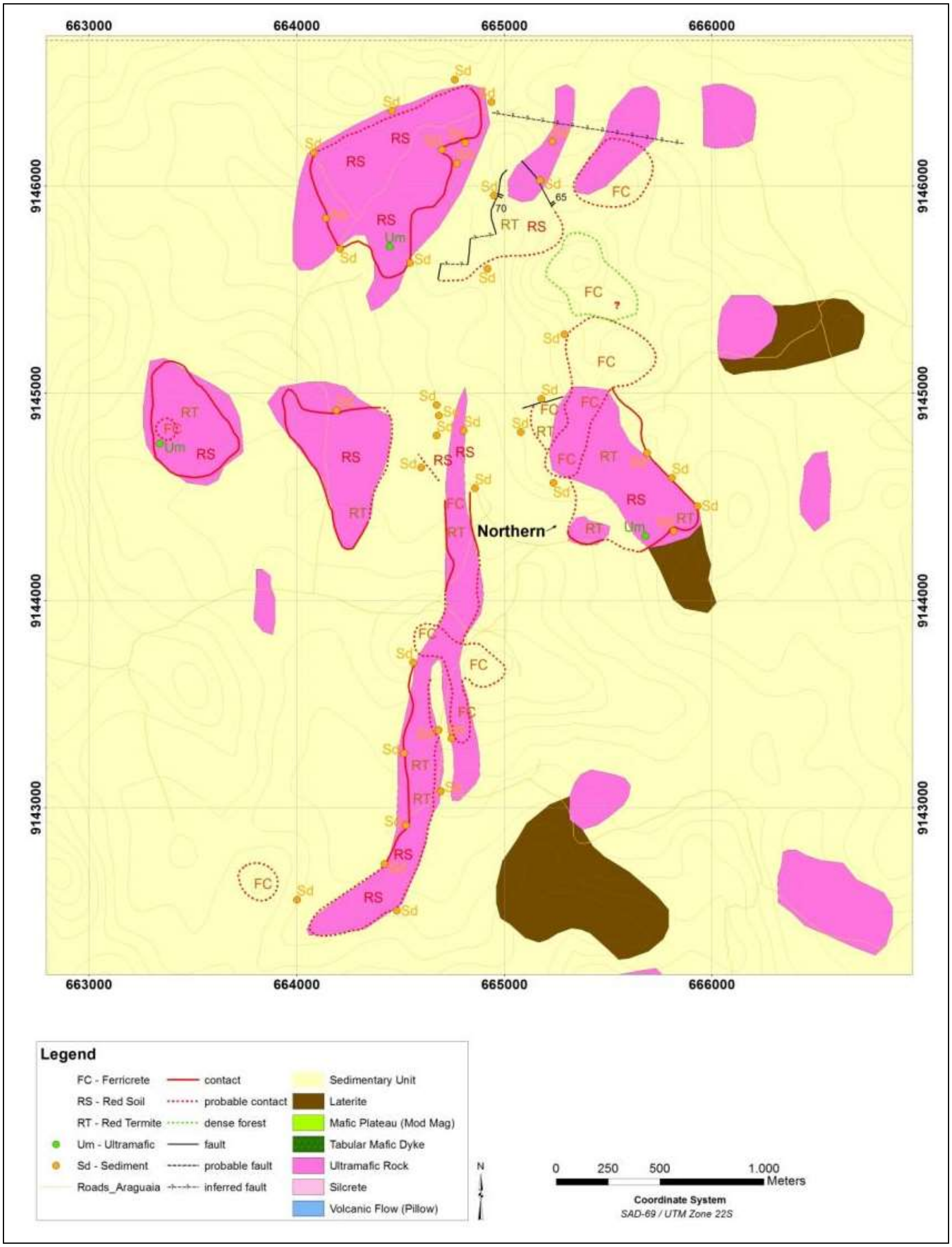


Note: SI: Silica; Um: Ultramafic (not to scale)
Source: Audet, M A, et al 2012

7.4.4 Lontra sector

There is limited outcrop in this sector and geology is largely interpreted from soil and termite mound colours (Figure 7.18). Sedimentary rocks and infrequent ultramafic / mafic rocks cover an area of 10 km^2. Hills in the eastern part consist of partly silicified sedimentary rocks, with smaller hills interpreted to consist of ultramafic and sedimentary rocks underneath the ferricrete cap. A few structures were found to host iron oxide enrichment and breccia instead of the massive silicification seen at Pequizeiro and Baião.

Figure 7.18 Geology map of North Lontra sector



Source: Audet, M A, et al 2012

8 Deposit types

The target mineralisation at HZM's Project are characteristic of typical nickel laterite deposits formed in a seasonally wet tropical climate, on weathered and partially serpentinised ultramafic rocks. Features of nickel laterites include:

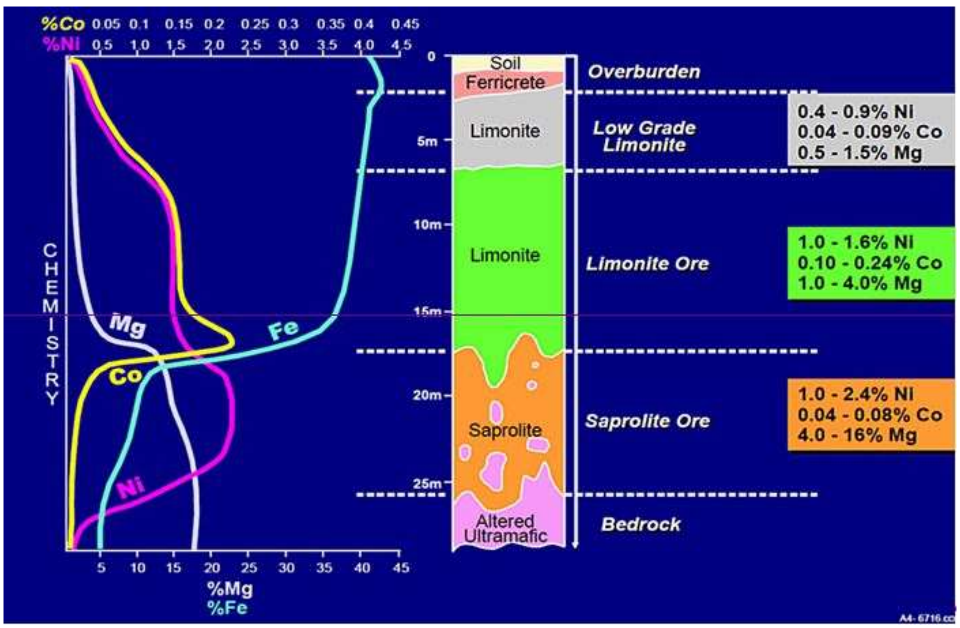
- The nickel is derived from altered olivine, pyroxene and serpentine that constitute the bulk of tectonically emplaced ultramafic oceanic crust and upper mantle rocks.
- Lateritisation of serpentinised peridotite bodies occurred during the Tertiary period and the residual products have been preserved as laterite profiles over plateaus/amphitheatres, elevated terraces and ridges/spurs.
- The process of formation starts with hydration, oxidation, and hydrolysis, within the zone of oxidation, of the minerals comprising the ultramafic protore.
- The warm/hot climate and the circulation of meteoric water (the pH being neutral to acid and the Eh being neutral to oxidant) are essential to this process. Silicates are in part dissolved, and the soluble substances are carried out of the system.
- This process results in the concentration of nickel in the regolith in hydrated silicate minerals and hydrated iron oxides; nickel and cobalt also concentrate in manganese oxides. The regolith hosting nickel laterite deposits is typically 10 – 50 m thick, but can exceed 100m.
- Concentration of the nickel by leaching from the limonite zone and enrichment in the underlying saprolite zones is also common. Leaching of magnesium +/- silicon causes nickel and iron to become relatively concentrated in the limonite zone. Nickel is released by re-crystallization and dehydration of iron oxy-hydrides and is slowly leached downwards through the profile, both vertically and laterally, re-precipitating at the base with silicon and magnesium to form an absolute concentration within the saprolite (Figure 8.1).
- The degree of the nickel concentration and the detailed type of regolith profile developed is determined by several factors including climate, geomorphology, drainage, lithology composition, and structures in the parent rock, acting over time.
- A typical laterite profile contains three distinct horizons (limonite, transition and saprolite). A schematic laterite profile for Baião and Pequizeiro is shown in Figure 8.2.

Exploration criteria is summarised from Brand et al (1996) as follows:

- Geological massifs with olivine-rich lithologies and their metamorphic derivatives, large enough to host nickel laterite deposits that will support low-cost, high-tonnage, open-cut mining operations, must initially be identified.
- Airborne magnetic surveys, regional mapping and known occurrences of lateritic nickel are useful to identify likely targets.
- Later, detailed geological and geophysical surveys may be needed to delineate olivine-rich lithologies and faulting that may represent sites for shallow, high-grade Mn-Co-Ni and garnierite mineralisation.
- Regolith landform mapping and reconnaissance drilling can be used to determine the nature and distribution of the regolith (i.e. whether in-situ, concealed or stripped) and those zones that host nickel enrichments.

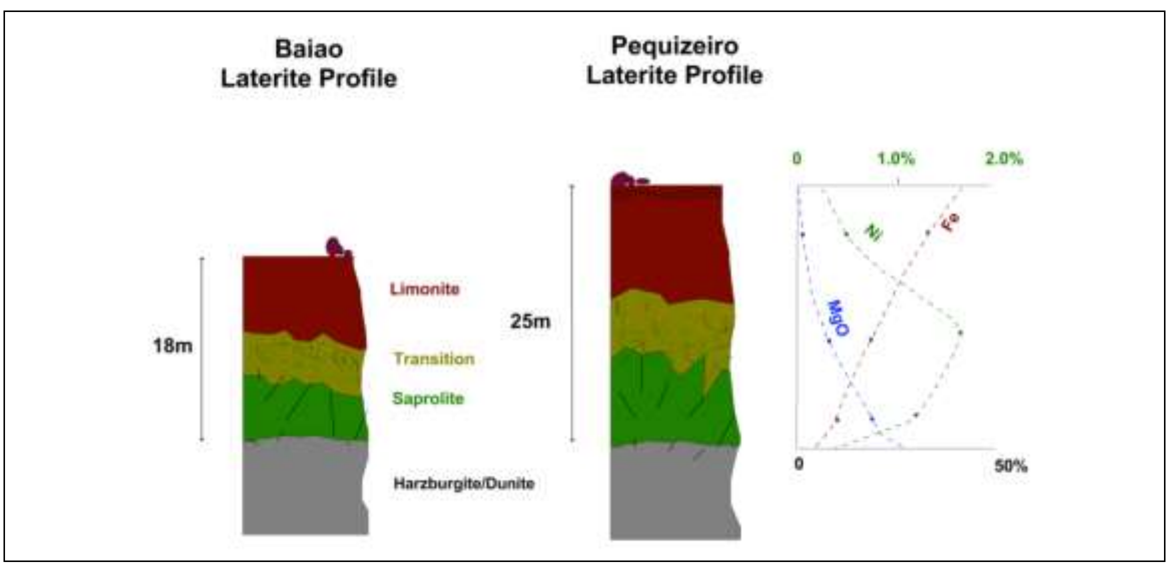
- Regional drilling and possibly soil sampling of in-situ regolith can be used to identify nickel halos (>0.5% Ni), and target the most prospective parts of a weathered ultramafic sequence.
- Follow-up drilling to delineate Ni-enriched zones will, in association with geochemistry and mineralogy, provide valuable information on the geological and metallurgical characteristics of any nickel laterite. For metallurgical purposes, it is useful to maintain a consistent element suite when analysing drill sample (Ni, Co, Mn, Cr, Mg, Fe, Si, Al) and ignition loss.

Figure 8.1 Chemical trends in schematic nickel laterite profile



Source: MALA Ground Penetrating Radar

Figure 8.2 Schematic laterite profile



Source: Audet, M A, et al 2012

9 Exploration

Drilling programmes are the main form of exploration conducted on the Project by HZM, and these are summarised in section 10. This section 9 presents relevant exploration other than drilling conducted by HZM. Exploration and drilling conducted by prior owners and operators is summarised in section 6 and details of their programmes can be found in Audet, M A, et al (2012); Barry, J.P. (2006).

9.1 Lontra area

In 2006 HZM commenced exploration on the Lontra area which is in the northwest of the current Project area. Non-drilling work included stream sediment, mapping, soil and rock sampling. A total of 2,024 stream sediment, soil and rock samples were taken. Garmin handheld GPS devices were used for field mapping control.

Surface exploration and mapping

Exploration was initiated by HZM in late 2006 with a regional low threshold, multi-element, fine fraction stream sediment survey. This led to the definition of seven anomalous zones of which three were considered priority nickel targets. Initial field reconnaissance indicated the presence of previously unmapped ultramafic lithologies and produced a rock sample, from a laterite gravel pit being used to obtain road base, with visible garnierite indicating the potential for lateritic nickel. Ground magnetometry surveys assisted the geological mapping.

In 2007, after formalising the Joint Venture on the Lontra Project, the stream sediment targets were followed up by regional (400 m x 80 m grid) multi-element soil sampling programmes.

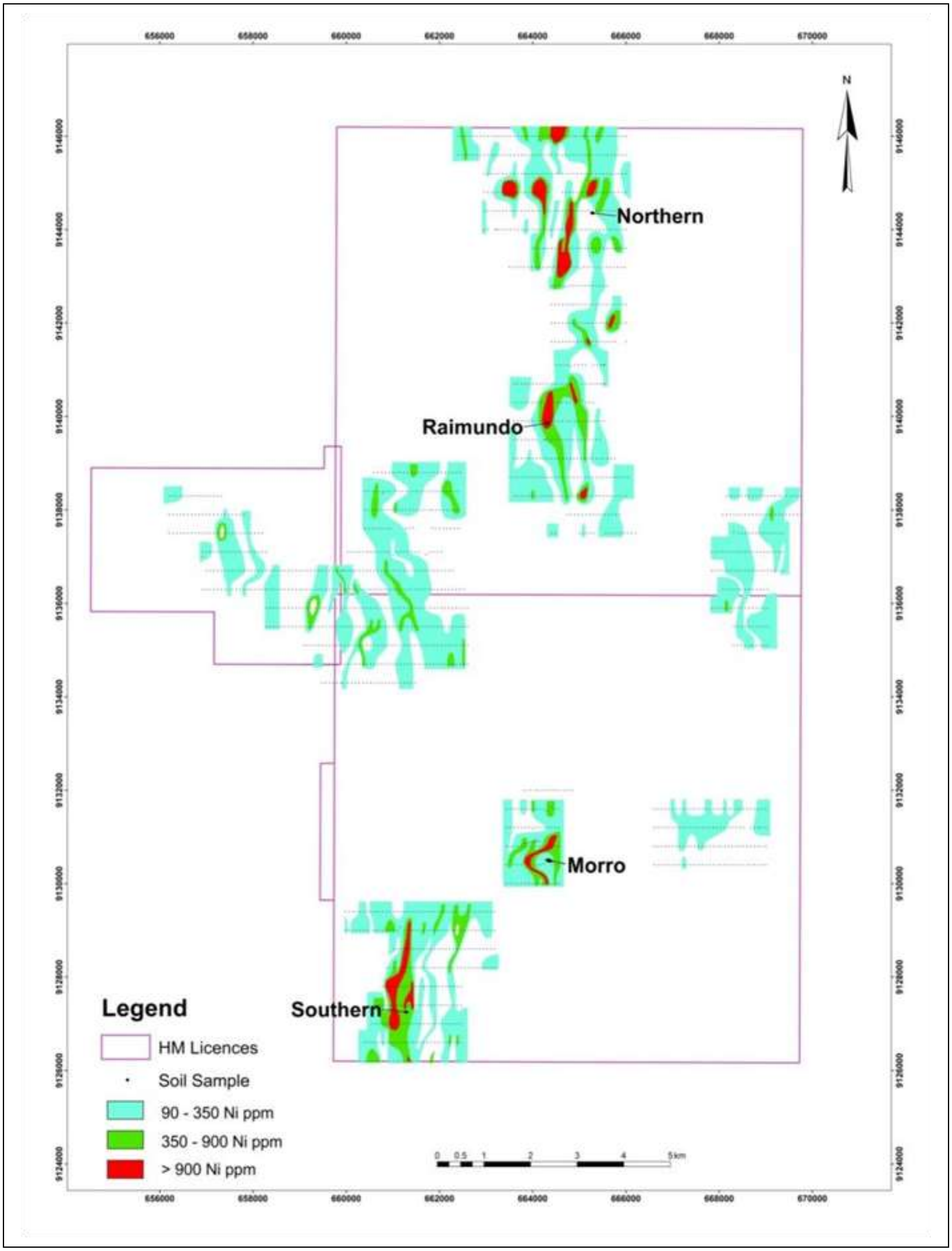
HZM soil geochemical survey grids for the Lontra project area are shown in Figure 9.1.

Through this work three principal areas of ophiolite emplacement with associated laterite development have been established, namely: Northern target; Raimundo target; and Southern and Morro target.

Targets are shown in Figure 9.1 and brief descriptions of the three main targets discovered and developed by HZM are given below:

- **Northern Anomaly**: The northern zone is a 3 km by 1.5 km area containing four anomalies, of which the main target is a 1,600 m by 250 m soil geochemical anomaly. The soil anomaly is over undulating terrain with dark red soils and termite mounds and is truncated to the northeast by wide flat residual lateritic plateaus.
- **Raimundo Anomaly**: 2 km to the south of the Northern anomaly the Raimundo anomaly has a core zone of 1,600 m by 1,000 m which became the focus of diamond drilling.
- **Southern and Morro Anomaly**: This zone gave some of the best results in a shallow auger programme despite the fact that many of the holes had to be abandoned before reaching the target depth due to the presence of silcrete or saprolite.

Figure 9.1 HZM Lontra licences soil geochemistry



Source: Audet, M A, et al 2012

10 Drilling

HZM has conducted several programmes of auger drilling and diamond core drilling at the Project. Shallow auger drilling was used prior to 2009 to define some exploration targets, while wide diameter auger drilling was used in 2011 to obtain bulk samples of mineralisation for metallurgical testwork from one deposit. Core drilling has been used to obtain samples for resource delineation in phased programmes and, separately, for geotechnical assessment of a potential open pit and plant site.

10.1 Auger drilling

10.1.1 Shallow auger drilling

In late 2007, a 124 hole shallow auger drilling programme was initiated by HZM at the Lontra area to evaluate the principal soil anomalies at Raimundo, Northern Zone and Southern Zone. Exploration success continued in 2007 with a number of mineralised nickel intervals being intersected in the auger drilling. However, the rising water table associated with the on-set of the rainy season and the limited ability of the auger to penetrate to the saprock zone meant that many holes had to be abandoned above or within the mineralised interval.

Figure 10.1 shows HZM auger drillhole coverage for the Lontra area. Holes were initially spaced on a 400 m x 80 m grid and few holes reached maximum depths beyond 12 m. Metres drilled totalled 921 m with an average depth of 7.4 m.

Data from these drillholes have not been used in mineral resource estimation.

10.1.2 Wide diameter auger drilling

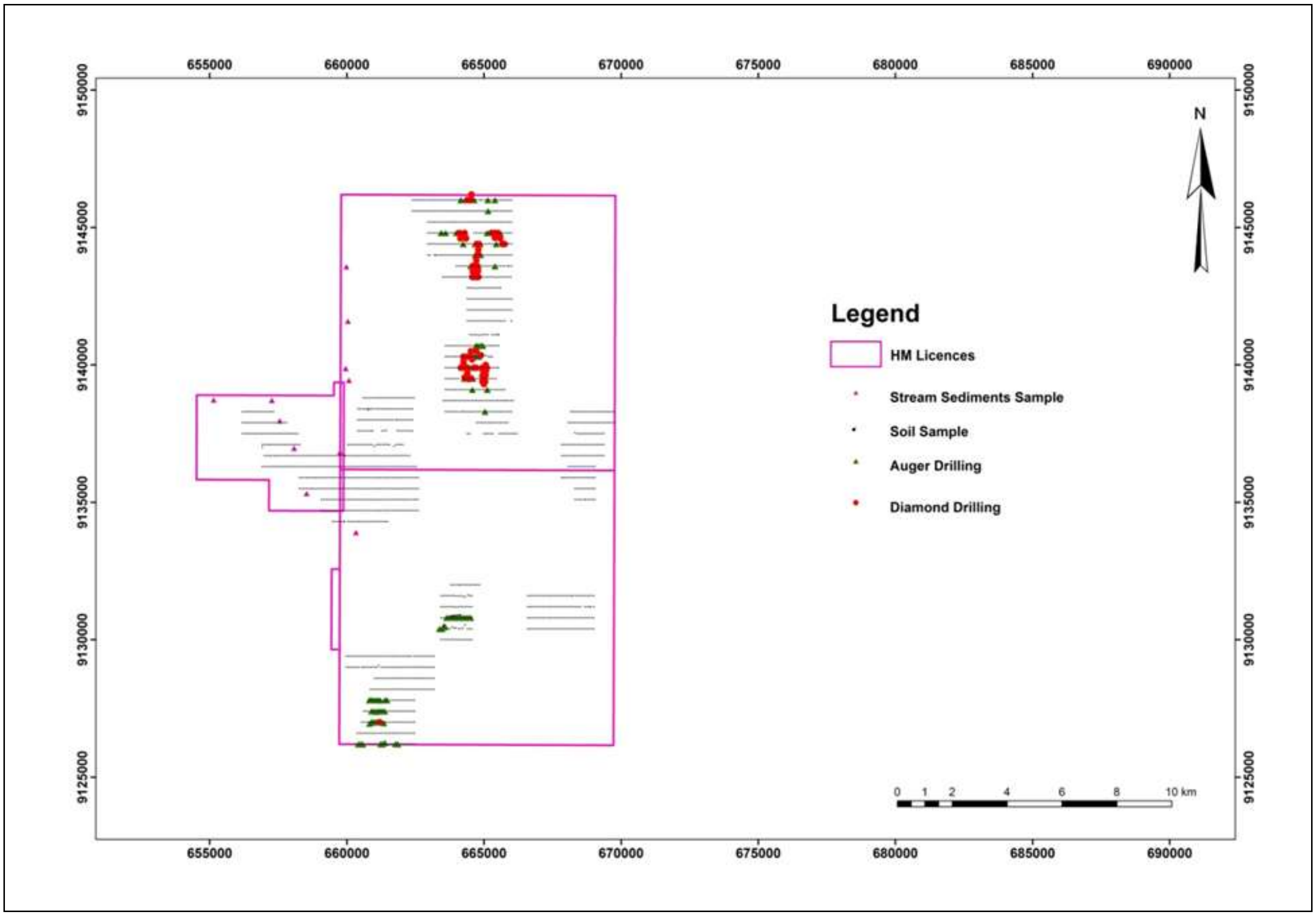
Audet et al (2012) reported that wide diameter auger drilling was used to collect approximately 130 dry tonnes of bulk sample from the Pequizeiro deposit for metallurgical testwork.

Drilling was by way of a truck mounted auger drill capable of a maximum depth of 20 m with a variety of bits including a wide diameter 1 m bit to be used for the bulk sampling, in 1 m intervals. At each location the auger holes were drilled in close proximity (about 2 m) of an earlier core drillhole to ensure that the target geology/geochemistry was collected. Positioning of the auger hole was supervised by the Project Manager and Project Geologists and organised by the Operations Manager/Technicians.

The auger drill was incapable of passing through “blocky ferricrete” containing fist sized and greater cemented blocks. An initial reconnaissance by the geology/technical team determined if these blocks were present, and an excavator removed the ferricrete, which is only found to a maximum depth of about 4 m.

The author verified the remainder of the samples are stored in 200-litre sealed plastic drums in a warehouse in Conceição do Araguaia.

Figure 10.1 HZM Lontra drill location map



Source: Audet, M A, et al (2012)

10.2 Diamond core drilling

10.2.1 Phase 1

In 2008 HZM initiated the first of three phases of diamond drilling. In total 63 diamond drillholes were completed totalling 1,299.5 m to test the Northern (31 holes) and Raimundo Zone (31 holes) target anomalies. One exploratory hole was completed on the Southern anomaly.

Within the programme vertical holes were drilled to 15-25 m in depth, ensuring that the saprock-fresh rock interface was intersected. Drillhole spacing was as follows: on 400 m spaced lines with 80 m hole centres (for geological sections and interpretation); on 200 m x 200 m centres (for resource potential identification); and on 100 m x 100 m centres (in the Raimundo high grade zone for definition of grade variation).

The diamond drilling programme was carried out with the objective of demonstrating the existence of lateritic nickel mineralisation over a significant area.

The first phase holes were drilled by drill contractor, Pacheco e Filhos Ltda of Rio Grande do Sul, using a "Sullivan" diamond drill rig with conventional drilling techniques. The second phase was drilled by Mariana Drilling Inc. of Goiania, Goias, using a "BBS-10" drill rig. The holes were drilled with HWT rods resulting in HQ core. Core recoveries were closely monitored, with less than 90% recovery being questioned and less than 80% requiring the hole to be re-drilled.

Holes were drilled through the lateritic profile to fresh rock where, in general, the hole was stopped after 3-5 m of highly competent massive fresh rock in the first phase and at the contact in the second phase. Holes were typically between 15-25 m long, but did reach over 30 m in depth.

10.2.2 Phase 2

HZM re-commenced exploration drilling on the Araguaia Nickel Project (Combined Teck Araguaia and HZM Lontra Licences) in October 2010. The programmes were designed to infill the previous core drilling completed by Teck. As well as establishing various field camps near the target sites, to minimise travel for drill crews and field staff, HZM also established an exploration office in Conceição do Araguaia from September 2010 to co-ordinate all exploration activity.

An initial drilling programme was designed to infill the 200 m x 200 m pattern on the Pequizeiro West, Pequizeiro and Baião targets. Geosonda Sondagens Geológicas Ltda. drilled HQ3 core that was designed to first reduce the drill spacing to 141 m x 141 m (5-spot drilling) and then to further reduce the drill spacing on the Pequizeiro and Baião targets to 100 m x 100 m. In addition HZM conducted drilling at Pequizeiro and Baião, at a spacing of 25 m x 25 m, to determine grade variability, geological continuity and the drill spacing required for Inferred and Indicated resource definition.

From October 2010 to December 2011 HZM completed 539 drillholes for 13,261 m as part of Phase 2. The extent of drilling to 2012 is shown in Figure 10.2.

10.2.3 Phase 3

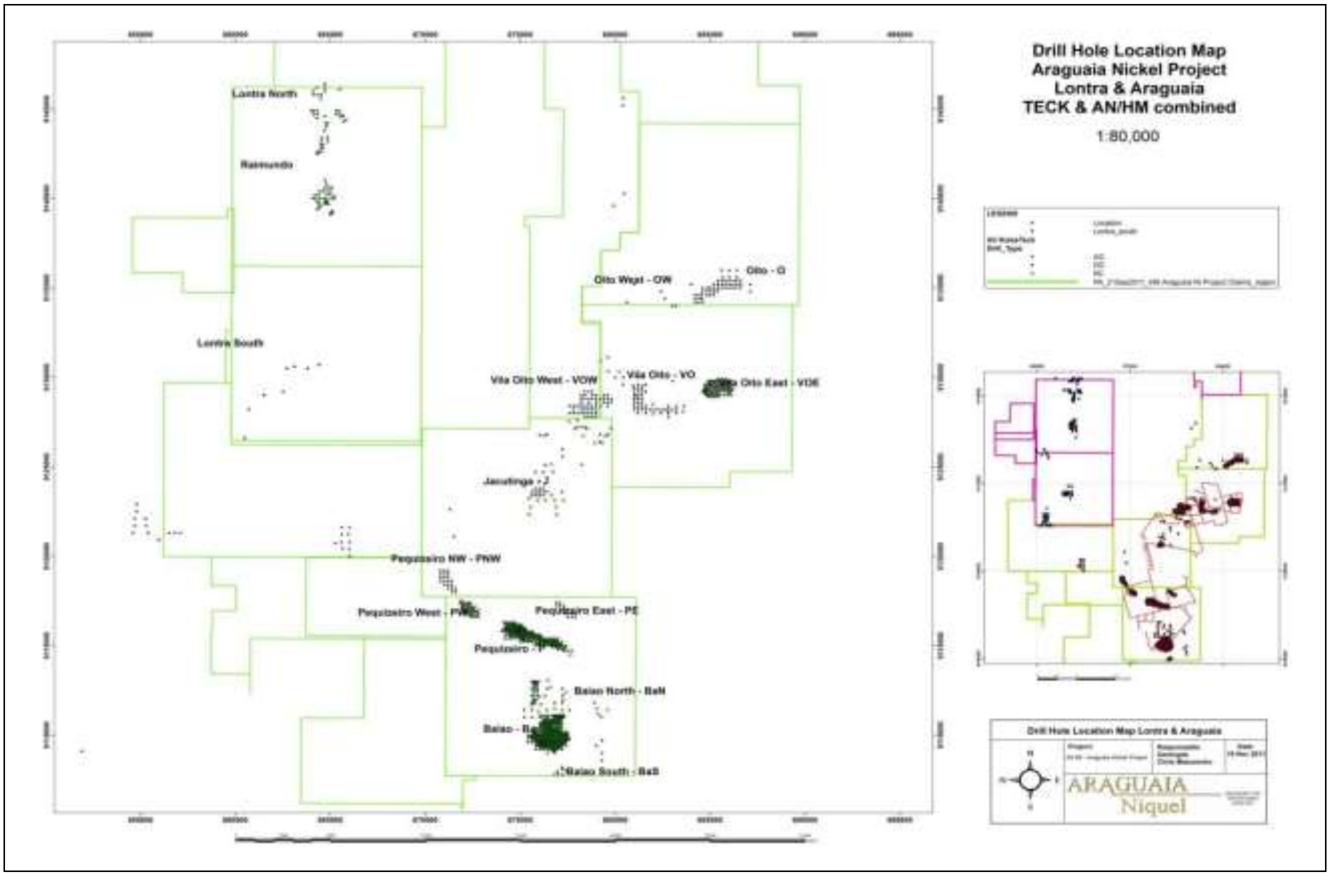
From September 2012 to April 2013 HZM conducted a Phase 3 mineral resource drilling programme. This programme was designed to complete infill drilling on 100 m x 100 m grids on the Jacutinga, Vila Oito West, Vila Oito, Vila Oito East and Pequizeiro West targets in order to convert Inferred resources to Indicated resource categories. 321 holes (9,309 m) were completed including 35 holes (1,186 m) on Jacutinga, 84 holes (1,669 m) on Vila Oito West, 133 holes (4,228 m) on Vila Oito, 44 holes (1,509 m) on Vila Oito East and 25 holes (717 m) on Pequizeiro West. HZM engaged drilling contractor Servitec Foraco to undertake core drilling with the provision of up to five rigs (Figure 10.2). Steven Heim PMP of Heim Consultoria acted as site Project Manager for HZM with technical support from F. Roger Billington P.Geo.

A summary of resource delineation drilling for seven targets that underpin pre-feasibility studies is provided in Table 10.1 (Figure 10.3). Drillhole locations and bedrock geology maps are provided in section 7.

Figure 10.2 Drillsite DDH-0684

Source: HZM 2013

Figure 10.3 Collar location map of the HZM Project to 2012



Source: Audet, M A, et al (2012)

Table 10.1 Summary of resource delineation drilling by HZM and Teck

Target	№	Metres drilled
Vila Oito West (VOW)	143	3,096.5
Vila Oito (VOI)	182	5,573.4
Vila Oito East (VOE)	127	3,901.7
Jacutinga (JAC)	59	1,720.9
Pequizeiro (PQZ)	219	6,114.9
Pequizeiro West (PQW)	60	1,626.0
Baião (BAI)	330	7,098.0
Total	1,120	29,131.4

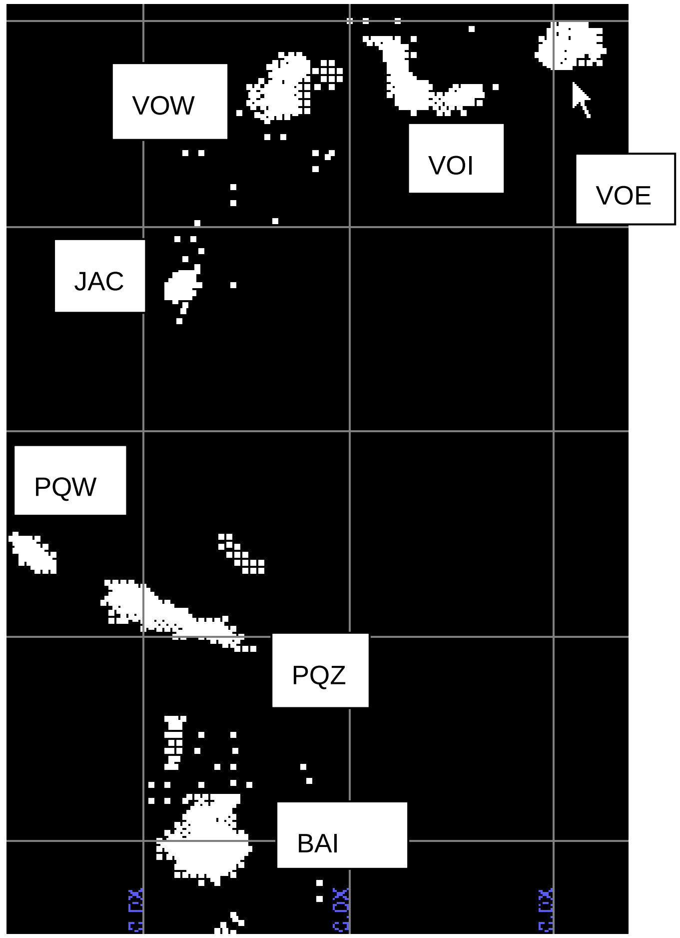
According to NI 43-101, HZM's Project meets the definition of an "advanced property" because Mineral Resource estimates and a Preliminary Economic Assessment have been reported (Audet et al., 2012a and 2012b). In line with NI 43-101 F1 instructions for advanced properties, drilling results are therefore not provided in this Technical Report. The reader is referred to Audet et al., 2012b for Phase 2 drilling results.

HZM has provided drill results by way of four news releases for the Phase 3 infill drilling programme (Table 10.2). In these news releases mineralised intervals are calculated by compositing of the nickel grades in individual drillholes across geological boundaries using a nickel cut-off of 1% with a minimum intercept length of 2.0 metres and a maximum length of internal waste of 2 metres. All holes were vertical and as these nickel laterite deposits are essentially flat-lying, all widths reported are essentially true widths.

Table 10.2 Phase 3 resource delineation drilling news releases

Date	Title
8 January 2013	Positive results from infill drilling programme & successful metallurgical testing
14 March 2013	New high grade nickel results from infill drilling programme
30 April 2013	New high grade nickel results from infill drilling programme
10 September 2013	Final drill results from infill drilling programme

Figure 10.4 Schematic plan-view of drillhole locations for 2013 PFS resource estimates



10.2.4 Geotechnical

As part of a geotechnical data collection programme, 12 diamond core drillholes designed by Snowden were drilled in 2013 in six potential open pit mining areas, totalling 386 m of HQ size diamond core (Table 10.3). All the holes were drilled vertically and cored from the surface with hole depths ranging from 20 m to 40 m. No oriented holes were drilled as it was considered that rock mass structure has no influence on preliminary slope design due to shallow depths of proposed pits limited mostly to the laterite profile.

Table 10.3 Details of pit geotechnical drillholes

Target Area	Hole ID	Easting (m)	Northing (m)	Elevation (m RL)	Dip	Depth (m)	No of samples	
							PSD & AL	UCS
Pequiziero	DDGT_001	674898	9115893	288	-90	45.2	6	1
	DDGT_002	674748	9115487	278	-90	30.2	3	0
	DDGT_003	676880	9115100	285	-90	35.1	5	3
	DDGT_004	672398	9117000	287	-90	25.2	4	0
Baião	DDGT_005	676200	9109600	270	-90	40.0	5	2
	DDGT_006	676200	9110701	270	-90	20.0	2	1
	DDGT_007	676800	9109695	265	-90	25.0	4	1
	DDGT_008	677513	9110090	253	-90	30.0	5	1
Jacutinga	DDGT_009	676000	9123600	268	-90	35.0	4	0
Vila Oito	DDGT_010	681000	9129196	271	-90	40.0	5	1
Vila Oito East	DDGT_011	685105	9129800	233	-90	30.0	5	0
Vila Oito West	DDGT_012	678600	9128600	286	-90	30.0	4	1

Note: PSD (Particle size distribution); AL (Atterberg Limit); UCS (Unified Soil Classification)

Geotechnical data collected included core interval data; lithology, degree of weathering, strength, rock quality designation, fracture frequency, core recovery and discontinuity data. These data have been entered into a geotechnical database for analysis and domain definition for the development of geotechnical models for the potential pit areas.

The selected samples were tested at the Engesolo Engenharia Ltda laboratory in Belo Horizonte, Brazil in July 2013.

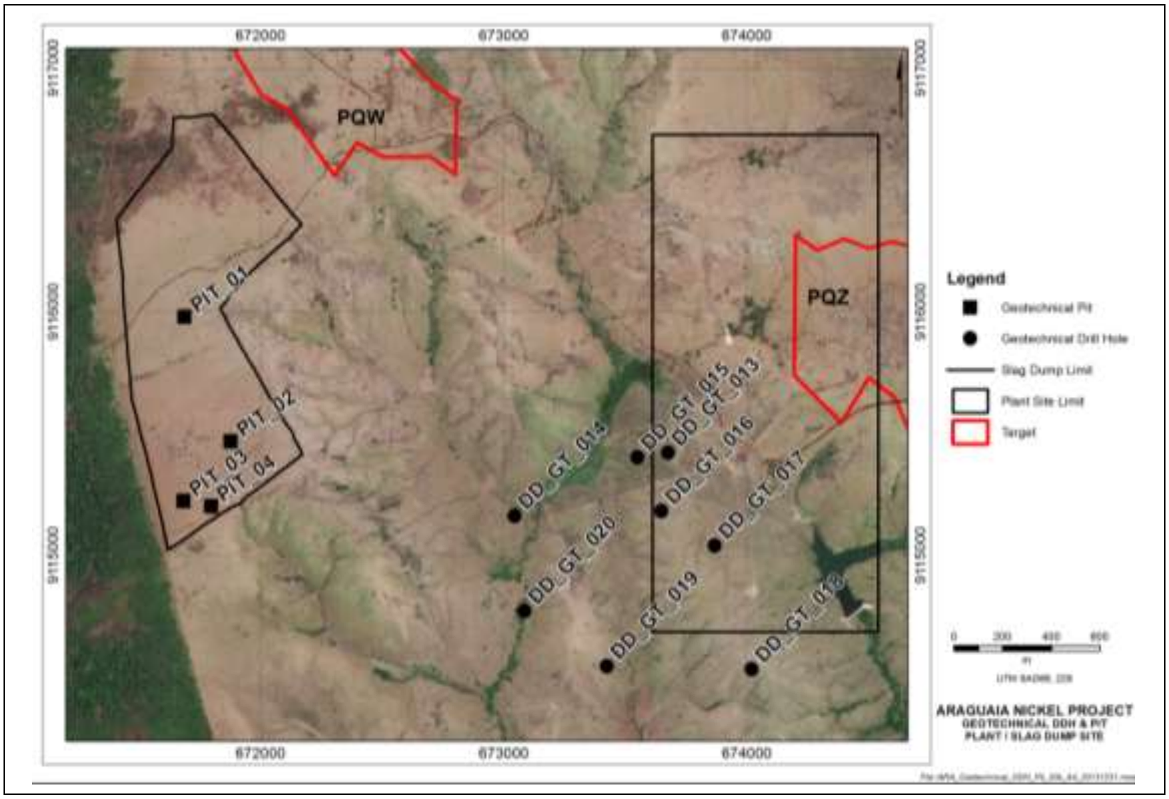
Eight diamond drillholes and several pits were also completed at potential plant and slag dump sites (Table 10.4and Figure 10.5). The drill core was assessed in a similar way to the other geotechnical drillholes at Engesolo Engenharia Ltda laboratory in August 2013.

Table 10.4 Details of plant site geotechnical drillholes

Target Area	Hole ID	Easting (m)	Northing (m)	Elevation (m RL)	Dip	Depth\ (m)	No of samples	
							PSD & AL	UCS
Plant site B	DDGT_013	673676	9115420	233	-90	53.5	3	0
	DDGT_014	673045	9115160	218	-90	19.9	3	0
	DDGT_015	673550	9115401	221	-90	22.04	5	0
	DDGT_016	673649	9115180	244	-90	25.7	3	0
	DDGT_017	673868	9115038	248	-90	24.2	4	0
	DDGT_018	674020	9114530	286	-90	22.63	4	1
	DDGT_019	673424	9114543	232	-90	24.15	2	1
	DDGT_020	673086	9114770	240	-90	18.2	3	1

Note: PSD (Particle size distribution); AL (Atterberg Limit); UCS (Unified Soil Classification)

Figure 10.5 Location of geotechnical drillholes and pits at potential plant and slag dump sites



10.3 Procedures

Procedures relevant to diamond core drilling for resource delineation are provided below and are summarised from HZM's Standard Operating Procedures (Horizonte Minerals 2012) and verified by Snowden.

- Drillhole programmes are approved by HZM's Project Manager in consultation with the Technical Advisor. Qualifications and experience for both roles are consistent with the definition of Qualified Person according to NI 43-101.
- Proposed drillhole locations are presented in an internal report accompanied by maps and sections with a cost estimate, together with target depths and objective.
- The HZM Project Manager leads a team comprising Operations Manager. Project Geologists, Technicians and Core Checkers.
- Proposed co-ordinates of drillholes are passed to the Operations Manager and Technicians for sighting in the field. The survey team instructed by the Operations Manager locates each position in the field using a total station system and identify each position with a survey tag. The total station used is a Sokkia Stratus integrated GPS L1 system with an accuracy of 5 mm horizontal and 10 mm vertical.
- With a handheld GPS, Technicians locate the correct survey tag for the hole in question using the "known" drillhole co-ordinates supplied by the Project Manager.
- Where required, access tracks and drill pads are cleared in line with licence conditions and to ensure a safe working area. Earthworks and tree cutting is kept to the minimum required for a safe working area.

- Drill rig set-up, positioning and levelling is checked by HZM staff prior to drilling.
- The Operations Manager and Project Manager monitor health and safety at the drill site.
- Drill rig activity and progress is monitored by HZM and recorded in production reports and compared with the drilling contractor's production record sheets at the end of each shift. Core Checkers record recovery / drill advance, hole completion and final depth, core box details, drill platform assessment.
- HZM require that minimum recovery requirements are met by the drilling contractor and holes finish in bedrock. Current requirements are a minimum recovery in mineralised zones is 85% over a 6 m run, and 3 m of bedrock is drilled at the bottom of each hole. HQ triple tube coring systems are mandatory.
- Technicians ensure, with Core Checkers, that the contractor correctly labels each core box with the following information: Target Name, Hole Identifier, Box Number, From and To, the direction of drill run and start and finish of the core in the box must also be marked. To ensure clarity, box identification data is marked on metal plaques using metal stamps. Core blocks showing the advance and recovery for each run are securely placed in the box, all core is wrapped in plastic.
- After drilling the hole is sealed and marked with a concrete cap which includes hole identifier, date, contractor and final depth. Once drilling is completed and the hole has been capped the survey team return and re-survey the capped position using a total station system as described above.
- HZM supervise core handling in the field and ensure that core boxes are properly secured and carefully transported from the field to the core shed. Core boxes are covered by a nailed lid and carefully transported to the core shed under a tarpaulin to protect the boxes from the weather.

10.4 Qualified Person's comment on drilling procedures

It is the author's opinion that the procedures used by HZM for core drilling were thorough and provided the appropriate level and quality of information required to interpret the laterite profile and to form the basis for Mineral Resource estimates. There is no apparent drilling or recovery factor that would materially impact the accuracy and reliability of the diamond core drilling results.

11 Sample preparation, analyses, and security

11.1 Sample preparation methods and quality control measures prior to dispatch of samples

HZM's Standard Operating procedures (Horizonte 2012) describes the handling of diamond drill core in the following steps which were verified by Snowden at the HZM's core shed in Conceição do Araguaia (Figure 11.1). All procedures are undertaken by HZM technicians and supervised by project geologists who have a minimum 2 years' experience in drill core and pit sampling:

- Details of new core boxes transported from the field are recorded in the core shed log book
- Quick logging of core is undertaken to define the boundaries of the main facies
- Samples are selected for density measurement that is performed in the core shed
- Plastic covering is removed from the core
- Core is half-split or sawn according to hardness (Figure 11.2)
- Sample intervals are defined and marked by metal tags on the core box
- QAQC samples are defined
- Core is logged
- Density test samples are returned to the core box and cut
- Core is sampled and bagged
- Quality control (QC) samples (blanks, standards) are added to the batch
- Field duplicate samples are selected
- Core is photographed wet
- The sample submission sheet is prepared
- Procedures are checked prior to the packing and dispatch of samples.

11.2 Sample splitting

Sampling of core starts after a hole has been completed and all core boxes have been transferred to the core shed where bulk density samples are identified, removed, tested and returned and the core is logged in detail.

Half core samples are taken for analysis, the other half core remains in the box for reference. The nominal sample length is 1.0 m and can vary from 0.25 m minimum to a maximum of 1.50 m according to adjustments by the geologist so the intervals do not cross lithological boundaries. Relic fragments of unweathered bedrock of less than 10 cm in length within the saprolite facies are sampled together with the facies in which it occurs. If exceeding 10 cm the fragment is sampled separately. Soft material is split using a paint scraper and hard core is cut with a diamond saw by HZM personnel.

Figure 11.1 HZM core storage

Figure 11.2 Splitting soft core with paint scraper

Samples are double bagged in plastic, and the sample number is written on both bags. Numbered sample tickets are also added to the inner plastic bag; three tickets are placed in the bag with each sample and sent to the laboratory. In the laboratory, two tickets stay with the reject material produced at various stages of preparation while the third follows the pulp through to analysis. The sample number is recorded along with the hole number and sample interval.

The sample is then weighed and this information is also recorded. Project geologists are responsible for ensuring that information is correctly recorded. Sample intervals are marked in the boxes, a metal tag with the sample number and in the case of QC samples the type and 1 ticket from the sample book all located at the start of the sample in consideration.

All primary samples have a unique number and sampling of any medium is accompanied by application of sample numbers from the series of standard sampling books of the type that contain 6 tear off tickets. All QC samples are also given primary sample numbers.

The numbers assigned to samples on the sampling cards are recorded on a spread sheet, along with drillhole identifier, sample interval, weight, sample batch and volume number. For QC samples the sample type is also recorded. Core samples are transferred from core boxes into sacks and placed on the sampling table. The sampling table is divided so that 3 batches may be produced at any one time but kept separate by taped lines. Samples are placed on the table in sequential order with sacks containing QC samples also placed in the batch at the correct point.

A batch consists of 42 samples including 5 control samples: a high nickel standard, a low nickel standard, a quartz blank, plus a pulp duplicate and in alternate batches, a crush duplicate or a field duplicate. Field duplicates are ¼ core samples. Instructions are sent to SGS to prepare crush and pulp duplicates at the relevant preparation stage.

11.3 Security measures

Standards are closely monitored, each standard has a colour assigned and each individual packet of that standard is marked with the same colour. The standards are placed in sacks which are not immediately sealed but left open for the geologist to verify the contents in the final checking procedure.

Once a batch is complete a final checking procedure is conducted. A HZM Project Geologist takes the sampling spread sheet relevant for the batch to be packed; initially the number and type of QC samples are checked and verified, the presence of standards and their colours are checked and the standards are sealed in their sacks. The samples are packed in 6 large sacks (volumes) each containing 7 samples. The Project Geologist counts off the 7 samples for a volume checking that the initial and final sample numbers as well as the sequence between correspond with the data recorded in the spread sheet for that volume.

Once the Project Geologist is satisfied that the volume is correct, the samples are double packed into two large sacks with company name, batch number and volume number are written on the outside. This procedure is repeated for the 6 volumes and the geologist then signs off on the batch for dispatch to SGS Geosol Goiania via HZM personnel.

The samples are transported by daily local transport and once samples arrive, at SGS Goiania, custody passes to SGS.

Analytical results are received from SGS in digital format via email, using a pre-defined Excel file format and a hardcopy signed analytical certificate.

Reference core is stored in core boxes sequentially by hole and box number onsite in Conceição do Araguaia. Pulp and crush rejects are returned after a 90 day period at SGS, pulp rejects are stored in wooden boxes and crush rejects in large plastic boxes sequentially batch by batch also onsite.

11.4 Bulk density measurements

The amount of density tests conducted on each facies type is continually monitored to ensure that an even spread of samples is taken across all facies. Bulk density testing is completed as soon after the core arrives in the core shed from the field as possible to avoid drying out of samples and subsequent reduction in volume

One 10 – 15 cm length of core is taken every 3.0 m run for bulk density testing of consolidated material. Where possible each density sample is taken in the first 10 to 15 cm of the analytical sample interval to avoid bias and is not taken across facies boundaries.

Core is weighed wet straight from the box, then in water (after the core is coated with wax) and after drying (Figure 11.3). Sample drying is by way of oven heating at 100°C for a minimum period of 12 hours. Sample position, number and length, facies type, as well as weight information, are recorded.

Prior to any weighing exercise the electronic scales are calibrated using a variety of checks. Five 'standard samples' with known bulk density values are also tested along with the core samples in order to check both accuracy and precision of the equipment. Currently 2 nylon, 2 aluminium and 1 PVC samples are used with known density values ranging from 1.15 to 2.72 g/cm^3.

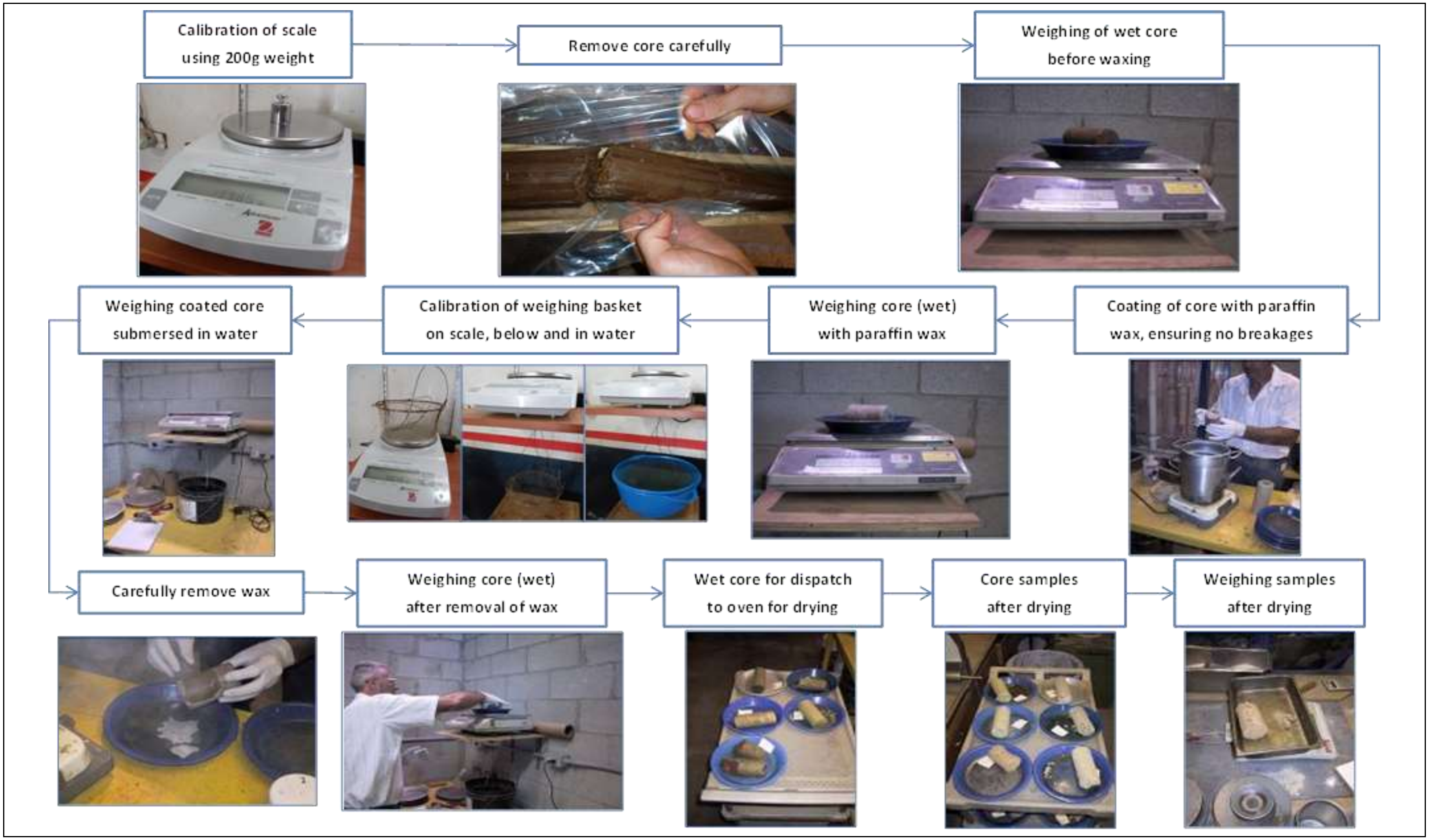
The standard samples approximate a core sample in terms of diameter and length and are weighed on the balance and in water in the same manner as core samples. One of the standard samples is chosen at random and tested with core samples to be tested on that day, the newly tested density value for the standard sample is immediately calculated in the core shed before the other core samples are placed in the oven for drying. If the newly tested density value is acceptably close to the known value for that standard sample the test is accepted and the samples are sent for drying. If the result of the standard test is unacceptable all the core samples are retested. The standard test is accepted if the newly tested density value is calculated to within ± 10%.

The temperature of water is taken and recorded; a value significantly above or below 28°C is avoided.

It should be noted that no external, independent bulk density analysis has been performed by HZM as it is considered that the results compare well to those from the Teck period of exploration and analysis.

A combination of HZM and Teck bulk density measurements, now totalling approximately 10,000 representative samples from each of the major laterite facies, has been used to derive the dry and wet bulk densities as well as moisture content. The result of this work is summarised in Table 11.1.

Figure 11.3 Density measurement procedure



Source: HZM, 2012

Table 11.1 Average dry, wet bulk densities, moisture contents and chemistry

Facies	№ Samples	SG Dry	SG Wet	H_2O%	Ni%	Co%	Fe%	MgO%	MnO%	SiO_2%	TiO_2%	Al_2O_3%	Cr_2O_3%	CaO%
Soil	726	1.73	2.1	18.61	0.14	0.04	27.78	0.15	0.54	27.85	1.53	17.95	1.39	0.01
Ferricrete	79	1.74	2.24	27.46	0.37	0.1	48.97	0.22	1.03	9.41	0.31	9.03	1.94	0.01
Limonite	1,538	1.38	1.89	30.41	0.91	0.12	35.96	2.29	0.96	21.36	0.58	10.27	2.16	0.05
Transition	1,780	1.25	1.68	29.94	1.21	0.05	17.79	11.97	0.46	45.02	0.25	4.72	1.18	0.13
Earthy Saprolite	436	1.17	1.69	36.13	1.41	0.04	14.81	17.91	0.35	42.07	0.28	4.44	1.05	0.22
Rocky Saprolite	2,026	1.43	1.82	25.67	0.99	0.03	10.44	25.74	0.25	42.64	0.2	3.78	0.72	0.15
Silicified Saprolite	124	1.57	1.95	24.32	0.51	0.03	8.94	7.8	0.27	67.06	0.2	4.07	0.56	0.38
Bedrock	959	2.26	2.4	6.86	0.28	0.01	5.91	34.55	0.12	40.9	0.06	1.5	0.42	1.27
Diorite	397	1.59	1.95	19.21	0.2	0.02	11.98	4.36	0.32	47.59	2.27	16.87	0.27	0.96
Sediment	1,762	1.63	1.98	18.35	0.05	0.01	8.7	1.82	0.35	58.68	1.29	15.96	0.15	0.29
CaO Rich	17	2.86	2.91	1.72	0.06	0.01	3.7	17.46	0.16	16.26	0.01	0.62	0.2	23.65
Dyke Al-rich	165	1.74	2.01	13.6	0.13	0.01	5.43	3.49	0.16	61.19	1.05	15.76	0.07	0.46
Quartz vein	11	2.18	2.32	6.33	0.03	0.01	1.82	0.47	0.05	94.26	0.07	1.03	0.08	0.02
Total	10,020													

11.5 Phase 3 sample preparation and analysis

11.5.1 Introduction

Half split core samples are crushed and pulverised at SGS laboratory in Goiania and the resultant pulps analysed routinely at SGS laboratory in Belo Horizonte using tetraborate fusion X-Ray Fluorescence.

11.5.2 Routine sample preparation and analysis

The following procedures are used for sample preparation of the half core samples submitted (SGS method reference: 'PREP_GY'):

- Weigh on receipt
- Dry for 12 hr at 105°C
- Weigh to determine moisture content
- Crush to 95% passing 2 mm
- Weigh to evaluate loss of material during crushing stage
- Sieve at 2 mm size to evaluate performance of crushing stage
- Split to approx. 300 g size
- Weigh 300 g sample
- Pulverise 300 g sample using carbon steel bowl to 85% passing 200 mesh
- Weigh to evaluate loss of material during pulverising stage
- Sieve at 2 mm size to evaluate performance of pulverising stage
- Split to 30 g aliquot ready for analysis
- The preparation laboratory also inserts 8 QC samples.

Once the prepared samples are received at SGS Geosol Belo Horizonte they are re-dried at 105°C (±5°C) before being riffle split and a pulp sample removed for analysis.

A glass fused disc is then prepared using lithium tetraborate to enable XRF analysis to be conducted for Co, Ni, Cu, Pb, Zn and other major oxides, as well as Loss-on-Ignition ('LOI') using thermogravimetric analysis by SGS method reference 'PHY01E'. Additional analysis for Co may be required for selected samples by four acid digestion ICP OES, but are only requested after receipt of results from XRF.

Samples are analysed by method reference 'XRF79C' which is one of the Nickel Laterite Packages offered by SGS. The suite of analysed elements and detection limits are given in Table 11.2.

Table 11.2 Suite of constituents for method XRF79C and PHY01E

Element	Detection Limit	Element	Detection Limit	Element	Detection Limit
Al_2O_3	0.1 %	Fe	0.007 %	Pb	0.01 %
Cu	0.01 %	P_2O_5	0.01 %	Cr_2O_3	0.01 %
Ni	0.008 %	Zn	0.01 %	MnO	0.01 %
TiO_2	0.01 %	Co	0.005 %	SiO_2	0.1 %
CaO	0.01 %	MgO	0.1 %	LOI	-45 %

In the event of the cobalt value in a sample exceeding 0.24% the sample is re-analysed by SGS method reference 'ICP41BB'. The suite of analysed elements and detection limits for this method has an increased upper detection limit for cobalt (8 – 10,000 ppm).

11.5.3 Check sample umpire analysis

Where primary analysis has been undertaken at the SGS Geosol in Belo Horizonte, check assays are conducted on selected samples at ACME laboratories, Canada.

Umpire samples comprising 30 g aliquots of the remaining pulp of the selected samples for analysis are analysed using an identical method to that used in the primary laboratory (i.e. tetraborate fusion/XRF). Umpire samples were submitted in batches of 40 to which were added 2 standard samples.

11.5.4 Laboratory certification

SGS is independent of HZM and the sample preparation and analytical laboratories are located as follows:

- Preparation:
  - SGS Geosol Goiana, Av Joao Leite 3209 – Q01 – L18-19.
  - Setor Santa Genoveva, Goiana, GO-CEP 74672-020.
- Analysis:
  - SGS Geosol Laboratorio Ltda, Av Mario Fonseca Viana, 120
  - Bairro Angicos, Vespasiano, MG.

SGS Geosol operates with the following Quality Management System certification:

- ISO 9001:2008; ISO 14001:2004 (ABS 32982 and ABS 39911.

ACME is independent of HZM and its laboratories are located at:

- Acme Analytical Laboratories (Vancouver) Ltd; 9050 Shaughnessy St., Vancouver BC V6P 6E5.

ACME operates with the following Quality Management System certification:

- ISO 9001:2008 for provision of assays and geochemical analyses.

11.6 Results of Quality Assurance/Quality Control (QA/QC)

Quality assurance describes the confidence in validity (i.e. data reflects what it is supposed to represent) and correct storage (i.e. data is stored accurately and may be recovered easily and without error) that is perceived for a given data set. Quality control procedures are in place by HZM to ensure that a high level of quality assurance is achieved.

11.6.1 Phase 3 results – standards, blanks, duplicates

Sampling of the resource drillhole core resulted in 9,178 samples to which 1,241 control samples were added for a total of 10,417 samples. Control samples account for 12% of the samples submitted for analysis.

Criteria for batch acceptance / rejection

Assay batches are passed or failed according to the following criteria:

- Analyses for nickel:
- Blank values must not exceed 200 ppm for Ni (2.5 x detection limit)
- Duplicate (pulp & crush) values must lie within 10% of the primary sample.
- Standard values fall within "boundary gates" as follows:
- OREAS standards – values must not exceed mean ±10%
- If a pass is not achieved then the analysis will be considered to have failed.
- A batch is accepted for entry into the database if:
- All standards, blanks, and duplicates pass, or
- Only a single Ni standard fails (exceed the boundary gates), in this case only the standard is re-analysed. If the fail is repeated the batch is deemed to have failed.
- A batch will be rejected, not entered into the database, and submitted for re-assay if:
- Both standards fail
- A blank fails
- A pulp/crush duplicate analysis fails.

If there is a discrepancy of ±10% for field duplicate results it is reported to the HZM Project Manager who decides on either reporting it further to the QP or depending on all other samples in the batch whether to proceed to include the batch in the data base.

Failure of blank/standard samples could be due to errors in the analytical machine and should be investigated at the analytical laboratory. Failure of pulp/crush duplicates could be due to problems at the preparation laboratory which should be investigated.

If a batch passes based on the Ni pass criteria the values for Fe, SiO_2 and MgO from the standard samples submitted in the batch are reviewed. If one or more of the values for these oxides exceeds the mean ± 10% the results for the batch are flagged for critical examination and the QP's determine if the batch passes or if re-assay is required.

Results

The assay results under review were reported in 254 certificates of analysis. Out of the 254 certificates 9 were rejected. The reasons for rejection, actions taken, and current status are summarised in Table 11.3. Three of the certificates were subsequently accepted where sample identifier switches were identified and corrected. The re-analysis of the samples for nine certificates returned acceptable values for the samples in question and the new certificates accepted.

Table 11.3 Quality control failures and actions

Month/ Year	Target	Laboratory Certificate Number	Accepted /Rejected	Reason not accepted – Action taken	Final Certificate Accepted
Oct/2012	JAC	GY1200889	Rejected	OREAS standards assay values incompatible with recommended values. On further analysis established that standards were switched. Accepted on switch reversal.	Accepted
Oct/2012	JAC	GY1200890	Rejected	Sample mix-up requiring repeat lab work. The re-analysis resolved the mix-up.	Accepted GY1201118
Jan/2013	VOI	GY1300003	Rejected	OREAS standards assay values incompatible with recommended values. On further analysis established that standards were switched. Accepted on switch reversal.	Accepted
Feb/2013	VOI	GY1300117	Rejected	Sample mix-up requiring repeat lab work. The re-analysis resolved the mix-up.	Accepted RE1300123
Feb/2013	VOI	GY1300118	Rejected	Sample mix-up requiring repeat lab work. The re-analysis resolved the mix-up.	Accepted RE1300124
Feb/2013	VOI	GY1300119	Rejected	Sample mix-up requiring repeat lab work. The re-analysis resolved the mix-up.	Accepted RE1300125
Mar/2013	VOE	GY1300139	Rejected	Blank assay value incompatible with recommended value. On further analysis established that Blank was switched with identifiable sample. Accepted on switch reversal.	Accepted
Apr/2013	VOW	GY1300244	Rejected	OREAS standards assay values incompatible with recommended values. On further analysis established that standards were switched. Accepted on switch reversal	Accepted GY1300195
May/2013	VOW	GY1300319	Rejected	Blank assay value incompatible with recommended value. On further analysis established that Blank was switched with identifiable sample. Accepted on switch reversal.	Accepted GY1300056

Duplicates

HZM used three types of duplicate samples in the Phase 3 drilling campaign:

- Field Duplicates

Field duplicate samples were designed to test the sample heterogeneity. They were taken at the rate of one every alternate batch dispatched i.e. 1 in 74 samples. Where a sample interval was selected for a field duplicate, the remaining ½ core from primary sample was further split in two, leaving a ¼ core material in core boxes.

- Crush Duplicates

Crush duplicates tested the variability at the crusher stage and again were taken at the rate of one every alternate batch i.e. 1 in 74 samples. Crush duplicates were not prepared onsite but were produced following instruction given to the preparation laboratory. The preparation laboratory had to split the selected sample post crushing into two equal samples viz., a primary and duplicate sample.

- Pulp Duplicates

Pulp duplicates tested the variability at the pulverisation stage and were taken at the rate of one every batch i.e. 1 in 37 samples. As for the Crush duplicates, the Pulp duplicates were not prepared onsite but by the preparation laboratory following instruction by HZM. The preparation laboratory had to split the selected sample post pulverising into two equal samples viz., a primary and duplicate sample.

A total of 497 duplicate samples were inserted in the HZM samples submitted during the Phase 3 drilling campaign for 4.8% of the total as follows; Field duplicates: 127, Crush duplicates: 124, and Pulp duplicates: 246.

With few exceptions, samples show satisfactory re-assay precision statistics for the whole range of data values with assay pairs showing less than 10% absolute difference between first and second assays.

11.7 Phase 3 results – umpire assay analysis

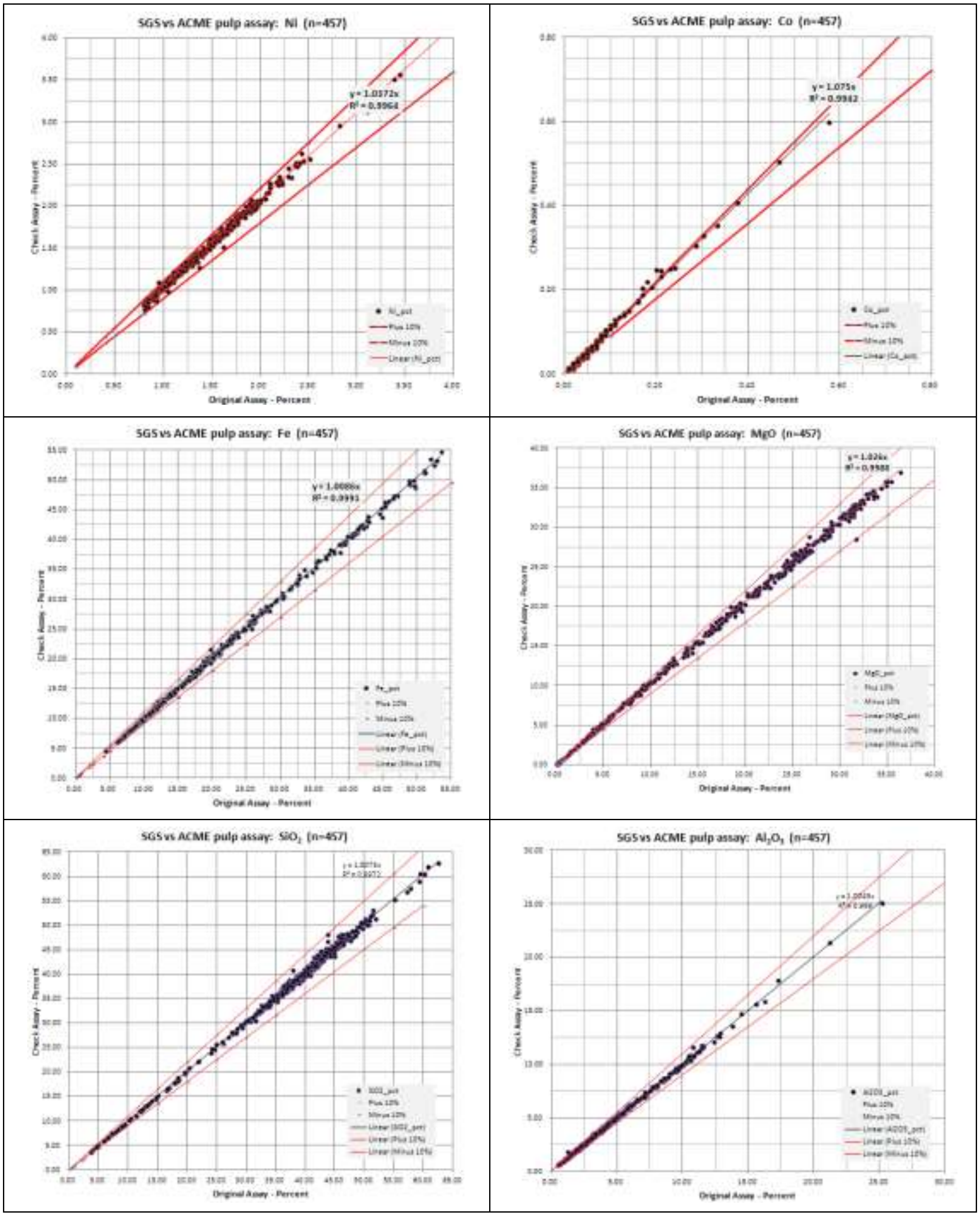
After receipt of the primary assay results 5% of the samples (from mineralised zones) were selected for umpire assay.

A minimum protocol would be for >90 % of the samples assayed at the umpire laboratory should give <10 % difference in nickel values relative to the primary laboratory.

In 2013 HZM dispatched 457 duplicate pulp samples (55 from JAC area; 37 from PQW area; 73 from VOE area; 202 from VOI area; 90 from VOW area) to ACME.

The results of the umpire analyses are presented in Figure 11.4 below and are acceptable to Snowden.

Figure 11.4 Results of umpire analyses



11.8 Author's opinion on the adequacy of sample preparation, security, and analytical procedures

The author concludes that the procedures are acceptable and that the resulting records are suitable for use in Mineral Resource estimation.

12 Data verification

12.1 Phase 3 data verification

The GEMS databases compiled and verified by Dr. Marc-Antoine Audet were provided to Snowden on 12 October 2012. Snowden, under the supervision of Andrew Ross FAusIMM, P.Geo, checked the databases and reconciled the drill data and resource block models with the information presented in Audet, M A, et al (2012).

Several field visits and reviews have been conducted by Snowden consultants. The first visit to the project site occurred from 22 to 24 November 2012 by Andrew Ross and Marcio Soares when drilling was underway on the Vila Oito area. A subsequent site visit was performed by Marcio Soares from 8 to 12 April 2013 where 27 of the Phase 3 drill collars were checked, representing 10% of the drillholes. Five drillhole cores in the logging facility were reviewed. The sample preparation and analytical facilities were also inspected by Marcio Soares: on 18 April 2013 the SGS Geosol sample preparation laboratory in Goiania was inspected; on 24 April 2013 the SGS Geosol analytical laboratory in Vespasiano was inspected. From 26 to 30 August 2013, Asoka Herath of Snowden inspected geotechnical drilling underway at the potential plant site.

HZM provided Snowden with drillhole related information on several occasions throughout the duration of the Phase 3 drill program, including: standard operating procedures; drill geological, survey and sample data files; analytical certificates; QA/QC results; and density measurements. In order to maintain consistency with the assigned facies defined by the chemical correlation matrix, Snowden requested HZM engage Dr. Marc-Antoine Audet to complete that assessment.

Data verification work of Phase 3 drilling included:

- Collar locations – independent checking of 10 % of collar locations
- Geological logging – independent checking of drill core logs
- Review of QA/QC procedures and results
- Review of facies assignment from sample chemistry.
- Review of density measurement procedures
- Review of data entry procedures and cross-check with analytical certificates
- Review of sample preparation and analytical laboratory procedures.

12.2 Qualified person's opinion on the adequacy of the data for the purposes used in the technical report

The author concludes that the data verification results provide assurance that the data is reliable and adequate for use in mineral resource estimation. Together with the findings of the QP for Phase 1 and 2 data, Snowden concludes that the Project resource database meets industry standards and is compatible with the JORC and CIM codes for public reporting.

13 Mineral processing and metallurgical testing

13.1 Introduction

The Project scenario under consideration is the development of a greenfield Fe-Ni smelter with a design production capacity of up to 15,067 tonnes per annum of nickel in Fe-Ni from laterite ore. The pyro-metallurgical plant will have one single processing line from Ore Receipts through to Granulation of the refined Fe-Ni final product and will be capable of processing 0.9 Mtpa (dry). For design purposes, the ore nickel grade adopted is 1.80% Ni. The Life-of-Mine (LOM) plan reflects one year when the grade increases to up to 1.9% Ni while the full LOM average grade is 1.66% Ni. It is important to note that over the LOM this proposed capacity was selected due to the potential availability of equipment that is already sized and constructed. As a result production is limited to the capacity of this equipment. The key project criteria for the base case are given in Table 13.1. An alternative higher capacity scenario was also evaluated.

Table 13.1 Key base case project criteria

ITEM	UNIT	VALUE
Ore throughput – year 1 to year 4	Mtpa (dry)	0.9
Ore grade – LOM average	% Ni	1.63
Ore grade – design	% Ni	1.80
Overall nickel recovery	%	93
Final metal Ni production - design	Mtpa Ni	15 067
Furnace power (per furnace)	MW	50
Ni grade in the final product metal	%	20.3
Plant configuration		One RKEF line
Refinery system		Ladle furnace
Final product		Granulated Fe-Ni

HZM has previously investigated processing by hydro-metallurgical methods and testwork for this approach is summarised in this section for completeness.

13.2 Sample selection and variability

13.2.1 Introduction

As part of the work by HZM, a testwork program to examine the Araguaia laterite material for treatment by the Rotary Kiln Electric Furnace (RKEF) process was carried out. A number of test samples were obtained which were considered by HZM to be representative for processing of this ore and the %Ni cut-offs selected were based on the Mineral Resource estimate available at the time of collection. Two sets of samples were used in the metallurgical testwork program which was carried out at both Xstrata Process Support (XPS) and FLSmidth (FLS). The material sent to XPS in 2011 was based on ¼ core samples, while the samples sent to FLS in 2012 were prepared from a large (130 dry tonne) bulk sample taken with a 1 m auger. The following section describes these test samples.

13.2.2 Samples sent to XPS

For the tests at XPS carried out in late 2011, six samples, 2 x 10 kg from each of the principal facies types (Limonite, Transition and Saprolite) were made up of selected ¼ core samples. The samples were representative of each of the facies from the Pequizeiro and Baião deposits, characterised at a 1.0% Ni cut-off. In total, 60 kg of sample material in partly dried condition was received at XPS. Each of the facies samples were homogenised at XPS.

Table 13.2 presents the assay data of these samples. It is seen that the assays of the homogenised samples match with those calculated.

Table 13.2 Assay data of samples sent to XPS for testing

Composite Sample	Targets & Samples	Facies	% in Resource	Ni %	Co %	Fe %	MgO %	SiO2 %	Al2O3 %	SiO2/MgO	Fe/Ni	Ni/Co
HM_14L_44T_42S_FLS	Target	Limonite	13.7%	1.43	0.117	34.68	4.73	20.10	9.17	4.25	24.20	12.29
	HM_14L_FLS	Limonite		1.89	0.140	31.17	4.38	30.23	5.96	6.90	16.49	13.50
	Target	Transition	43.8%	1.66	0.063	18.92	11.76	40.72	5.65	3.46	11.41	26.22
	HM_44T_FLS	Transition		1.57	0.067	20.08	11.69	37.83	5.44	3.24	12.82	23.50
	Target	Saprolite	42.5%	1.58	0.040	12.49	21.03	42.21	4.37	2.01	7.92	39.89
	HM_42S_FLS	Saprolite		1.27	0.052	13.83	21.60	38.23	5.23	1.77	10.87	24.61
	Target TOTAL	Composite	100%	1.59	0.060	18.35	14.74	38.53	5.59	2.61	11.52	26.34
	Sample TOTAL	Composite		1.52	0.074	19.84	13.77	36.70	5.46	2.66	13.03	20.60
HM_51T_49S_FLS	Target	Transition	50.7%	1.66	0.063	18.92	11.76	40.72	5.65	3.46	11.41	26.22
	HM_51T_FLS	Transition		1.59	0.070	20.76	10.63	37.77	5.53	3.55	13.09	22.65
	Target	Saprolite	49.3%	1.58	0.040	12.49	21.03	42.21	4.37	2.01	7.92	39.89
	HM_49S_FLS	Saprolite		1.68	0.047	15.40	18.27	38.73	4.71	2.12	9.16	36.02
	Target Composite	Composite	100%	1.62	0.052	15.75	16.33	41.45	5.02	2.54	9.73	31.39
	Composite Sample	Composite		1.64	0.056	17.54	15.21	38.35	5.03	2.52	10.68	29.33

13.2.3 Samples sent to FLS

Samples sent to FLS for testing during the fall of 2012 (also for testing by Feeco International and by KR Komarek) were taken from a 130 tonne bulk sample (dry weight). The material was taken with a 1 m auger in September 2011 from selected areas of the Pequizeiro deposit (Figure 13.1). The objective was to generate two blended samples (14% Limonite, 44% Transition and 42% Saprolite; and 51% Transition and 49% Saprolite) of approximately 3 tonnes each (wet) representative of the total resource estimate at a 1.2% Ni cut-off. A total of 6,099 kg of material (wet basis) was received at FLS for testing.

The resource chemistry is presented in Table 13.3, while Table 13.4 presents the assays obtained at FLS. It is seen that the chemistry of the material at FLS closely matched the target chemistry.

Table 13.3 Target laterite chemistry based on the resource (February 2012) at Pequizeiro and Baião at a 1.2% Ni cut-off

Composite Sample	Targets & Samples	Facies	% in Resource	Ni %	Co %	Fe %	MgO %	SiO2 %	Al2O3 %	SiO2/MgO	Fe/Ni	Ni/Co
HM_14L_44T_42S_FLS	Target	Limonite	13.7%	1.43	0.117	34.68	4.73	20.10	9.17	4.25	24.20	12.29
	HM_14L_FLS	Limonite		1.89	0.140	31.17	4.38	30.23	5.96	6.90	16.49	13.50
	Target	Transition	43.8%	1.66	0.063	18.92	11.76	40.72	5.65	3.46	11.41	26.22
	HM_44T_FLS	Transition		1.57	0.067	20.08	11.69	37.83	5.44	3.24	12.82	23.50
	Target	Saprolite	42.5%	1.58	0.040	12.49	21.03	42.21	4.37	2.01	7.92	39.89
	HM_42S_FLS	Saprolite		1.27	0.052	13.83	21.60	38.23	5.23	1.77	10.87	24.61
	Target TOTAL	Composite	100%	1.59	0.060	18.35	14.74	38.53	5.59	2.61	11.52	26.34
	Sample TOTAL	Composite		1.52	0.074	19.84	13.77	36.70	5.46	2.66	13.03	20.60
HM_51T_49S_FLS	Target	Transition	50.7%	1.66	0.063	18.92	11.76	40.72	5.65	3.46	11.41	26.22
	HM_51T_FLS	Transition		1.59	0.070	20.76	10.63	37.77	5.53	3.55	13.09	22.65
	Target	Saprolite	49.3%	1.58	0.040	12.49	21.03	42.21	4.37	2.01	7.92	39.89
	HM_49S_FLS	Saprolite		1.68	0.047	15.40	18.27	38.73	4.71	2.12	9.16	36.02
	Target Composite	Composite	100%	1.62	0.052	15.75	16.33	41.45	5.02	2.54	9.73	31.39
	Composite Sample	Composite		1.64	0.056	17.54	15.21	38.35	5.03	2.52	10.68	29.33

Figure 13.1 1 m diameter auger drill used to take the 130 tonne bulk sample at Pequizeiro in September, 2011

Table 13.4 Chemical analysis of the samples received at FLS

Lab No.:	120274M		Lab No.:	120276M	
Sample ID:	51/49 Mix		Sample ID:	14/44/42 Mix	
SiO_2	42.2	%	SiO_2	39.4	%
Al_2O_3	5.54	%	Al_2O_3	5.83	%
Fe_2O_3	25.2	%	Fe_2O_3	27.6	%
CaO	0.23	%	CaO	0.19	%
MgO	13.8	%	MgO	13.0	%
K_2O	0.014	%	K_2O	0.012	%
Na2O	0.109	%	Na2O	0.105	%
SO3	0.018	%	SO3	0.018	%
NiO	2.16	%	NiO	1.96	%
TiO_2	0.19	%	TiO_2	0.18	%
MnO	0.37	%	MnO	0.40	%
Loss @ 982° C	11.13	%	Loss @ 982° C	10.64	%
Total	100.96	%	Total	99.34	%

13.3 Metallurgical testwork review

During late 2011 and 2013, HZM contracted various organisations and metallurgical laboratories to conduct testwork.

The main testwork and reports produced by various companies are shown in Table 13.5 and are reviewed in this section.

Table 13.5 Testwork reports

Title/Description	Date	Company
Lab Testing of Horizonte Minerals Araguaia Nickel Laterite Deposits - Final Report	03 April 2012	Xstrata Process Support (XPS)
Slag Chemistry for the Smelting of Horizonte Laterite	29 July 2012	Kingston Process Metallurgy (KPM)
Report on Agglomeration Tests	15 January 2013	Feeco International (Feeco)
Liquidus Measurement of Fe-Ni Slag	26 April 2013	Kingston Process Metallurgy
Evaluation of Brazilian Araguaia Nickel Laterite	July 2013	FLSmidth(FLS)

The review highlighted some key issues which can impact on design and operational factors on a commercial scale processing plant. Comments from IGEO, who conducted the engineering design for the Project, based on their experiences on some of these issues are also incorporated.

13.3.1 Feeco

Testwork summary

A number of agglomeration tests were run in the pilot rotary agglomerator at Feeco on the 51% transition - 49% saprolite blend. The idea of doing these tests arose out of discussions with FLS that given the potential fine particle size of the material and the possible impact on kiln dusting, it was considered desirable to get a better understanding of the behaviour of the as-received laterite material during the rotary action in the dryer kiln. The tests on the unit at Feeco were able to simulate this physical action. The objectives of the tests were to observe the agglomeration behaviour of the as-received ore (nominally 38% moisture content) when processed through this type of unit over a range of test conditions.

Four barrels (corresponding to about 798 kg) of the 51 % transition - 49% saprolite blend of Araguaia material was shipped to Feeco from FLS prior to the work. A moisture determination made by Feeco on a sample taken at about the middle of one of the drums reported a moisture content of 38% H_2O.

The three variables that were changed in the tests were:

- the feed rate
- the drum rotational speed - each of these effectively affecting the residence time
- with and without a liner and lifters inside the drum.

The first two variables each affect the residence time of the ore in the agglomerator. The results of these preliminary tests were presented as a report and were complemented by subsequent testwork at FLS

The rotary agglomerator at Feeco (see Figure 13.2) essentially resembles a kiln and is a revolving, stainless steel, cylindrical drum sized 0.762 m diameter by 2.9 m long. The unit was equipped with a variable speed chain drive turning the supporting rollers and thus turning the drum. The unit had provision for feeding material at a known, controlled feed rate while at the discharge end there was a chute where the product emptied into plastic trays. Figure 13.3 shows the agglomerated product discharging from the agglomerator.

Tests were run with the inside of the drum both with and without a plasticised rubber lining and six small “lifters” bolted lengthwise along the inside of the unit. The rotary movement of the ore was considered to reasonably simulate the action within a dryer or kiln.

Figure 13.2 Feeco rotary agglomerator

Source: Feeco International Inc.

Figure 13.3 Agglomerated product discharge

Source: Feeco International Inc

Test results

It was visually observed that agglomerates were reasonably competent and that fines were generally contained within the agglomerates.

The drop test demonstrated good resilience of the agglomerated particles to breaking during the test. The round particles did however deform into flat disks during the tests.

Particle size distribution ranged from approximately 3 mm to 25 mm.

Agglomerated particles did tend to break up during exposure to water in a flat tray, however it was observed that the feed material tended to break up faster than the agglomerated particles.

Conclusions

From the observations made on the agglomerator product, it is evident that the rotary action of the drum was generally able to produce balled or agglomerated material from as-received wet Araguaia laterite ore over a range of feed rates and drum revolving speeds and independently of whether the liner was in place or not.

A review of the test results found that the agglomeration tests carried out gave important indications of the favourable behaviour of this ore to agglomeration. Equally importantly was the subjection of the agglomerated material to competency tests after drying.

Over the past years IGEO has done considerable work with engineering company Polysius on incorporating this concept of agglomeration in the design of the dryer and these have actually been transposed into commercial operating plant equipment. Focus has also been given to design of the internals of the kiln, not only to retain the agglomeration benefits as much as possible, but also to improve drying efficiency in the early stages of the kiln while at the same time taking care to minimise dust losses in the kiln off-gas.

It is also important to highlight that, apart from the dryer and kiln design, agglomeration can have an important influence on the effectiveness of the furnace operations. Mainly this is due to the benefits arising from improved charge permeability and also to the type of charge profile that can be attained around the electrodes. It was recommended to conduct pilot scale agglomeration tests to confirm the behaviour of as-received ore during any future pilot plant calcining and smelting campaign.

At this stage of the project it is planned to develop a flowsheet based on the IGEO experience from operations and testwork and with a focus on minimising capital costs.

For treatment of kiln off-gas dust from the precipitator at this stage it is planned to introduce the simplest flowsheet, but one which is known from experience does in fact work to a reasonable degree. In the feasibility study stage, it needs to be considered whether something more sophisticated such as the Barro Alto concept is adopted.

13.3.2 Pyrometallurgical laboratory testing by XPS

Testwork summary

As a part of testwork for the PFS, HZM supplied three different types of ore samples to XPS. XPS first performed a series of sample characterisation tests which included moisture measurements (free and crystalline), particle size analyses and chemical assays of the ore and calcine samples. After the initial phase of sample characterisation, a series of smelting tests were conducted at elevated temperature (1520 ^{0}C). At the end of each smelting test, the Fe-Ni alloy and slag products were collected, weighed and submitted for chemical analyses. As a part of the study, the potential recovery of Ni was also estimated based upon small scale batch tests.

To estimate the required processing energy, preliminary heat and mass balance calculations were also performed. Based upon these calculations, a simplified flow sheet was created. The indicated flow sheet included steps such as feed preparation, drying, calcining and smelting. Suggestions also were made for modifying the flow sheet towards the optimisation of energy consumption.

The report contains all the results from the laboratory tests and thermo-chemical calculations. Based upon XPS's experience in the production of Fe-Ni, some suggestions are also made.

Arising from the work the following items are highlighted:

- Particle size analyses indicated that the fineness of the ore suggests that material handling challenges are to be expected, particularly during wet weather:

  - If a rotary kiln based calcination step is chosen, significant agglomeration of the feed will be required to prevent dust issues.
  - Dusting during drying will also be an issue.
- A feed blending step is necessary in order to reach an optimum level of Fe/Ni ratio in the ore feed. The feed preparation step can also be tailored towards the production of Fe-Ni alloy or matte as the final products.
- Preliminary batch smelting tests were conducted at 1520^{0}C on three samples. Smelting tests were repeated to ensure reproducibility. An alloy grade of 30 wt% was targeted in each case. The tests were successful in producing this grade of alloy for two samples. The smelting tests with specific blend ratios were also successful in achieving targeted alloy grade. The Fe-Ni grades obtained in the tests varied from 14% Ni to 53% Ni.
- The possibility of producing a matte phase was also investigated. It should be pointed out that the market and margins for Fe-Ni alloy and matte are different.
- The possibility of utilising the sensible heat and chemical reduction potential of the off-gas from the smelting step opens up a number of flow sheet options to be explored and examples are discussed.
- Extensive thermo-chemical modelling was performed predicting the liquidus temperatures of all possible phases. Based upon preliminary heat and mass balance calculations, a basic flow sheet was developed to treat the Araguaia deposits. The energy requirements for each processing step of the flow sheet were calculated and presented. The amount of reductant to produce 30 wt% Ni alloy was also calculated. It should be pointed out that the basic flow sheet can be optimised to reduce the fuel and reductant consumptions. The opportunities for energy savings were also identified and discussed.

XPS also carried out a simplified heat and mass balance.

The testwork demonstrated that the SiO_2/MgO ratio has a significant impact on the slag liquidus temperature and should be monitored with regards to the design of the furnace, especially with regards to potential cooling system requirements. The three samples tested had SiO_2/MgO ratios of 6.7, 4.4 and 2.3 respectively.

IGEO carried out a review of the XPS test work and raised a number of concerns regarding the moisture content of the ore, LOI results and inconsistencies associated with the interpretation of this data. In consultation with HZM, however, the evaluation of this data for design purposes was resolved.

A valuable aspect of the XPS report was the focus on the thermodynamic properties of Fe-Ni slags and associated alloys. The criticality of the liquidus temperatures on operating conditions and design issues for the furnace will pose a significant challenge for the Araguaia project and mitigation of the associated risks will have to be a strong focus in any future more detailed study phases. Additionally, benchmarking data on other operations were provided in the XPS report for comparison, particularly in terms of slag liquidus characteristics. Having had association with a number of these operations, it is recommended that for this PFS, HZM endeavours to obtain more recent information and experiences on current kiln and furnace operations at Cerro Matoso. In conjunction with Hatch, operating staff at Cerro Matoso probably have had the most experience in dealing with furnace design and operations issues associated with ratios of SiO_2:MgO values above 2.0. From a previous site visit by IGEO it was understood that Cerro Matoso is probably now operating at SiO_2:MgO ratios of below 2.0.

13.3.3 KPM

Two reports were prepared by KPM and reviewed during the PFS.

13.3.4 KPM Report 29 July 2012

Summary

Simulations conducted by KPM indicated that XPS results were in close agreement with the thermodynamic calculations performed in this KPM study. The coke consumption in the electric furnace is expected to range from 3 to 5% addition (over the feed) in the electric furnace depending on the ore blend and also the level of pre-reduction of the calcine.

According to the thermodynamic (FACT) simulations with the resultant calcine, the proposed blend provides a challenge and an opportunity for the smelting in the electric furnace by providing a low slag liquidus temperature (lower than the Fe-Ni alloy liquidus), which will allow operating at a lower temperature, but at the same time will require considerations for the furnace design.

The impact of the SiO_2/MgO was mapped and the anticipated ratio in the proposed blend is 2.6 with FeO content in the range of 20% to 25% by weight in the slag. With these conditions, KPM suggests that no major problems should be experienced during the smelting of this laterite blend. The alumina content of the slag is in a range for having a low slag liquidus and the Cr_2O_3 level does not appear to be at critical level for creating accretion problems in the furnace. The reasonably low FeO level of the slag reduces the partial pressure of oxygen, hence minimising the soluble nickel in the slag. Under the present oxidant conditions, allowing the production of a 20% to 25% Ni alloy, a small portion of the nickel is expected to be in the form of soluble nickel (0.01wt% Ni in slag). The remaining nickel should be in the form of entrained metal. Total losses are expected to be at nearly 0.15% when looking at losses in similar operations.

KPM was also requested to perform a number of FACT calculations regarding the slag chemistry during the smelting of Araguaia laterite ore to ferro-nickel. The purpose of the study was two-fold:

- to simulate the results of a selected number of laboratory smelting tests carried out at XPS during 2011.
- to evaluate the smelting of a new laterite blend proposed by HZM, including an approximate estimate of the reduction requirements for a commercial plant to produce 20% to 25% Ni alloy, and in particular to carry out an evaluation of the slag chemistry and estimate the optimal operational temperatures. The latter part of the study also covers an evaluation of the key slag components (the SiO_2/MgO ratio, the %FeO and %Al_2O_3 levels in slag) and provides phase diagrams showing the impact of each parameter on the slag melting point. A discussion was also provided with respect to the slag superheat, furnace operating temperature, and comments on design criteria for the electric furnace.

Conclusions

- The results indicated that XPS experiment outputs are in close agreement with the thermodynamic calculations performed in this study. Based on the work here, the coke consumption in the electric furnace is expected to be in the range from 3% to 5% of the calcine feed to the electric furnace, depending on the feed blend and pre-reduction level of the calcine (Fe^{2+} and Fe^{3+} levels).
- The new blend proposed for the Araguaia laterites brings a challenge and an opportunity in the smelting of this ore by providing a low melting point slag, which will allow operating the electric furnace at lower temperatures of 1,510°C to 1,530°C comparing with other electric furnaces. The resulting high superheat required, however, to maintain the metal as liquid has to be considered in the furnace design. Another possibility is to add sulphur to the metal producing matte with lower liquidus temperature.
- The alumina content of the slag (~7%wt) also favours a low slag liquidus temperature. The reasonably low FeO level of the slag reduces the partial pressure of oxygen in the furnace, hence minimising the soluble nickel in the slag. A small portion of the nickel will be in the form of soluble nickel and some will be in the form of entrained metal.
- KPM sees no major technical constraint in smelting this type of blend in an electric furnace, provided that a careful furnace design is taken into account.

Other possible future activities were recommended. These include:

- Evaluation of the best drying and pre-reduction technologies for Araguaia laterite.
- Discussion with electric furnaces suppliers to evaluate the best design and have a CAPEX evaluation for the furnace. To consider copper coolers, deep immersion, electrodes configuration and electric heating (DC vs AC).
- Modelling of heat and mass balance for the Fe-Ni production plant with 2-3 configurations (pre-reduction, smelting and refining).
- Evaluation of sulphur and other impurities in Fe-Ni and layout of the Fe-Ni refining (literature review, experimental test and modelling), also evaluate the alternative to form matte as regards, lowering the liquidus temperature of the metal phase and at the present slag composition.

13.3.5 KPM Report 26 April 2013

Summary

KPM were also contracted by HZM to measure slag liquidus temperatures for a Fe-Ni smelting process. Differential thermal analysis (DTA) combined with thermo-gravimetry analysis (TGA) was used to determine the liquidus temperatures. In all, six different synthetic slag compositions were studied to determine the influence of alumina (Al_2O_3) concentration and silica/magnesia (SiO_2/MgO) ratio on the liquidus temperatures.

The results confirmed that the FACT calculations for this system provide an accurate representation of the liquidus within ~10°C (on the high side) of the measured values.

Conclusions

Six tests were completed to determine liquidus temperature at different slag compositions varying Al_2O_3 (0, 4 and 7.36%) and SiO_2/MgO ratio (2.3, 2.4, 2.55 and 2.7). Slag solidification is an exothermic process which could be detected by DTA. The liquidus was measured at two different rates, 5^{o}C/min and 20^{o}C/min but values obtained at a lower cooling rate (5^{o}C/min) were used for higher accuracy.

The liquidus temperature measured experimentally followed the same trend as the projected (FACT) values and consistently lower by 10-20^{o}C. The FACT values should be taken as an uppermost value for design purposes.

At constant SiO_2/MgO ratio (2.55), the liquidus temperature was found to decrease significantly when the Al_2O_3 level was increased. The liquidus temperatures were 1,383°C and 1,431°C at 7.36% and 4% Al_2O_3, respectively. The liquidus was above 1,450°C at 0% Al_2O_3, above the maximum allowable test furnace temperature. The liquidus temperature decreased from 1,405°C to 1,368°C as the SiO_2/MgO ratio was increased from 2.3 to 2.7.

The outcome of the KPM reports has provided valuable information on variation of slag liquidus temperatures with variations in the slag analyses anticipated in treating the ores from the Araguaia project. This information is absolutely critical in:

- evaluating the furnace operational conditions likely to be encountered
- designing the furnace with appropriate high intensity water cooling facilities for the walls.

The impact of slag viscosity on slag foaming and potential nickel losses to slag must also be evaluated. These issues are the most crucial to be addressed for the potential success of the Araguaia project.

13.3.6 FLS

Summary

A laboratory study was performed beginning in the third-quarter of 2012 to evaluate the physical and chemical properties of two nickel laterite ore blends. As a follow-up to further expand the results of that testing, HZM authorised additional testing of the 51/49 blend (see below) generated in Phase 1 testing, identified as FLS Lab# 120274M. Simultaneously, material from the first test phase was returned from Feeco agglomeration testing and subjected to FLS tumble testing.

For the testwork the supplied materials were combined into two blends mixed in proportions by wet weight as follows:

- Mix 1: 51% T and 49% S
- Mix 2: 14% L, 44% T and 42% S

The blended mix samples were relatively free flowing despite free moisture levels in the range of 34% to 38 wt%.

Ore physical analyses

The samples were analysed to determine free moisture content, bulk density, angle of repose, particle size distribution, drying curves and particle degradation (tumble testing). The free moisture levels were determined using a drying oven temperature of 105°C and were 37.70 wt% for Mix 1 and 34.53 wt% for Mix 2. Particle size distributions were determined by wet and dry sieve methods:

The dry sieve results include 100% passing 31.8 mm, 69.9% to 73.9% passing 6.4 mm and a D_{50} of about 4 mm to 4.5 mm. A D_{50} value of about 4 mm to 4.5 mm falls within the D_{50} range (1.2 mm to 9.5 mm) of other laterite ore samples that have been analysed by FLS.

The shift in D_{50} analyses between wet and dry material is significantly greater than for other ores tested by FLS and initially suggests an ore dusting potential exceeding that of lateritic ores currently processed in rotary dryer or kiln systems. This test does not account for the natural binding properties of the ore, which may limit the degree of particle size reduction that actually occurs during rotary drying and kiln firing. The natural binding properties of the Araguaia ore are very good when dried (similar to many clays), yielding hard agglomerates that are expected to be resistant to dust generation during processing in rotary systems. The end result is a dry particle size distribution and dusting potential comparable to numerous laterite ores currently processed in commercial RKEF operations.

Modified ASTM E279 tumble tests were performed on the two mixes following drying at 105°C and screening to 50.8 mm x 9.5 mm. The tumble test product was then screened at seven size ranges from +19 mm to below 90 µm. Overall degradation of the ore was slightly higher than saprolitic ores and less than limonitic ores tested by FLS. This data suggests a kiln dusting potential of 15% to 20% of dry feed input, somewhat higher than the majority of Fe-Ni rotary kiln operations at10% to 15% of dry feed input.

Drying curves were developed and the time required to achieve complete moisture reduction was similar for both sample mixes at approximately 30 minutes at 105°C.

A series of attrition tests (10 minute rotap) were performed to establish the agglomerated ore's resistance to degradation upon firing at various temperatures. The attrition behaviour of the ore was compared upon treatment at 105°C, 500°C and 1000°C. A comparison of the size distribution of the ore samples following the thermal treatment and rotap procedures indicate no change between 105°C and 500°C, and only a minor increase after treatment at 1000°C. This data suggests that the agglomerated ore will not experience significant degradation as it is heated in the rotary kiln process.

Ore chemical analyses

The chemical assays for the two mixes were determined by FLS (Table 13.6).

Table 13.6 Chemical analysis

Lab No.:	120274M		Lab No.:	120276M	
Sample ID:	51/49 Mix		Sample ID:	14/44/42 Mix	
SiO_2	42.2	%	SiO_2	39.4	%
Al_2O_3	5.54	%	Al_2O_3	5.83	%
Fe_2O_3	25.2	%	Fe_2O_3	27.6	%
CaO	0.23	%	CaO	0.19	%
MgO	13.8	%	MgO	13.0	%
K_2O	0.014	%	K_2O	0.012	%
Na_2O	0.109	%	Na_2O	0.105	%
SO_3	0.018	%	SO_3	0.018	%
NiO	2.16	%	NiO	1.96	%
TiO_2	0.19	%	TiO_2	0.18	%
MnO	0.37	%	MnO	0.40	%
Loss @ 982^o C	11.13	%	Loss @ 982^o C	10.64	%
Total	100.96	%	Total	99.34	%

The XRF analysis also indicated total chrome levels as Cr_2O_3 of 1.36 and 1.38 wt%.

Speciation work by SGS does not support the presence of significant Ni-chromite in the ore samples, but rather indicates that the nickel is dominantly contained within nontronite, goethite, hydrated Magnesium silicates (serpentine and chlorite) and Manganese oxide/hydroxides.

Ore thermal analysis

The mix samples were dried at 105°C and ground and were analysed via TGA – DSC. The total sample weight losses measured between ambient and 1,000°C were 13.09 and 12.99 wt% for Mix 1 and Mix 2 respectively.

The DSC test was continued to a maximum temperature of 1500°C in order to establish melt point temperatures. The following partial and primary melt temperature ranges were reported:

- Mix 1: 1,250°C to 1,340°C partial, 1,340°C to 1,420°C primary
- Mix 2: 1,270°C to 1,335°C partial, 1,335°C to 1,440°C primary

Ore sintering and reduction

Samples of the two mixes were dried, sized to minus 6.3 mm and blended with 5 wt% coal (ground to pass through 2mm) in preparation for sintering and Fe/Ni reduction tests. The coal used as reductant was previously used to simulate PT Inco kiln operations and includes a volatile content of 35 to 40 wt%. The coal properties are also similar to several major Colombian bituminous coal sources that are used by Fe-Ni producers operating in Brazil.

The sintering and reduction tests were performed in a tube furnace using peak temperatures of 1,100°C, 1,150°C and 1,200°C under a reducing atmosphere (85% nitrogen, 10% carbon dioxide and 5% carbon monoxide). A small degree of particle sintering was evident in both ore mixes following treatment at 1,150°C, followed by significant sintering/melting in select regions following treatment at 1,200°C. The majority of ores tested by FLS do not exhibit significant sintering until the temperature is >1,200°C. This data suggests that a lower calcine temperature should be considered to limit sinter formation in the rotary kiln. Based on practical operating experience a target calcine discharge temperature of 800-825°C should be considered for commercial system design purposes versus levels of 850-900°C considered for ores that do not exhibit significant sintering until >1,200°C.

Each sinter sample was analysed to establish Fe and Ni speciation and residual organic carbon. The level of Fe reduction achieved is considered normal for this test method while the Ni reduction level is lower than the normal range of 20% to 30%. The lower nickel reduction may be attributed to a portion of the nickel being combined as Ni-chromite. Speciation work subsequently conducted by HZM suggests that the nickel is associated with nontronite (a clay-like mineral) and some nickel is combined as complex silicates – both of which may also explain the lower reduction levels observed - while virtually no nickel was identified as associated with Ni-chromite.

Low residual carbon levels are attributed to the high carbon consumption resulting from the elevated iron content and suggest higher than typical reductant addition rates (typical 3% to 4% of dry ore input) are required in the rotary kiln to support high pre-reduction levels while maintaining an acceptable residual carbon level to support the electric furnace operation. HZM is considering additional laboratory reduction analysis utilising a higher reductant coal addition of 8.5 wt%.

Figure 13.4 FLS pilot kiln

Source: FL Smidth

Following the completion of this analysis HZM requested additional testing to further evaluate reduction behaviour using alternate temperatures and reductant coal addition rates. The sintering and reduction tests were performed in a tube furnace using peak temperatures of 800°C, 900°C and 1,150°C under a reducing atmosphere (85% nitrogen, 10% carbon dioxide and 5% carbon monoxide). Testing was conducted utilising 5 wt% coal at 900°C and 8.5% weight coal at all three temperatures.

As in the previous test burns, a small degree of particle sintering was evident in the ore mix beginning at a temperature of 1,150°C.

Each sinter sample was analysed to establish Fe and Ni speciation and residual organic carbon. Previously, the level of Fe reduction achieved was in what is considered a normal range for this test method while the Ni reduction level was lower than the normal range of 20% to 30%. Notable increases in both Fe and Ni reduction were achieved when combining processing at 1150°C with a higher reductant addition levels. Based on the sintering behaviour of the ore and considering pre-reduction levels achieved in existing commercial Fe-Ni kiln operations, FLS considers 10% Ni reduction and 60% Fe reduction levels to be acceptable targets for Fe-Ni line design purposes.

Figure 13.5 FLS Tumble or breakage test unit (modified ASTM Test E279)

Source - FLSmidth

Briquetting

Briquetting tests were carried out at KR Komarek to evaluate the briquetting properties of Mix 1. The following conclusion was drawn:

"Based on Komarek' s experience you can briquette the ore at around 15% moisture and we estimate that coal maybe a good addition for the briquettes, however at other plants the ore alone can be briquetted as long as the particle and moisture levels are kept within the Parameters of the pocket size and roll size."

Small samples of briquettes delivered to FLS demonstrated good green strength; they did not crush easily in one's hand nor break when dropped. A moisture analysis indicated a total free moisture level of 17.4 wt%. In the event that rotary agglomeration is not effective, initial tests suggest that briquetting is a viable option to transform rotary dryer product to a briquette suitable for rotary kiln processing. Additional larger scale briquette testing followed by pilot rotary kiln processing is required to confirm this.

Rotary drum agglomeration tests

HZM made arrangements with Feeco to evaluate the potential benefit of promoting particle agglomeration using a rotary device prior to subsequent rotary drying and kiln processing (modified ASTM E279 tumble tests). The material was dried at 105°C and screened to 50.8 mm x 9.5 mm prior to conducting the tumble test. Tumble test feed and product sieve analyses were carried out. While the coarser agglomerates degraded to a larger extent than the dried ore (static drying process), virtually no fines were generated during tumbling. This suggests a lower dusting rate in a rotary kiln operation as a result. Therefore, the design of the ore preparation and drying circuit should include provision to maximise ore agglomerate formation and strength development prior to the rotary kiln process.

FLS conclusions

- The Araguaia ore is characterised by a very fine natural particle size. The fine particles demonstrate binding properties similar to clays when dried, thereby yielding relatively hard agglomerates resistant to significant degradation and dusting. The end result is a dry particle size distribution and dusting potential comparable to numerous laterite ores currently processed in commercial RKEF operations.
- The onset of particle sintering is 50°C to 100°C lower than many lateritic ores evaluated by FLS. This suggests a limited achievable calcine temperature of 800 to 825°C during rotary kiln processing (versus 850 to 900°C), which will also limit the degree of iron pre-reduction that can be obtained.
- FLS suggests the use of 10% Ni and 60% Fe pre-reduction targets for commercial Fe-Ni line design and that higher degrees of pre-reduction will be difficult to achieve given the ore sintering behaviour and high iron content. Pilot rotary kiln testing is required to confirm final design criteria. Comments on the design data for pre-reduction are given in the review below.
- Briquetting appears to be a viable option for producing an agglomerated feed suitable for kiln processing to yield a granule calcine with acceptable dusting rates.
- Rotary drum agglomeration demonstrated the production of agglomerates resistant to fines generation during tumbling.
- The commercial system feed preparation and drying circuits must consider provisions for promoting ore agglomeration and strength development prior to the rotary kiln process.
- The results of this laboratory study suggest that the Araguaia ore is suited for rotary kiln processing in an RKEF system provided that proper agglomeration provisions are adapted and that lower calcine temperature and pre-reduction levels are considered in the electric furnace design. This statement does not consider the suitability of the ore for the electric furnace smelting process.
- Larger scale pilot testing is recommended to confirm the conclusions of this study. The testing will confirm: ore agglomeration behaviour, preferred method of drying/agglomeration, agglomerated ore behaviour in the rotary kiln, dusting rate, degree of Fe/Ni reduction achieved, peak calcine temperature, sintering behaviour and reductant consumption.

A review of the FLS test work has provided valuable information which, in turn, has provided an understanding of the behaviour of the ore in the context of materials handling and during calcining and smelting. Specific issues are commented on as follows:

- The as-received ore was stated to be relatively free flowing despite a free moisture content of 34% to 38%.

- Carrying out both wet and dry sieve analyses provides important information on the potential behaviour of lateritic ores during handling.
- Overall degradation of the tumble test product during screening was said by FLS to be slightly higher than other similar ores and FLS suggested a dusting potential of 15% to 20% of dry ore feed be considered. A value of 20% has been incorporated in the preliminary Process Design Criteria (PDC).
- With respect to the ore chemical analyses, of particular note for IGEO was the combined moisture value of approximately 11%. This is considered probably more realistic for design purposes than the approximately 7% reported in the XPS testwork report.
- From the ore sintering and reduction testwork results, FLS have recommended that reduction values of 10% for nickel and 60% for Fe be applied to the kiln design.
- It may be necessary to restrain the target temperature of the calcine discharge to 800-825^{o}C. For design purposes for the kiln, however, a target temperature of 850^{o}C has been defined for the kiln discharge calcine and 825^{o}C for the calcine feed to the furnace.
- FLS considers it may only be possible to achieve pre-reduction values of 10% for Ni and 60% for Fe. IGEO have defined 20% for Ni and 70% for Fe in the PDC but concur with FLS that this design issue must be confirmed in future pilot scale testwork.
- Testwork on briquetting showed that the addition of coal was necessary to obtain briquettes of suitable competency. It is believed that briquetting of fresh ore will require additional unnecessary capital expenditure and a simpler flowsheet is being proposed.
- It is noted that no carbon and silicon analyses of the alloy are provided.

13.3.7 Discussion on smelting testwork

The following section provides a discussion on the smelting testwork. In addition, as noted earlier, Hatch (Toronto) was requested to examine available information on the smelting characteristics of the Araguaia ore. Taking all information into account, IGEO then developed the proposed metal and slag characteristics as summarised in Table 13.7.

Slag Characteristics

The testwork carried out at XPS and KPM on Araguaia ore blends focussed considerably on the smelting characteristics of the slags produced from the ores because these slags differ significantly from most other current Fe-Ni operations in the world. It is important to fully understand the implications of these characteristics on both the furnace design and on the operational challenges which will be presented. Fortunately, due to the extensive amount of technical knowledge which has developed on nickel laterite smelting over the years, the fundamentals of the slag characteristics and impact of slag chemistry can be fully evaluated and the risks assessed.

Key factors which need to be considered are:

- The combined impact of the ratio SiO_2/MgO, FeO and Al_2O_3 contents of the slag on the liquidus temperatures and the viscosity of the slags. The chemical effects of high SiO_2 slag on the basic furnace refractory material must also be considered.
- The impact of impurities, particularly carbon and silicon, on the liquidus temperatures and viscosity of the metal. Also the risks must be considered of silicon reversion and associated extensive generation of heat from the reaction if silicon content is too high in the metal.

- The impact of the issues in 1 and 2 above can be affected/controlled to varying degrees by controlling the degree of reduction of Fe effected during the smelting process.
- Slag foaming is also an issue that can occur with viscous slags and silica and alumina play significant roles in this issue. Such foaming is also likely to increase nickel losses into the slag.
- Very importantly, the degree of iron reduction dictates the grade of nickel to be achieved in the final metal product as well the silicon and carbon content of the metal. Higher reduction will result in lower metal grades and in all the testwork evaluations, grades of nickel content of between 15% Ni and 25% Ni have been evaluated. Higher degrees of reduction also give the added benefit of higher nickel recoveries across the furnace.
- Basically, at the ratio of SiO_2/MgO of 2.29 and the FeO and the Al_2O_3 content anticipated in the Araguaia slags, the liquidus temperatures can be significantly depressed. However, minimum tapping temperatures of both metal and slag must be attained to ensure fluidity. This therefore can result in excessive superheating of the slag with possible undesirable consequences to the operation. It also will demand intensive water cooling systems be installed on the furnace sidewalls.

Metal liquidus temperature

- The XPS report considered the binary Fe-Ni system only and reported temperatures above 1500^{o}C. KPM did not include the effects of C, Si and S on the alloy and reported a nominal metal liquidus temperature of 1479^{o}C.
- It is noted that the studies carried out showed that a 50% degree of iron reduction (with respect to Fe/Ni ratio) is required to get substantial amounts of C and Si into the metal. These elements have an important influence on the metal liquidus temperature. Reduction of over 50% would result in high C, Si and Cr in the metal and a liquidus temperature of 1250-1350^{o}C.
- The studies showed that there is the possibility of the introduction of sulphur into the metal to reduce the metal liquidus temperature. The objective would be to produce a matte product with lower liquidus temperature than the Fe-Ni metal. This concept requires further treatment methods of the matte and therefore is not considered for the PFS phase of the Araguaia project.

Summary of liquidus temperatures

It was from consideration of all the data available that HZM, in conjunction with IGEO, agreed that the target metal grade from the furnace will be 20% nickel, and that this value was to be adopted for the PDC in PFS. Given this criterion, and evaluating all the relevant data from liquidus temperature determinations in the KPM testwork, the liquidus and tapping temperatures of metal and slag were defined as shown in Table 13.7. The resultant superheat contained in the metal and slag are factors which will have to managed from both an operational perspective and in designing of robust cooling systems required for the furnace walls. Attention is also being given to the furnace roof construction such that more rapid repairs can be carried out if the roof is subjected to high temperatures due to operating conditions within the furnace.

The tapping temperatures achievable due to the liquidus temperatures of metal and slag are criteria that will need to be verified in pilot scale work with a bulk sample in the next phase of the project.

Table 13.7 Estimated furnace metal and slag temperatures

	Araguaia Criteria
Ore Blend	LOM Plan Blend
Target crude Fe-Ni grade % Ni	20
Metal liquidus °C	1,440
Slag liquidus °C	1,380
Metal tapping temperature °C	1,470
Slag tapping temperature °C	1,550
Metal - Slag delta T °C	80
Metal superheat °C	30
Slag superheat °C	170
Nickel furnace recovery %	95.0

13.3.8 Other testwork conducted

During earlier stages of the project hydrometallurgical testwork was also conducted, with two approaches reviewed:

- Atmospheric tank leach
- Bottle roll tests to simulate heap leaching.

Sulphuric acid was selected for both types of leaching tests. Acid consumption was found to be high and these options were not further pursued after the initial testwork was completed.

Atmospheric tank leach tests

Preliminary atmospheric tank leach tests were undertaken under different test conditions to investigate the leachability of Araguaia blended ore with respect to the effect of particle size, solid/liquid ratio, acid strength, leaching temperature, leach time on metal recoveries and acid consumptions. A composite feed ore blend consisting of 15% limonite, 45% transition and 40% saprolite core drill samples was used to carry out the 21 batch tests.

The batch tests clearly established that Araguaia laterite ore was leachable in tanks with promising results. Nickel leach rates were encouraging with up to 65% of nickel extracted within the first hour and 89% extraction achieved in 4 hours leaching. Ranges of 70-89% recovery for nickel and 68-93% recovery for cobalt, with acid consumptions of 500 – 750 kg/t, were achieved in 16 of the 21 tests.

Bottle roll leach tests

Sixteen bottle roll tests were performed at WAI test laboratories using separate limonite, transition and saprolite samples as well as a composite ore blend. The tests were carried out under atmospheric pressure and ambient temperature. Solid/liquid ratio was 1:10 using 150 grams of feed ore for each test. Two ore sizes: -6.35mm and -3.35mm and two acid concentrations: 75 and 100g/l were tested for each type of ore.

At the end of 91 days leaching, the transition and saprolite samples produced nickel recoveries in excess of 85% and the composite feed ore blend generated nickel recoveries in the order of 85-90%. As was expected, nickel recoveries for the pure limonite samples were lower (51%). Acid consumptions varied in the range of 400-800 kg/t ore for different ore types and test conditions.

13.4 Key recommendations from the testwork conducted

- For a further stage of the project, pilot plant scale agglomeration, calcining and smelting testwork should be conducted:
  - Confirmation of correlations in key data such as Fe-Ni metal grade and Fe:Ni ratio in ore against nickel recovery in smelting.
  - The criticality of the liquidus temperatures on operating conditions.
- Evaluation of the best drying and pre-reduction technologies for Araguaia laterites.
- The impact of slag viscosity on slag foaming and potential nickel losses to slag must also be evaluated.
- Modelling of heat and mass balance for the Fe-Ni production plant with 2-3 configurations (pre-reduction, smelting and refining).
- Evaluation of sulphur and other impurities in Fe-Ni and layout of the Fe-Ni refining.
- The following issues from the FLS testwork are relevant to future operational and design criteria for Araguaia:
  - The as-received ore was stated to be relatively free flowing despite a free moisture content of 34% to 38%. It is the opinion of IGEO that this moisture content will decrease during handling and storage on a commercial operation and thus a value of 30% has been incorporated in the PDC.
  - Carrying out both wet and dry sieve analyses provides important information on the potential behaviour of lateritic ores during handling. It has been found that the fines in some ores tend to agglomerate and then subsequently disintegrate upon sun drying and this has to be considered in the design of the front end crushing plant.
  - Overall degradation of the tumble test product during screening was said by FLS to be slightly higher than other similar ores and FLS suggested a dusting potential of 15% to 20% of dry ore feed be considered. From IGEO experience the value of 20% is a practical figure to apply and this has been incorporated in the preliminary PDC.
  - The attrition tests carried out on the agglomerates after firing at temperatures 105°C, 500°C and 1 000°C provided valuable information on competence of the ores at high temperatures. The reality, however, is that in a commercial size kiln other abrasion factors prevail and occur over a much longer period with retention times of approximately 3 hours. Nevertheless, the FLS results obtained of minimal increase in degradation between 105°C and 1 000°C is an important observation.
  - From the ore sintering and reduction testwork results, FLS have recommended that reduction values of 10% for nickel and 60% for Fe be applied to the kiln design. For the purposes of design specifications for equipment suppliers, IGEO has defined the pre-reduction values for Ni and Fe as 20% and 70% respectively and these have been incorporated into the preliminary PDC. It is recommended, however, that these be confirmed in larger scale tests in the next phase of the project.

- From sintering observations FLS have recommended that for a commercial scale kiln the calcine temperature be constrained to 800-825°C. For design purposes, however, IGEO has defined a calcine temperature at kiln discharge of 850°C and a calcine temperature of 825°C feeding into the furnace. This requires verification on a pilot scale.
- Testwork on briquetting showed that the addition of coal was necessary to obtain briquettes of suitable competency. It is believed that briquetting of fresh ore will require additional unnecessary capital expenditure and a simpler flowsheet is being proposed.
- Carbon and silicon analyses of the alloy must be monitored.

* In addition, from the work carried by HZM, the following points were noted for consideration during the feasibility stage of the project:
  - Whether addition of MgO to the furnace feed is a strategy that should be considered for this phase of the Araguaia project to modify the slag characteristics.
  - Further experimental work to determine the slag liquidus temperature in the expected compositional range.
  - Further evaluation of metal liquidus temperatures at different impurity levels.
  - The need to conduct slag leachability testing for environmental purposes and establish the need or otherwise for lining the slag dump.
  - Power density and operation in a shielded arc or immersed electrode mode are key factors in the design and operation of the furnace.
  - A marketing study is required to define the grade of Fe-Ni to be targeted for the Araguaia project. This is key to defining the slag characteristics and thus furnace design and operational conditions.

14 Mineral Resource estimates

14.1 Summary

Mineral Resource estimates are currently reported for the nickel laterite deposits at Araguaia in Table 14.1. At a cut-off grade of 0.95% Ni, a total of 72 Mt at a grade of 1.33% Ni is defined as Indicated Mineral Resource and a further 25 Mt at a grade of 1.21% Ni is defined as Inferred Mineral Resource.

Mineral Resources reported for the PFS deposits were prepared under the supervision of Mr. Andrew F. Ross and reviewed by Mr Richard Sulway. Both are employees of Snowden.

Mineral Resources for other deposits in the project area were prepared by Dr. Marc-Antoine Audet and were reported in Audet, M A, et al (2012). The other deposits are Pequizeiro NW, Oito Main, Lontra North and Raimundo for which Inferred Mineral Resources are reported. These other deposits were not considered in the PFS.

Snowden is unaware of any issues that materially affect the mineral resources in a detrimental sense.

14.2 Mineral Resource estimates for the PFS based on drilling up to Phase 3

The HZM Phase 3 drilling resulted in updated databases for areas PQW, VOE, JAC, VOW and VOI. So that consistency was applied, Snowden also re-estimated mineral resources for the BAI and PQZ areas.

The estimates were prepared in the following steps:

- data preparation
- geological interpretation and horizon modelling
- establishment of block models and definitions
- compositing of assay intervals
- exploratory data analysis and variography
- Ordinary Kriging estimation method and Parameters
- model validation
- calculation of dry density
- classification of estimates with respect to JORC guidelines
- resource tabulation and resource reporting.

Details of the resource estimation work were reported to HZM in Ross (2013). Details are summarised below.

Table 14.1 Mineral Resources for Araguaia as at March 2014 by material type (0.95% Ni cut-off grade)

Araguaia	Category	Material type	Tonnage (kT)	Density (t/m3)	Contained Ni metal (t)	Ni (%)	Co (%)	Fe (%)	MgO (%)	SiO_2 (%)	Al_2O_3 (%)	Cr_2O_3 (%)
Sub-total	Indicated	Limonite	11,560	1.35	137,790	1.19	0.127	36.50	2.76	19.45	9.48	2.61
Sub-total	Indicated	Transition	24,110	1.19	346,920	1.44	0.060	19.87	11.36	41.19	5.05	1.38
Sub-total	Indicated	Saprolite	36,310	1.32	473,960	1.31	0.034	11.82	23.67	42.27	3.62	0.85
Sub-total	Inferred	Limonite	8,830	1.34	100,310	1.14	0.097	35.85	3.94	19.77	9.48	1.78
Sub-total	Inferred	Transition	9,340	1.28	122,040	1.31	0.053	20.34	13.94	37.80	5.31	1.20
Sub-total	Inferred	Saprolite	7,190	1.41	84,370	1.18	0.033	12.07	23.92	41.46	4.16	0.80
TOTAL	Indicated	All	71,980	1.28	958,660	1.33	0.058	18.48	16.19	38.25	5.04	1.31
TOTAL	Inferred	All	25,350	1.34	306,730	1.21	0.063	23.40	13.29	32.56	6.43	1.29

Note: Totals may not add due to rounding. Mineral Resources are inclusive of Mineral Reserves.

14.2.1 Data provided and preparation

HZM provided Snowden with a series of GEMS project databases that were compiled by Dr. Marc-Antoine Audet for resource estimates completed in 2011. Following receipt of the Phase 3 drilling data, Snowden updated the relevant tables by importing collar surveys, sample identifiers and assays for the new infill drilling. Validation routines were run inside GEMS to identify any discrepancies such as duplicate or missing records, and no significant issues were identified.

14.2.2 Geological interpretation and horizon modelling

The supplied geological maps in PDF format were used to guide the 3D interpretation of digital surfaces to constrain the distribution of drilled limonite, transition and saprolite horizons between the surface topography and bedrock. This process was applied to the PQW, JAC, VOW, VOI, and VOE areas.

The initial step required verification of the supplied surveyed drillhole collar elevations with the surface topography digital terrain models (DTM) provided by HZM. Any discrepancies were rectified by pressing the drillhole collar against the surface topography DTM.

The major constituent chemistry of each sample was used by Dr. Marc-Antoine Audet to assign a facies code (one of limonite, transition, saprolite or fresh rock) for each sample interval. The resultant codes were then grouped and internally adjusted by Snowden to ensure a logical sequence of horizons existed for each drillhole. These were imported into a GEMS table and the base elevation of each horizon (limonite, transition and saprolite) was used to generate 3D surfaces by way of the Laplace algorithm provided by GEMS.

Horizon surface extrapolations were constrained by a distance of 25 m from the perimeter drillholes. In the case of VOE, a 3D wireframe was interpreted for a barren dyke, based on the supplied geological map and drillhole information.

Snowden retained the triangulated horizon surfaces for BAI and PQZ that were modelled by Dr. Marc-Antoine Audet, since there are no additional Phase 3 drillholes for these areas.

Horizon surfaces and coded drillhole assays were exported from GEMS and then imported to Datamine Studio 3 software for compositing, block model construction and estimation. Variogram analysis was undertaken in Snowden Supervisor V8 software.

A consistent set of codes was used to define limonite (100), transition (200), saprolite (300), fresh rock (500) and dyke (450, at VOE).

14.2.3 Block model definitions

The final model extents are listed in Table 14.2. The sample density (drillhole spacing 100 mE by 100 mN) was considered when selecting the parent cell size of 25 m x 25 m x 2 m (XYZ). In the vertical direction, the parent cell size is 2 mRL reflecting the likely mining bench height.

Table 14.2 Block model definitions

Model definition	VOW	VOI	VOE	JAC	PQZ	PQW	BAI
X Origin (mE)	677300	680500	684600	675500	673700	671700	675000
Y Origin (mN)	9127000	9127600	9128800	9123100	9114300	9116500	9108900
Z Origin (mRL)	200	100	150	150	200	220	180
Maximum Easting (mE)	679700	683600	686200	676300	678100	672900	677700
Maximum Northing (mN)	9129400	9129800	9130000	9123950	9116600	9117600	9113300
Maximum Elevation (mRL)	400	400	300	350	310	310	400

Sub-celling to 6.25 m x 6.25 m x 0.5 m (XYZ) was employed to honour the horizon wireframes.

14.2.4 Compositing of assay intervals

The drillhole data was composited downhole prior to running the estimation process using a nominal 1 m sample interval to minimise any bias due to sample length. Assigned density values were used to weight the composites as was done previously by Dr. Marc-Antoine Audet. The compositing was run within the coded horizon fields to ensure that no composite intervals crossed any lithological or grade boundaries. To allow for uneven sample lengths within each of the horizons, the Datamine composite process was run using the variable sample length method. This adjusts the sample intervals, where necessary, to ensure all samples are included in the composite file (i.e. no residuals) while keeping the sample interval as close to the desired sample interval as possible.

The compositing process was checked by:

- Comparing the lists of horizon domain values in the raw and composite files, which matched.
- Comparing the sample length statistics in the raw and composite files. The two total length values matched and the mean composite interval was 1 m.

14.3 Exploratory data analysis - summary statistics

Basic statistical parameters for elements and oxides (as % grade) for each area are provided in Table 14.3 to Table 14.9. An assessment of the Coefficient of Variation (CV – ratio of the standard deviation to the mean) parameter resulted in the decision to cap selected constituents (CaO, MgO, Co) during grade estimation for some horizons. The top-cut values are provided in Table 14.10.

Table 14.3 Grade characteristics for VOW

Horizon	Constituent	Number of Composites	Minimum (%)	Maximum (%)	Mean (%)	Variance	CV
100	Al_2O_3	743	1.06	21.73	10.52	16.51	0.39
100	CaO	743	0.00	0.25	0.02	0.00	1.32
100	Co	743	0.00	0.61	0.10	0.01	0.81
100	Cr_2O_3	743	0.04	6.92	2.48	1.39	0.47
100	Fe	743	2.82	52.52	33.55	106.37	0.31
100	MgO	743	0.05	30.00	1.48	7.08	1.80
100	MnO	743	0.01	4.42	0.78	0.34	0.75
100	Ni	743	0.01	2.64	0.71	0.27	0.74
100	SiO_2	743	3.76	93.11	25.17	275.13	0.66
200	Al_2O_3	429	0.05	24.49	4.64	19.16	0.94
200	CaO	429	0.00	3.88	0.14	0.19	3.17
200	Co	429	0.00	0.18	0.05	0.00	0.66
200	Cr_2O_3	429	0.07	6.27	1.24	0.60	0.62
200	Fe	429	1.68	46.95	17.38	50.14	0.41
200	MgO	429	0.05	36.48	10.80	53.48	0.68
200	MnO	429	0.02	7.28	0.44	0.31	1.26
200	Ni	429	0.01	3.07	1.06	0.32	0.54
200	SiO_2	429	11.05	94.00	47.21	185.41	0.29
300	Al_2O_3	893	0.05	20.18	2.14	5.34	1.08
300	CaO	893	0.00	10.37	0.12	0.48	5.59
300	Co	893	0.00	0.15	0.03	0.00	0.57
300	Cr_2O_3	893	0.03	4.40	0.76	0.18	0.56
300	Fe	893	0.88	35.96	9.99	15.90	0.40
300	MgO	893	0.11	38.43	26.02	84.80	0.35
300	MnO	893	0.02	1.88	0.23	0.03	0.78
300	Ni	893	0.02	3.29	0.77	0.15	0.50
300	SiO_2	893	29.89	96.20	45.61	150.69	0.27

Table 14.4 Grade characteristics for VOI

Horizon	Constituent	Number of Composites	Minimum (%)	Maximum (%)	Mean (%)	Variance	CV
100	Al_2O_3	1731	1.53	27.51	13.03	22.14	0.36
100	CaO	1731	0.00	1.71	0.01	0.00	3.54
100	Co	1731	0.00	1.12	0.09	0.01	1.21
100	Cr_2O_3	1731	0.22	5.61	2.03	0.80	0.44
100	Fe	1731	5.85	55.47	37.02	73.22	0.23
100	MgO	1731	0.05	34.60	0.73	6.60	3.51
100	MnO	1731	0.02	8.16	0.94	0.76	0.93
100	Ni	1731	0.01	3.64	0.40	0.13	0.89
100	SiO_2	1731	3.08	73.68	18.55	108.97	0.56
200	Al_2O_3	505	0.82	25.10	5.54	15.35	0.71
200	CaO	505	0.00	0.68	0.07	0.01	1.35
200	Co	505	0.01	0.33	0.05	0.00	0.68
200	Cr_2O_3	505	0.23	3.89	1.03	0.26	0.49
200	Fe	505	5.59	53.08	20.35	66.12	0.40
200	MgO	505	0.05	30.90	10.88	61.76	0.72
200	MnO	505	0.04	4.55	0.55	0.21	0.83
200	Ni	505	0.07	5.24	1.00	0.36	0.60
200	SiO_2	505	9.36	75.30	42.62	113.41	0.25
300	Al_2O_3	1507	0.17	27.30	3.17	10.02	1.00
300	CaO	1507	0.00	18.48	0.20	1.52	6.09
300	Co	1507	0.00	0.14	0.03	0.00	0.54
300	Cr_2O_3	1507	0.02	2.84	0.69	0.08	0.43
300	Fe	1507	2.92	40.28	11.07	19.37	0.40
300	MgO	1507	0.05	37.48	25.45	74.93	0.34
300	MnO	1507	0.03	2.82	0.26	0.03	0.69
300	Ni	1507	0.01	4.80	0.92	0.25	0.55
300	SiO_2	1507	15.41	89.93	42.80	61.58	0.18

Table 14.5 Grade characteristics for VOE

Horizon	Constituent	Number of Composites	Minimum (%)	Maximum (%)	Mean (%)	Variance	CV
100	Al_2O_3	932	1.83	30.60	14.62	29.88	0.37
100	CaO	932	0.00	3.84	0.04	0.02	3.90
100	Co	932	0.00	0.50	0.06	0.01	1.28
100	Cr_2O_3	932	0.01	5.67	1.40	0.95	0.70
100	Fe	932	3.03	57.92	28.84	161.21	0.44
100	MgO	932	0.05	22.13	0.69	2.53	2.29
100	MnO	932	0.00	7.91	0.57	0.35	1.05
100	Ni	932	0.01	2.70	0.28	0.15	1.38
100	SiO_2	932	3.94	82.50	30.07	289.67	0.57
200	Al_2O_3	464	1.19	26.38	5.15	18.33	0.83
200	CaO	464	0.00	1.04	0.17	0.03	0.98
200	Co	464	0.01	0.23	0.06	0.00	0.57
200	Cr_2O_3	464	0.02	10.94	1.25	0.63	0.63
200	Fe	464	4.28	53.68	19.58	51.38	0.37
200	MgO	464	0.26	30.26	8.99	44.34	0.74
200	MnO	464	0.03	3.62	0.45	0.12	0.75
200	Ni	464	0.12	2.35	1.15	0.27	0.45
200	SiO_2	464	9.70	78.74	44.73	96.03	0.22
300	Al_2O_3	883	0.25	20.29	3.31	13.18	1.10
300	CaO	883	0.00	7.84	0.18	0.44	3.72
300	Co	883	0.00	0.24	0.03	0.00	0.67
300	Cr_2O_3	883	0.04	5.70	0.75	0.16	0.53
300	Fe	883	3.07	44.35	10.78	16.20	0.37
300	MgO	883	0.05	36.99	24.04	83.38	0.38
300	MnO	883	0.01	1.42	0.24	0.03	0.70
300	Ni	883	0.11	3.36	0.96	0.19	0.45
300	SiO_2	883	11.28	82.50	43.98	50.58	0.16

Table 14.6 Grade characteristics for JAC

Horizon	Constituent	Number of Composites	Minimum (%)	Maximum (%)	Mean (%)	Variance	CV
100	Al_2O_3	280	0.47	21.88	11.38	19.97	0.39
100	CaO	280	0.00	0.40	0.04	0.00	1.36
100	Co	280	0.00	0.61	0.10	0.01	0.85
100	Cr_2O_3	280	0.18	5.28	2.02	0.77	0.44
100	Fe	280	3.76	56.42	30.11	122.15	0.37
100	MgO	280	0.05	18.14	1.03	3.76	1.88
100	MnO	280	0.02	7.53	0.88	0.73	0.97
100	Ni	280	0.07	4.04	0.55	0.24	0.89
100	SiO_2	280	1.29	1.90	1.49	0.04	0.13
200	Al_2O_3	104	0.11	17.70	4.42	11.77	0.78
200	CaO	104	0.00	0.68	0.13	0.02	1.05
200	Co	104	0.00	0.28	0.06	0.00	0.73
200	Cr_2O_3	104	0.06	5.52	1.35	0.67	0.60
200	Fe	104	1.78	49.52	21.43	89.88	0.44
200	MgO	104	0.86	24.42	11.82	44.28	0.56
200	MnO	104	0.05	2.19	0.63	0.18	0.68
200	Ni	104	0.06	3.39	1.31	0.40	0.49
200	SiO_2	104	11.70	93.50	39.67	199.39	0.36
300	Al_2O_3	452	0.12	19.40	2.40	6.82	1.09
300	CaO	452	0.00	0.53	0.10	0.01	0.93
300	Co	452	0.01	0.21	0.03	0.00	0.77
300	Cr_2O_3	452	0.07	2.92	0.68	0.11	0.48
300	Fe	452	1.48	35.74	10.60	21.45	0.44
300	MgO	452	0.10	37.10	23.32	119.35	0.47
300	MnO	452	0.03	2.31	0.28	0.05	0.79
300	Ni	452	0.03	3.55	0.89	0.35	0.66
300	SiO_2	452	18.70	96.32	47.45	203.26	0.30

Table 14.7 Grade characteristics for PQZ

Horizon	Constituent	Number of Composites	Minimum (%)	Maximum (%)	Mean (%)	Variance	CV
100	Al_2O_3	1604	1.54	28.64	15.62	25.34	0.32
100	CaO	1604	0.01	7.45	0.04	0.11	9.12
100	Co	1604	0.01	0.42	0.09	0.00	0.71
100	Cr_2O_3	1604	0.01	6.04	2.27	0.92	0.42
100	Fe	1604	3.11	58.47	34.06	93.47	0.28
100	MgO	1604	0.10	34.30	0.90	4.52	2.36
100	MnO	1604	0.02	4.79	0.93	0.35	0.64
100	Ni	1604	0.01	2.67	0.47	0.14	0.80
100	SiO_2	1604	2.44	83.03	18.87	164.52	0.68
200	Al_2O_3	675	0.82	28.40	6.01	19.27	0.73
200	CaO	675	0.01	0.47	0.08	0.01	1.11
200	Co	675	0.01	0.21	0.06	0.00	0.45
200	Cr_2O_3	675	0.21	3.44	1.37	0.25	0.37
200	Fe	675	6.77	45.25	18.83	38.00	0.33
200	MgO	675	0.10	29.88	10.49	38.43	0.59
200	MnO	675	0.04	1.40	0.44	0.04	0.47
200	Ni	675	0.14	4.23	1.65	0.36	0.36
200	SiO_2	675	12.30	79.09	41.70	90.92	0.23
300	Al_2O_3	1851	0.45	24.84	4.26	14.33	0.89
300	CaO	1851	0.01	3.07	0.09	0.04	2.32
300	Co	1851	0.01	0.16	0.03	0.00	0.54
300	Cr_2O_3	1851	0.05	3.22	0.88	0.14	0.42
300	Fe	1851	4.68	39.90	11.51	15.94	0.35
300	MgO	1851	0.52	37.92	22.43	70.44	0.37
300	MnO	1851	0.05	1.98	0.26	0.02	0.54
300	Ni	1851	0.16	5.78	1.16	0.38	0.53
300	SiO_2	1851	14.23	81.11	43.02	54.09	0.17

Table 14.8 Grade characteristics for PQW

Horizon	Constituent	Number of Composites	Minimum (%)	Maximum (%)	Mean (%)	Variance	CV
100	Al_2O_3	468	2.09	33.53	17.09	23.53	0.28
100	CaO	468	0.00	0.29	0.02	0.00	1.44
100	Co	468	0.00	0.57	0.04	0.00	1.19
100	Cr_2O_3	468	0.01	4.51	0.94	0.57	0.81
100	Fe	468	4.48	50.01	26.59	84.29	0.35
100	MgO	468	0.01	3.36	0.63	0.34	0.92
100	MnO	468	0.01	1.09	0.13	0.02	1.13
100	Ni	468	8.41	75.30	30.73	166.67	0.42
100	SiO_2	369	0.18	19.10	5.40	7.43	0.50
200	Al_2O_3	369	0.00	4.25	0.18	0.11	1.86
200	CaO	369	0.01	0.25	0.06	0.00	0.58
200	Co	369	0.02	4.14	1.39	0.23	0.34
200	Cr_2O_3	369	1.97	53.78	21.97	66.98	0.37
200	Fe	369	0.05	26.64	6.32	16.39	0.64
200	MgO	369	0.03	3.41	1.13	0.25	0.45
200	MnO	369	4.63	96.20	43.27	160.94	0.29
200	Ni	300	0.10	18.63	3.76	10.69	0.87
200	SiO_2	300	0.01	3.63	0.16	0.11	2.02
300	Al_2O_3	300	0.01	0.16	0.03	0.00	0.70
300	CaO	300	0.02	2.13	0.79	0.13	0.45
300	Co	300	2.13	28.60	10.74	12.97	0.34
300	Cr_2O_3	300	0.10	35.89	19.95	85.04	0.46
300	Fe	300	0.03	4.70	0.38	0.19	1.13
300	MgO	300	31.18	95.10	48.56	118.85	0.22
300	MnO	300	0.57	1.47	0.99	0.00	0.07
300	Ni	411	0.28	26.37	6.47	51.57	1.11
300	SiO_2	411	0.01	10.50	0.97	3.80	2.02

Table 14.9 Grade characteristics for BAI

Horizon	Constituent	Number of Composites	Minimum (%)	Maximum (%)	Mean (%)	Variance	CV
100	Al_2O_3	2408	2.21	28.94	12.62	15.31	0.31
100	CaO	2408	0.01	8.87	0.04	0.05	5.86
100	Co	2408	0.00	0.55	0.10	0.00	0.68
100	Cr_2O_3	2408	0.07	8.87	3.11	1.69	0.42
100	Fe	2408	3.96	53.29	37.05	51.36	0.19
100	MgO	2408	0.10	35.15	1.61	9.22	1.88
100	MnO	2408	0.05	6.12	0.86	0.22	0.55
100	Ni	2408	0.02	3.64	0.72	0.23	0.66
100	SiO_2	2408	2.78	77.47	16.21	96.04	0.60
200	Al_2O_3	857	0.71	22.20	5.30	14.69	0.72
200	CaO	857	0.01	2.76	0.08	0.03	2.22
200	Co	857	0.01	0.22	0.05	0.00	0.58
200	Cr_2O_3	857	0.12	6.09	1.36	0.51	0.52
200	Fe	857	4.17	52.40	18.33	41.89	0.35
200	MgO	857	0.54	37.60	14.42	46.35	0.47
200	MnO	857	0.04	1.71	0.43	0.04	0.48
200	Ni	857	0.21	5.30	1.30	0.45	0.52
200	SiO_2	857	5.10	80.60	40.49	90.66	0.24
300	Al_2O_3	1786	0.10	20.50	3.36	9.30	0.91
300	CaO	1786	0.01	13.50	0.14	0.43	4.67
300	Co	1786	0.00	0.20	0.03	0.00	0.55
300	Cr_2O_3	1786	0.03	5.23	0.86	0.16	0.47
300	Fe	1786	1.08	45.36	11.38	15.43	0.35
300	MgO	1786	1.16	37.92	24.34	55.82	0.31
300	MnO	1786	0.02	1.89	0.26	0.02	0.57
300	Ni	1786	0.06	4.08	0.89	0.22	0.53
300	SiO_2	1786	12.14	96.20	43.27	79.13	0.21

Table 14.10 Top-cuts applied during grade estimation

Area	Horizon	CaO Top-cut %	Number affected %	MgO Top-cut %	Number affected %	Co Top-cut %	Number affected %
VOW	100	0.14	0.50	10.00	1.50	-	-
VOW	200	0.50	3.30	-	-	-	-
VOW	300	0.35	2.80	-	-	-	-
VOI	100	0.15	1.00	7.20	1.90	-	-
VOI	300	0.70	1.90	-	-	-	-
VOE	100	0.36	0.01	9.00	0.10	-	-
VOE	300	0.80	2.70	-	-	-	-
JAC	100	0.31	0.40	8.00	1.10	-	-
PQZ	100	0.25	0.40	12.00	0.60	-	-
PQZ	300	1.00	1.10	-	-	-	-
PQW	100	0.14	1.50	4.00	-	0.30	0.20
PQW	200	0.50	2.20	-	-	-	-
PQW	300	1.00	1.30	-	-	-	-
BAI	100	0.30	0.40	15.00	0.50	-	-
BAI	200	1.20	0.50	-	-	-	-
BAI	300	4.00	0.60	-	-	-	-

14.4 Estimation

14.4.1 Variography

Variograms were generated to assess the grade continuity of the various constituents and as inputs to the Ordinary Kriging algorithm used to interpolate grades. Snowden SuperVisor v.8 software was used to generate and model the variograms.

Laterite deposits occur often as low-lying hills, with the laterite profile effectively following the profile of the hill and weathering fronts. This, together with variable thicknesses of individual horizons and vertical grade trends, results in undulating geometries which present issues for effective grade interpolation using traditional 3D methods. Snowden therefore elected to use the Datamine Unfold process to address the impact of the undulations on the modelling of variograms and the estimation of grades. Unfolding improves the grade estimation process as it transforms the sample coordinates to assist in preserving vertical grade trends. This allows variogram analysis and grade estimation to be carried out using the pre-folding coordinates, which are then converted back to the folded (local) coordinate system. The unfolding process results in more samples being available for variogram modelling and grade estimation than would have been the case if standard resource estimation methods were used.

Variogram models

Variograms for unfolded nickel (Ni), cobalt (Co), iron (Fe) and oxide constituents (Al_2O_3, CaO, Cr_2O_3, MgO, MnO, SiO_2) were developed for each horizon and area, provided the data density was sufficient to support robust variograms. In the case of JAC variograms were adopted from the adjacent VOW deposit, with the major direction of continuity adjusted to 050°. In the case of PQW, variograms were adopted from the adjacent PQZ deposit, with similar changes to the directions of continuity.

All variograms were modelled using the following general approach:

- The drillhole composites were unfolded and modelled using the unfolded coordinate fields.
- All variograms were standardised to a sill of one.
- Variograms were modelled using spherical variograms with a nugget effect and three structures. Snowden found that most nuggets derived from the down-hole direction were exceedingly low which is expected in the nickel laterite environment.
- The variograms were evaluated using normal scores variograms rather than traditional variograms. This method produces a clearer image of the ranges of continuity in skewed data sets. The nugget and sill values were then back transformed to traditional variograms using the discrete Gaussian polynomials technique (Guibal et al, 1987).

14.4.2 Estimation method and parameters

Datamine software was used to unfold the composite data and estimate grades using ordinary kriging. Grades were estimated using variogram models for each attribute grouped by Horizon. Hard boundary conditions were used to preserve the chemistry of each Horizon.

14.4.3 Search parameters

For each area, the same search ellipse ranges and axis rotations were used with each of the grade estimates in order to keep the ratios of the various constituents (metal balance) as consistent as possible. The search ellipse axis lengths were derived from the variogram modelling.

14.4.4 Search ellipse strategy

The distribution and density of the various attribute values within each of the domains is quite variable in areas around the edges of the mineralisation and for the Transition Horizon which is often thin and highly variable thickness. As such if a single search ellipse was applied for the estimation process then a significant proportion of cells within the interpreted Horizons would not get informed for all of the grade fields. To ensure that each cell within the Horizons includes an estimated grade value, a dynamic search volume approach was used:

14.4.5 Estimation settings summary

The key search ellipse and estimation parameters are summarised in Table 14.11.

Table 14.11 Estimation parameters

Estimation setting	Description/setting
Final model names	vow050713v1.dm; voi050713v1.dm; voe270613v1.dm; jac050713v1.dm; pq060613v1.dm; pqw230513v1.dm; ba140613v1.dm
Drillholes	Unfolded and coded drilling data in Datamine format with top cuts applied for selected variables
Boundary conditions	Hard Horizon boundaries for all estimates
Top cuts	Per Ross (2013)
Search ellipsoid	Per Ross (2013)
Method	Ordinary kriging (parent cell estimation) with unfolding)
Variograms	Per Ross (2013)
Dynamic search volumes	Yes
Minimum number of samples – volume 1	5
Maximum number of samples – volume 1	30
Search volume 2 factor	1
Minimum number of samples – volume 2	2
Maximum number of samples – volume 2	30
Search volume 3 factor	2
Minimum number of samples – volume 3	1
Maximum number of samples – volume 3	30
Octant searching	No
Block discretisation (XYZ)	8 by 8 by 1

14.4.6 Model validation

The estimates were validated using:

- A visual comparison of the block grade estimates and the drillhole composite data.
- Generation of vertical section and plan-view plots of the estimates, and naïve composite grades, along with the number of composite samples available (slice plots). Due to the regular drill spacing the naïve composite grades are effectively declustered.
- A global comparison of the average composite (naïve) and estimated grades.
- A comparison of the correlations between constituents within the input composite data and the block model grade estimates.

The conclusions from the model validation work are:

- Inspection of the slice plots shows, for regions where there are substantive input composite numbers, good agreement in grade trends.
- Visual comparison of the model grades and the corresponding drillhole composite grades shows a good outcome (< 10 % difference) for almost all constituents. The exceptions are: VOE 100 for CaO; PQZ 100 for MgO; BAI 100 for CaO, MgO, 200 for CaO, Ni. Of these Ni is within 11 % difference which is not considered material.
- The estimated models adequately preserve the correlations observed in the input statistics.

14.5 Calculation of dry density

A combination of HZM and Teck density measurements, now totalling approximately 10,000 representative samples from each of the major laterite facies is summarised in Table 11.1.

Snowden investigated the relationship between measured density and major chemistry of the density samples and derived formulae to allow block dry density to be calculated from major chemistry block estimates. Samples were sorted by Horizon and Excel's LINEST (multiple linear regression) formula was then applied to create an equation for each Horizon. The regression prediction was checked by calculating the regressed density values from actual assays: checking that the average calculated results were equal to the average of the density measurements; and comparing the predicted values with the actual density measurements by scatterplots.

14.6 Mineral Resource classification

JORC guidelines

Table 14.12 and Table 14.13 list Snowden's assessment of the criteria that were considered when classifying the 2013 Araguaia PFS resource estimates in accordance with the JORC Code (2012 edition) guidelines.

Table 14.12 JORC Code (2012) Table 1 Section 1 – Sampling techniques and data

Item	Comments
Sampling techniques	The data used for resource estimation is based on the logging and sampling of diamond core drilling (100% of the sample data).
Drilling techniques	The drilling was completed using vertical core holes. Vertical drillholes are appropriate given the strike and dip of the mineralisation.
Drill sample recovery	HZM has required the drilling contractor to redrill where recoveries were less than 85 %, thus ensuring the recoveries in the provided database are adequate.
Logging	Almost all of the geological information has been obtained by the logging of drill samples, and supplemented by surface geological mapping and interpretation of geophysical surveys. Logging of drillhole samples was done with sufficient detail to meet the requirements of resource estimation and mining studies, and in accordance with HZM Standard Operating Procedures.
Sub-sampling	Cores were sampled at 1 m intervals. Half split core samples are crushed and pulverised at SGS laboratory in Goiania and the resultant pulps analysed at SGS laboratory in Belo Horizonte using tetraborate fusion X-Ray Fluorescence. Full QA/QC procedures are implemented, including the insertion of standards, duplicates and blanks.
Quality of assay data and laboratory tests	Snowden's analysis of the QAQC data (standards, blanks, duplicates) and assessment made by HZM did not identify any significant issues which could be material to the resource estimate.
Verification of sampling and assaying	Drilling from earlier phases was verified by independent Qualified Persons. In 2013 HZM dispatched 457 duplicate pulp samples to the ACME Laboratory in Canada for umpire check analyses. Original assay was completed by SGS in Brazil and the same analysis method (tetraborate fusion XRF) was applied at both laboratories. A reasonable level of accuracy has been demonstrated.
Location of data points	The project area is centred about the following co-ordinates: WGS 84 Latitude 07° 54' 9.0" South; UTM SAD 69 22S 9126200mN; and WGS 84 Longitude 49° 26' 1.8" West; UTM SAD 69 22S 672700mE. Collar locations were surveyed using a DGPS (precision +/- 10 cm) by Independent Licenced Surveyor. Elevation differences between drillhole collars and the supplied topography DTM were checked and eliminated by pressing the collars to the DTM.
	No downhole surveys were collected for the drilling. Mitigating this issue to a large extent is the fact that most of the drilling consists of shallow vertical core holes and the drill rig alignment is checked by HZM staff prior to drilling.
Data spacing and distribution	Drilling was completed along a set of oriented sections. The drillhole spacing is essentially 100 m apart with two small areas on areas PQZ and BAI drilled to 25 m spacing for variogram analysis. These drillhole spacings are sufficient to establish the degree of geological and grade continuity necessary to support the resource classifications that were applied.
	The drilling was composited downhole using a 1.0 m interval which corresponds to the dominant assay interval.
Orientation of data in relation to geological structure	The location and orientation of the Araguaia drilling (vertical) is appropriate given the geometry and orientation (horizontal) of the laterite mineralisation.
Sample security	All sampling and data collection is handled by HZM personnel and the drill core is subsequently transferred into core boxes. Drill core is stored in a secure facility in Conceição do Araguaia. Sample security procedures are provided in section 11.3. Pulp and crush rejects are returned after a 90 day period at SGS, pulp rejects are stored in cardboard boxes and crush rejects in large plastic boxes sequentially batch by batch also onsite.
Audits and reviews	The drilling database was reviewed by Snowden and sufficient cross-checks with assay certificates, drill core and logging, collar surveys were undertaken to confirm that the data is suitable for use in mineral resource estimation.

Table 14.13 JORC Code (2012) Table 1 Section 3 – Estimation and reporting of Mineral Resources

Item	Comments
Database integrity	The new Phase 3 drilling data was supplied to Snowden in Microsoft Excel spread sheets and then imported into an existing GEMS Project database by Snowden. Internal validation checks were made by Snowden and any discrepancies were corrected in consultation with HZM.
Geological interpretation	Snowden believes that the local geology is well understood as a result of work undertaken by HZM and Dr Marc-Antoine Audet in respect of chemical classification of rock types. The contacts between laterite horizons have been interpreted based on a combination of logging and geochemistry as described in Section 6. Alternative interpretations of the mineralisation are unlikely to significantly change the overall volume of the Horizons.
Dimensions	The Araguaia mineralisation estimated by Snowden consists of seven areas. Descriptions of the deposits are provided in Section 4.3. Maximum and average thickness of the laterite horizons are provided in section 7.
Estimation and modelling techniques	Unfolded ordinary block kriging using hard boundary domains, with sub-celling to accurately reflect horizon contacts. The deposits have been estimated previously by Dr Marc-Antoine Audet in GEMS software using an unwrinkling approach.
Moisture	All tonnages have been estimated as dry tonnages.
Cut-off parameters	The nickel mineralisation was reported above a 0.95% nickel cut-off grade.
Mining factors and assumptions	It is assumed the deposits will be mined using open cut methods.
Metallurgical factors and Assumptions	None. Metallurgical test work reported in Section 13 indicates there is a reasonable prospect for metal recovery using current technologies.
Environmental factors or assumptions	These are discussed in Section 20 “Environmental Studies, Permitting, and Social or Community Impact.
Density	There are sufficient bulk density measurements (water displacement method) to relate major chemistry to density by linear regression. Block estimates of dry density were calculated from block grade estimates..
Classification	The resources have been classified based on continuity of both the geology and the nickel grade along with the drillhole spacing. Additionally the information summarised in this table has been used to support the resource classification categories of Indicated and Inferred..
Audits and reviews	The Snowden models compiled in 2013 have not been independently reviewed in detail but have been discussed with HZM’s Technical Advisor Mr F R Billington
Accuracy and confidence	The resource was classified by taking into consideration the confidence in the continuity of nickel grades and the confidence in the geological interpretation.

14.6.1 Mineral Resource classification scheme

The Phase 3 Mineral Resource estimates were classified and reported in accordance with the 2012 Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves (the JORC Code). The resource is reported above a 0.95% nickel cut-off grade to enable comparison with estimates reported at the completion of Phase 2 drilling (Audet M.A. et al., 2012). A nickel cut-off grade of 0.9% is supported by economic analysis by Snowden.

Reasonable prospects for metal recovery by current technologies were reported by HZM:

- In Q1 2013 HZM reported the completion of an 18 month comprehensive metallurgical test programme, which included work by FLSmidth, the global leader in high temperature kiln technology, Xstrata Process Support ('XPS') and Kingston Process Metallurgy ('KPM'). The test programme was designed to evaluate the expected ore performance in RKEF processing, and HZM were pleased to report that Araguaia ore was found to be suitable for treatment using this proven technology. Rotary kiln processing is a key step in the well-established RKEF pyro-metallurgical process that HZM are aiming to utilise at Araguaia. Smelting tests carried out by XPS on a number of ore blends showed that smelting Araguaia laterite can produce Fe-Ni alloy and a low nickel slag. This work and additional testing by KPM confirmed the electric furnace conditions when producing a 15 to 20% Ni grade of Fe-Ni, and further confirmed the suitability of the RKEF process for producing a marketable grade of Fe-Ni.

The resource classification criteria were developed based on an assessment of the following items:

- Nature and quality of the drilling and sampling.
- Drilling density.
- Confidence in the understanding of the underlying geological and nickel grade continuity.
- Analysis of the QAQC data. Confidence in the estimate of the mineralised volume.
- The results of the model validation.
- Table 1 Section 1 and 3 of the JORC code.

The resource classification scheme adopted by Snowden for the 2013 Araguaia mineral resource estimate was based on the following:

- Mineralisation was classified as Indicated where the drilling density was 100 mE by 100 mN (or less).
- Mineralisation delineated using a drilling density larger than 100 mE by 100 mN and up to about 150 m spacing was classified as Inferred.
- Mineralisation delineated using sparse spacings was not classified.

14.6.2 PFS Mineral Resource reporting

The classified Mineral Resources for the PFS have been reported using a 0.95% nickel cut-off grade and are provided in Table 14.14.

Table 14.14 PFS Mineral Resource estimates reported at 0.95% Ni cut-off

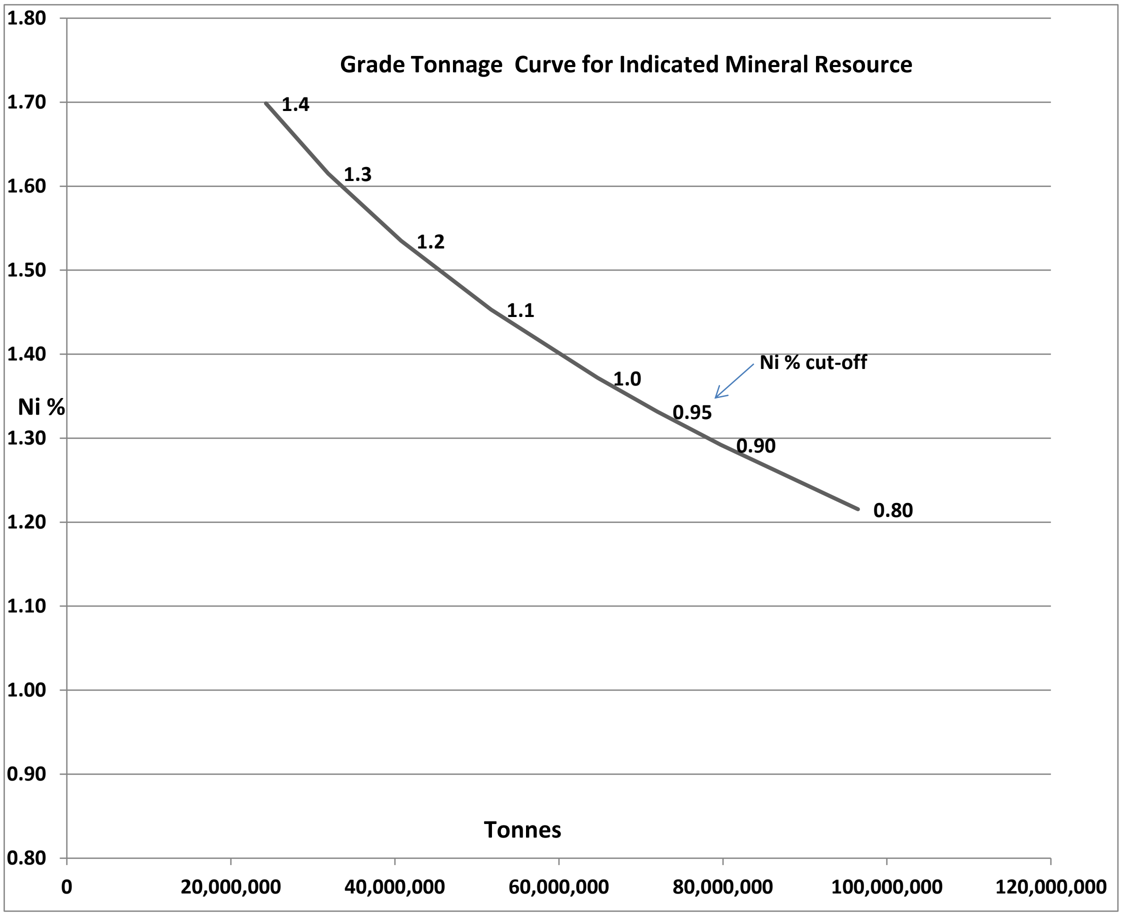
PFS Area	Category	Material type	Tonnage (kT)	Density (t/m3)	Contained Ni metal (t)	Ni (%)	Co (%)	Fe (%)	MgO (%)	SiO_2 (%)	Al_2O_3 (%)	Cr_2O_3 (%)
VOW	Indicated	Limonite	2,490	1.34	31,150	1.25	0.121	35.15	2.42	23.21	8.06	2.67
VOW	Indicated	Transition	2,740	1.17	37,180	1.36	0.050	19.27	12.01	42.80	3.89	1.44
VOW	Indicated	Saprolite	1,870	1.27	21,430	1.15	0.033	12.23	23.28	42.58	2.97	0.97
VOW	Inferred	Limonite	590	1.36	7,160	1.21	0.128	37.64	2.14	20.50	7.98	2.89
VOW	Inferred	Transition	910	1.17	11,580	1.27	0.057	18.94	11.73	43.75	3.72	1.43
VOW	Inferred	Saprolite	800	1.33	8,760	1.09	0.030	11.52	26.88	40.77	2.64	0.92
VOI	Indicated	Limonite	1,230	1.33	14,710	1.20	0.191	38.28	2.50	19.61	7.87	2.08
VOI	Indicated	Transition	2,880	1.21	40,160	1.39	0.060	20.06	12.26	40.85	4.77	1.18
VOI	Indicated	Saprolite	8,050	1.34	104,930	1.30	0.032	11.88	25.50	41.13	3.09	0.74
VOI	Inferred	Limonite	240	1.32	2,780	1.14	0.114	33.40	2.74	28.09	7.26	1.94
VOI	Inferred	Transition	490	1.18	6,370	1.30	0.054	20.81	10.96	41.19	4.55	1.24
VOI	Inferred	Saprolite	810	1.37	10,160	1.25	0.032	11.80	24.18	42.43	3.82	0.73
VOE	Indicated	Limonite	510	1.31	6,130	1.20	0.156	37.21	2.01	22.99	7.70	2.14
VOE	Indicated	Transition	3,160	1.16	43,250	1.37	0.062	20.67	10.05	42.44	4.34	1.37
VOE	Inferred	Saprolite	310	1.30	3,490	1.13	0.035	12.49	17.36	45.24	5.80	0.72
JAC	Indicated	Limonite	480	1.30	5,850	1.23	0.134	37.03	2.40	22.59	7.80	2.22
JAC	Indicated	Transition	870	1.14	12,340	1.42	0.071	23.19	12.14	36.77	4.03	1.47
JAC	Indicated	Saprolite	1,710	1.36	23,800	1.39	0.046	11.75	23.27	45.00	2.63	0.78
JAC	Inferred	Limonite	40	1.37	450	1.09	0.118	36.07	2.26	23.91	7.30	2.08
JAC	Inferred	Transition	80	1.19	1,230	1.44	0.064	18.63	12.91	41.63	4.39	1.20
JAC	Inferred	Saprolite	400	1.30	5,650	1.42	0.042	11.75	21.54	45.31	2.62	0.80
PQZ	Indicated	Limonite	1,440	1.34	16,730	1.17	0.067	35.21	2.61	19.06	11.63	2.28
PQZ	Indicated	Transition	5,540	1.19	94,060	1.7	0.035	19.12	10.9	40.62	6.1	1.36
PQZ	Indicated	Saprolite	12,510	1.32	173,530	1.39	0.095	11.68	22.68	42.1	4.45	0.89
PQZ	Inferred	Limonite	250	1.37	2,630	1.06	0.069	33.94	3.71	18.7	12.1	2.35
PQZ	Inferred	Transition	420	1.22	6,160	1.46	0.034	21.17	13.71	34.83	6.83	1.53

PFS Area	Category	Material type	Tonnage (kT)	Density (t/m3)	Contained Ni metal (t)	Ni (%)	Co (%)	Fe (%)	MgO (%)	SiO_2 (%)	Al_2O_3 (%)	Cr_2O_3 (%)
PQZ	Inferred	Saprolite	1,110	1.43	13,540	1.22	N/A	10.99	27.06	39.8	4.68	0.82
PQW	Indicated	Limonite	N/A	N/A	N/A	N/A	0.058	N/A	N/A	N/A	N/A	N/A
PQW	Indicated	Transition	2,730	1.14	32,930	1.21	0.037	21.79	6.64	43.19	5.47	1.41
PQW	Indicated	Saprolite	670	1.34	7,840	1.18	N/A	11.5	19.99	46.49	3.99	0.84
PQW	Inferred	Limonite	N/A	N/A	N/A	N/A	0.064	N/A	N/A	N/A	N/A	N/A
PQW	Inferred	Transition	310	1.12	3,410	1.1	0.052	22.75	6.55	41.49	5.2	1.34
PQW	Inferred	Saprolite	40	1.29	440	1.05	0.113	12.73	18.6	45.14	4.67	0.79
BAI	Indicated	Limonite	5,420	1.37	63,210	1.17	0.057	36.94	3.12	17.17	10.25	2.86
BAI	Indicated	Transition	6,190	1.22	87,000	1.41	0.031	18.99	13.72	40.24	5.07	1.44
BAI	Indicated	Saprolite	6,340	1.31	77,990	1.23	0.083	12.33	23.37	42.55	3.46	0.93
BAI	Inferred	Limonite	770	1.35	8,350	1.08	0.049	32.62	4.7	23.37	8.97	2.66
BAI	Inferred	Transition	270	1.18	3,130	1.15	0.029	24.87	11.83	33.02	6.05	1.63
BAI	Inferred	Saprolite	150	1.29	1,650	1.11	0	12.67	24.14	39.2	4.85	1
Total	Indicated	All	72,000	1.28	958,700	1.33	0.058	18.48	16.19	38.25	5.04	1.31
Total	Inferred	All	8,400	1.21	102,400	1.21	0.058	19.86	15.11	37.25	5.32	1.41

Note: Mineral Resources are inclusive of Mineral Reserves. Totals may not add due to rounding

Figure 14.1 provides a grade-tonnage curve for a range of nickel cut-offs for the Indicated Mineral Resources.

Figure 14.1 Grade tonnage curve for Indicated Mineral resource



14.7 Other deposits within the project area

Mineral Resources were estimated by Dr. Marc-Antoine Audet using block estimation by Inverse Distance at the power of 2 (ID2) interpolation methodologies on 25 × 25 × 2 m blocks (Audet et al, 2012). The deposits are: Pequizeiro NW (PQNW), Oito Main (Oito), Lontra North (Lontra 1 – 4 or Northern) and Raimundo. The locations of these deposits are shown in Section 7 and Section 10.

A geochemical correlation matrix was defined in order to assign a 'GeoFacies' to each individual sample in the database. Bulk density values (wet and dry) and moisture content were assigned based on facies.

Three-dimensional models these deposits were created using surveyed holes. The models integrate the concept of geological horizons (limonite, transition and saprolite) to create a 3D block model. For each deposit, a surface geological constraining envelope was generated using drillhole data as well as information from geological mapping.

The estimates were previously reported in Audet, M A, et al (2012) and are classified as Inferred Mineral Resources (Table 14.15). These resources are not considered in the Pre-Feasibility Study discussed in this Technical Report.

There are no Mineral Resource estimates for other prospects (Morro, Southern, Oito West and Pequizeiro East) due to insufficient drill sample information.

Table 14.15 Non-PFS Mineral Resource estimates reported at 0.95% Ni cut-off

Area (Non-PFS)	Category	Material type	Tonnage (kT)	Density (t/m3)	Contained Ni metal (t)	Ni (%)	Co (%)	Fe (%)	MgO (%)	SiO_2 (%)	Al_2O_3 (%)	Cr_2O_3 (%)
PQNW	Inferred	Limonite	1,000	1.35	11,060	1.11	0.083	35.72	4.88	18.27	9.74	2.47
PQNW	Inferred	Transition	700	1.34	9,220	1.32	0.050	21.19	14.00	33.46	7.05	1.58
PQNW	Inferred	Saprolite	320	1.55	3,480	1.09	0.035	14.15	23.58	38.83	3.48	1.07
Oito	Inferred	Limonite	2,860	1.34	31,840	1.11	0.123	37.69	2.67	18.20	10.51	2.32
Oito	Inferred	Transition	3,470	1.35	47,820	1.38	0.051	19.41	13.90	40.28	5.09	1.24
Oito	Inferred	Saprolite	2,890	1.46	33,210	1.15	0.031	12.11	22.68	42.35	4.63	0.76
Lontra 1	Inferred	Limonite	510	1.32	5,680	1.11	0.055	31.10	6.85	25.54	7.69	0.47
Lontra 1	Inferred	Transition	40	1.27	350	1.00	0.04	23.46	17.38	31.48	3.64	0.75
Lontra 1	Inferred	Saprolite	-	-	-	-	-	-	-	-	-	-
Lontra 2	Inferred	Limonite	360	1.33	4,310	1.21	0.074	38.52	3.18	14.22	8.83	3.18
Lontra 2	Inferred	Transition	100	1.31	1,100	1.11	0.038	19.29	20.66	29.65	4.19	4.54
Lontra 2	Inferred	Saprolite	20	1.47	170	1.13	0.03	14.34	25.87	35.11	3.61	1.23
Lontra 3	Inferred	Limonite	930	1.33	10,910	1.18	0.082	34.93	4.65	19.69	9.34	0.15
Lontra 3	Inferred	Transition	670	1.28	8,240	1.23	0.054	21.20	18.23	29.82	5.95	0.64
Lontra 3	Inferred	Saprolite	20	1.44	210	1.06	0.042	13.79	26.09	34.74	3.34	0.85
Lontra 4	Inferred	Limonite	240	1.32	2,920	1.23	0.082	38.15	5.27	16.56	7.60	0.12
Lontra 4	Inferred	Transition	160	1.30	2,070	1.28	0.054	19.93	20.04	30.92	5.56	0.66
Lontra 4	Inferred	Saprolite	230	1.47	2,680	1.19	0.045	14.96	26.24	33.63	3.32	0.66
Raimundo	Inferred	Limonite	1,030	1.32	12,140	1.18	0.072	35.07	5.29	20.3	9.28	0.19
Raimundo	Inferred	Transition	1,310	1.28	16,000	1.23	0.046	21.47	17.6	31.4	5.84	0.57
Raimundo	Inferred	Saprolite	90	1.35	940	1.08	0.034	15.16	26.33	34.39	3.59	0.67
Total	Inferred	All	16,900	1.36	204,360	1.21	0,065	25.16	12.38	30.23	6.99	1.23

15 Mineral Reserve estimates

Mineral Reserves, which are a subset of the Mineral Resources described in Section 14 have been prepared for the Project as part of the PFS. The Reserves use the Base Case assumptions, designs and parameters defined predominantly in Section 16 and also other relevant sections of this report.

In accordance with the CIM Definition Standards on Mineral Resources and Mineral Reserves (as adopted and amended), Mineral Reserves are classified as either “probable” or “proven” Mineral Reserves and are based on Indicated and Measured Mineral Resources only. No Mineral Reserves have been estimated using Inferred Mineral Resources

15.1 Summary

The estimation of Mineral Reserves used the recently completed estimate of Indicated Resources for the Project as reported in section 14 of this report.

A Mineral Reserve of 21,204 kt (dry) at an average grade of 1.66% Ni was estimated. The detailed breakdown of the Mineral Reserve by deposit is presented in Table 15.1

Table 15.1 March 2014 Mineral Reserve estimate

Class	Deposit	Ore Dry Mass (kt)	Ni (%)	Fe (%)	Al_2O_3 (%)	SiO_2/MgO
Probable	Baião	3,520	1.67	17.41	4.58	2.56
Probable	Pequizeiro	9,300	1.70	15.58	5.39	2.56
Probable	Pequizeiro West	380	1.57	20.38	4.63	4.29
Probable	Jacutinga	960	1.81	15.13	2.96	2.11
Probable	Vila Oito East	2,450	1.55	15.97	3.73	2.22
Probable	Vila Oito	3,580	1.63	14.61	3.63	2.05
Probable	Vila Oito West	1,020	1.59	19.35	4.25	3.32
	Total Probable	21,200	1.66	16.01	4.59	2.44
	Proven	NIL	N/A	N/A	N/A	N/A
	Total Proven and Probable	21,204	1.66	16.01	4.59	2.44

This Mineral Reserve is calculated on the basis of currently available information. Snowden strongly recommends a test pit(s) to assess in-situ grade reconciliation to the resource model, incidence of barren rocks in the saprolite, mining recovery and mining dilution.

15.2 Disclosure

Mineral Reserves reported in Section 15 were based on the PFS undertaken under the direct supervision of Mr Anthony Finch who is a Qualified Person as defined in NI 43-101, an employee of Snowden. Snowden is independent of HZM.

15.2.1 Known issues that materially affect mineral reserves

The Mineral Reserves could be affected by changes in metal price, capital and operating costs, metallurgical performance, infrastructure requirements, permitting or other factors. These factors are discussed in other sections of this report. The major risks to the Mineral Reserves are factors that either effect the costs to exploit resource or the revenues received for the products produced.

As discussed below a break-even grade for was calculated as 1.04% Ni given the commodity price assumption. The actual nickel cut-off grade used was 1.4% (for operational reasons), so this provides degree of robustness for the reserve estimation.

The metallurgical testwork has indicated that the minerals can be economically recovered using existing technology and methodology. Metallurgical performance has a direct effect on the revenue received and increase or decreases in performance will change the amount of metal recovered and hence the revenue received.

Permitting is not expected to be a material risk to the project as there have been no indications to date that there are any social, regulatory or community issues that cannot be managed through best practice operating standards and/or risk management planning and mitigation measures. Permitting remains a risk to the reserves until the granting of the mining licence as part of the outcomes of the feasibility studies and the successful submission of the permitting and licence to operate requirements that will be outcomes of the final social, environmental and community studies.

There are no perceived infrastructure risks to the reserve estimation process. The infrastructure is either existing or of a relatively standard type to install during construction of the project.

15.3 Assumptions, methods and parameters

15.3.1 Pit optimisation

The pit optimisation parameters used to develop the mineral reserves are shown in Table 15.2.

Table 15.2 Pit optimisation parameters

Parameter	Unit	Value
Slope angle	Degrees	35
Waste mining cost	USD/dmt	3.00
Ore mining cost	USD/dmt	9.00
Processing cost	USD/dmt feed	127.8
Processing NI recovery	%	93
Process feed grade constraints	Ni Minimum grade %	1.2
	Iron grade range %	15 to 16.5
	Al_2O_3 grade range %	4.0 to 5.5
	SiO_2/MgO ratio	2.2 and 2.6
Commodity price	USD/t of recovered Ni	15,000
Administration costs	USD/dmt of feed	11.20

Using the above assumptions, the marginal cut-off grade for the Project is calculated to be 1.04% Ni. However, this grade was not applied for mine planning due to the following reasons:

- The overall ore feed is off-specification for the plant.

In particular, the overall iron grade is too high. Therefore, the higher iron grade rock types (limonite and transition) had elevated cut-off grades applied to reduce their impact on the average iron grades whilst maintaining the highest possible grade bring the overall grade to within specification.

- The mine life is long and impacts negatively on net present value.

Applying a higher cut-off grade reduces the footprint of the pit, increases the average grade and avoids mining lower grade material that would not be processed for a number of years.

Through iteration Snowden determined that the appropriate cut-off grades to meet the grade specifications (including minimum nickel feed grade) were:

- Limonite – 1.69% Ni
- Transition – 1.4% Ni
- Saprolite – 1.4% Ni.

These cut-off grades were applied in the pit optimisation. For scheduling lower grade Saprolite (>1.1% Ni) was processed at the end of the mine life.

Prior to optimisation the resource model was re-blocked to account for dilution. A detailed discussion of the process and the outcomes of this re-blocking is presented in Section 16.6.

Using the re-blocked resource models, and the parameters detailed herein, a number of pit shells were generated, ranked by revenue, with factors between 30% and 200% of the base case revenue assumptions. From these shells Snowden selected the revenue factor 1 pit for the mining inventory calculation. A summary of the pit optimisation inventory is provided in Table 15.3.

Table 15.3 Pit optimisation results

Deposit	Pit Dry Mass (kt)	Waste Dry Mass (kt)	Strip Ratio (w:o)	Feed Dry Mass (kt)	Ni (%)	Fe (%)	Al_2O_3 (%)	SiO_2/MgO	Cash Flow (USD M)[2]
Baião	17,816	14,383	4.19	3,433	1.69	17.69	4.74	2.59	256.3
Pequizeiro	27,648	18,835	2.14	8,813	1.72	15.72	5.40	2.61	751.1
Pequizeiro West	2,350	1,944	4.80	405	1.58	20.13	4.66	4.29	23.3
Jacutinga	2,943	2,014	2.17	930	1.83	15.26	2.98	2.11	93.8
Vila Oito East	11,040	8,888	4.13	2,152	1.59	16.48	3.64	2.28	131
Vila Oito	16,356	13,195	4.17	3,161	1.68	15.05	3.65	2.12	235.0
Vila Oito West	5,523	4,537	4.60	3,161	1.61	19.57	4.27	3.41	62.2
Total	83,676	63,795	3.21	19,881	1.69	16.29	4.63	2.50	1,553

15.3.2 Pit design

The pit design used smoothed pit shells from the optimisation, altered for the removal of small satellite pits.

The reconciliation of volumes and value between the pit shell selected for design and the design itself is shown in Table 15.4. This reconciliation is deemed to be appropriate.

Table 15.4 Design inventory comparison

Item	Design	Pit Shell	Difference (%)
Total pit dry mass (kt)	81,166	83,676	(3.0%)
Waste dry mass (kt)	61,631	63,800	(3.4%)
Feed dry mass (kt)	19,583	19,881	(1.5%)
Ni (%)	1.69	1.69	(0.0%)
Strip ratio (w:o)	3.14	3.21	(2.1%)
Cash flow ($M)	1,542	2,955	(0.7%)

After the pit optimisation, additional lower grade saprolite material (1.1% Ni to 1.4% Ni) was included in the inventory to be processed at the end of the mine life. This increased the inventory to 21.2 Mt at 1.66% Ni and reduced the strip ratio to 2.83 (waste:ore). All inventories reported beyond this point include this lower grade material in the inventory.

The resultant designs and schedules are detailed in Section 16.

[2]*Cash flow is operating only, and is calculated based upon design assumptions and does not consider capital, interest or tax. This number should be seen as a point of comparison only and not a measure of absolute viability.*

15.3.3 Mineral Reserve classification

The classification categories of Probable and Proved Ore Reserve under the JORC Code are equivalent to the CIM categories of Probable and Proven Mineral Reserve (CIM, 2010).

Includes:

- nickel spot price of USD 19,000 / tonne flat for LOM
- iron spot price of USD 150 / tonne flat for LOM
- Fe-Ni ratio of 20.3% Ni and 79.7% Fe
- royalties for the combined Fe-Ni product is 2% of the costs (up to the furnace)
- selling costs - there are no selling costs as product is being sold at the mine gate.

16 Mining methods

The mine design and accompanying schedules, detailed herein, are based upon the Base Case of 0.9 Mtpa run of mine yielding 327.4 kt of Fe-Ni product.

An alternative mine plan and schedule was developed for a higher plant throughput case of 2.7 Mtpa yielding 50.65 Mt Fe-Ni product. This option is referred to in Section 22.

16.1 Geotechnical investigation - summary

The geotechnical data collection programme comprised:

- Geotechnical logging of diamond drill cores from the 12 drillholes from Baião, Pequizeiro Jacutinga and Oito (three areas) deposits and sampling for geotechnical laboratory testing.
- HMZ geologists conducted geotechnical logging of 386 m of HQ size diamond core according to the Snowden logging procedures. All the holes were drilled vertically and cored from ground surface with depths ranging from 20 m to 40 m.
- Soil particle size distribution and Atterberg limits and rock UCS tests were carried out at the Engesolo soils laboratory in Brazil as part of the geotechnical investigation programme.

16.1.1 Engineering geology

The engineering geology is complicated as a result of the varying proportions of limonite, transition and saprolite zones in the laterite profile. The thicknesses of these materials change rapidly both laterally and vertically across the deposit sites. The average thicknesses of the laterite profile in the proposed mining areas lie between 18.3 m to 27.7 m.

The inferred base of weathering lies below the maximum depth of the planned pits. The majority of the pit slopes will be developed within the laterite profile but occasionally the toe sections of deeper pit slopes may be developed through highly weathered rock mass.

16.1.2 Ground water

The regional water table generally lies within the saprolite horizon. Clay rich zones can be erratically developed in the saprolite (common in laterite profiles), and therefore there is potential to develop perched water tables above the regional water table, within the weathered profile during the rainy season.

Dewatering will significantly improve mining conditions including trafficability. Although total depressurisation of clay dominant material zones may not be possible, it is expected that substantial dewatering may be achievable by pumping from the highly transmissive weathered/fractured rock mass below the laterite profile.

16.1.3 Geotechnical model

Principal geotechnical domains have been identified and defined in terms of soil or rock mass characteristics. These domains are summarised in Table 16.1:

- Typically, residual lateritic clays have formed beneath a hard ferricrete cap. At depth, the weathering profile consists of rocky saprolite above the fresh ultramafic bedrock.
- The weathering profile above the fresh ultramafic rock mass is highly variable, consisting of a mixture of weak and harder material that can be classified into three broad geotechnical domains, namely an upper ferruginous zone, lateritic-clay zone and a basal ultramafic saprolite zone.
- The lateritic zone can be delineated in to three separate geotechnical domains, based on their grain size and index properties namely; a limonite domain consisting of Sandy Silty CLAY (classified as ML and MH on plasticity characteristics), transition domain consisting of Sandy Clayey SILT (MH) and saprolite domain consisting of Sandy Clayey SILT (MH). The latter two domains have increased amounts of smectite clays and active clays. They are expected to deteriorate under wet-dry cycles due to potential swelling and shrinking.

The rapid changes in material types and the large spacing between the diamond drillholes used to capture geotechnical information, result in increased uncertainties in the geotechnical model. Nevertheless, the confidence level of the geotechnical model is considered to be adequate for the current PFS.

Table 16.1 Summary of geotechnical domains

Geotechnical domain	Thickness	Strength	Description
Top soil & Ferricrete (hardcap)	Avg. 3 m	Soft to med. strong	Top soil: Reddish brown, loose to medium dense ferruginous gravelly sand with some silt and clay. Organic matter and plant roots present. Ferricrete: Reddish brown, loose to medium dense, mostly cemented, vuggy ferruginous gravelly sand with some silt and clay.
Limonite	Avg. 9.9 m *24 to 45 m	Very soft to stiff	Reddish brown to yellowish brown, soft to stiff, sandy clayey silt with some gravel. PI- avg.-18 (11 to 29); LL- avg.-56 (43 to 70); Classed ML and MH. Mostly inactive clays.
Transition	Avg. 5.1 m *18 to 59 m	Very soft to stiff	Light green, green and brown colour depending on smectite content. Soft to stiff, sometimes friable, Sandy Silty Clay. PI- avg.-45 (19 to 69); LL- avg.-107 (44 to 159); Classed MH. Mostly normal clays.
Saprolite (Earthy saprolite)	Avg. 8.2 m *28 to 59 m	Soft to stiff and very weak	Brown to reddish to green, mostly altered serpentine rock. High in smectite. Fine grained zones classified as sandy silty clay; classed MH by plasticity. PI- avg.-55 (23 to 70); LL- avg.-115 (52 to 150); mostly normal but significant amount of active clays.
Weathered and fractured rock mass (Rocky saprolite)	Variable	Very weak to weak	Highly fractured and weathered rock mass, consisting sections of extremely weathered rock. High alteration along fractures. Groundwater aquifer.
Fresh rock	variable	strong	Generally massive serpentinite

*- Maximum thickness range in different deposits

16.2 Slope stability

Potential modes of instability in the laterite and weathered ultramafic domains forming the interim and final slopes of both pits will depend on the scale of the slope. The most likely failure mode in the inter-ramp and/or overall slope scale is likely to be rotational sliding through the laterite and weak rock mass.

Slope recommendations

Deterministic stability analyses have been undertaken for rotational sliding for batter scale and overall slope scale. Sensitivity analyses were conducted for different groundwater conditions. At the current PFS level of study the slopes were designed for a target factor of safety of 1.2. The slope angles can be adjusted in the next phase of study when the shear strength parameters for different geotechnical domains are better defined.

Given the weak nature of the materials it is advisable to develop the pit slopes with short batters. Steeper batter face angles can be achieved with reduced batter heights. Depending on the choice of batter height for the optimisation of productivity and the prevailing groundwater conditions, following batter face angles are recommended for design. For the PFS designs, 80° and 55° batter face angles are recommended for batter heights of 5 m and 10 m respectively as presented in Table 16.2.

Overall design slope angles depend on the slope height and achievable depressurisation. It is expected that some depressurisation would occur adjacent to the slope hence 39° and 32° maximum overall slope angles are recommended for slope heights 20 m and 30 m respectively as presented in Table 16.3.

Table 16.2 Recommended batter configurations under different groundwater conditions

Batter configuration	Low PS		Mid PS		High PS	
Batter height	5m	10m	5m	10m	5m	10m
Face angle	80°	55°	65°	40°	55°	NA

Table 16.3 Recommended overall slope angles

Slope (°)	20 m high slope		30 m high slope	
	No drawdown	Some drawdown adjacent to slope	No drawdown	Some drawdown adjacent to slope
Design overall slope angle	33°	39°	29°	32°

16.2.1 Waste dump slope design

The waste dumps will consist predominantly of limonite material. The dump stability assessment indicates that a 19° overall slope over 30 m height will be stable under worst case conditions. However, due to the uncertainties associated with the adopted shear strength parameters, the overall slope angle used in the current design; 3H:1V (18.4°) is endorsed for a 30 m dump height.

16.2.2 Trafficability

The trafficability assessment indicates that sheeting will be required in the weaker laterite domains to improve trafficability. The recommended sheeting thicknesses depending on the type of trucks used are presented in Table 16.4.

Table 16.4 Required sheeting thicknesses

Sheeting thickness requirements								
Truck Type	Limonite		Transition		Earthy Saprolite		Rocky Saprolite	
	Case1 Sheeting thickness (m)	Case 2 Sheeting thickness (m)	Case1 Sheeting thickness (m)	Case 2 Sheeting thickness (m)	Case1 Sheeting thickness (m)	Case 2 Sheeting thickness (m)	Case1 Sheeting thickness (m)	Case2 Sheeting thickness (m)
Cat 735B	0.38	0.41	0.61	0.67	0.72	0.78	0.3	0.32
Cat 740B	0.4	0.44	0.64	0.7	0.75	0.82	0.31	0.34
Cat 772	0.55	0.6	0.88	0.97	1.03	1.13	0.43	0.47

Note: Case1 - 6 monthly major road maintenance, Case 2 - 12 monthly major road maintenance

The unconsolidated pisolitic ferricrete (PF) is considered to provide good sheeting material. It is recommended to stockpile this material for this purpose.

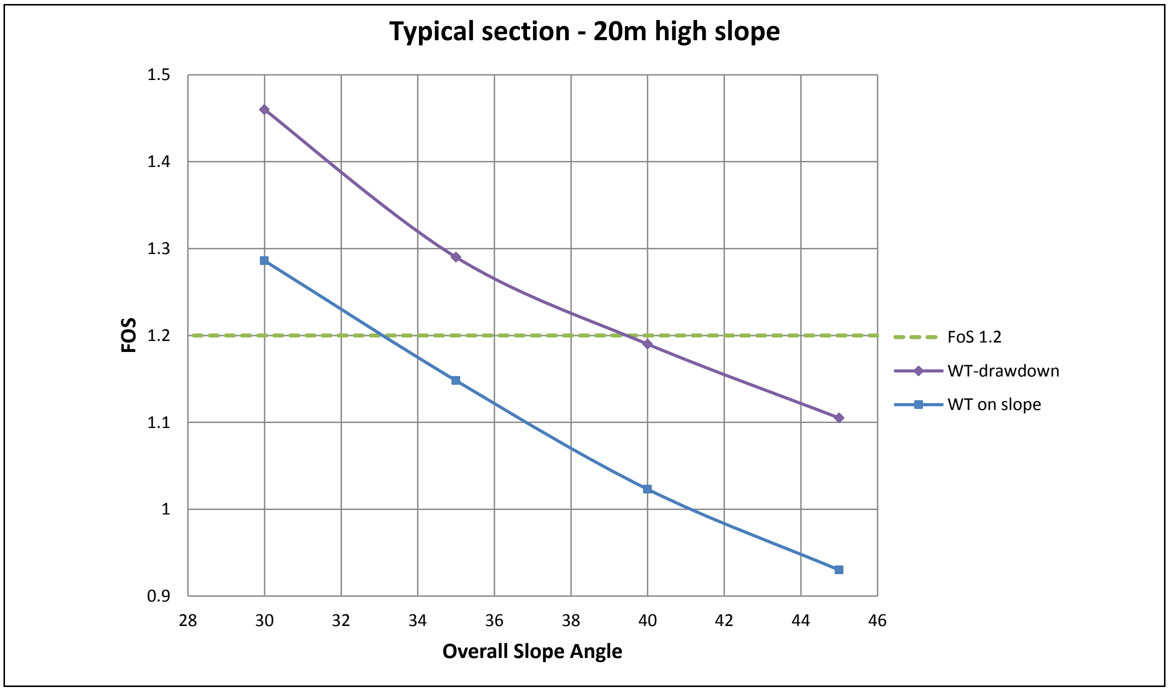
16.2.3 Geotechnical review of proposed pit design

Design checks for the slopes in each pit have been conducted using the "Factor of Safety – Overall slope angle" design charts developed using typical sections. To develop the design charts, sensitivity analyses have been conducted using two depressurisation scenarios:

- Scenario 1 – No water table drawdown occurs, piezometric line intersects and lies on the slope
- Scenario 2 – Some drawdown occurs adjacent to the slope.

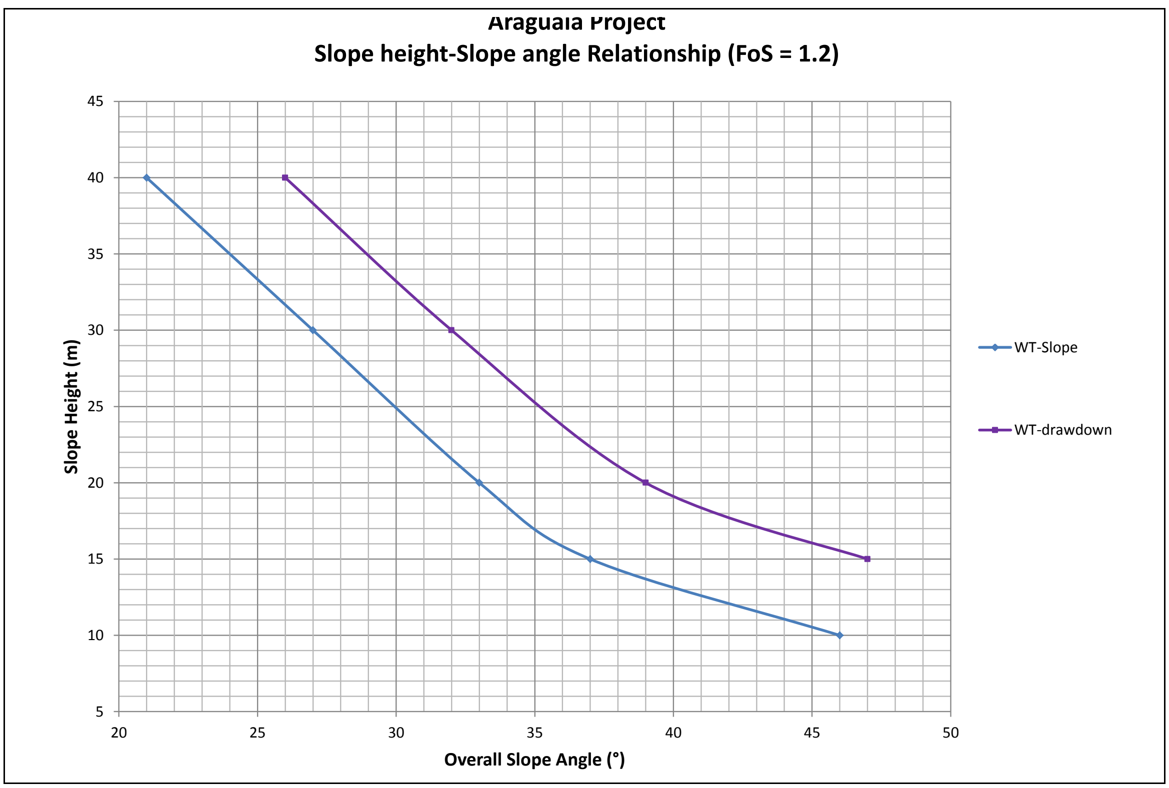
In Figure 16.1 the stability analyses results for a 20m high slope is presented.

Figure 16.1 Stability anlysis results for a 20m high slope



The slope stability analyses results for different slope heights have been used to develop the design charts as presented in Figure 16.2.

Figure 16.2 Slope height-overall slope angle relationship using typical sections



Design checks

The stability of the slopes of each pit has been checked using the design charts. The geology, groundwater conditions and the depressurisation potential have been considered in this assessment and the applicable Factor of Safety (FoS) for each slope section was determined.

The majority of the slopes checked have met the target FoS of 1.2 and are in fact conservatively designed. The few slopes which do not comply with the 1.2 target FoS, also have FoS values > 1.1, which is acceptable at the PFS level of accuracy[3]. It is expected that at the next level of study (feasibility) more detailed assessments will be made on potential groundwater depressurisation and dewatering.

16.2.4 Recommendations for further work

- The geotechnical models for the Araguaia project targets are of a preliminary nature due to limited geotechnical drilling. It is recommended to update the geotechnical models in the next stage of study with:
  - More geotechnical drilling in data limited sections
  - Detailed definition of the laterite profile including the main material zones and their sub facies including basic engineering properties
  - Detailed definition of the special variability of the laterite profile and sub zone.
- Specialised laboratory tests (e.g. tri axial and direct shear) to better define the shear strength Parameters of the weaker material zones
- Laboratory compaction and Californian bearing ratio (CBR) tests on ferricrete material to ascertain suitability as sheeting material
- Study the mine dewatering and depressurisation potential to confirm the design assumptions.

16.3 Mine planning methodology

All seven pits were designed through a standard process of pit optimisation, waste dump design and pit design.

Pit optimisations were completed in Whittle 4X software, an industry standard package. This software determines the economic limits of each deposit after accounting for estimated revenues and costs associated with mining each resource and waste block and the maximum allowable slope angles. The results of the pit optimisation were pit shells which were used for subsequent mine planning.

The pit surfaces were used to derive volumes for waste dump placement. Feedback from all relevant stakeholders was used to determine a waste disposal concept for each pit, including both ex-pit and in-pit options. After calculating the volumes of each waste type, waste dumps were designed to contain this material and minimise required haulage distances as much as possible.

[3] *Guidelines for Open pit Slope design, Editors: Reed, J. and Stacey, P. 2010, CSIRO Publishing, Collingwood, Vic, Australia*

Because all of the deposits are near surface (less than 25 m), it was determined that comprehensive ramp designs would not be required, consequently smoothed pit shells were used as the basis for design. However, the major external road networks were explicitly designed.

Each pit was split into a number of panels for scheduling. These are explained in more detail in the relevant sections below.

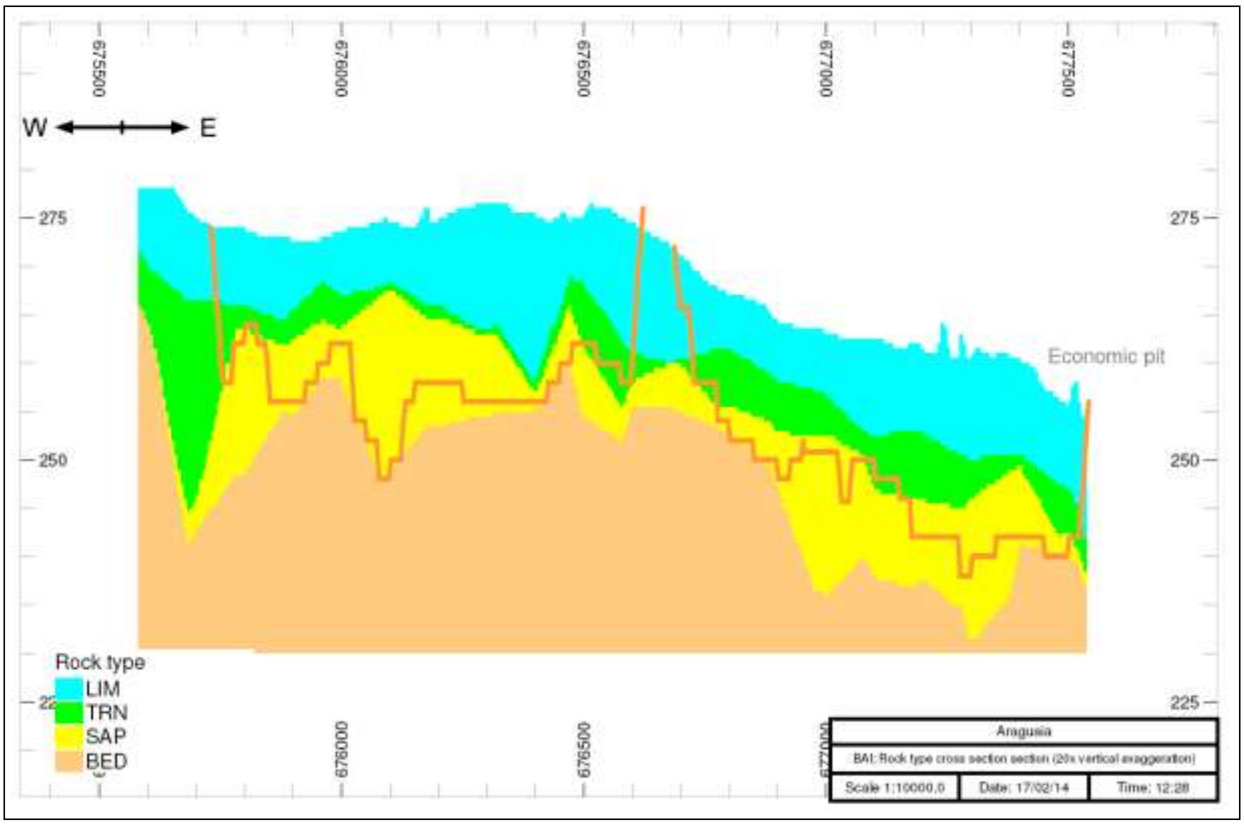
16.4 Mining concept

16.4.1 Excavation

Each of the deposits is proposed to be mined with typical truck and excavator mining. Snowden is of the opinion that other options (such as scraper/dozer systems or bucketwheel excavator and similar) are unlikely to give a material improvement in project economics.

Excavation through the lateritic profile typically encounters five rock types (soil, iron cap, limonite, transition, saprolite). A typical cross section is shown in Figure 16.3.

Figure 16.3 Typical cross section showing rock types



Soil

Soil is typically pushed by a dozer. An excavator is used to load it into trucks. Trucks then haul the soil to the appropriate stockpiles for later use on rehabilitation, or possibly direct to active rehabilitation areas. At the dump sites, other dozers may be used to arrange and manage the stockpiles.

Iron cap (ferricrete)

The iron cap, which is immediately below the soil, is brittle. Once the cap is fractured and penetrated, it can be readily dug. It is assumed that the cap will successfully be broken by ripping and/or by high break-out force hydraulic excavators. Beneath the cap, there is often a more granular unconsolidated pisolitic iron rich material. This can be excavated with the iron cap. The iron rich layer averages 2 m thick with a maximum of about 5 m. It closely follows the geomorphology of the land surface.

The excavator and trucks can traffic directly on the iron cap which comfortably supports the machines. Hence there is little required road works. The excavator operator can readily see the contact between the iron material and the underlying limonitic clays.

The unconsolidated pisolitic ferricrete will be used as construction material. It will often be hauled directly to a construction site such as platforms, road ways, and water control embankments. If there is any surplus it will be stockpiled for later construction use. The pisolitic ferricrete may require primary crushing prior to use as a roadway material.

Limonite

The next layer in the profile is a limonite rich clay. This material typically increases in nickel grade and water content with depth. This material is freely dug with an excavator. The limonite layer at Araguaia averages about 4 m in depth.

While the top of the limonite may be marginally trafficable with machinery, it is rarely possible towards the bottom. Accordingly, sheeted ‘finger’ roads will be required (see Section 16.4.2). Trucks will travel to the face on these finger roads. Generally the loading unit will operate from the bench above (4 m above) and load into the trucks. In some situation dozers will be employed to push material to the loading units.

The lower surface of the limonite is usually visually recognisable. The grade boundary and limonite/transition boundary will require moderately selective mining methods. The limonite grade boundary is smooth and gradational and thus diluting materials taken from above the grade boundary will only be slightly below the cut-off grade. The lower contact boundary against the transition material will not negatively affect the nickel grades (rather it will improve them) but it will increase the limonite grades in MgO and SiO_2. This will be monitored closely as it may cause blending problems in the feed preparation to the plant.

Transition

Transition is practically all ore grade. It averages 3 m thickness up to a maximum of 5 m. The transition lower contact pinches and swells into the saprolite, and thus does not appear everywhere. Low spots are generally where ground water concentrates and are thus very wet.

Transition has a unique chemistry and will be carefully separated from the other ore types prior being fed to the plant. While the contact with the overlying limonite is fairly clear, the lower contact with saprolite is more gradational. However, due to its relatively thin nature, transition is expected to mix with its neighbouring rock types during mining. Selective mining of the transition will be undertaken to control the Fe, MgO, SiO_2 and other chemical concentrations of the process feed. The nickel concentration of the limonite (above) and the saprolite (below) is often similar at the respective transition boundary. Thus, there will be insignificant nickel dilution. However, the other elements often change over a short vertical interval and are of greater concern from a dilution perspective. It is inevitable that some transition will report to the other ore streams (limonite and saprolite) and vice versa. This has been accommodated in mine planning through the re-blocking of the grades at these boundaries.

Transition material is generally of high water content and not trafficable without sheeting. Accordingly, in most cases transition will be mined predominantly from the bottom limonite bench. This implies that the excavator is reaching down and removing the last of the limonite and then the transition from more or less the same position. Where mining downwards from the last limonite bench is not safe or practical, it will be necessary to establish finger roads and bench access just below the transition in the top levels of the saprolite. The excavator will then be scraping down the face and loading trucks on the same level. Dozers will often be utilised to help feed the loading unit and clean off the limonite and transition layers appropriately.

Saprolite

Saprolite (SAP) is nearly all ore and averages 4 m in thickness (to the economic pit floor). However the floor (which is the pit bottom) is defined as a nickel cut-off boundary and, in some instances, reaches the bedrock contact. The nature of saprolite is a variable mix of particle sizes from fines to larger rocks. The presence of these rocks increases with depth and is captured in the drill logs differentiating between “earthy saprolite”, which contains more clays, and “rocky saprolite” which contains less clays. The saprolite has very high water content.

Saprolite will be mined separately due to its particular chemistry. The MgO content of the plant feed is largely controlled by the saprolite component.

Saprolite will be mined in a similar manner to the limonite with finger roads established on 4 m benches.

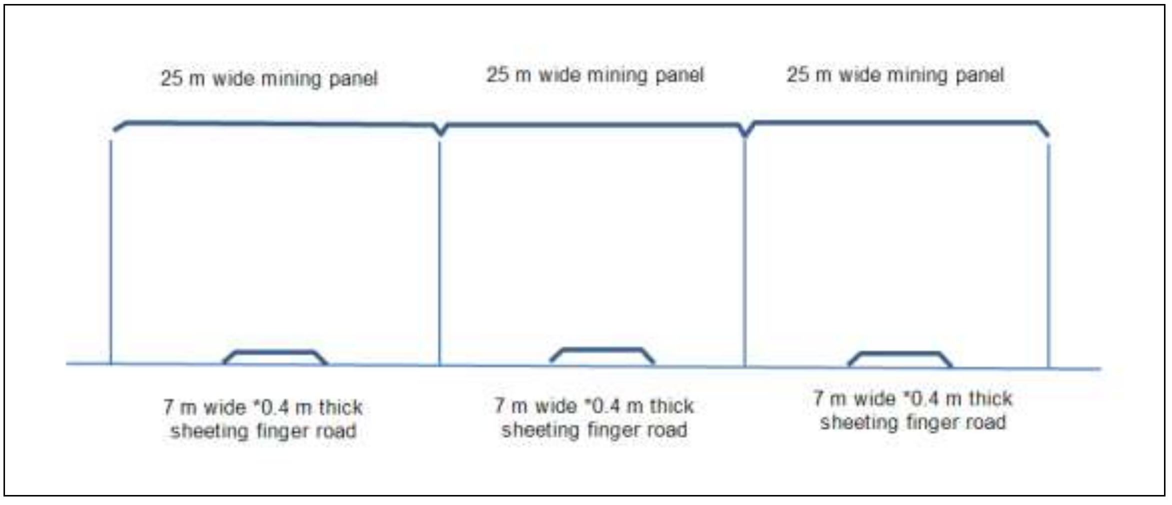
In some areas, the economic pit outline approaches the bedrock. The final floor will be a combination of Ni grade cut-off limit and bedrock (where the SAP is still above cut-off at the contact). The floor will be an uneven non-planar surface possibly with some pinnacles and troughs. This may pose operational difficulties with access and water management. Overall, the floors of the pits have a gradient. As a general philosophy, mining will advance uphill so as to allow the machinery to work away from the low point where water will collect.

At the time of this report, the exploration drilling has not encountered significant barren "core stones" or pinnacles typical of some other nickel laterite deposits. These tend to increase towards the bedrock interface. Similarly, high grade nickel occurrences in thin cracks and altered faces of rocks near the bedrock contact have not been seen. Boulder sized core stones and pinnacles, should they occur, can cause a number of operating challenges from mining to processing. Test pits down to the bedrock surface, will need to be completed in future studies to confirm that these features are not present, and, if they are identified, incorporated in future mining and processing strategies and plans.

16.4.2 Trafficability

Snowden has completed a preliminary assessment of the trafficability of the materials and suggested road construction guidelines based on the geotechnical properties of these materials (See Section 16.2.2). Sheeting quantities have been estimated using an average sheeting thickness of 0.4 m for a 40 t capacity articulated truck. Mining slices of approximately 25 m wide have been assumed based on the dig envelope of an 48 t excavator. With a finger road width of 7 m in each mining slice of 25 m, this equates to 28% of the bench surface being sheeted. (Figure 16.4).

Figure 16.4 Typically sheeting requirement



For a bench height of 4 m, the mass of sheeting is approximately an extra 6% over and above the mass of the bench.

Sheeting will be sourced from the unconsolidated pisolitic ferricrete "iron cap" which is present above the limonite over much of the deposit. Analysis of drillhole logging shows an average thickness of this material (within the economic pits) to vary between 0.9 m and 2.8 m (Table 7.2).

Given the pits are less than 20 m deep (on average) it was determined that the amount of pisolitic ferricrete material available is sufficient to supply the sheeting requirements of the mine and provide some excess material for aiding in construction of main haulage roads and improving stockpile and waste dump trafficability.

16.4.3 Ore transport

Ore will be transferred from each deposit to a near mine stockpile. There are four near mine stockpiles associated with each of the mining zones. These are listed in Table 16.5. Ore is hauled from these stockpiles via highway trucks to the run of mine (ROM) facility, located near the plant.

Table 16.5 Mining zones

Mining Zone	Deposits
Vila Oito zone	Vila Oito East, Vila Oito, Vila Oito West
Jacutinga zone	Jacutinga
Pequizeiro zone	Pequizeiro, Pequizeiro West
Baião zone	Baião

Ore will be stockpiled according to rock type (limonite, transition and saprolite) and nickel grade (mineralised waste, low grade and high grade). The classification system is shown in Table 16.6.

Table 16.6 Ore grade bins

Rock Type	Mineralised Waste[4] (MW)	Low Grade (LG)	High Grade (HG)
Limonite	>0.8%	-	
Transition	0.8% - 1.4%	-	>1.4%
Saprolite	0.8% - 1.1%	1.1% - 1.4%	>1.4%

There is a relatively strong visual differentiation of the various rock types which will aid in grade control. Grade control drilling (on a 12.5 m by 12.5 m pattern) will be used to provide vertical grade differentiation between mineralised waste, low grade and high grade. Additionally, this will assist in determining the economic depth of the pit.

Rock type is also used to control the non-nickel chemistry of the ore (Fe, MgO, SiO_2, Al_2O_3). Piles for each rock type will build up (and be homogenised through strategic dumping and dozing) over the course of a month or more to reduce the variability of the chemistry contained within each pile. A record of the average grade of the pile is maintained for later blending. When the pile is closed off it is available for ore haulage to the ROM facility.

The high grade stockpiles will typically be retained for a period of less than a quarter. Low grade stockpiles may remain for up to 20 years. Mineralised waste is not processed in this mine plan although may be considered for processing after the low grade stockpiles are depleted.

[4] *All Inferred Resources above 0.8% Ni are treated as mineralised waste.*

A ROM shed will be built at the process facility where ore will be dumped and dozed into finger piles for each rock type in order to homogenise it at the appropriate ratio prior to loading into the grizzly. The possibility of direct tipping of some material may be considered during operations but has been excluded from this study. If prolonged rainy periods are experienced, limiting the haulage of material to the facility, it may be necessary to draw from the nearby Pequizeiro mine stockpiles during this time to maintain feed.

Some small stocks of specific blend materials (high iron, low iron etc.) will be stored in piles near the ROM facility to allow the plant some flexibility to manage variations in grade that will inevitably occur.

16.4.4 Waste disposal

Waste rock, (defined as nickel grade less than 0.8%), is disposed in waste dumps and back into the mined out pits. Each mining area has a waste dump. Snowden has applied a pit backfill factor of 35% of the mass. This is based on:

- Initial mining: It is required to mine out some of the pit to provide space to backfill into. In the meantime waste needs to be stored externally.
- Scheduling logistics: In order to mine the highest grades first to improve the project economics, mining takes place from a number of disjointed separated areas. Thus there might not be a simple path from the waste excavation face to the nearest area to backfill in the pit
- Sterilisation: there remains a significant amount of the pit floor that contains mineralised material that may be mined later. Placing waste on top of these areas reduces the potential viability of this option, thus reducing the potential life of the operation (and the flow on to local jobs).

The 35% factor mitigates these risks by allowing for the largest footprint for the operation. It is likely that the actual percentage backfilled is higher and the external waste footprints are smaller.

It is not proposed to re-handle all the external waste dump material back into the pits at the end of the mine life.

High quality access roads and drainage will be maintained for ex-pit- waste dumps throughout the mine life. Waste will be pushed to the middle of cells (not left to accumulate by the finger roads) to eliminate ponding during the wet season. Additional dozing time has been allocated for these activities.

16.5 Modifying factors

The parameters used for Mineral Reserve determination are provided below. These are more conservative than those used in the economic evaluation. In this way the reserve is relatively robust with respect to fluctuations in costs, recoveries, and commodity prices.

16.5.1 Resource model

Snowden used the resource models shown in Table 14.11. Details on these models are provided in Section 14.

16.5.2 Geotechnical parameters

A 35° overall wall angle was applied for pit optimisation. The derivation of this was as a consequence of geotechnical drilling, and analysis of typical pit wall lithologies. (Section 16.1)

16.5.3 Mining parameters

A mining cost of USD_3/dmt for waste and USD_9/dmt for ore (incorporating stockpile reclamation) was applied for optimisation.

16.5.4 Processing parameters

Based on preliminary modelling, a processing cost of USD127.8/dmt for optimisation and a nickel recovery of 93%, which incorporates processing, smelting and refining. This is applied for all material processed. There is no screening of ore at the mine, thus all material is feed for processing. Detail on the testwork and process designs that support these assumptions are supplied in Section 13.

A range of process grade constraints for various ore constituents was used for the Base Case:

- Nickel grade greater than 1.2%
- Iron grade between 15.0% and 16.5%
- Al_2O_3 grade between 4.0% and 5.5%
- SiO_2/MgO ratio between 2.2 and 2.6.

These constraints were determined as part of the process design which is detailed in Section 17.

16.5.5 Sales parameters

The processing option for this study is RKEF which produces a refined ferronickel alloy. Consequently, a value of USD15,000/t of recovered nickel was used for the optimisation. This price incorporates:

- nickel spot price of USD 19,000 / tonne flat for LOM
- iron spot price of USD 150 / tonne flat for LOM
- Fe-Ni ratio of 20.3% Ni and 79.7% Fe
- royalties for the combined Fe-Ni product is 2% of the costs (up to the furnace)
- selling costs - there are no selling costs as product is being sold at the mine gate

16.5.6 Administration costs

A cost of USD11.20/t dry feed was applied for administration. This is equivalent to USD10.0 Mpa.

16.5.7 Cut-off grades

Using the above assumptions, the marginal cut-off grade for the project is calculated to be 1.04% Ni. This grade was not applied for mine planning due to the following reasons:

The overall ore feed is off-specification for the plant. In particular, the overall iron grade is too high. Therefore, the higher iron grade rock types (limonite and transition) had elevated cut-off grades applied to reduce their impact on the average iron grades whilst maintaining the highest possible grade bring the overall grade to within specification.

The mine life is long and impacts negatively on net present value. Applying a higher cut-off grade reduces the footprint of the pit, increases the average grade and avoids mining lower grade material that would not be processed for a number of years.

Through iteration Snowden determined that the appropriate cut-off grades to meet the grade specifications (including minimum nickel feed grade) were:

- Limonite – not processed
- Transition – 1.4% Ni
- Saprolite – 1.4% Ni.

These cut-off grades were applied in the pit optimisation and scheduling, lower grade Saprolite (>1.1% Ni) was processed at the end of the mine life.

16.6 Mining model

Each of the resource models had parent cell size of 25 mE by 25 mN by 2 mRL with a minimum subcell of 6.25 mE by 6.25 mN by 0.5 mRL.

To apply dilution Snowden re-blocked the models to the parent cell size of 25 mE by 25 mN by 2 mRL. This method mimics the natural mixing processes of mining and allows for local dilution of all elements, which is important for the process feed blending. The resultant mining mineral inventory is shown in Table 16.7.

Table 16.7 Reblocked Indicated Resource summary at 1% Ni cut-off (Transition and Saprolite material only)

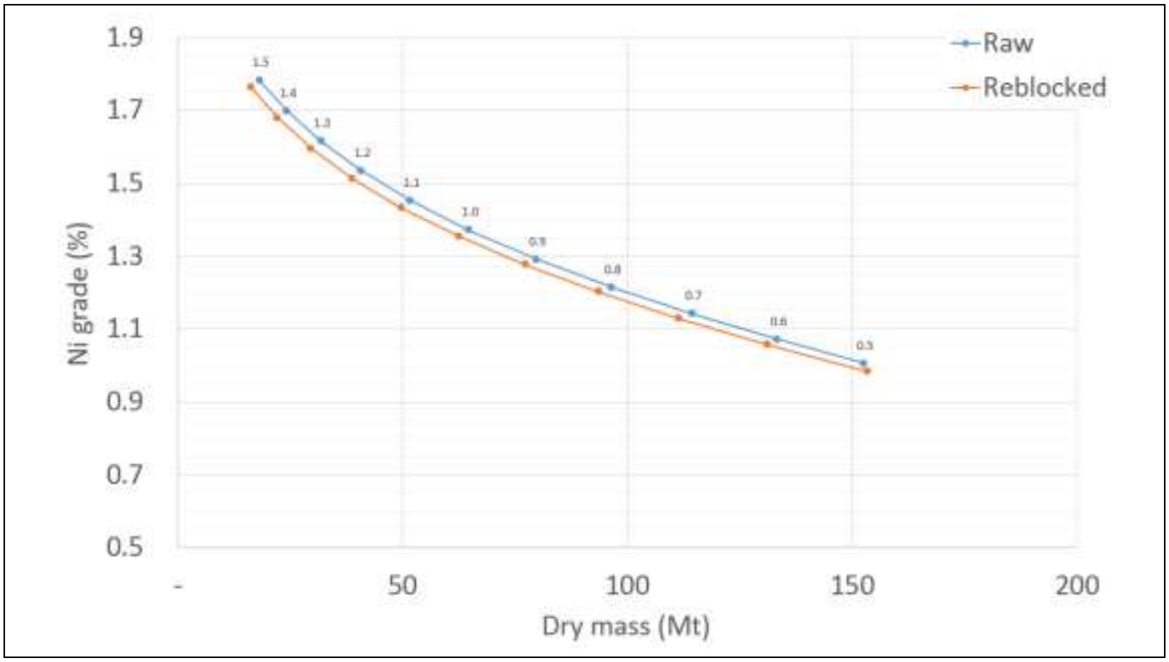
Deposit	Dry Mass (kt)	Ni (%)	Fe (%)	Al2O3 (%)	SiO2/ MgO
Baião	15,194	1.30	22.2	6.06	2.48
Pequizeiro	18,077	1.46	16.5	5.75	2.31
Pequizeiro West	2,500	1.23	19.6	5.13	4.62
Jacutinga	2,513	1.43	18.5	3.80	2.33
Vila Oito East	7,474	1.31	16.7	3.97	2.30
Vila Oito	10,528	1.34	17.2	4.09	1.99
Vila Oito West	6,252	1.28	23.3	5.14	3.14
Total	62,537	1.35	18.9	5.17	2.40
Comparison to raw model at 1% Ni cut-off	(3.4%)	(1.3%)	2.8%	3.6%	0.3%

The re-blocking procedure appears to reduce the material above cut-off but this is simply associated with decreasing block grade before applying the cut-off grade. Figure 16.5 shows that for a common tonnage the nickel grade is reduced by about 4% as a result of re-blocking at the cut-off grade range of interest, indicating effective dilution of approximately 4% on grade (after considering the grade of nearby blocks). Snowden deems this to be appropriate given that:

The low strip ratio means that there is not a high incidence of waste blocks adjacent to ore blocks. The majority of dilution is likely to occur at the rock type boundaries.

The grade of diluting blocks is likely to be similar to the grade of the ore blocks i.e. there may be higher than 4% dilution but it has grade that is much higher than zero which minimises the impact on recovered grade.

Figure 16.5 Grade-tonnage curve comparison for nickel grade

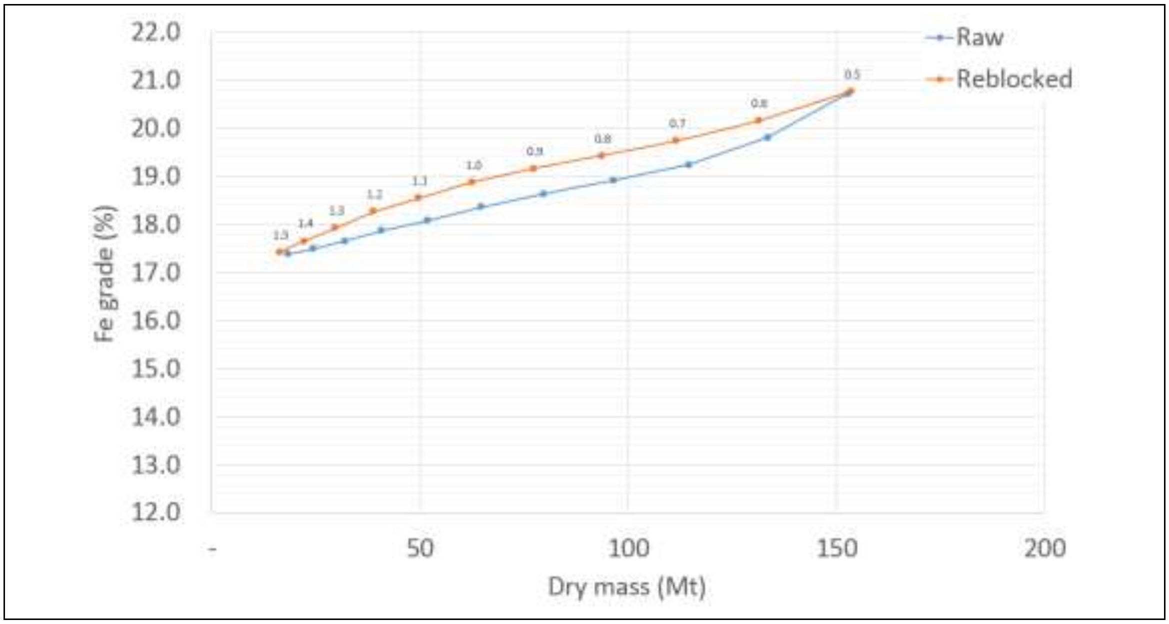


Note: data labels indicate nickel grade cut-off

Figure 16.4 to Figure 16.8 show the impact of re-blocking on other important elements and ratios. To summarise the important outcomes:

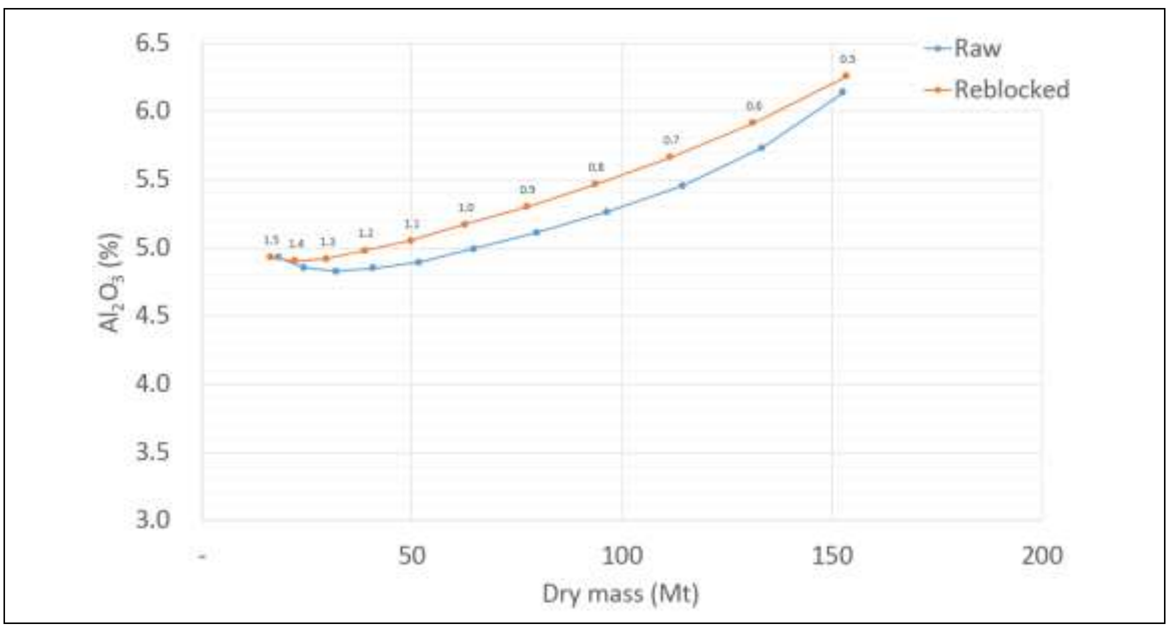
- Iron and Al_2O_3 grade is increased as high iron/Al_2O_3, high grade limonite dilution is included at the limonite/transition boundary
- SiO_2/MgO ratio is largely unaffected.

Figure 16.6 Grade-tonnage curve comparison for iron grade



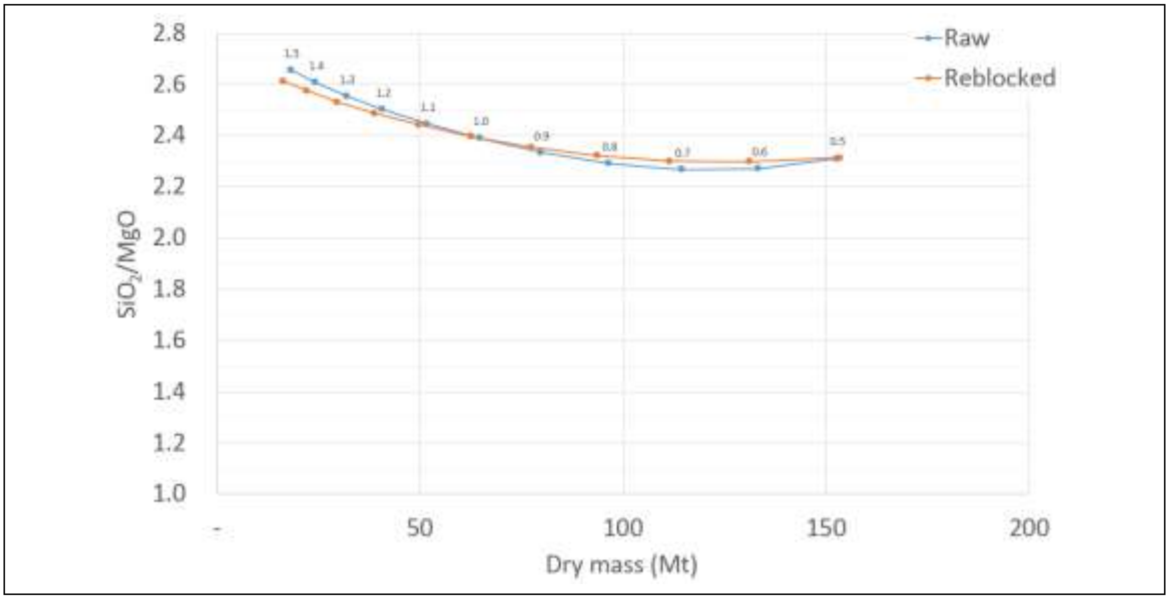
Note: data labels indicate nickel grade cut-off

Figure 16.7 Grade-tonnage curve comparison for Al2O3 grade



Note: data labels indicate nickel grade cut-off

Figure 16.8 Grade-tonnage curve comparison for SiO_2/MgO ratio



Note: data labels indicate nickel grade cut-off

No further mining recovery or grade factors were applied for mine planning.

For pit optimisation, this model was then split into 5 mE by 5 mN by 2 mRL blocks (or the same grade) so that the selected wall angles could be achieved in the optimisation.

16.7 Pit design

The pit design used smoothed pit shells from the pit optimisation (Section 15.3), altered for the removal of small satellite pits. This was deemed by Snowden to be appropriate for pits with no ramp requirements. It is likely that the actual pit floor will be dictated by operating conditions as they are mined, although the quantities mined from each will be similar to those above.

The reconciliation of volumes and value between the pit shell selected for design and the design itself is shown in Table 16.8. This reconciliation is deemed to be appropriate.

Table 16.8 Design inventory comparison

Item	Design	Pit Shell	Difference (%)
Total pit dry mass (kt)	81,166	83,676	(3.0%)
Waste dry mass (kt)	61,631	63,800	(3.4%)
Feed dry mass (kt)	19,583	19,881	(1.5%)
Ni (%)	1.69	1.69	(0.0%)
Strip ratio (w:o)	3.14	3.21	(2.1%)
Cash flow ($M)	1,542	2,955	(0.7%)

For scheduling, the designs were split into mining panels. These panels are shown for each deposit in Figure 16.9 to Figure 16.15, with key statistics reported in Table 16.8 to Table 16.16.

Figure 16.9 Baião mining panels

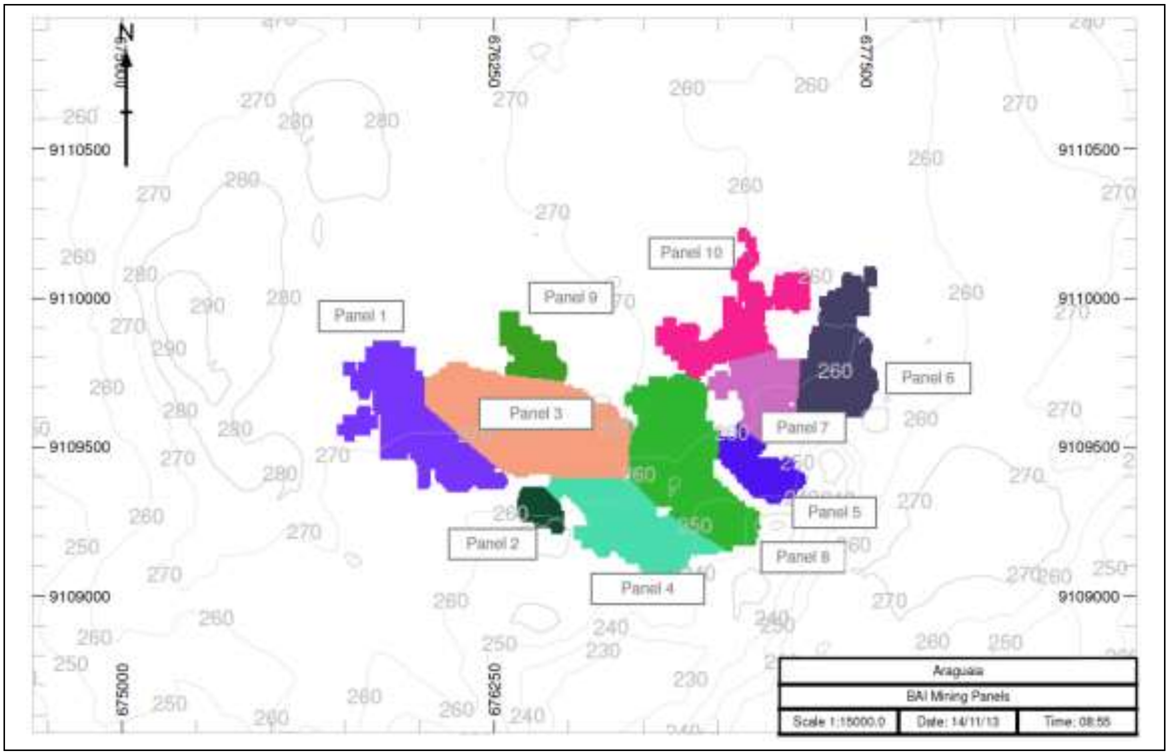


Table 16.9 Baião mining panel summary

Panel	Panel Size Dry Mass (kt)	Waste Dry Mass (kt)	Strip Ratio (w:o)	Ore Dry Mass (kt)	Ni (%)	Fe (%)	Al_2O_3 (%)	SiO_2 / MgO
1	2,160	1,625	3.04	534	1.60	18.0	4.66	2.37
2	354	290	4.56	64	1.49	16.7	4.62	2.64
3	4,033	3,094	3.30	939	1.67	17.2	4.42	2.23
4	2,131	1,814	5.72	317	1.56	15.1	4.84	2.97
5	655	539	4.67	115	1.48	16.7	5.75	2.93
6	1,991	1,619	4.35	372	1.76	15.0	3.56	2.53
7	1,142	941	4.67	202	1.55	20.2	4.76	3.39
8	3,091	2,402	3.48	690	1.85	17.8	4.70	2.84
9	418	351	5.24	67	1.59	24.1	7.97	1.81
10	1,234	1,008	4.47	225	1.54	19.0	4.14	2.83
Total	17,208	13,683	3.88	3,524	1.67	17.4	4.58	2.56

Figure 16.10 Pequizeiro mining panels

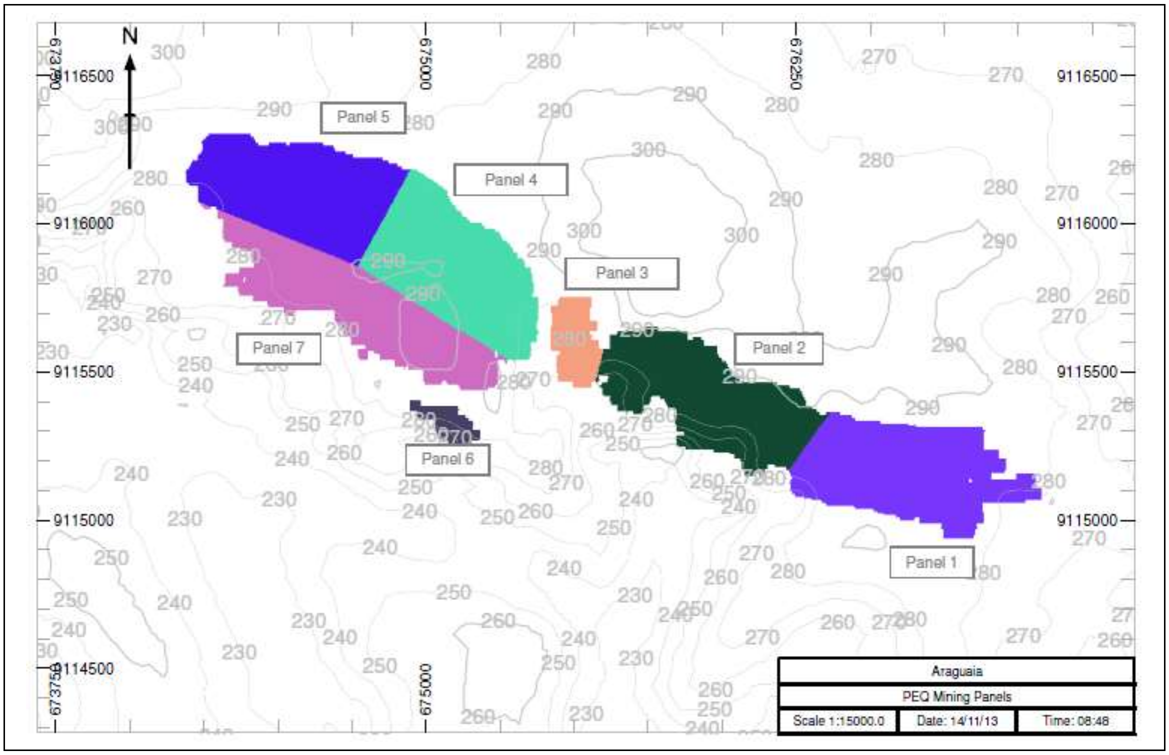


Table 16.10 Pequizeiro mining panel summary

Panel	Panel Size Dry Mass (kt)	Waste Dry Mass (kt)	Strip Ratio (w:o)	Ore Dry Mass (kt)	Ni (%)	Fe (%)	Al_2O_3 (%)	SiO_2 / MgO
1	5,523	4,161	3.05	1,363	1.76	17.7	6.65	3.17
2	4,524	2,671	1.44	1,853	1.73	15.1	6.05	2.14
3	513	337	1.91	177	1.53	16.9	5.96	2.46
4	6,203	4,097	1.94	2,107	1.72	14.9	5.38	3.72
5	6,329	4,114	1.86	2,215	1.70	15.0	3.88	2.04
6	212	114	1.17	98	1.44	14.7	4.71	2.09
7	4,289	2,806	1.89	1,483	1.58	16.0	5.63	2.45
Total	27,594	18,299	1.97	9,295	1.70	15.6	5.39	2.56

Figure 16.11 Pequizeiro West mining panels

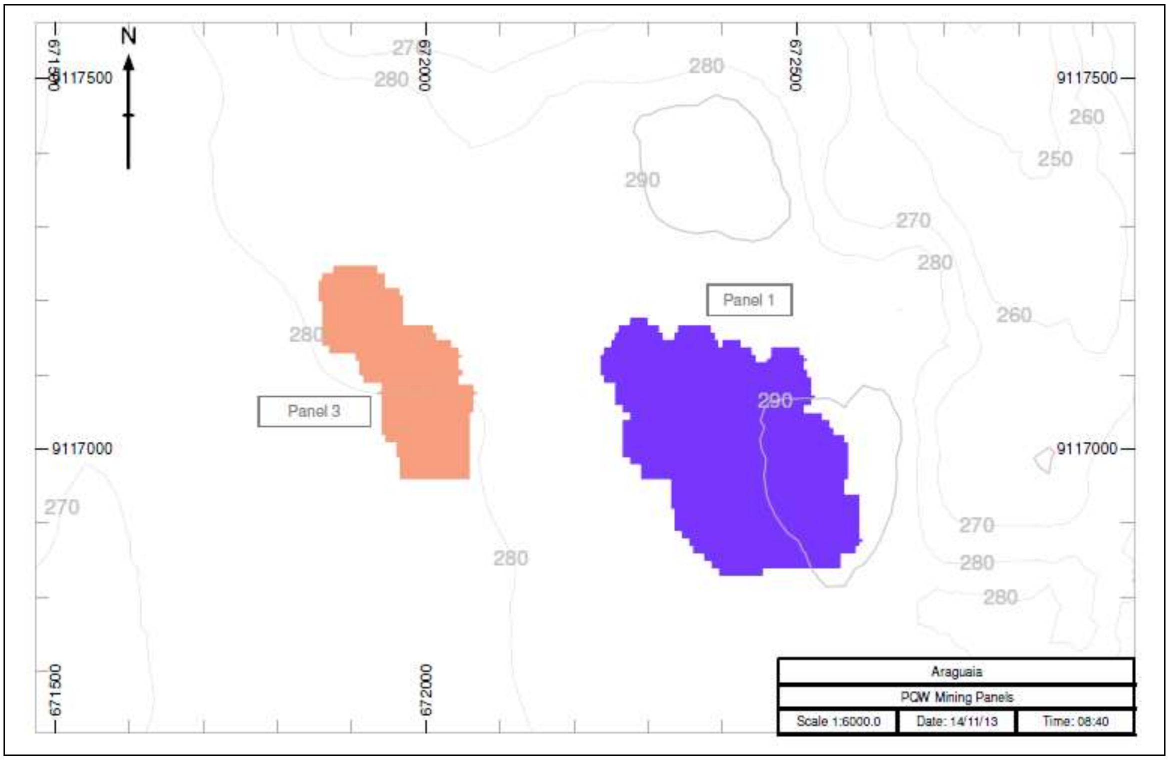


Table 16.11 Pequizeiro West mining panel summary

Panel	Panel Size Dry Mass (kt)	Waste Dry Mass (kt)	Strip Ratio (w:o)	Ore Dry Mass (kt)	Ni (%)	Fe (%)	Al_2O_3 (%)	SiO_2 / MgO
1	1,447	1,129	3.55	318	1.57	20.2	4.55	4.04
3	527	465	7.55	62	1.56	21.6	5.02	6.30
Total	1,973	1,594	4.20	379	1.57	20.4	4.63	4.29

Figure 16.12 Jacutinga mining panels

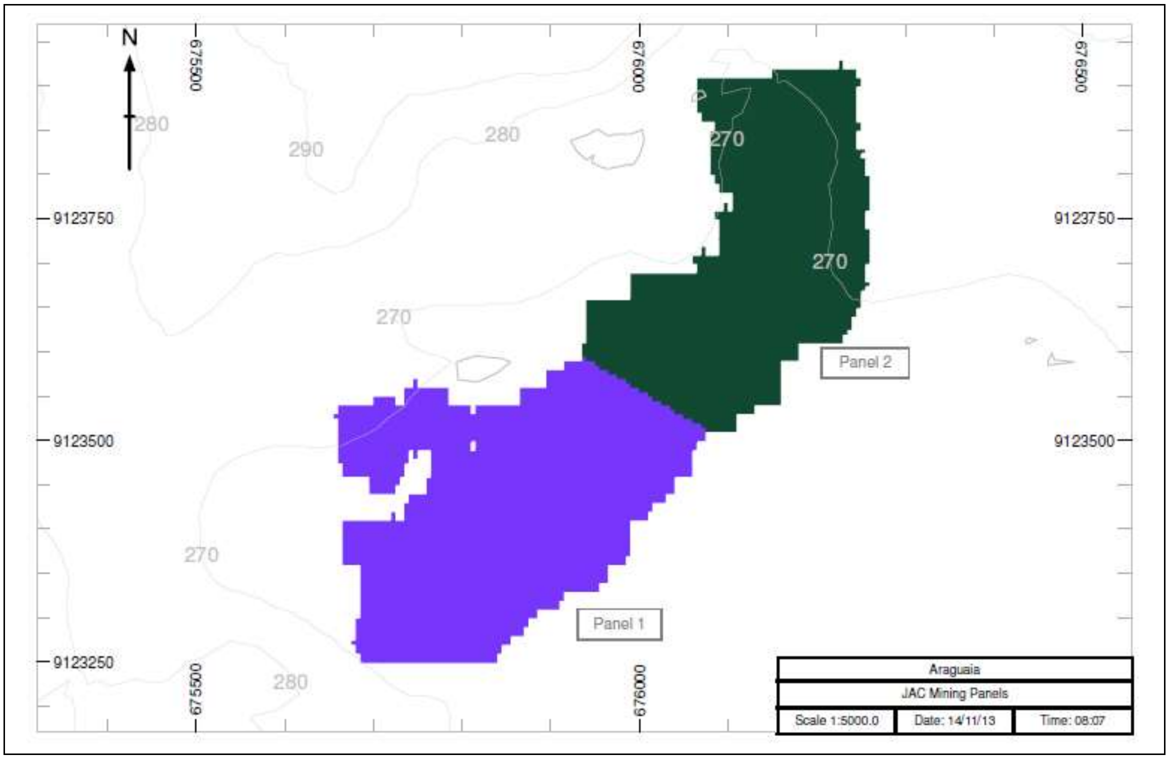


Table 16.12 Jacutinga mining panel summary

Panel	Panel Size Dry Mass (kt)	Waste Dry Mass (kt)	Strip Ratio (w:o)	Ore Dry Mass (kt)	Ni (%)	Fe (%)	Al_2O_3 (%)	SiO_2 / MgO
1	1,544	938	1.55	605	1.86	14.0	2.76	2.06
2	1,395	1,043	2.96	352	1.74	17.1	3.31	2.22
Total	2,939	1,982	2.07	957	1.81	15.1	2.96	2.11

Figure 16.13 Vila Oito East mining panels

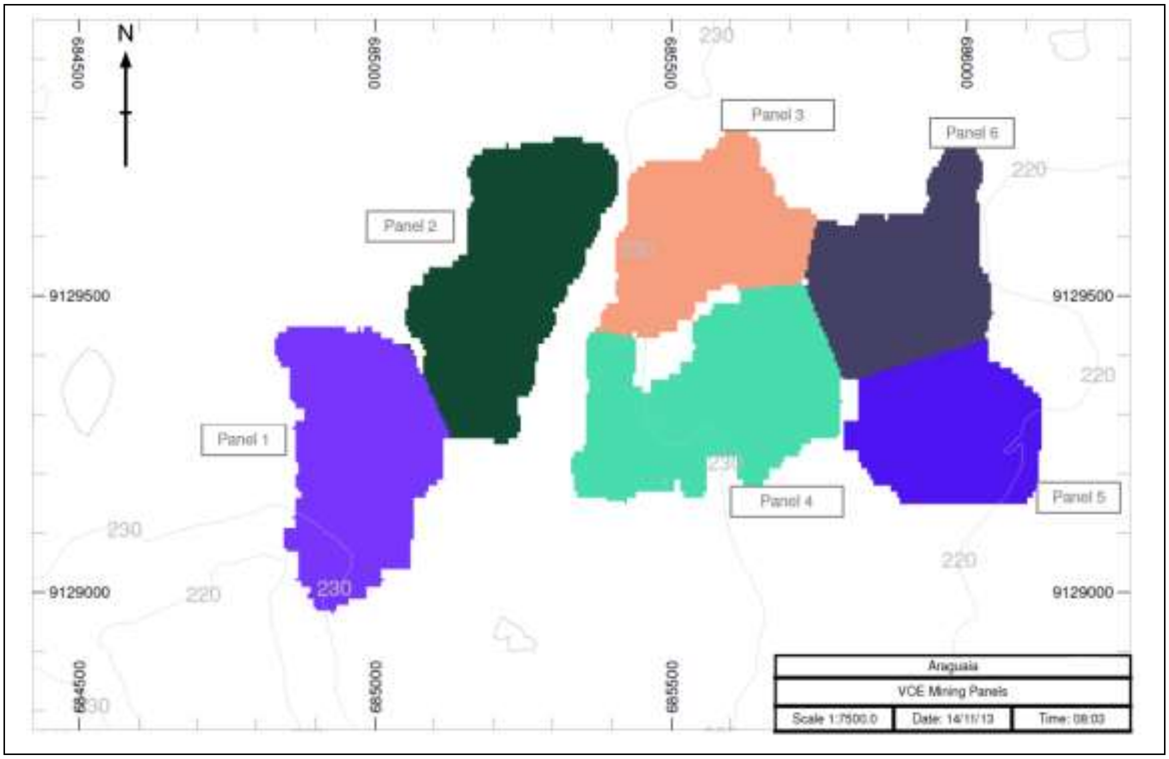


Table 16.13 Vila Oito East mining panel summary

Panel	Panel Size Dry Mass (kt)	Waste Dry Mass (kt)	Strip Ratio (w:o)	Ore Dry Mass (kt)	Ni (%)	Fe (%)	Al_2O_3 (%)	SiO_2 / MgO
1	1,786	1,380	3.40	406	1.58	19.4	4.04	2.39
2	2,525	2,109	5.08	416	1.53	18.8	3.81	2.59
3	1,716	1,467	5.89	249	1.56	17.6	3.31	3.07
4	1,991	1,508	3.12	483	1.58	13.8	2.65	1.97
5	1,524	997	1.89	527	1.49	13.4	5.40	2.27
6	1,499	1,131	3.07	368	1.55	14.5	2.58	1.70
Total	11,041	8,592	3.51	2,449	1.55	16.0	3.73	2.22

Figure 16.14 Vila Oito mining panels

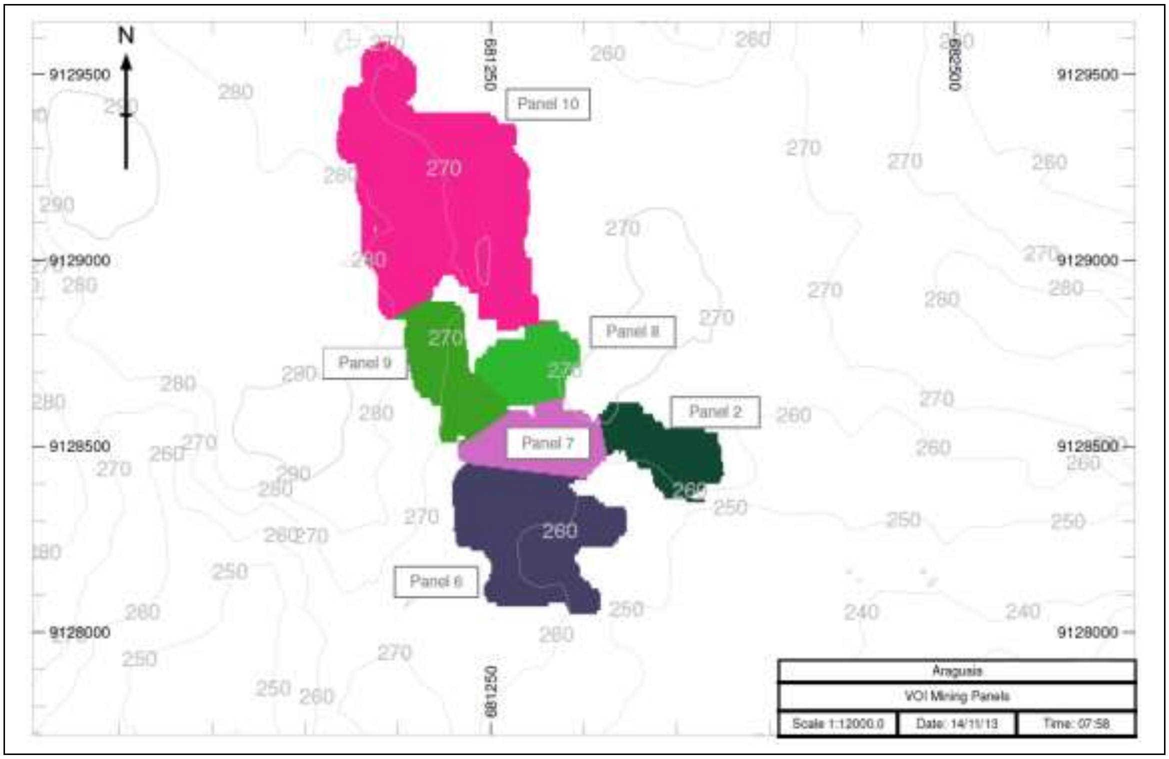


Table 16.14 Vila Oito mining panel summary

Panel	Panel Size Dry Mass (kt)	Waste Dry Mass (kt)	Strip Ratio (w:o)	Ore Dry Mass (kt)	Ni (%)	Fe (%)	Al_2O_3 (%)	SiO_2 / MgO
2	946	792	5.15	154	1.93	19.0	4.87	2.87
6	3,460	2,662	3.34	798	1.51	12.1	3.85	1.85
7	1,278	1,022	4.00	256	1.70	16.8	4.16	2.01
8	752	616	4.53	136	1.59	20.3	3.66	1.91
9	1,454	1,151	3.80	303	1.41	13.6	3.47	1.78
10	7,065	5,133	2.66	1,933	1.68	14.8	3.40	2.17
Total	14,956	11,377	3.18	3,579	1.63	14.6	3.63	2.05

Figure 16.15 Vila Oito West mining panels

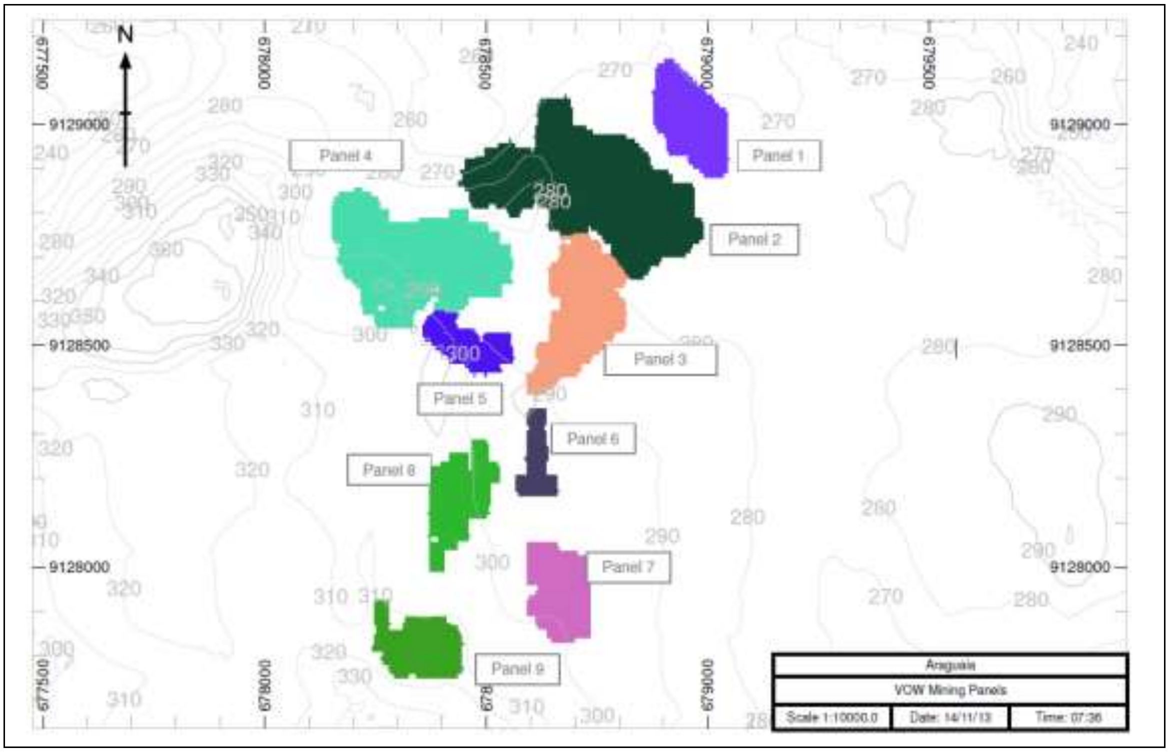


Table 16.15 Vila Oito West mining panel summary

Panel	Panel Size Dry Mass (kt)	Waste Dry Mass (kt)	Strip Ratio (w:o)	Ore Dry Mass (kt)	Ni (%)	Fe (%)	Al2O3 (%)	SiO_2 / MgO
1	518	470	9.97	47	1.54	20.2	4.27	3.91
2	2,154	1,828	5.61	326	1.64	20.1	5.54	4.01
3	665	544	4.49	121	1.70	20.0	3.79	3.97
4	968	684	2.40	285	1.58	18.9	3.41	3.44
5	174	148	5.64	26	1.47	17.7	3.19	3.88
6	100	90	8.93	10	1.50	22.4	3.62	3.45
7	383	308	4.13	75	1.60	16.9	3.09	1.85
8	284	215	3.08	70	1.42	18.3	4.31	2.35
9	298	236	3.82	62	1.51	19.8	4.03	2.55
Total	5,544	4,523	4.43	1,021	1.59	19.3	4.25	3.32

16.9 Mineral Reserve

A Probable Mineral Reserve of 21,204 kt (dry) at 1.66% Ni was estimated. The detailed breakdown of the Mineral Reserve by deposit is presented in Table 16.16.

Table 16.16 Probable Mineral Reserve

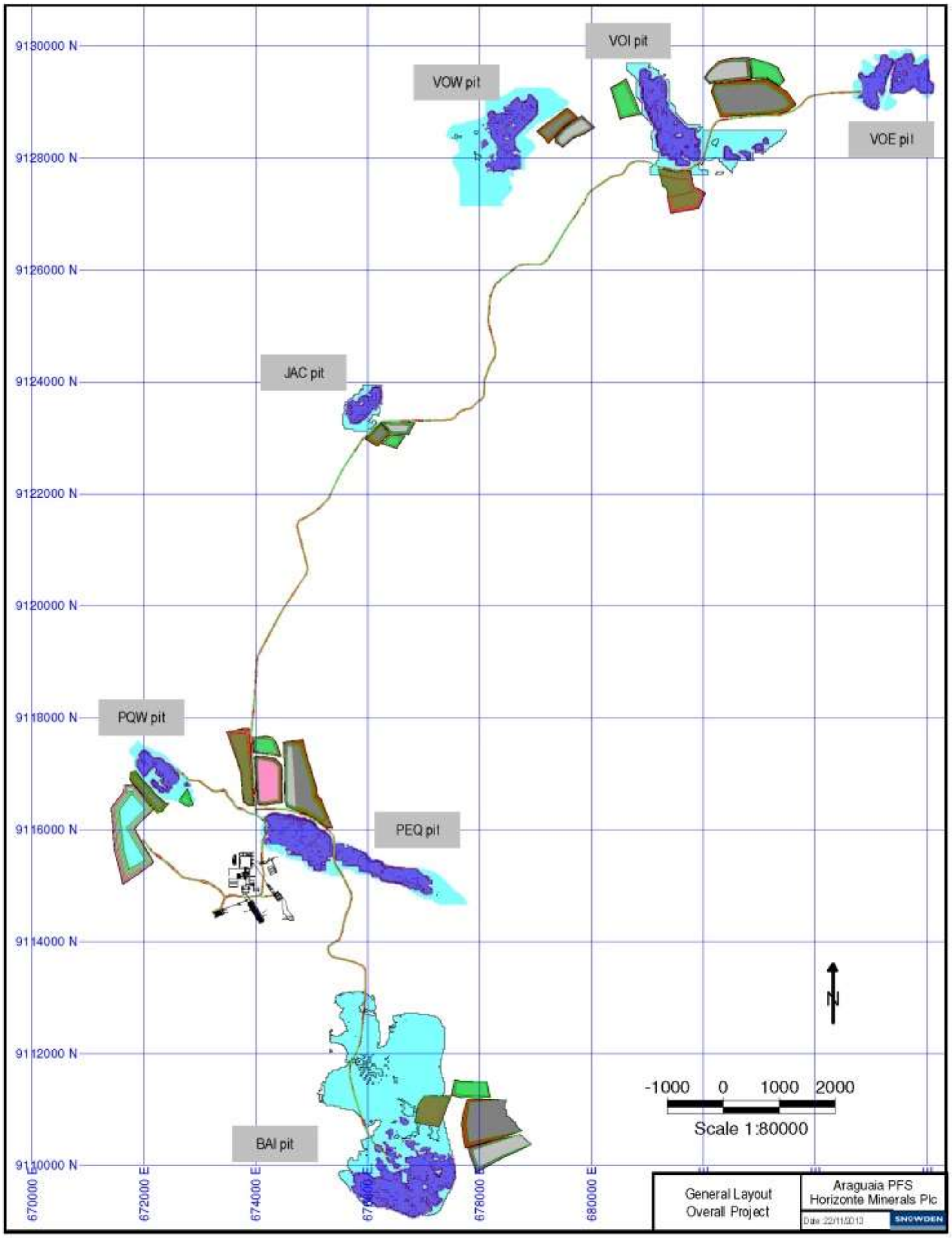
Deposit	Ore Dry Mass (kt)	Ni (%)	Fe (%)	Al_2O_3 (%)	SiO_2/MgO
BAI	3,524	1.67	17.41	4.58	2.56
PEQ	9,295	1.70	15.58	5.39	2.56
PQW	379	1.57	20.38	4.63	4.29
JAC	957	1.81	15.13	2.96	2.11
VOE	2,449	1.55	15.97	3.73	2.22
VOI	3,579	1.63	14.61	3.63	2.05
VOW	1,021	1.59	19.35	4.25	3.32
Total	21,204	1.66	16.01	4.59	2.44

This Mineral Reserve is calculated on the basis of currently available information. Snowden strongly recommends a test pit(s) to assess in-situ grade reconciliation to the resource model, incidence of barren rocks in the saprolite, mining recovery and mining dilution.

16.10 Site layout

The overall mining configuration layout is shown in Figure 16.16.

Figure 16.16 Overall mining configuration



16.10.1 Basis

The goals of the layout are to:

- Minimise haulage cost through minimised haul distances.

- Minimise the disturbance footprint and thus associated restoration and water management catchment basin areas.
- Minimise the number of disturbed drainage basins.
- Minimise impingement on forest zones.
- Facilitate the safe and efficient movement of material between the various sources and destinations at the appropriate time.
- In general the dump heights have been minimised creating a larger footprint. This reduces uphill haul, and is expected to reduce restoration costs, enhances dump stability, and in the case of shared dumps, reduces travel distance from respective pits.

The key design assumptions applied are:

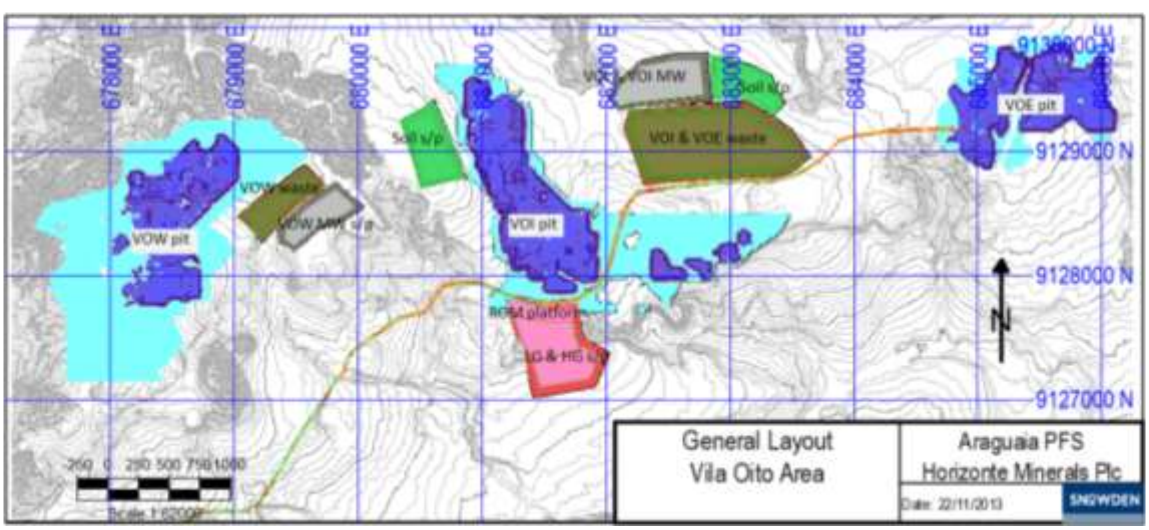
- Material allocations as shown in Table 16.17.
- For the waste dumps Snowden has assumed that 65% of the mined waste will be placed in the external ex-pit dumps with the balance (35%) back filled into the pit excavation (See Section 16.4.4).
- Dump slopes are modelled with a single uniform slope of 3H:1V (18.4°).
- The in-situ block model densities averages have been reduced for swell in the dumps by 0% for top soil, and 18% for all other rock types in line with swell factors seen in other laterite operations. This assumption should be confirmed at the next level of study.

Both top soil and sub-soil are removed and stockpiled from over the pits. Only topsoil is removed from the waste dumps. The total thickness of soil (topsoil and subsoil) within each of the pits (as measured from the drillhole database logging) ranges between 0.6 m and 1.6 m for the deposits. The split of subsoil and topsoil has not been modelled. Therefore, a 0.4 m topsoil layer was assumed but this should be confirmed in subsequent studies.

16.10.2 Vila Oito zone

The Vila Oito zone encompasses zones VOI, VOE and VOW. The proposed layout is shown in Figure 16.17. Care has been taken to site the facility so as to largely avoid the forested zones.

Figure 16.17 Vila Oito zone layout



The waste and MW (mineralised waste) for VOI and VOE areas are combined in a common dump while VOW has its own dumps. A single common LG/HG stockpile is planned for this zone. A ROM platform is shown integrated with the ore stockpile alongside the haul road. Active ROM materials will be homogenised/blended between pits and transferred into the on-highway trucks for transport to the plant. Soil is stockpiled in two areas.

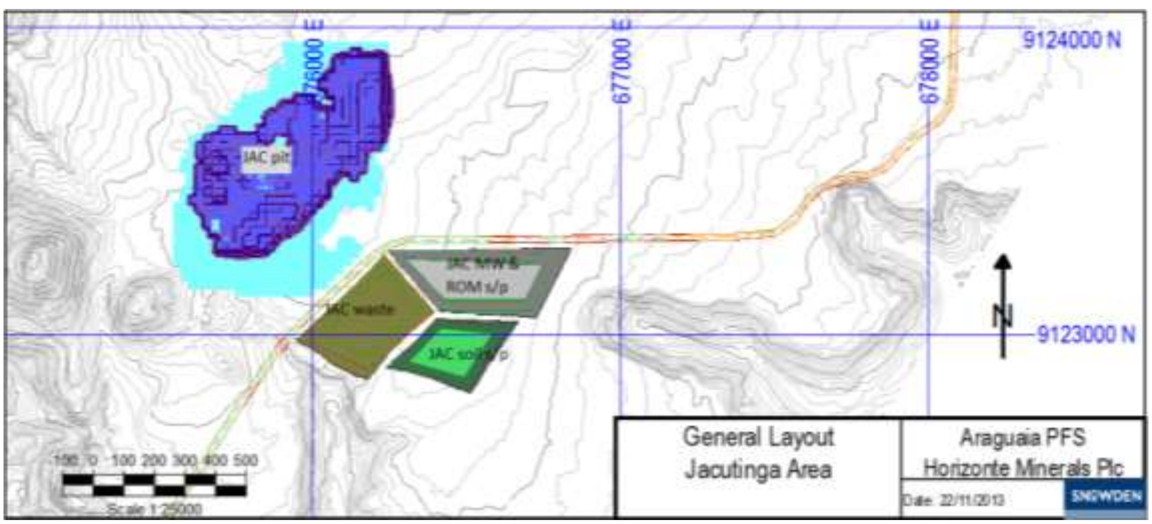
The deposits are located near the crests of the hills on quasi-plateaus. The creeks have incised these plateaus and eroded away the mineral deposits. Hence, the pits tend to be near the top of the drainage basins. The dumps and stockpiles are placed laterally or downstream of the pits such that the entire disturbed area drains into a few catchment pond locations. The proposed layout avoids interfering with the drainages to the north. All drainage is to the south where water management controls will be located.

16.10.3 Jacutinga zone

This small Jacutinga zone is located between the North area and the plant site (Figure 16.18). It has its own waste dump and a 1 Mm^3 mineralised waste (MW) stockpile as shown.

Drainage is to the SE and the entire disturbed area can be controlled with water management structures south of the dumps.

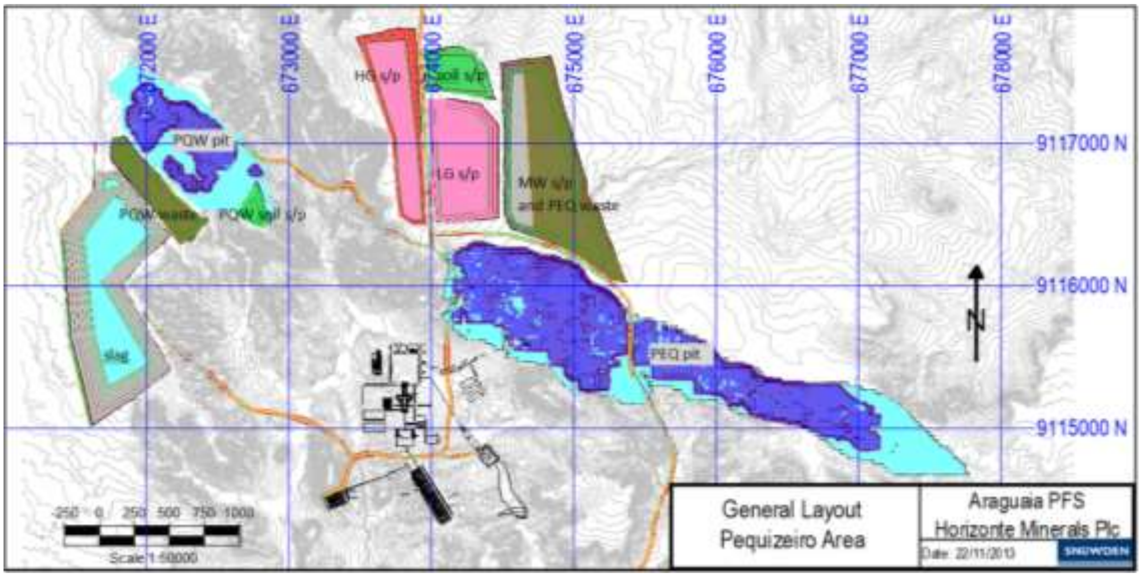
Figure 16.18 Jacintuga zone layout



16.10.4 Pequizeiro zone

The Pequizeiro zone comprises the large mining zones of PEQ near the plant site (Figure 16.19) as well as the smaller PQW deposit. Each pit has its own waste dump but share common ore stockpiles. The HG stockpile area is oversized relative to the current mine plan volumes. The extra space can be used for blending and homogenising the ores from the various zones. MW from the central pits is accommodated in the large PEQ waste dump. Space is limited at the PQW zone due to the slag dump occupying much of the available area, hence MW from PQW is accommodated in the PEQ dump site. The PEQ zone is constrained by a forest area along the north eastern flank of the pit.

Figure 16.19 Pequizeiro zone layout

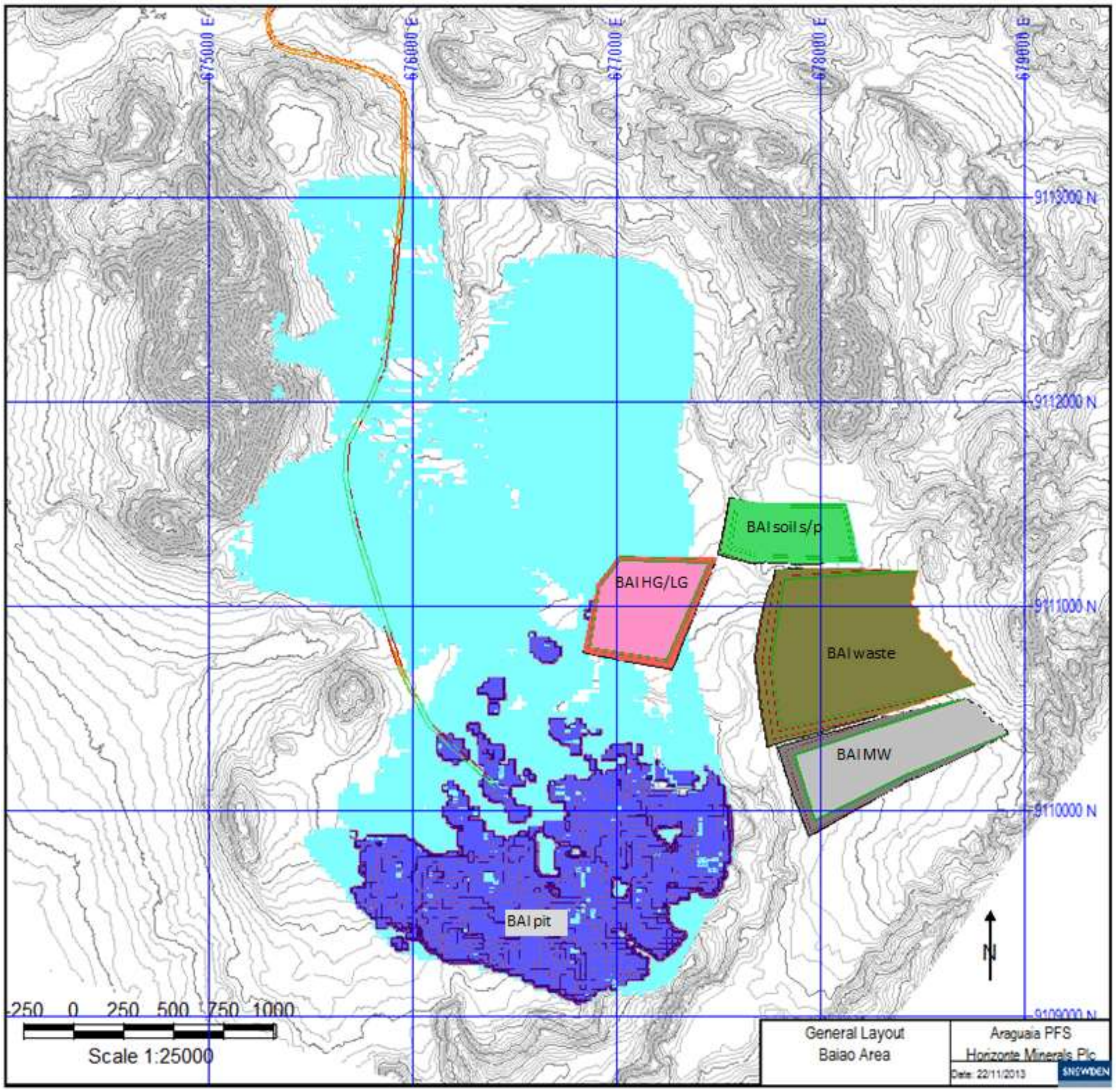


The PEQ pit and dumps are located near the headwaters of a northeast draining catchment (towards the forest). There is a potential sediment pond site in the gully east of the waste dump within an existing cleared area. The PQW area will be designed to drain southeast towards the plant where there will be sediment control structures in the catchments. The PQW waste dump drainage will be kept separate from the slag drainage.

16.10.5 Baião zone

The Baião zone comprises the BAI pit (Figure 16.20). This area is heavily impacted by the sterilisation zone. The waste dump is located northeast of the currently planned pit. Stockpile ore is assumed to be trucked to the central area stockpiles when required. Although this stockpile is shown overlaying potential ore zones, it is assumed that the stockpile can be reclaimed before the ore is recovered. MW is shown stockpiled adjacent to the waste dump.

Figure 16.20 Baião zone layout



Drainage out of the pit and waste dump can be controlled and collected in the creek to the south. However, if the pit were to expand, the pit and potential dump sites would span multiple creeks and require extra water management controls.

16.10.6 Stockpile and dump capacities

Stockpile and waste dump design capacities are shown in Table 16.17.

Table 16.17 Dump capacities (m^3)

	Required volume (Mm^3)	Design volume (Mm^3)
Vila Oito zone		
VOI / VOE waste	7.9	7.8
VOI / VOE MW	2.7	2.8
HG/LG stockpile	1.7	2.2
Topsoil VOE	1.7	1.7
Topsoil VOI/VOW	1.0	1.3
VOW MW	1.9	2.7
VOW waste	1.4	1.4
Jacutinga zone		
JAC stockpile	0.6	0.6
JAC MW	0.7	0.8
JAC waste	0.4	0.5
JAC soil	0.6	0.6
Baiao zone		
BAI stockpile	1.1	1.5
BAI soil	4.7	4.7
BAI waste	2.9	2.9
BAI MW	0.4	0.5
Pequizeiro zone		
PEQ waste / MW	9.9	10.1
LG stockpile	0.5	0.4
PEQ soil	1.1	1.1
HG stockpile	1.1	2.0
PQW soil	0.1	0.3
PQW waste	0.7	0.7

PEQ stockpiles are oversized as it is possible that a certain volume of additional material may be added from other mining areas. The stockpiles have not been modelled into separate ‘fingers’ by rock type or grade bins at this time. It is assumed that such divisions would be achieved by dumping in distinct flagged areas within the overall dump footprint. The volumes shown are the maximum LOM storage requirements and in operations, the actual volume on any stockpile would be less.

16.11 Trunk roads

The trunk roads are shown in Figure 16.16. The layout links all the zones to the plant. The roads mostly follow the higher ridge areas and avoid the gullies and the forests. They do not exceed 10% grade. In general, the terrain is flat and cuts/fills are minor (with the exception of the Southern trunk road which has a 60m elevation drop in gullies). Some culverts will be required. The trunk road has been designed at 20 m wide. This is more than adequate for two way on–highway truck traffic. However off-highway trucks may use the route and there may be other services (power lines) along the right of way. At this time it is assumed that highway trucks will be used to haul ore to the plant area from the Baião, Jacutinga and north zone stockpiles. All other haulage is to be undertaken by off-highway trucks.

16.12 Mining schedule

16.12.1 Basis

Software

The mining schedule was completed in Snowden's Evaluator scheduling software, which is a Mixed Integer Linear programming based tool. It is driven by the maximisation of net present value in the presence of physical quantity and grade constraints.

Resolution

The project was scheduled on the basis of panels. These panels are shown in Section 1.4. Within each panel, a number of "bins" are generated on the basis of rock type and nickel grade. Each bin within a panel is assumed to be consumed at the same rate.

Time horizons

The production schedule was completed in quarterly increments over the life of the project.

Constraints

Processing

A number of processing constraints were applied to the schedule:

- A ramp up period of 13 months has been used
  - Quarter 1 – 124 ktpq
  - Quarter 2 – 171 ktpq
  - Quarter 3 – 203 ktpq
  - Quarter 4 – 216 ktpq
  - Quarter 5 – 223 ktpq
  - Quarter 6 onwards – 225 ktpq.
- Grade constraints
  - Fe grade between 15.0% and 16.5%
  - Al_2O_3 grade between 4.0% and 5.5%
  - SiO_2/MgO ratio between 2.2 and 2.6.

Mining

As the project is driven by processing requirements, no hard mining constraints were applied. The expansive, shallow geometry is such that there is flexibility for both mining sequence and mining rate. Constraints were used to smooth:

- Overall mining rate.
- Mining rate in each mining zone.
- Ore haulage from the outlying deposits.

The mining of Baião was deferred until Year 3 and Jacutinga, Vila Oito East, Vila Oito, and Vila Oito West were deferred until Year 5 to allow for roads and infrastructure to be put in place over time and defer expenditure on this infrastructure to a period when positive cash flows are being generated.

An additional goal was set to minimise the number of simultaneously active mining pits and panels.

Within these rules, flexibility was provided as to how to distribute capacity amongst the pits over time.

There were no constraints imposed to limit the size of low grade stockpiles.

Schedule progression

A visual representation of the mining advance in each pit is shown in Figure 16.21 to Figure 16.27.

Figure 16.21 Baião mining advance

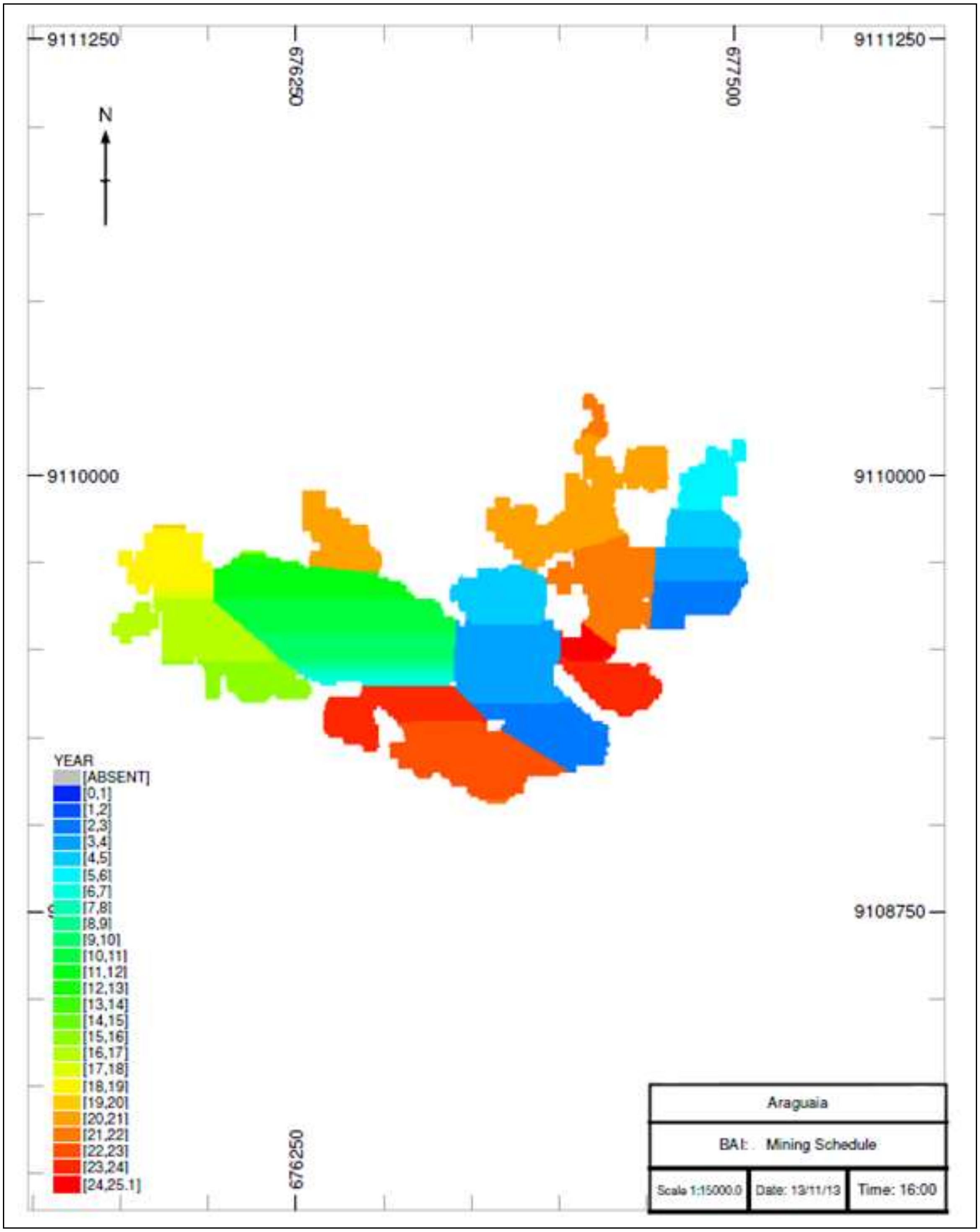


Figure 16.22 Pequizeiro mining advance

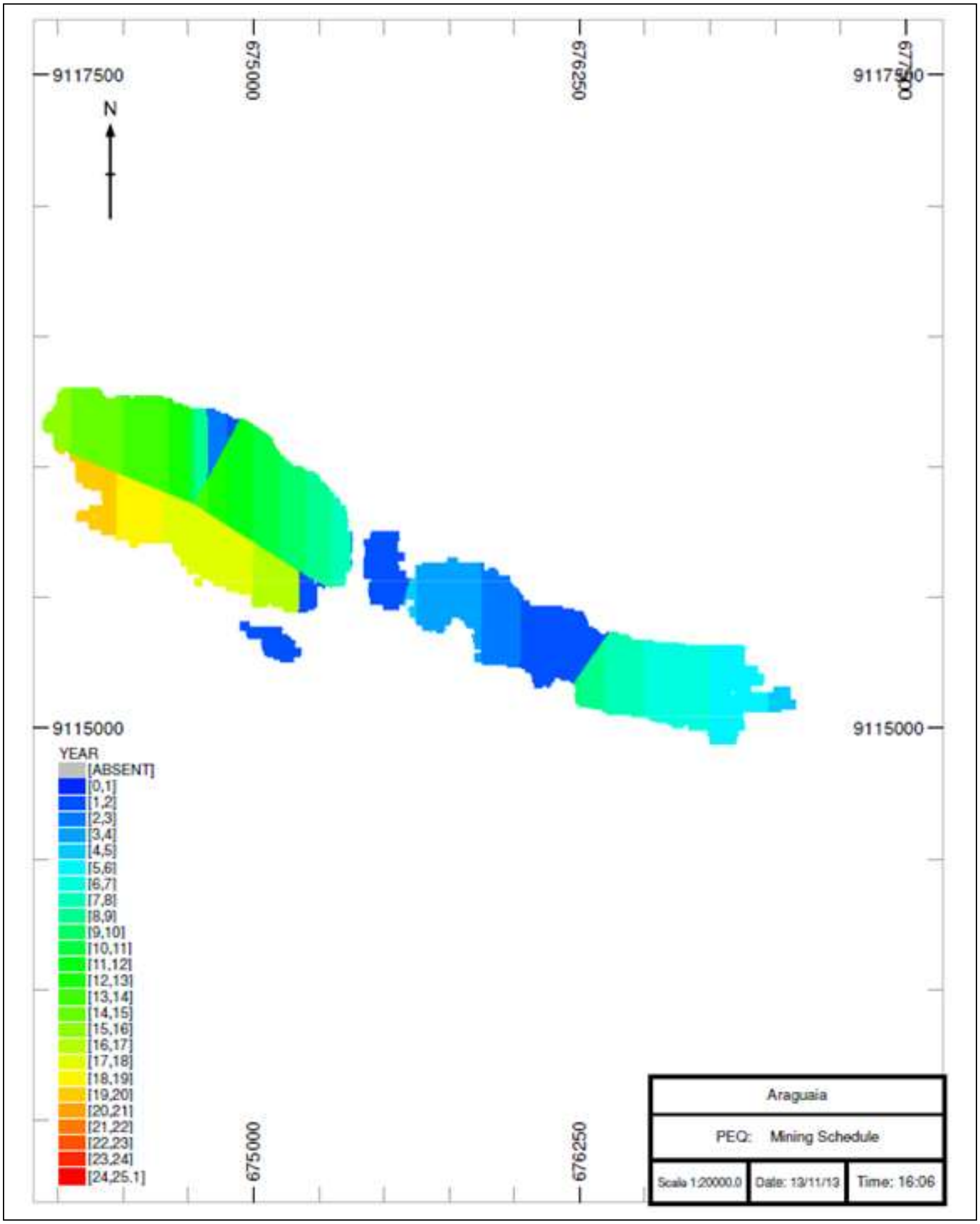


Figure 16.23 Pequizeiro West mining advance

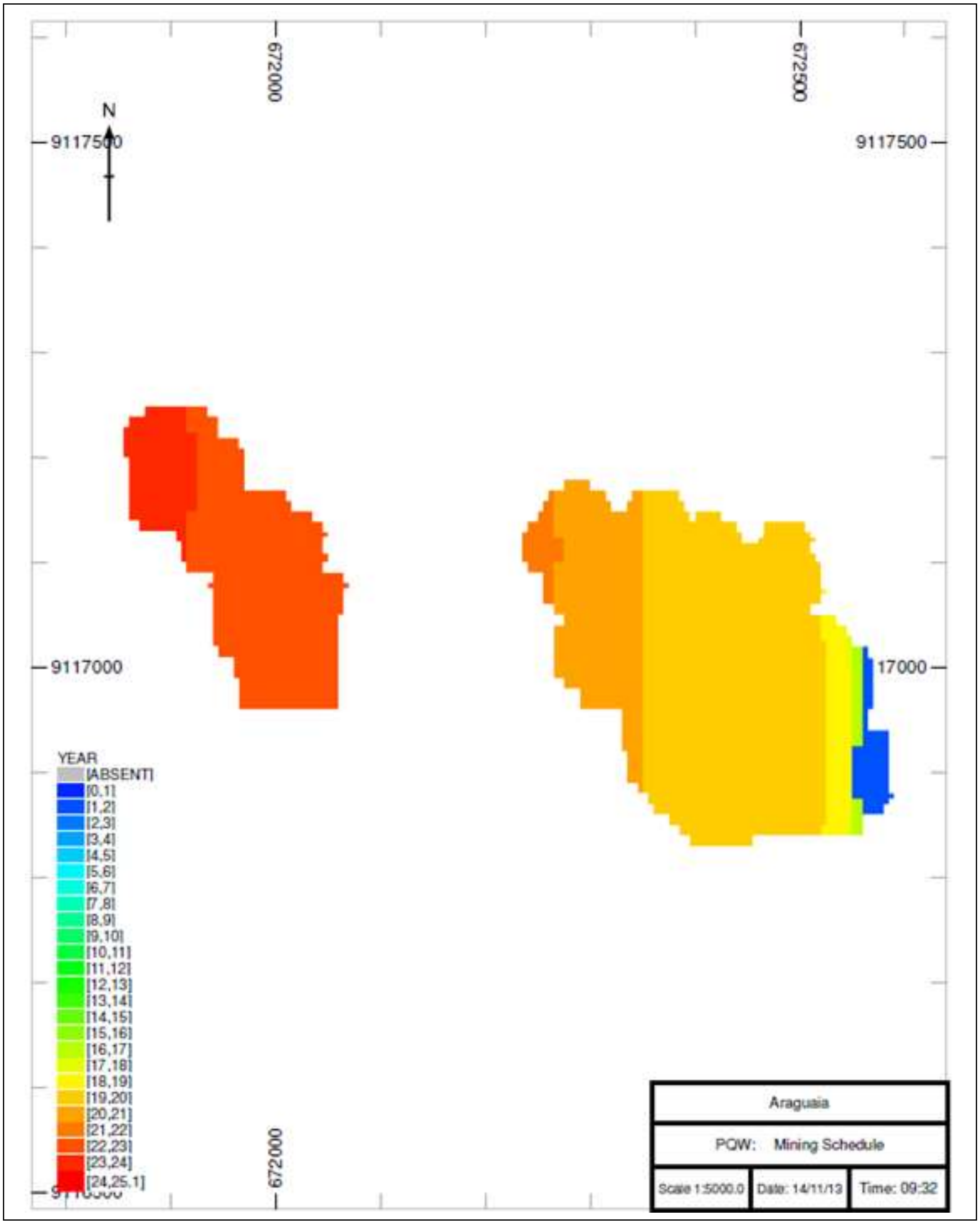


Figure 16.24 Jacutinga Mining Advance

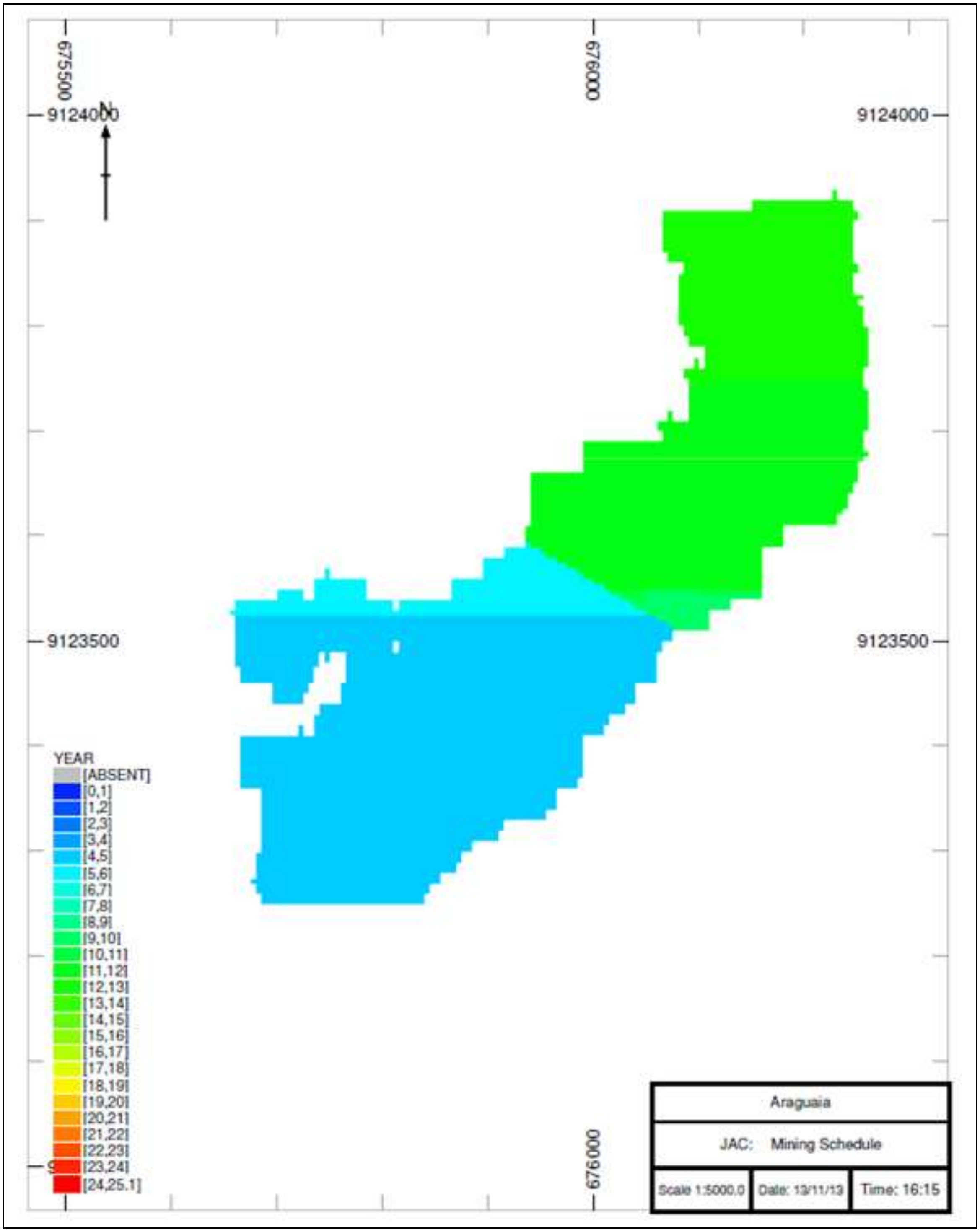


Figure 16.25 Vila Oito East mining advance

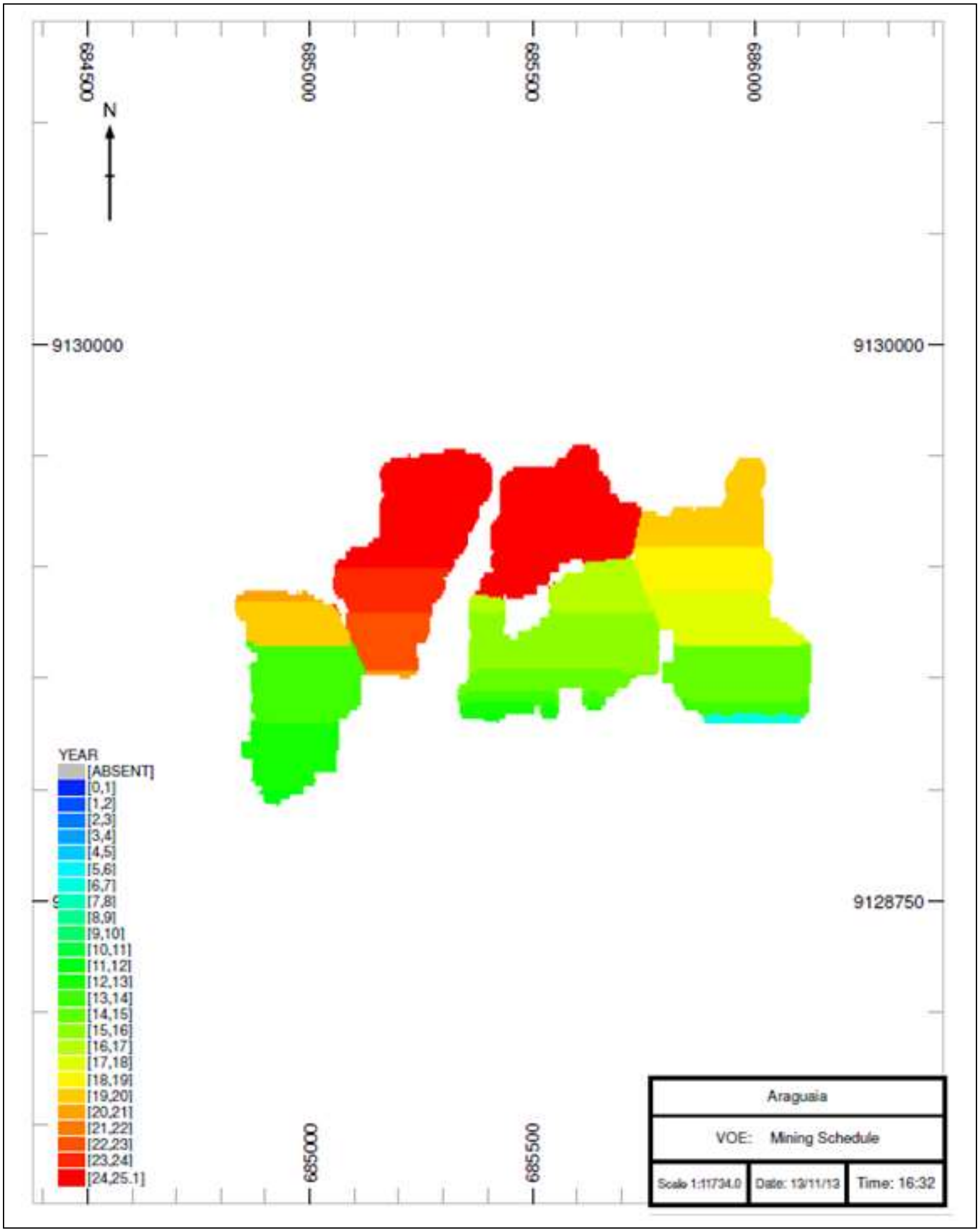


Figure 16.26 Vila Oito mining advance

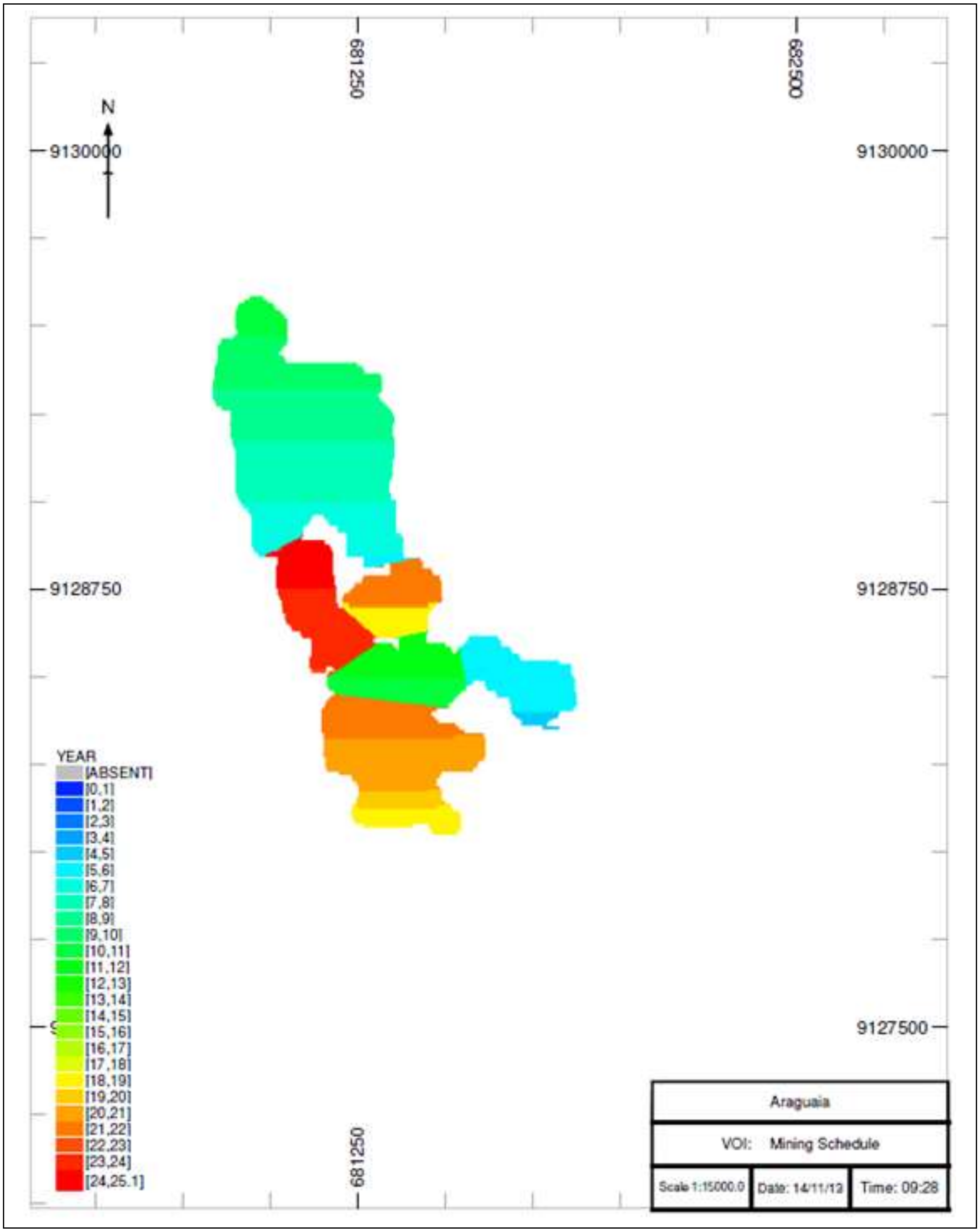
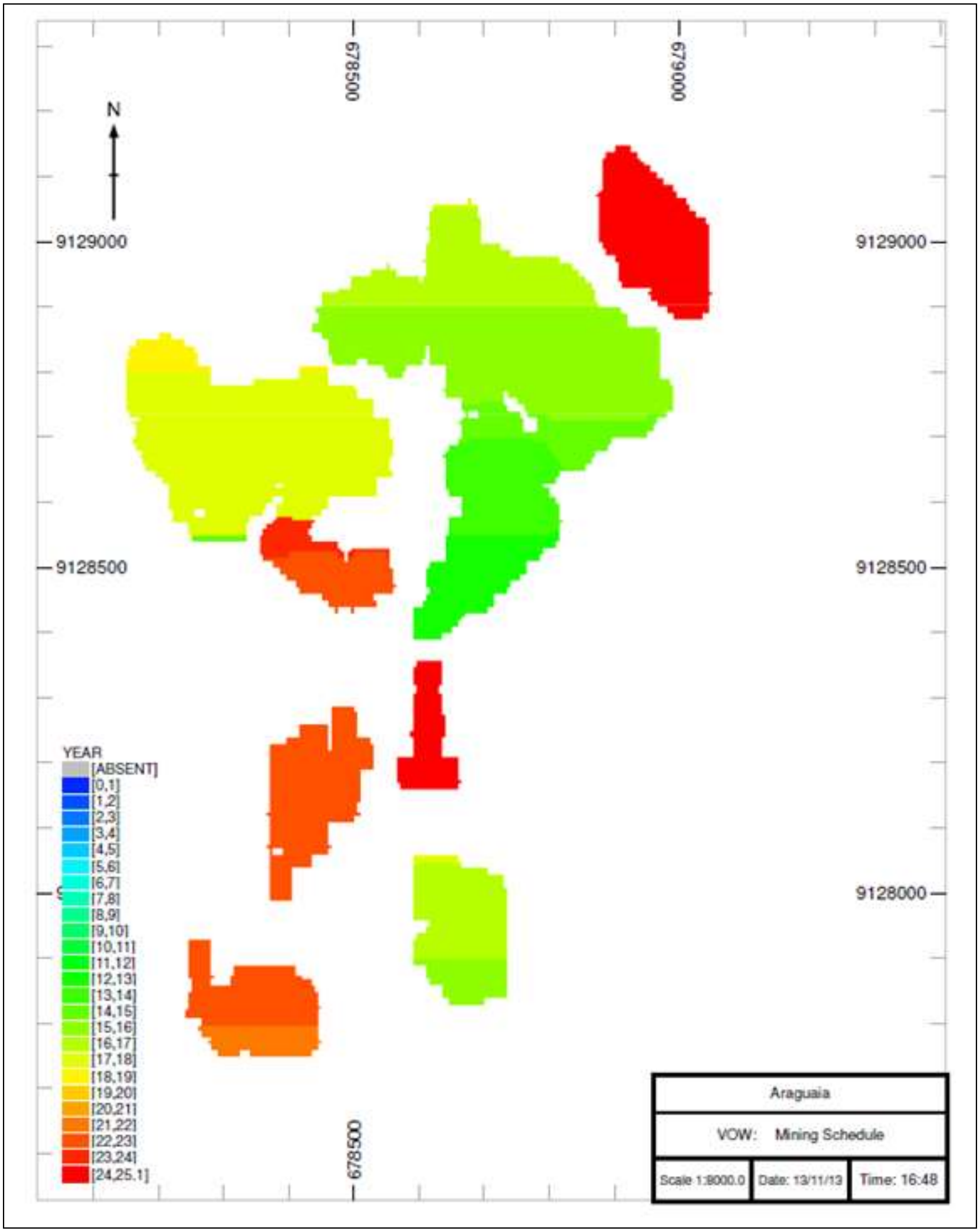


Figure 16.27 Vila Oito West mining advance

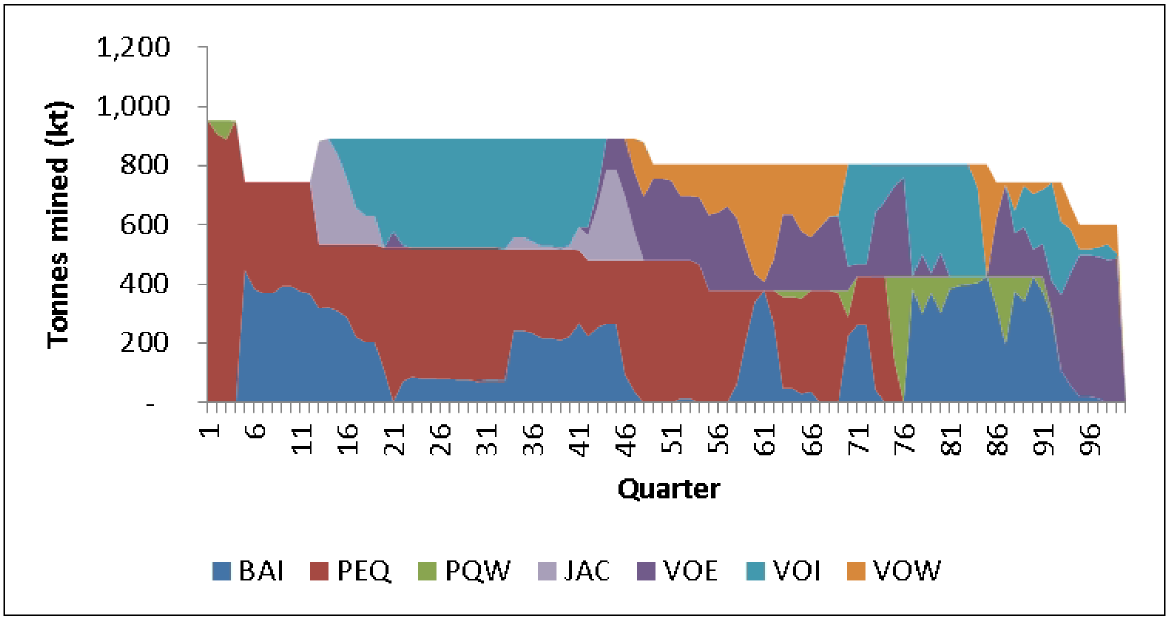


16.13 Schedule report

16.13.1 Mining schedule

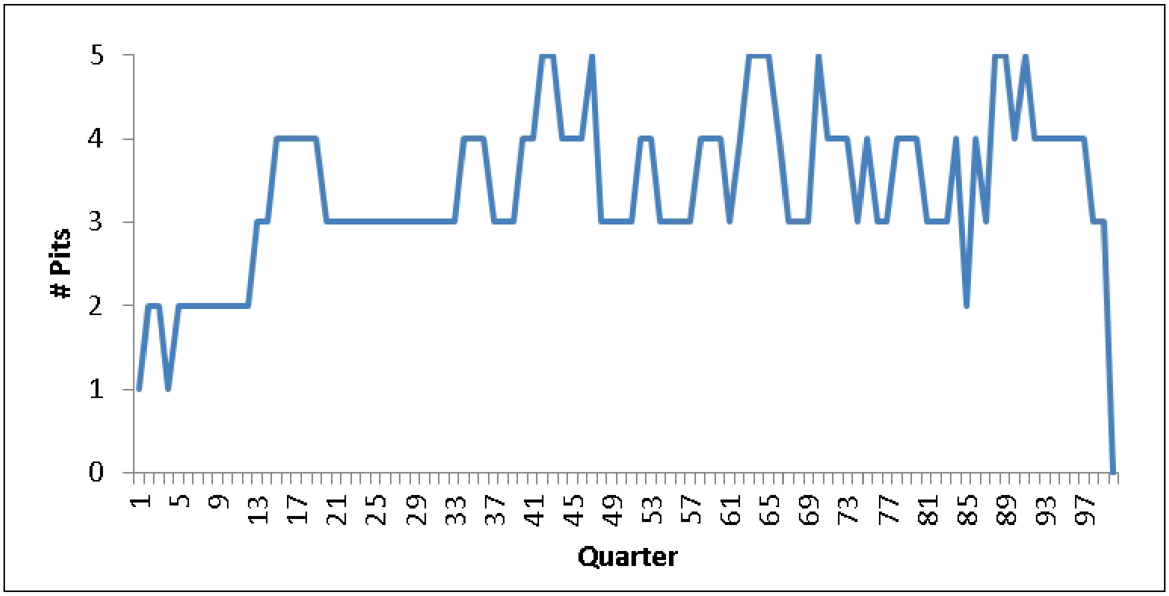
The overall Base Case open pit mining schedule is shown in Figure 16.28. Early mining focuses on Pequizeiro and Baião. When the outlying deposits come online the mining rate is split between each deposit and varies over time on a quarterly basis.

Figure 16.28 Mining schedule by deposit



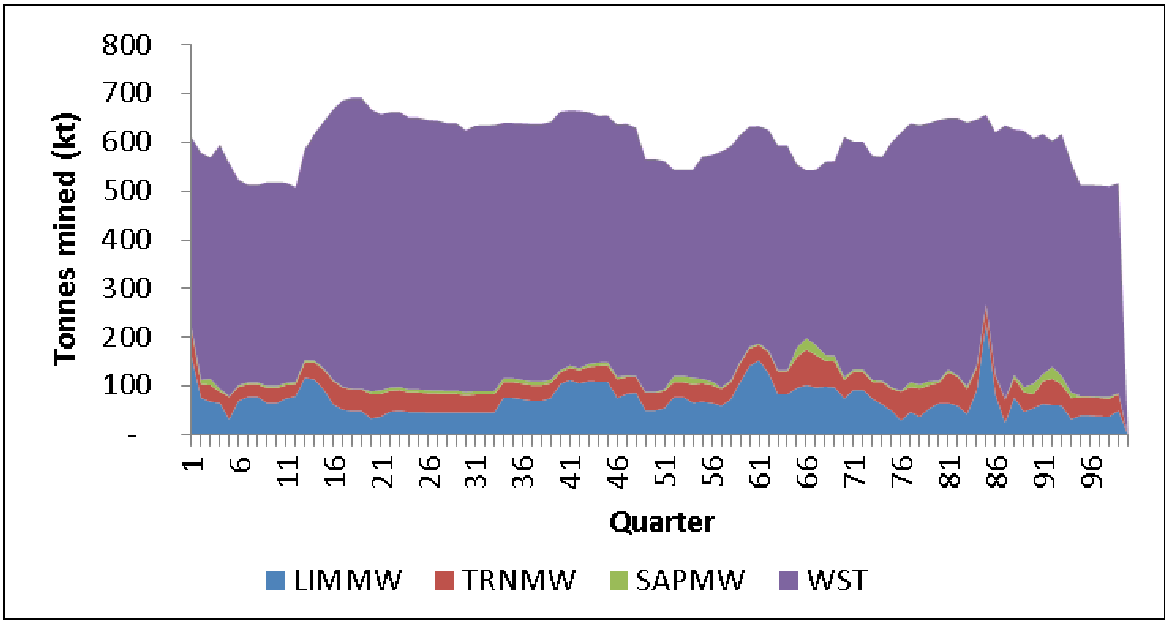
Throughout mining the maximum number of active pits in any quarter is five (Figure 16.29). However, during most of the life the number of active pits is usually three or four. Thus, the schedule provides for minimal movement of excavators within quarters.

Figure 16.29 Simultaneously active pits



The waste is broken into pure waste and mineralised waste. Mineralised waste is any material not processed that is above 0.8% Ni. This includes Inferred Resources. A schedule breakdown of the waste is provided in Figure 16.30.

Figure 16.30 Waste schedule



A summary of the overall mining quantities is shown in Table 16.18.

Table 16.18 Base Case mining schedule by deposit (dry mass ore and waste)

Year	BAI (kt)	PEQ (kt)	PQW (kt)	JAC (kt)	VOE (kt)	VOI (kt)	VOW (kt)	Total (kt)
1	-	3,701	111	-	-	-	-	3,812
2	1,572	1,408	-	-	-	-	-	2,981
3	1,529	1,452	-	-	-	-	-	2,981
4	1,238	901	-	1,228	-	191	-	3,559
5	740	1,388	-	316	-	1,124	-	3,567
6	242	1,849	-	-	63	1,412	-	3,567
7	320	1,772	-	-	-	1,475	-	3,567
8	302	1,790	-	-	-	1,475	-	3,567
9	798	1,275	-	107	-	1,387	-	3,567
10	872	1,202	-	26	15	1,453	-	3,567
11	1,014	945	-	647	187	772	-	3,567
12	397	1,529	-	615	716	-	298	3,555
13	15	1,911	-	-	1,034	-	260	3,220
14	15	1,692	-	-	960	-	553	3,220
15	612	906	-	-	730	-	971	3,220
16	745	730	43	-	645	-	1,057	3,220
17	69	1,421	28	-	839	2	860	3,220
18	749	763	99	-	415	1,021	172	3,220
19	44	958	702	-	1,110	405	-	3,220
20	1,361	-	344	-	165	1,350	-	3,220
21	1,584	-	120	-	-	1,434	81	3,220
21	1,328	-	376	-	651	76	603	3,035
21	1,434	-	150	-	473	843	73	2,973
21	211	-	-	-	1,589	434	379	2,613
	15	-	-	-	1,448	100	239	1,802
Total	17,208	27,594	1,973	2,939	11,041	14,956	5,544	81,256

16.13.2 Ore mining schedule

Ore is mined at rates above the specified processing capacity (Figure 16.31). This is to provide the highest grade/best blend material possible whilst mining through less desirable material to expose it. This is demonstrated in Figure 16.32 where, in many of the periods, the mined grades are outside process specification. This indicates that material is being stockpiled for later blending.

Low grade ore material is stockpiled for processing at the end of the project life. The maximum stockpile size (dry mass) gets to 1.5 Mt (Figure 16.33). The largest stockpiles are built in the Vila Oito zone.

Figure 16.31 Ore mining schedule

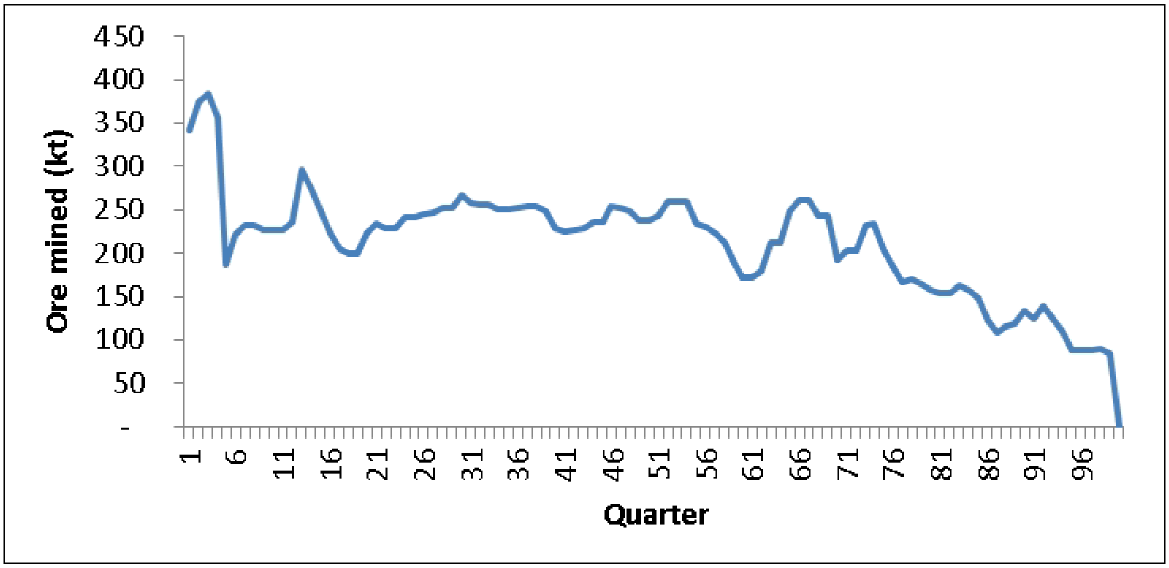


Figure 16.32 Ore mining grades

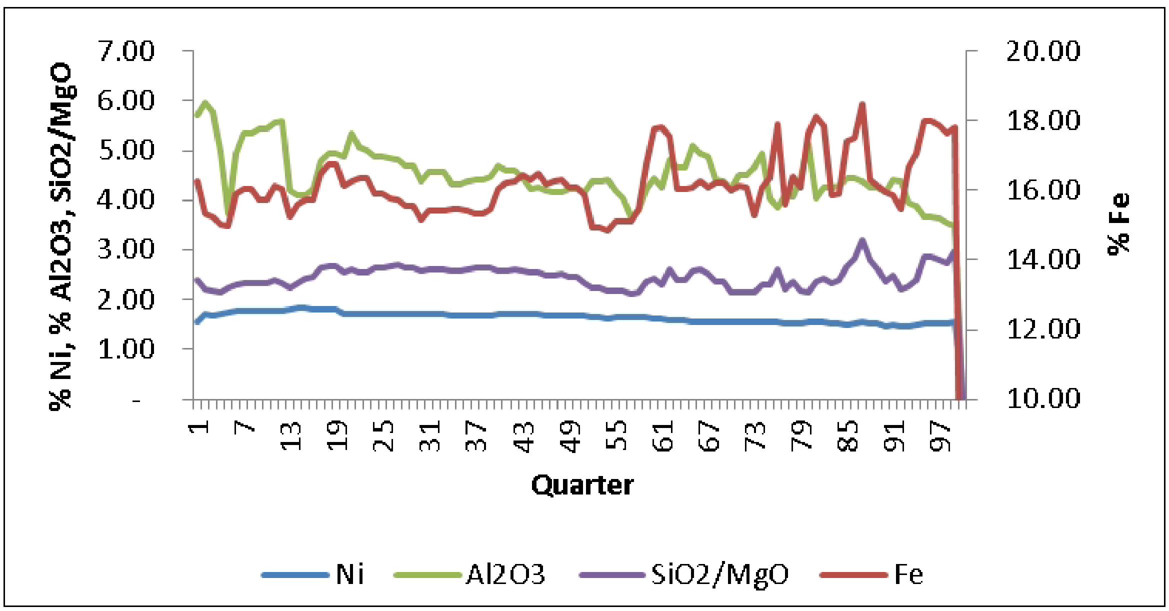
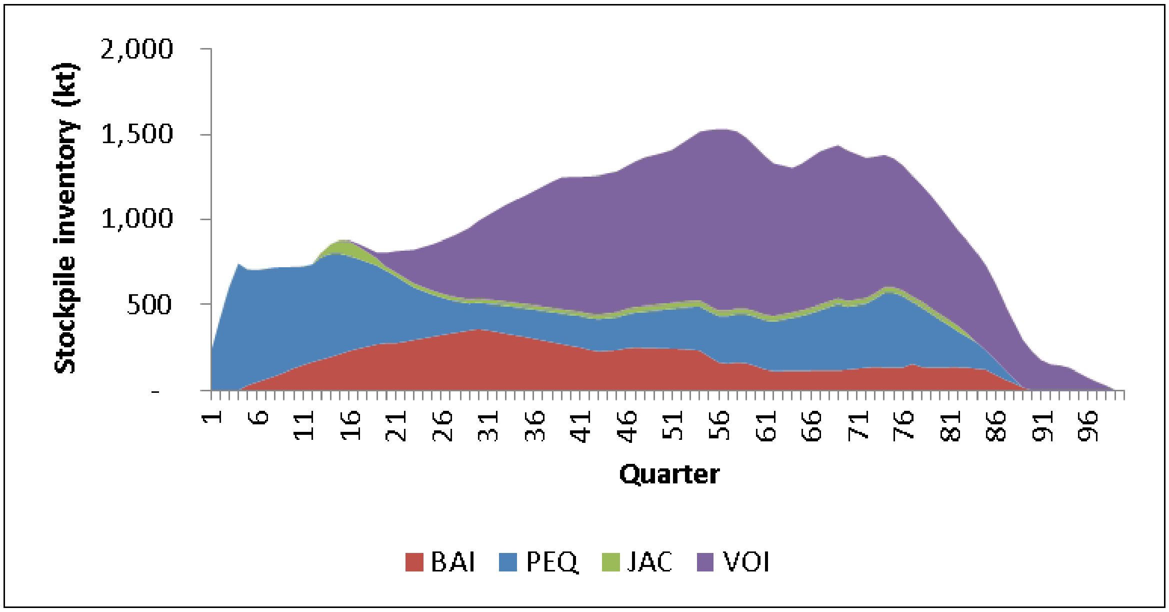


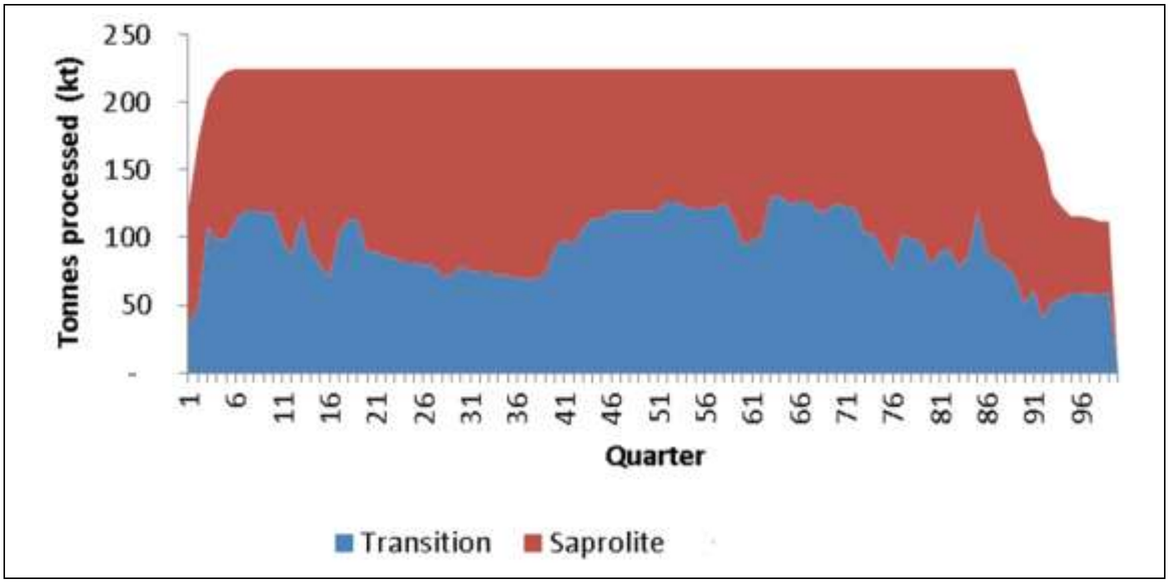
Figure 16.33 Stockpile schedule



16.13.3 Processing schedule

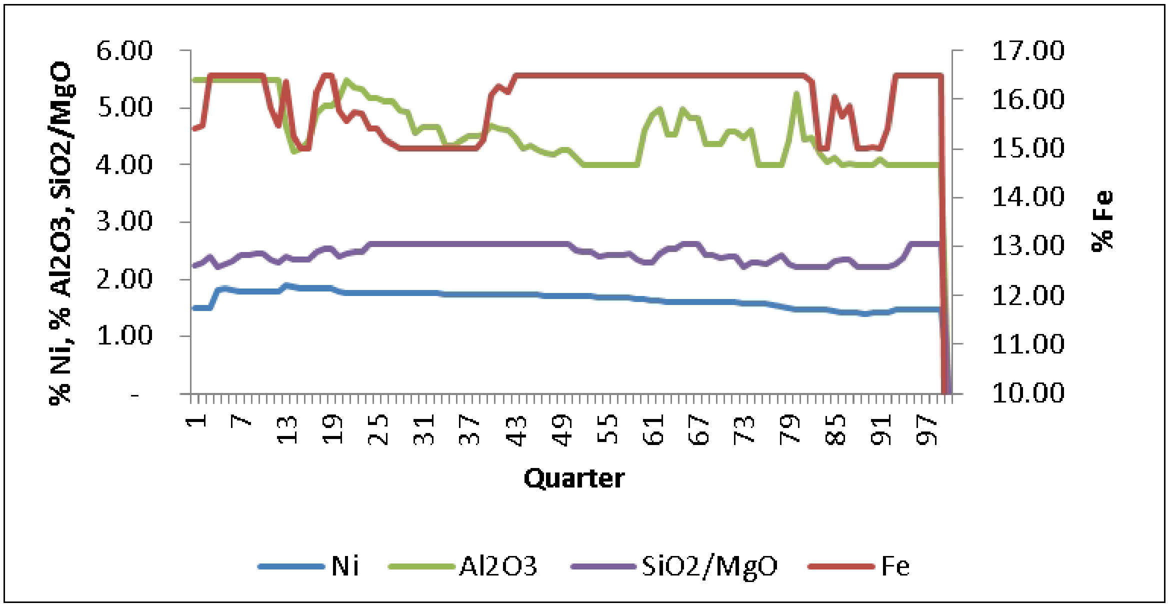
The designed processing capacity is met in all periods until the end of the project (Figure 16.34). The feed is quite evenly split between transition and saprolite rock types. Limonite contributes a small amount to the feed but could be considered opportunistic feed when the iron grades of other material processed during the period is below specification.

Figure 16.34 Processing schedule by rock type



The process feed schedule maintains grades within the specified tolerances in each quarter (Figure 16.35). However, there are a number of quarters where the lower or upper thresholds are met for each element or ratio, particularly iron grades which are at maximum levels in every quarter. Tight operational controls will need to be established to ensure compliance with targets over shorter time intervals.

Figure 16.35 Processing grade schedule



The annual processing schedule is presented in Table 16.19.

Table 16.19 Processing schedule

Year	Dry Feed (kt)	Ni (%)	Fe (%)	Al_2O_3 (%)	SiO_2/MgO
1	714	1.60	16.06	5.50	2.29
2	898	1.82	16.50	5.50	2.35
3	900	1.79	16.08	5.50	2.39
4	900	1.86	15.40	4.40	2.36
5	900	1.82	16.24	5.03	2.49
6	900	1.76	15.61	5.33	2.50
7	900	1.76	15.17	5.09	2.60
8	900	1.75	15.00	4.70	2.60
9	900	1.75	15.00	4.44	2.60
10	900	1.74	15.32	4.55	2.60
11	900	1.73	16.36	4.50	2.60
12	900	1.72	16.50	4.25	2.60
13	900	1.70	16.50	4.16	2.55
14	900	1.69	16.50	4.00	2.43
15	900	1.66	16.50	4.15	2.38
16	900	1.61	16.50	4.73	2.46
17	900	1.60	16.50	4.75	2.55
18	900	1.59	16.50	4.48	2.40
19	900	1.58	16.50	4.28	2.27
20	900	1.51	16.50	4.43	2.31
21	900	1.46	15.72	4.30	2.20
22	900	1.41	15.65	4.04	2.30
23	769	1.41	15.10	4.02	2.20
24	486	1.47	16.50	4.00	2.44
25	339	1.47	16.50	4.00	2.60
Total	21,206	1.66	16.01	4.59	2.44

16.13.4 Product schedule

A schedule of recovered nickel in the final product is shown in Figure 16.36 and Table 16.20. After ramp-up the production rate reaches approximately 15 ktpa. This is maintained for about two years. After this, production drops steadily to 10 ktpa by the end of the project life.

Figure 16.36 Recovered nickel schedule

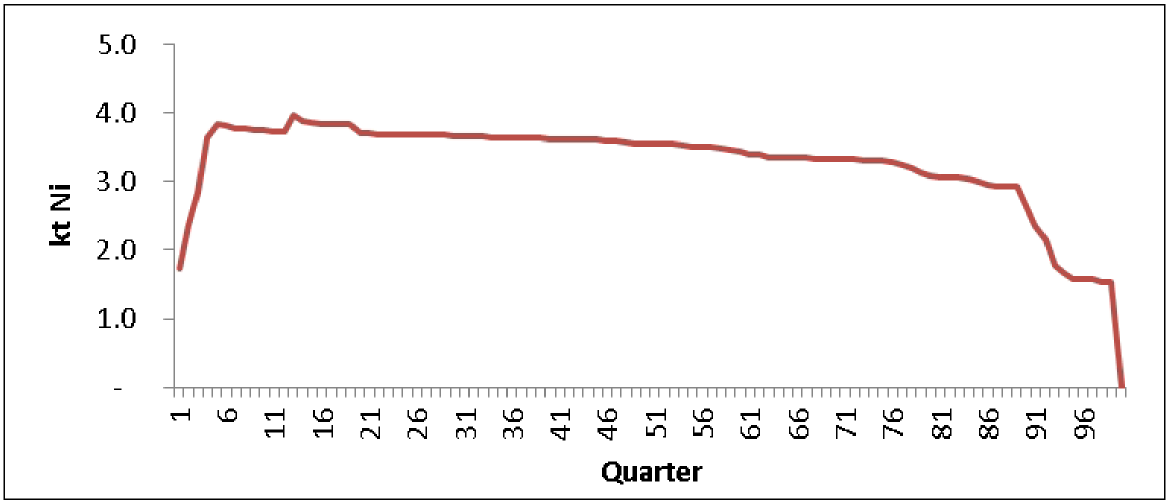


Table 16.20 Product schedule

Year	Nickel kt	Year	Nickel kt
1	10.6	14	14.1
2	15.2	15	13.9
3	15.0	16	13.5
4	15.6	17	13.4
5	15.2	18	13.3
6	14.8	19	13.2
7	14.7	20	12.7
8	14.7	21	12.3
9	14.8	22	11.8
10	14.6	23	10.1
11	14.5	24	0.6
12	14.4	25	0.5
13	14.2		
		Total	327.4

16.14 Cost estimate

16.14.1 Equipment supplier preference

Snowden does not recommend any specific equipment manufacturer. Where Snowden has nominated particular models of equipment, these should be considered to be representative of an equipment class only (i.e. size and productivity).

16.14.2 Scope

The scope for the mining cost estimate includes capital and operating costs for the following activities:

- Load and haul of ore, waste and top soil
- Grade control
- Re-handling ore from stockpiles ending at the ROM shed
- Mine services (dewatering, road construction and maintenance)

The costs are inclusive of:

- Fuel costs
- Maintenance costs (inclusive of GET, consumables, tyres, accidental damage and contingency)
- Labour costs (operators, maintenance personnel and mining management, technical and administration)
- Sampling costs
- Leasing costs / equipment purchase costs (including transportation)
- Excluded from this cost estimate is:
- Haulage roads (and haulage) from the plant to the slag dump
- Closure costs other than movement of material (covered in environmental cost estimate)
- Slag dump construction costs (covered in infrastructure cost estimate).

16.14.3 Accuracy

The cost model is a P50% estimate, implying a 50% probability of being too high or too low.

The mining cost estimate is supported by:

- quotations by vendors for key equipment from a large equipment dealer in Brazil
- estimation of haulage profiles for each source and destination over the life of the project
- The supporting evidence is sufficient such that Snowden is confident that this cost is estimate is accurate to within 25% for the given inputs.

16.14.4 Cost modelling methodology

The cost model is based on assigning all mining costs to units of equipment. For each piece of equipment, an hourly cost is derived through a number of calculations and benchmarks. All costs are converted to hourly equipment costs (including sampling cost, and administration costs). Where possible, hourly costs are supported by quotation from equipment vendors. The hourly cost is multiplied by the required machine hours to derive a total cost (which is grouped by year, source, destination and material type). The hourly costs are adjusted on the basis of quotes received (where available). Machine requirements are determined through the tonnage demand (and haulage profile estimation) and the relevant equipment productivities. More detail on this methodology is provided in the following sections.

16.15 Equipment selection

There are two fleets included the Base Case cost model:

- A mining fleet, for load and haul activities. The final destination for this fleet is the mine waste dump or stockpile.
- An ore haulage fleet to rehandle ore from the mine stockpiles to the ROM pad. For Pequizeiro articulated trucks are used due to the short haul; other deposits use highway trucks for haulage.

The primary pieces of equipment are shown in Table 16.21.

Table 16.21 Primary mining equipment

Function	Mining fleet	Ore haulage
Loader	48 t OW[5] excavator	50 t OW front end loader (FEL)
Haulage	35 t CP[6] articulated truck (ADT)	35 t CP on-highway truck
Grade control		16 t OW drill

The ancillary equipment used is shown in Table 16.22.

Table 16.22 Ancillary equipment

Equipment	Model	Tasks
Track dozer	39 t OW	Assisting loading, road construction and maintenance
Wheel dozer	29 t OW	Day works
Water truck	35 t CP	Dust suppression
Grader	21 t OW	Road construction, road/stockpile maintenance
Roller	16 t OW	Road construction/maintenance
Refuelling truck	35 t CP	Refuelling
Tyre handler	24 t OW	Tyre changes
Diesel pump	75 HP[7]	Dewatering

One representative machine model has been selected for each machine type for costing. In subsequent studies, the substitution of some smaller machines to provide operational flexibility to deal with a range of conditions may be considered.

[5] *Operating weight.*
[6] *Rated payload (capacity)*
[7] *Horsepower*

16.15.1 Off-highway truck selection

Snowden considered three candidate trucks: two articulated dump trucks (35 t CP and 40 t CP) as well as a 50 t CP rigid body truck. Trafficability analysis (Section16.2.2) showed that for the rigid body truck a step change (approximately 25%) in sheeting thickness would be required for roads due to the increased bearing pressure associated with fewer tyres. Given that transporting, dumping, and grading of the sheeting material is an additional expense (for the pits as well as roads, dumps and stockpiles), rigid body trucks were excluded from consideration.

Snowden selected the largest articulated truck available, the 40 t CP. The selection of this truck:

- Provides a six pass loading cycle with the selected excavator
- Achieve a manageable 15 to 20 trucks at peak requirement
- Minimise sheeting requirements.

16.15.2 On-highway truck selection

Nominally, a 35 t CP on-highway truck was selected for inter-pit haulage. For the target material movement this results in approximately 20 trucks. This number is suitable to fit in the confined space of the ROM pad, and provides flexibility in sourcing a range of materials from the various mine stockpiles for blending.

16.15.3 Excavator selection

The excavator was sized to match the mining trucks, the selectivity of mining, meet the production rate, and to provide flexibility of mining multiple areas simultaneously. The 48 t OW excavator provides a six pass cycle for the 35 t CP ADT and is capable of loading down to a flitch height of 2 m. The 1.2 m wide bucket is capable of mining to the selectivity required by the project. Six excavators are required to meet the production rate, providing some flexibility to mine multiple areas.

16.15.4 Front end loader selection

A 50 t OW FEL was selected to provide sufficient clearance to load the trucks comfortably. This class provides three pass loading into the highway trucks.

16.16 Mining management

Snowden considered both owner operator and contractor mining alternatives for each of the fleets.

Typically, contractor mining results in higher operating cost due to the expensing of equipment capital in the form of lease costs and the profit margin that is charged. The capital cost is limited to the cost of infrastructure and mobilisation.

A number of important factors were considered:

- **Capital cost:** is a common motivator for contractor mining. The processing capital costs of the Araguaia project are orders of magnitude higher than the potential mining capital cost and are as not considered as a major driver for this decision.

- **Operating cost:** contractors will charge a margin over their costs whereas owner-operator will not. This is often compensated partially by productivity and manning differences.
- **Grade control:** Whilst the operating cost of mining is less than 20% of the overall project operating cost, mining is a very important aspect of mining for nickel laterite projects. The requirement to blend a range of elements for the RKEF process circuit makes the control of blend imperative. It is difficult to incentivise a contractor to blend and manage grade in such a way that smooth feed to the plant is attained. An owner-operator, whilst perhaps being less efficient, will place greater emphasis of ore quality.

An estimate of the capital cost and operating cost for each fleet is shown in Table 16.23. Each of these estimates was built up from vendor quotes supplied for key equipment. No contractor quotations were sourced for this study, but are recommended prior to the Feasibility Study. Snowden have assumed a contractor margin of 20% on labour, maintenance, and management and leasing. No margin was applied to fuel costs as it is assumed that HM will procure this directly. A lease rate of 8% was applied for estimating ownership costs.

Table 16.23 Owner-operator versus contractor comparison

	Life Of Mine Capital Cost ($M)	Life Of Mine Operating Cost ($M)	Total Cost ($M)
Owner operator	74	407	481
Contractor	5	553	558

Either of these options is technically feasible. For this project, contractor mining was selected to minimise capital cost.

16.17 Productivity

Productivities are calculated on a tonnes per engine hour basis. This incorporates de-rating factors for:

- 50 minute hour which accounts for the conversion from peak productivity to average productivity
- Waiting and queuing time (stated in following sections where applied)
- 20% time allowance for unproductive time on primary equipment – time spent on rework or other activities that were not planned
- Low speed limits for trucks (stated in following sections where applied)

16.17.1 Loading

In total, four loaders are used for the various activities of the project. Table 16.24 summarises the key inputs.

Table 16.24 Loader inputs

Item	Units	Excavator	FEL
Bucket capacity	Loose cubic metres (lcm)	2.6	7.7
Bucket capacity	Wet metric tonnes (wmt)	5.0	11.4
Pass time	sec	35	60

Additionally, material properties are shown in Table 16.25.

Table 16.25 Material properties

Material	In-situ Dry Density (t/m^3)	Swell Factor	Moisture Content	Carryback
Limonite	1.61	15%	31%	15%
Transition	1.22	20%	31%	15%
Saprolite	1.30	20%	37%	15%

Table 16.26 summarises the average productivity of each loading unit over the life of the project, along with the material demand for that material and loader combination.

Table 16.26 Loader productivity

Material		LIM	TRN	SAP
Excavator	Total LOM dry mass (kt)	53,773	14,847	12,633
	Productivity (dmt/hr)	174	134	143
FEL	Total LOM dry mass[8] (kt)	0	9,358	11,847
	Productivity (dmt/hr)		181	166

16.17.2 Hauling

Two different truck types were used for the project. Their properties are shown in Table 16.27.

Table 16.27 Truck inputs

Item	Units	Off-Highway Truck	Highway Truck
Mass capacity	wmt	32.7	35.0
Empty machine mass	tonnes	32.4	16.0
Engine power	kW	333	321
Dumping time	sec	90	90
Spotting time	sec	45	45
Queuing time	sec	135	135
Travel time factor[9]	%	95	95

[8] *Considers that many of the ore tonnes are rehandled twice.*
[9] *Travel time considers the correction of speed to account for acceleration, braking for turns or passing. This increases the travel time and decreases the average fuel burn.*

Rolling resistances of 2% for major haul roads and 5% for finger roads in the pit were applied in determining speeds and fuel burns.

A speed limit of 20 km/h (loaded) and 30 km/h (unloaded) was applied in and around the operating pits. A maximum speed of 50km/h (loaded) and 60km/h (unloaded) was applied for Interpit haulage.

Haulage distances and gradients were estimated on the basis of major haulage roads designed by Snowden.

Speeds (and fuel burns) were calculated on the basis of the operating mass (loaded or unloaded), engine power, road gradient and rolling resistance. The calculations are derived from manufacturer rimpull curves for typical examples of the truck classes selected.

A summary of haulage productivity by deposit and material type (averaged over the life of the project) is shown in Table 16.28.

Table 16.28 Truck productivity by source and destination

Source	Destination	Dry Mass Moved (kt)	Productivity (dmt/hr)	One Way Haulage Distance (m)	Cycle Time (min)
Mining fleet					
BAI pit	BAI stockpile	3,524	33	1,500	16
BAI pit	BAI Mineralised waste	3,398	37	1,500	15
BAI pit	BAI waste	10,285	46	1,000	13
PEQ pit	PEQ stockpile	9,295	37	1,200	14
PEQ pit	PEQ mineralised waste	2,846	40	1,200	14
PEQ pit	PEQ waste	15,453	46	1,000	13
PQW pit	PEQ stockpile	379	33	1,500	16
PQW pit	PEQ mineralised waste	182	25	2,500	21
PQW pit	PQW waste	1,412	58	500	10
JAC pit	JAC stockpile	957	50	500	11
JAC pit	JAC mineralised waste	603	56	500	10
JAC pit	JAC waste	1,378	58	500	10
VOE pit	VOI stockpile	2,449	18	4,000	29
VOE pit	VOI mineralised waste	1,419	26	2,500	21
VOE pit	VOE waste	7,173	35	1,800	17
VOI pit	VOI stockpile	3,579	40	1,000	13
VOI pit	VOI mineralised waste	1,581	35	1,500	16
VOI pit	VOI waste	9,796	50	800	12
VOW pit	VOI stockpile	1,021	22	3,000	24
VOW pit	VOW mineralised waste	1,596	45	1,000	13
VOW pit	VOW waste	2,926	53	700	11
Reclaim fleet					
BAI stockpile	ROM pad	3,524	26	7,000	23
JAC stockpile	ROM pad	957	23	8,000	25
VOI stockpile	ROM pad	7,050	15	15,000	41
PEQ stockpile	ROM pad	9,674	62	400	8

16.17.3 Grade control drilling

Grade control drilling is currently assumed as 12.5 m by 12.5 m. This drilling will be a combination of reverse circulation and core drilling. Drilling will be from the surface to bedrock through the entire lateritic profile. This averages some 15 m but can be up to 40 m. Although the top layers are waste which do not need sampling, it is typically easier to drill off the solid iron cap surface and also there can be 'ore' bands worth stockpiling high in the profile within the 'waste'. Two metre sample intervals (on average) have been assumed. This is considered appropriate because in more cases ore selection can be made through the visual differentiation of rock types.

Grade control drilling will take place well in advance of mining activities so that the results can be included in the final mine dig plans. A productivity of 50 linear metres of drilling per shift has been calculated. Grade control drilling will occur on a campaign basis and be completed on day shift only.

16.18 Equipment requirements

16.18.1 Operating hours

Snowden's estimate of annual equipment operating hours is summarised in Table 16.29, taking account of mechanical availability, lost shifts (holidays, bad weather), shift downtime (breaks, pre-start, refuelling) and a maximum utilisation (non-mechanical downtime). Engine operating hours are referred to as SMU hours, as they are based on the service meter unit reading. Equipment costs have been calculated using SMU hours. For practical purposes, SMU hours can be equated to operating hours.

The availability of equipment has been lowered to account for a number of days per year (53) where the mine is not trafficable due to heavy rain, This accounts for the average number of days in the region with >10 mm daily rainfall, measured over the past 50 years.

Table 16.29 Operating hours

Equipment	Mechanical Availability (%)	Shifts Per Day	Shift Length (Hours)	Lost Shifts (Per Year)[10]	Shift Downtime (Per Shift)	Maximum Utilization (%)	Operating Hours (Hrs Per Year)
Excavator	86	3	8	186	1	60	3,283
FEL	86	3	8	186	1	30	1,641
Off-highway truck	86	3	8	186	1	90	4,925
On-highway truck	86	3	8	186	1	90	4,925
Drill	86	1	8	186	1	70	1,277
Track dozer	86	3	8	186	1	75	4,104
Wheel dozer	86	3	8	186	1	50	2,736
Grader	86	3	8	186	1	75	4,104
Roller	86	3	8	186	1	50	2,736
Tyre handler	86	3	8	186	1	50	2,736
Water truck	86	3	8	186	1	25	1,368
Fuel truck	86	3	8	186	1	50	2,736
Diesel pump	86	3	8	-	-	15	938

16.18.2 Fleet equipment requirements (load, haul, drill)

Minimum equipment requirements were determined on the basis of calculated productivities and operating hours. Requirements were then altered to smooth the fleet, with a subsequently adjusted utilisation.

[10] *Includes rain delays as well as public holidays.*

Table 16.30 Primary equipment requirements

Year	Excavator	FEL	Off-highway truck	On-highway truck	Drill
1	6	1	17	0	2
2	6	2	17	2	2
3	6	2	17	2	2
4	6	3	17	5	2
5	6	3	17	5	2
6	6	3	17	5	2
7	6	3	17	5	2
8	6	3	17	5	2
9	6	3	17	5	2
10	6	3	17	5	2
11	6	3	17	5	2
12	6	3	17	5	2
13	5	3	16	5	2
14	5	3	16	5	2
15	5	3	16	7	2
16	5	3	16	7	2
17	5	3	16	7	2
18	5	3	14	7	1
19	5	3	16	7	1
20	5	3	16	7	1
21	5	3	16	7	1
22	5	3	16	7	1
23	5	3	16	8	1
24	5	3	16	8	1
25	5	3	16	8	1

16.18.3 Ancillary

Ancillary equipment requirements were selected on the basis of a ratio to the load, haul and drill fleet size. Consideration was made to enable mining from at least four of the deposits simultaneously (separated by a distance).

Table 16.31 Ancillary equipment requirements

Year	Track Dozer	Wheel Dozer	Grader	Roller	Tyre Handler	Water Truck	Fuel Truck	Diesel Pump
1	6	1	2	1	1	2	1	6
2	6	1	2	1	1	2	1	6
3	6	1	2	1	1	2	1	6
4	6	1	2	1	1	2	1	6
5	6	1	2	1	1	2	1	6
6	6	1	2	1	1	2	1	6
7	6	1	2	1	1	2	1	6
8	6	1	2	1	1	2	1	6
9	6	1	2	1	1	2	1	6
10	6	1	2	1	1	2	1	6
11	6	1	2	1	1	2	1	6
12	6	1	2	1	1	2	2	6
13	5	1	2	1	1	2	2	6
14	5	1	2	1	1	2	2	6
15	5	1	2	1	1	2	2	6
16	5	1	2	1	1	2	2	6
17	5	1	2	1	1	2	2	6
18	5	1	2	1	1	2	2	5
19	5	1	2	1	1	2	2	5
20	5	1	2	1	1	2	2	5
21	5	1	2	1	1	2	2	5
22	5	1	2	1	1	2	2	5
23	5	1	2	1	1	2	2	5
24	5	1	2	1	1	2	2	5
25	5	1	2	1	1	2	2	5

16.19 Non-equipment requirements

16.19.1 Manning

Manning is separated into three key categories: operators, maintenance operators, and technical/admin staff.

Operators are allocated at the ratio of 1.3 operator hours per engine hour. The factor relates to the need for spotters, trainees and general labourers. The available hours for each operator (as it relates to engine hours) are determined from Table 16.32:

Table 16.32 Operator hours parameters

Item	
Shift roster	6 days on, 2 days off
Shift length	8 hours
Number of crews	4
Off days (weather/leave/holidays)	70 days
Lost time per shift	1 hour
Operator hours per year	1,433 hours

Additionally, 3.0 sampling person hours are required per drill hour.

Maintenance operators are calculated as a ratio to equipment hours. This factor is dependent on the size and type of equipment and is derived from Snowden internal benchmarks.

In addition, manning is allowed for management, technical and administrative staff. The staffing needs of the mining vary with time, as pits start and stop. The peak demand for each staff type of staff is shown in Table 16.33 to Table 16.39, below.

Table 16.33 Mining general management

Title	Peak demand
General manager – mining	1
Secretary	1
Data clerk	1

Table 16.34 Mining supervisory staff

Title	Peak demand
Mine Superintendent	1
Mine Foreman	3
Shift Boss	8
Mobile Maintenance Superintendent	1
Mobile Maintenance Foreman	2
Mobile Maintenance Planner	2
Secretary	1
Data clerk	1

Table 16.35 Mining technical services staff

Title	Peak demand
Technical services manager	1
Chief mine planning engineer	1
Chief geologist	1
Senior mining engineer	1
Senior Geologist	2
Mining engineer	3
Geologist	4
Senior Surveyor	1
Surveyor	3
Survey assistant	3
Dispatcher	3
Dispatch technician	1
Secretary	1
Data Clerk	1

Table 16.36 Mining safety and training staff (all mines)

Title	Peak demand
Manage Mine Safety	1
Mine safety officers	2
Mine training officers	2

Table 16.37 Manning requirements

Year	Operators	Maintenance	Technical, admin and management	Total
1	133	42	53	227
2	115	36	53	204
3	115	36	53	204
4	135	42	53	230
5	135	42	53	230
6	138	43	53	234
7	137	43	53	233
8	138	43	53	234
9	141	44	53	238
10	142	44	53	239
11	141	43	53	237
12	143	44	53	240
13	138	43	53	234
14	140	43	53	237
15	145	44	53	243
16	147	45	53	245
17	151	46	53	250
18	143	44	53	239
19	146	45	53	244
20	140	42	53	235
21	139	42	53	235
22	140	42	53	236
23	134	41	53	227
24	121	37	53	210
25	87	26	40	153

16.19.2 Fuel

Fuel burns were derived from quotations supplied by equipment vendors. For the haulage functions these were adjusted to account for haulage profiles. A summary of the fuel burns is shown in Table 16.38. A schedule of fuel requirement is shown in Table 16.39.

Table 16.38 Fuel burns

Equipment	Average Fuel Burn (L/hr)	Equipment Hours (hr x 1000)	Fuel Burn (ML)
Excavator	36.0	391.5	14.1
FEL	46.4	94.8	4.4
Off-highway truck	20.3	1,680.1	34.1
On-Highway truck	15.0	525.9	7.9
Drill	44.8	23.9	1.1
Track dozer	38.6	486.3	18.8
Wheel dozer	42.8	48.6	2.1
Grader	18.6	176.5	3.3
Roller	13.3	48.6	0.6
Tyre handler	15.3	48.6	0.7
Water truck	19.3	57.8	1.1
Fuel truck	19.3	66.2	1.3
Diesel pump	11.8	117.4	1.4
Total			90.8

Table 16.39 Fuel requirement schedule

Year	Fuel Required (ML)	Year	Fuel Required (ML)
1	3.7	14	3.9
2	3.1	15	3.9
3	3.2	16	3.9
4	3.6	17	4.1
5	3.6	18	3.8
6	3.7	19	4.0
7	3.7	20	3.6
8	3.7	21	3.6
9	3.7	22	3.7
10	3.8	23	3.5
11	3.7	24	3.3
12	3.9	25	2.4
13	3.8	Total	90.8

16.19.3 Explosives

No explosives are planned for this project. Hard ripping will be used where necessary.

16.19.4 Sampling

Grade control drillholes are to be sampled at 2 m intervals for all ore and mineralised waste. A schedule of sampling requirements is shown in Table 16.40.

Table 16.40 Samples required

Year	Samples Required (x1000)	Year	Samples Required (x1000)
1	5	14	4
2	3	15	4
3	4	16	4
4	4	17	5
5	3	18	4
6	4	19	3
7	4	20	3
8	4	21	3
9	4	22	3
10	4	23	3
11	4	24	2
12	4	25	1
13	4	Total	88

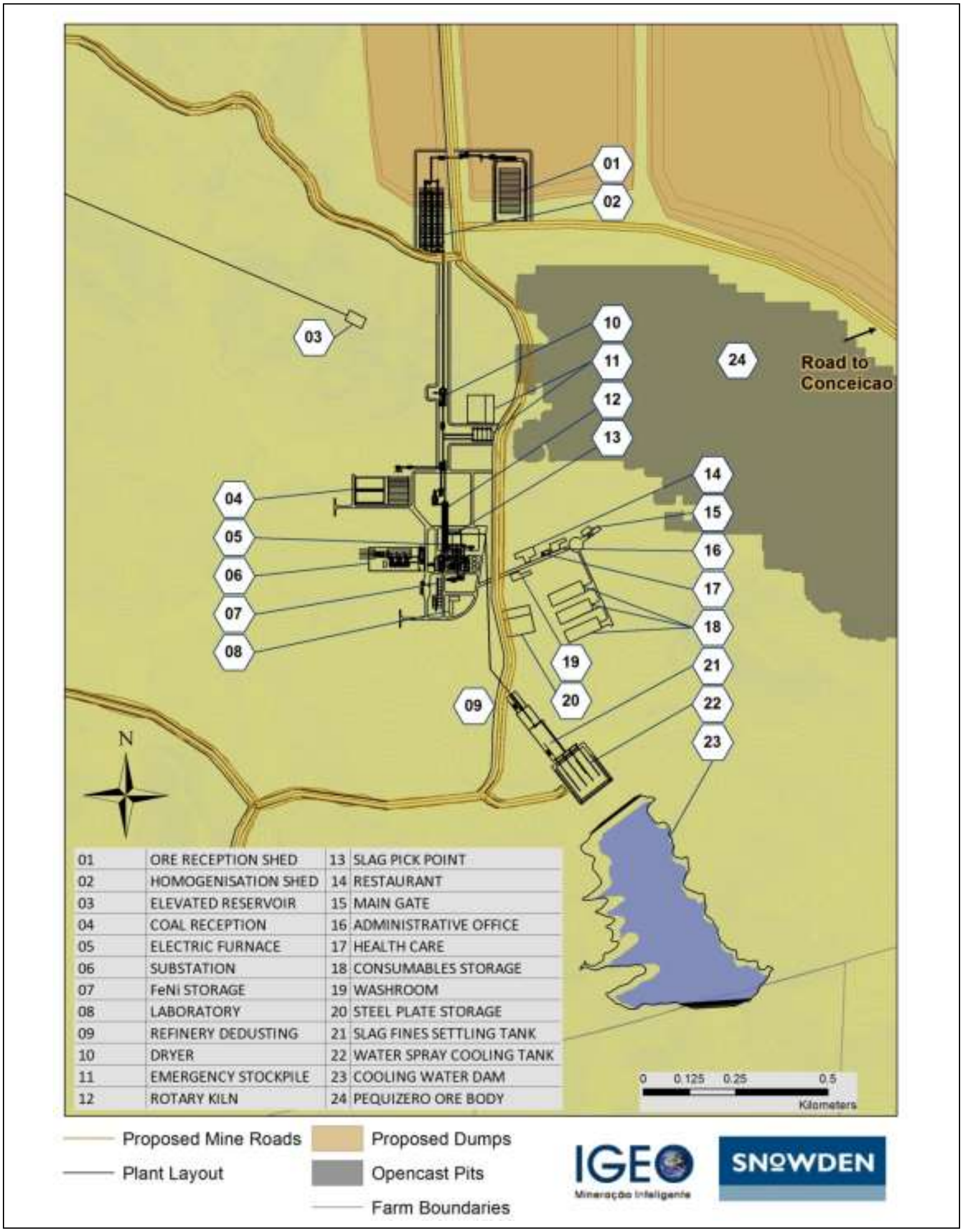
17 Recovery methods

17.1 Introduction

The Base Case plant will have one single processing line from ore receipts through to granulation of the refined Fe-Ni final product and will be capable of processing 0.9 Mtpa (dry) at full capacity to produce 15 ktpa of Ni, utilising the RKEF process. For design purposes the nickel grade adopted is 1.80% Ni.

The plant will have one primary and one secondary crushing station, one ore homogenisation facility and one production line comprised of one rotary dryer, a tertiary crushing station, a rotary kiln, a 50 MW (nominal) smelting electric furnace and a refining ladle furnace, coupled to a metal granulation and metal conditioning area and a metal recovery from refining slag plant.

Figure 17.1 Plant site location and orientation



Source: IGEO

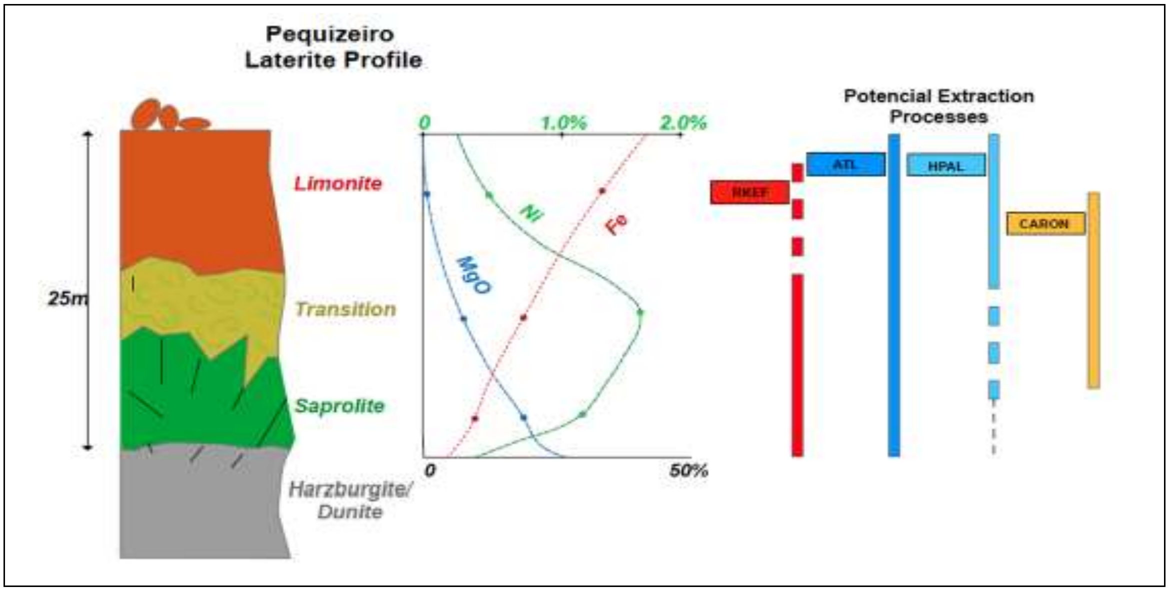
17.2 Process Selection

Two metallurgical processes for the treatment of Ni laterites were considered. These are:

- Pyro-metallurgical processes
- Hydro-metallurgical processes

The general features of the typical laterite profile and metallurgical processing categories are illustrated in Figure 17.2. Typically, limonite material has been treated hydro-metallurgically (High Pressure Leach or HPAL), while the saprolite and transition materials have been treated either hydro-metallurgically (e.g. pressure leach of the limonite followed by an atmospheric leach of the saprolite) or pyro-metallurgically, normally by the RKEF process.

Figure 17.2 Laterite profile and potential metallurgical processes



Source: Audet, M A, et al 2012

As part of the metallurgical testwork on the Araguaia laterite ore, both pyro-metallurgical and hydro-metallurgical test work was undertaken. This testwork subsequently confirmed the preferred suitability of the conventional RKEF process for the treatment of the Araguaia ore to produce Fe-Ni, and this process was adopted for the PFS by HZM.

17.3 Process description

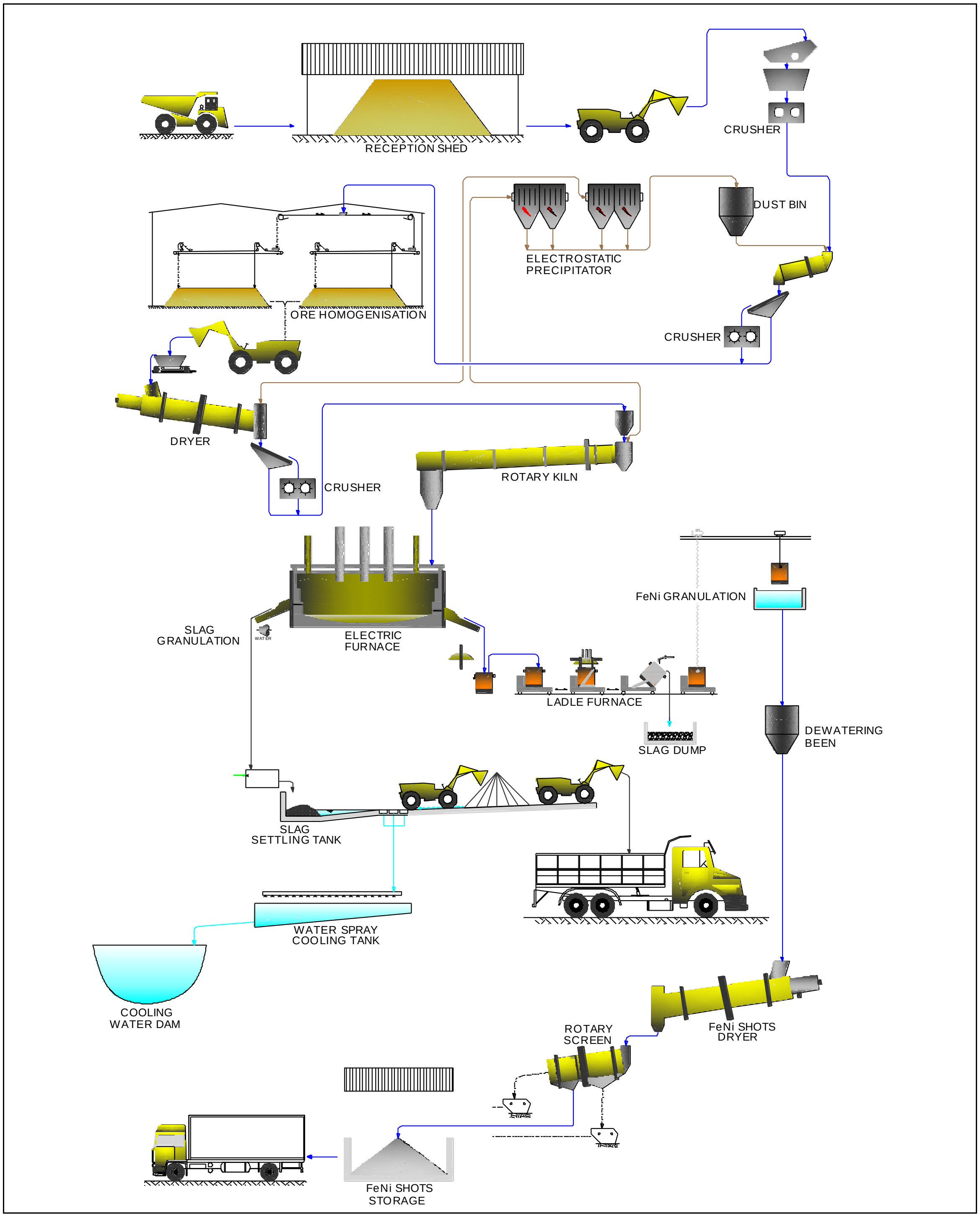
As part of the PFS, two RKEF plant capacity scenarios were evaluated:

- Base Case: A single line 0.9 Mtpa RKEF installation, producing approximately 15 ktpa nickel as Fe-Ni
- Option 1: Twin line RKEF installations of 1.35 Mtpa each producing a total of approximately 40 ktpa Ni as Fe-Ni

The process description for the Base Case, together with block flow diagrams and key design criteria of each process section is presented here to provide a clear understanding of the project scope and main characteristics of the metallurgical process. The main sections for the Project are depicted in Figure 17.3.

The Process Design Criteria (PDC) was developed from the throughput data for the 0.9 Mtpa Base Case. Option 1 twin line RKEF plant was also developed for comparative purposes and subsequent evaluation to identify the preferred option (Base Case 0.9 Mtpa).

Figure 17.3 Process flow diagram



Source: IGEO

The initial stages encompass ROM ore reception and blending of ore to meet metallurgical processing requirements, ore preparation, where the ore is sized to match the subsequent metallurgical process requirements. The ore is then homogenised, partially dried and fed to the kiln with the addition of a reductant material. In the kiln, the ore is completely dried, calcined to remove chemically combined moisture, and partially pre-reduced. Kiln dust is recycled to the process before the secondary crushing stage. Calcine from the kiln is then transferred into an electric furnace for the separation of the metal and slag at high temperature. The metal is conveyed in ladles to the refining stage. The refining oxidised slag is granulated with water, while the reducing slag is transported molten and disposed of in a specific site. The rationale for adopting these treatment methods is explained later in this document. The final Fe-Ni product is granulated with water, screened, dried and stockpiled prior to dispatch to the market.

The process flow diagram is described in subsequent sections.

17.3.1 ROM ore reception

The ROM will be stockpiled for the purpose of initial blending of different ore types to establish and maintain planned plant feed a pre-established metallurgical characteristic for feeding the plant. ROM will be reclaimed by excavator and loaded onto trucks, which will haul the material to the ore reception shed, from where it will be rehandled by front-end loader for feeding onto a shaking grizzly. This concept has been developed in order to maintain ROM flexibility and achieve a crushing section availability of 75% whilst ensuring more compatible flows of ore and dust on the conveyor feeding the contact drum.

17.3.2 Ore crushing and homogenisation

The shaking grizzly will feature a 500 mm x 500 mm gap. Oversize material will be removed with the aid of the same front-end loader used for feeding the plant, or otherwise, discharged on the floor by the shaking effect of the grizzly. The undersize (-500 mm) stream will be stored in a 20 m^3 chute from which it will be extracted by a conveyor belt for discharge into a mineral sizer type crusher of 250 mm gap. The crushed material will be discharged onto a conveyor belt and then sampled by means of a shuttle conveyor before being discharged onto the feed conveyor of the dust mixing drum. .

The rotary-type dust mixing drum will be used for providing contact between fresh ore and recycled rotary kiln dust and thus will promote a degree of agglomeration. This concept provides an inexpensive means of handling the kiln off-gas dust and has another major benefit in preventing the adherence of sticky ore fines onto downstream conveyor belts. The recycled dust may be wetted, depending on the fresh ore moisture level, prior to drum addition. Dust generated from the contacting process will be conveyed to a baghouse and recycled to the contacting drum.

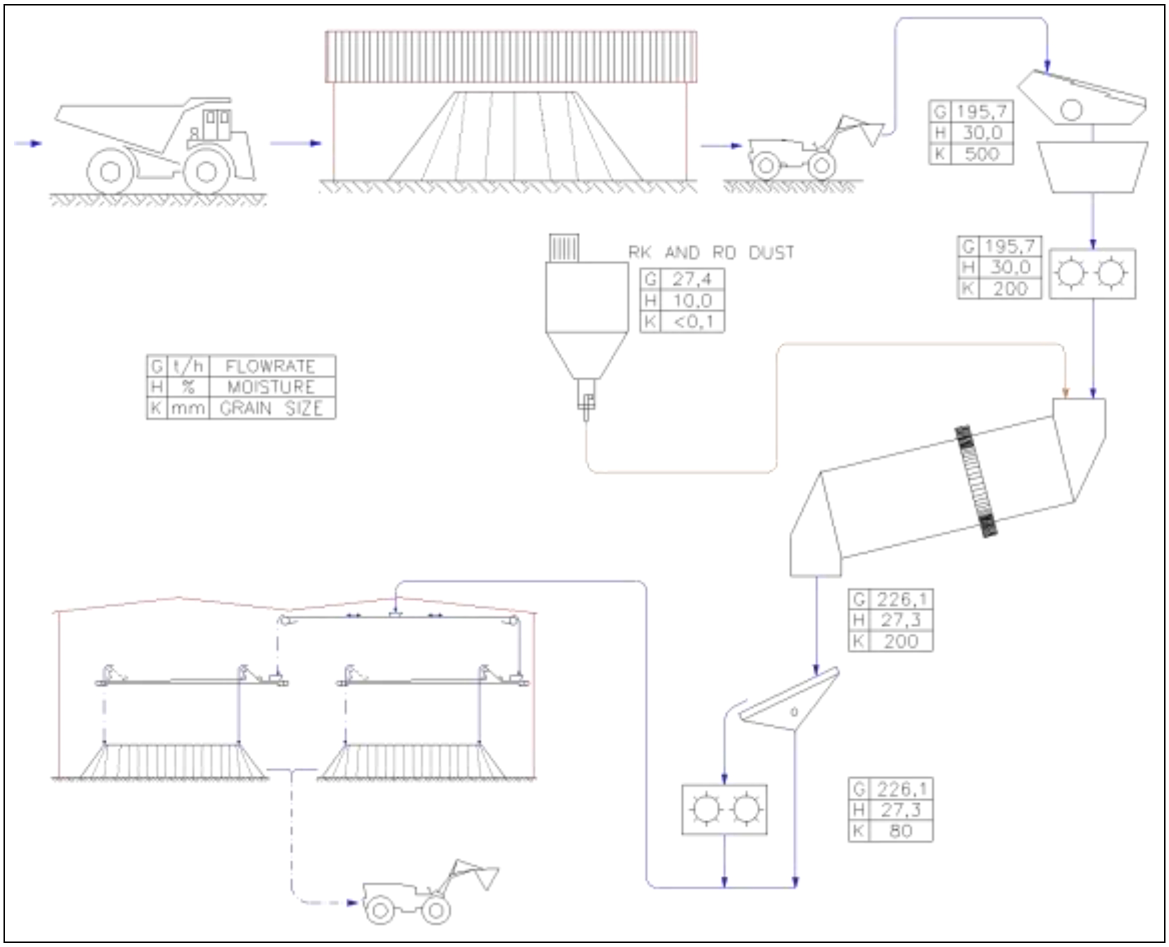
Ore mixed with dust will be conveyed to a vibrating screen with 80 mm mesh opening, operating in open circuit with a crusher. The oversize fraction will be conveyed to, and crushed by, a roll crusher of 80mm gap. The screen undersize and the crushed product will be combined and conveyed to an ore stacker for deposition onto a homogenisation stockpile. The storage yard will feature two piles, each with a capacity of approximately 22 000 wet tonnes of ore which is sufficient for eight days total plant feed capacity. When one pile is being created the other is feeding the plant to allow for a continuous operation.

The homogenisation of ore is a critical process function in order to minimise variation and provide consistent feed to the smelting furnaces. In the case of the Araguaia ore, this feature is particularly important due to the anticipated smelting characteristics of the slag to be produced in terms of liquidus temperatures. Key criteria of the Ore Preparation section are defined in Table 17.1.

Table 17.1 Key criteria of crushing and dust recycling – Base Case

Item	Unit	Value
Fresh Ore feed rate – design	dry mt/hr	137.0
Fresh Ore feed rate – nominal	wet mt/hr	195.7
Moisture content	%	30
Dust recycling – storage to contactor drum	dry mt/hr	27.4
Dust moisture content - maximum	%	10
Dust recycling – storage to contactor drum	wet mt/hr	30.4
Primary crusher type	-	Sizer
Secondary crusher type	-	Toothed double roll
Dust recycling equipment	-	Contacting drum
Secondary screen	-	Vibrating rubber lined
Ore homogenisation capacity	wet mt	2 x 22 000
Ore handling for homogenisation		Stacker - reclaimer

Figure 17.4 Primary and secondary crushing and homogenisation flowsheet



Source: IGEO

17.3.3 Ore drying and tertiary crushing

Reclamation of wet ore from the homogenising piles will be by means of a single reclaimer and conveyed to the rotary dryer feed bin. From here it will be extracted from the bin by a weigh-belt feeder, and then transferred to the dryer feed chute. The effect of the recycled dust mixed in the ore will ensure that the material flows readily through the chute, which is to be designed so as to promote a smooth and adequate material flow as well. A steady feed rate thereby improves the operational control of the drying process, which is a key requirement for optimising energy utilisation, as well as facilitating material agglomeration.

The discharge from the dryer will feed through the tertiary screening and crushing station and thereafter will feed into the kilns feed bin. In order to maintain availability for the kiln in the event of maintenance, an emergency stockpile of dried and crushed ore will be incorporated. The reclaiming system will be by means of front-end loader and hopper for recycling to the kiln feed bin.

The dryer will feature a co-current configuration with a combustion chamber designed for burning pulverised coal, and having fuel oil for backup and start up.

The dryer will be designed so as to optimise the agglomeration process. Therefore it will feature a retention dam at the discharge to attain the optimum filling degree. Heat transfer will be controlled and the off-gas temperature will be such that it avoids condensation in the electrostatic precipitator, which will be used for collecting dust present in the off-gas.

The target moisture of the dryer product will be 18% to 20%.

Dried ore from the dryer will then be screened on a vibrating screen with 30 mm mesh gauge. The oversize will be crushed in a roll crusher with 30 mm setting. This top size of 30 mm has been found to be a good criterion for optimising heat transfer and reaction kinetics in the kiln while trying to minimise dust losses in the kiln off-gas.

Key criteria of this section are defined in Table 17.2 and Table 17.3, as well as Figure 17.5.

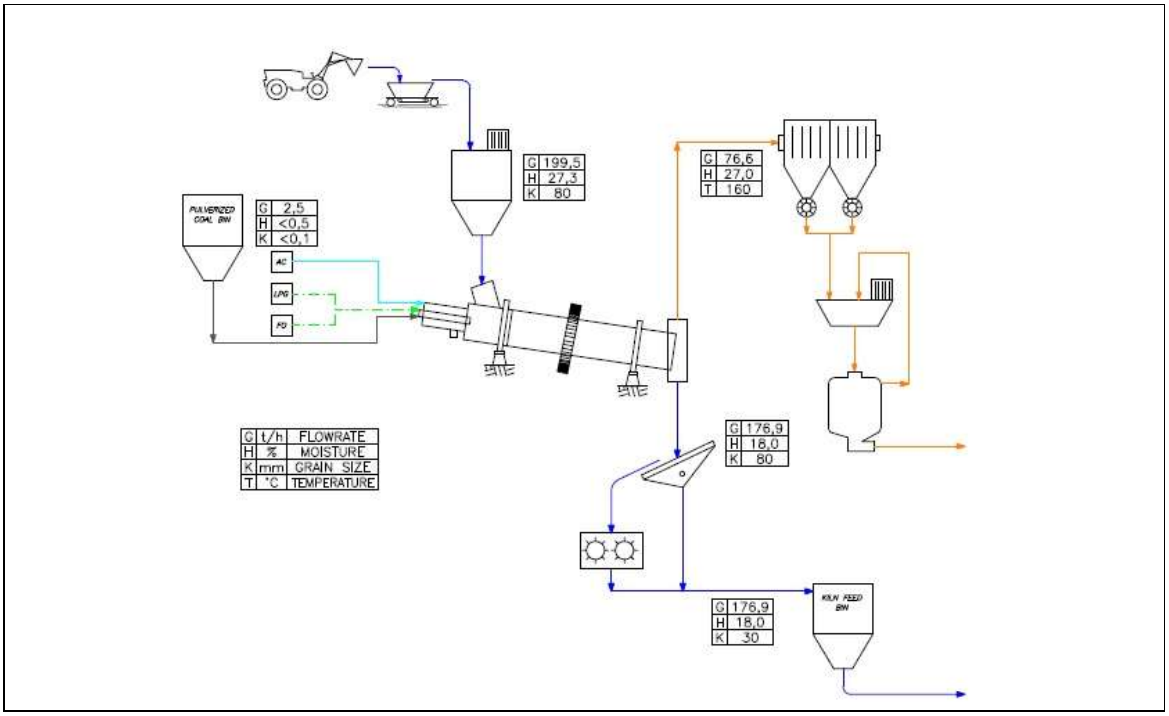
Table 17.2 Key drying parameters – Base Case

Item	Unit	Value
Ore + Dust feed rate – one line	wmt/hr	199.5
Feed moisture content	%	27.3
Dryer throughput – ore + dust	dmt/hr	145.0
Product moisture content	%	18
Dryer type		Co-current – rotary
Fuel type		Pulverised coal
De-dusting system		ESP

Table 17.3 Tertiary crushing key parameters – Base Case

Item	Unit	Value
Ore + dust feed rate	wmt/hr	176.9
Ore + dust feed rate	dmt/hr	145.0
Circuit type		Direct, open
Crusher type		Double roll - toothed

Figure 17.5 Drying and tertiary crushing flowsheet



17.3.4 Calcining

The material reclaimed from the kiln feed bin will be combined with reductant coal of < 12,5 mm from the crushed coal storage bin. A weightometer on the ore feed will govern the set feed rate of addition of coal which is controlled by a variable speed rotary valve. The kiln feed will be conveyed to the kiln feed chute. The three main phenomena which occur in the rotary kiln are: drying, calcining and pre-reduction. The kiln will be constructed with 3 retention dams – one at the drying section, the second at approximately the end of the calcining process, and the last one at the discharge end. As a consequence, the material will interact for a longer period of time, thereby enhancing the metallurgical process, as well as avoiding the generation of excess fines. The retention time in the kiln is generally around 3 hours. The kiln will be equipped with lifters at the feed end section for the purpose of recovering heat from the off-gas stream. The lifter arrangement will be designed, however, such as to also minimise dust carry over by the off-gas stream. The heat required for the kiln operation will be supplied by a burner of specialised design located at the discharge end of the kiln, thus providing the counter current heating for attaining the required temperature profile along the length of the kiln. The burner is designed such as to be able to use pulverised coal and can also use fuel oil as an alternative or in case it is necessary to stabilise the kiln environment. This dual purpose burner thus provides flexibility in future decisions on the most economic fuel to be utilised.

The hot calcine at the kiln discharge will pass through a screening device for removing any oversize material that may cause blockages in downstream equipment. It will be a trommel with 80 mm mesh opening, or even larger if proved to be acceptable for the hot calcine discharge valves in the kiln calcine bin as well as the furnace feed system. The undersize material will be discharged into a refractory-lined bin, from where it will be extracted through a transfer chute into a refractory-lined, 15 m³ container, which will be located on a transfer car. All equipment associated with the calcine transfer and furnace feed systems are described later.

The kiln off-gas is conveyed to the electrostatic precipitator at approximately 300°C. Collected dust will be pneumatically conveyed to the dust recycling bin at the secondary crushing plant. Clean off-gas will be partially recycled to the coal milling facility, and the rest will be discharged into the atmosphere. Key criteria of this section are depicted in Table 17.4, as well as Figure 17.6.

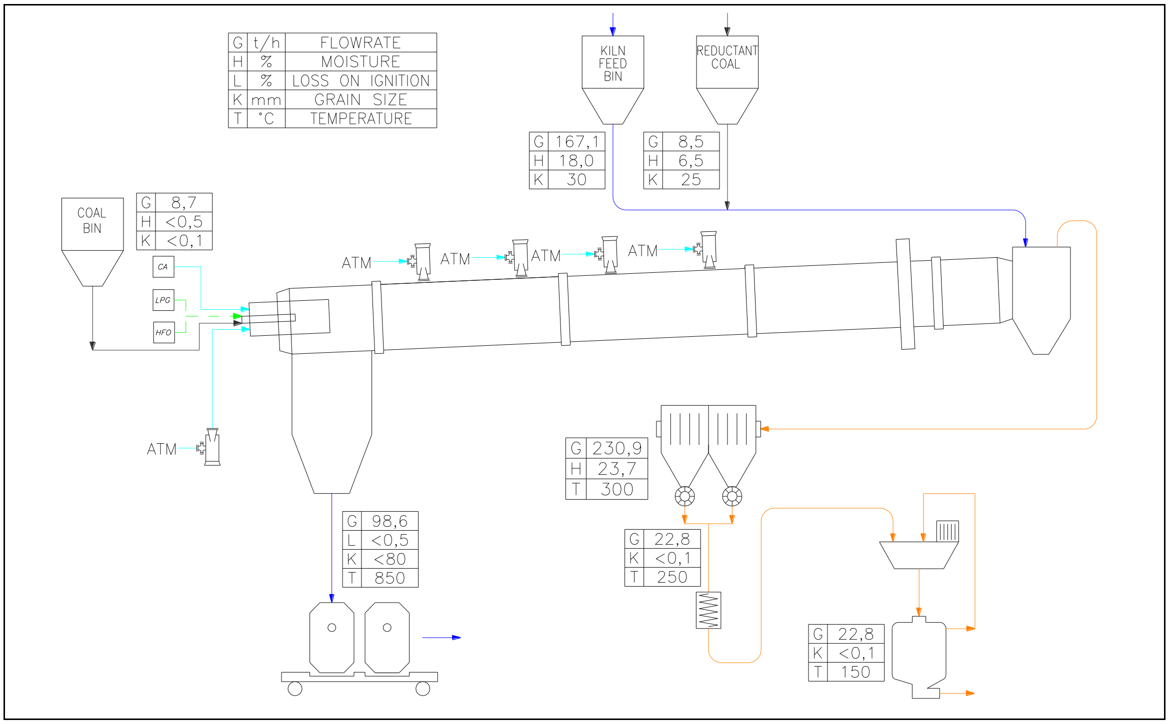
Table 17.4 Calcining key criteria – Base Case

Item	Unit	Value
Ore + dust feed rate	wmt/hr	167.1
Ore + dust feed rate	dmt/hr	137.0
Ore + dust moisture content	%	18.0
Fresh ore feed rate	dmt/hr	114.2
Reductant coal feed rate	dmt/hr	9.1
Reductant coal feed rate	wmt/hr	9.8
Coal moisture content	%	6.5
Calcine production	mt/h	98.6
Degree of Ni pre-reduction	%	20
Degree of Fe pre-reduction	%	70

With reference to the findings of the testwork carried out at FLS, the following issues are highlighted in terms of kiln design:

- From the testwork results at FLS, it was observed that due to the characteristics of the ore blend material it may be necessary to restrain the target temperature of the calcine discharge to 800°C to 825°C. For design purposes for the kiln, however, IGEO has defined a target temperature of 850°C for the kiln discharge calcine and 825°C for the calcine feed to the furnace. It is strongly recommended that these temperatures are confirmed on a larger bulk sample on pilot plant scale in the next stage of the project.
- Partly due to temperature constraints mentioned above but combined with the mineralogy of the ore, FLS considers it may only be possible to achieve pre-reduction values of 10% for Ni and 60% for Fe. Again for design purposes for the kiln, however, IGEO have defined 20% for Ni and 70% for Fe in the PDC.
- FLS ascertained that from agglomeration behaviour of the ore in comparison to other ores tested, dusting rates in the kiln off-gas of 15% to 20% could be anticipated. IGEO have applied a rate of 20% in the PDC.

Figure 17.6 Calcining flowsheet



17.3.5 Smelting

The furnace selected for a project comparable to the Base Case proposed a circular furnace with an 18 m internal hearth diameter for processing the 0.9 Mtpa (dry) ore per year. Figure 17.7 shows a cross-cut of a typical electric arc furnace. Final selection of the furnace will have to be reviewed in a trade-off study during the next phase of the project, comparing a circular furnace with a rectangular furnace.

Figure 17.7 Cross section of a typical electric arc furnace

17.3.6 Slag characteristics

The testwork carried out at Xstrata Process Support (XPS) and Kingston Process Metallurgy (KPM) on Araguaia ore blends focussed considerably on the smelting characteristics of the slags produced from the ores because the composition of Araguaia slag differs from that of slag produced at other RKEF operations. Key factors which need to be considered are:

- The combined impact of the ratio SiO_2/MgO, FeO and Al_2O_3 contents of the slag on the liquidus temperatures and the viscosity of the slags. The chemical effects of high SiO_2 slag on the basic furnace refractory material must also be considered.
- The impact of impurities, particularly carbon and silicon, on the liquidus temperatures and viscosity of the metal. Also the risks must be considered of silicon reversion and associated extensive generation of heat from the reaction if silicon content is too high in the metal. Slag foaming is also an issue that can occur with viscous slags.
- The impact of the issues in 1 and 2 above can be affected/controlled to varying degrees by controlling the degree of reduction of Fe effected during the smelting process.
- The degree of iron reduction dictates the grade of nickel to be achieved in the final metal product as well the silicon and carbon content of the metal. Higher reduction will result in lower metal grades and in all the testwork evaluations, grades of nickel content of between 15% Ni and 25% Ni have been evaluated. Higher degrees of reduction also give the added benefit of higher nickel recoveries across the furnace.

- At the ratio of SiO_2/MgO of 2.29 and the FeO and the Al_2O_3 content anticipated in the Araguaia slags, the liquidus temperatures can be significantly depressed. However, minimum tapping temperatures of the slag must be attained to ensure fluidity. This therefore can result in excessive superheating of the slag and metal with possible undesirable consequences to the operation. It also will demand intensive water cooling systems be installed on the furnace sidewalls.

Hatch (Toronto) was requested to examine available information on the smelting characteristics of the Araguaia ore. It was from consideration of all the data available that HZM in conjunction with IGEO agreed that the target metal grade from the furnace to adopt for the PDC will be 20% nickel. Given this criterion, and evaluating all the relevant data from liquidus temperature determinations in the KPM testwork, the liquidus and tapping temperatures of metal and slag were then defined as shown in Table 17.5. The resultant superheat contained in the slag is a factor which will have to be managed from both an operational perspective and in designing of the cooling system required for the furnace walls. Attention is also being given to the furnace roof construction as explained in section 17.3.7.

The tapping temperatures achievable due to the liquidus temperatures of metal and slag are criteria that will need to be verified in pilot scale work with a bulk sample in the next phase of the project.

Table 17.5 Estimated furnace metal and slag temperatures

	Araguaia Criteria
Ore Blend	LOM Plan Blend
Target crude Fe-Ni grade % Ni	20
Metal liquidus °C	1,440
Slag liquidus °C	1,380
Metal tapping temperature °C	1,470
Slag tapping temperature °C	1,550
Metal - Slag delta T °C	80
Metal superheat °C	30
Slag superheat °C	170
Nickel furnace recovery %	95.0

17.3.7 Smelting process description

Each furnace will feature two calcine transfer cars and each car will contain two calcine containers. The calcine transfer car will move the container to the lifting position, where a lifting tower (one per car) will perform its hoisting function to the top of the furnace feed bins.

It has been determined from the energy balance that the specific power consumption is approximately 511 kWh per metric tonne calcine. On this basis, the required power rating for the furnace will be approximately 50 MW for smelting 98.6 mtph calcine. This power consumption is based on a slag tapping temperature of 1,550^{o}C and metal tapping temperature of 1,470^{o}C.

The hot calcine, containing the required amount of residual reductant, will be stored in the furnace feed bins, which will be strategically positioned in order to obtain a suitable distribution and embankment of the charge inside the furnace. The calcine feed into the furnace will be monitored in accordance with the slag temperature and, therefore, the slag temperature inside the furnace will require monitoring as well. This has proven on other operations to be a critical issue to be monitored.

Metal tapping will be carried out through one of two tapholes, positioned at a convenient location along the circumference of the furnace. Slag will be tapped at a location approximately diametrically opposite of the metal tapholes (see Figure 17.7) and will be granulated with water. Any entrained slag fines in the water will settle in an appropriately designed settling tank, and will be reclaimed by front-end loader and truck and transported to the slag dumping area. Water will be ducted to a cooling system and dam, from where it will be pumped to an elevated reservoir with a capacity of 6,000 m^3.

The working hearth refractory will be designed with tapered tongue and grooved, high magnesia bricks. Underneath the working hearth, a tapered safety hearth will be constructed with high thermal conductivity magnesia bricks. The cooling system for the bottom floor will be air, with dedicated fans. The upper sidewall will be fixed to the shell by means of a wall hold down system, or other design as furnished by the particular supplier. Side wall cooling in the region of the molten slag depth will be effected by copper cooler components.

The furnace process (primary) off-gas will be collected by a dedicated system at a temperature of 1,000°C to 1,200°C at the furnace roof outlet. It will be cooled down and then conveyed to a baghouse, designed for obtaining a high efficiency of particulate collection that will comply with the Environmental Design Criteria. An emergency off-gas system will discharge the raw off-gas to atmosphere while the furnace power is reduced to a minimum for overcoming the upset condition.

The hot calcine discharge transfer points into the transfer containers and into the furnace feed bins will also have a dedicated (secondary) dust extraction system. De-dusting gases collected from the metal and slag tapping positions will be used for cooling the primary off-gas, and then undergo the same handling process.

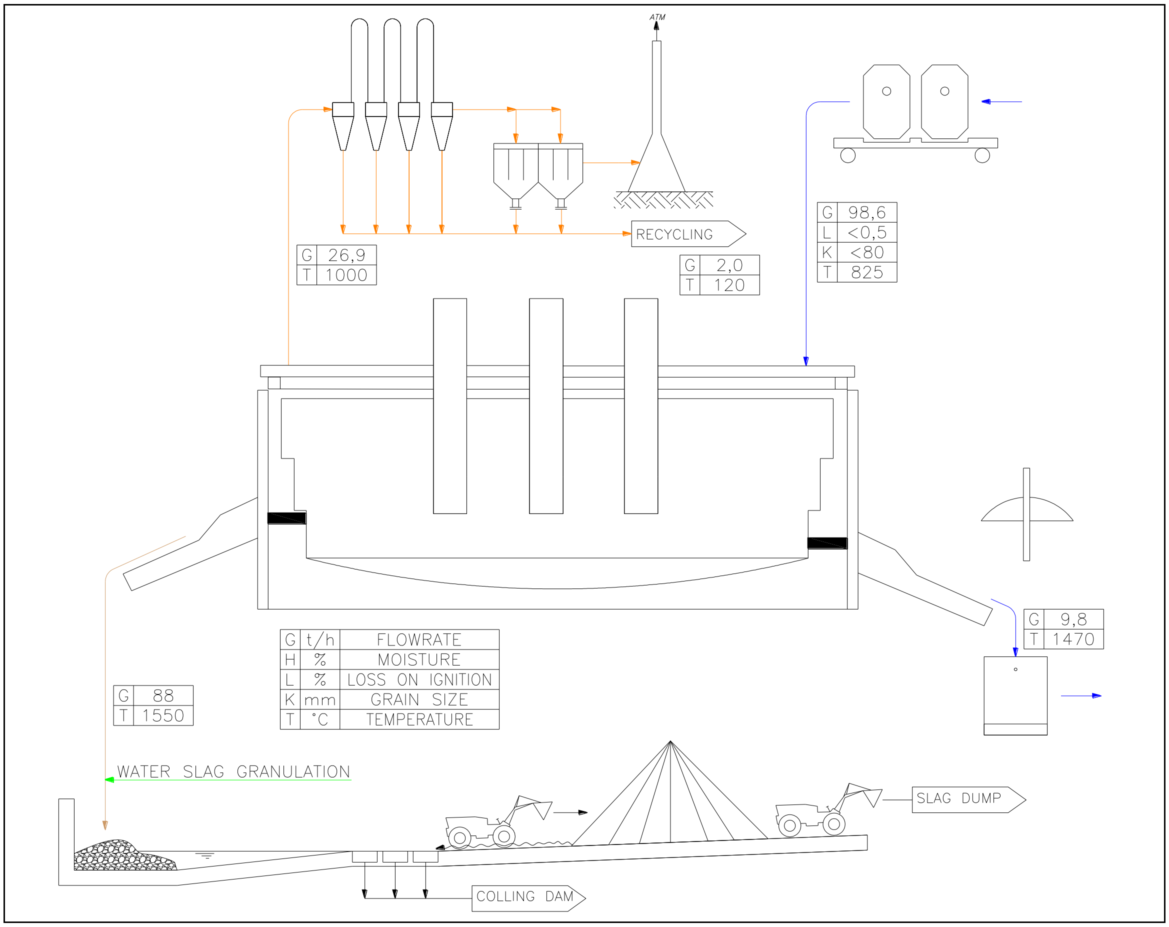
The roof will be constructed with high alumina bricks and the central region around the electrodes will be manufactured with special ramming material. This is to ensure that maintenance can be performed very readily with minimum shutdown time and minimum exposure of the workers to the harsh environment. The design concept will be such that no water cooling of the furnace roof will be required.

Key characteristics of this section are depicted in Table 17.6 as well as Figure 17.8.

Table 17.6 Smelting key parameters – Base Case

Item	Unit	Value
Furnace feed material		
Calcine feed rate	t/hr	98.6
Calcine temperature at furnace	°C	825
Furnace characteristics		
Geometry		Circular
Internal diameter	m	18.0
Electrodes configuration		3 - triangle
Electrode diameter	m	TBC
Number of transformers		3, one per phase
Power	MW	50
Slag region cooling		Copper elements
Metal characteristics		
Metal production -design	t/hr	9.8
Tapping Temperature	°C	1,470
Liquidus temperature	°C	1,440
Ni content	%	20.0
Slag characteristics		
Average slag production rate	t/hr	88
Tapping mode	h/day	15-20
Tapping Temperature	°C	1,550
Liquidus Temperature	^{o}C	1,380
Dedusting systems		
Type (both systems)		Baghouse

Figure 17.8 Smelting flowsheet



17.3.8 Refining

Crude metal will be tapped at 1,470 °C from the furnace tapholes into a preheated ladle. A last-minute preheater will be used at the final ladle position at the launder for maintaining the ladle temperature. Furnace metal contains mainly carbon, phosphorous, a small amount of silicon and sulphur as impurities. Ferrosilicon (FeSi) will be added into the ladle when and if required, in order to avoid porous plug blockage, and into the metal launder, in order to improve metal flowability through the launder. FeSi will also generate energy in the ladle due to the reaction with oxygen to be blown into the metal through a refractory-lined or water-cooled lance, depending on whether the metal is being tapped or blown.

Nitrogen will be blown into the metal through a porous plug at the ladle bottom for agitating the metal, in order to ensure an even temperature distribution through the melt and improve reaction kinetics. Nitrogen blowing is important for enhancing process reactions, and is therefore maintained through the entire refining process. During oxygen blowing, lime and fluorspar (if required) are added in order to remove phosphorous into the slag, thus shortening the subsequent process cycle time. At the end of metal tapping, the ladle containing up to 25 tonnes of hot metal, will be positioned on a transfer car by an overhead crane and transferred to the heating station for conditioning of the oxidised slag. This will be removed off the surface of the metal by a skimming machine and tilting of the ladle.

Once free of the oxidising slag, the molten bath will then be deoxidised with FeSi and aluminium, and a lime-rich slag will be developed for the desulphurisation process. Dissolved oxygen and temperature are key parameters to be controlled and, under favourable conditions, calcium-silicon (CaSi) cored aluminium wire will be injected deep into the bath to speed up the desulphurisation process. At the end of the cycle, slag will be skimmed off and a metal sample will be collected. Should the analysis match the required specification, the ladle will be submitted to a final heating stage to a temperature of 1,630 °C before metal granulation. Should the metal specification not be achieved, a further slag is formed and skimmed off prior to the final heating stage followed by granulation.

During the refining process, the oxidising slags will be granulated with water, or allowed to cool naturally, while the reducing slags will be collected in slag pots and deposited in a specific area for hydration. This yields spontaneous breaking of the slag blocks and facilitates recovery of any entrained metal. Metal characteristics are defined in Table 17.7. The Ni/Co and Ni/Cu ratios are adequately above market specifications.

Table 17.7 Metal characteristics

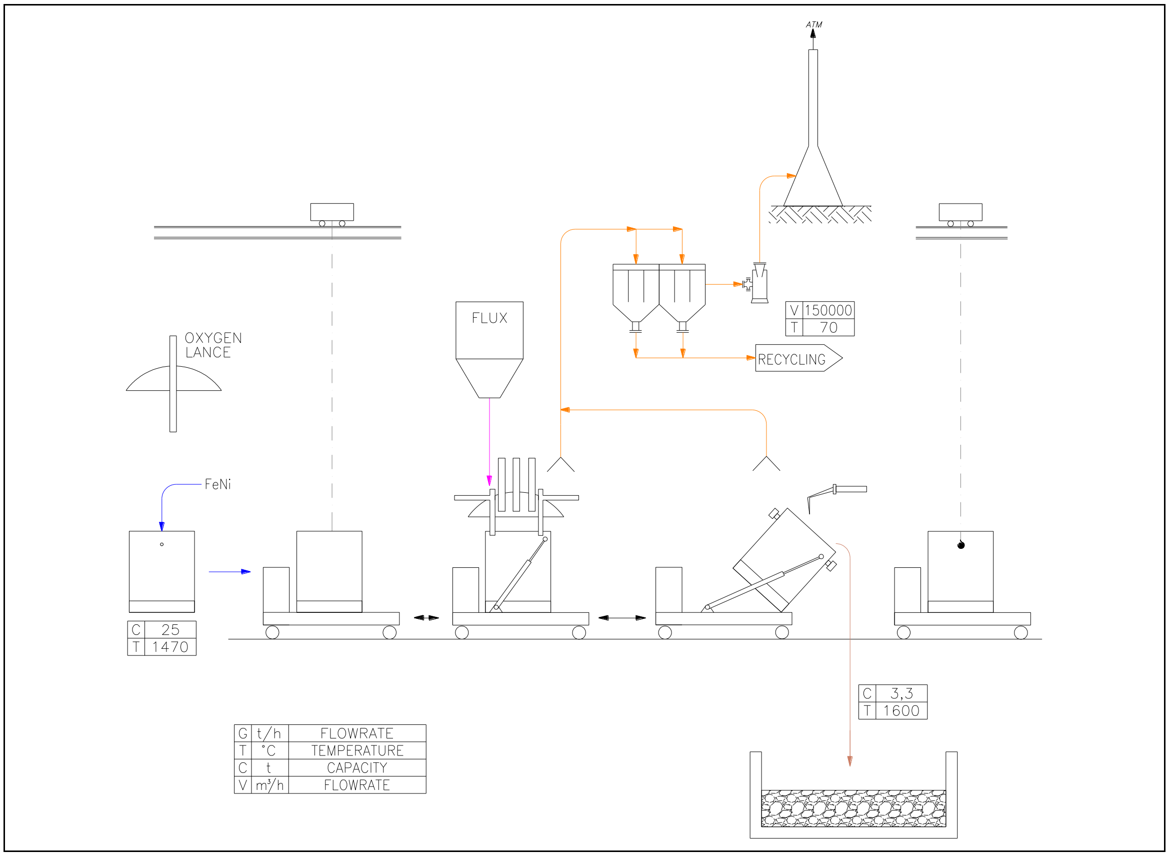
Elements	Unit	Crude Metal	Refined Metal
Ni	%	20.0	20.3
Si	%	< 0.3	< 0.04
C	%	< 1.0	< 0.04
P	%	< 0.1	< 0.03
S	%	< 0.5	< 0.04
Cu	Ni/Cu	> 100	> 100
Co	Ni/Co	> 30	> 30

The refining station design criteria are indicated in Table 17.8. The flowsheet is depicted in Figure 17.9.

Table 17.8 Refining furnace key design characteristics – Base Case

Item	Unit	Value
Specific power input	kWh/t Fe-Ni	160
Design Power – estimated - TBC	MVA	5.5
Type		Ladle furnace
Ladle capacity	t	25
Crude metal temperature	°C	1,470
Refined metal temperature	°C	1,630
De S slag reagents :		
Lime	Kg/t Fe-Ni	87
CaF_2	Kg/t Fe-Ni	3.0
Al	Kg/t Fe-Ni	10
FeSi	Kg/t Fe-Ni	3
CaSi	Kg/t Fe-Ni	2.1
De P slag reagents:		
Lime	Kg/t Fe-Ni	15
Fluorite	Kg/t Fe-Ni	0.5
Oxygen	Nm3/tFeNi	10
Nitrogen	Nm3/tFeNi	0.5

Figure 17.9 Refining flowsheet



17.3.9 Metal granulation and product conditioning

Refined metal in the ladle at 1,630 °C will be transferred to the granulation facility and will be positioned on a movable platform by the overhead crane. The metal will discharge through a sliding gate installed at the bottom of the ladle at a controlled flow rate of approximately 1.0 t/min directly into one of two water tanks which are equipped with strategically positioned high pressure water nozzles for obtaining the granulated metal.

The granulated metal will be contained in the two tanks, half in each one and will be extracted by the overhead crane at the end of the process and transferred to a dewatering bin/screen, and finally fed into a rotary dryer. The dried material will be screened into product sizes as required by the market. The sized material will be bagged or transferred to a stockpile for dispatching. Generally fines fractions of < 2 mm are not accepted by clients and will be recycled directly into the ladle at the heating station.

Table 17.9 and Table 17.10 define the key criteria for this process section while Figure 17.10 and Figure 17.11 depict the flowsheet for metal granulation and product conditioning respectively.

Table 17.9 Metal granulation key characteristics

Item	Unit	Value
Metal granulation		
Granulation rate	t/min	1.5
Granulation time	min	35
Metal temperature	°C	1,630

Table 17.10 Metal conditioning key characteristics

Item	Unit	Value
Metal drying and screening		
Dryer feed rate	t/hr	25
Screen size	mm	2, 5 and 50
Product conditioning		
Configuration		Big bags / bulk
Storage capacity	t	10,000

Figure 17.10 Refined metal granulation flowsheet

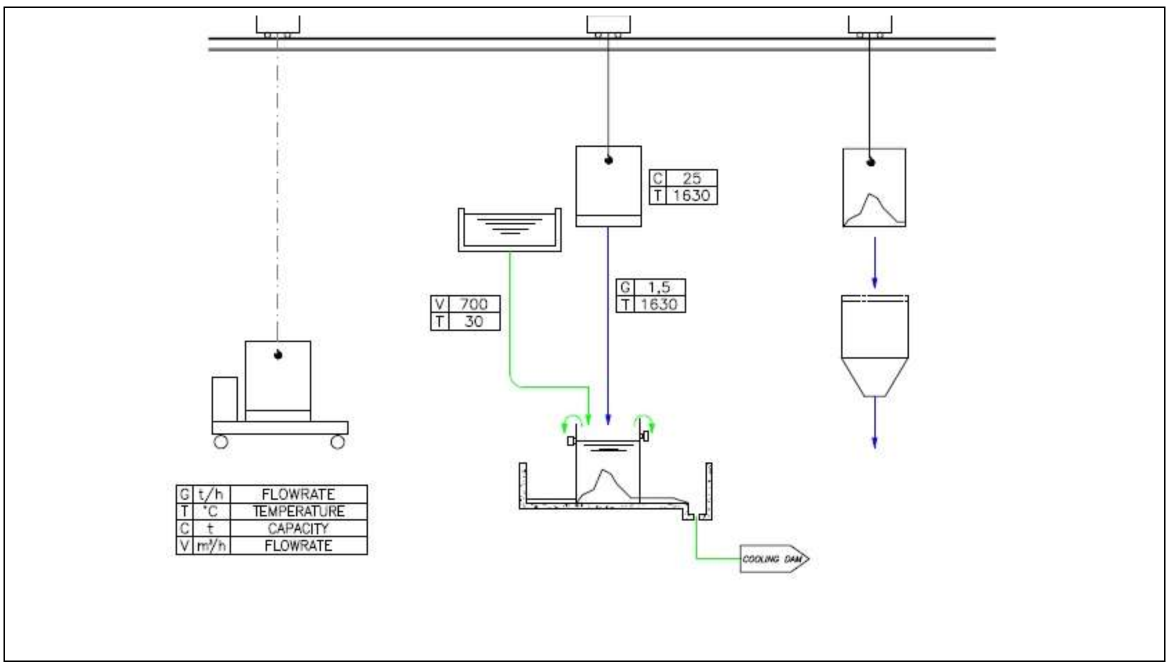
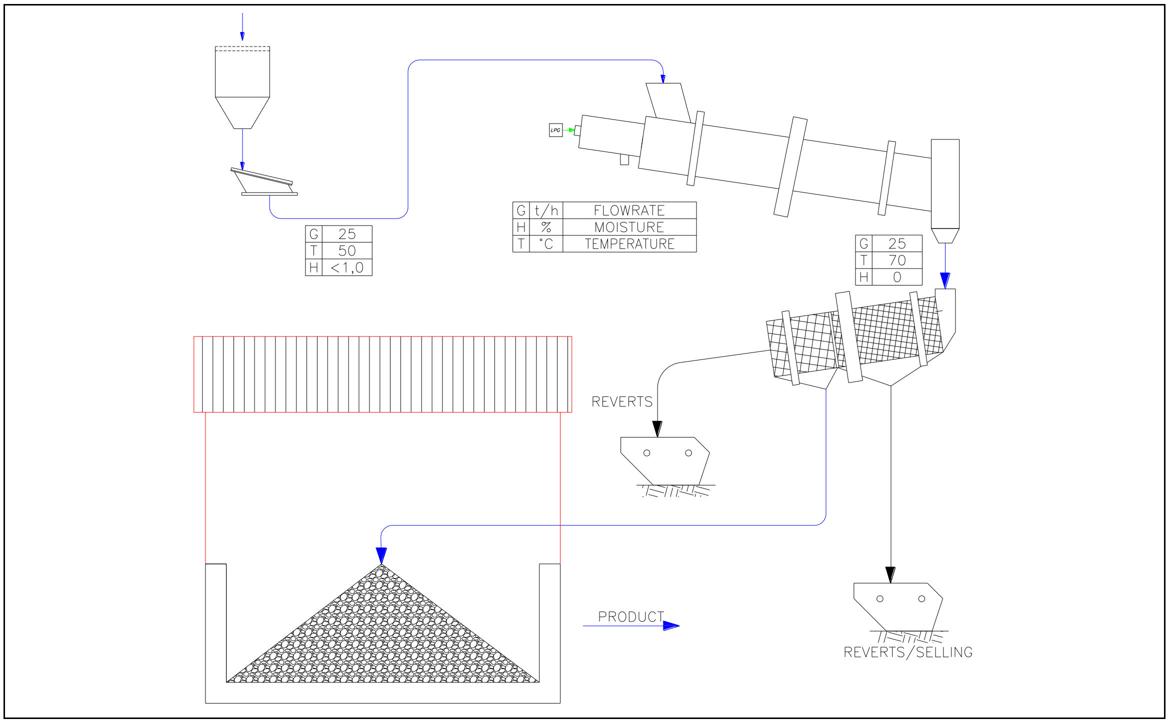


Figure 17.11 Refined metal conditioning flowsheet



17.3.10 Auxiliary process installations

The main auxiliary facilities that are required for the processing of laterites for Fe-Ni production are:

- Coal preparation plant (for fuel and reductant)
- Dust handling systems.

Utilities systems are described further in this document.

17.3.11 Coal preparation

The coal preparation plant will be designed for supplying both the pulverised fuel coal and reductant according to the key criteria indicated in Table 17.11 and the flowsheet is depicted in Figure 17.12. The plant will be designed with capacity for supplying both the dryer and the kiln.

Table 17.11 Coal preparation key characteristics

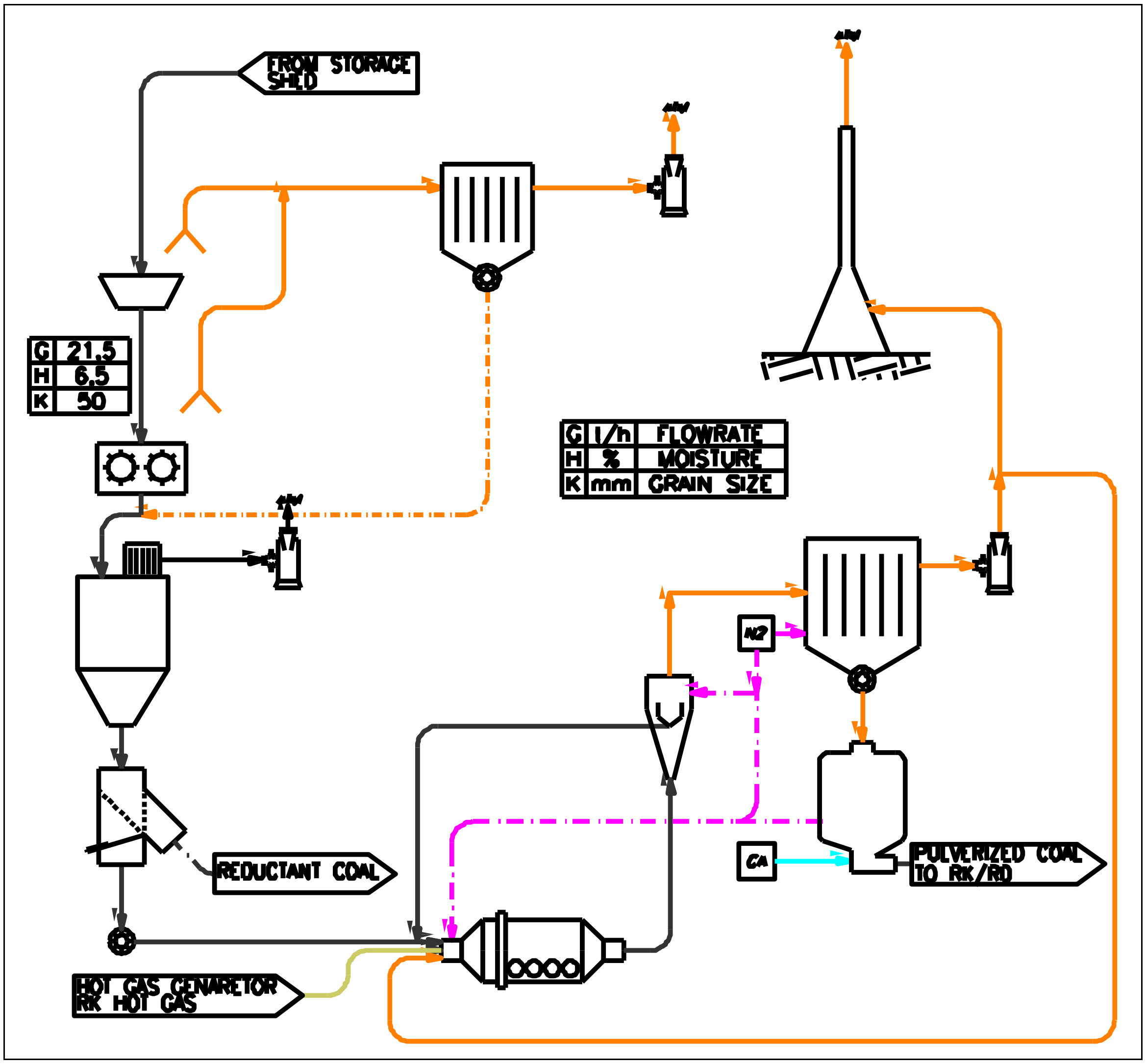
Item	Unit	Value
Fuel coal to the dryer burner	dmt/year	23,623
Fuel coal to the kiln burner	dmt/year	68,760
Reductant coal	dmt/year	72,000
Total coal consumption	dmt/year	164,383
Total coal consumption	wmt/year	175,811

Coal will be transported to site and discharged onto a receiving hopper with grizzly, located under a shed. Coal will then be extracted from the hopper and transported to the coal stacker or tripper conveyor which will pile it in a covered storage area with capacity for 30,000 m^3 (wet). From one of the two piles a front-end loader will reclaim the coal and discharge it into the coal crusher feed hopper.

Coal will be bulldozed from the crusher feed hopper into a roll crusher which will reduce the as-received coal top size to 12.5 mm. The crushed coal will then be conveyed to either the coal milling feed bin or the reductant coal bin. The handling system will operate with diverters linked to level indicators such that when one bin is full the crushed coal will be diverted to another bin. If all bins are full, the coal crushing and conveying system will be stopped.

Coarse coal will be conveyed to a ball mill designed such as to reduce coal top size to 170# (or 90 µm). Clean calcining off-gas will be used to dry the ground coal to 0.5% moisture so that it can be used as burner fuel for the kiln and dryer units. In the absence of clean off-gas, an oil-fired hot gas generator will be available. Pulverised coal (coal mill product) will be classified in a dynamic separator and sent to the coal milling baghouse. While coarse particles will be returned to the mill feed, pulverised coal will be pneumatically conveyed to the consumption points, namely the dryer and kiln. The dryer and kiln will have one bin and dosing system. From the feed bins Coriolis/Pfister dosing pumps will feed the dryer and kiln burners individually.

Figure 17.12 Coal preparation flowsheet



17.3.12 Dust handling systems

Dust contains high nickel grades and therefore, in order to obtain high metallurgical recoveries for the overall plant (i.e. above 90%), it is paramount to implement a successful, proven and reliable dust handling and recovery system. Dust will be mainly generated and thus collected at the following points: dryer ESP, kiln ESP, furnace baghouse (primary and secondary), and refinery baghouse.

The refinery dust will be discharged into a bucket and combined together with the smelter dust for recycling directly to the smelting furnace due to its very high Ni content (approx.5 – 7%).

Dryer dust will be collected independently and also conveyed to the rotary kiln dust bin for transporting, both together, to the dust storage bin located upstream of the secondary crushing. From this storage bin, dust will be wetted and added to the dust mixing drum in order to be reintroduced back to the process.

17.3.13 Utilities

The utilities areas and services incorporated for the project are:

- Fuel oil storage
- Diesel storage
- LPG storage/distribution
- Water systems
- Söderberg paste handling
- Refinery reagents handling
- Compressed air generation; and
- Oxygen and nitrogen storage/generation.

Each of them will be described as follows.

17.3.14 Fuel oil storage

Fuel oil will be used in start-up and flame stabilisation conditions in both the dryer and the kiln. The storage location, therefore, has to be close to these consumers. The system will feature one operating and one standby pump for each consumption point.

Oil will be also used for the emergency/start-up hot gas generator at coal milling.

17.3.15 Diesel storage

Diesel will be used by plant and mine mobile equipment. One area for each site (mine and plant) will be envisaged. Fuel supply will be performed by tank trucks as required.

17.3.16 LPG storage

LPG will be consumed in the dryer and kiln pilot burners, last-minute pre-heaters, ladle dryer, ladle pre-heater, refined Fe-Ni drying, the metallurgical laboratory, and the canteen. Therefore, two storage areas are envisaged: one for the non-process areas and another for the process areas.

It is suggested that the client pursue an "over-the-fence" agreement, so that capital costs are minimised.

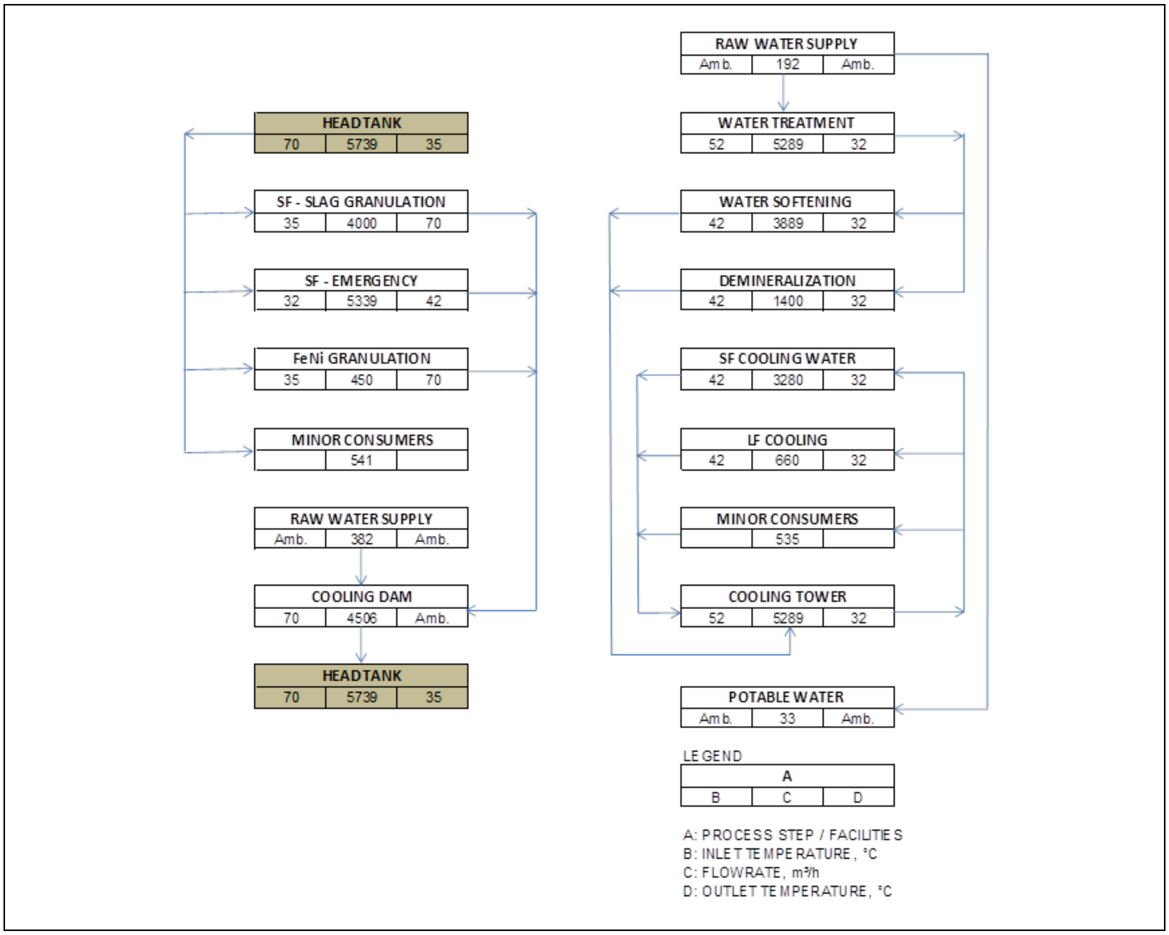
17.3.17 Water systems

The water systems comprise:

- Raw water supply and make-up
- Water treatment
- Electrodes cooling water system
- Copper plates cooling , slag and metal copper block cooling
- Slag granulation cooling
- Elevated reservoir.

The preliminarily block diagram for the different water systems as shown in the Water Balance document is depicted as follows:

Figure 17.13 Block flowsheet of water system



17.3.18 Söderberg paste handling

Söderberg paste for the electric furnace electrodes will be hoisted to the electrode floor for handling and addition into the electrode casings.

17.3.19 Refinery reagents handling

Refinery reagents will be transported to site in bulk bags whenever possible.

17.3.20 Compressed air

Compressed air will be used for instruments, dust conveying, and general purpose use. Conveying systems will be independent from instrument/plant air systems. All compressors will feature the same size so as to minimise spares inventory.

17.3.21 Oxygen and nitrogen

Oxygen is used in refinery lancing and for metal and slag tapping. Nitrogen is used for porous plug stirring, emergency lancing at the refinery aisle and inertisation of the coal bins. In order to reduce capital costs, it is suggested that the project pursue supplying alternatives "over-the-fence".

17.4 Process design criteria

The PDC were developed by IGEO for the Base Case with contributions from several key team members. Table 17.12 defines the source codes for the original source of the data as reflected in the various tables in this section.

Table 17.12 Data source code

Source Code	Origin
HM	Horizonte Minerals
MB	Mass Balance
EB	Energy Balance
TBC	To be confirmed

17.1.1 Overall plant data – Base Case

The Base Case data as used for the development of the mass balance and PDC is shown in Table 17.13.

Table 17.13 Overall plant design criteria – Base Case

Item	Source	Unit	Nominal value	Design value
Ore Characteristics				
Angle of repose of the wet ore		degree	50	50
Angle of repose of the mix ore-dust		degree	40	40
Bulk Density of ore (dry)		t/m^3	1.37	1.37
Ni - Nominal value - LOM average	Snowden	%	1.41	1.41
Ni - Design value - early years LOM plan			1.80	1.80
Fe	HM	%	16.50	16.50
Fe : Ni ratio - nominal		ratio	11.70	
Fe : Ni ratio - design		ratio		9.17
Co	HM	%	0.050	0.050
SiO_2	HM	%	40.22	40.22
MgO	HM	%	17.60	17.60
SiO_2 / MgO	HM	%	2.29	2.29
Al_2O_3	HM	%	4.57	4.57
CaO	HM		0.08	0.08
S	TBC - HM	%		
P	TBC - HM	%		
K_2O	TBC - HM	%		
Na_2O	TBC - HM	%		
SO_3	TBC - HM	%		
TiO_2	TBC - HM	%		
MnO	HM	%	0.39	0.39
Cr_2O_3	HM	%	1.16	1.16
LOI at 982°C	HM	%	11.89	11.89
Plant throughput				
One Line Flowsheet Capacity		Mtpa (dry)	0.9	0.9
Nickel in the ore - NOMINAL		% Ni	1.41	
Nickel in the ore - DESIGN		% Ni		1.80
Nickel in ore feed to plant - NOMINAL		tpa	12,690	
Nickel in ore feed to plant - DESIGN		tpa		16,200
Nickel recovery - overall		%	93.0	93.0
Nickel production - NOMINAL - refined Fe-Ni	MB - D396	t/yr	11,803	
Nickel production - DESIGN - refined Fe-Ni	MB - D396	t/yr		15,067

17.4.1 Ore receipt, crushing and homogenising

The design criteria for primary and secondary crushing are shown in Table 17.14 and for homogenising the design criteria are shown in Table 17.15.

Table 17.14 Design criteria for ore receipt and crushing – Base Case

Item	Source	Unit	Nominal value	Design Value
Primary crushing				
Operating hours		h/yr	6,570	6,570
Availability		%	75	75
Fixed grizzly gap		Mm	500 x 500	500 x 500
Ore moisture content		%	30	30
Fresh ore Feed rate		Mtpa	0.9	0.9
Fresh ore Feed rate		Tpa (wet)	1,285,714	1,285,714
Fresh ore Feed rate		dry t/hr	137.0	137.0
Fresh ore Feed rate		wet t/hr	195.7	195.7
Crusher type		type	sizer	sizer
Crusher gap		Mm	200	200
Kiln dust recycling - FROM dust storage				
Operating hours		h/yr	6,570	6,570
Availability		%	75	75
Temperature		°C	150	150
Dust moisture content - dust bin		%	0	0
Dust moisture content at drum inlet		%	10	10
Recycling load - % of dry fresh ore feed		%	20	20
Dust addition		dry tpa	180,000	180,000
Dust addition		wet tpa	200,000	200,000
Dust addition		dry mt/hr	27.4	27.4
Dust addition		wet mt/hr	30.4	30.4
Total Feed to the contacting drum – ore + dust		tpa (dry)	1,080,000	1,080,000
Total Feed to the contacting drum – ore + dust		tpa wet	1,485,714	1,485,714
Total Feed to the contacting drum – ore + dust		dry mt/hr	164.4	164.4
Total Feed to the contacting drum – ore + dust		wet mt/hr	226.1	226.1
Moisture content in the mix			27.3	27.3
Secondary crushing				
Operating hours		h/yr	6,570.0	6,570.0
Availability		%	75.0	75.0
Feed to the screen		wet mt/hr	226.1	226.1
Moisture content in the mix		%	27.3	27.3
Screening size		Mm	80.0	80.0
O/S fraction moisture content		%	15.0	15.0
O/S fraction partition		%	30.0	30.0
		wet mt/hr	58.0	58.0
U/S fraction		%	70.0	70.0
		wet mt/hr	168.1	168.1
U/S fraction moisture content		%	31.6	31.6

Table 17.15 Design criteria for homogenising – Base Case

Item	Source	Unit	Nominal value	Design Value
System		Stacker/ Reclaimer	Stacker/ Reclaim	Stacker/Reclaim
Number of stockpiles		#	2	2
Angle of repose		Degrees	34	34
Ore bulk density		wet t/m3	1.25	1.25
Type of stockpile protection			covered	covered
Feed TO stockpile		wet mt/hr	226.1	226.1
		wet mt/day	5427	5427
Stockpile capacity - each		Days	8	8
		wet tonnes	43418	43418
Reclaiming capacity FROM stockpile to Dryer		t/hr	199.5	199.5
Operating hours		#	7446	7446
Availability		%	85	85

17.4.2 Rotary dryer and tertiary screening

The design criteria for the rotary dryer are shown in Table 17.16and for tertiary screening and crushing in Table 17.17. At the time of preparing this report a decision regarding the sizing of the associated emergency ore stockpile was still pending.

Table 17.16 Design criteria for the rotary dryer – Base Case

Item	Source	Unit	Nominal value	Design Value
Rotary Dryer + Tertiary Screening				
Dryer dimensions	TBC	diameter m		
Dryer dimensions	TBC	length m		
Fresh ore treated PER dryer line		Mtpa (dry)	0.9	0.9
%moisture in ROM ore		%	30	30
Fresh ore treated PER dryer line		tpa (wet)	1,285,714	1,285,714
% recycled dust		% of dry ore	20	20
%moisture in recycled dust to contact drum		%	10	10
Total recycled dust PER line		tpa (dry)	180,000	180,000
Total recycled dust PER line		tpa (wet)	200,000	200,000
Total feed rate PER Dryer - ore + dust		Tpa (dry)	1,080,000	1,080,000
Total feed rate PER Dryer - ore + dust		Tpa (wet)	1,485,714	1,485,714
% moisture in dryer feed - total ore + dust			27.3	27.3
Operating hours		h/yr	7446	7446
Availability		%	85	85
Fresh ore feed to dryer		dry mt/hr	120.9	120.9
Fresh ore feed to dryer		wet mt/hr	172.7	172.7
Dust feed to dryer		dry mt/hr	24.2	24.2
Dust feed to dryer		wet mt/hr	26.9	26.9
Total feed to dryer		dry mt/hr	145.0	145.0
Total feed to the dryer		wet mt/hr	199.5	199.5
Moisture in the feed to the dryer		%	27.3	27.3
Target moisture in the dryer product		%	18	18
Dryer product		wet t/hr	176.9	176.9
		dry t/hr	145.0	145.0
Dryer water removal		t/hr	22.6	22.6
Arising dust		%	1.0	1.0
		dry t/hr	1.5	1.5
Dust particle size		mm	< 0,2	< 0,2
Dust moisture		%	3.0	3.0

Table 17.17 Design criteria for tertiary screening and crushing - Base Case

Item	Source	Unit	Nominal value	Design Value
Operating hours		h/yr	7,446	7,446
Availability		%	85	85
Type of screen		model	vibrating	vibrating
Feed to the screen		wet t/hr	176.9	176.9
Feed to the screen		dry t/hr	145.0	145.0
Screening size		mm	30	30
O/S fraction		%	30	30
O/S fraction tonnage		dry t/hr	43.5	43.5
O/S fraction		wet t/hr	51.2	51.2
U/S fraction tonnage		wet t/hr	125.7	125.7
Moisture content in the mix		%	18	18
Moisture content in the O/S		%	15	15
Moisture content in the U/S		%	20.1	20.1
Crusher type		type	roll	roll
Crusher gap		mm	30	30
Emergency dried ore stockpile				
Storage capacity	TBC	h	20	20
Storage capacity		t	3,341	3,341

17.4.3 Rotary kiln

The rotary kiln is one of the key installations for the treatment of nickel laterites in a pyrometallurgical process and is, as a result, of significant importance. A schematic for a typical rotary kiln is shown in Figure 17.14 with the design criteria for the Project rotary kiln shown in Table 17.18.

Figure 17.14 Rotary kiln - typical schematic

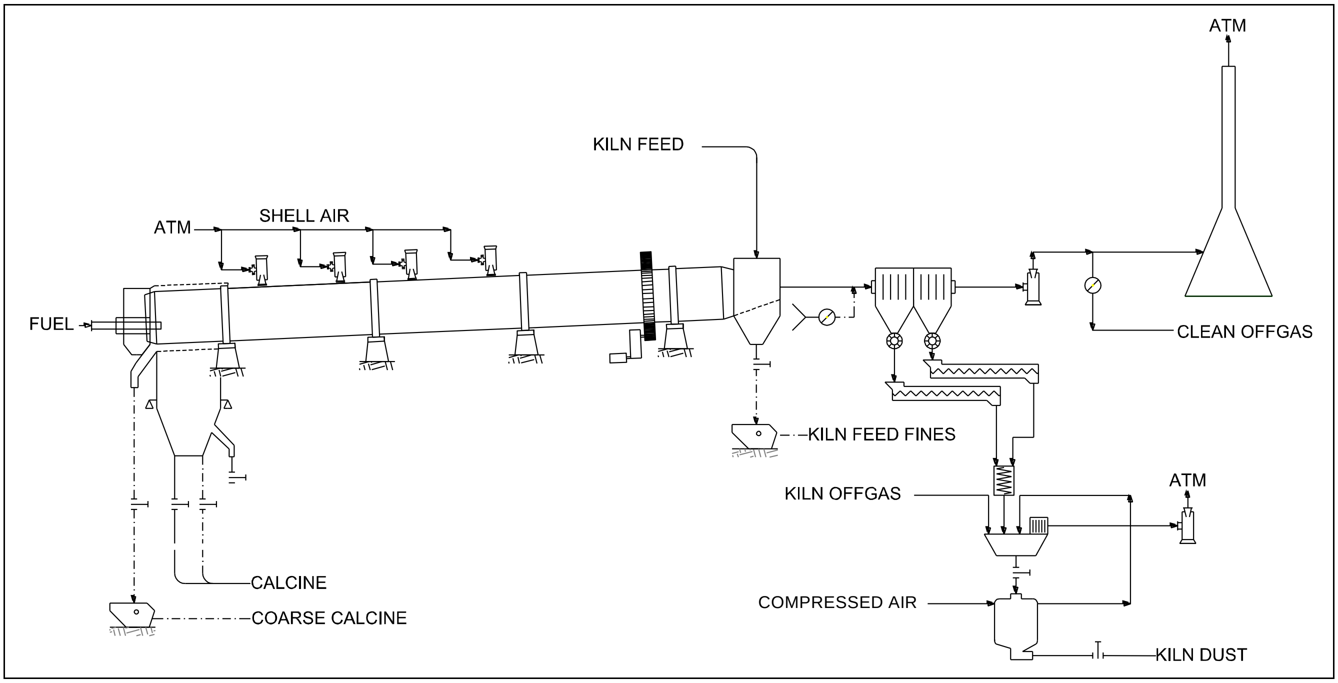


Table 17.18 Design criteria for rotary kiln – Base Case

Item	Source	Unit	Nominal value	Design Value
Kiln dimensions		diameter m	5.5	5.5
Kiln dimensions		length m	120.0	120.0
Operating hours		h/yr	7,884	7,884
Availability		%	90	90
Feed bulk density		t/m^3	1.25	1.25
Calcine product bulk density		t/m^3	1.20	1.20
Moisture content in the kiln feed		%	18.0	18.0
Arising dust - % of fresh ore - design		%	20	20
Ratio calcine / dry fresh ore	MB	ratio	0.86	0.86
Fresh ore feed rate to kiln		dry mt/hr	114.2	114.2
Recycle dust feed rate to kiln		dry mt/hr	22.8	22.8
Ore + dust feed rate		wet mt/hr	167.1	167.1
Ore + dust feed rate		dry mt/hr	137.0	137.0
Reductant coal feed rate		dry mt/hr	7.9	7.9
Reductant coal feed rate - % of fresh ore feed		%	7.0	7.0
Fixed carbon - % of fresh ore feed		%	4.4	4.4
Calcine production	MB - B284	t/hr	98.6	98.6
Ni content in the fresh ore		%	1.41	1.80
Nickel content in the calcine		%	1.64	2.09
Kiln Nickel recovery		%	98	98
Pre-reduction (Fe)		% Fe^{2+}/Fe_t	70	70
Pre-reduction (Ni)		$\%Ni^0/Ni_t$	20	20
Target Residual LOI		%	0.5	0.5
Target % fixed carbon in calcine		%	2.3	2.3
Calcine temperature at discharge		°C	850	850
Calcine temperature at feed to furnace		°C	825	825
Kiln off-gas dust recovery system		Type	ESP	ESP
Kiln dust temperature		°C	300	300
Kiln dust transportation temperature		°C	150	150
Kiln off-gas flow rate		Nm^3/hr	195,000	195,000
Kiln off-gas temperature		°C	350	350

17.4.4 Coal treatment and fuel oil

The design criteria for the coal preparation areas used both as reductant as well as fuel in the process, are shown in Table 17.19. The design criteria for fuel oil are shown in Table 17.20.

Table 17.19 Design criteria for coal production plant – Base Case

Item	Source	Unit	Nominal value	Design Value
Coal calculations				
Coal calorific value		kcal/kg	7,300	7,300
Granulometry - maximum		mm	80	80
Coal % carbon - Ultimate analyses		%	81.3	81.3
Coal % Fixed Carbon		%	62.6	62.6
Reductant				
Free moisture in the coal		%	6.5	6.5
Fixed carbon - FC - in the coal		%	81.3	81.3
Coal consumption - dry kg coal/t dry fresh ore	MB E203	dry kg/dry t ore	69.5	69.5
Coal feed rate to kiln feed		dry t/hr	7.9	7.9
Coal consumption - wet t/hr		wet t/hr coal	8.5	8.5
Fuel Coal - Kiln				
Fuel consumption @ 100% Partition	EB H68	MJ/t dry fresh ore	2005	2005
Coal calorific value		MJ/t coal	30,514	30,514
Coal consumption - % of total energy		%	90	90
Coal feed rate to kiln burner		dry t/hr	8.7	8.7
Coal consumption - kg/t fresh ore	EB C57	dry kg/t dry fresh ore	76.4	76.4
Fuel coal - Dryer				
Fuel consumption - base total wet feed		MJ/t total wet feed	411	411
Coal partition		%	90	90
Coal feed rate to the burner		dry t/hr	2.54	2.54
Coal consumption - wet kg/t wet feed	EB E27	kg/t total wet feed	13.6	13.6
Coal preparation plant				
Coal consumption - overall		t/hr	19.2	19.2
Coal consumption - overall		t/yr	150,202	150,202
Operating hours	TBC	h/yr	7,446	7,446
Coal production		t/hr	20.2	20.2

Table 17.20 Heavy fuel oil – Base Case

Item	Source	Unit	Nominal value	Design Value
Fuel Oil calorific value		kcal/kg	9,800	9,800
Fuel oil calorific value		MJ/t	40,964	40,964
Energy required - percentage of total - KILN		%	10	10
Energy required in the KILN	EB G129	MJ/t dry fresh ore	201	201
Energy required - percentage of total - DRYER		%	10	10
Energy required in the DRYER	EB G135	MJ/t total wet feed	41.1	41.1
Fuel consumption in the KILN - fresh ore		t/hr - of dry fresh ore	0.6	0.6
Fuel consumption in the KILN - fresh ore		kg/t dry fresh ore	4.9	4.9
Fuel consumption in the DRYER		t/hr of total wet feed	0.20	0.20
Fuel consumption in the DRYER		kg/t of total wet feed	1.00	1.00

17.4.5 Electric furnace and associated equipment

The electric furnace used for smelting the calcined ore feed is the most significant equipment item in the pyrometallurgical treatment of nickel laterite ores. A typical graphic depicting such a furnace is shown in Figure 17.15. As a result of the importance of the furnace, the design criteria for the furnace are of critical importance to the overall process design. Table 17.21 shows the design criteria as specified by IGEO for the electric furnace.

Figure 17.15 Typical electric arc furnace schematic

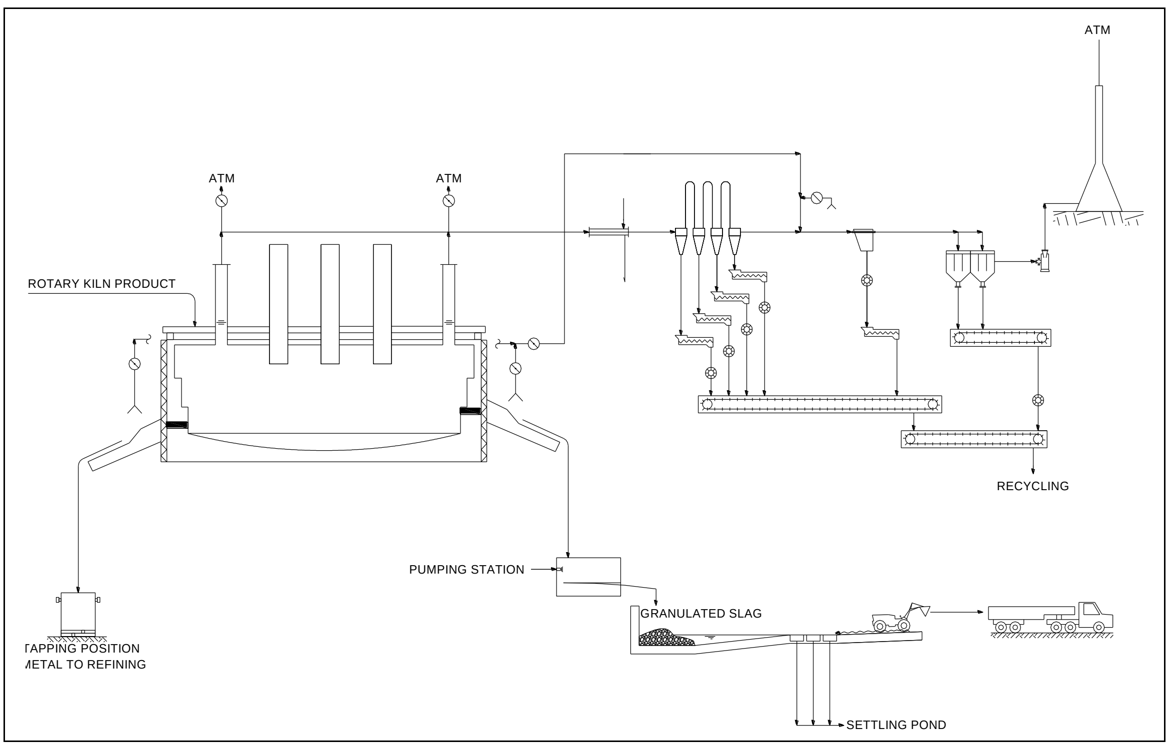


Table 17.21 Design criteria for electric furnace – Base Case

Item	Source	Unit	Nominal value	Design Value
Availability - RK - EF combined		%	90	90
Operating hours		h/yr	7,884	7,884
Ni recovery across furnace	MB E289	%	95	95
Calcine feed		t/hr	98.6	98.6
		% Ni	1.64	2.09
Fixed carbon in the calcine feed		% C	2.30	2.30
Mass ratio slag / calcine	MB	ratio	0.893	0.893
Slag product				
Slag tapping temperature		°C	1,550	1,550
Slag liquidus temperature		°C	1,380	1,380
Slag superheat delta T		°C	170	170
Ave rate over 24 hrs		t/hr	88	88
Ni content of slag	MB D303	% Ni	0.09	0.12
Tapping mode		hours per day	15-20	15-20
Analyses		%FeO	18.8	16.8
	MB	%SiO_2	52.0	53.6
	MB	%MgO	22.7	23.5
	MB	%Al_2O_3	5.9	6.1
	MB	Ratio SiO_2/MgO	2.29	2.28
	TBC	% Cr_2O_3		
Fe reduction across furnace		%	31.70	40.50
Metal product				
Metal tapping temperature		Tapping temp. °C	1,470	1,470
Metal liquidus temperature		Liquidus temp. °C	1,440	1,440
Metal superheat delta T		°C	30	30
Ni		%	20.0	20.0
C max		%	1.00	1.00
Si max		%	0.30	0.30
P max		%	0.10	0.10
S max		%	0.50	0.50
Cu	TBC	%	< 0.20	< 0.20
Co	TBC	%	< 0.67	< 0.67
Ni/Cu		ratio	> 100	> 100
Ni/Co		ratio	> 30	> 30
Fe-Ni production	MB C295	ave t/hr over 24hrs	7.7	9.8
Fe-Ni production	MB C294	Fe-Ni t/yr	60,217	76,873
Ni contained in Fe-Ni	MB D294	nickel t/yr	12,043	15,375

Item	Source	Unit	Nominal value	Design Value
Energy consumption	EB C132	kWh/t calcine	501	511
Energy consumption		kWh/h	49,399	50,385
Furnace power density	TBC	kW/m^2	230	230
Furnace characteristics				
calculated area required		m^2	215	219
circular furnace internal diam		M	18.0	18.0
rectangular furnace. length	TBC	M		
rectangular furnace width	TBC	M		
transformer rating	TBC	MVA	63.0	63.0
furnace power	TBC	MW	49	50
wall cooling		copper elements	copper elements	copper elements
Slag granulation				
water / slag - regular		ratio t/t	15	15
water / slag - design		ratio t/t	20	20
pressure - minimum		Bar	7	7
design water flow rate		m^3/hr	1,761	1,761
Moisture content		%	6	6
Size		Mm	< 6	< 6
Angle of repose dry		degree	35	35
De-dusting system				
Furnace gas temp at outlet	TBC	°C	1,000	1,000
process gas & slag / metal	TBC	Nm^3/hr	80,000	80,000
Calcine feed bins	TBC	m^3/hr	40,000	40,000
metal and slag launders	TBC	m^3/hr	TBC	TBC
Filter type		type	bag filter	bag filter
Solids recovery		% of calcine feed	2.0	2.0
Solids recovery		mt/hr	2.0	2.0

17.4.6 Refining

Table 17.22 Design criteria for refining – Base Case

Item	Source	Unit	Nominal value	Design Value
Operating hours		h/yr	8,322	8,322
Availability		%	95	95
Ni production in Fe-Ni	MB D396	t/yr	11,803	15,067
Refined Fe-Ni production	MB C396	t/yr	58,141	74,222
Ni Recovery Refining	MB D401		98.0	98.0
Crude metal temperature		°C	1,470	1,470
Refined metal temperature		°C	1,630	1,630
Refined metal composition				
Ni		%	20.3	20.3
C max		%	0.04	0.04
Si max		%	0.04	0.04
P max		%	0.03	0.03
S max		%	0.04	0.04
Cu		%	< 0,50	< 0,50
Co		%	<0,67	<0,67
Ni/Cu		ratio	> 40	> 40
Ni/Co		ratio	> 30	> 30
Ladle furnace				
Capacity		t	25	25
Freeboard		m	1.0	1.0
Refractory lining		type	basic	basic
Refractory consumption		kt/t Fe-Ni	5.7	5.7
Number of heats		heats/yr	2,409	3,075
Number of heats		heats/day	6.6	8.4
Energy consumption		kWh/t Fe-Ni	160	160
Furnace power	TBC	MW	5.6	5.6
Power Factor	TBC	ratio	0.85	0.85
	TBC	MVA - design	7.0	7.0
Gases				
Oxygen addition		Nm^3/t	10	10
Oxygen purity		%	95	95
Nitrogen addition		Nm^3/t	0.5	0.5
Reagents				
Lime - Total DeS and DeP		kg/t	102	102
FeSi		kg/t	3.0	3.0

Item	Source	Unit	Nominal value	Design Value
CaSi		kg/t	2.1	2.1
Al		kg/t	10	10
CaF2		kg/t	3	3
De-dusting system		m³/hr	150,000	150,000
Refining Slagging system				
Oxidising slag handling		water granulated	water granulated	water granulated
Oxidising slag		t/heat	1.6	1.6
Reducing slag handling		slag pot	slag pot	slag pot
Total slag production		t/heat	3.3	3.3
Total slag production		t/yr	11,803	15,067

17.4.7 Metal granulation

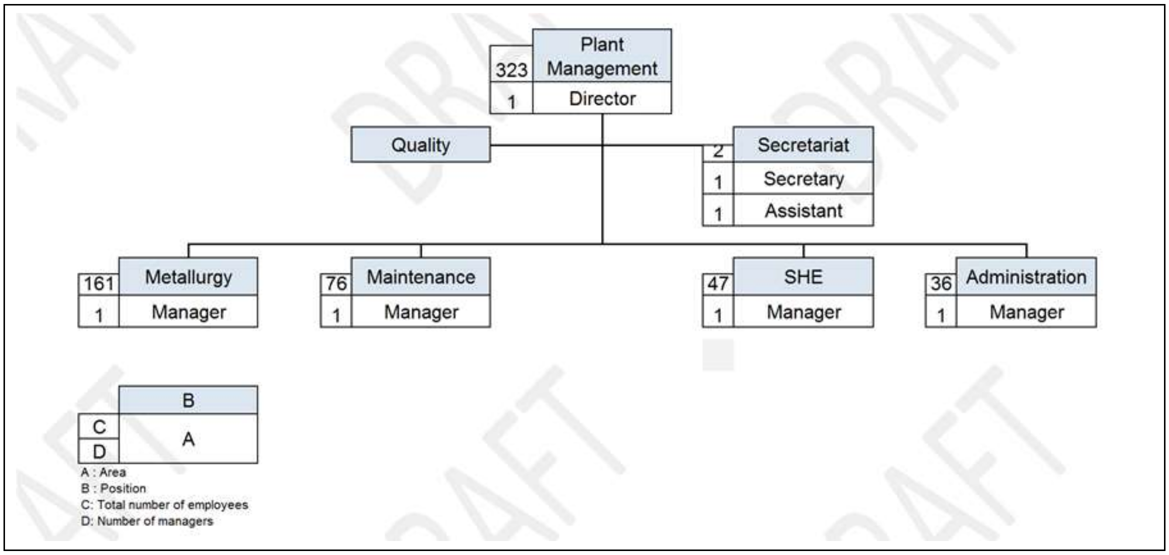
Table 17.23 Design criteria for metal granulation – Base Case

Item	Source	Unit	Nominal value	Design Value
Operating hours		h/yr	8,322	8,322
Availability		%	95	95
Technology		supplier	Uddeholm	Uddeholm
Refined metal grade - %Ni	MB D395	% Ni	20.3	20.3
Granulation rate		t/min	1.5	1.5
Temperature		°C	1,630	1,630
granulation time		minutes	35	35
Granulation tank height		m	4	4
Granulation tank diameter		m	3	3
Water for granulation				
Temperature		°C	30	30
Flowrate		m³/hr	700	700
Pressure		bar, minimum	6	6
Granulated Metal system				
Transfer system from tank		compressed air	compressed air	compressed air
Temperature		°C	50	50
Dewatering			hopper	hopper
Granulated metal dryer				
type		Rotary	Rotary	Rotary
Feedrate		t/hr	25	25
Final moisture		%	0.6	0.6
Fuel		gas / diesel		
Granulated metal screening				
Type		Rotary	Rotary	Rotary
Screen size		mm	2,5 and 50	2,5 and 50
Fines Fraction < 2 mm		%	3	3
Fines fraction		t/heat	0.75	0.75
Final Product storage capacity	TBC	t	3,000	3,000
Final Product storage capacity	TBC	days	18	14
Final product Lots	TBC		4	4

17.5 Manpower Requirements

The human resource requirement for (steady state) for the management and operation is given in Figure 17.16.

Figure 17.16 Human Resource Requirement



17.6 Mass and energy balances

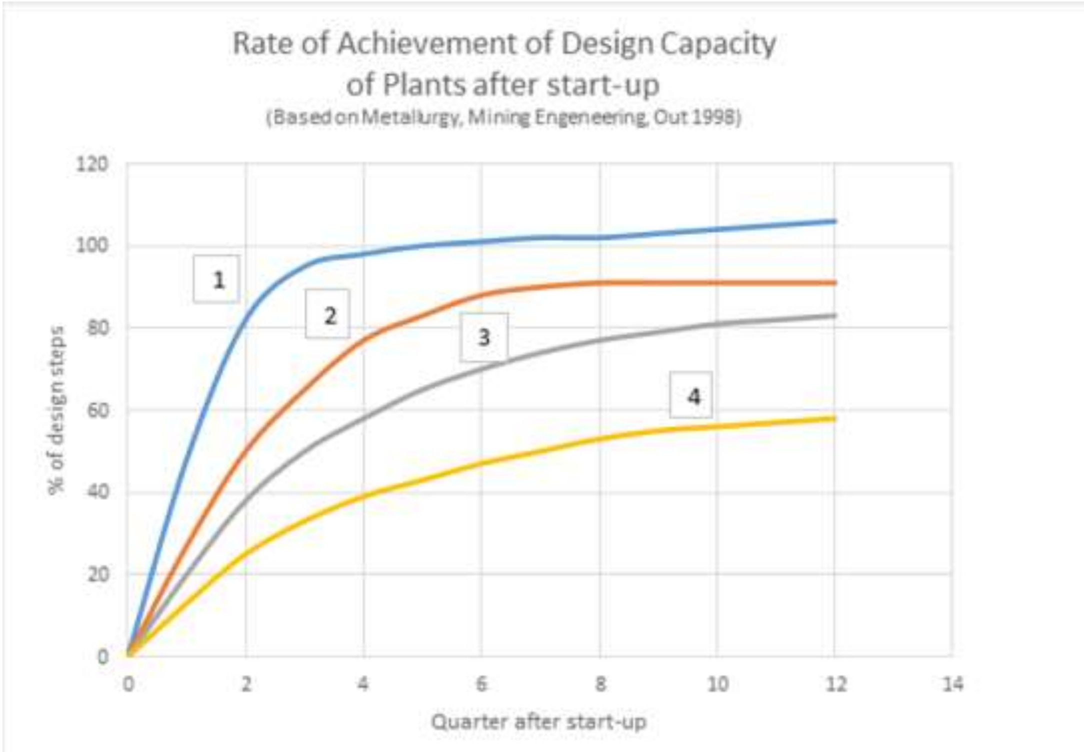
The process mass balances have been developed from an IGEO model and the energy balances from IGEO data base of information which includes data from other similar projects. These will be subject to verification when the budget proposals have been received from the main equipment suppliers for the Project but it is anticipated that the information in the IGEO balances conform to the level of accuracy of this pre-feasibility study.

17.7 Ramp-up

In determining what would be a reasonable ramp up period for Araguaia, consideration was given to ramp up curves developed by McNulty who had carried out extensive evaluations of ramp up rates actually achieved on projects. Fundamentally McNulty had classified the curve types depicted in Figure 17.17 as follows:

- **Type 1**: Process with mature technology which had been implemented in other projects.
- **Type 2**: Such a project was developing a prototype and with which the testing carried out was incomplete.
- **Type 3**: Similar to Type 2 but with very limited pilot testing and little knowledge of variation likely to occur in the feed material.
- **Type 4**: Characteristics similar to Types 2 and 3 but with a more complex flowsheet.

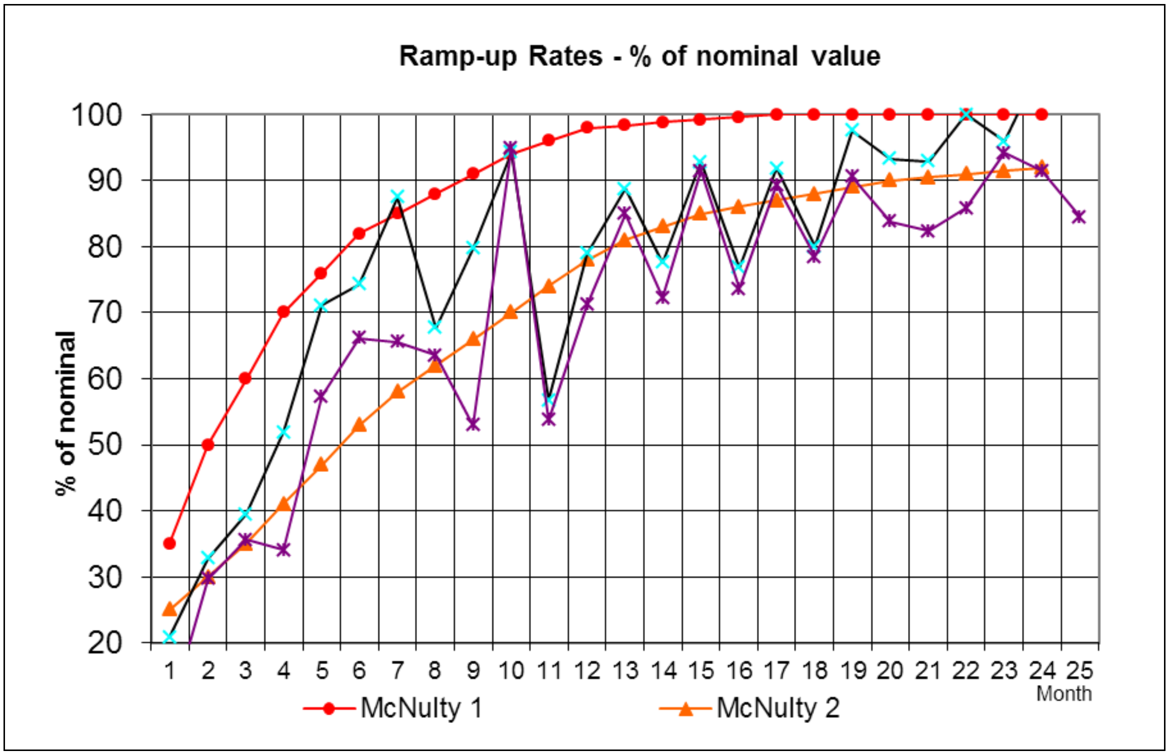
Figure 17.17 McNulty curves of plant ramp



Source: McNulty

For the Project, a ramp up curve between McNulty Types 1 and 2 is considered. These curves are depicted in the graphs in Figure 17.18 together with another but smaller Fe-Ni plant (Project A) which has been in operation for some 12 years.

Figure 17.18 McNulty curves 1+2 and Project A



Source: McNulty

Consequently the proposed ramp-up period for the Base Case is shown in Table 17.24. The Base Case depicts a rapid ramp-up rate because the size of furnace envisaged (50MW) has been well proven in many operations internationally.

Table 17.24 Preliminary processs ramp-up - Base Case

Year	Month	Month Tonnage % of Nominal	Ni Recovery Actual %	Ni Produced % of Ni Input
1	1	50	45	22.5
	2	55	65	35.8
	3	60	75	45.0
	4	69	78	53.8
	5	76	81	61.6
	6	83	83	68.9
	7	87	85	74.0
	8	90	87	78.3
	9	93	88	81.8
	10	95	90	85.5
	11	100	90	90.0
	12	100	93	93.0
2	13	100	93	93.0
	14	100	93	93.0
	15	100	93	93.0
	16	100	93	93.0
	17	100	93	93.0
	18	100	93	93.0
	19	100	93	93.0
	20	100	93	93.0
	21	100	93	93.0
	22	100	93	93.0
	23	100	93	93.0
	24	100	93	93.0

Notes: 1) Nominal plant feed 900ktpa dry ore
2) Ni production = % of Ni input in ROM ore

18 Project infrastructure

18.1 Summary

The scope of infrastructure described within this section includes the Project site requirements at Araguaia as well as the existing road infrastructure, identified as the preferred transport route to the Port at Itaqui in the city of São Luís for the import of coal for Project. Considerations have been given to the potential for using the port of Itaqui for the export of Fe-Ni product but this PFS considers the sale of Fe-Ni at the mine gate. Rail has also been described as there may be potential to utilise this infrastructure in future years but it does not form part of the PFS design solution.

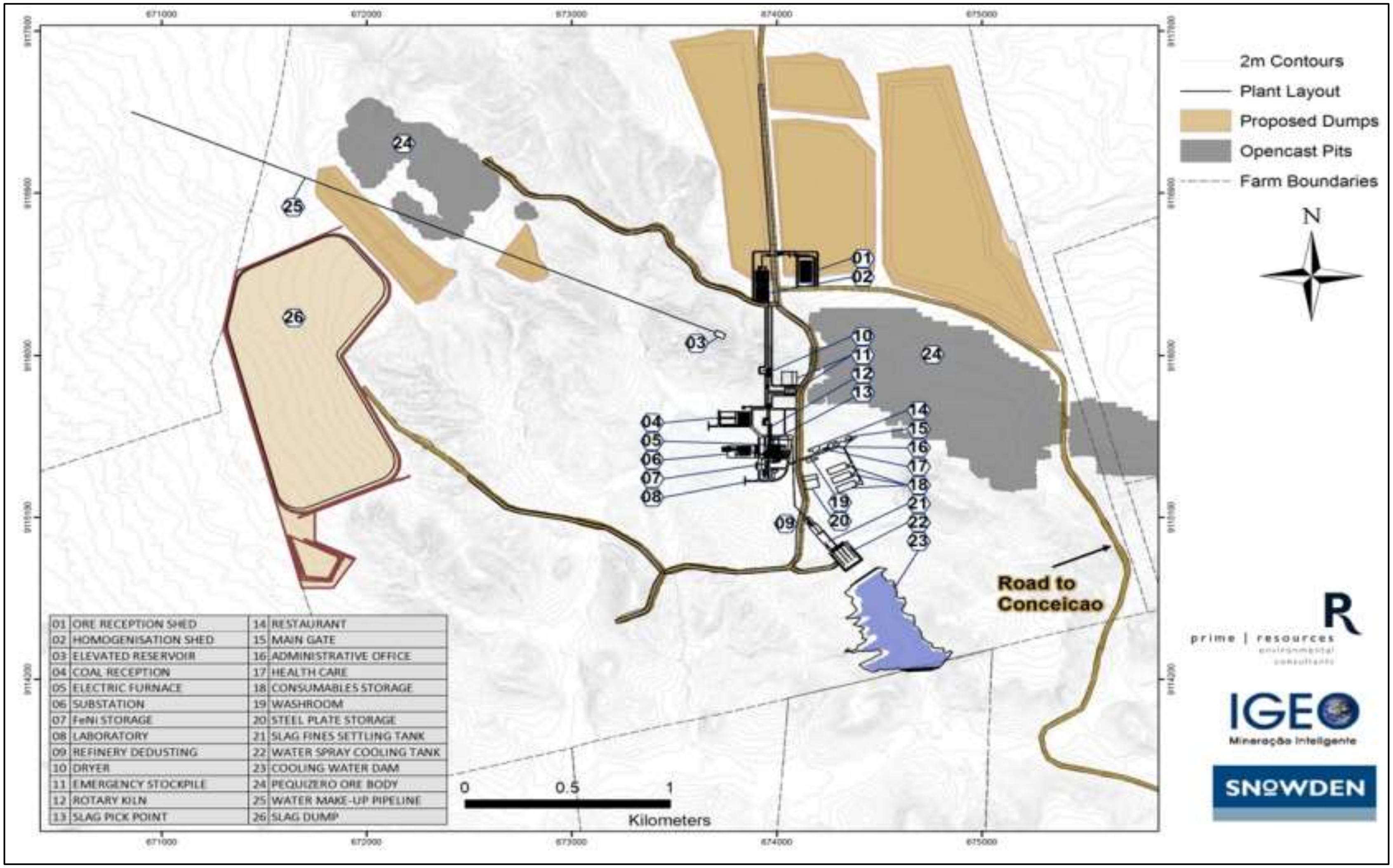
The proposed infrastructure for the Project will include:

- Access and site roads
- Water supply
- Coal storage facility
- Slag storage facility
- Security and fencing
- Water cooling facility
- Water treatment and minesite sewage
- Fire-fighting system
- Employee housing and transportation
- Data and communications infrastructure
- Power supply

The mining area is expansive and key infrastructure is associated with the processing plant and smelter facilities.

The Project site layout is presented in Figure 18.1

Figure 18.1 Site layout



Source: PRIME and IGEO

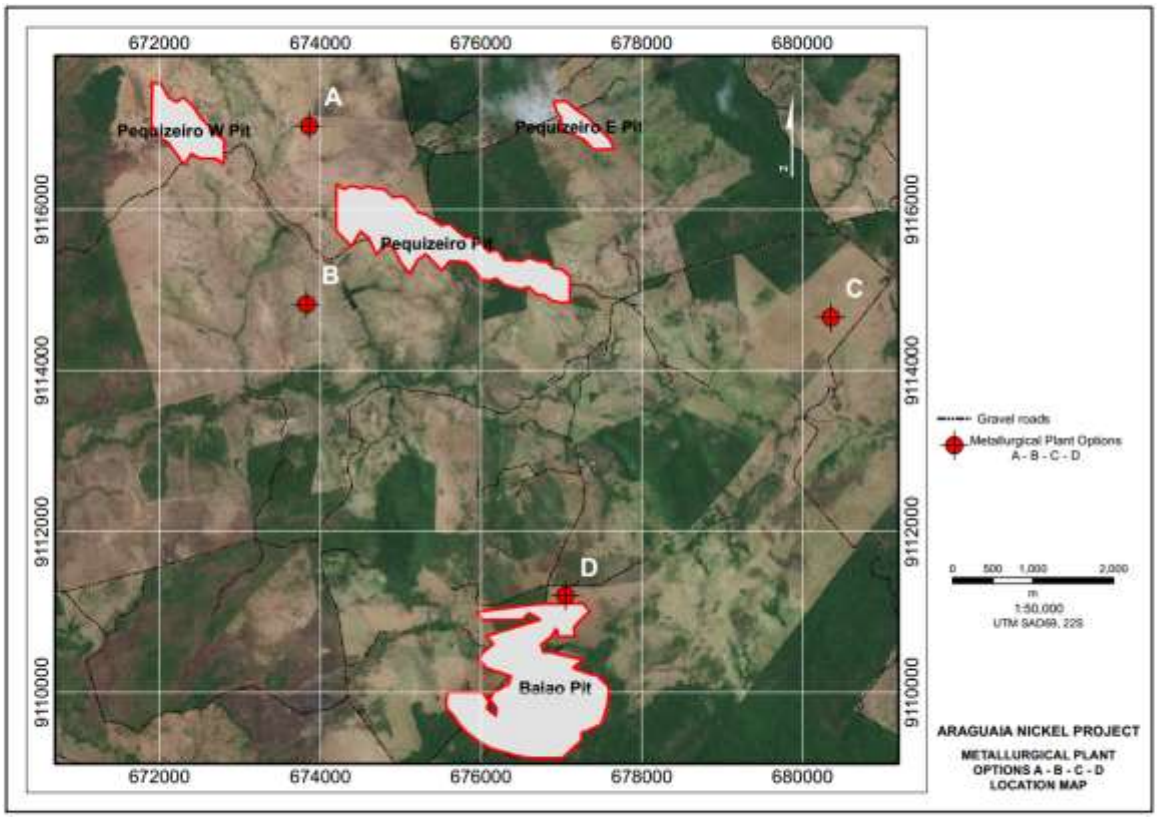
18.2 Plant site selection

The location of the plant site was selected on the basis of a trade-off which considered:

- environmental impact
- logistics and mine waste disposal
- topographical suitability
- ore transportation distances
- slag dumping facilitation
- suitability of location for provision of water cooling facility

Four locations were considered A, B, C and D as shown in Figure 18.2. Location B was selected as the optimal location.

Figure 18.2 Location of plant location options



18.3 Roads

The Project site is located approximately 200 km from the main BR-153 highway which is a Federal highway providing one of the key transport corridors for the area (Figure 18.3). The Project is supported by an existing road system within the state of Pará which includes a network of dirt tracks currently used for access by local farms.

Figure 18.3 Regional road infrastructure

Source: IGEO-Google Maps

This PFS considers the existing road system for all in-bound and out-bound logistics from the port of Itaqui in the city of São Luís and the Project site for the:

- supply of coal – 160,000 tpa (wet)
- supply of all consumables

plus

- potential transportation of Fe-Ni product – 74,222 tpa

Coal will be imported from the port of Itaqui which is 1150 km from the Project. Coal could also be imported from Vila do Conde Port in Barcarena as required. For the purposes of this PFS, Fe-Ni product is sold at the mine gate; however there is potential synergy between the deliveries of coal, from the port to the mine, with an opportunity to transport Fe-Ni product on the return journey for export.

The consumables required for the Project are indicated in Table 18.1.

Table 18.1 Annual consumption of main consumables

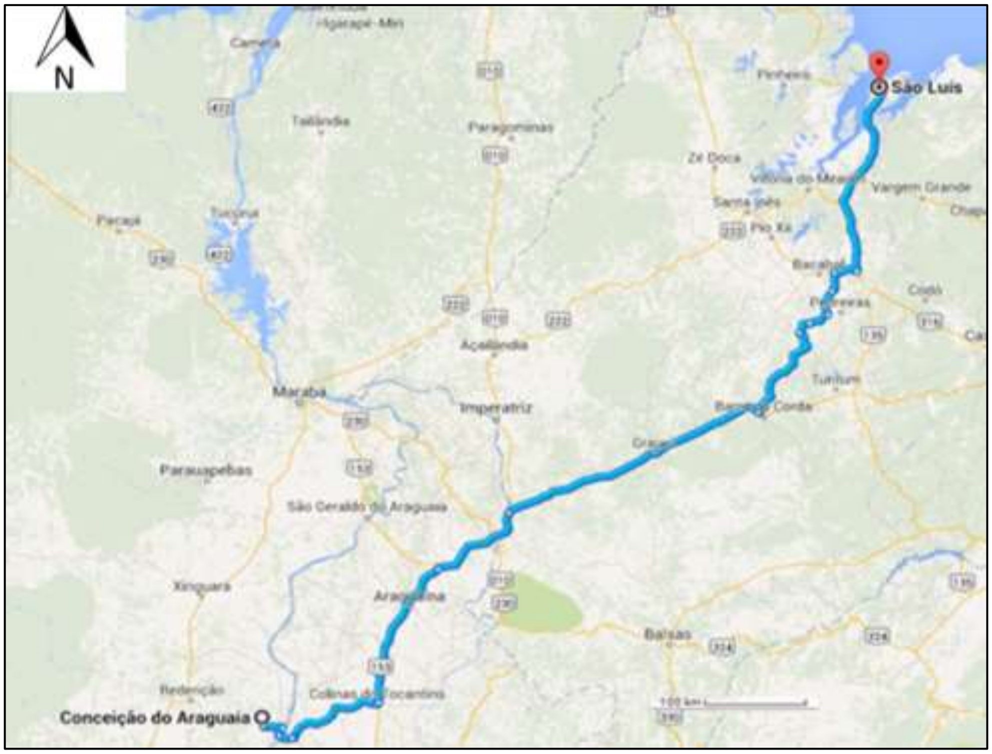
Description	Unit	Value
Fuel oil	t/year	6,302
LPG	t/year	181
Oxygen	Nm^3/year	681,942
Nitrogen	Nm^3/year	75,771
Lime	t/year	5,304
Magnesium oxide	t/year	606
Ferrosilicon	t/year	227
Fluorspar	t/year	1,061
Aluminum	t/year	758
CaSi wire	t/year	152
Coal	dry t/year	142,438

Other consumables include Söderberg electrode paste/bricks, various refractories (kiln, smelter, and refinery), spare parts, employee uniforms and PPE, etc.

The existing road system was considered the most cost-effective solution following a detailed options study that included road, rail and waterways. Figure 18.4 shows the considered road transport corridor.

A consideration for Fe-Ni product would be the potential economic benefits of increasing the utility of bulk haulage to include coal transport to the mine site and Fe-Ni product to port, on the return journey to port.

Figure 18.4 Road transport corridor to São Luís (port of Itaqui)



Source: IGEO-Google Maps

18.3.1 Main access road

Access to the Project site will be made by linking in with the existing road, the PA449 which is the main road connecting Conceição do Araguaia and Floresta do Araguaia, via a purpose built 44 km sealed road giving access for the town of Conceição do Araguaia where it is anticipated that the majority of the workforce will live and be sourced from. This road will provide access for all consumables for the Project site as well as all construction materials and machinery. Figure 18.5 shows the proposed location for the access road.

Figure 18.5 Map showing proposed access road to the Project site

Source: IGEO-Google Maps

18.3.2 Trunk roads

These roads provide access and haulage routes between the various open pits and will be designed to support on–highway truck traffic (35 t trucks) and off-highway traffic (40 t trucks). These are described in more detail in Section 16.15.

18.3.3 Ancillary plant site roads

Plant site roads will be developed to facilitate plant requirements. These will be sealed roads.

18.4 Rail

The rail network does not form part of the proposed PFS design solution. There may be potential utility subject to future rail developments. For the purposes of this study, rail has been included for information purposes only.

The region is supported by an integrated rail network which is owned and operated by Valec as described in Section 5.5.3.

Rail transport has been considered in the PFS as an option for bulk transport of coal and Fe-Ni. This would have involved trucking to and from the railhead at Colinas (200 km east of the Project), which is the main rail terminal for the North-South (FNS) railway. The North-South railway connects to the Carajás railway (200 km to the North) at Açailandia town which goes directly to Itaqui port.

The rail option has been excluded from this Base Case on economic grounds in favour of road haulage.

18.5 Port facilities

As stated, the existing network of federal highways establishes the transport corridor for all Fe-Ni, coal and other consumables. The transport solution for this PFS considers utilising bulk road haulage along these highways from the Project to the port at Itaqui (Figure 18.6).

Figure 18.6 Satellite image of the port at Itaqui

Source: Google Earth

The port administration is the responsibility of the Empresa Maranhense de Administração Portuária (EMAP) which administers ports across the states of Maranhão, Tocantins, SE of Pará, N of Goiás and NE of Mato Grosso).

The port at Itaqui, is reached by roads (BR-135 and BR-222) and railway (CFN – Northeastern Railway Company and EFC Carajás Railway Company and North - South railway). It may also be reached by river. These transport options were considered in an option study for the PFS with road haulage being the preferred option.

The port wharf is 1,616 m in length with depth varying from 9 m to 21.5 m, distributed in 7 bays and includes the facilities which are detailed in Table 18.2 and equipment as detailed in Table 18.3.

Table 18.2 Available facilities at the port of Itaqui

Facility	No	Description	Capacity
Warehouse 1	1	Covered facility	7500 m^2
Warehouse 2	1	Temporary covered facility for bulk solids	3000m^2
Fuel storage tanks	50		21,000 m^3
LPG tanks	2		8,680 m^3

Table 18.3 Available equipment at the Port of Itaqui

Equipment	No	Description	Capacity
Ship-loaders	2	bulk material handling	
Reach stackers	2	container handling	
Mobile crane	1	material handling	64t
Crane (tracked)		material handling	6.3t
Fork-lift	20	material handling	

The port facility has two private terminals, one of them is owned by VALE for iron ore from Carajás and Fe-Ni from Onça e Puma. The wharf has a capacity for 450,000 DWT ships, one opened storage yard with 125,000 m^2 for iron and manganese ore. The other terminal, which 252 m long, is owned by Alcoa, Billiton, and Novelis for bulk material shipping, such as bauxite and aluminium ingots.

IGEO has not held discussions with the relevant port authorities. However, it is understood that the port authorities will have capacity for the project requirements as the port facilities at Itaqui has been underutilised.

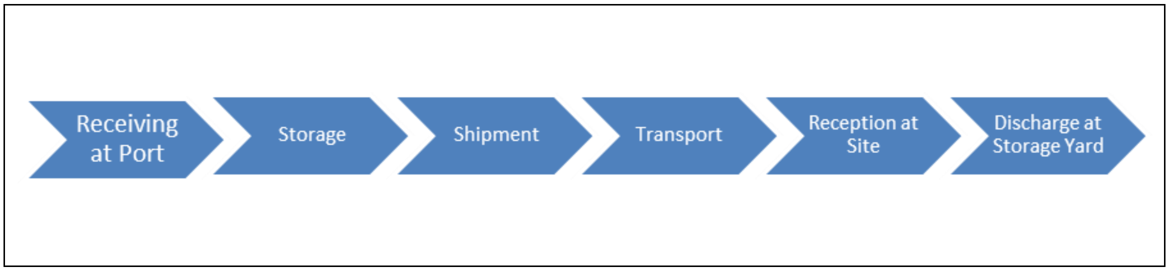
18.6 Supply chain solution (Coal and Fe-Ni)

The supply chain solution for coal (and Fe-Ni) is described in the following sections.

18.6.1 Coal supply

The coal supply chain for the Project includes coal reception at port, port operation, storage, shipping, transport, and receiving and unloading at Project site, as depicted in Figure 18.7.

Figure 18.7 Coal supply chain



There are two port options for coal importation. The first one is Itaqui Port, in São Luís (MA), and the second one is Vila do Conde, in Barcarena (PA).

Storage at the port is provided by a cargo terminal, and then the coal is transported to site, where it is received and unloaded into the coal storage yard. Each alternative route has been considered in this study.

Coal handling at the port will be carried out by third parties.

18.6.2 Fe-Ni export

The PFS considers selling Fe-Ni product from the mine gate. No costs have been included for the transportation of Fe-Ni product to the port or storage and shipment thereof. However, the PFS did consider potential solutions which would be elaborated in subsequent studies.

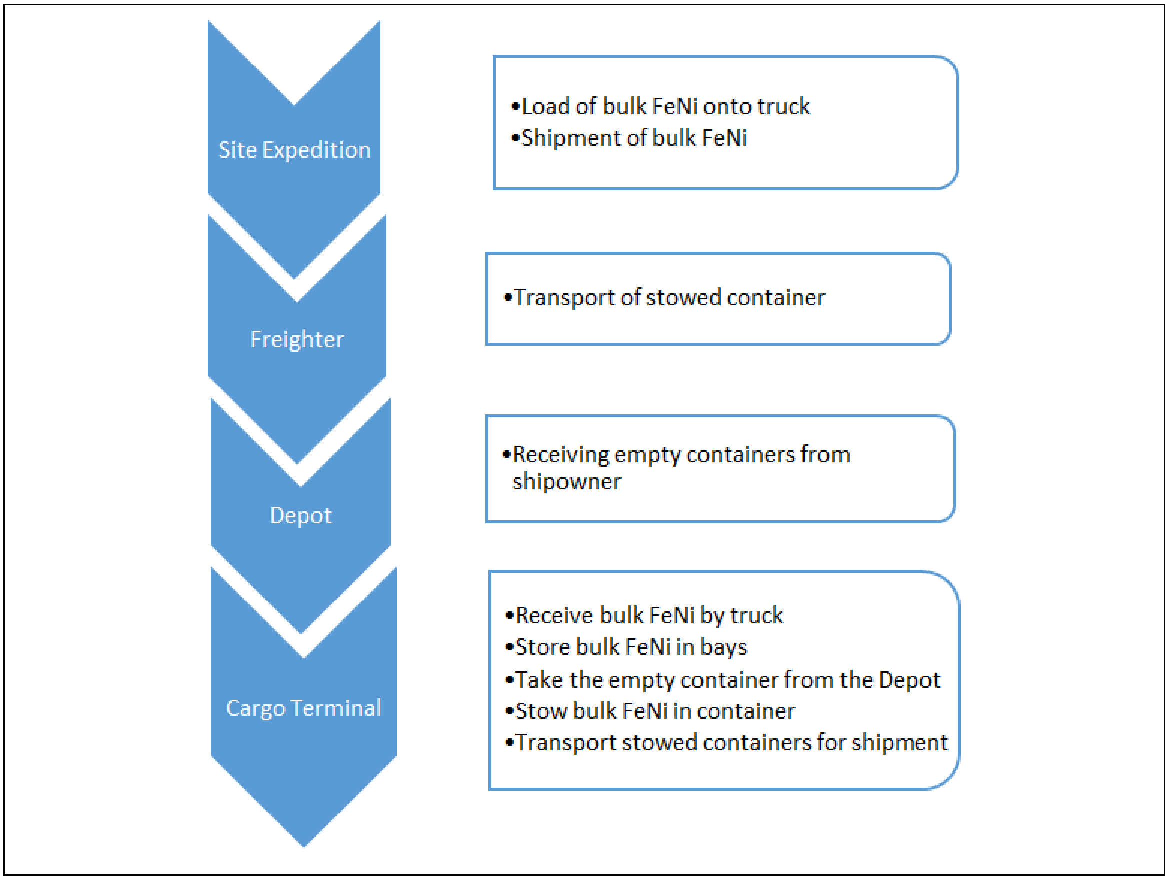
Two alternatives were considered for exporting Fe-Ni product from the Project site:

- container stowage at Project site and then road haulage to port
- road haulage to port and stowage performed by a logistics operator in a cargo terminal

The preferred solution was to consider road haulage of Fe-Ni to port and then to utilise a logistical operator to load containers then onto ships – this process is described below.

The process consists of renting the containers, which will be stored in a depot by a logistical contractor. Fe-Ni is delivered in bulk to the cargo terminal by road and stored in storage bays ready for export. The logistics contractor will then collect the empty containers from the depot and load them with Fe-Ni product and stow them, seal them and, after concluding administrative procedures, ship them to the port terminal for further ship loading, as per Figure 18.8.

Figure 18.8 Fe-Ni export process, container stowage at port



Similarly to thermal coal supply, the battery limits considered in this study include haulage from the Project site to the Itaqui port. An alternate port would be Vila do Conde Port in Barcarena (PA) but only the port at São Luís is considered here. Other assumptions include 100% of production exported through the above mentioned ports, and maximum Fe-Ni production to be exported of 74,222 tpa.

No costs were considered for container rental, as the current market practice provides 15 days free of charge. In addition, there may be opportunity to negotiate improved terms with the transport providers due to the large volumes being considered.

Fe-Ni handling at Port would be conducted by third parties.

18.6.3 Supply chain synergy

This PFS considers the savings provided by road haulage for coal and Fe-Ni shipment. The proposed logistical solution for this PFS considers the delivery of coal by truck and then loading of empty trucks with Fe-Ni product for road haulage to the Port. In this manner, economies will be made by increasing the utilisation of road haulage.

The coal will be received and unloaded at the Port, transported to the cargo terminal, where the coal is stored and then loaded onto trucks, and transported to the site. At the project site, the coal is unloaded into coal storage facility and then washed to prevent contamination, loaded with Fe-Ni and then taken by truck to the cargo terminal at the port of Itaqui. At the cargo terminal, Fe-Ni will be stored in prepared bays and then stowed in 20 ft containers ready for export. The containers will then be hauled by truck to the port terminal to be loaded aboard the ship.

18.7 Water supply

Water will be supplied via a 10 km purpose built pipeline from the river Arraias do Araguaia where a pumping station will provide adequate water requirements for the processing plant and smelter, as required, as make up water.

The management of the water used for processing is such that all water, whether it is for cooling or for metal and slag granulation, will be contained in a closed system, with the cooling water dam the main reservoir where all water flows to. The cooling water dam receives the water at 40 C, after being partially cooled in a spray cooling tank, used for cooling of slag and metal granulation water where around 60% to 70% of the total energy input to the electrical furnace is contained. It is cooled down to 30°C to 35°C before being pumped back to the elevated reservoir for recycling. Water make up is performed at an elevated reservoir which will further reduce the water temperature.

Plant and potable usage has been estimated for the Base Case as indicated in Table 18.4 (expressed as m^3/hr). The estimate is for a Base Case processing rate of 0.9 Mtpa and use of a pre-constructed plant.

Table 18.4 Base Case water requirements

Use	Average (m^3/h)	Maximum (m^3/h)	Peak (m^3/h)
Raw	130.97	123.96	137.97
Softened	42.52	38.44	46.59
Demineralised	12.84	12.84	12.84
Potable	33.33	33.33	33.33
Total	219.66	208.58	230.74

Water requirements in addition to process usage will be for road works, dust suppression and mobile plant wash-down during the wet season and is estimate at between 30 m^3/hr and 40 m^3 /hr.

Other potential water sources are discussed in Section 18.12.2.

18.8 Coal storage facility

A covered coal storage facility will be provided and will consist of a flat area for the containment of 30,000 m^3 of coal. This designated area will be compacted and engineered to accommodate a drainage scheme, appropriate bunding and containment to accommodate and mitigate against potential pollution from coal run-off into the immediate environment.

Coal will be transported to site and directly discharged onto a receiving hopper with grizzly, located in a shed. Coal will then be extracted from the hopper and transported to the coal stacker or tripper conveyor which will then discharge the coal to a covered storage area with capacity for 30,000 m³ (wet). A front-end loader will reclaim the coal from one of the two stockpiles and discharge it into the coal crusher feed hopper.

18.9 Slag storage facility

Three sites were identified as potential slag storage areas. These sites were further investigated with regards to the Brazilian national legislation's Protected Areas National System, which establishes environmentally protected zones. The selected site is described herein. Figure 18.9 shows the area in consideration.

Figure 18.9 View of selected slag dump site

Source: HZM, 2013

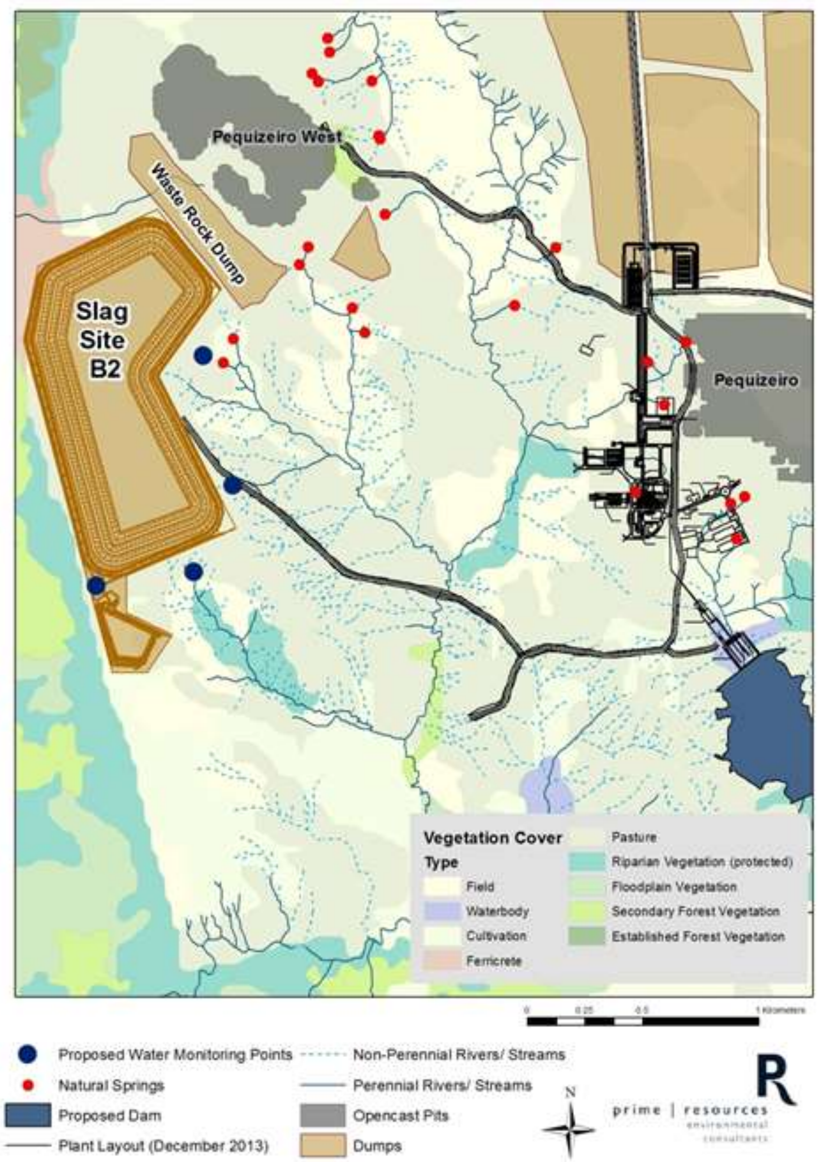
The total volume of slag produced over the 25 years is estimated at 9.93 Mm^3. The proposed slag disposal facility has a total surface area of approximately 78.7 ha of which the in-situ material will be ripped and re-compacted to form the base layer and liner. There will be four benches, each with a vertical height of 15m, a final maximum design elevation of 60 m from base and an overall slope of 1V:3H. The slag disposal facility has a maximum total storage capacity of 25.80 Mm^3 at a placed density of 1.67 m^3. Figure 18.10 shows the location of the slag dump in relation to the plant site.

The slag storage area layout includes a toe drain that runs along the north western and western perimeters of the area as well as a toe wall to form the boundary of the slag storage area. Adjacent to the slag storage area is the drainage management area consisting of a catchment paddock, drains and a solution trench. The catchment paddock drains are partitioned by catchment paddock division walls every 50m.

A solution trench is positioned on the western edge of the slag storage area. The solution trench leads to a silt trap and water retention dam, located to the south of the slag storage area. A decant outlet, located within the water retention dam, is connected to a pump system, which enables water to be pumped from the water retention dam back to the plant for reuse. A storm water diversion trench is proposed to flank the water retention dam along its north east perimeter. Figure 18.11 provides a detailed plan of the slag dump and water retention dam.

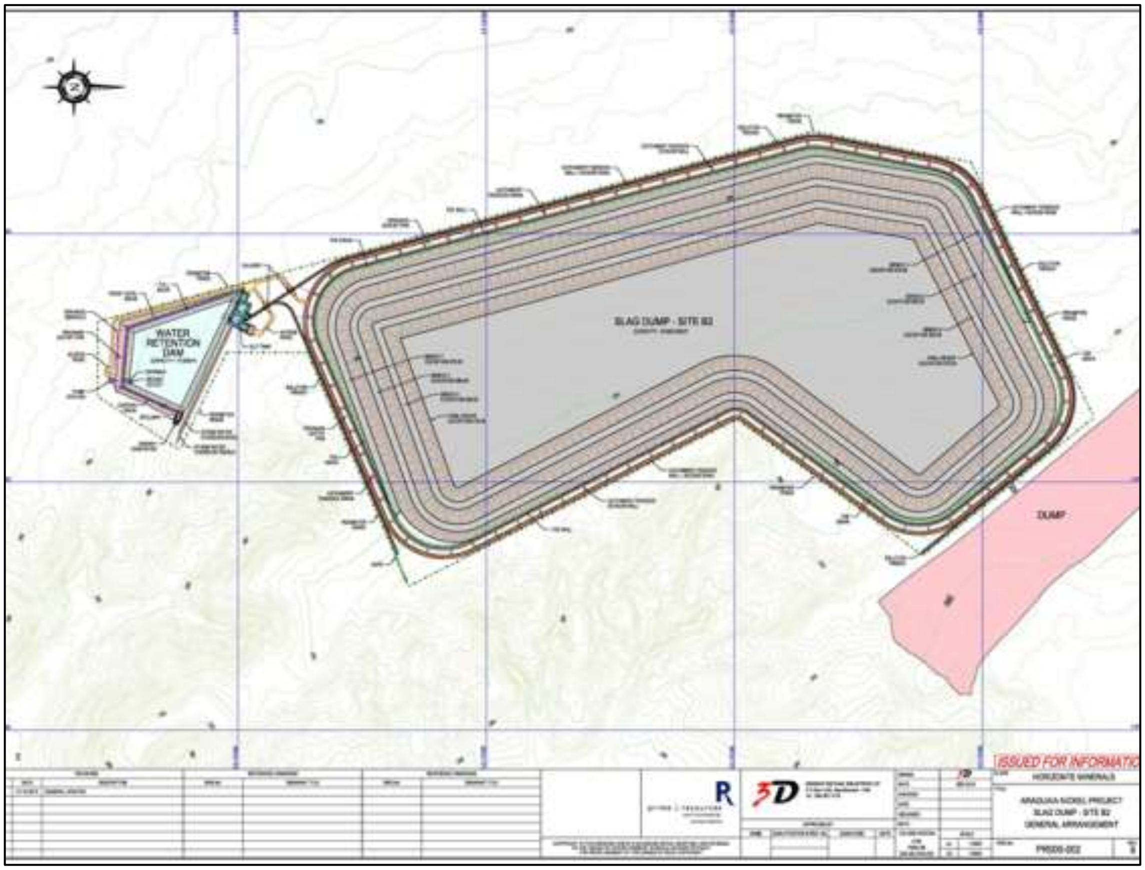
A geotechnical evaluation for the siting and design of the slag dump was completed as part of the PFS.

Figure 18.10 Location of slag dump in relation to plant



Source: Prime

Figure 18.11 Detailed plan of slag dump and water retention dam



Source: Prime

18.10 Security and fencing

The production area will be delimitated by a fence with higher security required at the production centres including key fixed assets. The requirement is not only to minimise any potential security risk but also to minimise any potential livestock entering the area and posing a danger.

An entrance checkpoint and barrier will be provided at main access points.

During the construction period, it is anticipated that the main contractors will be responsible for all security features to minimise any delays by the potential theft of key equipment. This is a normal feature for all developing mine sites and has been considered.

18.11 Water cooling facility

The purpose of the water cooling facility is to reduce the temperature of the quenching water used during slag granulation. Below the slag fines settling tank, water sprays are used in a spray cooling tank to reduce the temperature of the slag granulation water to below 40 °C degrees.

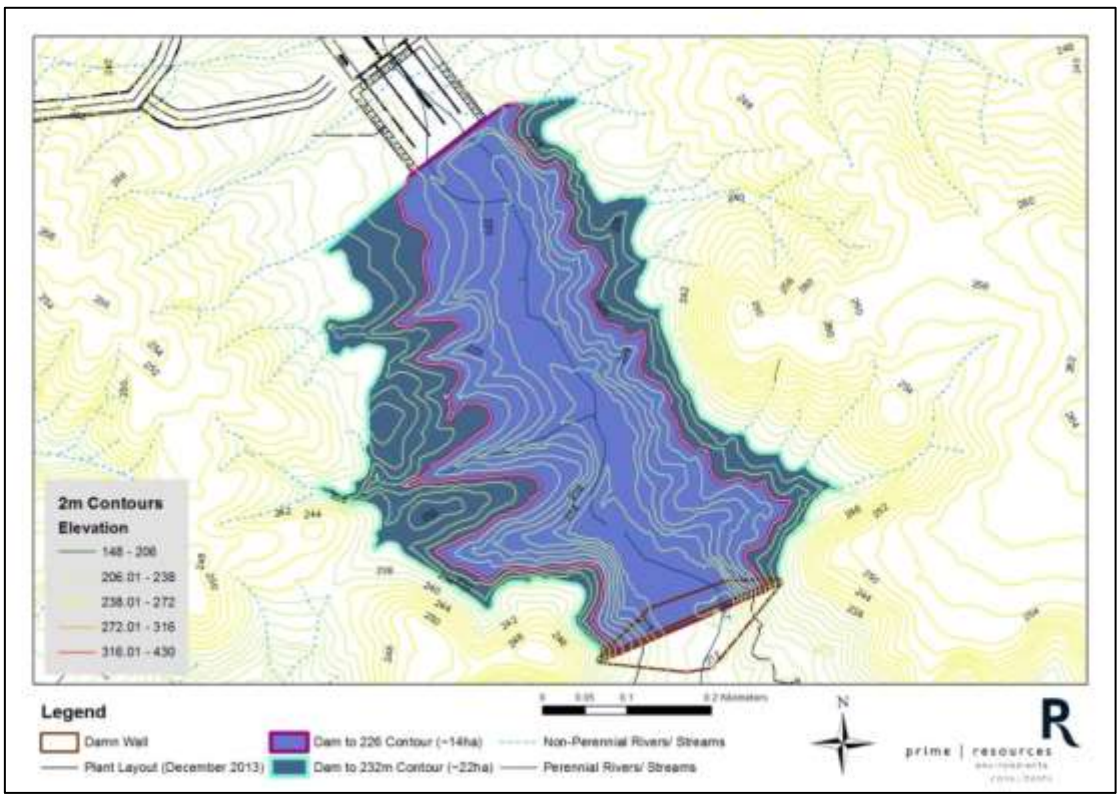
The area selected for the Base Case is immediately to the South of the Plant facility and is shown in Figure 18.1.

Cooling water from the spray cooling tank overflows into a large water cooling dam, built across the valley below the spray cooling tank. This water cooling dam is large (~15 Ha) with a substantial surface area to allow the temperature of the water to reduce from approximately 40°C to around 30°C.

The proposed water cooling dam construction is to be a homogeneous earthfill embankment dam, with a side channel stormwater overflow (or spillway) to manage water flow from excessive rain such that the dam itself never overflows. The level in the dam is controlled by a pumping system from the water pumping station at the Arraias do Araguaia River. At the embankment of the dam, a pumping station recycles the cooled water to the elevated reservoir or directly to the slag granulation nozzles for slag granulation, metal granulation, minor consumers and emergency cooling of high temperature components of the furnace.

The construction of the earthfill embankment will be from selected sandy-clay material sourced from inside the dam basin itself, thereby also increasing the overall capacity of the dam basin by using the material from inside the basin and along the flanks of the dam. The upstream slope (1:3) of the earthfill embankment will be covered in a geo-fabric and rip-rap from the selected ferricrete boulders sourced on site. The downstream slope (1:3) will be grassed. Along the downstream toe of the embankment, there will be a series of drainage systems to monitor and collect any seepage through the embankment and manage water flow. The side channel spillway will be designed for a 1:100 year flood event. Stormwater inflow into the dam will be limited to rainwater falling in the dam. A detailed plan of the water cooling facility is provided in Figure 18.12.

Figure 18.12 Detail of water cooling facility



Source: Prime

18.12 Water treatment and minesite sewage

18.12.1 Water requirements

The three main project water requirements are for:

- metallurgical processing and mine usage
- haul road maintenance and ROM dust suppression
- potable supply.

18.12.2 Water sources

Four principal water sources have been identified:

- river water
- mine dewatering of groundwater inflow
- rain catchment on plant waste dumps and the open pit
- catchment from the slag dump enclosure.

Mine dewatering

No hydraulic testing has been conducted to provide a method of modelling the rate of increase in groundwater inflow with mine depth or lateral progress.

Estimates based on inferred values of storativity and transmissivity for the rock formation to a depth of 40m suggest that groundwater inflows will range from 10 m^3/hr in the first year, to 40 m^3/hr by year twenty.

Open pit and mine infrastructure rain catchment

Estimates on recoverable rainfall runoff suggest that wet season runoff could provide an average of 116 m^3/hr. This supply would be highly variable over short periods of time, depending on rainfall intensity factors. A more detailed estimate on catchment probability will be possible on availability of more detailed climate data in subsequent studies.

Slag storage impoundment

This impoundment has potential to store water and allow for an even rate of extraction when compared with the more intermittent pumping regime that will be required to cope with sporadic storm catchment to the pit.

It is proposed that the slag dump will have a clay-lined base which will allow for the accumulation of rainfall. Controlled seepage, through walls and floor, will be directed to a catchment dam and returned for plant use. Controlled seepage through the clay-lined base will enter the groundwater system, some of which will resurface as downstream spring flow and will be available for pumping back to the plant as required.

Recoverable water from a designed slag dump decant system has been estimated to range between 40 m^3/hr to 50 m^3/hr.

Minesite sewerage

The preliminary design for a sewerage treatment system has been sized for an estimated peak workforce of 400 people expected for the operational phase of the project. Sewage will be treated septic tanks, sewers and filters, duly sized for service throughout the project life time.

For the construction phase, the use of the same sewage treatment system as in the operational phase is being considered.

18.13 Fire-fighting system

All the main plant areas of the Project will be equipped with fire-fighting and prevention systems, in accordance with guidelines set by the Brazilian Standards Association (ABNT). All equipment, fixed and movable assets will be equipped with fire-fighting and prevention systems to ensure safety of all employees and equipment.

The following components are part of the fire-fighting and prevention systems, according to the classification of risks involved in each facility and for specific utilization.

- hydrant system with hose points
- automatic sprinkler systems
- foam-based systems
- fixed CO_2 extinguishing systems
- portable fire extinguishers
- fire water storage tank
- fire truck

18.14 Employee housing and transportation

For the purposes of this PFS, ablutions, refectory and associated administration and management facilities will be provided on site. It is anticipated that the town of Conceicao do Araguaia, which is 35 km from the proposed site, will provide the main accommodation centre for all employees.

Transportation will be provided by a regular bus service which could be outsourced to a local contractor.

18.15 Communications

The mine will be designed to accommodate world-class communication facilities. The project is provided with existing communication facilities in the immediate vicinity which will be extended to accommodate the Project as required.

Enterprise resource planning (ERP) systems would be considered to ensure efficient procurement, time management and overall resource planning for the facility. This would accommodate efficient in-bound and out-bound logistics to minimise cost of inventory of product and consumables.

18.16 Power supply

18.16.1 Overview of the Brazilian energy market

Between 2003 and 2004, the federal government determined the basis for a new model for the Brazilian electricity sector, which is defined by Laws nº 10,847 and 10,848 of 15 March, 2004, and Decree nº 5,163, of 30 July, 2004.

In institutional terms, the new model established the creation of the agencies as shown in Table 18.5.

Table 18.5 Federal agencies for Brazilian electricity supply managment

Agency	Mandate
Energy Research Company (EPE),	Long-term planning of the electricity sector
Electricity Sector Monitoring Committee (CMSE),	Monitoring and assessment of the security of power supply
Electric Energy Trading Chamber (CCEE)	Maintain continuity with respect to commercialization of electricity in Brazil and to monitor and control the activities of the Wholesale Electricity Market (MAE).

With respect to the commercialization of electricity and its overall supply, two entities were established for purchasing and selling electricity: the Regulated Contracting Environment (ACR), where Electricity Distribution and Generation Agents participate; and the Free Contracting Environment (ACL), where Generation Agents, Traders, Importers and Exporters of electricity and Free Consumers participate.

18.16.2 Power distribution and transmission (national perspective)

The electricity distribution system in Brazil is regulated by ANEEL, which is governed by directives established by laws approved by the National Congress and the decrees established by the Federal Government.

The private sector is responsible for 80% of the power distributed in the country. By the end of March 2013, the Brazilian electricity transmission network had established 107,400 km of power grid.

The transmission system for the North region supplies the states of Pará and Tocantins. These are described below:

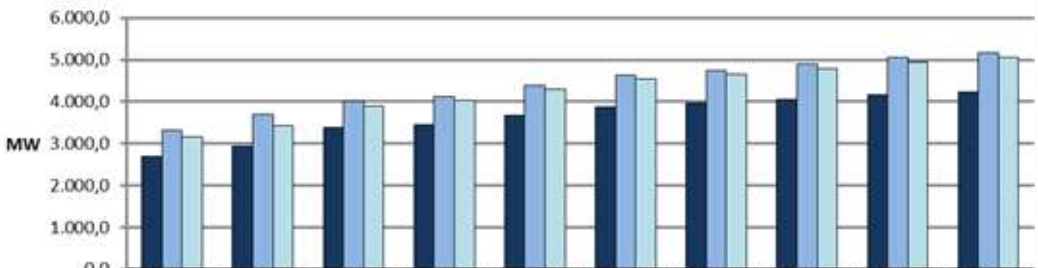
18.16.3 Pará state power capacity and distribution

Power distribution for the state of Pará is provided from the grid at voltage levels between 500 kV and 230 kV.

The total amount of currently installed and operating power in the state of Pará, in the area supplied by the electrical grid or Sistema Integrado Nacional (SIN), is approximately 8,866 MW, 95.4% of it being provided by hydropower. The load profile for the state of Pará forecast for 2012-2021 considers three levels (heavy, medium and light) which are presented in Figure 18.13. The average annual growth of power requirements is approximately 5.4%.

For clarification, Light, Medium and Heavy refers to a period of time during a typical day. It is expected that Heavy may be Lower than Medium due to the fact that the Heavy period corresponds to the period of time 19:00 up to 22:00, and a corresponding higher price of the energy. It would be cost-efficient operational practice to consider lowering the power consumption of furnaces during this period in order to reduce cost.

Figure 18.13 Load evolution, Pará state



	2012	2013	2014	2015	2016	2017	2018	2019	2020	2021
Light	2.690,5	2.928,5	3.376,0	3.446,3	3.680,3	3.880,4	3.967,6	4.054,2	4.172,0	4.240,5
Medium	3.317,0	3.687,0	4.006,9	4.122,4	4.396,0	4.638,4	4.753,7	4.895,9	5.057,2	5.164,9
Heavy	3.154,8	3.428,3	3.897,3	4.033,6	4.304,3	4.539,7	4.663,0	4.786,6	4.942,4	5.047,1

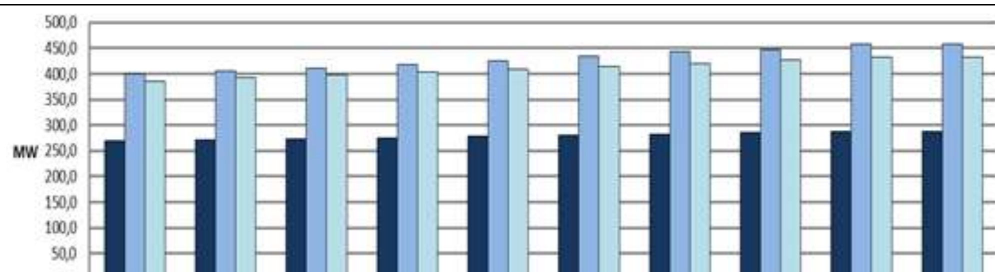
Source: MME, 2013

18.16.4 Tocantins state power capacity and distribution

The power distribution for the state of Tocantins is provided by a Basic Grid at voltage levels of 500 kV and 230 kV, the main supply point being Miracema SE at 500 KV. The total amount of currently installed and operating power in the state of Tocantins is around 1,500 MW, 100% of it being hydropower.

The load requirements for the state of Tocantins foreseen for the period 2012-2021, for three levels (heavy, medium and light) is presented in Figure 18.14. It can be seen that the average annual growth of the heavy load is approximately 1.3%.

Figure 18.14 Load evolution, Tocantins state



	2012	2013	2014	2015	2016	2017	2018	2019	2020	2021
Light	269,3	271,5	273,6	275,9	278,1	280,5	282,8	285,0	286,9	286,9
Medium	400,4	405,1	411,0	417,9	425,2	434,4	442,6	446,7	458,3	458,3
Heavy	385,3	392,0	397,8	403,3	409,0	414,8	420,0	426,4	432,5	432,5

Source: MME, 2013

18.16.5 Electrical energy source

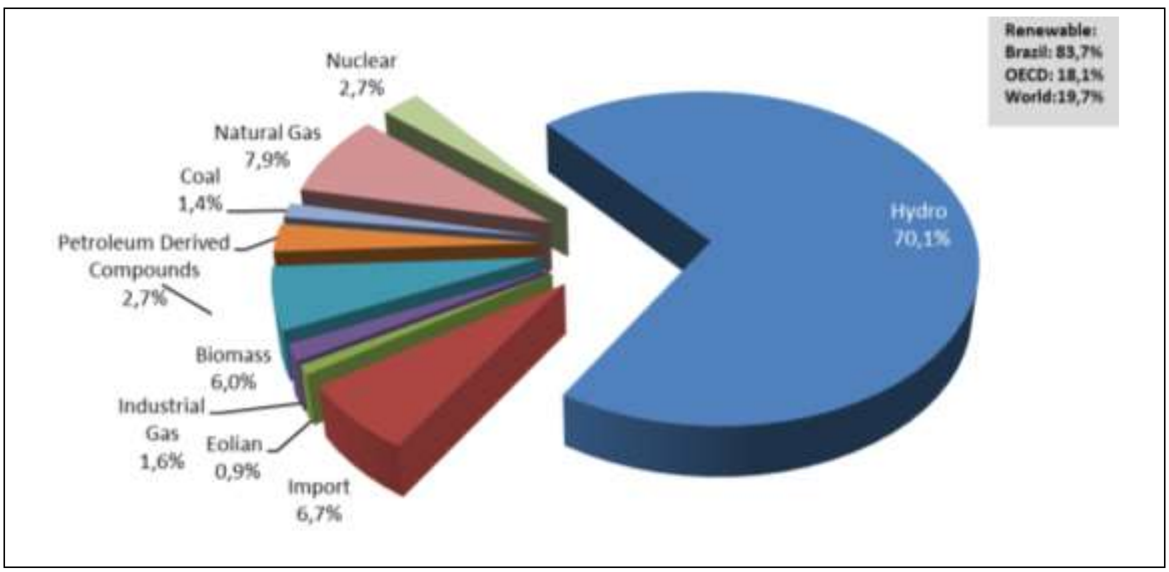
Table 18.6 gives the total amount of electricity generated in Brazil from various sources. This index is known as Electrical Energy Internal Offer (OIEE) which is depicted for the period of 2011 and 2012 in Figure 18.15. Most of the electricity in Brazil is produced via hydroelectric generation.

Table 18.6 2012/2011 Sources of electrical power in GWh

Source	GWh		12/11 %	Structure %	
	2011	2012		2011	2012
Hydraulics	428.333	415.342	-3,0	75,5	70,1
Nuclear	15.659	16.038	2,4	2,8	2,7
Natural Gas	25.095	46.760	86,3	4,4	7,9
Mineral Coal	6.485	8.422	29,9	1,1	1,4
Oil Products	12.239	16.214	32,5	2,2	2,7
Biomass	32.791	35.296	7,6	5,8	6,0
Industrial Gas	8.451	9.376	10,9	1,5	1,6
Wind	2.705	5.050	86,7	0,5	0,9
Import	35.886	40.254	12,2	6,3	6,8
TOTAL	567.644	592.752	4,4	100	100

Source: MME, 2013

Figure 18.15 Brazilian supply of electrical energy (OIEE) breakdown



Source: MME, 2013

18.16.6 Estimated Project power requirements

Electrical power will be provided by the existing 500 kV line at Colinas, approximately 110 km from the Project site. An alternate shorter line (35 km) was considered (Paráuapebas – Miracema), but the timing for existing Government plans for this particular line is beyond the present project timeline, but should be reviewed again when planned construction occurs.

The substation at Colinas is owned by Eletronorte who maintain the connection to Serra da Mesa, which is part of the North System. Energy supply capacity is considered robust and in the near future, another transmission line from Belo Monte will be established thereby shortening the distance from110 km to 35 km by the year 2018.

Power requirement

The electrical demand for sizing of the substation and the transmission line was established from the estimated electrical load which is summarized in Table 18.7.

Table 18.7 Nominal and peak load demand

Substation	Areas	Nominal MW	Peak MW
300-SE-001	Ore preparation Primary crushing Dust recycling Crushing Homogenizing shed and reclaiming	1.66	1.75
360-SE-001	Ore drying Tertiary screening and crushing Emergency stockpile Coal handling and crushing	1.54	1.62
410-SE-001	Rotary kiln (calcination) Kiln off-gas handling Dust handling Fuel oil	1.64	1.72
440-SE-001	Smelting Refining Metal granulation Product storage, handling and dispatch Technical office Stores Restaurant Refueling station / Wash and lubrication bay Laboratory Dressing room Reception and gate First aid/ Work health/ safety Workshop	8.91	9.36
440-SE-101	Smelting	51.00	51.00
470-SE-101	Refining	1.22	1.22
610-SE-001	Water supply	1.50	1.58
620-SE-001	Cooling water facility	5.00	5.25
TOTAL		72.48	73.49

18.16.7 Connection to the 500 kV power grid

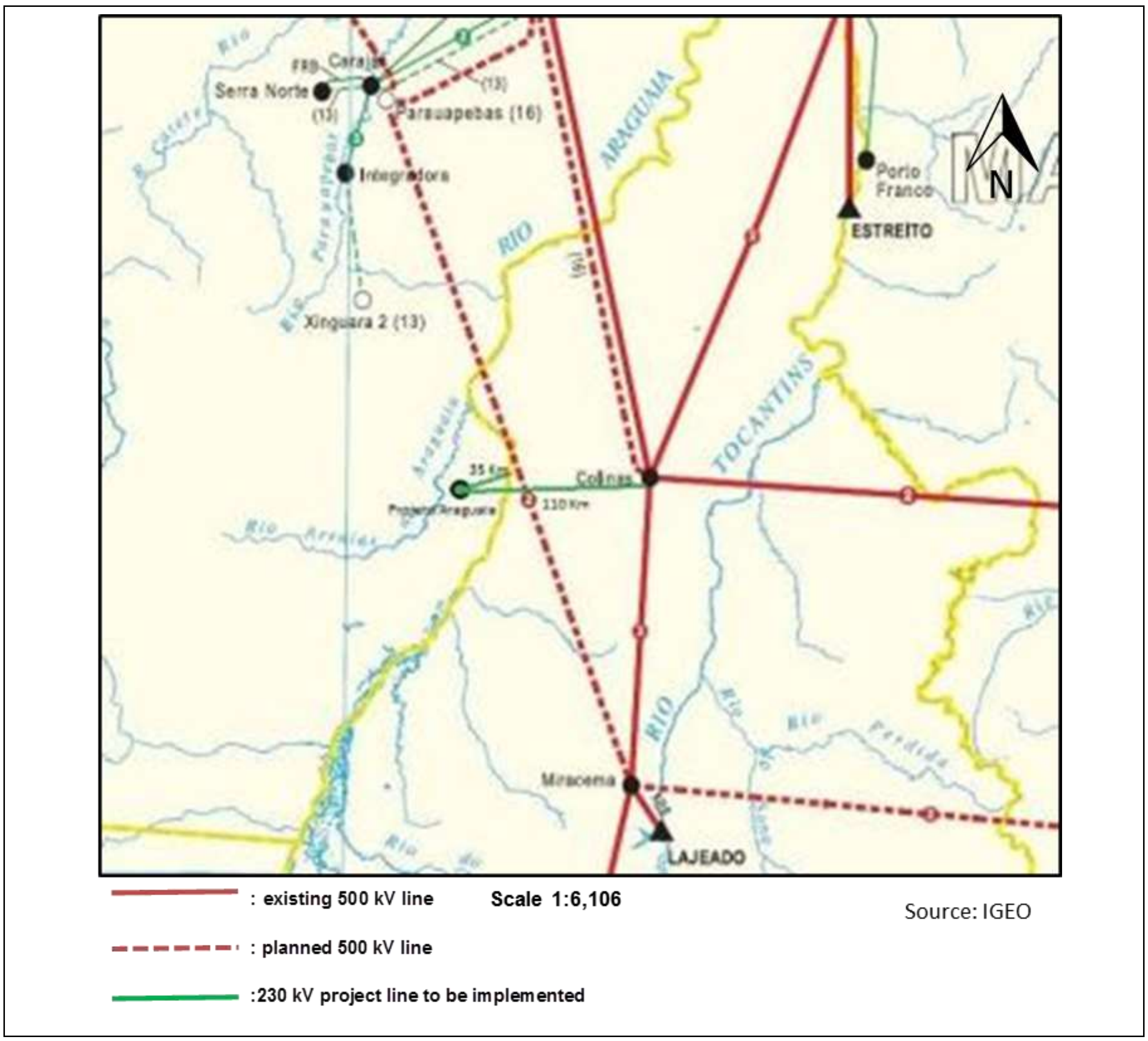
The study has determined that a 500/230 kV substation at Colinas will be required, together with a 110 Km transmission line at 230 kV for the supply of power to the Project site. In addition, a 230/13.8 kV substation will also be required at the Project site.

An alternative solution for of a 500kV transmission line was also considered, but was eliminated on the basis of environmental impact following discussions with ANEEL.

An additional power option included the planned 35km-long Paráuapebas-Miracema 500 kV transmission line, which would be developed close to the Project site. However, the current plans are for implementation is in 2016 and was therefore eliminated as an alternative power solution. However, it is suggested that this is reviewed during the next stage of the Project

Figure 18.16 shows the proposed 110 km transmission line as well as the shorter 500 kV future possibility planned for 2016.

Figure 18.16 Power Transmission to the Project site



18.16.8 Energy supply system configuration

The implementation of the transmission line for the project will involve the following:

- Construction of a 500/230 kV substation having an approximate area of 1 ha, next to the existing Colinas substation. This substation will be similar to the existing Colinas substation, as depicted in Figure 18.17 and Figure 18.18.
- Construction of a 110 km long, 230 kV transmission line to be defined in greater detail during subsequent engineering studies, together with consultation with the power transmission concession holder.
- Construction of a 230/13.8 kV substation at the Project site.

The 500/230 kV substation

The substation configuration features two transformers and manoeuvring assemblies. This arrangement includes a standby transformer to maintain transformer availability in support of the maintenance requirement (typically 45 days) and risk of damage.

Figure 18.17 and Figure 18.18 shows a satellite view of the existing Colinas substation and a ground-level view of the same substation which will be similar to the substation proposed for the Project.

Figure 18.17 Existing Colinas substation – Satellite view

Source: Google Earth

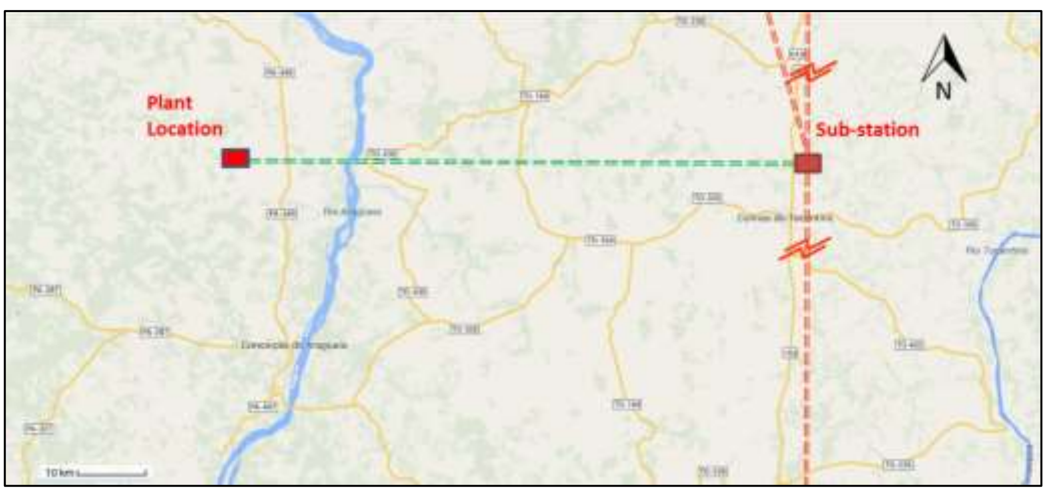
Figure 18.18 Existing Colinas substation

Source: IGEO

230 kV transmission line

The 110 Km transmission line between the Colinas sub-station and the Project is illustrated in Figure 18.19.

Figure 18.19 Colinas substation to Project power transmission line connection



Source: Google Maps

Based on similar projects in the region, an average of two transmission towers per Km was estimated. The final route will depend on subsequent studies to be conducted together with discussions with the concession holder.

230/34.5 – 13.8 kV substation

As noted, the substation for the Base Case would be configured with output voltage of 13.8 kV with two transformers, one as a stand-by.

18.16.9 Conclusion

The proposed 230 kV, 110 Km transmission line from Colinas and the proposed sub-station arrangement are required for the Project. The capital costs are provided in Section 21.0.

Further, it is understood that the proposed 230kV transmission line would be readily approved and regulated by the relevant Government agencies. Figure 18.16 shows the shorter Paráuapebas - Miracema transmission line (35 km) remains a future possibility but not certain for the proposed time frame of the Project.

19 Market studies

19.1 Fe-Ni market study

A study into the ferronickel market was completed for Horizonte by CRU Strategies ('CRU') in the fourth quarter of 2012.

19.2 Sources of nickel supply

According to Sherritt / CRU (January 2014), global mined nickel production in 2013 was 1.9 Mt (of contained nickel), with about 60 % originating from sulphide ores and 40% from laterites. Sulphide ores account for about 30% of global resources and as the larger nickel sulphide deposits are exhausted, the proportion of Fe-Ni as a source of nickel units in the market is forecast to increase, because most new projects coming on stream are predominantly nickel laterite, processed through either hydro-or pyrometallurgical routes, the latter primarily producing Fe-Ni.

According to Sherritt / CRU (January 2014), the fastest growing source of supply in 2013 was nickel pig iron ('NPI'), which accounted for approximately 425,000 t. NPI is a lower grade substitute for finished nickel and almost all production is located in China.

About 70% of Chinese NPI production and 50% of total Chinese nickel production utilised ore from Indonesia in 2013. The raw materials export ban implemented by the Indonesian government in January 2014 has the potential to remove about 300,000 tpa of nickel from the market.

19.3 Outlook for nickel consumption

While there is no substitute for nickel units in stainless steel and other applications, a number of primary nickel unit types are interchangeable e.g. refined nickel for Fe-Ni in stainless steel mills. Global nickel demand is closely linked to trends in industrial production – according to Sherritt / CRU (January 2014), primary nickel demand is expected to increase at an average annual rate of approximately 6% to 2018. Primary demand for non-stainless applications is expected to rise by approximately 4% per annum, driven in particular by demand in non-ferrous alloys and batteries.

19.4 Principal applications of Fe-Ni and market position of Araguaia product

In this study the Project plans to produce a 20% nickel in Fe-Ni product.

While the Fe-Ni market is not subdivided around Ni grade, there is however a distinction between the ferronickel and nickel pig iron (NPI) markets. Ni grade is not the only factor and impurities are a significant component affecting product attractiveness.

With a proposed Fe-Ni of 20 % Ni the Project's product is higher grade than typical NPI material (mainly 8-12%Ni) but at the lower end of Fe-Ni product grades. Stainless steel mills in China will usually input nickel units as NPI and Ni cathode or Fe-Ni and Ni Cathode or a mix of all three. Depending on the exact stainless steel product that the mill is producing, the feed mix will differ. This is also the case outside China, however NPI is only used in Chinese mills.

It is anticipated that the 20% nickel in Fe-Ni product propsoed to be produced by the Project will be more attractive than a 30% Ni product when iron payablility is less than 100% as the volume of non-payable iron received is potentially greater. Furthermore a 20% Fe-Ni avoids the stainless mill having too many iron units and needing to buy extra cathode to balance the feed compared to purchasing a lower grade Fe-Ni.

Any stainless steel mill can in theory change its Ni feed product as long as the appropriate balance in the feed mix is established. European and other non-Chinese mills have not had success in dealing with the processing of NPI impurity levels and as such do not use NPI, which means that those mills are seen as a market for the Project's product.

While the optimal Fe-Ni product for a consumer is totally dependent on their stainless steel mill feed mix and there being no standard, optimal grade, the 20% Fe-Ni product proposed from the Project sits comfortably in the Fe-Ni product range available in the market.

19.5 Nickel price forecasts

The following price forecast metrics were utilised in establishing the Base Case economic evaluation presented in Section 22 of this report.

1. Consensus Economics Inc Ni price forecast December 2013
2. Historic 10 year nickel price – Nickel Institute http://nickelinstitute.org/

Consensus forecast as published by Consensus Economics Inc. in December 2013, and based upon analysts expectations across a range of financial institutions, as set out below in Table 19.1.

Table 19.1 Consensus Economics forecast[11]

Nickel (US$/tonne)	2017	2018	2019-2023 (nominal)	2019-2023 (real)
Consensus (Mean)	19,348	20,061	22,519	19,109
High	24,251	24,568	26,446	22,443
Low	15,135	16,467	18,188	15,432
Standard deviation	2,766	2,754	3,238	2,409
Number of forecasts	16	14	9	11
Liberium Capital				
Citigroup	24,000	24,000	24,000	
Macquarie Bank	22,000	23,999		
Wilson HTM	19,842	21,495	23,756	19,486
Commonwealth Bank	22,908	24,568	26,446	22,046
CIMB Group	18,250	18,700	18,700	16,119
RBC Capital Markets	24,251		23,126	20,944
Investec	18,188	18,188	18,188	16,563
Econ Intelligence Unit	18,521	19,006		
IHS Global Insight	20,465	22,153	25,710	22,443
BNP Paribas				
Barclays Capital				18,200
Societe Generale	17,500	19,000		
Credit Suisse	18,000			20,000
Oxford Economics	16,200	17,398		
UBS	20,173	20,678		18,078
Numis	16,497	17,009	18,356	15,432
Morgan Stanley	17,637	18,188	24,392	20,890
Euromonitor International	15,135	16,467		

Historic Average Ni Prices are were sources from Bloomberg and presented in Table 19.2 below. The Ni price, over the past 10 years, has averaged US$ 20,283 /t.

[11] *Survey Date, December 16, 2013 © Copyright Consensus Economics Inc.*

Table 19.2 Historical annual Ni metal price

Year	Mean: 10 years Ni Price: actual US$/t
2004	13,854
2005	14,766
2006	24,196
2007	37,118
2008	21,004
2009	14,711
2010	21,811
2011	22,843
2012	17,526
2013	14,992
Average	**20,283**

19.6 Iron credit forecast

The proportion of the iron content that stainless mills typically pay for varies considerably according to market conditions, fluctuating between 0 and 100% in recent years, with mills unwilling to pay for some or any of content in a surplus market, with the opposite being true in an under supplied market.

Table 19.3 illustrates typical Fe credits for a 17% Ni in Fe-Ni product in recent years:

Table 19.3 Typical Fe credits for a 17% Ni, Fe-Ni product.

	LME 3 month price	FeNi Gross Weight per tonne	Ni in Fe Ni %	Ni component	Fe element remaining	Fe credit per tonne
	$/lb	$ / tonne Fe Ni		$ / tonne Fe Ni	$ / tonne Fe Ni	$ / tonne Fe in FeNi
2008	9.66	3,964	17%	3,619	344.5912	415
2009	6.68	2,648	17%	2,503	145.1376	175
2010	9.92	3,792	17%	3,717	75.1744	91
2011	10.4	3,896	17%	3,897	-0.672	- 1
2012	8.03	3,009	17%	3,009	0.3196	0
Source: CRU Strategies						

As can be seen from Table 19.3, 2008 and 2009 and 2010 can be characterised as under supplied, and 2011 and 2012 over supplied.

The iron credit applied in the financial evaluation of $ 150 /t iron in Fe-Ni is close to the average for the past 5 years of $ 136 /t Fe in Fe-Ni and assumes a tightening in supply, consistent with the anticipated improvement in the Ni price from recent and current levels.

The QP has reviewed the above and that the results support the assumptions in the technical report.

19.7 Economic evaluation price

As can be seen from the two Ni price metrics set out above, the consensus forecast Ni price of US$19,348/t is broadly comparable to the 10-year historic average of US$20,283/t, the average across the two sets of figures of US$19,816/t.

For the economic evaluation in this study, the Base Case uses a lower figure of US$19,000/t ($8.62/lb Ni) and an iron credit of US$150/t ($0.06/lb Fe). A fixed price for both Ni and Fe has been used through the life of mine.

20 Environmental studies, permitting, and social or community impact

20.1 Introduction

This chapter considers the environmental and social aspects of the project, based upon work completed to date, with a view to compliance, from an international best practice and Brazilian perspective, for fulfilment of regulatory requirements for permitting and licencing.

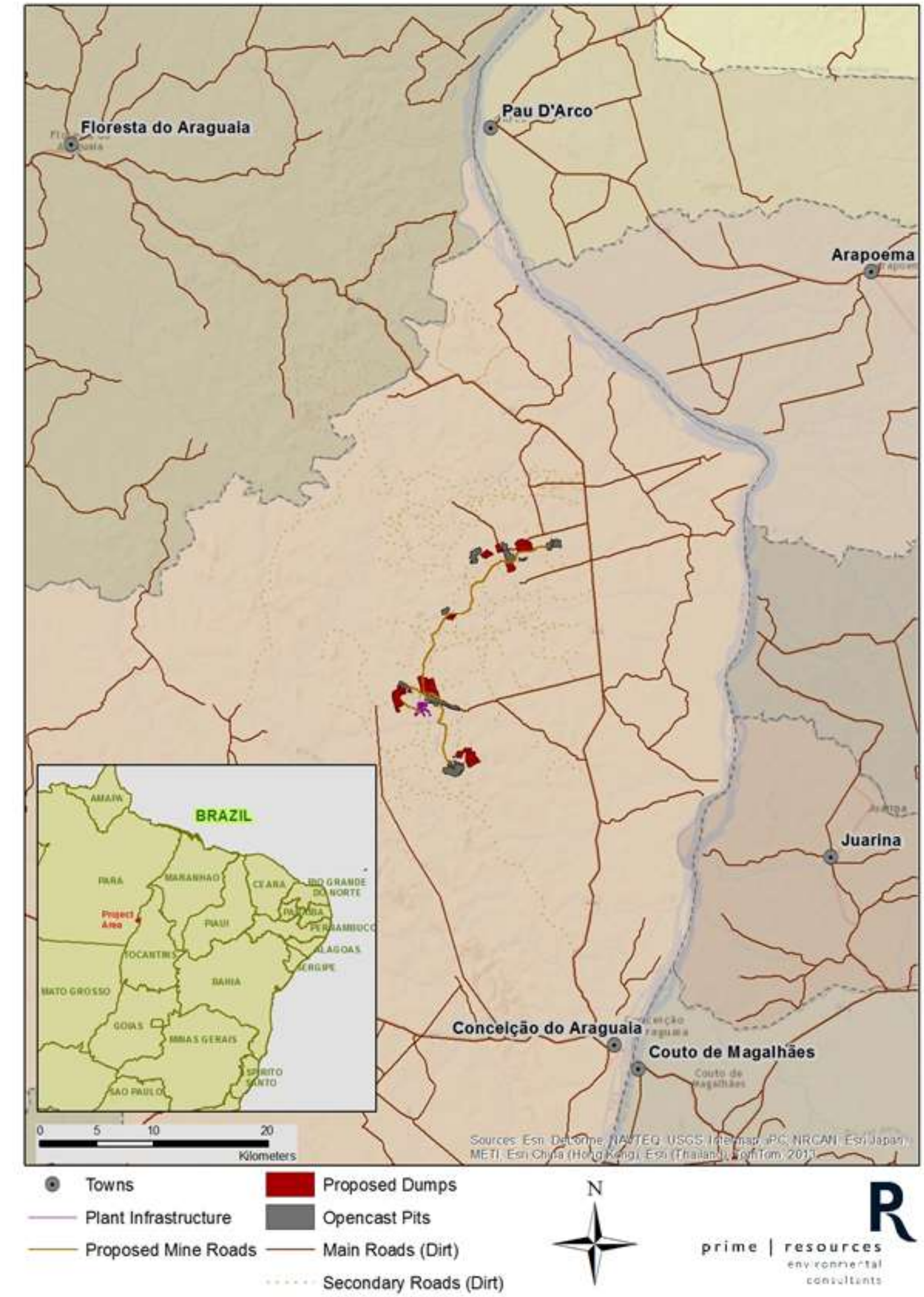
20.2 Project background

Pará state had a GDP of R$ 77.8 billion in 2010, representing 2.1 % of Brazil's total GDP. The major industrial activities to the economy of Pará are mining (iron, bauxite, manganese, limestone, gold and tin), timber, agriculture, industry and tourism.

Project area of influence has been defined in terms of national legislation. The area of influence covers the intersection between the project and the terrain (direct impact resulting from footprint), the local watercourses directly affected by the project, and the catchment / basin affected indirectly by the project. The full extent and therefore impact of the project has not yet been assessed because of the early project phase. The full area of influence, as defined by Performance Standard 1, will be considered during the Environmental Impact Assessment (EIA-RIMA).

The project area is characterised by flat-topped highlands with a downward slope across the project area from west to east towards the Araguaia River (refer to Figure 20.1). The region's climate is categorized as being equatorial super-humid, with wet summers and dry winters with periodic droughts. Historically, the area in consideration has been cleared to support the agricultural industry. The vegetation profile in the proposed mining area is comprised of savannah, semi-deciduous forest, riparian forest and ombrophillous forest. Local farming, which operates at a commercial level, consists of cash crops including rice, maize / corn and pineapples.

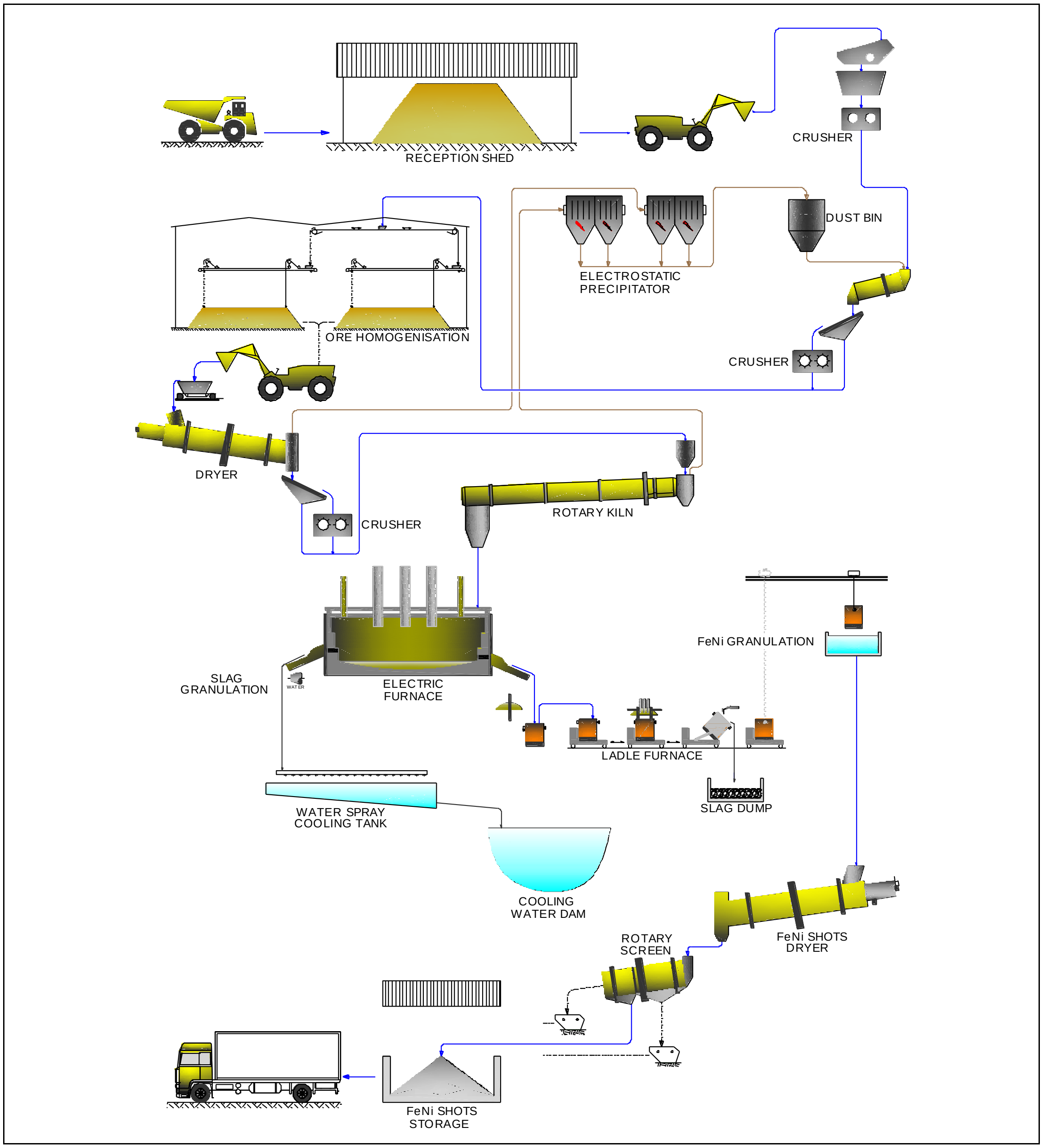
Figure 20.1 Project location



20.3 Project description

ROM ore will be mined by means of mechanical excavation and hauled by truck to intermediate and pre-blending stockpiles. Blended ore will be reclaimed from these stockpiles by means of a backhoe or front-end loader and hauled by 20 tonne trucks to the processing plant. Fe-Ni production will take place using the RKEF process which involves the preparation, drying, calcining, smelting of the ore and refining of the produced Fe-Ni and ultimately the granulation of the smelting slag for disposal (see Figure 20.2). The granulated slag is then transported by truck to the proposed slag disposal facility.

Figure 20.2 Process flow diagram for the Project



From initial geochemical testing by SGS, it has been determined that slag produced by the smelting and granulation process will be comprised primarily of SiO_2 (50 %), MgO (24 %), Fe_2O_3 (14.3 %) and Al_2O_3 (10.6 %) with trace elements of Co (60 ppm), Cr (3 120 ppm), Cu (62.6 ppm), Li (68 ppm), Mo (8 ppm), Ni (948 ppm), V (109 ppm), Y (409 ppm) and Zn (119 ppm).

The Base Case is for a 25 year Life of Mine (LoM) with ore processing capacity of 900 ktpa and 674 ktpa slag production respectively.

The total volume of slag produced over the 25 years is estimated at 9.93 Mm^3. This will be accommodated in a waste dump as part of a waste management programme. The slag dump will be lined with a clay liner which preliminary geotechnical testing indicates as suitable after engineering and compaction. High clay content material is readily available on site.

20.4 Legal framework

The investigations undertaken have described the regulatory framework for Social and Environmental Impact Assessment (SEIA) within Brazil (locally known as the *Estudo de Impacto Ambiental* - EIA) together with international obligations for the Project. These include the International Finance Corporation (IFC) Performance Standards (PS) and World Bank Group Environmental, Health and Safety (EHS) Guidelines.

20.5 Permitting

20.5.1 Permitting requirements

According to Resolution 237/97, issued by the National Environmental Council, three consecutive environmental permits must be requested: Preliminary Licence, Installation Licence, and Operating Licence. Refer to Table 20.1for further details.

Table 20.1 Approvals and permits required

Permit	Responsible authority/Legal basis	Objectives
Exploration Licence (*Alavara de Pesquisa)*	National Department of Mineral Production (DNPM)	Allows investigation into the mineral potential of the licence area.
Preliminary Licence (*Licença Prévia*)	Special Secretariat for the Environment (SEMA)	Granted at the conceptual stage. Approves the location, design, and environmental sustainability of the project. Establishes the basic requirements and conditions to be met in the next stage of implementation.
Installation Licence (*Licença de Instalação*) (LI)	SEMA	Authorizes the construction of the project in accordance with the approved EIA-RIMA and supporting documentation. The proponent must implement the mitigation measures relevant to construction and must ensure that monitoring occurs.
Operating Licence (*Licença de Operação*) (LO)	SEMA	Authorizes project operation after effective compliance with the other licenses and mitigation measures has been verified.
Land Use Authorisation	Conceição do Araguaia Municipality	Amends land use authorisation to industrial / mining.
Vegetation Removal Authorisation (*ASV*)	SEMA	For areas that will be cleared of vegetation. Valid for 2 years.
Transport, use and storage of chemicals	National Traffic Council (CONTRAN) and Ministry of Labour	Addresses environmental and health risk.
Water Licence (*Outorga*)	SEMA	Addresses impacts on water resources.
Mining Licence	DNPM	Obtained on approval of the Plan for Economic Development (PAE) and LO authorization.

20.5.2 Permitting status

Horizonte Minerals has in place

1) An Exploration Licence for the property which is valid until 26 February 2016;

2) An Operating Licence (for exploration), which is valid until 21 May 2014. A request for renewal or extension was submitted to SEMA on 17 January 2014; and

3) A Water Licence. A request for renewal was submitted to SEMA on 17 January 2014.

20.5.3 Permitting schedule

Refer to Table 20.2.

Table 20.2 Permitting schedule

Permit	Responsible authority	Schedule / Timing
Exploration Licence (*Alavara de Pesquisa)*	DNPM	Apply for prior to exploration. Valid for 3 years (nickel).
Preliminary Licence (*Licença Prévia*)	SEMA	Must have in place an Exploration Licence to apply for LP. An EIA-RIMA must be prepared and registered with SEMA. EIA-RIMA is presented to communities and authorities at a public meeting. Authorities approve or reject LP based on EIA-RIMA. Valid for 3 years.
Installation Licence (*Licença de Instalação*) (LI)	SEMA	LP documentation must be presented. Environmental Control Plan (*PCA*) must be prepared. If PCA is approved, then LI is granted. Construction may begin, with certain conditions. Valid for 3 years.
Operating Licence (*Licença de Operação*) (LO) for Mining	SEMA	LI documentation must be presented. Authorities must be satisfied that construction occurred as per LI and any conditions. Allows mining, processing, and sale of minerals. Valid for 4 years; may be renewed.
Vegetation Removal Authorisation (*ASV*)	SEMA	Applied for as and when required. Valid for 2 years.
Transport, use and storage of chemicals	O Conselho Nacional de Trânsito (CONTRAN) and Ministry of Labour	Applied for as and when required.
Water Licence (*Outorga*)	SEMA	Applied for as and when required. Valid for 2 years; renewable.
Final Exploration Report (*RFP*)	DNPM	Report on technical and financial feasibility of project, based on exploration results. If approved, proponent has one year to apply for Mining Licence.
Plan for Economic Development (PAE)	DNPM	Requires an approved Final Exploration Report (RFP) stating technical and financial feasibility of the project. Valid for 1 year. Renewable for 1 year.
Mining Licence	DNPM	Obtained on approval of the Plan for Economic Development (PAE) and LO authorization.

20.6 Environmental and social investigations

Horizonte Minerals have to date undertaken investigations into climate, air quality, noise, soils, flora, hydrology, geohydrology, protected areas, traffic and socio-economics, The baseline studies have not all been completed and still require additional investigation. Additional baseline studies also need to be undertaken – these include cultural heritage, resettlement and fauna. No information on these aspects was available at the time of compiling this report.

Upon completion, the baseline studies will meet the requirements of international best practice.

20.7 Summary of environmental baseline conditions

Baseline environmental conditions are summarised:

Climate

The region's climate is categorized as being "equatorial super-humid", with an average annual temperature of 26.3° C (DBO Engineering, 2013). Relative humidity in Conceição do Araguaia is considered elevated with a defined rainy season between November and May, and a dry season from June through October. There is almost no rainfall during the dry season months. The annual rainfall for the region is approximately 2 000 mm (DBO Engineering, 2013). The rate of evaporation and transpiration in the project area is considered to be lower than, or at most equal to, precipitation (WALM, 2013).

Air quality

Dust monitoring has been undertaken at Oito, Gorgulho and Santa Maria from July 2012 to June 2013. Results indicate very high dust emissions at Gorgulho in August 2012 and May 2013, and at Oito in October 2012, which may be partially due to clearing of land by burning during the dry season (March to October). These values do not exceed the CONAMA standards of 2 500 $\mu g/m^3$ for ambient dust, but do exceed the IFC target values of 50 to 150 $\mu g/m^3$ for prevention of air quality deterioration. The monitored dust values are higher than the ambient limits provided by Brazil.

Noise

Noise monitoring was conducted at 10 sites within and around the project area. Based on these results, the more rural sampling points have average daytime noise levels between 41.3 and 47.3 dB. Sample points along a major road have average values of 50.8 to 56.3 dB, likely elevated as a result of traffic noise. The monitored noise values are higher than the ambient limits provided by Brazil.

Topography

The project area is characterised by flat-topped highlands and sloping surfaces towards the major river systems (KH Morgan and Associates, 2013). There is a downward slope across the project area from west to east towards the Araguaia River. The highest elevation of the project area is 360 mamsl and the lowest elevation is 217 mamsl.

Soils

There are four major soil types in the whole project area:

Alfissols are mineral soils, with non-hydromorphic soil horizons (light colour, characteristic of the loss of clay, iron or organic matter), underlain by horizons which have a reddish to yellowish colour due to an iron oxide content below 15 %. The clay content of the soil increases with depth. These soils contain low values of Ca, Mg and K, with a base saturation of below 50 %, and are acidic soils.

Gleysols are formed under conditions of prolonged soaking (high water table) in the floodplains near the Araguaia River. These soils are poorly drained, strongly acidic, and shallow. The hydromorphic conditions cause a mottled appearance of reddish-yellow or red within the soil profile. These soils normally have less than 2.5 % of organic matter content in the surface or topsoil horizon. The topsoil horizon has a thickness of approximately 46 cm. These soils are susceptible to erosion.

Entisols have their soil horizon directly on rock or rock fragments with a diameter greater than 2 mm (gravel, cobbles and boulders). Entisols are not hydromorphic soils and they have little resistance to weathering. These soils are heterogeneous in terms of their chemical, physical and mineralogical characteristics. The texture and colour is closely related to the source material.

Dystrophic soils consist of mineral material (oxisols) with a horizon of >50 cm. The soil horizon is situated 200 to 300 cm, sometimes 2m, below the surface. In general, oxisols are non-hydromorphic and porous. The red-yellow colour of the soil is related to the removal of silica and the exchangeable cations (Ca^{2+}, Mg^{2+}, K^{+}), leading to enrichment with iron oxides and aluminium (aggregating agents). The oxisols have a low erosion potential due to their depth and their uniform texture.

The agricultural potential of the soils within the project area is considered moderate (WALM, 2013).

Land use

Much of the project area has been historically cleared for pastures and agriculture. The extent of original forest remaining within the project area will be determined during ongoing investigations.

Flora

The area of influence can be classified as transitional forest, with vegetation characteristic of both the Dense Forest Biome and the Amazon Biome. Plant species identified included those which could adapt to both these biomes, as well as species exclusive to either of the biomes.

The vegetation profile in the area of influence is comprised of the Savannah (characterised by xerophyte vegetation), Semi-Deciduous Forest (occurring in water-scarce areas), Riparian Forest (along the Pau d’Arco and Arraias do Araguaia rivers) and Ombrophillous Forest (transitional forest).

In total, 248 species were identified within the project area. Of the identified species, ten are considered to be vulnerable according to SEMA. Two of these (*Cedrela fissilis* (cedar) (endangered) and *Cedrela odorata* (white cedar) (vulnerable), are protected in terms of the IUCN Red List of Threatened Species. It must still be determined (during ongoing investigations) whether there are protected species in the areas that are proposed to be cleared of vegetation.

Hydrology

The project area falls within the Araguaia-Tocantins Hydrological Basin. The major surface water resources within the project area include the Araguaia River and tributaries thereof including the Salobro River and the Arraias do Araguaia River (DBO Engineering, 2013).

Regional drainage is west-to-east towards the Araguaia River. The entire region is drained by the Araguaia River basin, with the sub-basins of the Pau D'Arco and Arraias do Araguaia Rivers being especially relevant since the latter is a direct tributary of the Araguaia River. Several other watercourses, including the streams of Grotão São Domingos, Grotão do Ferro, Grotão do Ouro, and Grotão do Régis, also flow into the Araguaia River via the left bank (DBO Engineering, 2013).

The major rivers have extensive deposition of sandy alluvium within wide sheets of shallow water, retained by low sandbars. Part of the headwaters of tributaries of these two systems drain from the nickel mineralised project areas to Rio Arraias.

Surface water sampling has been undertaken at 29 locations within the project area between November 2011 and October 2012. Cyanide, chromium (total), phosphate, mercury, selenium and vanadium all exceed the Class I CONAMA limits (drinking water, aquatic health, crop cultivation) at almost all the sampling points. It is likely that this is related to the geology of the area, but may impact on water management during project operation, to ensure that these values are not increased or concentrated further.

Geohydrology

Groundwater occurs at a shallow depth of a few metres throughout the region. The saturated oxidised rocks are clayey, have high water storage, and generally low to very low water transmissivity. Permanent to semi-permanent springs occur at the headwaters of most small creeks and maintain permanent to semi-permanent stream flow and/or saturated stream beds.

This piezometric level relationship indicates that the lower groundwater system under Pequizeiro is downward draining to the regional groundwater system and is an influent drainage (fed by precipitation) as compared to effluent flow by the phreatic upper aquifer (recharging surface water). It is possible that these drainage systems may alternate between effluent and influent flow between rainfall recharge periods.

Groundwater samples were collected monthly from nine sampling points (wells / boreholes) between April 2012 and August 2013. Results indicate that groundwater is not suitable for human consumption, because of the high values of total dissolved solids (TDS), lead, cobalt, iron, nickel, and nitrogen exceeding the CONAMA limits for drinking water. pH is also acidic in some areas, possibly those sample points near surface springs (due to interaction of water with organic matter).

Protected areas

Pará has 83 protected areas, under the responsibility of SEMA. No protected areas were identified within the project's area of influence. There are however several local areas considered important and protected in terms of the Protected Areas National System (Law 9985/2000). Areas relevant to the general project area include riparian zones of perennial watercourses and springs (unless they are intermittent).

Traffic

A traffic survey was conducted on the PA449 road, which is the main road connecting Conceição do Araguaia and Floresta do Araguaia (Horizonte Minerals, 2013). It is an important road for the transportation of local produce. The road is approximately 110 km long and classified as a class II vicinal (local) road, and passes 3 km to the east of the project area.

A 24-hour traffic count was conducted over three days, in September 2013. The morning and afternoon peak hours were found to be between 07h00 to 08h00 and 17h00 to 18h00, respectively. The maximum peak hour traffic recorded was 61 vehicles, 40 of which were motorcycles.

20.8 Summary of social baseline conditions

Baseline social conditions are summarised:

According to the 2010 Census, the total population in Conceição do Araguaia is 46,206, with a population density of 7.81 per km^2. The population has grown by 5 % since 2000. Approximately 71 % (32 464) live in urban areas and 29 % (13 093) live in rural areas.

There are slightly more men (51 %) than women (49 %). The majority of the population consists of residents aged 0 to 19 years (approximately 48 %), with only approximately 6 % of the population being over 60 years of age. The predominant ethnic group is the brown / mulatto ethnic group (61 %) followed by white (29 %), black (9 %) and indigenous (0.3 %).

The overall literacy rate in 2011 was 87.4 %, with women being slightly more literate (88.5 %) than men (87.1 %). More students attend primary education than secondary education.

The majority of employment is within the commercial sector (48 %) followed by agriculture / farming (24 %). Crops planted include pineapple, rice, beans, manioc (cassava), watermelon, corn and soy beans; of which pineapple, corn and soy bean are the greatest value crops. In terms of livestock husbandry, 69 % is cattle and 24 % is poultry; the rest is made up by horses, buffalo, donkeys, mules, pigs, goats and sheep.

At the end of 2011, there were 2 816 formal jobs, most of them within the commercial sector, with a greater number of men employed in this sector than women.

The majority of the families (71 %) living in Conceição do Araguaia live above the poverty line, having a per capita monthly income of more than BRL 140, and 86.7 % of families are supported by a single breadwinner. In the urban areas the average monthly income was R$ 529.27, compared to R$ 261.34 in the rural areas.

The municipal road network has 52 routes and most of them are impassable due to poor condition and disrepair. The morning and afternoon peak hours on the PA449 road, connecting Conceição do Araguaia and Floresta do Araguaia, were found to be between 07h00 to 08h00 and 17h00 to 18h00, respectively. The maximum peak hour traffic recorded was 61 vehicles, 40 of which were motorcycles. The dominant mode of transport is by motorcycle (including scooters), making up 70 % of the vehicles in the municipality.

The major land use in the municipality is agriculture. Most of the agricultural establishments are 20 to 50 hectares in size. The status of land tenure and ownership is unknown although it is anticipated that much of this land is owned by the current farmers and/or residents.

Less than half of the dwellings in Conceição do Araguaia have access to running water. The main water supply sources are wells or springs located on the properties. Other sources include rain water stored in cisterns, water trucks, dams or streams. Of the 12 853 dwellings, 1.6 % have basic adequate basic sanitation facilities, 71.4 % have semi-adequate basic sanitation facilities, and the remaining dwellings have inadequate sanitation facilities (latrines or disposal into watercourses).

In Conceição do Araguaia, waste is collected from 67 % of the dwellings. Where waste is not collected, it is burned or buried on the property, or disposed of into vacant lots or rivers. The municipality does not have a controlled landfill site or an adequate location for the disposal of waste, so it is disposed of in a general dump.

20.9 Stakeholder engagement

The process of stakeholder engagement has to date only considered the identification of stakeholders and the compilation of a Stakeholder Engagement Plan by Integratio. Stakeholders include government departments (agriculture, land reform, nature conservation, heritage, and mining), NGOs, community leaders and residents of both municipalities. Villages within the municipalities include Vila Seringa, Vila Jonco / Lote Oito, Vila Chapéu de Palha, Vila Canarãma, and the Fazenda Jacutinga region within Conceição do Araguaia; and Vila Mendonça, Vila Tabuleiro and Vila Bom Jesus within Floresta do Araguaia.

20.10 Resettlement planning

It is likely that the project will require the purchase of land and relocation of certain households. The extent of resettlement is not yet fully defined but it is currently being investigated. Resettlement is expected to be voluntary, with land being purchased from relevant landowners, and impacts on livelihoods being compensated. No forced relocation will occur as a result of this project. Integratio has been appointed to undertake this investigation.

There are no regulated compensation rates in Brazil, and actual values will be determined upon negotiation with the various parties. A calculation has however been undertaken based on Integratio's experience in similar projects. This calculation considers only the land surface that will be impacted on (within direct footprint and buffer area around the mine), and a standard rate applied to land value. A total of 6 332 ha of land has been highlighted as the area of influence, and an estimated value of US$ 2 166 per ha has been provided by Integratio. A basic cost of land purchase has been calculated at US$ 13.7 M. This costing is oversimplified and does not consider the cost of lost dwellings or livelihoods (where residents may not be landowners), actual valuation of property, number of households or individuals affected, and defined land required for purchase. It also does not consider the option of leasing land in some cases.

20.11 Rehabilitation and closure

The Constitution of Brazil requires that areas disturbed by mining activities must be restored by the mining concession holders. WALM is compiling the Environmental Impact Assessment (EIA-RIMA) as required by Brazilian authorities and as part of that, likely within the Environmental Control Plan, they will address rehabilitation and closure objectives in terms of national legislation. Financial provision for ongoing environmental management, as part of the Environmental Control Plan, will be addressed in the EIA-RIMA.

20.11.1 Mine closure

The following sections provide a philosophy of approach for closure activities for the mine pit, processing and smelter facilities.

Mine pit closure

The mining system will be designed for closure such that mined-out areas will be reclaimed as a continual process as an integral requirement of mine planning and production. However, it is not proposed to re-handle all the waste material back into the pits at the end of the mine life.

It is anticipated that all topsoil will be stockpiled for future replacement to allow for re-vegetation subsequent to mining activities.

With mine closure, and the cessation of all mining activities, the infrastructure within the mine production area will be removed. A sequencing of closure events will be necessary for an effective closure plan.

Processing and smelter facility closure

It is anticipated that decommissioning and demolition of all industrial structures will be undertaken.

Closure costs

An estimate of closure-related costs has been calculated. The closure cost has been based on international best practice parametric estimation rates.

The financial provision for closure is US$ 47 M.

20.12 Planning for feasibility

The feasibility study will be undertaken in terms of international best practice, i.e. Equator Principles, International Finance Corporation (IFC) Performance Standards, and World Bank Environmental, Health and Safety (EHS) Guidelines. In addition, the project will also comply with the required Brazilian national legislation. WALM has been appointed to complete the Environmental Impact Assessment (EIA-RIMA), and to address any Brazilian licensing requirements.

Completion of baseline specialist studies

The following issues are currently being investigated but were not completed in time for inclusion into the PFS report:

- Resettlement
- Fauna
- Cultural heritage.

These baseline studies are due to be completed before the end of 2014 and will be incorporated into the feasibility study.

Some of the baseline studies need to be further expanded. These include:

- Vegetation – to determine species composition of areas that will be cleared of vegetation, and to determine whether any conservation important species will need to be relocated.
- Geohydrology – the specialist study must be expanded to address the other areas of the mine. The current focus has been on the central Pequizeiro area. In addition, springs have not been sampled for hydrochemistry, and this will be undertaken during the next sampling season.

Social and environmental impact assessment

In order to comply with international best practice, the following aspects must still be addressed within the EIA-RIMA:

- Identification and assessment of impacts - All the specialist studies undertaken to date, including those that are still due to be completed, must identify and assess the potential environmental and social impacts and risks of the project, including any cumulative and third party impacts. Greenhouse gas emissions, climate change and transboundary effects must be addressed. The impact identification process will also take account of the outcome of the engagement process with affected communities.
- Management and mitigation of impacts - Management programmes will include mitigation and performance improvement measures to address the identified environmental and social risks and impacts of the project. Mitigation must ensure that the project operates in compliance with applicable laws and regulations, and meets the requirements of International Finance Corporation (IFC) Performance Standards 1 to 8.
- Stakeholder engagement - Stakeholder engagement is an ongoing process that should occur early in project planning phase, and continue throughout the Life of Mine. Stakeholders have already been identified, and a Stakeholder Engagement Plan compiled. Integratio must ensure that this plan fully meets the requirements of international best practice.

21 Capital and operating costs

21.1 Cost estimation overview

The Base Case for this study assumes an ore processing rate of 0.9 Mtpa after an initial ramp up period. The ore processing methodology is the pyro-metallurgical conversion of a nickel bearing laterite ore into a Fe-Ni product using the RKEF process that will be sold at the mine gate.

The pre-production capital costs in the cashflow model have been allocated 30% in year 1 and 70% in year 2 of the 2 years of construction. Deposits for equipment will be required on the high value long lead items in year 1 and the balance of the purchase price will be required in year 2.

The capital and operating cost estimates were prepared or advised by the following groups:

- Mining Snowden
- Process plant IGEO (Reviewed by Snowden)
- Onsite infrastructure IGEO (Reviewed by Snowden)
- Offsite infrastructure IGEO (Reviewed by Snowden)
- Environmental Prime Resources
- Social Prime Resources
- Corporate G & A Horizonte (Reviewed by Snowden)
- Royalties Earnest & Young (E&Y)
- Taxation E & Y
- Closure & Salvage Horizonte (Reviewed by Snowden)

The total estimated pre-production capital cost to engineer, procure, construct and manage all of the works required to bring the Base Case into initial production is $582 million.

The objective of the Base Case study is to provide:

- the basis of a detailed economic evaluation of the project
- the basis of further studies to optimise the return on the investment
- a framework to increase confidence in the investment
- the basis for future studies.

Several option studies were undertaken that included an option for a higher plant throughput of 2.7 Mtpa (Option 1) of ore in addition to a mining fleet owner operator (Case 2) at both feed rates

21.2 Exclusions

The following items are not included in the capital and operating estimations:

- Cost of studies and work prior to commencing equipment purchase and construction. However an estimated cost has been included to be written off in the taxation section
- Import duties, withholding and purchase taxes based on advice from E & Y
- Foreign currency exchange rate fluctuations
- Risks due to potential government policy changes, labour disputes or permitting delays.

21.3 Capital costs

The capital costs have been estimated following a series of indicative prices received from major equipment manufacturers and suppliers. The capital costs have been provided in detail and for this report are summarised in Table 21.1 and Table 21.2. All of the capital costs have been reviewed by Snowden.

Import duties on capital equipment not sourced or manufactured in Brazil are high and as such work has been done to minimise or where possible eliminate importing items not supplied originally or manufactured in Brazil. On advice from E & Y, no import duties have been applied on capital equipment.

Table 21.1 Base Case pre-production capital costs

Item	$ Million
PLANT DIRECT	376.088
Site preparation	4.988
Plant ancillaries	10.950
Plant utilities	21.425
Ore preparation(included drying)	65.663
Coal preparation	26.631
Calcining	55.883
Smelting	147.187
Refining	43.360
PLANT INDIRECT	38.206
Temporary facilities and services/communication	2.119
Freight	6.528
Vendor representatives	0.448
Spare parts	4.093
EPCM services	24.000
Third party engineering	1.019
OWNER COSTS	18.313
Pre-production employment and training	2.793
Project and construction management	3.000
Construction power and catering	4.200
Insurance	4.820
Corporate travel & services	2.000
Security	1.500
INFRASTRUCTURE	56.034
Water supply	3.510
Road to site	21.739
Substation 500KV - 230KV	9.265
Power line 230 KV	12.826
Substation 230KV - 34.5KV	8.694
Slag Storage Facility	5.242
Site clearance	0.939
Earthworks & excavations	2.506
Drainage	0.612
Concrete structures	0.169
Miscellaneous	0.035
Fixed charge items	0.554
Remote location factor	0.427
Social	6.000
Mining	5.000
First fills and spares	1.200
Contingency at 15%	76.092
Total pre-production capital costs	582.176

The social cost has been adjusted to reflect that the full project area cost will not be required in the Base Case.

Table 21.2 Base Case production capital costs

Item	$ Million
Mining & plant sustaining	43.313
Annual allocation	1.750
Total LOM sustaining	43.313
Closure (2 Years)	20.000
Total production capital cost	63.313
Salvage	1.400

The anticipated life of the major capital equipment is longer than the ore processing years for the Base Case, the allocated sustaining capital to maintain the plant designed throughput has been adjusted to reflect this and is considered adequate.

21.1 Operating costs

The operating costs have been estimated following a series of indicative prices received from suppliers and also the extensive databases held by the major contributors. All of the operating costs have been reviewed by Snowden and are presented in summary in Table 21.3.

Import duties on supplies not sourced or manufactured in Brazil are high and as such work has been done to minimise or where possible eliminate importing items not supplied originally or manufactured in Brazil. On advice from E & Y, no import duties have been applied on operating supplies.

IGEO appointed an independent consultant to provide advice on power costs. It was determined that the power cost base for the PFS was 8.5 cents per kWhr and the rate applied to the analysis was exclusive of taxes.

Table 21.3 Base Case operating costs

Item	$ Million	$/Tonne - Ore[12]
Mining (Contractor)	552.998	26.08
Mine & haul ore	137.228	6.47
Mine & haul waste	289.407	4.82
Fixed costs	126.363	5.96
Mining cost / tonne ore mined		26.08
Mining cost / tonne material moved		6.81[13]
Processing	2,641.667	124.57
Ore preparation - variable	6.591	0.31
Pre furnace – fixed	545.339	25.72
Pre furnace – variable	1,060.125	49.99
Post kiln – fixed	168.014	7.92
Post kiln – variable	861.597	40.63
Off-site overheads	99.000	4.67
Total Operating Costs	**3,293.665**	**155.32**

The ore feed to the process plant comes from a number of pits, some of which are remote to the plant. The cost of hauling the ore from these pits and the pre blending has been included in the processing costs pre-furnace.

21.2 Royalties (CFEM)

The Compensation for Exploitation of Mineral Resources (CFEM) is a royalty payment akin to a tax, created by the Federal Constitution as a compensation to the States and Municipalities for the economic use of mineral resources in their territory.

CFEM is payable by legal entities in the mining industry that exploit or extract mineral resources and are payable upon sale of the mining product. The CFEM rate applied to the entity's net revenues varies in accordance with the mining product, but cannot exceed 3% of gross revenues, after deducting insurance, tax, and transportation costs. For nickel, the applicable CFEM rate is 2%.

The CFEM revenues are divided as follows:

- 10% for the Federal Government (DNPM and IBAMA);
- 2% for the FNDCT (National Fund for Scientific and Technological Development);
- 23% for the state (or Federal District) from which the mineral is extracted;
- 65% for the municipality from which the mineral is extracted. In the case of where two or more municipalities are involved, the CFEM must be paid to each municipality pro rata to the volume of extraction carried out in each of them.

[12] Except where noted
[13] $/t of total material (ore and waste)

The CFEM in Brazil is a tax on costs to a point in the process where the ore has not undergone a physical change. The calculation of the CFEM is carried out using 2% of these accumulated costs of production.

In the case of a pyro metallurgical project such as the Project the calculation of the CFEM includes the following costs: mining, stockpiling, crushing, coal preparation, administration, maintenance and some environmental costs up to and including calcining. The addition of all these costs gives a value that will be multiplied by 2%. Based on this the life of mine royalties used in the analysis are presented in Table 21.4, below.

Table 21.4 Royalties (CFEM)

Item	$ million
CFEM	43.301

21.3 Taxation

The tax treatment adopted in the financial evaluation was based upon advice taken from E & Y in Brazil. The Brazilian government is making efforts to improve the climate for foreign investment as it seeks to develop a more market-oriented economy. Incentives are therefore available for exporters and for which it is believed the Project would be eligible.

Most incentives apply to new investments and are offered by Federal, State or Municipal governments. These generally include substantial reductions in taxes (mainly State VAT, ICMS, as well as corporation tax), utility charges and other expenses. Federal incentives (generally income tax reductions) are available for investments in less developed areas such as where the Project is located.

Depreciation is calculated on a straight line method over 10 years.

The Project has been assessed as eligible for the various existing fiscal incentives available and these have been assumed where applicable in quantifying the tax burden on the project

Summarised below are the principal taxes relevant to the project:

(i) **Federal Corporate Income Taxes:** There are 2 income taxes - the corporate income tax and the social contribution tax on profits. They are imposed on similar taxable bases.

Brazilian corporate income tax (‘IPRJ’) is charged at a 15.25% rate, with a surtax of 10% applicable to profits exceeding R$240,000 a year.

In addition, Brazil imposes a social contribution tax (‘CSLL’) on corporate profits. The social contribution tax works similarly to income tax and it is charged at a 9% rate. Ordinary tax losses may be carried forward with no time limit for offset against future operating profits.

Application of SUDAM incentive to IPRJ (25% Corporate Income Tax). A 75% reduction in the 25%-rate IPRJ would be available to the project through application to SUDAM (Superintendência do Desenvolvimento do Amazônia – the Development Superintendency for the Amazonia Region), valid for 10 years from the initiation of commercial production. Coupled with the 9% social contribution tax on corporate profits, this has the net effect of reducing the effective rate of corporation tax from 34% to 15.25% over the first 10 years of operations.

The taxation regime applied in the cashflow model allows for the initial 10 years of production at a taxation rate of 15.25% of the taxable income. Following this period taxation is payable at the rate of 34% of the taxable income calculated after deducting all operating expenses and depreciation of capital items.

(ii) **Gross Revenues Taxes**: PIS and COFINS are federal taxes charged on gross revenues, on a monthly basis, under 2 regimes: cumulative and non-cumulative.

Current PIS and COFINS tax provisions were implemented in December 2002 (Law 10,637/02) and December 2003 (Law 10,833/03). As a result of such rules, the PIS and COFINS rates were set at 1.65% and 7.6% respectively and a credit mechanism introduced

Imports – PIS and COFINS are generally imposed on the import of goods and services at a combined rate of 9.25%.

(iii) **Indirect Taxes:** Both the Federal and State governments impose value-added tax (VAT) type taxes in Brazil. Each manufacturing plant or branch of a Brazilian company is generally considered an autonomous tax unit for both federal and state VAT purposes.

Federal Value Added Tax (IPI) Federal VAT (Imposto sobre Produtos Industrializados or IPI) is charged on imports of goods, on the first sale of imported goods and on transactions involving manufactured goods. Exports are tax exempt. The tax rate varies depending on the product traded and ranges from 0% to 365%. IPI paid on an import transaction or on local acquisitions generally becomes a tax credit to offset IPI charged on subsequent transactions. Special rules apply to the import and sale of fixed assets.

State Value-Added Tax (ICMS): State VAT (Imposto sobre operações relativas à circulação de mercadorias e sobre prestações de serviços de transporte interestadual e intermunicipal e de comunicações or ICMS) is levied on the import of goods and on the movement out of imported and manufactured goods, even if between branches of the sale legal entity. Exports are tax exempt. ICMS paid on imports as well as on local acquisitions generally becomes a tax credit to offset ICMS due on subsequent transactions. Special rules apply to the offset of ICMS tax credits associated with the acquisition of fixed assets.

ICMS tax rates vary according to the state where the company and the acquirer of the goods are located. Imports are generally subject to a 17% or 18% rate, while local transactions are subject to rates varying from 7% to 18%. Transactions involving taxpayers located in the states of the North, Northeast and Centre-West regions and Espírito Santo are subject to a 7% rate, while a 12% rate applies to transactions involving companies located in states of the Southeast and South regions.

ICMS is also charged on the provision of transportation services, communication and electricity.

(iv) **Application of Indirect Taxes – Capital and Operating expenses:** Where benefits and special regimes exist for indirect taxes, it has been assumed that these are applicable to the Project.

It has been assumed in this study that any credits built up on input taxes are offset against Federal taxes on inputs. This will require confirmation through a more detailed tax evaluation, including negotiations with the relevant authorities.

(v) **Other taxes relevant to the Araguaia Project**

It is assumed that the majority of capital equipment will be sourced in Brazil.

In the eventuality of items being imported where these are not available in Brazil, it is assumed that the 'Ex Tarifário" benefit is granted. This reduces the rate of import duty usually to 2%, or 0% where the Government considers that the fixed assets are of high importance for the development of the Brazilian economy. The 0% rate has been assumed.

Furthermore, the rate can be reduced to 0% if the fixed assets are imported from a country which has a trade agreement with Brazil (such as MERCOSUR) and provided that the goods have certificate of origin with an also 0% taxation in the country of origin. For this project an initial value of $15 million was allowed for previous expenditure not deducted.

The calculation of total tax paid in the financial model is presented in Table 21.5, below.

Table 21.5 Taxation

Item	$ million
Taxation	668.051

21.4 Trade off studies

Case 1: 2.7 Mtpa, contractor mining and haulage

A trade off study using contractor mining and haulage was undertaken to test the economics of the project at a plant throughput of 2.7 Mtpa (Option 1). The capital and operating costs are shown in Table 21.6 below.

Table 21.6 Base Case – Option 1 contractor

Item	Base Case (0.9 Mtpa) $ million	Option 1 (2.7 Mtpa) $ million
Capital	**645.488**	**1,593.131**
Pre-production capital costs	582.176	1,436.134
Production capital costs	63.313	156.997
Operating	**3,293.665**	**6,728.469**
Mining	552.998	733.581
Processing	2,641.667	5,884.891
Corporate overheads	**99.000**	**109.997**
Royalties (CFEM)	**43.301**	**84.841**
Taxation	**668.051**	**1,132.568**

Case 2: owner operator

A further study was undertaken using an owner operator mining and haulage fleet to provide a comparison with the Case 1 contractor study. The results are shown in Table 21.7 below.

Table 21.7 Base Case – Option 2 owner operator

Item	Base Case (0.9 Mtpa) $ million	Option 2 (2.7 Mtpa) $ million
Capital	**718.425**	**1,739.730**
Pre-production capital costs	611.739	1,486.105
Production capital costs	106.686	253.625
Operating	**3,106.320**	**6,436.750**
Mining	406.985	534.134
Processing	2,600.335	5,792.619
Corporate overheads	**99.000**	**109.997**
Royalties (CFEM)	**39.554**	**79.007**
Taxation	**700.321**	**1,166.497**

22 Economic analysis

22.1 The basis for choosing the Base Case

Several cases were studied to identify the financial implications to the Project.

The studies were:

- Plant throughput of 0.9 Mtpa using contractor mining and ore haulage
- Plant throughput of 2.7 Mtpa using contractor mining and ore haulage
- Plant throughput of 0.9 Mtpa using owner operator mining and ore haulage
- Plant throughput of 2.7 Mtpa using owner operator mining and ore haulage.

The Base Case chosen was the 0.9 Mtpa throughput using both contractor mining and ore haulage. The reasons for choosing this model are:

- The 2.7 Mtpa throughput returned the highest NPV8 however in doing so all of the reserves of ore at the Project were consumed leaving no room for flexibility over time
- The capital requirement for the 0.9 Mtpa model was significantly lower than the 2.7 Mtpa model
- The models using owner operator returned a higher NPV8 than the contractor models however the pre-production capital cost was reduced with the contractor model and this cost was offset by higher mining costs over the life of the mine
- The IRR was similar for all models
- The unit KPI were similar for all models
- The breakeven price for Ni was slightly lower for the 0.9 Mtpa than the 2.7 Mtpa cases.

22.2 Base Case economic headline results

The following Table 22.1 and Table 22.2 provide the project headline results before and after taxation:

Table 22.1 Base Case economic model headline results before taxation

Item	Unit	Value
Net Cashflow	$M	2,433.933
NVP_8	$M	730.067
IRR	%	22
Production year payback	years	4.1

Table 22.2 Base Case economic model headline results after taxation

Item	Unit	Value
Net Cashflow	$M	1,765.882
NVP_8	$M	519.233
IRR	%	20
Production year payback	years	4.4

22.3 General criteria

Snowden prepared an economic cashflow and financial analysis model based on inputs derived from mining and processing schedules as well as capital and operating cost estimates including royalties (CFEM) for the Base Case. The model was prepared from mining schedules estimated on a quarterly basis for the first 4 years of production and then annually for the remaining project life. All inputs are consolidated annually in this report.

The cashflow model was based on the following:

- 100% equity ownership by HZM
- Costing from 1 February 2013
- 2 year pre-production period for plant construction
- No cost escalation
- All costs reported in USD and where costs were estimated in Brazilian Reais the exchange rate used was 2.2 Reais to the USD.

The objective of preparing the cashflow model was to:

- collate all of the inputs for the following disciplines into a single model:
  - Mining
  - Processing
  - Metallurgical
  - Metal pricing
  - Pre-production capital costs
  - Production sustaining capital
  - Operating costs
  - Environmental costs
  - Social costs
  - Rehabilitation and closure costs
  - Royalties (CFEM) and taxation.
- be flexible to enable options to be valued
- provide sufficient information to management so that they are supported in any decision making process
- provide the basis for future studies.

The economic cashflow model was then interrogated to determine the following values pre and post taxation:

- Headline values:
  - Net cashflow
  - Net Present Value at 8% discount rate (NPV_8)
  - Internal Rate of Return (IRR).
- Key Performance Indicators:
  - Operating costs
  - Total costs
  - Production payback years.
- Breakeven nickel prices based on:
  - Net Cashflow
  - $NPV_{8.}$

22.4 Economic model inputs

Table 22.3, shows the inputs were used in the economic cashflow model:

Table 22.3 Base Case economic model inputs

Item	Unit	Value
Pre-Production period	Years	2.0
Life of project production	Years	24.75
LOM ore mined and processed	kt	21,206
LOM waste mined	kt	60,050
LOM Average Ni grade	%	1.66
LOM Average Fe grade	%	16.01
LOM Average Ni recovery	%	93.0
LOM Average Fe recovery	%	37.9
LOM Average product Ni grade	%	20.3
LOM Average product Fe grade	%	79.7
Plant throughput	Mtpa	0.9
LOM Ni Price	$/t	19,000
LOM Fe price	$/t	150

The LOM Ni price of $19,000 was used following a review of a consensus report produced by Consensus Economics Inc. in December 2013 and is based on forward price estimations from 19 analyst groups. The mean forward prices are shown in Table 22.4 and are discussed in more detail in Section 19.

Table 22.4 Forward consensus Ni prices

Years	Mean $/t Ni
2017	19,348
2018	20,061
2019 – 2023 (real)	19,109

Source: Consensus Economics Inc., December 2013

A review of was also made of historical prices, shown in Table 22.5.

Table 22.5 Historical Ni prices

Previous years	Mean $/t Ni
5	19,574
6	22,505
9	20,926
10	19,796

Source: Bloomberg

22.5 Economic model results

The economic cashflow model results are presented in Table 22.6 to Table 22.8, below.

Table 22.6 Base Case economic model headline results before taxation

Item	Unit	Value
Net Cashflow	$M	2,434
NVP_8	$M	730
IRR	%	22
Production year payback	years	4.1

Table 22.7 Base Case economic model results before taxation

Item	Unit	Value
LOM Ni recovered	kt	327
LOM Fe recovered	kt	1,286
Avg. Ni production at 0.9 Mtpa ore	ktpa	14.0
Avg. Fe production at 0.9 Mtpa ore	ktpa	54.5
LOM Ni revenue before royalties	$M	6,222.128
LOM Fe revenue before royalties	$M	192.858
Estimated salvage value	$M	1.400
Total revenue	$M	6,416.387
Total costs	$M	3,982.454
Operating cashflow	$M	3,078.021

Table 22.8 Base Case economic model headline results after taxation

Item	Unit	Value
Net Cashflow	$M	1,766
NVP_8	$M	519
IRR	%	20
Production year payback	years	4.4

22.6 Production summary

The physical production of the project Base Case is based on the mining, processing and recovery of metal. The net cashflow is before taxation and the results are shown in Table 22.9 and Table 22.10.

Table 22.9 Production physicals by period

Project year	Ore mined (kt)	Ore processed (kt)	Ni grade (%)	Ni in product (kt)	Fe in product (kt)
3	1,457	714	1.60	11	43
4	874	898	1.82	15	56
5	918	900	1.79	15	55
6	1,042	900	1.86	16	52
7	828	900	1.82	15	55
8	933	900	1.76	15	53
9	984	900	1.76	15	52
10	1,033	900	1.75	15	51
11	1,011	900	1.75	15	51
12	984	900	1.74	15	52
13	919	900	1.73	14	56
14	994	900	1.72	14	56
15	981	900	1.70	14	56
16	986	900	1.69	14	56
17	797	900	1.66	14	56
18	773	900	1.61	14	56
19	1,017	900	1.60	13	56
20	842	900	1.59	13	56
21	856	900	1.58	13	56
22	658	900	1.51	13	56
23	630	900	1.46	12	54
24	495	900	1.41	12	53
25	518	769	1.41	10	44
26	411	486	1.47	7	30
27	263	339	1.47	5	21

Table 22.10 Project financials by period

Project year	Revenue ($M)	Costs ($M)	Net cashflow ($M)	Prorata cash cost / lb Ni	Prorata cash cost / t Ni
1		169	-169		
2		414	-414		
3	208	121	86	4.96	10,943
4	297	138	158	3.97	8,749
5	293	139	154	4.02	8,872
6	304	142	162	3.97	8,746
7	297	141	156	4.05	8,929
8	288	142	147	4.19	9,229
9	288	142	146	4.20	9,249
10	287	142	145	4.21	9,282
11	285	142	143	4.24	9,340
12	284	142	142	4.25	9,374
13	283	142	141	4.26	9,394
14	282	142	139	4.30	9,485
15	278	142	137	4.33	9,543
16	276	142	135	4.36	9,623
17	273	142	131	4.42	9,956
18	265	142	124	4.55	10,224
19	263	143	120	4.62	10,025
20	262	141	121	4.59	10,181
21	260	142	118	4.65	10,115
22	249	140	109	4.78	10,541
23	241	140	101	4.93	10,877
24	233	139	93	5.10	11,247
25	198	126	72	5.39	11,889
26	131	97	33	6.27	13,812
27	93	74	18	6.66	14,685
28		10	-10		
29		8	-8		
Totals	6,416	4,651	1,766	4.48	9,883

The prorata cash costs include all direct operating costs plus royalties (CFEM). The costs are allocated in proportion to the value of Ni to the total value of the product produced.

22.7 Key performance indicators (KPI)

The Base Case LOM KPI's before taxation are presented in Table 22.11, below.

Table 22.11 Base Case KPI's before taxation

Item	Unit	Value
Value of product sold	$/t ore	302.58
Cash cost	$/t ore	157.36
Total cost	$/t ore	187.80
Production year payback	year	4.1
Prorata cash cost	$/lb Ni	4.48
Prorata cash cost	$/t Ni	9,883
Prorata total cost	$/lb Ni	5.35
Prorata total cost	$/t Ni	11,795
Brooke Hunt methodology C1 cost	$/lb Ni	4.16
Brooke Hunt methodology C1 cost	$/t Ni	9,166

The prorata cash costs include all direct operating costs plus royalties (CFEM), the prorata total costs include the prorata cash costs plus capital costs. The costs are allocated in proportion to the value of Ni to the total value of the product produced.

The Brooke Hunt methodology C1 costs include all direct operating expenses but do not include royalties (CFEM).

22.8 Sensitivity analysis

The economic cashflow model was used to prepare a sensitivity analysis for the NPV_8 for the Base Case before taxation. The sensitivity analysis was completed on the following variables:

- Grade of Ni
- Grade of Fe
- Recovery of Ni
- Recovery of Fe
- Price of Ni
- Price of Fe
- Pre-production capital
- Production capital
- Mining cost
- Processing cost
- Overhead cost.

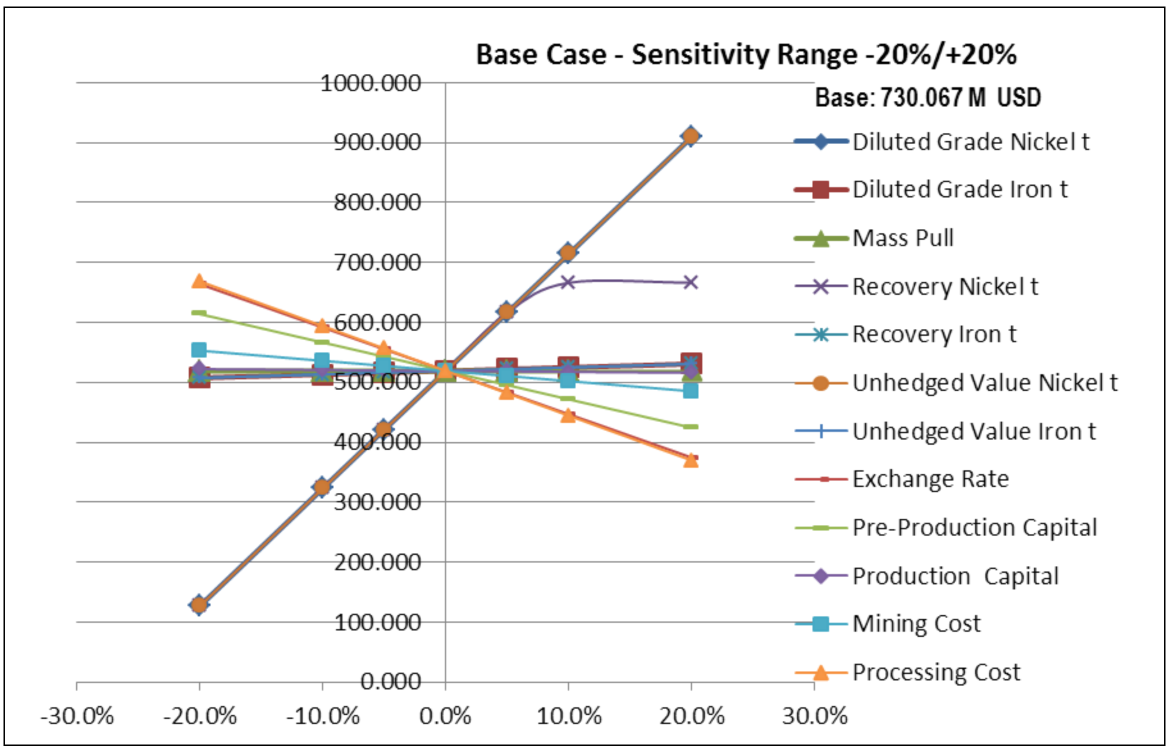
The sensitivity analysis determines how the NPV_8 is affected with changes to one variable at a time while holding the other variables constant. The results of the sensitivity analysis are presented in Table 22.12 below. In this table the B/E represents the breakeven and it indicates the change in the variable that will bring the project NPV_8 to $0.000. Elasticity is a measure of sensitivity that indicates for a 1% change in the variable what change in the NPV_8 will occur. A value greater than 1 indicates that the change in the variable will have a higher value change in the NPV_8 than the change in the variable and indicates a higher sensitivity to change.

Table 22.12 Sensitivity table for NPV_8 – before taxation

	-20%	-10%	-5%	0%	5%	10%	20%	B/E	Elasticity
Grade Ni	231	480	605	730	855	980	1229	-29%	3.4
Grade Fe	715	723	726	730	734	738	745	-	0.1
Mining reserves	678	723	728	730	750	773	813	-	0.1
Recovery Ni	231	480	605	730	855	918	918	-29%	3.4
Recovery Fe	715	723	726	730	734	738	745	-	0.1
Price Ni	231	480	605	730	855	980	1229	-29%	3.4
Price Fe	715	723	726	730	734	738	745	-	0.1
Pre-production capital	836	783	757	730	704	677	624	-	0.7
Production capital	734	732	731	730	729	728	726	-	0.0
Mining cost	774	752	741	730	719	708	686	-	0.3
Processing cost	922	826	778	730	682	634	538	76%	1.3
Overhead cost	750	740	735	730	725	720	710	-	0.1

The sensitivity chart is shown in Figure 22.1 below and it covers a range of variable changes from -20% to +20%. The flat line in the recovery of Ni indicates that the recovery cannot exceed 100%.

Figure 22.1 Sensitivity graph



22.9 Breakeven analysis

A Base Case before taxation breakeven analysis was undertaken on the Ni price for both NPV_8 and net cashflow. This analysis is conducted on the sensitivity analysis data and provides the nickel price which will bring either the NPV_8 or the net cashflow to $0. The results of this analysis are presented in Table 22.13.

Table 22.13 Breakeven analysis before taxation

Item	Unit	Breakeven Price
Net cashflow	$/t Ni	11,568
NPV_8	$/t Ni	13,442

22.10 Trade off studies

Case 1: 2.7 Mtpa, contractor mining and haulage

A trade off study using contractor mining and haulage was undertaken to test the economics of Option 1. Table 22.14 and Table 22.15 shows the headline values before and after taxation.

Table 22.14 Case 1 economic model headline results before taxation

Item	Unit	Base Case 0.9 Mtpa	Option 1 2.7 Mtpa
Net cashflow	$M	2,434	4,603
NVP_8	$M	730	1,637
IRR	%	22	24
Production year payback	years	4.1	3.6
Breakeven Ni price on NPV_8	$/t	13,442	13,633

Table 22.15 Case 1 economic model headline results after taxation

Item	Unit	Base Case 0.9 Mtpa	Option 1 2.7 Mtpa
Net cashflow	$M	1,766	3,470
NVP_8	$M	519	1,204
IRR	%	20	21
Production year payback	years	4.4	3.9
Breakeven Ni price on NPV_8	$/t	13,977	14,060

Case 2: owner operator

A further study was undertaken using an owner operator mining and haulage fleet (Option 2) to provide a comparison to the Case 1 study. The results are shown in Table 22.16 and Table 22.17 that show the headline values before and after taxation.

Table 22.16 Case 2 economic model headline results before taxation

Item	Unit	Option 2 0.9 Mtpa	Option 2 2.7 Mtpa
Net cashflow	$M	2,552	4,754
NVP_8	$M	762	1,685
IRR	%	22	24
Production year payback	years	4.2	3.6
Breakeven Ni price on NPV_8	$/t	13,200	13,474

Table 22.17 Case 2 economic model headline results after taxation

Item	Unit	Option 2 0.9 Mtpa	Option 2 2.7 Mtpa
Net cashflow	$M	1,852	3,587
NVP_8	$M	541	1,238
IRR	%	20	21
Production year payback	years	4.4	3.9
Breakeven Ni price on NPV_8	$/t	13,768	13,922

23 Adjacent properties

There is no information from adjacent properties applicable to the Araguaia Project for disclosure in this report.

24 Other relevant data and information

The authors and Qualified Persons are unaware of any other data or information that would be relevant to this Technical Report.

25 Interpretation and conclusions

25.1 Interpretations and conclusions

- The PFS mineral resource estimate is based on diamond core drillhole data and classified with respect to JORC Code (2012) guidelines.
- Reliability of the sample assay data is within acceptable limits for mineral resource estimation.
- There remains an opportunity to identify measured resource estimates through additional infill drilling and test pits to reduce the resource risk with respect to the economics of the Project (payback period).
- Due to the relatively small amount of sub-economic material overlying the resource, it is felt that all of the reported resources show reasonable prospects for economic extraction.
- There are no known factors related to metallurgical, environmental, permitting, legal, title, taxation, socio-economic, marketing or political issues which could materially affect the mineral reserve estimate.
- The metallurgical process would produce the target Fe-Ni alloy product at greater than 93% nickel recovery over a range of varying feed characteristics.
- Test samples and the comprehensive test work completed to date, for all considered lithologies, are considered to be representative of the mineral resource within the Project deposit areas.
- The mining system described for the exploitation of the Project is considered robust, low cost and achievable in terms of supporting the production requirements as detailed in this report. There is the potential for large barren “stones” to be encountered in the bottom of the mining pits; these could present certain difficulties when mining in the pit bottoms. It is recommended that test pitting be carried out to determine the frequency and extent of these stones and explore methodologies to handle them.
- The Project is in an area of Brazil that is well-supported by existing infrastructure.
- The proposed 230 kV transmission line would be readily approved and regulated by the relevant Government agencies.
- In summary, this PFS has established the Project as economically viable. A complete risk assessment has been conducted which considers the entire value chain of the Project as described within this report. As part of a risk management process, these risks will be addressed as part of the subsequent set of studies, including the feasibility study.
- Snowden concludes that the Project should be considered for further engineering evaluation through a feasibility study to refine the economics through better engineering definition.

25.2 Risks and opportunities

A full risk and opportunity assessment was completed for the Project.

A risk assessment was undertaken for the PFS that focused on the key elements of the value chain. These were:

- Resource estimation
- Mining
- Metallurgy & Process (Ore preparation, dryer, calcining, smelting, refining/granulation)
- Plant and Layout
- Infrastructure
- Geotechnical (Open Pit, Plant Area)
- Hydrology and Hydrogeology
- Social and Environmental
- Tenure and licencing
- Cashflow model (Financial model integrity).

Table 25.1 Risks and opportunities identified.

	Risks						
Category	High	Medium High	Medium	Medium Low	Low	Opportunities	Issues
Resource	0	0	0	0	0		1
Mining	0	2	3	3	0	2	2
Process & Metallurgy	23	28	22	21	5		
General	7	10	8	4	1		
Ore Prep	6	2	3	0	1		
Dryer	2	2	0	3	1		
Calcining	2	7	3	5	0		
Smelting	5	5	6	6	2		
Refining/Granulation	1	2	2	3	0		
Plant and layout	1	2	1	1	0		
Infrastructure	0	0	1	2	0		
Geotechnical	0	2	9	0	0		
Open Pit	0	0	9	0	0	1	
Plant Area	0	1	0	0	0	2	
Slag Dump	0	1	0	0	0	2	
Hydrology and Hydrogeology	2	4	6	2	1		
Social & Environmental	1	2	1	0	0		1
Tenure and licencing	5	1	0	0	0		
Cashflow model	4	1	3	0	0		
Total	**36**	**42**	**46**	**29**	**6**	**7**	**4**

From this assessment, Snowden has established five prevailing risk themes:

- Management and technical competency in relation to the high technical skill-sets required to manage and maintain a complex metallurgical process.
- Complexity with regard to the mineralisation and subsequent beneficiation and recovery of nickel product.
- Data requirements, acquisition, procedures and modelling.
- The water theme has a number of key questions which need to be satisfied.
- The tenure and licencing theme.

These are common themes are discussed below.

The management and technical competency theme

This relates to the technical knowledge, ability and willingness of potential employees (management and workforce) to commission and maintain a complex nickel plant with high operational tolerances in terms of throughput, control and instrumentation and maintenance.

Mitigation of this will require a human resource management response through human capital planning to ensure that appropriately skilled and experienced employees are employed at the required time. This is of high importance during the commissioning and ramp-up stages of the project.

The complexity theme

This relates to the complexities of mineability and treatability of nickel laterites.

The mitigation effort required will necessitate the simplification (where possible) of design and operational features, such as monitoring and control procedures, for both mining and metallurgical processes. The requirement would be design for useability and maintainability to ensure that operational readiness considers the simplification of processes both in design and in operational execution and management.

The data theme

This theme covers timely data acquisition that is sufficiently well scoped that it covers all later needs.

The PFS has identified a number of data and information requirements which need to be satisfied before the FS starts. The mitigation effort requires that all required data and information is acquired in a timely manner. This establishes the value of information and data as a necessary condition for efficient project delivery.

The water theme

This theme relates to the mine being in a part of the world exposed to seasonal variations of heavy rainfall and drought which poses subsequent design requirements in terms of controlling moisture within excavated material (ore or waste), productivity aspects relating to excavation, together with potential inundation, etc.

The mitigation effort requires the establishment of a water management capability which would encompass the overall mine. This would include operational features of mining which will have to consider a suitable response to maintain (or exceed) planned productivities for earthmoving machinery. There are a number of nickel laterite mines throughout the world and consideration of best practical solutions should be considered in subsequent studies.

The tenure and licencing theme

Delays of permitting caused by delays in approval of various permits and key documentation will have a direct impact on the critical path leading to production and therefore revenue generation.

The mitigation effort required is based on the inherent need to ensure advanced planning of key documentation for permits and permissions is achieved in a manner that is both achievable and reflects the execution schedule of the Project. It is Snowden's opinion that HZM maintains close engagement with the relevant authorities to ensure timely delivery of permits and permissions.

26 Recommendations

26.1 Geology

Snowden makes the following recommendations with respect to geology:

- Further resource investigations: Further work should be completed to provide additional confidence in the resource prior to commencing the Feasibility Study.
- These investigations should include:
  - Infill drilling to define a Measured Resource. Drill spacing to 50 m x 50 m is recommended, however in areas of geological complexity a drill spacing of 25 m x 25 m is recommended.
  - Test mining and trenching to better understand short scale variability.

26.2 Mining

Snowden makes the following recommendations with respect to mining:

Test mining:

- Mining conditions in nickel laterites can be varied and challenging. Test mining should be completed in selected mining areas, across the full laterite profile to better understand these conditions, specifically within the context of large stones that may be encountered in the pit bottom. This work will allow for a more accurate estimation of mining costs.
- This Mineral Reserve is calculated on the basis of currently available information. Snowden strongly recommends a test pit(s) to assess in-situ grade reconciliation to the resource model, incidence of barren rocks in the saprolite, mining recovery and mining dilution.

Blending and materials handling study:

- An integrated study into the location and orientation of the plant in relation to the ROM shed, the blending strategy prior to the plant along with the possibility of direct tipping material should be completed prior to the Feasibility Study.

Estimation of pisolitic ferricrete supply:

- Future resource models should estimate the volume of pisolitic ferricrete within the resource, such that accurate estimates of sheeting supply can be ascertained.

Sheeting demand:

- A more detailed model of sheeting demand should be developed, including sheeting requirements for ex-pit items such as roads, stockpiles and waste dumps.

Waste dump optimisation:

- Further work should be completed to trade-off the amount of material that is backfilled into the pit during mining, and also the location and geometry of the external waste dumps with respect to sterilisation, scheduling logistics, permitting, water management and stability.

More detailed scheduling:

- In the Feasibility Study, scheduling at a greater resolution and which included the consideration of changes in pit, dump, and road availability in certain wetter months.

Sourcing local equipment:

- Local mining equipment suppliers should be sought to gain quotations for the equipment specified in this report.

Source contractor quotations:

- Budget contractor quotations should be sought in order to validate the current mining cost estimate.

Government liaison:

- HZM should liaise with the Brazilian Government to ensure that the assumed taxation exemptions will be awarded.

Review of the site layout:

- The project team should revisit site and mark out areas disturbed by the operation to ensure that these locations are appropriate.

Mine water management plan:

- A detailed plan and design should be developed to manage water and effluent from the mine.

Geotechnical:

- The geotechnical models for the Project are of a preliminary nature due to limited geotechnical drilling. It is recommended to update the geotechnical models in the next stage of study with:
  - More geotechnical drilling in data limited sections:
  - Detailed definition of the laterite profile including the main material zones and their sub facies including basic engineering properties
  - Detailed definition of the special variability of the laterite profile and sub zone.
- Specialised laboratory tests (e.g. tri axial and direct shear) to better define the shear strength Parameters of the weaker material zones
- Laboratory compaction and CBR tests on ferricrete material to ascertain suitability as sheeting material
- Study the mine dewatering and depressurisation potential to confirm the design assumptions.

26.3 Metallurgy

26.3.1 Testwork

- For a the next stage of the project, it is recommended that particularly pilot plant calcining and smelting testwork should be planned to confirm some of the conclusions from this report. Importantly, such pilot scale tests will be designed to confirm correlations in key data such Fe-Ni metal grade and Fe-Ni ratio in ore against nickel recovery in smelting. The XPS testwork data, however, does provide important basic information with which to optimize the design of a pilot testwork programme.
- Having had association with a number of these operations, IGEO would recommend that for this PFS study, HZM endeavours to obtain more recent information and experiences on current kiln and furnace operations at Cerro Matoso. In conjunction with Hatch, operating staff at Cerro Matoso probably have had the most experience in dealing with furnace design and operations issues associated with ratios of SiO_2:MgO values above 2.0. From a previous site visit, however, it was understood that Cerro Matoso is probably now operating at SiO_2:MgO ratios of below 2.0.

26.3.2 Future work

Future work in this project could include the following:

- Evaluation of the best drying and pre-reduction technologies for the Projects’ laterites.
- Discussion with electric furnaces suppliers to evaluate the best design and have a CAPEX evaluation for the furnace. To consider copper coolers, deep immersion, electrodes configuration and electric heating (DC vs AC).
- Modelling of heat and mass balance for the Fe-Ni production plant with 2-3 configurations (pre-reduction, smelting and refining).
- Evaluation of sulphur and other impurities in Fe-Ni and layout of the Fe-Ni refining (literature, experimental test and modelling), also evaluate the alternative to form matte to lowering the liquidus temperature of the metal phase and at the present slag composition.

26.3.3 Recommendations

Following the proof of concept with laboratory trials at XPS and slag chemistry modelling at KPM, the following activities should be considered to advance in the next stage of the Project, being the feasibility study:

- Evaluation of the best drying and pre-reduction technologies for Horizonte laterite.
- Completion of a technical review including targeted plant visits and discussions with electric furnace suppliers as required to evaluate the best design, including documenting of current operating experience of similar plants as those proposed for the Project.
- Evaluate the best CAPEX appropriate for the design, which should consider copper coolers, deep immersion, electrode configuration and electric heating (DC vs AC). Discussion with electric furnaces suppliers to evaluate the best design and have a CAPEX evaluation for the furnace.
- Modelling of heat and mass balance for the Fe-Ni production plant with 2-3 configurations (pre-reduction, smelting and refining).

- Evaluation of sulphur and other impurities in Fe-Ni and layout of the Fe-Ni refining (literature review, experimental test and modelling), also evaluate the alternative to form matte as regards, lowering the liquidus temperature of the metal phase and at the present slag composition.
- A techno-economic evaluation should be performed, after completion of the above items.
- A 100 t (dry), bulk ore sample, or as a blended ore sample, or blended ore sample representative of the Araguaia laterite deposit, and suitable for pilot testing as recommended in Section 26.3.1 should be taken.. Smaller samples (of the order of 100 kg each) of the different types of ore (limonite, transition and saprolite) should also be taken for future smaller scale testwork.

Completion of a pilot scale test campaign incorporating the following features:

- Kiln drying of the representative bulk sample, including testing the in-kiln agglomeration action and the characteristics of the ore.
- Reduction kiln testing under conditions targeted to produce calcine suitable for the production of 20% Ni in Fe-Ni upon smelting.
- Electric furnace smelting test at 20% Ni in Fe-Ni
- A review of the quality of the Fe-Ni produced in the above electric furnace smelting test and development of the refining practice to refine this metal product.
- The development of a suitable heat and mass balance for the above steps.

26.4 Environmental and Social

The feasibility study will be undertaken in terms of international best practice, i.e. Equator Principles, International Finance Corporation (IFC) Performance Standards, and World Bank Environmental, Health and Safety (EHS) Guidelines. In addition, the project will also comply with the required Brazilian national legislation. WALM has been appointed to complete the Environmental Impact Assessment (EIA-RIMA), and to address any Brazilian licensing requirements.

26.4.1 Resettlement planning

It is likely that the project will require the purchase of land and relocation of certain households. The full extent of resettlement is not yet known but it is currently being investigated.

26.4.2 Completion of baseline specialist studies

The following issues are currently being investigated but were not completed in time for inclusion into this report:

- Resettlement
- Fauna
- Cultural heritage.

In addition, the following baseline studies need to be expanded upon:

- Vegetation – to determine species composition of areas that will be cleared of vegetation, and to determine whether any conservation important species will need to be relocated.
- Geohydrology – the specialist study must be expanded to address the other areas of the proposed mine. The current focus has been on the central Pequizeiro area. In addition, springs have not been sampled for hydrochemistry, although it is anticipated that this will be undertaken during the next sampling season.

Additional specialist studies may be required, and are dependent on the outcome of the stakeholder engagement process.

26.4.3 Social and environmental impact assessment

In order to comply with international best practice, the following aspects must be addressed within the Environmental Impact Assessment (EIA-RIMA):

Identification and assessment of risks and impacts

The project's potential impacts and risks must be addressed in terms of the area of influence:

- Area likely to be affected by the project activities and facilities that are a component of the project.
- Impacts from unplanned but predictable developments caused by the project that may occur later or at a different location.
- Indirect project impacts on biodiversity or on the ecosystem.
- Associated facilities, including facilities would not have been constructed or expanded if the project did not exist and, without which, the project would not be viable.
- Cumulative impacts that result from the incremental impact, on areas or resources used or directly impacted by the project, from other existing, planned or reasonably defined developments at the time the risks and impacts identification process is conducted.
- Where possible, the risks and impacts identification process will extend to those associated with primary supply chains.
- Greenhouse gas emissions, climate change and trans-boundary effects must be addressed. The impact identification process will also take account of the outcome of the engagement process with affected communities.

Risk Management and mitigation of impacts

Future management programmes to include mitigation and performance improvement measures to address identified environmental and social risks and impacts of the project. Mitigation must ensure that the project operates in compliance with applicable laws and regulations, and meets the requirements of International Finance Corporation's Performance Standards 1 to 8.

The mitigation hierarchy must favour avoidance of impacts over minimisation, and where residual impacts remain, compensation and/or offsets will be recommended wherever technically and financially feasible.

The management programme must indicate measureable events (as mitigation performance targets) where possible, with elements that can be tracked over defined time periods, and with estimates of resources and responsibilities required for implementation.

The following aspects must be incorporated into the risk management programme:

- Organisational capacity and competency
- Emergency preparedness and response
- Monitoring and review
- Rehabilitation and closure.

Stakeholder Engagement

Stakeholder engagement should be undertaken as per the Stakeholder Engagement Plan.

27 Certificates

27.1 Andrew F. Ross

CERTIFICATE of QUALIFIED PERSON

(a) I, Andrew F. Ross, Senior Principal Consultant of Snowden Mining Industry Consultants Pty Ltd., 87 Colin St., West Perth, Western Australia, do hereby certify that:

(b) I am a co-author of the technical report titled Araguaia Prefeasibility Study and dated 25 March, 2014 (the 'Technical Report') prepared for Horizonte Minerals.

(c) I graduated with an Honours Degree in Bachelor of Science in Geology from the University of Adelaide in 1972. In 1985 I graduated with a Master of Science degree in Geology from James Cook University of North Queensland. I am: a Fellow of the Australasian Institute of Mining and Metallurgy; a member of the Australian Institute of Geoscientists; licenced as a Professional Geoscientist with APEG (British Columbia). I have worked as a geologist continuously for a total of 42 years since graduation. I have been involved in resource evaluation consulting for 19 years, including resource estimation of nickel laterite deposits for at least 5 years. I have read the definition of "qualified person" set out in NI 43-101 ("the Instrument") and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements of a "qualified person" for the purposes of the Instrument.

(d) I visited the Araguaia Nickel Property from 22 to 24 November 2012.

(e) I am responsible for the preparation of sections 6, 7, 8, 9, 10, 11, 12, and 14 of the Technical Report, and contributed to preparation of sections 1, 2, 25 and 26 of the Technical Report.

(f) I am independent of the issuer as defined in section 1.5 of the Instrument.

(g) I have no prior involvement with the property that is the subject of the Technical Report.

(h) I have read the Instrument and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

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

Dated at Perth WA this 25 March 2014

Andrew Ross, BSc (Hons), MSc, MAIG, FAusIMM, CP (Geo), MGAA.

27.2 Harald Muller

CERTIFICATE of QUALIFIED PERSON

(a) I, Harald Muller, Senior Principal Consultant and Divisional Manager - Metallurgy of Snowden Mining Industry Consultants Pty Ltd., 87 Colin St., West Perth, Western Australia, do hereby certify that:

(b) I a the co-author of the technical report titled Araguaia Prefeasibility Study and dated 25 March, 2014 (the 'Technical Report') prepared for Horizonte Minerals.

(c) I graduated with a Bachelors degree in Chemical Engineering from Pretoria University and a Masters degree in Business Leadership from the University of South Africa. I am a Fellow of AusIMM, a Fellow of IChemE, a Chartered Engineer and a registered Professional Engineer, as well as a Fellow of SAIChE. I have worked as a metallurgist continuously for a total of 30 years since my graduation from university. I have worked in the process and project development of nickel laterite projects for at least 5 years.

(d) I have read the definition of 'qualified person' set out in National Instrument 43-101 ('the Instrument') and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements of a 'qualified person' for the purposes of the Instrument. I have been involved in mining and metallurgy related consulting practice for 3 years, including development and review of nickel laterite projects such as the Mirabela operation and the Dutwa Laterite project.

(e) I have not made a current visit to the Araguaia Nickel Property.

(f) I am responsible for the preparation of sections 13 and 17 of the Technical Report, and contributed to preparation of sections 1 and 18 of the Technical Report..

(g) I am independent of the issuer as defined in section 1.5 of the Instrument.

(h) I have not had prior involvement with the property that is the subject of the Technical Report.

(i) I have read the Instrument and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

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

Dated at Brisbane QLD this 25 March 2014

Harald Muller, B Eng (Chem), MBL, FAusIMM, FIChemE, FSAIChE, C Eng, Pr Eng.

27.3 Anthony Finch

CERTIFICATE of QUALIFIED PERSON

I, Anthony Finch, Senior Principal Consultant of Snowden Mining Industry Consultants Pty Ltd., 87 Colin St., West Perth, Western Australia, do hereby certify that:

(b) I am a co-author of the technical report titled Araguaia Prefeasibility Study and dated 25 March, 2014 (the 'Technical Report') prepared for Horizonte Minerals.

(c) I graduated with a degree in Mining Engineering from the University of Queensland in Australia in 1986. I am a Professional Engineer in the Province of British Columbia, number 164687, and a Member of the Australasian Institute of Mining and Metallurgy, number 103583 and a Certified Professional (Mining) of that Institute. Since graduation I have had 27 years continuous experience in the mining industry, in both operations and consulting, in various roles of increasing seniority. I have worked in hard rock underground mining, including precious metals, for over ten years, and in open pit mining for over ten years. By reason of my education, affiliation with a professional association, and past relevant work experience, I fulfil the requirements to be considered a "qualified person", as described in Section 1.1 of National Instrument 43-101, Standards of Disclosure for Mineral Projects (NI 43-101).

(d) I visited the Araguaia Nickel Property from 23 to 27 July 2013.

(e) I am responsible for the preparation of parts of sections 1, 2, and 3, and for sections,15,16,19 and 21-27 of the Technical Report.

(f) I am independent of the issuer as defined in section 1.5 of the Instrument.

(g) I have no prior involvement with the property that is the subject of the Technical Report.

(h) I have read the Instrument and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

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

Dated at Brisbane, QLD this 25 March 2014

Anthony Finch, P.Eng., MAusIMM (CP Mining), B.Eng., B.Econ.

27.4 Peter Theron

CERTIFICATE of QUALIFIED PERSON

(a) I, Peter J Theron, Principal Environmental Consultant of Prime Resources (Pty) Ltd Environmental Consultants; 70 – 7th Avenue, Parktown North, Johannesburg, South Africa, do hereby certify that:

(b) I am the co-author of the technical report titled Araguaia Prefeasibility Study and dated 25 March, 2014 (the 'Technical Report') prepared for Horizonte Minerals.

(c) I graduated with a Bachelor Degree in Civil Engineering from the University of Pretoria in 1985 and a Graduate Diploma in Environmental Engineering from the University of Witwatersrand in Johannesburg in 1995. I am a Member of the South African Institute of Mining and Metallurgy and a Professional Engineer. I have worked as an environmental and civil engineer consultant to the mining industry, continuously for a total of 28 years since graduation. I have been involved in international environmental consulting for the last 17 years. I have read the definition of "qualified person" set out in NI 43-101 ("the Instrument") and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements of a "qualified person" for the purposes of the Instrument.

(d) I visited the Araguaia Nickel Property from 23 to 27 July 2013.

(e) I am responsible for the preparation of sections 20 of the Technical Report, and contributed to preparation of parts of sections 1, 2, and 3, 25, and 26 and for sections 4, 5, 18, Technical Report.

(f) I am independent of the issuer as defined in section 1.5 of the Instrument.

(g) I have no prior involvement with the property that is the subject of the Technical Report.

(h) I have read the Instrument and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

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

Dated at Johannesburg, South Africa this 25 March 2014

Peter Theron, Pr Eng, SAIMM, B.Eng (Civil Eng), G.D.E

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